

Report to:



**TECHNICAL REPORT ON THE
KUTCHO PROPERTY –
BRITISH COLUMBIA, CANADA**

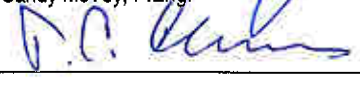
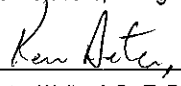
Project No. 0752370200-REP-R0002-00

Report to:



TECHNICAL REPORT ON THE KUTCHO PROPERTY – BRITISH COLUMBIA, CANADA

AUGUST 2007

Prepared by	 Sandy McVey, P.Eng.	Date	<u>9 AUG 2007</u>
Reviewed by	 Tom Stubens, P.Eng.	Date	<u>9 Aug 2007</u>
Authorized by	 Peter Wells, A.Sc.T, B.Comm.	Date	<u>9 August 2007</u>

WARDROP

Suite 800, 555 West Hastings Street, Vancouver, British Columbia V6B 1M1
Phone: 604-408-3788 Fax: 604-408-3722 E-mail: vancouver@wardrop.com

REVISION HISTORY

REV. NO	ISSUE DATE	PREPARED BY AND DATE	REVIEWED BY AND DATE	APPROVED BY AND DATE	DESCRIPTION OF REVISION
00	Aug. 9, 07	S.M. Aug. 8, 07	T.S. Aug. 8, 07	K.D. Aug. 8, 07	Final version issued to client

TABLE OF CONTENTS

1.0	SUMMARY	1-1
2.0	INTRODUCTION	2-1
3.0	RELIANCE ON OTHER EXPERTS	3-1
4.0	PROPERTY DESCRIPTION AND LOCATION	4-1
5.0	ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY	5-1
6.0	HISTORY	6-1
7.0	GEOLOGY.....	7-1
7.1	GEOLOGICAL SETTING	7-1
7.1.1	REGIONAL GEOLOGY	7-1
7.1.2	PROPERTY GEOLOGY	7-1
7.2	DEPOSIT TYPES.....	7-5
7.3	MINERALIZATION.....	7-7
7.3.1	MAIN DEPOSIT.....	7-7
7.3.2	SUMAC DEPOSIT	7-10
7.3.3	ESSO DEPOSIT	7-11
7.3.4	OTHER MINERALIZATION.....	7-11
7.4	EXPLORATION	7-12
7.5	DRILLING	7-12
7.6	SAMPLING METHOD AND APPROACH	7-17
7.7	SAMPLE PREPARATION, ANALYSES, AND SECURITY	7-17
7.7.1	HISTORICAL DATA	7-17
7.7.2	CURRENT DATA.....	7-19
7.7.3	CORE SAMPLING	7-19
7.7.4	ASSAYING	7-19
7.8	DATA VERIFICATION, QUALITY CONTROL, AND QUALITY ASSURANCE	7-20
7.8.1	HISTORICAL DATA	7-20
7.8.2	VALIDATION OF SML HISTORICAL ANALYSES	7-21
7.8.3	VALIDATION OF EMC HISTORICAL DATA	7-29
7.8.4	CURRENT DATA.....	7-29
7.8.5	CHECK ANALYSES	7-30
7.9	ADJACENT PROPERTIES.....	7-32
8.0	MINERAL PROCESSING AND METALLURGICAL TESTING.....	8-1
9.0	MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES.....	9-1

9.1	MINERAL RESOURCE ESTIMATE	9-1
9.1.1	DRILL HOLE DATABASE	9-1
9.1.2	SAMPLE ASSAY STATISTICS	9-1
9.1.3	GEOLOGIC (ROCK) MODELLING	9-4
9.1.4	DENSITY	9-6
9.1.5	COMPOSITING	9-6
9.1.6	COMPOSITE STATISTICS	9-7
9.1.7	OUTLIERS AND DECLUSTERING	9-10
9.1.8	TOPOGRAPHIC SURFACES	9-10
9.1.9	VARIOGRAM ANALYSIS	9-10
9.1.10	GEOLOGIC MODELLING	9-11
9.1.11	MODELLING PARAMETERS	9-11
9.1.12	MINERAL RESOURCE CLASSIFICATION	9-12
9.1.13	MINERAL RESOURCE SUMMARY	9-13
9.1.14	MODEL VALIDATION	9-14
9.1.15	VISUAL COMPARISON	9-14
9.1.16	GLOBAL STATISTICAL COMPARISON	9-14
9.1.17	LOCAL COMPARISON OF BLOCK AND DECLUSTERED COMPOSITE GRADES	9-14
9.1.18	MINERALOGICAL AND METALLURGICAL CORRELATION – MAIN DEPOSIT AND ESSO DEPOSIT	9-17
10.0	OTHER RELEVANT DATA AND INFORMATION	10-1
11.0	INTERPRETATION AND CONCLUSIONS	11-1
12.0	RECOMMENDATIONS	12-1
13.0	REFERENCES	13-1
14.0	CERTIFICATE OF QUALIFIED PERSONS	14-1

LIST OF APPENDICES

APPENDIX A	KUTCHO PROJECT 2006 DIAMOND DRILLING PROGRAM QA/QC STUDY
APPENDIX B	MEMORANDUM ON SPECIFIC GRAVITY DETERMINATIONS AND ESTIMATIONS FOR THE KUTCHO PROJECT
APPENDIX C	VARIOGRAMS

LIST OF TABLES

Table 4.1	Claim Listings	4-2
Table 7.1	Summary of Drill Holes	7-12
Table 7.2	Basic Statistics for the Main Deposit – Historical Drill Hole Assays	7-16
Table 7.3	Thickness and Grades of Adit and Surrounding Drill Holes	7-21
Table 7.4	Comparison of Intersections for EMC and Proximal WKM Drill Holes	7-29
Table 9.1	Summary SG Statistics	9-6
Table 9.2	Summary of Variogram Parameters	9-11
Table 9.3	Summary of Modelling Parameters	9-12
Table 9.4	Resource Summary	9-13

LIST OF FIGURES

Figure 4.1	Property Location Map	4-3
Figure 4.2	Claim Map	4-4
Figure 7.1	Regional Geological Setting.....	7-2
Figure 7.2	Schematic Cross Section of the KSA through the Kutcho Deposit Area	7-2
Figure 7.3	Property Geology with Historical Claim Outline and Surface Projection of Sulphide Deposits	7-4
Figure 7.4	Schematic Illustration of Reconstructed Property Stratigraphy – Vertical Exaggeration Approximately 10x	7-4
Figure 7.5	Computer Generated Image of the Three Known Massive Sulphide Deposits, Drill Holes and Surface Topography of the Kutcho Project.....	7-7
Figure 7.6	Cross-Section through the Central Part of the Kutcho Deposit.....	7-9
Figure 7.7	Internal Stratigraphy of the Main Deposit — Vertical Exaggeration Approximately 10x.....	7-10
Figure 7.8	Longitudinal Section Projected to Surface Showing Drill Hole Pierce Points in the Main Deposit	7-14
Figure 7.9	Vertical Projection of Longitudinal Section of the Esso Deposit Drill Hole Pierce Points	7-15
Figure 7.10	Cu Comparison	7-22
Figure 7.11	Zn Comparison.....	7-23
Figure 7.12	Ag Comparison	7-24
Figure 7.13	Au Comparison	7-25
Figure 7.14	S% Comparison	7-26
Figure 7.15	Cu Standard Results for Historical Core Intersection Resample	7-27
Figure 7.16	Zn Standard Results for Historical Core Intersection Resample.....	7-27

Figure 7.17	Ag Standard Results for Historical Core Intersection Samples.....	7-28
Figure 7.18	Cu and Zn Blank Results for Historical Core Intersection Samples	7-28
Figure 7.19	Blank Results for Cu and Zn – 2006 Drilling Program.....	7-30
Figure 7.20	Plot of Copper Analyses by Chemex and EcoTech.....	7-31
Figure 7.21	Zn Comparison for Five Resampled Drill Holes	7-31
Figure 9.1	Main Zone Assay Statistics	9-2
Figure 9.2	Esso/Sumac Assay Statistics.....	9-3
Figure 9.3	Perspective View of Main Geologic Solid – Looking Southeast.....	9-5
Figure 9.4	Typical Section (437700E) with NSR Values and Solid – Looking East	9-5
Figure 9.5	Main Zone 2.5 m Composite Statistics	9-8
Figure 9.6	Esso/Sumac Zones 2.5 m Composite Statistics.....	9-9
Figure 9.7	Main Zone Resource Classification – Looking Southeast.....	9-13
Figure 9.8	Main Zone Swath Plots Cu	9-15
Figure 9.9	Esso/Sumac Zone Swath Plots Cu	9-16

GLOSSARY

ABBREVIATIONS AND ACRONYMS

ALS Chemex Laboratories	Chemex
American Reserve Mining Corporation.....	ARMC
Atna Resources Limited	Atna
Barrick Gold Corporation	Barrick
coefficient of variation.....	COV
Copper Equivalent.....	CuEq
EcoTech Laboratory Limited.....	EcoTech
electromagnetic	EM
electromagnetic geophysical surveys	UTEM
Environmental Assessment	EA
Esso Minerals Canada	EMC
Footwall Zone	FWZ
Homestake Canada Limited	Homestake
induced coupled plasma.....	ICP
King Salmon Allochthon	KSA
McElhanney Engineering Services Limited	MESL
mineral title opinion.....	MTO
National Instrument 43-101	NI 43-101
net smelter return	NSR
potential acid generating	PAG
Quality Assurance/Quality Control.....	QA/QC
specific gravity	SG
Sumac Minerals Canada	SMC
Sumac Mines Limited	SML
Teck Cominco Limited	TCL
Tuff-Argillite Unit.....	TAU
volcanic hosted massive sulphide	VHMS
volcanogenic massive sulphide	VMS
Wardrop Engineering Incorporated.....	Wardrop
Western Keltic Mines Incorporated.....	WKM

UNITS OF MEASURE

Centimetre	cm
Cubic centimetre	cm ³
Cubic foot.....	ft ³
Cubic inch	in ³
Cubic metre.....	m ³
Day	d
Days per week	d/wk
Degree	°
Degrees Celsius.....	°C
Degrees Fahrenheit	°F
Diameter	Ø
Foot.....	ft
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than.....	>
Hectare (10,000 m ²).....	ha
Horsepower.....	hp
Hour	h or hr
Inch	"
Kilo (thousand).....	k
Kilogram.....	kg
Kilograms per cubic metre	kg/m ³
Kilograms per hour.....	kg/h
Kilograms per square metre.....	kg/m ²
Kilometre.....	km
Kilometres per hour.....	km/h
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Less than	<
Litre.....	L
Litres per minute	L/m
Megapascal.....	MPa
Megawatt	MW
Metre.....	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne).....	t
Micrometre (micron)	µm
Miles per hour	mph
Milligram	mg
Milligrams per litre	mg/L
Millilitre.....	mL
Millimetre	mm
Million.....	M
Million tonnes	Mt
Minute (plane angle)	'
Minute (time).....	min
Month.....	mo
Ounce	oz
Parts per billion	ppb
Parts per million	ppm

Percent	%
Percent moisture (relative humidity).....	% RH
Pound(s)	lb
Second (plane angle).....	"
Second (time).....	s
Short ton (2,000 lb)	st
Specific gravity.....	SG
Square centimetre.....	cm ²
Square foot	ft ²
Square inch.....	in ²
Square kilometre.....	km ²
Square metre	m ²
Thousand tonnes	kt
Tonne (1,000 kg).....	t
Tonnes per day	t/d
Tonnes per hour.....	t/h
Tonnes per year.....	t/a
Week.....	wk
Weight/weight	w/w
Wet metric ton.....	wmt
Yard	yd
Year	y

1.0 SUMMARY

The Kutcho Creek project area is situated 100 km east of the town of Dease Lake, and 330 km north of Smithers in northern BC. The claims cover an area of approximately 5,500 ha. Western Keltic Mines Inc. (WKM) owns the claims through two separate purchase agreements. One agreement is with Barrick Gold Inc. (a subsidiary of Barrick Gold Corporation) and AMI Resources Inc., who had 80% and 20% ownership, respectively, in all of the claims except the 16 SMRB claims and the 30 KC claims. The other agreement is with Sumac Mines Inc., a subsidiary of Sumitomo Metal Mining Co. Ltd. The claims are subject to net smelter return (NSR) royalties; in the case of the Barrick claims the NSR is 2%, and in the case of the Sumac claims the NSR is 3% beginning 36 months after achieving commercial production. Additional web-based Mineral Titles Online (MTO) cell claims were staked to fill gaps created during the conversion process and to protect the local access road and airstrip, including potential explosives manufacturing and storage areas.

3D geological solid models of the Main, Esso, and Sumac deposits were developed by WKM and used as the bounding domains for grade interpolation in this estimate. The general methodology was as follows:

- Cross sections were plotted and checked against historical sections for accuracy of location and metal grades.
- Polygons were generated on the sections to represent potentially mineable material, generally a US\$15 cutoff was used for the Main deposit and US\$35 cutoff was used for the Esso deposit; however, some lower grade material was included to maintain continuity in the down-dip direction and to maintain a minimum 3 m horizontal thickness. No cutoff grade was used for the Sumac deposit, as the shell was matched directly to the outline of the massive sulphide rock type.
- The sectional polygonal interpretations were linked together to create 3D geologic solids that were then used to code the rock type block model. The geology model was then used to constrain the grade interpolation.

The 3D geological models of the Main, Esso, and Sumac Zones were used to code the rock type model in Gemcom GEMS modelling software. The 3D solids were then used to tag blocks falling within the solids, with the percentage being stored to allow accurate reporting of volumes (partial block model). Additional 3D solids and surfaces were used to create a potentially acid generating (PAG) model for use in waste planning.

Separate block models were developed for the Main Zone and for the Esso/Sumac Zones because of the significant strike distance between them. Ordinary kriging was used for all areas.

The specific gravity (SG) data used for resource estimation purposes is a combination of measured values and calculated values, and was tabulated by interval in the assay database table. Sumac Mines Limited (SML) conducted SG determinations in the field using a “weigh in water, weigh in air” method, and also had ALS Chemex Laboratories (Chemex) of North Vancouver perform density measurements on uncrushed core from holes KT040 through KT049. Additionally, SML went back to the old core and carried out SG measurements in the field on the main or significant grade parts of the intersections in most (but not all) of the holes drilled prior to hole 40.

The calculated SGs are based primarily on various formulas relating the metal and sulphide content to SG, as documented in a March 2007 memorandum titled “Memorandum on Specific Gravity Determinations and Estimations for the Kutcho Project” that is attached as Appendix B. Sulphur and SG have a very strong correlation in the data that has both analyses, which is the majority of the database. Tim Carew, P.Geo., has reviewed this report and considers the conclusions and methodology reasonable.

The mineral resource has been classified using the mean distance to the samples used to interpolate a particular block, and the kriging variance. An average distance of 30 m and relative kriging variance of 0.6 (based on examination of kriging variance relative to pierce points in the plane of the deposit) was used to identify Measured material. The average distance was increased to 70 m for Indicated material and an average distance greater than 70 m identified Inferred material.

Mineral resources were classified in accordance with definitions provided by CIMM as stipulated in NI 43-101. The Kutcho mineral resources are classified as Measured, Indicated, and Inferred. Based on these parameters, Tim Carew, P.Geo., estimated that the Main Deposit, the Sumac Deposit, and the Esso Deposit contain approximately 17.7 Mt of resource in the indicated and measured categories, grading 1.71% copper, 2.36% zinc, 0.34 g/t gold, and 27.45 g/t silver. Additionally, all three deposits contain 11.8 Mt of inferred mineral resource grading 1.0% copper, 1.58% zinc, 0.17 g/t gold, and 15.55 g/t silver.

2.0 INTRODUCTION

Western Keltic Mines Inc. (WKM) has commissioned Wardrop Engineering Inc. (Wardrop) to prepare a Technical Report for the Kutcho Project in northern British Columbia, Canada. Previous studies, including engineering and design studies, were completed by Wright Engineers Limited in April 1985.

Tim Carew, P.Geo., who visited the property on 11 June 2007, completed the mineral resource estimate. Wardrop began work in April of 2007, which entailed incorporating this estimate into a new Technical Report for the Kutcho deposits.

3.0 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report disclaim responsibility for reliance on information provided by the following experts who are not Qualified Persons.

In preparing this report, Wardrop Engineering Inc. (Wardrop) has relied on information provided by Western Keltic Mines Inc. (WKM) for matters pertaining to environmental, socioeconomic, and permitting issues. Wardrop did not carry out a title search for the property, but have relied on information provided by Miller Thomson, LLP, WKM's legal counsel.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Kutcho Creek project area is situated 100 km east of the town of Dease Lake, and 330 km north of Smithers in northern BC (Figure 4.1). The property appears on NTS map sheet 104I/1, and geodetic coordinates for the center of the claim area are 58°12'N and 128°22'W. The claims cover an area of approximately 5,500 ha. As a result of overlap between historical claims, the sum of the individual claim areas are much greater than the actual total claim area. Claims are shown in Figure 4.2 and listed in Table 4.1 below. Western Keltic Mines Inc. (WKM) owns the claims through two separate purchase agreements. One agreement is with Barrick Gold Inc. (a subsidiary of Barrick Gold Corporation) and AMI Resources Inc., who had 80% and 20% ownership, respectively, in all of the claims except the 16 SMRB claims and the 30 KC claims. The other agreement is with Sumac Mines Inc., a subsidiary of Sumitomo Metal Mining Co. Ltd. The claims are subject to net smelter return (NSR) royalties; in the case of the Barrick claims the NSR is 2%, and in the case of the Sumac claims the NSR is 3% beginning 36 months after achieving commercial production. Most of the original claims have been converted to the new web-based Mineral Titles Online (MTO) cell claims administered by the Ministry of Energy, Mines and Petroleum Resources. Additional MTO cell claims were staked to fill gaps created during the conversion process and to protect the local access road and airstrip, including potential explosives manufacturing and storage areas.

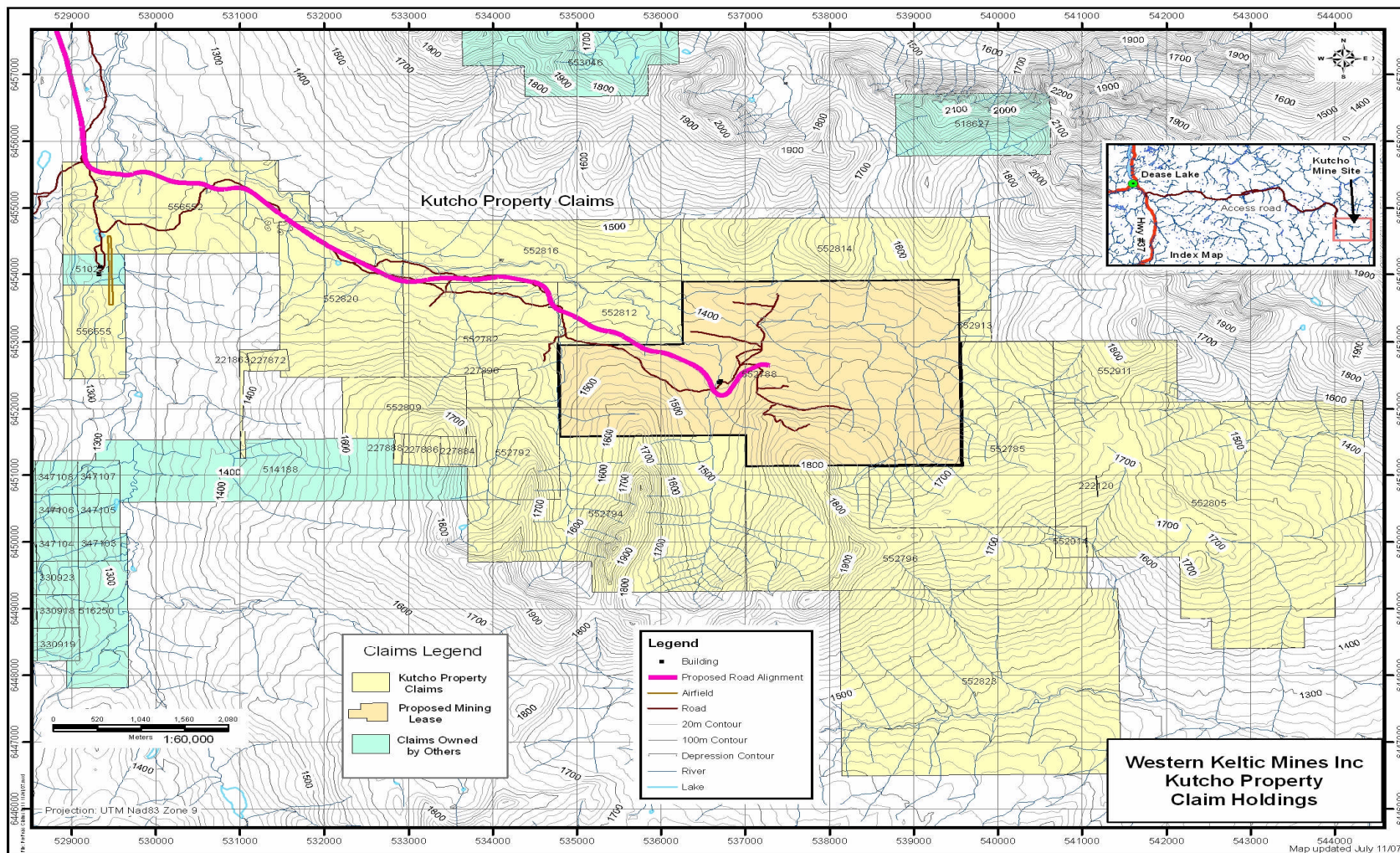
Table 4.1 Claim Listings

Tenure No.	Tenure Type	Claim Name	Owner	Map #	Good To Date	Status	Mining Division	Area (ha)	Tag #
221863	Mineral	LIN NO 1 FR	140550 (100%)	104I028	2017/Jan/31	GOOD	LIARD	25.0	38345
222120	Mineral	JEFF 114 FR	140550 (100%)	104I019	2011/Jan/31	GOOD	LIARD	25.0	72858
227872	Mineral	LIN #11	140550 (100%)	104I028	2017/Jan/31	GOOD	LIARD	25.0	459823M
227884	Mineral	KC 3	140550 (100%)	104I028	2017/Jan/31	GOOD	LIARD	25.0	248603M
227886	Mineral	KC 5	140550 (100%)	104I028	2017/Jan/31	GOOD	LIARD	25.0	248605M
227888	Mineral	KC 7	140550 (100%)	104I028	2017/Jan/31	GOOD	LIARD	25.0	248607M
227896	Mineral	KC 18	140550 (100%)	104I028	2010/Jan/31	GOOD	LIARD	25.0	248618M
552782	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		306.871	
552785	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		409.337	
552788	Mineral		140550 (100%)	104I	2010/Jan/31	GOOD		1091.244	
552792	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		153.501	
552794	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		597.091	
552796	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		494.787	
552805	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		1074.737	
552809	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		136.422	
552812	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		136.373	
552814	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		357.899	
552816	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		306.769	
552820	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		340.915	
552823	Mineral		140550 (100%)	104I	2017/Jan/31	GOOD		921.827	
552911	Mineral	PASS1	140550 (100%)	104I	2008/Feb/27	GOOD		136.406	
552913	Mineral	ADD1	140550 (100%)	104I	2008/Feb/27	GOOD		17.048	
552914	Mineral	ADD2	140550 (100%)	104I	2008/Feb/27	GOOD		17.062	
556552	Mineral	ADD3	140550 (100%)	104I	2008/Apr/17	GOOD		374.877	
556555	Mineral	ADD4	140550 (100%)	104I	2008/Apr/17	GOOD		102.288	

Figure 4.1 Property Location Map



Figure 4.2 Claim Map



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

Access to the property is by fixed-wing aircraft and helicopter from Smithers or Dease Lake to the 900 m gravel airstrip located at the junction of Kutcho and Andrea Creeks. The deposit area of the property is connected to the airstrip by a 10 km road. Currently this road has had culverts removed and is only passable to four-wheel drive trucks with good clearance. Four-wheel drive access to the property via the tote road to Dease Lake is available during the late summer and early fall, but is somewhat dependant upon weather due to extensive muddy sections.

The property is located within the Cassiar Mountains, just to the north of the continental divide between the Arctic and Pacific watersheds. The area is moderately rugged with elevations ranging from 1,400 to 2,200 metres. Most of the area is alpine with tree line at approximately 1,500 metres. Snow cover can persist for nine months of the year, particularly on shady north-facing slopes. Winters are cold and dry, while summers are cool and moist. Average annual temperature is -1°C, and approximately half of the average annual precipitation of 50 cm occurs as snow.

Structural fabric and two periods of glaciation have produced an intersecting pattern of east-west and north-south ridges and valleys. The major valleys are commonly filled with a deep layer of glacial till and outwash gravels.

6.0 HISTORY

Mineralization on what was to become the Kutcho property was first discovered in 1968 by a joint venture exploration operated by Imperial Oil Ltd. The discovery was made by prospecting in response to anomalous stream sediment samples collected during a regional drainage survey. Twenty claims were staked by W. Melnyk directly over the as of yet undiscovered main Kutcho sulphide deposit. These claims were allowed to lapse when the other partners in the joint venture declined to fund further exploration. After the statutes of the joint venture agreement expired, Imperial Oil returned to the area in 1972 in order to re-stake the area. However, Sumac Mines Ltd. (Sumac) had conducted stream sediment sampling earlier that season, and in response to anomalous samples R. Britten staked eight “two-post” claims along the anomalous stream and eight more claims along the geological strike direction, resulting in the cruciform claim outline overlying the western part of the main Kutcho sulphide deposit. Imperial Oil (later becoming Esso Minerals Canada Ltd.) staked a much larger area encompassing Sumac’s claims.

Beginning in 1973 both Sumac and Esso carried out exploration work, and their early successes prompted additional staking which resulted in claim boundaries roughly as they are today. Diamond drilling commenced in 1974, and by 1982 approximately 60,000 metres had been drilled by both companies, defining three sulphide lenses. During this time Esso also drilled a number of exploration targets in other areas of the property with moderate success. Environmental, metallurgical, and engineering studies were begun by both groups in 1980. A partnership agreement on engineering and development work was signed by Esso and Sumac in 1983, made retroactive to 1981 (the year Sumac began work driving the adit in order to collect a 100 tonne bulk sample). The agreement was essentially a 50/50 joint venture for development work, and culminated in a prefeasibility study by Wright Engineers Limited in 1985. This study indicated an 11% internal rate of return when using a copper price of US\$0.95. Given the risk factors involved and long-term price projections for copper below the US\$0.95 level, the companies put the project on hold pending further exploration results. A limited amount of exploration work was done on Esso’s claims to the south of the main mineralized trend between 1985 and 1988; however, this work and the numerous geophysical surveys that had been undertaken indicated limited potential for additional open pit mineralization.

In 1989, Esso sold most of its mining assets to Homestake Canada Ltd. (Homestake). In 1990, Homestake optioned the Kutcho property to American Reserve Mining Corporation (ARMC), who funded a \$1.1 M exploration program (Homestake remained the operator) which included 7,031 m of drilling in 28 holes, mostly in outlying target areas (Homestake remained the operator, thereby earning a 20% interest). Exploration successfully confirmed the presence of extensive areas of favourable geology and alteration indicative of hydrothermal activity, but failed to

discover zones of potentially economic mineralization. For example, 10 km southwest of the Kutcho deposit, a narrow zone of cryptocrystalline massive pyrite with a strike length in excess of 5 km was intersected in four widely-spaced drill holes, but was barren of base or precious metals. ARMC carried out engineering studies but did no further exploration work, relinquishing the option in 1993 while retaining a 20% interest in Homestake's property. The property was optioned to Teck Cominco Ltd. (TCL) in 1992. TCL carried out deep penetration electro-magnetic geophysical surveys (UTEM) over the Esso West Zone with the goal of defining additional conductors along the Kutcho trend. Due to extensive cover of conductive argillaceous units in the hanging wall, the UTEM system was unable to detect the Esso West deposit or other conductors at depth, leading TCL to drop their option. Homestake was purchased by Barrick Gold Corp. (Barrick) in 2003.

Extensions of the Kutcho stratigraphy to the west have been staked and worked by various companies in the past. Shortly after the discovery of the Kutcho deposits, Noranda staked the Kutcho formation to the west of Kutcho Creek. Noranda conducted geophysical surveys and carried out a small drilling program. The claims were allowed to lapse and were re-staked in 1995 by Gary Belik. Mr. Belik carried out a detailed mapping program and optioned the claims to Atna Resources Ltd. (Atna) in 1997. Atna conducted UTEM geophysical surveys and an extensive drilling program. Results of Atna's work were mixed, and although no deposits were discovered, significant but weak to moderately mineralized alteration zones were intersected. Structural complexity and lack of clear geophysical targets prevented additional work and the option was terminated.

Negotiations by Western Keltic Mines Inc. (WKM) to purchase the property from Barrick and Sumitomo were initiated in 2003 and concluded in early 2004. WKM carried out diamond drilling within the Main and Esso deposits during 2004 to confirm historical results and obtain material for metallurgical studies. A second round of drilling by WKM in 2005 tested the Main deposit's potential for up-dip and down dip extensions, as well as western extensions to the Esso deposit. The Sumac deposit was also drilled in 2005 to test for higher grade zones. A third round of drilling in 2006 focused on infill drilling within the five-year pit area of the Main deposit. The Kutcho property was entered into the Mine Development Review Process in 2006 and Environmental Assessment (EA) studies were initiated to provide baseline data for provincial and federal EA reviews.

7.0 GEOLOGY

7.1 GEOLOGICAL SETTING

7.1.1 REGIONAL GEOLOGY

The Kutcho property lies within the King Salmon Allochthon (KSA), a narrow belt of Permian-Triassic island arc volcanic rocks and Jurassic sediments, sandwiched between two northerly-dipping thrust faults: the Nahlin fault to the north, and the King Salmon fault to the south (Figure 7.1). Penetrative foliation and axial planes of major folds are parallel to these east-west trending bounding faults. The belt of volcanic rocks is thickest in the area where it hosts the volcanogenic massive sulphide (VMS) deposits, partly due to primary deposition, but also to stratigraphic repetition by folding and possibly thrusting. The KSA is terminated to the east (near the eastern edge of the property) by the Kutcho strike-slip fault (Gabrielse, 1978), but extends to the west for hundreds of kilometres. However, Kutcho Formation rocks thin to the west, and do not occur or are rarely exposed 10 km to the west of Kutcho Creek.

Stratigraphy of the KSA consists primarily of the Kutcho Formation, which is overlain by the limestone of the upper Triassic Sinwa Formation, which in turn is overlain by sediments (predominately argillite) of the Lower Jurassic Inklin Formation. Major folds are delineated by the Sinwa limestone and, where the Sinwa is absent, by the contact between the Kutcho and Inklin Formations (Figure 7.2).

7.1.2 PROPERTY GEOLOGY

STRATIGRAPHY

Stratigraphy of the Kutcho property has been described by Bridge (1984), Thorstad (1983), and Holbek (1985) and will only be briefly reviewed here. A property plan map is given in Figure 7.3, and a generalized reconstructed stratigraphic section is presented in Figure 7.4. Stratigraphy is best understood in the upper part of the Kutcho Formation, where units are better exposed and drill information is available. The footwall stratigraphy (particularly away from the deposit area) is not well understood.

The lowest rocks in the section are exposed on the southern ends of Imperial and Sumac Ridges and include interbedded basalts, basaltic tuffs and wackes, rhyolitic lapilli tuffs, and possible trondhjemite. The mafic rocks are fine to very fine-grained, chloritic, equigranular to weakly porphyritic, and are commonly given the field term of greenstone. The lapilli tuffs are pale grey and siliceous, and commonly contain very fine quartz phenocrysts and lenticular fragments from 0.5-3.0 cm in length. Textures

can only be seen on weathered (but lichen-free) surfaces. The trondhjemite unit is somewhat equivocal. It is described by Pearson and Pantaleyev (1975) and Bridge et al. (1983) as a fine-grained, equigranular, plagioclase-rich unit, but it is very similar to some of the tuffaceous units as well. A weak but pervasive carbonate-chlorite-pyrite or propylitic alteration of this unit is subtle but discernible.

Figure 7.1 Regional Geological Setting

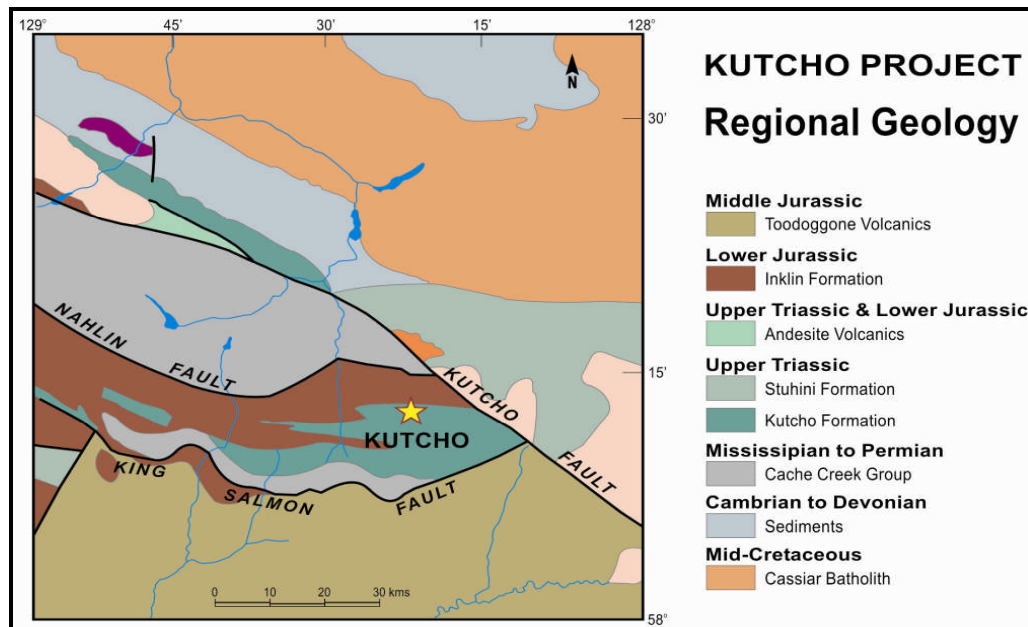
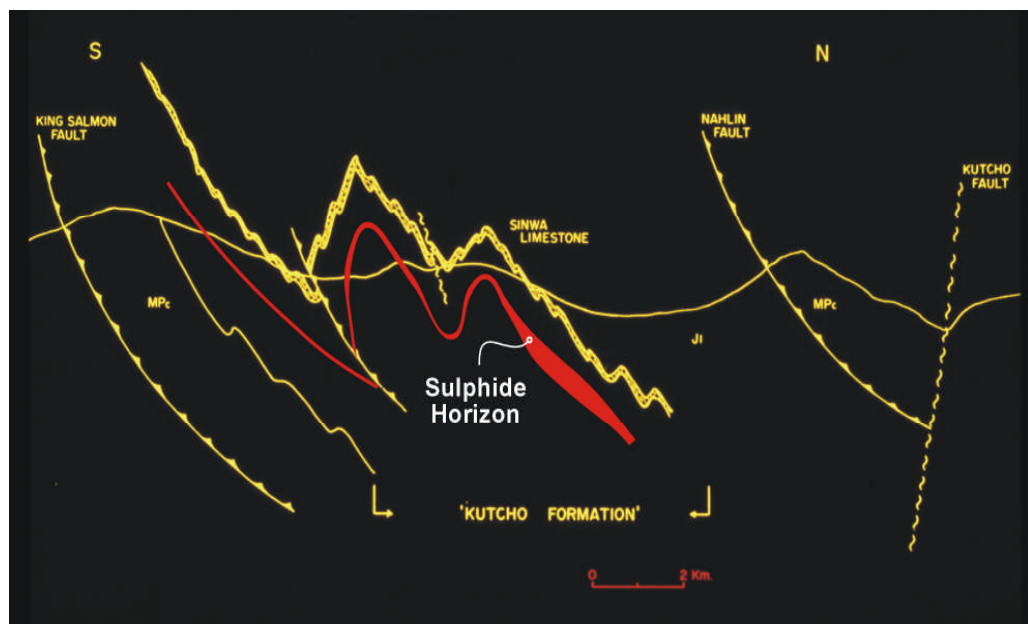


Figure 7.2 Schematic Cross Section of the KSA through the Kutcho Deposit Area



Rocks overlying the greenstone-lapilli tuff package have been termed the “ore-sequence,” and consist of lapilli tuffs, crystal-lithic tuffs, and quartz and quartz-feldspar crystal tuffs. Away from the deposit area these units tend to be thin, interbedded, and variably but weakly altered. Fine quartz-crystal ash tuff with silica-rich laminations and rare thin zones of ferroan dolomite typically mark the distal exhalative zone. The sulphide zones occur at (or near to) the contact between footwall lapilli tuff and hanging wall quartz-crystal tuff. In general, both lapilli fragments and phenocrysts are much coarser grained in the vicinity of the deposits, becoming progressively finer grained to the south and west. The quartz-feldspar crystal tuff is quartz-rich near the deposits, becoming more feldspar-rich to the south. A large zone of feldspar crystal tuff with almost no free quartz occurs a few hundred metres south of the sulphide zones, and whether this unit is footwall, hanging wall, or a facies equivalent to the quartz-feldspar crystal tuff is as yet indeterminate. An interesting feature is the occurrence of a coarse breccia texture within the quartz-feldspar crystal tuff immediately over the sulphide zones. The breccia fragments are typically subround, from 2–30 cm in size, and are identical to crystal tuff matrix except for an increase in the amount of epidote from 1–2% to closer to 10%. This feature has been interpreted to be a debris flow of semi-consolidated crystal tuff, shed from a flow dome complex and trapped in the graben or half-graben like structure which hosts the sulphide lenses.

Rocks between the ore sequence and the overlying conglomerate unit are referred to as the Tuff-Argillite Unit (TAU), and consist of gabbroic to basaltic intrusive sills and dykes, greywackes, and argillite. In the area of the deposit the gabbroic units are commonly coarse-grained and are commonly referred to as metagabbro. Higher in the section and both to the east and west of the Kutcho deposit this mafic unit becomes much finer grained, and an intrusive origin is not so clearly identified. The amount of argillite increases in a westerly direction, supporting the concept that this direction is towards the marine basin. The base of the TAU is interpreted to be a thrust fault, and there are numerous other fault zones within the unit as noted in drill cores and the adit. The basal thrust plane does not cause significant offset of the Sinwa limestone in the fold nose to the west, implying a scissor-type action with increasing movement to the east.

Overlying the TAU and truncating it to the west is the Kutcho Conglomerate. This unit is a heterolithic, fragment-supported conglomerate composed of subrounded clasts, ranging in size from 1–38 cm (long axis), and derived from all of the underlying lithologies. The conglomerate is conformably overlain and transitional into the Sinwa limestone, which in turn appears to be conformably overlain by Jurassic aged Inklin Formation argillite. (However, it is quite possible that there could be a contact between Kutcho Formation argillite and Inklin Formation argillite higher in the section, which would be difficult to spot and could be unconformable).

The Kutcho Formation is of Upper Triassic to uppermost Permian in age. Thorstad (1983) determined an Upper Triassic age on the basis of rubidium-strontium dating of volcanic rocks and regional stratigraphic constraints. Subsequent work by F. Childe

at the Mineral Deposit Research Unit of The University of British Columbia in 1996 suggests an age range from Lower Triassic to uppermost Permian.

Figure 7.3 Property Geology with Historical Claim Outline and Surface Projection of Sulphide Deposits

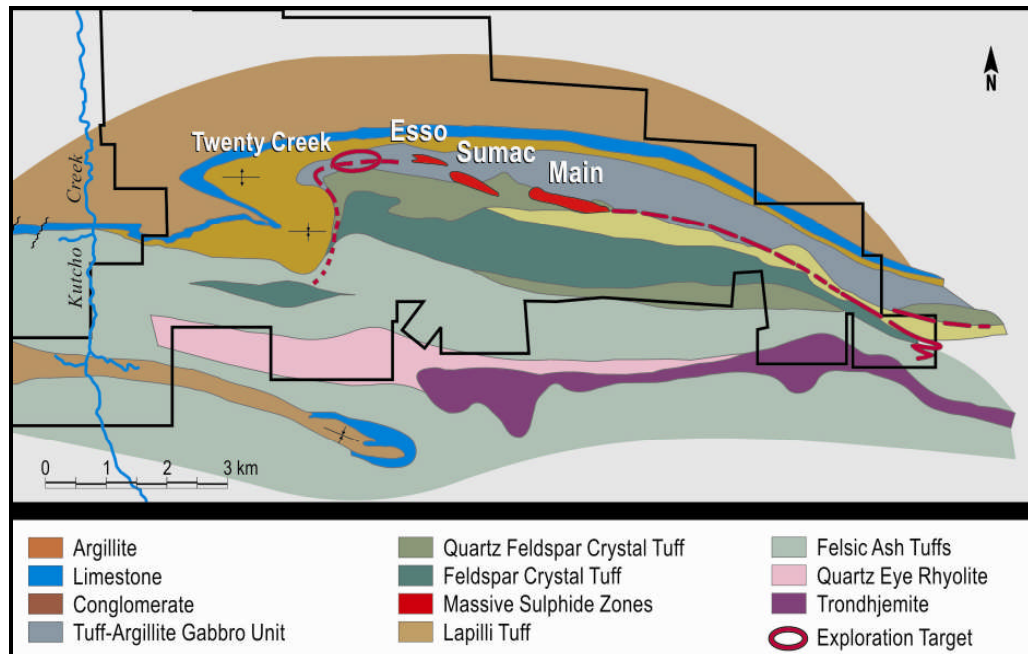
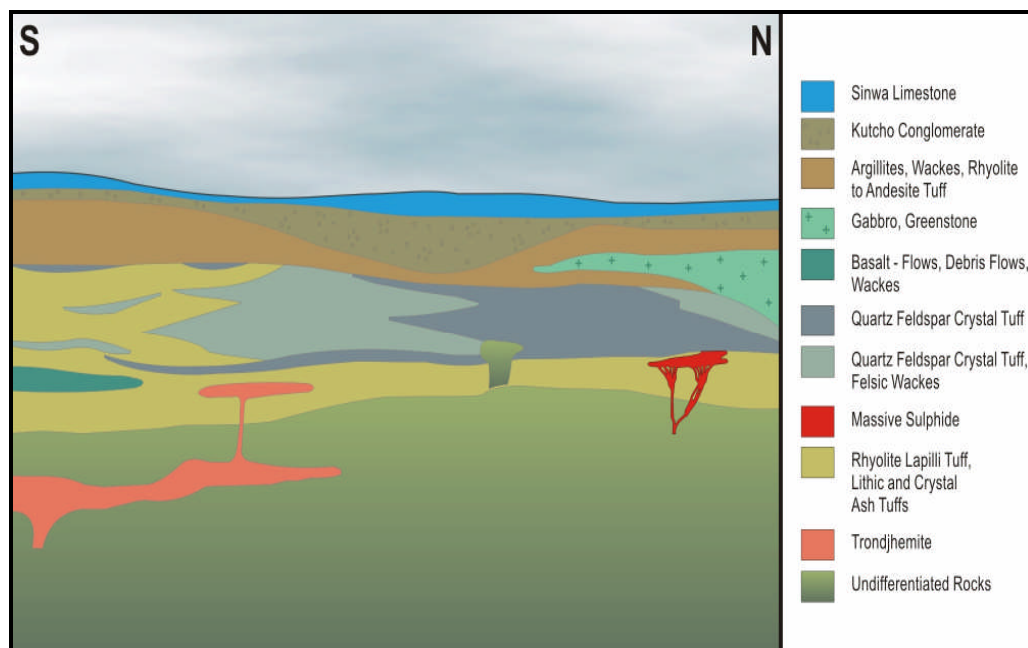


Figure 7.4 Schematic Illustration of Reconstructed Property Stratigraphy – Vertical Exaggeration Approximately 10x



STRUCTURE

Rocks of the Kutcho Formation are characterized by penetrative axial planar foliation that has a relatively constant strike direction of 270–290° with northerly dips from 45–65°. Minor but systematic changes in foliation from the east to west suggest low amplitude buckling of the fold axes. There appears to be a tendency for the dip of the foliation to decrease with structural depth, indicating that the axial planes are convex to the south.

Folds are open to tight, asymmetrical, inclined, and verging to the south. Fold plunges range from 0–30° in a westerly direction. Folds are most evident in well-bedded competent units, and therefore spatial distribution of the fold data is heavily biased to the western property area, where these units predominate.

Two aspects of the structure that critically affect stratigraphic interpretations are the number and size of foliation parallel thrust faults, and the degree to which the folds are propagated through the stratigraphic sequence. Neither of these aspects can be determined independently, and thus there remains considerable scope to reinterpret the stratigraphic position of various units locally. Foliation parallel thrust faults are difficult to detect from surface outcrop, but can be inferred from missing stratigraphy, contact geometry, shearing, and topographic evidence. Faults of this type are consistent with the deformation style and are considered to be prevalent over the property area.

Fold hinges outlined by the Sinwa limestone unit on Conglomerate Ridge (immediately east of Kutcho Creek) are difficult to trace in an easterly direction. Structural data (Holbek, 1985) indicate that the folds are cylindrical and therefore should be continuous within the depth of exposed stratigraphy. However, lithological competency contrasts are likely to result in disharmonic folding (Holbek and Heberlein, 1986), causing discontinuity of the axial plane towards the core of the fold. Stratigraphically thicker units will tend to produce a series of lower amplitude folds toward the core of the structure, which may explain why the fold axes so clearly outlined by the limestone unit on the western part of the property are not at all evident to the east, in the vicinity of the Sumac and Main deposits. Therefore, a certain degree of flexibility needs to be maintained regarding structural and stratigraphic interpretations in the vicinity of the sulphide deposits.

7.2 DEPOSIT TYPES

Mineralization of the Kutcho project is part of the volcanogenic massive sulphide (VMS) or volcanic hosted massive sulphide (VHMS) family of deposits. These deposits are major sources of copper, zinc, lead, silver, and gold around the world. Speculation about the origin of these deposits goes back to the mid-1850s, when various French and English scientists postulated chemical precipitation from seafloor volcanic activity (Stanton, 1991). In the early 19th century, Japanese workers documented astute observations of the sulphide textures preserved in the Kuroko

deposits of Japan and the association of these deposits with rhyolite domes and articulated the “submarine sinter theory”. However this work did not seem to attract much attention, and genetic theories or models of ore formation of this deposit type did not gain international acceptance until similar observations were published by others worldwide in the 1950s and ‘60s (Ohmoto and Skinner, 1983). Discovery of the Red Sea brine deposits in 1965 provided substantial impetus for the proponents of the “submarine exhalative” model. A certain amount of controversy between syngenetic and epigenetic theories continued through the 1970s, but with the advent of deep-sea submersibles and the filming of black and white hydrothermal vents (or “smokers”) in volcanic rift zones on the seafloor, scientific models went to a new level of realism and detail.

VMS deposits have been classified into various subtypes, depending upon the composition of the host rocks and the mineralization, and the tectonic setting of origin. The Kutcho deposits are VMS deposits of the Kuroko type or felsic volcani-siliciclastic depending upon the classification scheme. Mineralization is related to felsic volcanism in island arc or back-arc tectonic settings. Perhaps the most significant feature of VMS deposits from an exploration perspective is their tendency to occur in clusters. Larger VMS camps can have up to 25 discrete deposits, and mineralized districts are common.

Features of the Kutcho deposits suggest that they formed at (or very near to) the water-seafloor interface in a structurally controlled depression, likely a half-graben type structure. The Kutcho deposits have some uncommon features: the absence of lead and barite is likely due to the low potassium content of the volcanic host rocks (and presumably the associated rhyolite dome), and the abundant carbonate is probably of exhalative origin.

Alteration associated with VMS deposits is well documented, and provides a valuable exploration tool in that the area of alteration is much larger (by a factor of up to 10–100) than the actual sulphide deposit, thereby providing a much larger exploration target. Extensive studies of the alteration around the Kutcho deposit have been undertaken, and the chemical composition of the alteration is well zoned about the hydrothermal vent areas. This zonation allows the use of geochemical analysis of drill core within the alteration zone to provide vectors towards the hydrothermal vent area and, hopefully, the sulphide deposits.

Geophysical techniques such as electromagnetic (EM) and gravity surveys are useful for locating conductors or possible sulphide concentrations. EM methods can be used in airborne and ground surveys, but can also be used down drill holes to locate “off-hole” conductors, thereby effectively increasing the search area of a drill hole. A large number of airborne and ground-based geophysical surveys have been completed on the Kutcho property and all high-priority targets have been tested; however, there are many lower priority targets that still require additional follow-up.

7.3 MINERALIZATION

There are three known deposits that comprise the Kutcho project and form a westerly plunging linear trend (see Figure 7.5). From east to west, the deposits are termed the Main (previously known as Kutcho), Sumac, and Esso deposits. The Main deposit comes to surface at its eastern end, whereas the Esso deposit occurs at depths about 400 m below surface.

Figure 7.5 Computer Generated Image of the Three Known Massive Sulphide Deposits, Drill Holes and Surface Topography of the Kutcho Project

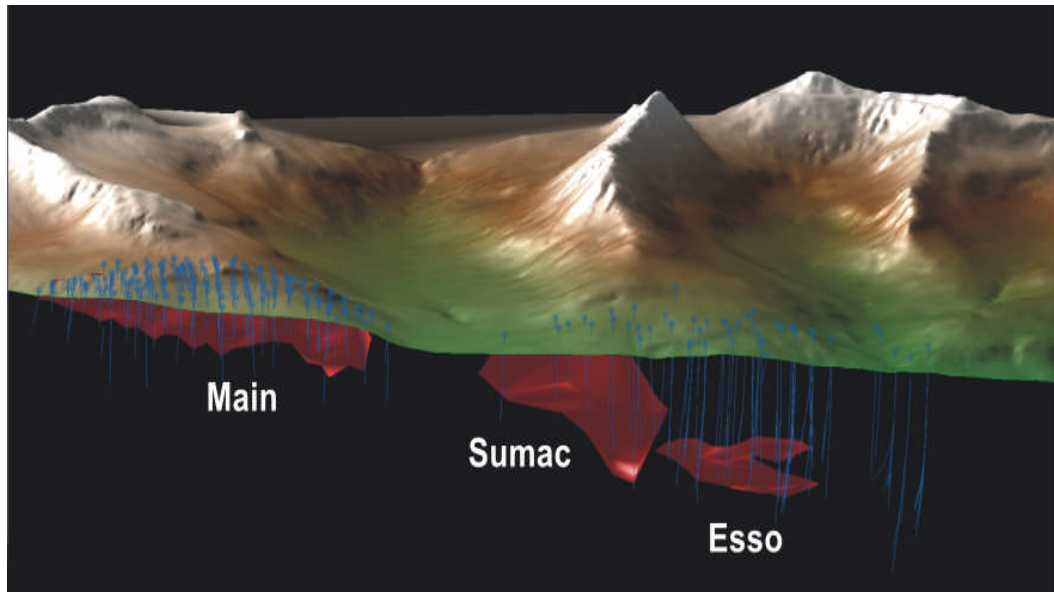


Image view is to the South

7.3.1 MAIN DEPOSIT

The Main deposit has an elliptical, lenticular shape with approximate dimensions of 1,500 m long, 260 m wide (down-dip), and 36 m maximum thickness. The long axis of the deposit plunges to the west at about 12°, just slightly less than the regional fold axes. The deposit is conformable with stratigraphy, dipping moderately to the north. There is a gentle warping of the deposit, such that the dip of the deposit changes from east to west and north to south. The shallowest dip (about 38°) occurs at the southeastern edge and becomes progressively steeper (to about 63°) at the northwestern edge. In general, the up-dip edge of the sulphide lens is narrow and pinches out, whereas the down-dip edge is thicker and interlayered with tuffaceous rock, giving the deposit an approximate flattened arrowhead shape in cross-section (Figure 7.6).

In detail, the Main deposit is composed of three depositional cycles. The cycles are interpreted to begin with pyritic mineralization, which then becomes more copper-rich and finally zinc-rich. The cycles are variably separated by siliceous or carbonate exhalative material and minor volcanic ash and detritus. Interpretation of the shape

of the sulphide zone, taken together with the observed volcanic and depositional textures of the enclosing rocks, suggests that the sulphide mineralization was deposited in a structural depression, likely a half-graben type structure. Fine mineralogical layering and sulphide-ash, sulphide-silica, or carbonate inter-layering, as well as framboidal and “snowball” textures in both the sulphide and carbonate minerals, suggests quasi-sedimentary deposition at the seawater seafloor interface. Polished section analysis indicates that very little sulphide recrystallization has taken place.

Sulphide mineralogy of the deposit is relatively simple, consisting of pyrite, chalcopyrite, sphalerite, and bornite, with minor sulphide minerals chalcocite, tetrahedrite, diginite (and related minerals), galena, idiaite, hessite, and electrum. Gangue minerals include quartz, dolomite, ankerite, sericite, gypsum, and anhydrite. Fluorite and barite have been observed, but do not occur in volumetrically significant amounts.

The internal stratigraphy of the Main deposit was determined by detailed drill core logging (Holbek and Heberlein, 1986) along a single longitudinal section of drill holes and is given in Figure 7.7. The deposit appears to have formed from three hydrothermal-depositional cycles that begin with barren pyrite which grades into a copper-rich middle and zinc-rich top. Depositional cycles are commonly separated by layers of exhalative quartz and/or carbonate and minor volcanic ash; however, continued hydrothermal activity results in sulphide replacement mineralization, which tends to blur grade boundaries in some areas. Additional features also cause complexity to the internal sulphide stratigraphy, such as an irregular depositional surface, localized slumping of sulphide mineralization or chimney collapse, and late stage (post depositional) hydrothermal activity. Areas of late overprinting by oxidized copper species and enrichment in precious metals are interpreted as indicators of vent areas, and occur along a linear trend on the down-dip side of the deposit, with two “hot-spots” near each end of the deposit. However, no well-defined areas of classical footwall stringer mineralization have been identified by drilling.

The upper contact of the sulphide mineralization is sharp, with almost no sulphide minerals occurring in the hanging wall rocks with the exception of scattered coarse grains of porphyroblastic pyrite. However, alteration of feldspar to sericite and carbonate in the hanging wall is intense, and occurs for up to 50 m above the sulphide contact. It is common for a small shear zone to occur at the sulphide-schist contact, which varies in thickness from 20–200 cm, and in many drill holes carries some grade. The base of the deposit consists of nearly barren massive pyrite with interstitial quartz. The contact between ore and the footwall pyrite zone can be either gradational or sharp. Below the footwall pyrite zone is quartz-sericite schist with bands of generally barren massive to semi-massive pyrite. The footwall pyrite content diminishes with depth away from the deposit, but extends to a maximum depth of 200 m below the central part of the deposit. Although the footwall material appears incompetent in the drill core (“poker chip” cleavage), it holds up very well in the underground adit.

KUTCHO PROJECT - MAIN DEPOSIT - SECTION 538000E

Legend:

- OVBD: OVERBURDEN
- GBBR: METAGABBRO
- QFXT/BX: QFXT BRECCIA
- QFXT: QUARTZ FELDSPAR CRYSTAL TUFF
- LXTF: CRYSTAL LAPILLI TUFF
- SEXL: SILICA CARBONATE EXHALITE
- MSSX: MASSIVE SULPHIDES
- XLAT: CRYSTAL LAPILLI ASH TUFF

Scale: 0 50 100 m

Orientation: N (North) S (South)

Section Line: 30m E of section

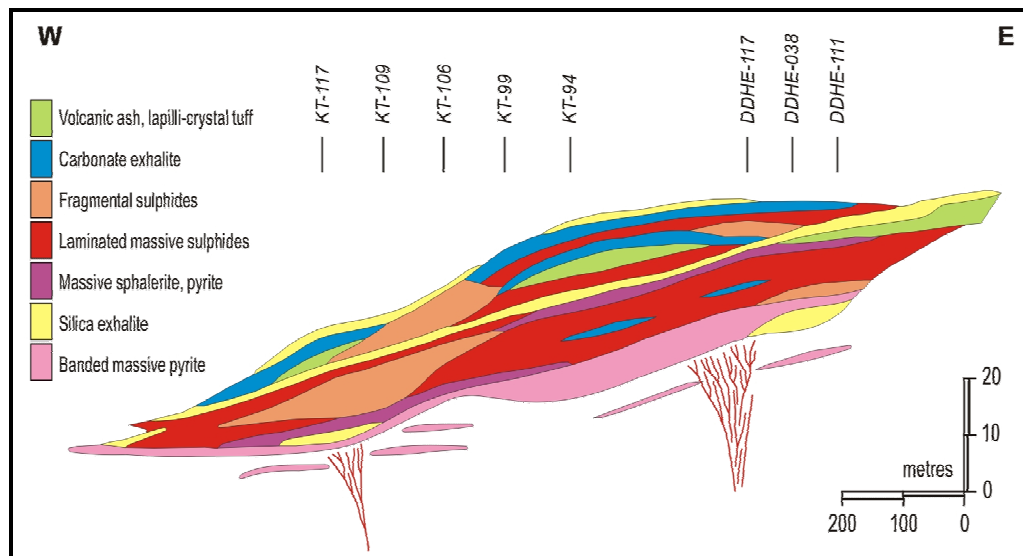
Key Features: E033, E038, E035, E083, E054, E007, WK0414, WK0415, XLAT, QFXT, GBBR, LXTF, SEXL, MSSX, XLAT.

Data Points (Approximate values):

- 1.15, 3.24, 9.0, 0.14
- 1.08, 0.18, 26.0, 0.22
- 0.76, 1.28, 27.0
- 3.09, 2.96, 51.0, 0.70
- 2.01, 2.03, 35.0
- 1.59, 2.78, 30.0, 0.49
- 5.95, 2.21, 135
- 1.87, 0.84, 30.0
- 3.43, 1.60, 48.0, 0.73
- 0.86, 0.22, 13.0
- 3.03, 5.65, 00.0

Western Keltic Mines Inc.

Figure 7.7 Internal Stratigraphy of the Main Deposit — Vertical Exaggeration Approximately 10x



7.3.2 SUMAC DEPOSIT

The Sumac deposit has not received much attention historically, due to its relatively low grades. The shape of the deposit is primarily taken from conductance contours generated by a chargeability geophysical survey carried out during the mid-1980s. A chargeability survey is carried out by putting an electrical current into a sulphide zone and measuring the change in the magnetic field due to electrical flow through the conductive (sulphide-rich) rocks. A total of 14 drill holes at 100–200 m spacing define the Sumac deposit. Better intercepts include 1.45% Cu, 2.56% Zn, and 23.7 g/t Ag over 26.1 m, and 1.37% Cu, 1.9% Zn, and 26.2 g/t Ag over 23.4 m.

The Sumac deposit is nearly (but not quite) continuous with the Esso deposit (across the historical property boundary), but sits within a local depression relative to the Main and Esso deposits. The historical estimate for the Sumac Zone, based on a polygonal method, is quoted by Sumac Minerals Canada (SMC) and Esso Minerals Canada (EMC) (1983) as 5.3 million tonnes grading 1.09% copper, 1.62% zinc, and 14.4 g/t silver.

An additional four drill holes were completed in the Sumac deposit in 2005 by Western Keltic Mines Inc. (WKM). These holes, drilled in the western part of the deposit, provided two of the best intersections within the deposit and helped establish the western end of the deposit. Reinterpretation of the Sumac drill data suggests that the core of the deposit has a much steeper plunge than previously suspected, indicating that historical drilling in the deposit's eastern end was likely too deep and opening up more area for test drilling (see Figure 7.8). The Sumac deposit is finely banded but massive and competent, and has the highest sulphide content (+90%) of the three deposits. Alteration of the host stratigraphy around it is very similar to that of the other two deposits.

7.3.3 *ESSO DEPOSIT*

The Esso deposit was discovered as a natural consequence of following the trend in mineralization through the Main and Sumac deposit areas. The Esso deposit occurs at depths of between 400–520 m below surface. The Esso deposit, like the others, is an elongate lens shape with current dimensions of approximately 580 m in length, up to 100 m in width, and up to 21 m in thickness. In general, there is zonation both in thickness and in grades from the central deposit area.

An historical resource estimate by EMC (Didur, 1980) using the sectional method had the following results: 1.63 million tonnes grading 3.42% Cu, 6.5% Zn, 62.7 g/t Ag, and 0.53 g/t Au in the main part, with 0.46 million tonnes grading 2.1% Cu, 3.13% Zn, 46.5 g/t Ag, and 0.43 g/t Au in the south wing zone. A subsequent historical resource estimate by EMC (for which documentation is unavailable) indicated a resource of 1.5 million tonnes grading 3.37% Cu, 5.71% Zn, 63.4 g/t Ag, and 0.54 g/t Au.

Historical drill holes are spaced at approximately 10–30 m along sections that are variably spaced, usually at 60 m but some at 120 m (Figure 7.9). WKM completed 16 drill holes and wedge branches in the Esso deposit area in 2005, of which half were within the deposit and half were peripheral to the deposit. Current and historical drilling is sufficient to define the overall size and geometry of the deposit. Minor changes along the deposit edges are possible with additional drilling, but these would not be significant with respect to tonnage, grade, or mine planning.

7.3.4 *OTHER MINERALIZATION*

Other zones of mineralization include the Footwall Zone (FWZ), and the Jenn Area. The FWZ occurs approximately 100 m stratigraphically below the Main deposit and slightly up-dip and to the east of the centre of the Main deposit. The FWZ is relatively narrow, at 2–5 m thick, and relatively zinc-rich. An historical resource estimate by Didur (1979) using a polygonal method is 230,000 tonnes grading 1.47% Cu, 5.52% Zn, 43.7 g/t Ag, and 0.4 g/t Au. The mineralization was only systematically drilled up to the historical Esso-Sumac property boundary, but a number of drill holes by SMC (and more recently WKM) demonstrate that the FWZ does not extend for significant distances to the west or down-dip of its current position.

The Jenn Area is on the eastern end of the property and received a fair amount of attention by EMC. Although significant alteration and some local mineralization were intersected in a number of drill holes, no resources have been defined in the Jenn Area. There are some structural and stratigraphic complexities in the region that make interpretation of drill data less than straightforward; however, airborne geophysical surveys did not detect significant untested conductors in the area, suggesting that any potential area deposits would be at considerable depths.

7.4 EXPLORATION

Historical exploration programs previously carried out on the Kutcho Property are described in Section 6.0: History. Exploration work, consisting primarily of resource drilling, has been carried out by WKM in 2004, 2005, and 2006, and is currently ongoing. Most of this work is described in other sections of the report, primarily the following Section 7.5: Drilling.

7.5 DRILLING

Approximately 88,000 m of diamond drilling in 365 drill holes and 43 wedge branches has been completed on the property. Most of the drilling was completed between 1974 and 1983 (by EMC and SMC) with an additional 7,031 m in 28 exploration holes completed by ARM and Homestake in 1990, and nearly 16,000 m of drilling in 87 holes completed by WKM between 2004 and 2006.

The Main deposit is defined by 221 drill holes (Table 7.1). The Esso deposit is defined by 33 pilot holes and an additional 43 wedge branches for a total of 76 intercepts, of which 48 are within the deposit shell or wire frame. The general shape of the Sumac deposit has been determined from both drill data and geophysical data, as there are only 14 drill holes in and around the deposit.

Table 7.1 Summary of Drill Holes

Company	Main Deposit	Sumac Deposit	Esso Deposit	Exploration/ Other	Total
SMC	102	10		17	129
EMC	49		24 (+39*)	48	121 (+39*)
ARM	2			26	28
WKM	68	4	11 (+4*)	4	87 (+4*)
Total	221	14	35 (+43*)	95	365 (+43*)

* wedge branches off pilot holes

Locations for drill holes that define the Main deposit are displayed on a plan map which also shows an approximate vertical projection of the deposit (Figure 7.8).

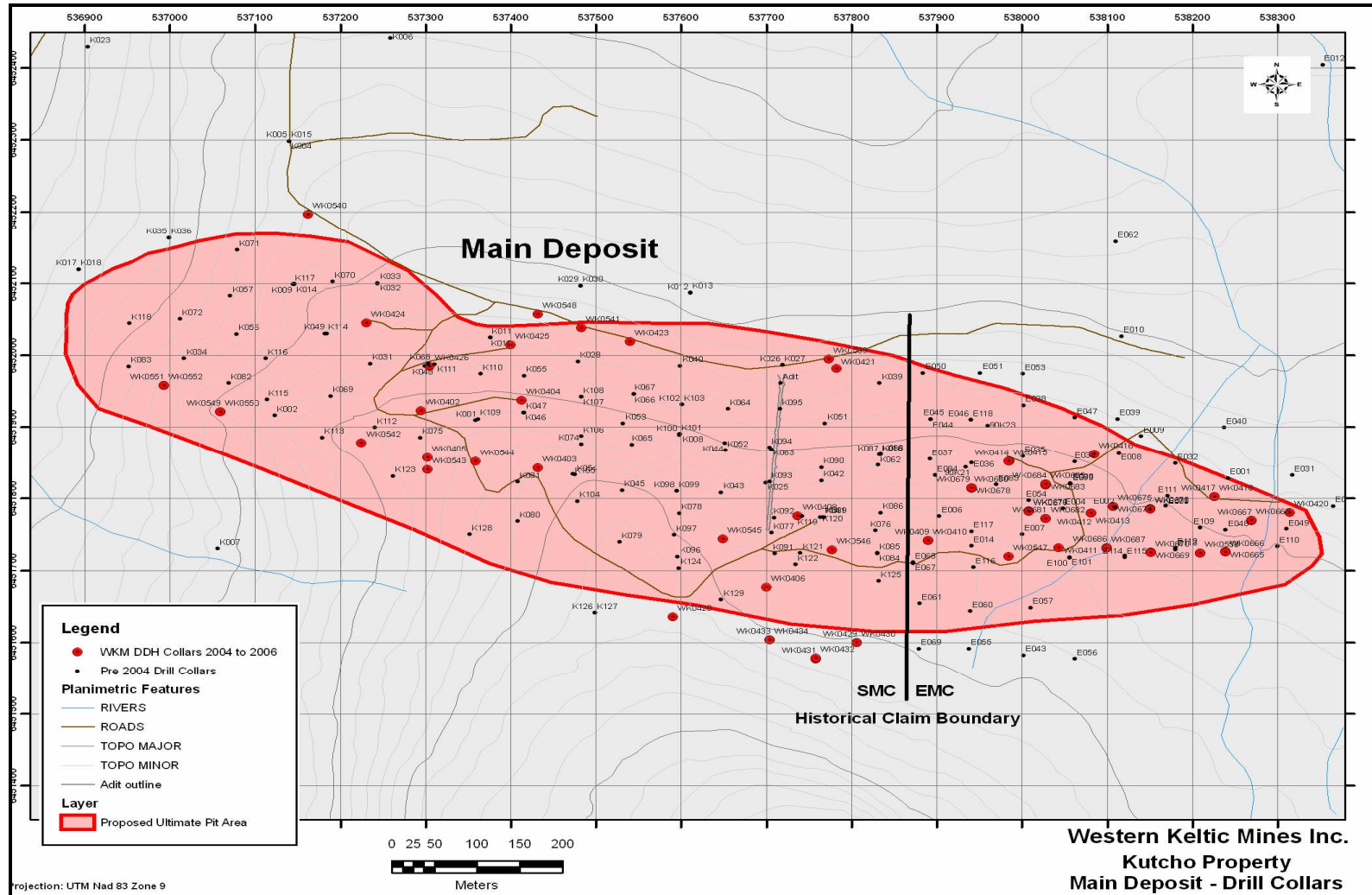
Drill collars and claim locations were surveyed periodically during exploration programs by McElhanney Engineering Services Limited (MESL) until 1983; all later WKM drill holes and many of the historical drill holes were surveyed or resurveyed by MESL in September 2006. Previous surveys were surveyed in mine-grid coordinates, whereas the recent survey was carried out in UTM coordinates, and survey points that were not resurveyed were converted to UTM grid points by MESL.

Most of the drill sites have been reclaimed, although many collars are still locatable and most of the drill core is stored on the property. A new core storage area was constructed for the Esso deposit drill core in 1985, and all of it was moved there over the following six years. The Sumac deposit core was stored on core racks in the

area between the Main and Sumac deposits. Due to decomposition of these racks, the core was recently removed and cross-stacked nearby, and the core racks were dismantled.

Approximately half of both Sumac and Esso deposits drill core was re-logged in 1984 and 1985 (Holbek and Heberlein, 1986) using the GEOLOG system. WKM has recovered all available data (in both digital and paper formats) and merged it with current data into two master databases; one for resource work and a total exploration database.

Figure 7.8 Longitudinal Section Projected to Surface Showing Drill Hole Pierce Points in the Main Deposit



The map displays the Kutcho Property with two main deposits highlighted in pink: the Esso Deposit to the west and the Sumac Deposit to the east. Numerous drill collar locations are marked with red dots (VKM DDH Collars 2004 to 2006) and black dots (Pre-2004 DDH Collars). The map includes a grid with UTM coordinates (Easting: 535100 to 536500; Northing: 6451900 to 6452900). A legend in the bottom left corner defines the symbols for drill collars, planimetric features (rivers, roads, topographic lines), and deposits. A scale bar (0 to 200 meters) and a north arrow are located in the bottom right. The map is titled 'Kutcho Property Esso & Sumac Deposits Drill Collars' and includes the text 'WESTERN KELTIC MINES INC.' and 'Projection: UTM Nad 83 Zone 9'.

Initial drilling in the Main deposit was carried out on 120 m spaced sections with drill holes spaced approximately 60 m along section lines. This spacing was subsequently reduced to approximately 30 m spaced drill intersections along 60 m spaced sections, and recently WKM has been reducing the drill spacing in selected areas to 30 m or less. Historical drill hole diameters are mostly BQ (38 mm) and recoveries were generally very good with only rare core loss in minor fault zones. The more recent drilling has been a combination of HQ and NQ in the Main deposit and NQ (or BQTW in wedge branches) within the Sumac and Esso deposits. Most holes were drilled at -45° to -60° in order to intersect mineralization at close to 90°. Due to strong foliation dipping to the north, even vertical holes tend to flatten and cut the mineralization roughly perpendicular to its dip.

The historical drill hole database for the Main deposit contained assays for copper, zinc, silver, and specific gravity (SG), with most holes containing assays for gold and approximately 60% containing assays for sulphur. Historical drill data for the Sumac and Esso deposits contain results for copper, zinc, silver, gold, and specific gravity. There are 4,569 assay intervals within the total resource database, of which 1,589 are new (WKM). Of the remaining 2,978 historical assay intervals, 2,061 are in the Main deposit, 443 from the Esso side of the deposit, and 1,618 from the Sumac side. The assay intervals were generally longer on the Esso side, while the Sumac data commonly contained shorter intervals based on sulphide mineralogy. Comparative statistics for the two historical data subsets are given in Table 7.2.

Table 7.2 Basic Statistics for the Main Deposit – Historical Drill Hole Assays

	Company	SG	% Cu	% Zn	Ag g/t	Au g/t
Minimum	EMC	2.670	0.00	0.00	0.00	0.00
Maximum	"	4.690	18.1	34.00	376.46	14.743
Mean	"	3.469	1.574	2.090	27.793	0.396
Median	"	3.180	0.870	0.530	14.400	0.137
Stan. Dev.	"	0.566	2.161	3.500	42.104	1.210
Minimum	SMC	2.560	0.01	0.00	0.00	0.00
Maximum	"	4.890	35.50	41.90	942.860	5.691
Mean	"	3.605	1.164	1.609	20.061	0.212
Median	"	3.420	0.610	0.360	8.230	0.103
Stan. Dev.	"	0.672	1.791	3.350	38.539	0.449

The SMC data set has a greater range than the EMC data set, partly due to samples having come from a larger area, but also because the individual sample lengths are smaller (a minimum 0.2 m and maximum 1.5 m, compared to Esso's minimum 0.4 m and maximum of 5.4 m). The difference in grades reflects the higher grade of the up-dip eastern part of the Main deposit, but is also a function of SMC taking more shoulder samples of lower grade.

7.6 SAMPLING METHOD AND APPROACH

The first drill holes into the Main deposit were carried out nearly simultaneously by EMC and SML in the summer of 1974, although within different areas of the deposit. The first two seasons of drilling were experimental, with relatively wide-spaced drill holes used to determine approximate extents of the mineralization. In 1976, SML carried out a chargeability geophysical survey, which provided clear indications of the size of the conductive mineralization, as well as indications of the Sumac deposit. Shortly thereafter both companies adopted a grid approach to the location of drill holes. A common grid for both companies was implemented later, with north-south grid lines spaced at 60 m. The 60 m spacing (as opposed to the more common 50 m) was chosen as it allowed for four subdivisions (if required) and better approximated the 100 and 200 foot line spacing used by EMC for initial soil geochemical surveys. Consequently, almost 90% of the drill holes within the Main deposit are located along the grid lines. Similar approaches were applied to the Sumac and Esso deposits.

In 2004, initial large diameter (HQ) core drilling in the Main deposit was carried out by WKM to verify historical data and to obtain metallurgical samples. The approach to drill hole locations for this program was to obtain a distribution of drill holes covering the entire deposit area, with specific drill hole locations placed where they would result in infilling areas of lower drill hole density. Because the HQ drill was a skid-mounted rig, drill hole locations were restricted to areas accessible to such a rig with minimal road building. The 2006 drilling program on the Main Zone was designed to infill the starter pit area on approximate 30 m centres.

Sampling methods for drill core were similar for all of the exploration phases on the property. Core size varied (as discussed in earlier sections) and sampling of the core using a mechanical splitter was initially used by both SML and EMC, with SML switching to a diamond saw after the first nine drill holes, and EMC switching after approximately 30 drill holes. Splitting by diamond saw has been used ever since. Sample selection within mineralized drill core is more significant, and is discussed in detail in the following section.

7.7 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Data verification and quality control and quality assurance for the Kutcho Project was studied extensively by AMEC (Chong, 2007) who worked closely with company personnel in order to determine that all historical data was suitable for use in resource or reserve estimates and that modern data met high standards for quality assurance and quality control.

7.7.1 HISTORICAL DATA

Historical drill hole data (originally from Esso) were obtained from Barrick in the form of digital ASCII files derived from the original tapes used by Wright Engineers during

their work on the prefeasibility study. The drill hole data was edited in ASCII format in order to load in Surpac Vision, a program from Surpac Software International. Data was plotted on sections and compared to sections within EMC and SMC reports and some of the original drill logs that contained assay information. Not all holes were in the database, and those that weren't were entered manually from the drill logs. In a few cases survey information was incorrect or missing, and had to be corrected or obtained from the original drill logs. Original assay lab reports were available for most of the EMC data, but were not available for the SML data, and the typed drill logs were relied upon for verification.

Neither EMC nor SMC documented their sampling and analytical protocol, but both companies typically carried out their work to the highest industry standards of the time. Some of the EMC drill logs have duplicate sample data written adjacent to the original samples, suggesting that duplicate samples were a standard procedure. The duplicate samples displayed normal expected variations between duplicates, and it is probable that the added values were re-analyses of sample pulps by a different lab. There is no documentation of any analytical problems.

Both EMC and SML used the same drilling company (Arctic Diamond Drilling) and the same drills (Longyear 38 Flyable). Core size was BQ and recoveries were normally excellent. Due to the deposits striking approximately east-west and dipping approximately 45° to the north, most drill holes were drilled with azimuths of 180° and inclinations from -45° to -90°. The steeper drill holes tended to curve into the foliation, consequently holes that began as vertical would flatten with depth, and the degree of drill hole curvature could on occasion be considerable. The drill hole flattening was common to EMC, SML, and ARM drilling, but not as significant for WKM due to changes in drill and bit technology.

Both EMC and SML logged core at camps on site, marked samples, and split core for analytical samples. EMC used a manual splitter for the first three years and then switched to a diamond saw (preferable for massive sulphide mineralization), while SML used a manual splitter for the first year (nine holes) and a diamond saw thereafter. EMC tended to take long samples of about 2–3 m lengths (to a maximum of 5.4 m) in the early years of exploration, but took shorter, more geologically constrained samples in the later years. Additionally, in much of the early sampling by EMC, the geologists did not take shoulder samples, so the samples tend to start and end in significant grade. Based on current experience with the deposit, it is unlikely that EMC missed any significant grade; however, grade of potential dilution will be more difficult to estimate. SML sampling was good, with samples selected on the basis of geology and sulphide mineralogy, average sample lengths of about 1 m, and collection of shoulder samples. EMC used Min-En Labs of North Vancouver (bought out by Bondar-Clegg in the 1990s) for almost all of their analytical work, while SML used ALS Chemex Laboratories (Chemex), also of North Vancouver. Check samples for EMC were carried out by Terramin Labs in Calgary and SGS Lakefield Research Ltd. in Lakefield, Ontario.

Almost all of the sulphide intersections from EMC core were used for metallurgical research and are not available for inspection or resampling. In contrast, SML quartered their core for metallurgical samples, and most of the remaining core was available on the property.

7.7.2 CURRENT DATA

WKM carried out diamond drilling programs on the Kutcho Project in 2004, 2005, and 2006. The 2004 program in the Main deposit consisted of 21 HQ drill holes which were drilled to verify historical drilling and to collect material for metallurgical testing: ten NQ drill holes and four BQTW wedge branches in or near the Esso deposit, and seven HQ drill holes in the Footwall Zone. The 2005 drill program consisted of 16 NQ drill holes in the Main deposit; four NQ drill holes in the Sumac deposit; four HQ drill holes and two BQTW wedge branches in the Esso deposit and two other drill holes. The 2006 drill program consisted of 23 BQTW drill holes within the eastern part of the Main deposit. Sampling methodology and analytical work is similar for all three programs.

7.7.3 CORE SAMPLING

Drill core was transported from the drill by truck or helicopter to the core logging facilities, situated adjacent to the Kutcho Airstrip approximately 8 km west of the drill area. Core was geologically logged and marked for splitting, with approximately 70% of the deposit intersections also selected for metallurgical study, with the core sawn in half and the half-core metallurgical samples collected by a geologist and sealed in a labelled, nitrogen-filled bag. The half-cores were then sawn into quarter core samples which the geologist collected, measured their specific gravities, placed the samples in plastic bags, and heat-sealed the bags closed. The process was the same for core not being used for metallurgical samples, except that half-core samples were collected and placed into plastic bags, which were then sealed, but not packed in nitrogen. Sealed sample bags were then placed into labelled rice bags for shipment to the laboratory. Sample shipment bags were transported to Dease Lake or Smithers by aircraft and picked up at the aircraft hangar by the trucking company for transport to the laboratory.

7.7.4 ASSAYING

All routine analysis of drill core was carried out by Chemex of North Vancouver. Core samples (including blanks) were ground to 80% passing 100 mesh, and were analyzed (including standards) using induced coupled plasma (ICP) methods for 33 elements following an aqua regia digestion. Copper, zinc, and silver values above the ICP detection limits (10,000 ppm for copper and zinc; 100 ppm for silver) were assayed by atomic absorption methods following an aqua regia digestion. All samples were analyzed for gold by fire assay in 30 g sub-samples, and sulphur was analyzed by Leco furnace.

Most core samples were measured for SG unless the rock was so highly altered as to render a measurement meaningless. SGs were measured in the field by weighing the sample using an Ohaus scale in air and in water, with the SG calculated using the formula:

$$\frac{\text{Weight in air}}{(\text{weight in air} - \text{weight in water})}$$

Sample weights were accurate to the nearest tenth of a gram.

Blanks and standards were inserted into the sample stream at a rate of one per 10–20 samples and numbered consecutively with the core samples. Field blanks consisted of non-mineralized limestone and rhyolite crush. The WKM internal field standards were prepared in 2004 by collecting 30 kg of mineralized material from the dumps adjacent to the adit. The material was visually hand-sorted into low, medium, and high-grade material, and shipped to Pioneer Laboratories in Richmond, BC for grinding and homogenization, with aliquots placed in laboratory supplied pulp bags. The WKM standards were assayed by three independent laboratories to ensure that the samples were fully homogenous and to determine the grades for the standards. For the 2006 drill program assays, commercial standards at three different copper and zinc grades were obtained from Canadian Resource Laboratories, and used in conjunction with the WKM internal standards.

7.8 DATA VERIFICATION, QUALITY CONTROL, AND QUALITY ASSURANCE

7.8.1 HISTORICAL DATA

Data verification of historical data has been carried out in a number of different ways, including verification of the digital data by double data re-entry of historical drill logs and original assay sheets (for EMC data), statistical analysis and comparisons of various data sets, resampling of drill core (SML), and twinning holes (EMC).

In the prefeasibility study Wright Engineers noted that there was no discrepancy in grades or metal ratios across the property boundary. They also noted that the two drill holes adjacent to the bulk sample were comparable in grade to the 100 tonne bulk sample. Table 7.3 shows the grades obtained from the 4 by 3 m adit compared with the surrounding drill holes. The drill hole grades are based on a cutoff that was not applied to the adit sampling, and there is a trend to higher copper in the up-dip and higher zinc in the down-dip direction. A length-weighted average grade from the two holes adjacent to the adit (KT092 and KT025) is very similar to the length-weighted adit grades; however, in the adit sample the slightly lower grades combined with the greater thickness suggests that there is probably dilution from the margins of the mineralization.

Table 7.3 Thickness and Grades of Adit and Surrounding Drill Holes

Sample		True Width*	% Cu	% Zn	Ag g/t
KT077 (above)	DH	5.0	2.49	2.14	28.0
KT092 (above)	DH	9.5	2.05	2.28	26.2
Pilot Mill Feed	Adit		1.91	2.06	31.8
Round by Round	Adit	13.7 m**	1.83	1.97	30.4
KT025 (below)	DH	11.6	1.81	3.19	25.9
KT093 (below)	DH	10.0	1.56	3.36	22.1
Drill Hole (DH) Average		9.0	1.98	2.74	27.8

* based on a 1% copper equivalent cutoff

** No cutoff applied

Analytical results of two holes drilled in the Kutcho lens during the ARM/Homestake program in 1990 did not show a discrepancy in metal grades with nearby holes that had been drilled up to 10 years earlier. Likewise, the WKM drill holes within the Main deposit generally gave very similar results to the average of the three nearest surrounding drill holes, with the exception that precious metal grades showed an approximate average 10% increase, likely due to improved analytical techniques.

7.8.2 VALIDATION OF SML HISTORICAL ANALYSES

In order to verify historical drill results for the SML drill holes (where original certificates are not available), WKM resampled the mineralized core drilled by Sumitomo between 1974 and 1982. There are 129 SML drill holes in the project area, of which approximately 105 are within the Main deposit. SML core had been previously resampled in 2004, and the new results were very similar to historical ones; however, no CRM or standards were included during that investigation, so a second resampling program was undertaken (as per AMEC Americas Ltd. recommendations). In order to provide corroboration for the historical data a total of 137 samples were collected from 18 drill holes, spanning the range from K001 to K094. SML core was sawn core and all of the remaining half-core or quartered core was collected for re-analysis. The half-core was not quartered due to relatively poor core and core-box conditions after 30 years in the elements.

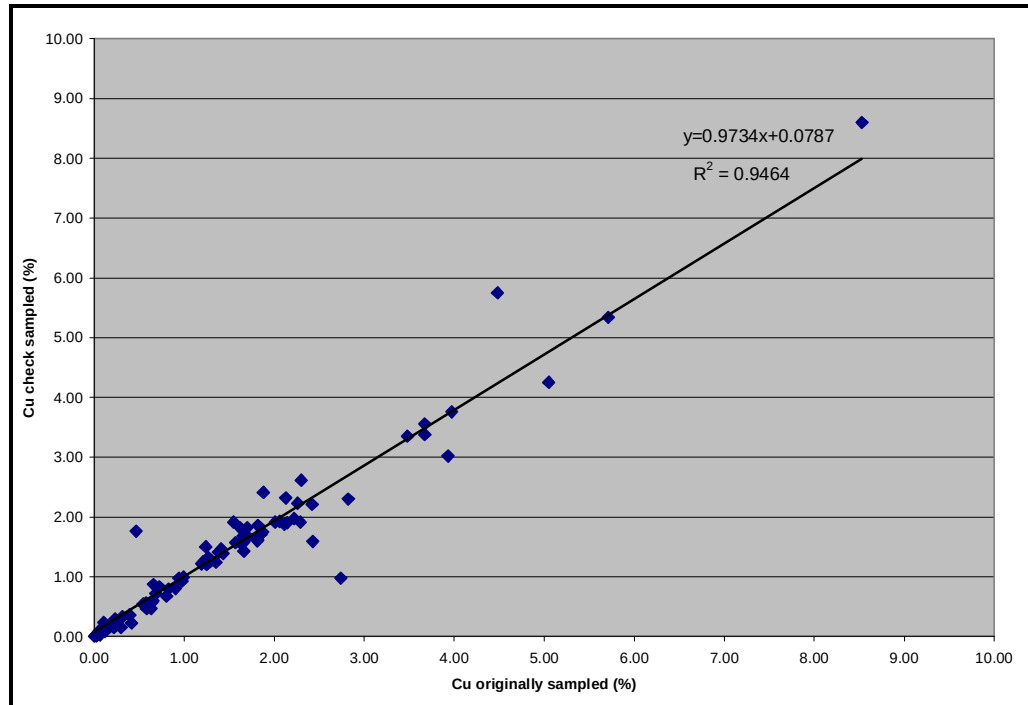
A comparison between the historical data and the new assays is presented in Figure 7.10 through Figure 7.14 for copper, zinc, silver, gold, and sulfur, respectively. See Appendix A (Table 6-1) for the supporting data table.

Slopes and intercepts have been calculated using the Reduction to Major Axis method, which does not introduce underlying assumptions about which data set is correct, nor does it assume any dependence or independence between the sample pairs (as a linear regression does). This method also allows for differences in population variance between the two data sets.

The formula used for these calculations is $y = mx + b$ (or $b = y - mx$), where b represents the intercept. Both x and y are known numbers (the mean values of the data sets), and m is derived from the standard deviation of the y data set divided by the standard deviation of the x data set.

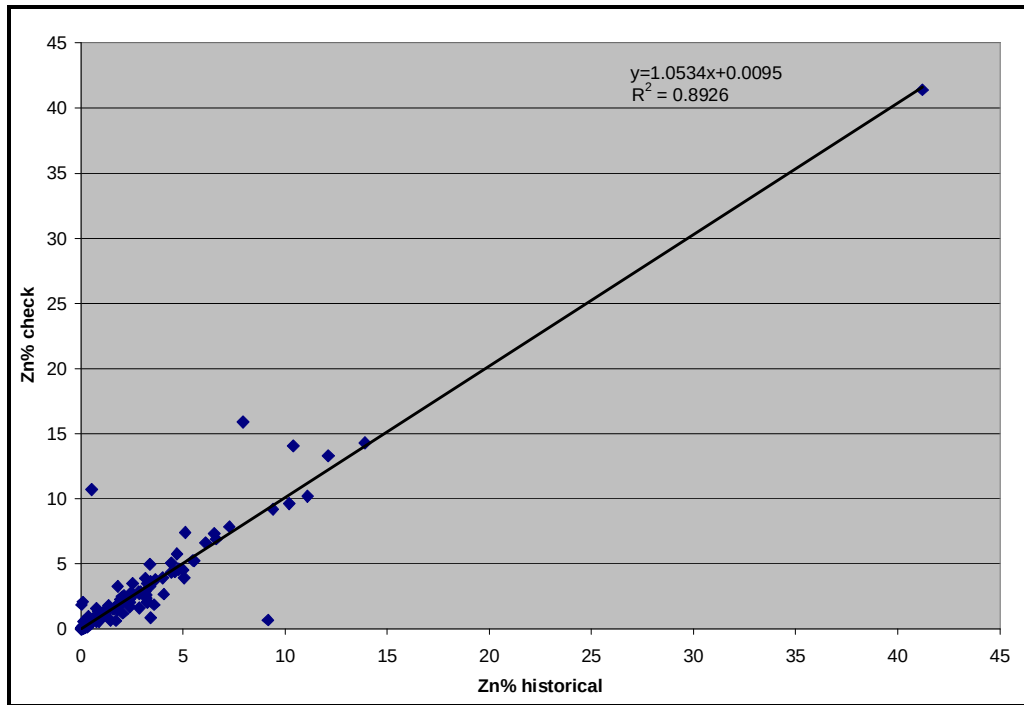
For copper, $x = 1.55$, $y = 1.58$, and $m = 0.9350$ ($STDEVy = 1.607331$, $STDEVx = 1.71899$), therefore $b = y - mx = 0.1285$.

Figure 7.10 Cu Comparison



The above figure and those following show that the two data sets correlate well for copper and zinc. The reproducibility of historical assays is limited by several factors. Split core shifting in the boxes through handling, and footage markers not always being legible, led to difficulties in collecting identical sample intervals. As well, improved lab techniques may yield different results, as lower detection limits are now available. However, given these challenges, the correspondence between data sets is remarkably good. The few outliers in Figure 7.10, Figure 7.11, and Figure 7.12 can be explained by either normal analytical variability or the previously noted challenges in obtaining true sample duplicates.

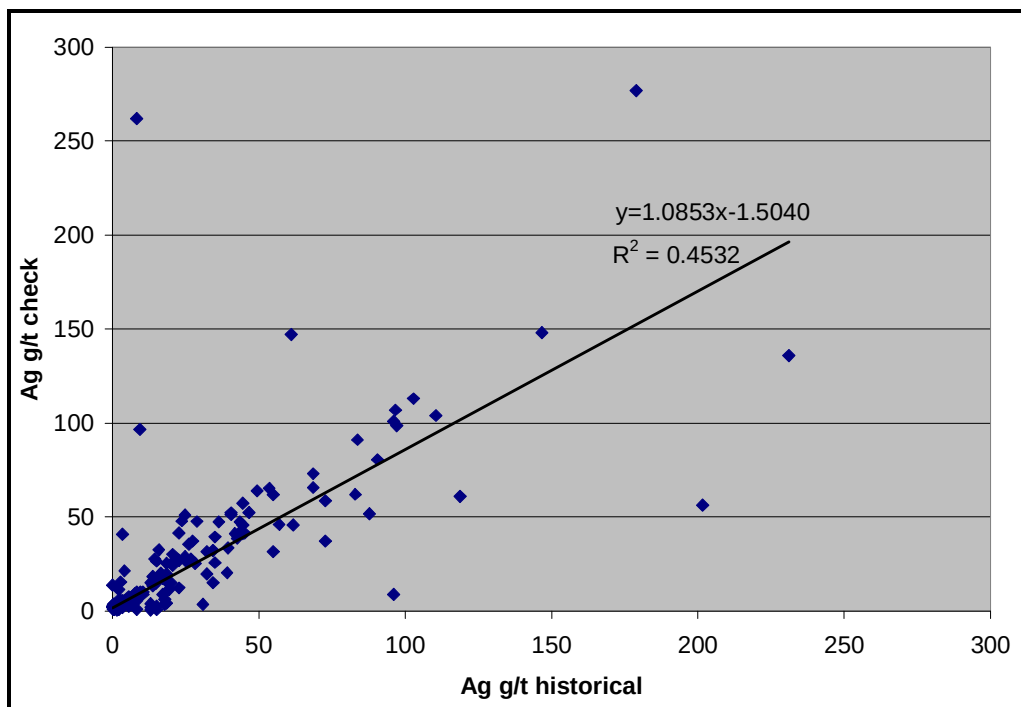
Figure 7.11 Zn Comparison



Using the formula from page 7-22 and the values for zinc, $x = 2.48$, $y = 2.65$, and $m = 1.0534$ ($STDEVy = 4.544127$, $STDEVx = 4.31382$), therefore $b = y - mx = 0.0095$.

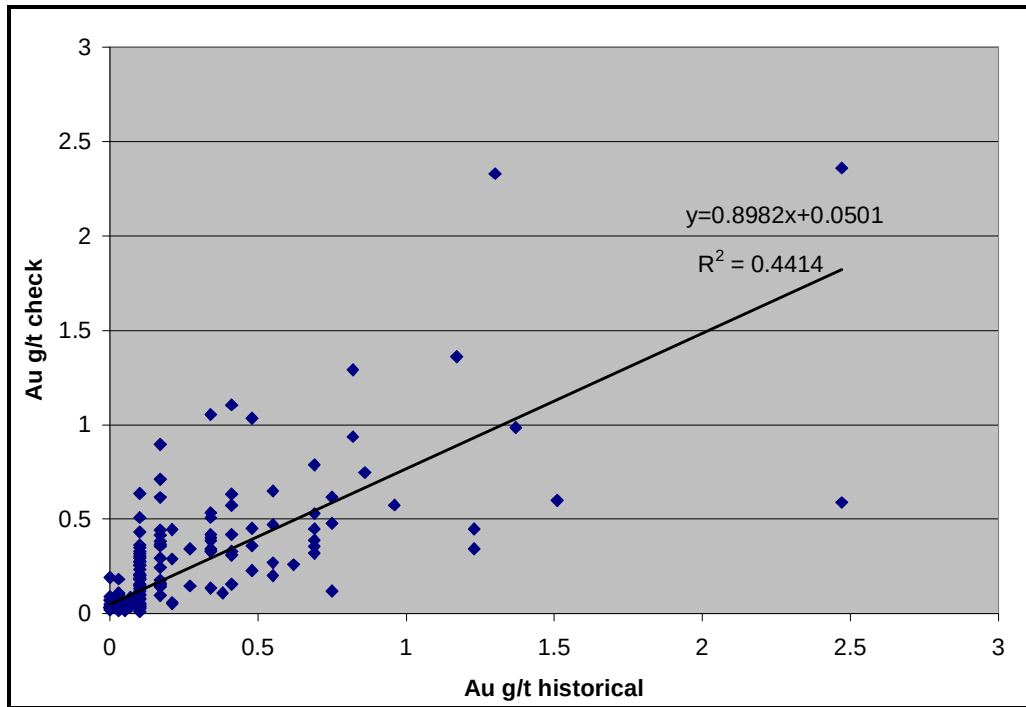
With only a few exceptions, the agreement for zinc results between historical and new data sets is very good. The most dramatic variances are from drill hole KT001, where both the near 10% and sub 1% zinc points could, in fact, be the same sample interval, but were possibly incorrectly designated during resampling. The other two outliers cannot be easily explained, but may be due to incomplete sample intervals available for resampling leading to a higher-grade bias induced by a few high-grade zinc core pieces within the remaining sampled core.

The chart for silver (Figure 7.12) contains more outliers and a wider spread in the correlation cluster. At least two outliers are related to the KT001 sampling error noted for zinc. Others could be explained by a silver “nugget” effect, where silver-rich core pieces may have been included in the historic and resampled cores. The overall correlation between historic and resampled core is reasonable, with a slight increase in overall grade likely due to improvements in contemporary analytical methods and equipment.

Figure 7.12 Ag Comparison

Using the formula from page 7-22 and the values for silver, $x = 31.38$, $y = 32.55$, and $m = 1.0853$ ($STDEVy = 42.71559$, $STDEVx = 39.36$), therefore $b = y - mx = -1.5040$.

Figure 7.13 Au Comparison

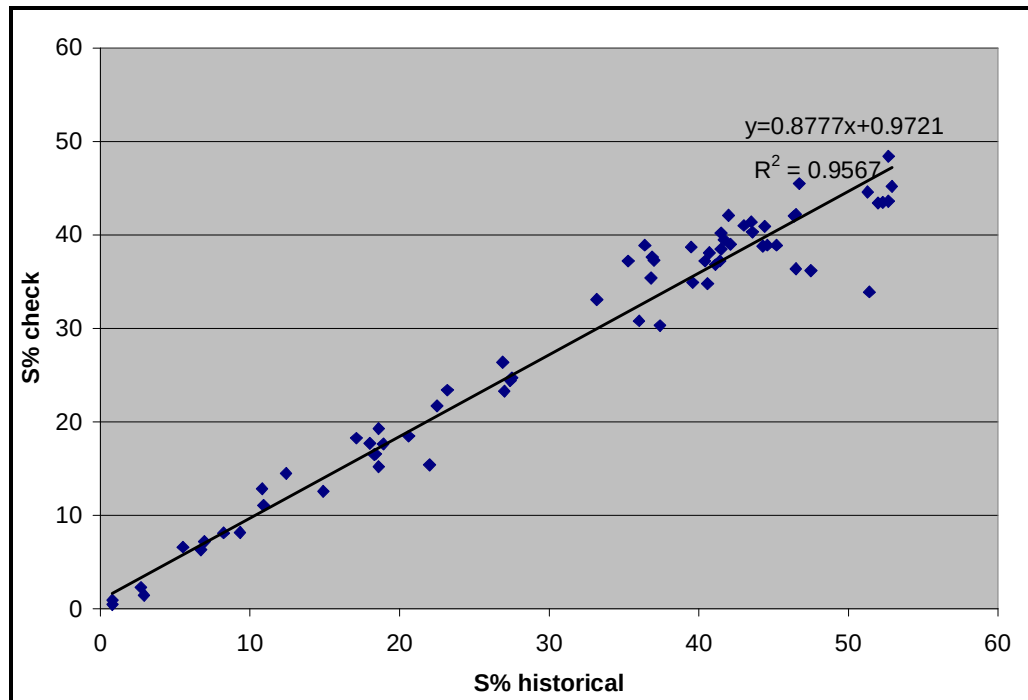


Using the formula from page 7-22 and the values for gold, $x = 0.31$, $y = 0.33$, and $m = 0.8982$ ($STDEVy = 0.369982$, $STDEVx = 0.41189$), therefore $b = y - mx = 0.0501$.

Correlation of gold results is severely limited by gold detection limits and historical reporting of gold assays in oz/t to three decimal places. Combined with a predictable “nugget” effect exacerbated by the small historic core volumes, only a moderate correlation is revealed; however, average grades would be approximately equal, with slightly higher values expected from the modern analysis.

Figure 7.14 shows that the two data sets correlate well for the sulphur % values. Given the mineralized interval's massive pyrite nature, a good correlation would be expected, and significant deviations would demonstrate errors in sampling over a non-mineralized interval.

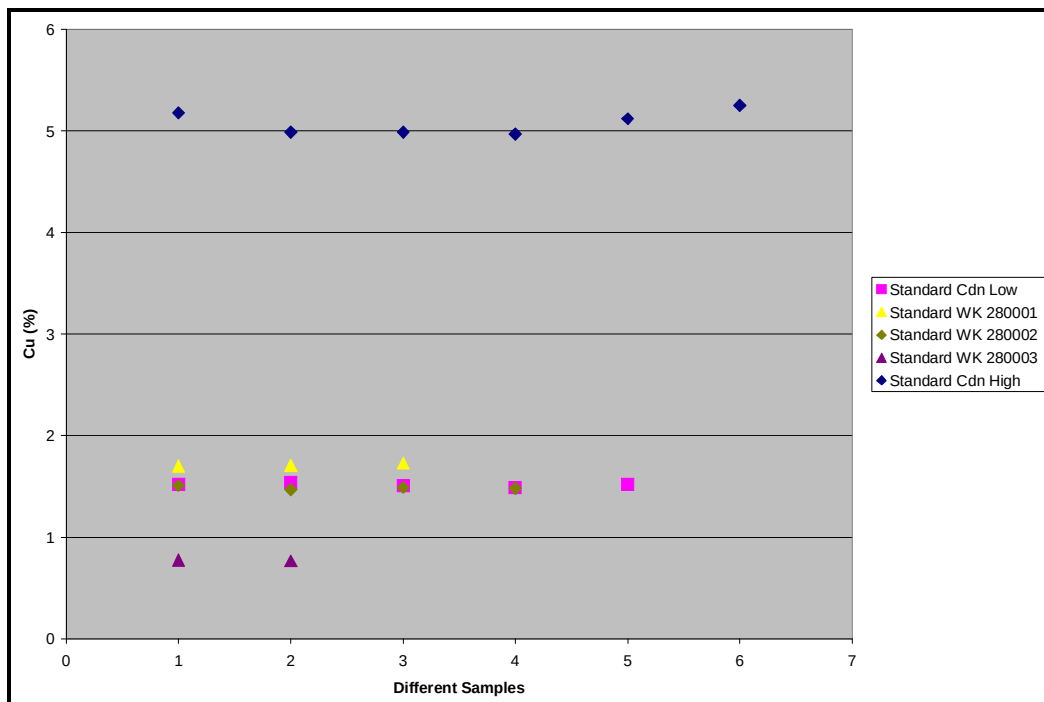
Figure 7.14 S% Comparison



Using the formula from page 7-22 and the values for sulphur, $x = 31.44$, $y = 28.57$, and $m = 0.8777$ ($STDEVy = 13.73373$, $STDEVx = 15.6477$), therefore $b = y - mx = 0.9721$.

A separate standard and blank Quality Assurance/Quality Control (QA/QC) study has been done for the historical core intersection resampling. Since all the standards were part of one sample submission batch, they are plotted as different samples. The results are shown in Figure 7.15, Figure 7.16, and Figure 7.17. The data tables are attached in Appendix A (Table 6-2 and 6-3).

Figure 7.15 Cu Standard Results for Historical Core Intersection Resample



The values for the standards shown in the charts above and below are within recommended ranges and confirm analytical accuracy.

Figure 7.16 Zn Standard Results for Historical Core Intersection Resample

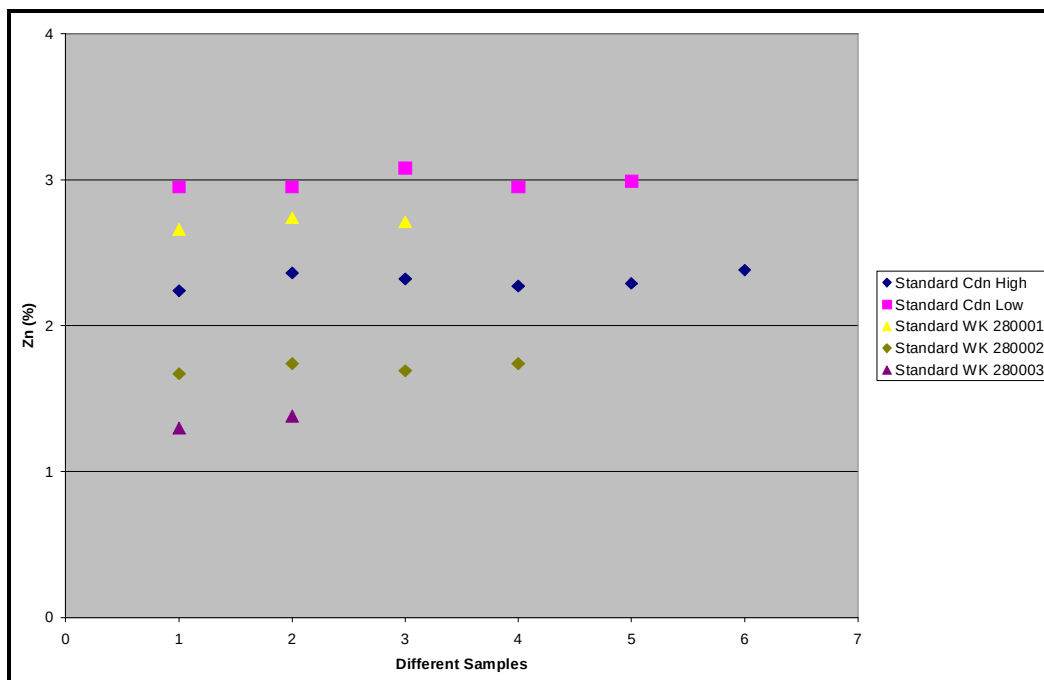
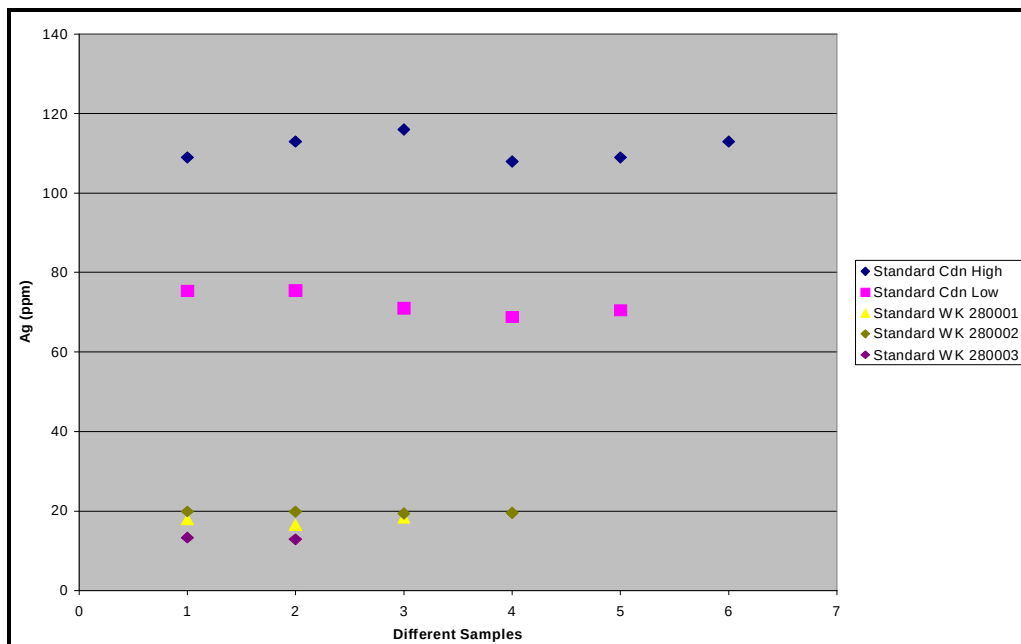
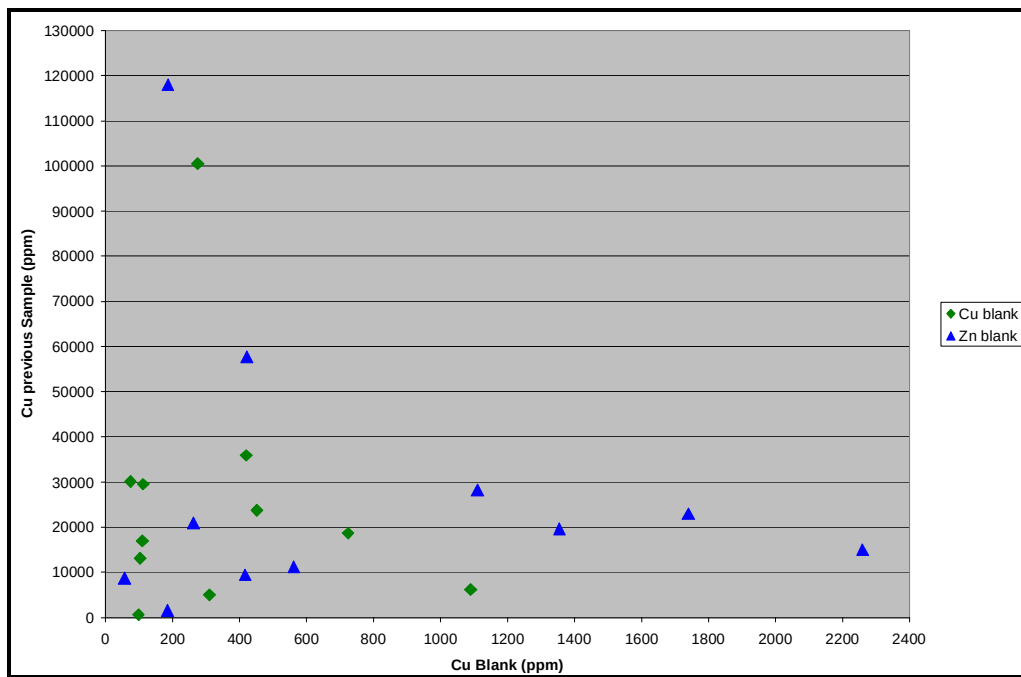


Figure 7.17 Ag Standard Results for Historical Core Intersection Samples



Blanks for historical core resampling consisted of crushed quartz. The copper and zinc values in the blank samples are plotted against the copper and zinc values of the samples preceding the blanks. The results for the blanks can be seen in Figure 7.18 below. Data tables are attached in Appendix A (Table 6-4 and 6-5).

Figure 7.18 Cu and Zn Blank Results for Historical Core Intersection Samples



As with blanks submitted during the 2006 drill sampling program, cross-contamination is evident, but the absolute value of that contamination is within the range of analytical variability for most ore-grade samples. Unequal chart axis ranges exaggerate the cross-contamination effect, and illustrate that it is more difficult to keep zinc from affecting the following sample. The results are suggestive of incomplete labware cleaning between sample preparations.

7.8.3 VALIDATION OF EMC HISTORICAL DATA

No sulphide intersections from EMC core were available for resampling; this was not such a critical issue, however, as analytical certificates were available for most of the data. There has also been sufficient new drilling in the EMC part of the Main deposit to alleviate any concerns about historical analytical quality. Nevertheless, at the insistence of AMEC, WKM attempted to twin three historical holes. Due to different deflections between the historical and modern drilling, it was difficult to keep the twin intersections proximal to one another, so distances between the parts of the drill holes vary from 3–15 m. In general, the twin holes confirm the location of the mineralization, and show a general correspondence to grade variation or internal sulphide stratigraphy. But because sample intervals differ and the drill holes cut the deposit at different angles, there is enough grade variation that it is difficult to draw conclusions about the analytical quality of EMC data. However, the twin holes do confirm the presence and general tenor of mineralization.

Table 7.4 Comparison of Intersections for EMC and Proximal WKM Drill Holes

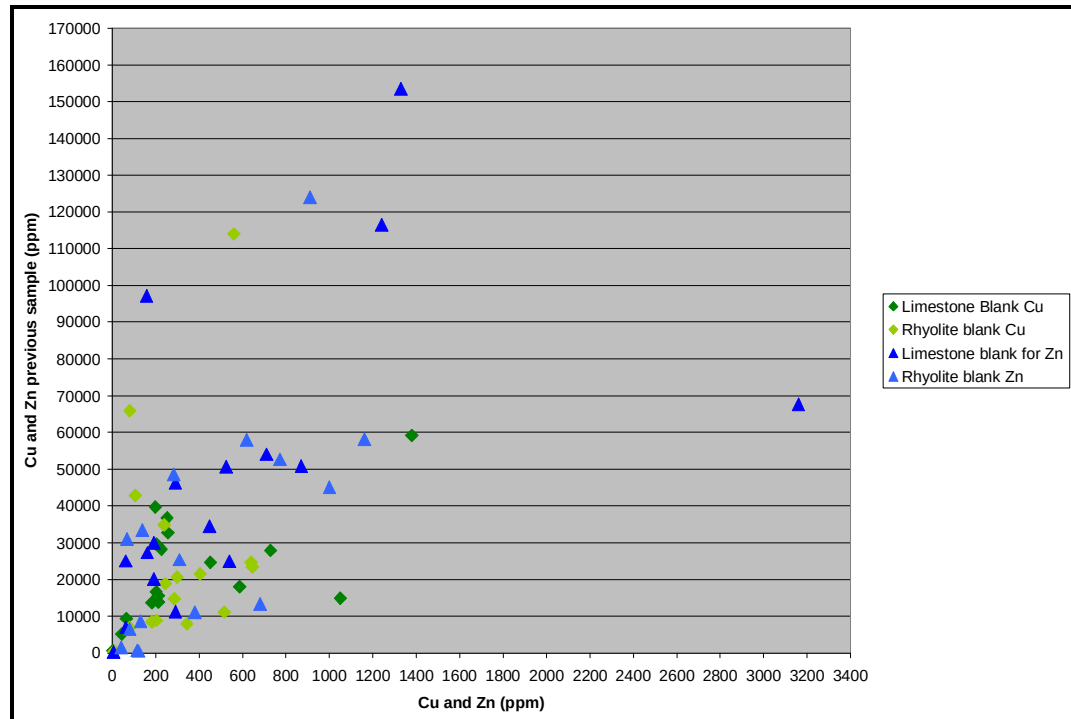
Drill Hole	From (m)	To (m)	Thickness (m)	Cu%	Zn%	Ag g/t	Au g/t
E002	50.9	76.1	25.2	1.17	3.22	20.5	0.20
WK06-74	52.4	75.0	22.6	1.11	4.36	18.5	0.22
E036	96.8	120.2	23.5	1.75	2.73	30.7	0.42
WK06-79	79.3	100.6	21.3	2.40	4.08	36.3	0.43
E054	65.3	84.7	19.4	1.40	1.53	27.5	0.36
WK06-76	57.9	77.8	18.8	1.13	0.94	14.3	0.18

7.8.4 CURRENT DATA

The QA/QC program at the Kutcho project has evolved since 2004. Initially, analytical quality was monitored through the use of company standards and blanks and re-analysis of 5% of the data set at another laboratory. In 2006, additional work was undertaken to ensure data homogenization and conformance to recommended values of the company and project standards. Three different commercially certified standards were added to the program, a full range of standards and blanks were included in the re-analysis of samples by a different lab, and five drill holes were sampled and analyzed twice, providing information on sampling and analytical variance and precision. With the exception of minor contamination of the blank samples, no issues with the quality of the analytical data were identified. Analysis of blank samples showed a general correlation of the value of the blank with the grade

of the preceding sample, strongly suggesting some cross-contamination or carry over. Further investigation and discussion with the lab confirmed that a certain amount of carry over is common, and, in the case of soft massive sulphide mineralization, may be more pronounced. With the possible exception of a single sample, the amount of carry-over would be insignificant to separation of waste-ore, deposit interpretation, or resource estimates; however, better cleaning between samples by the lab is also a possible solution.

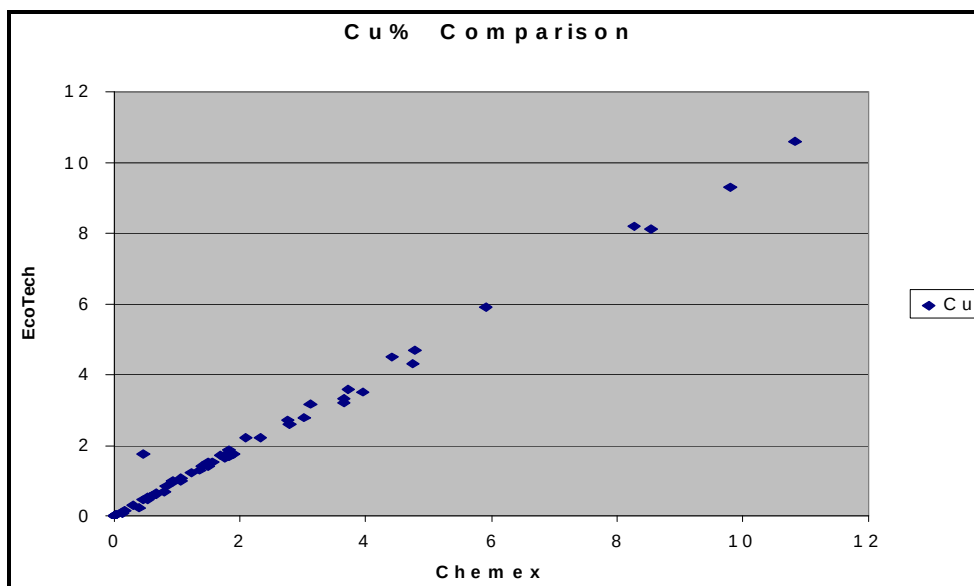
Figure 7.19 Blank Results for Cu and Zn – 2006 Drilling Program



7.8.5 CHECK ANALYSES

A total of 59 pulps (including some blanks and standards) were pulled from Chemex and sent to EcoTech Laboratory Ltd. (EcoTech) in Kamloops for re-analysis. Samples were collected from every 2006 drill hole and every analytical shipment, as well as being collected to span a range of copper and zinc values. Results for all elements were plotted on X-Y scatter plots (copper included below, Figure 7.20). The scatter plots show that there is very little difference between the two laboratories' results.

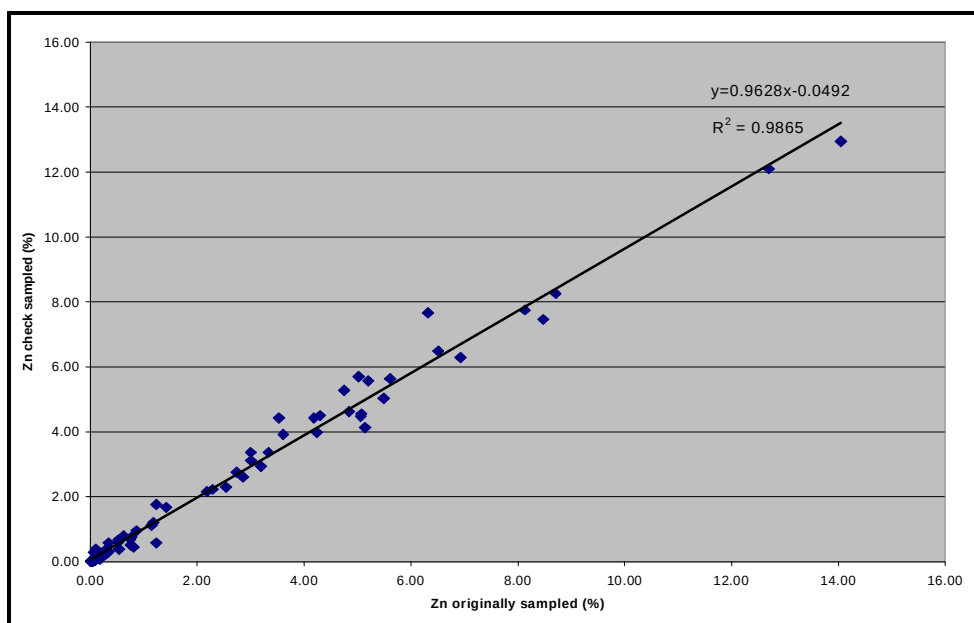
Figure 7.20 Plot of Copper Analyses by Chemex and EcoTech



Resampled drill core was analyzed at the same laboratory (Chemex), and displays good reproducibility. A chart comparing analyses for zinc is shown below.

Using the formula from page 7-22 and the values for zinc from Figure 7.21, $x = 2.06$, $y = 2.03$, and $m = 0.9628$ ($STDEVy = 2.79$, $STDEVx = 2.90$), therefore $b = y - mx = 0.0492$.

Figure 7.21 Zn Comparison for Five Resampled Drill Holes



7.9 ADJACENT PROPERTIES

There is only one property adjacent to WKM's Kutcho property. A number of claims, some of which are contiguous with WKM's claims, were staked to cover the westerly extension of the Kutcho formation by Gary Belik. These claims cover areas that were previously worked by both EMC and Noranda. In 1997, Atna Resources Ltd. optioned the Belik claims, and carried out UTEM surveys and a diamond drilling program. The program identified favourable stratigraphy which was altered and weakly mineralized, but did not encounter economic mineralization.

8.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Material pertaining to this section was not relevant to the mineral resource work on the Kutcho Project.

9.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

9.1 MINERAL RESOURCE ESTIMATE

9.1.1 *DRILL HOLE DATABASE*

Information on the drill hole database on which the mineral resource estimate is based is included in Section 7.0: Geology.

9.1.2 *SAMPLE ASSAY STATISTICS*

The statistics of the assay intervals within the mineralized zones are discussed below. Figure 9.1 shows the statistics and histogram for the assays from the Main Zone, and Figure 9.2 shows the statistics and histogram for the assays from the Esso and Sumac Zones.

Figure 9.1 Main Zone Assay Statistics

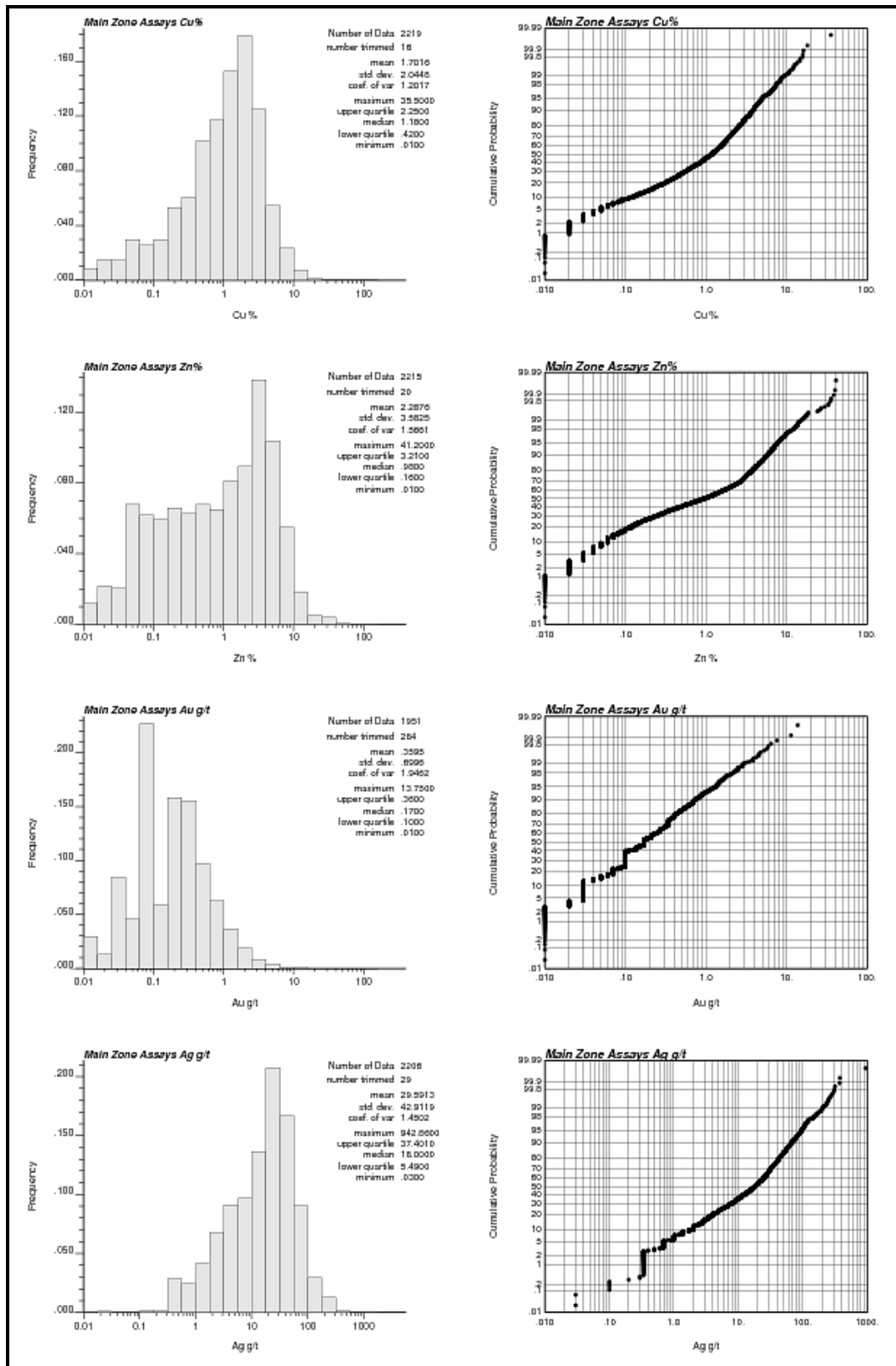
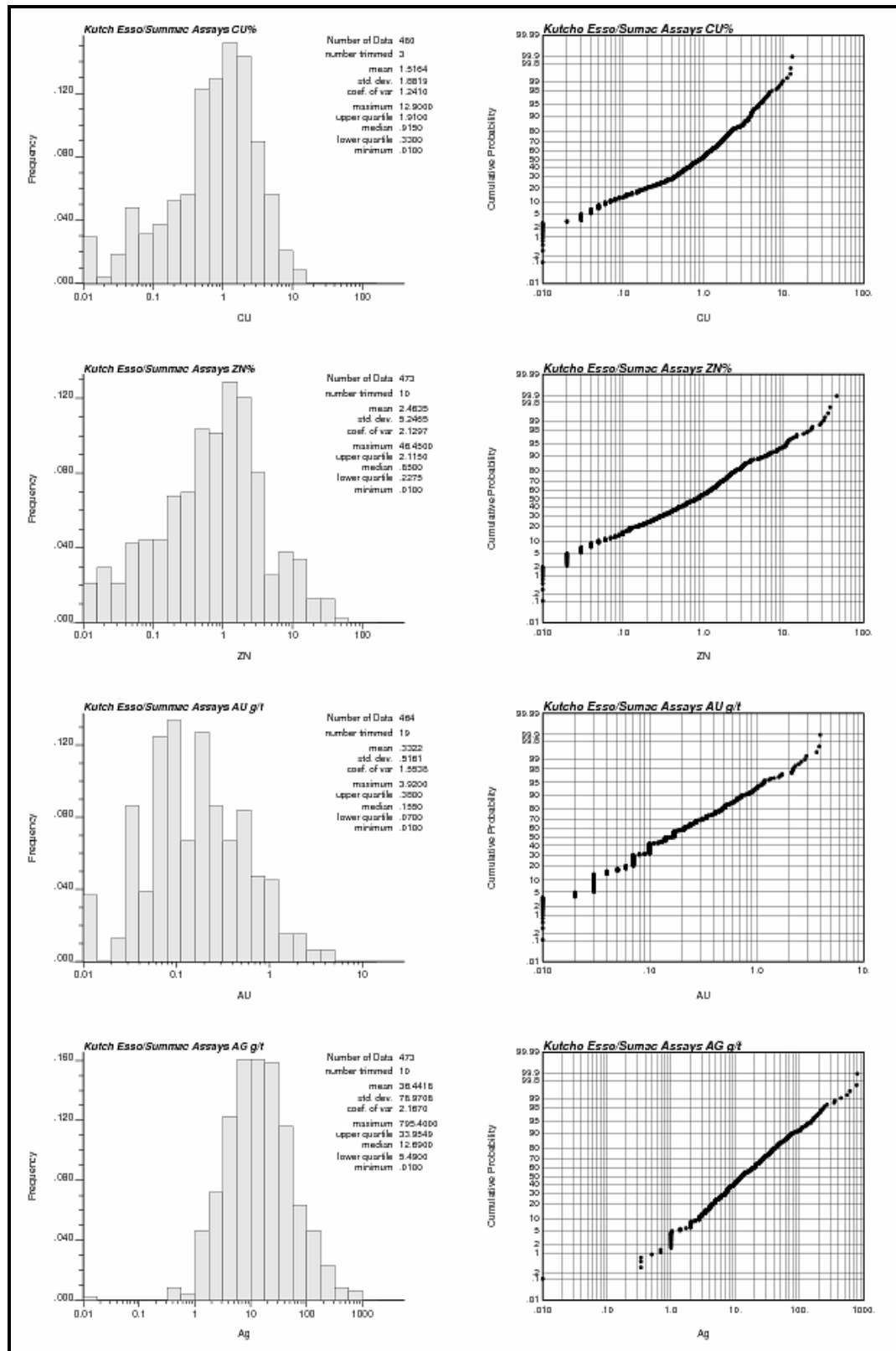


Figure 9.2 Esso/Sumac Assay Statistics



The Main Zone assay data is approximately lognormally distributed in all metals, and displays a fair amount of variability. The gold distribution is strongly positively skewed, with a coefficient of variation (COV) of 1.77. All of the metals exhibit evidence of bimodal populations, with both copper and zinc being distinctly bimodal, as evidenced by flexures in the lognormal probability plots. These plots also illustrate some pronounced breaks in the higher percentiles. Although these breaks can be used to determine thresholds for the capping of high-grade outlier samples before compositing, the qualified person opted to utilize a high-grade restriction approach, as detailed in Section 9.1.6: Composite Statistics.

The statistics for the Esso and Sumac Zones exhibit similar features to those of the Main Zone, although there is a definite increase in variability, as seen in elevated COV values. There are, however, far fewer data available in these zones.

9.1.3 *GEOLOGIC (ROCK) MODELLING*

3D geological solid models of the Main, Esso and Sumac deposits were developed by Western Keltic Mines Inc. (WKM) and used as the bounding domains for grade interpolation in this estimate. The general methodology was as follows:

- Sections were plotted and checked against historical sections for accuracy of location and metal grades.
- Polygons were generated on the sections to represent potentially mineable material, generally a US\$15 cutoff was used for the Main deposit and US\$35 cutoff was used for the Esso deposit; however, some lower grade material was included to maintain continuity in the down-dip direction and to maintain a minimum 3 m horizontal thickness. No cutoff grade was used for the Sumac deposit, as the shell was matched directly to massive sulphide rock type.
- The sectional polygonal interpretations were tied together to create 3D geologic solids that were then used to code the rock type block model used to constrain the grade interpolation.

A 3D perspective view of the Main 3D solid is shown in Figure 9.3, and a representative section in Figure 9.4.

9.1.4 DENSITY

The specific gravity (SG) data used for resource estimation purposes is a combination of measured values and calculated values, and was tabulated by interval in the assay database table. Sumac Mines Limited (SML) conducted SG determinations in the field using a “weigh in water, weigh in air” method, and also had ALS Chemex Laboratories (Chemex) of North Vancouver perform density measurements on uncrushed core from holes KT040 through KT049. Additionally, SML went back to the old core and carried out SG measurements in the field on the main or significant grade parts of the intersections in most (but not all) of the holes drilled prior to hole 40.

The calculated SGs are based primarily on various formulas relating the metal and sulphide content to SG, as documented in a March 2007 memorandum titled “Memorandum on Specific Gravity Determinations and Estimations for the Kutcho Project” that is attached as Appendix B. Sulphur and SG have a very strong correlation in the data that has both analyses, which is the majority of the database. Wardrop has reviewed this report and considers the conclusions and methodology reasonable.

The summary statistics of the SG data are shown in Table 9.1, where SG = Best Choice adopted, SGm = Measured SG, SGe = Estimated SG, and SGc = Calculated SG.

Table 9.1 Summary SG Statistics

Parameter	SG	SGm	SGe	SGc	SGe* Main	SGc* Main	SGe Esso
# of samples	4395	2864	715	2922	295	140	127
Minimum	2.60	2.60	2.60	2.72			
Maximum	5.40	5.40	4.86	4.98			
Mean	3.464	3.549	3.286	3.451	3.430	3.430	3.140
Variance	0.433	0.465	0.323	0.390			
Standard Deviation	0.658	0.681	0.568	0.624			
Skewness	0.636	0.383	1.300	0.640			
Kurtosis	1.957	1.670	3.374	1.957			
COV	0.189	0.191	0.172	0.180			

* EMC data Main deposit (E001-E069)

9.1.5 COMPOSITING

The drill hole assay data was composited into 2.5 m down-the-hole composites within the Main Zone, Esso, and Sumac 3D solids. This composite length was selected with reference to the block dimensions of 5 x 5 x 5 m, and to the occurrence of relatively narrow portions of all of the solids. Composites that were <1 m long (40%) were discarded. Given that the sectional interpretations were largely snapped to assay intervals, not many of these residuals were generated.

9.1.6 COMPOSITE STATISTICS

The 2.5 m mineralized zone composites were then tagged with the rock code from the appropriate 3D solid, and this data was extracted for analysis and modelling. The statistics of the composites within the mineralized zones are discussed below. Figure 9.5 shows the statistics and histogram for the composites from the Main Zone, and Figure 9.6 shows the statistics and histogram for the composites from the Esso and Sumac Zones.

Figure 9.5 Main Zone 2.5 m Composite Statistics

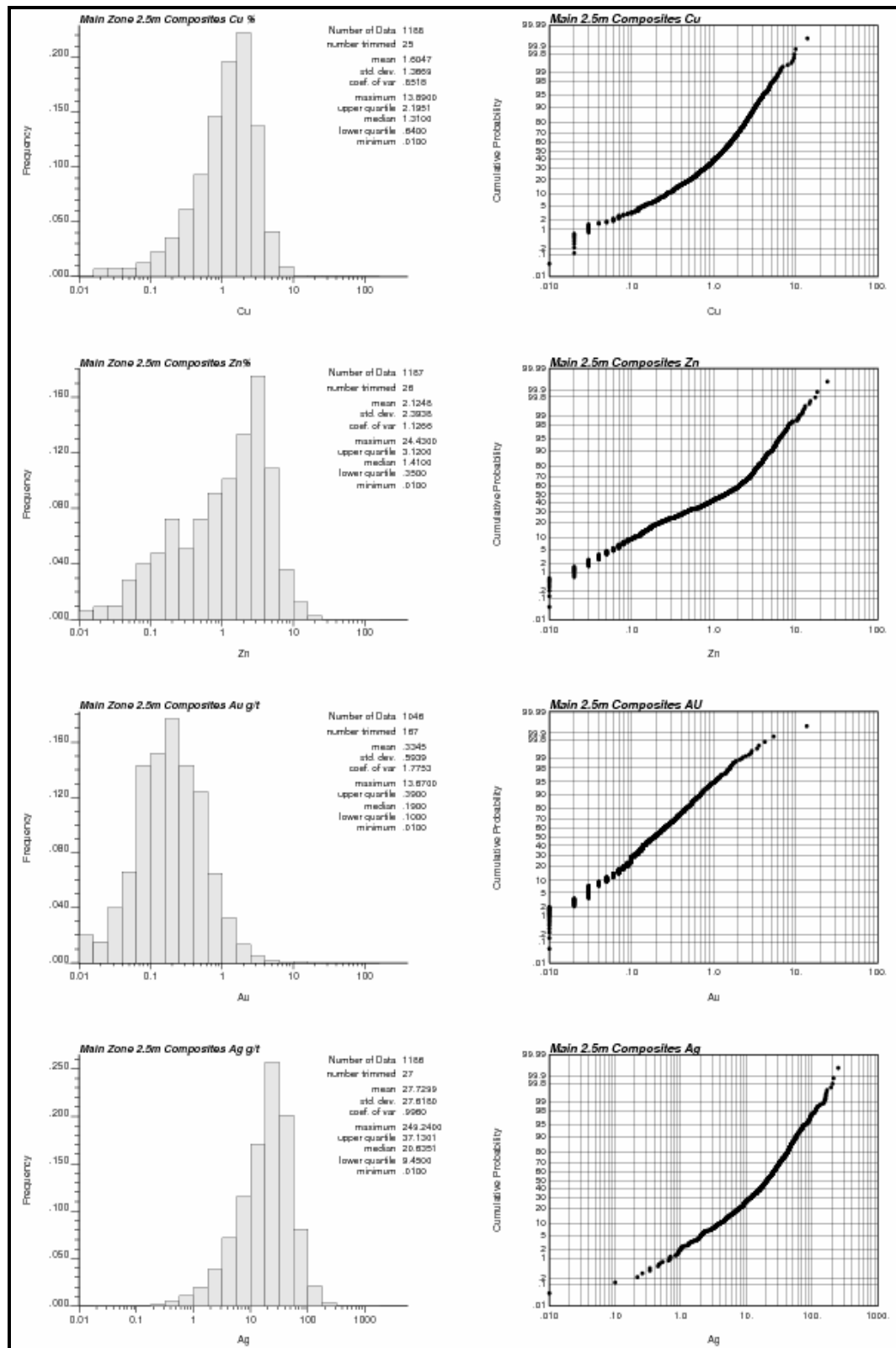
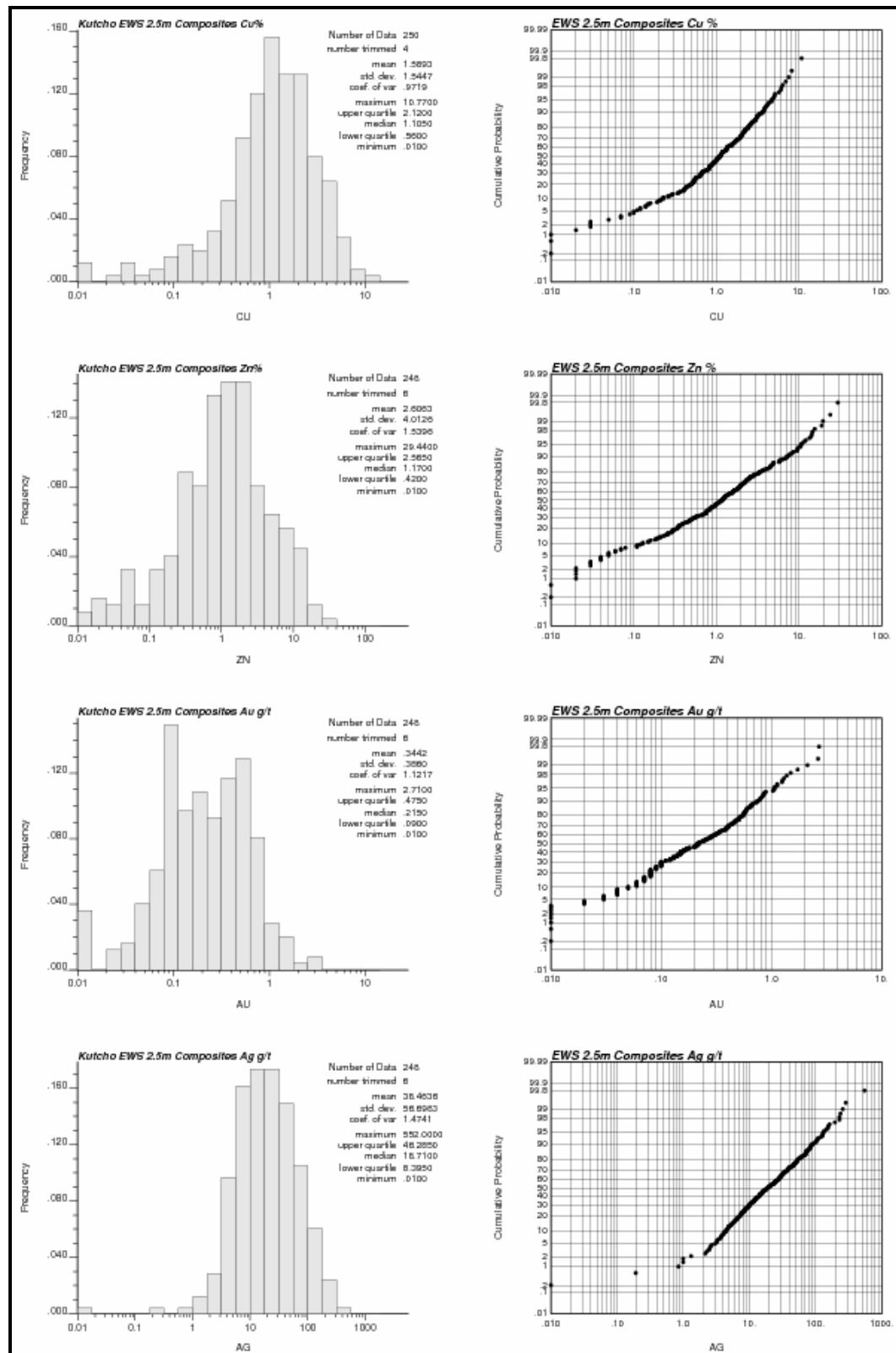


Figure 9.6 Esso/Sumac Zones 2.5 m Composite Statistics



9.1.7 *OUTLIERS AND DECLUSTERING*

The Main Zone composite data is approximately lognormally distributed, and relatively well behaved for most metals, with a COV of 0.85–1.13 for copper, zinc, and silver. The gold distribution is strongly positively skewed, with a COV of 1.77. The gold distribution is unimodal, as opposed to the other metals, which exhibit evidence of bimodal populations. The lognormal probability plots for the same sample set illustrate some pronounced breaks in the higher percentiles. Based on these breaks, values were selected for high-grade restriction in interpolation. High-grade search restrictions were used to constrain isolated high-grade composites, both to reduce the metal at risk and to prevent unwanted smearing of high-grade values into low-grade areas. The method imposes a smaller search ellipse (or shorter range of influence) on composites above a threshold value. Composites with values greater than the established thresholds were not included in the estimation neighbourhood if they were not captured within the smaller search ellipse. This method is considered to be a better approach than high-grade capping, because it allows isolated high-grade composites to have an influence on local blocks, while simultaneously preventing unwarranted smearing of high-grade.

The statistics for the Esso and Sumac Zones exhibit similar features to those for the Main Zone, although there is a definite increase in variability, as seen in elevated COV values. It should be noted, however, that there is considerably less data available in these zones.

Clustering would be present if a disproportionately large percentage of intercepts are concentrated within a limited area of higher grade mineralization. Given the overall spacing of the drilling, declustering of the data was not considered necessary in this case (noting also that ordinary kriging was used for grade interpolation).

9.1.8 *TOPOGRAPHIC SURFACES*

The bounding topographic surfaces used in the resource estimates were generated from a one metre topographic contour map created by McElhanney Engineering Services Limited and supplied by WKM.

9.1.9 *VARIOGRAM ANALYSIS*

There are several mathematical functions that measure variability in space; the qualified person finds that the correlogram, which is a variant of the semi-variogram, generally provides a stable estimate of spatial continuity. The correlogram measures the correlation coefficient between two sets of data, comprising values at both the heads and tails of vectors with similar direction and magnitude. For ease of modelling, the correlogram value is subtracted from one and is presented in a similar graphical form to the variogram. In this report the correlograms presented this way are referred to as variograms. Directional variograms were generated for the Main Zone composites using the SAGE2001 program, in 30° increments of azimuth and 15° increments of dip. The variogram parameters are summarized in Table 9.2, and the variograms are illustrated in Appendix C. Variogram analysis of the Esso/Sumac

composites was inconclusive due to the low number of available samples. Given the similarity between the deposits, the Main Zone variography was extended to these zones.

Table 9.2 Summary of Variogram Parameters

Main Zone Element	Nugget Effect	Spherical Components						
		Sill	Ranges					
			Maximum		Intermediate		Minimum	
			m	Dir./Dip	m	Dir./Dip	m	Dir./Dip
Cu	0.30	0.56	100	83/6	10	194/74	8	172/-15
		0.14	264	277/7	43	7/-5	30	62/82
Zn	0.28	0.49	9	323/17	7	286/-69	6	229/12
		0.23	101	332/68	26	94/12	11	8/-18
Au	0.74	0.23	298	336/-17	20	236/-29	12	272/56
		0.03	842	111/15	292	28/-22	97	171/-63
Ag	0.61	0.22	93	91/-3	24	133/85	6	1/3
		0.17	164	138/34	35	57/-13	25	345/53

9.1.10 GEOLOGIC MODELLING

The 3D geological models of the Main, Esso, and Sumac Zones were used to code the rock type model in Gemcom GEMS modelling software. The rock type model was first initialized with the “Air” rock code, and then all blocks below the topographic surface were set to a background rock code. The 3D solids were then used to overprint blocks falling within the solids, with the percentage being stored to provide accurate volumetrics reporting (partial block model). Additional 3D solids and surfaces were used to create a potential acid generating (PAG) model for use in waste planning. The 2.5 m composites falling within the solids were tagged with corresponding rock codes, to allow for the use of “hard” boundaries (i.e., only composites falling within the modelled zones can be used for interpolating blocks coded by the solids).

9.1.11 MODELLING PARAMETERS

Table 9.3 shows the modelling parameters used to generate the grade models. Separate block models were developed for the Main Zone and for the Esso/Sumac Zones because of the significant strike distance between them. Ordinary kriging was used for all areas, and three passes were utilized. The initial pass used search distances at approximately half of the variogram range, with the second pass using search distances at twice the previous values. The third pass utilized search distances double those of the previous pass. The search ellipsoid orientation was kept constant for all zones. The maximum of two samples per hole dictated that at least two holes were used to meet the minimum requirement of three samples. A nearest neighbour model using the same search parameters was also generated for comparison purposes.

Following interpolation of the copper, zinc, gold, and silver grade models, a model of Copper Equivalent (CuEq) was generated using the following formula:

$$\text{CuEq} = (\% \text{Cu} \times 1) + (\% \text{Zn} \times 0.414) + (\text{g/t Ag} \times 0.00256) + (\text{g/t Au} \times 0.119)$$

This equivalent is calculated using prices of US\$2.36/lb copper, US\$1.38/lb zinc, US\$534.36/oz gold, and US\$9.78/oz silver. Metallurgical recoveries are 87% copper, 79% zinc, 31% gold, and 37% silver.

Table 9.3 Summary of Modelling Parameters

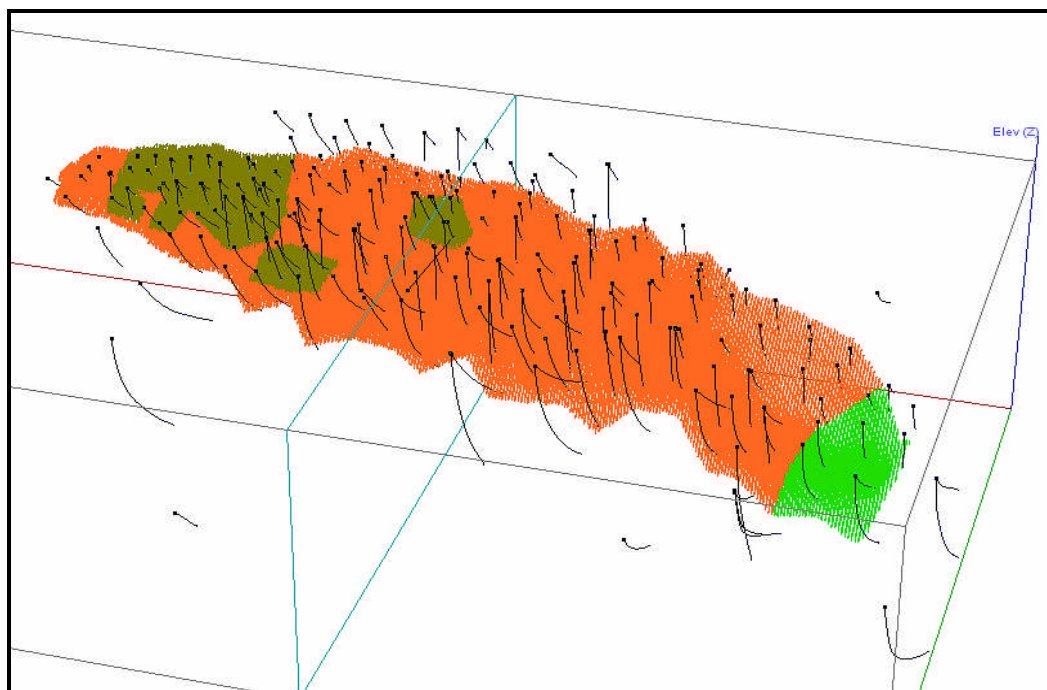
Pass	Azimuth/Dip		Search Distance		Z-Search Direction	Restriction Threshold	Max/ Hole	Min/Max Comps
	X- Direction	Y- Direction	X-	Y-				
Cu Pass 1	100/0	10/-45	50	50	20	15 %	2	5/12
Cu Pass 2	100/0	10/-45	100	100	40	15 %	2	3/12
Cu Pass 3	100/0	10/-45	200	200	80	15 %	2	3/12
Zn Pass 1	100/0	10/-45	50	50	20	20 %	2	5/12
Zn Pass 2	100/0	10/-45	100	100	40	20 %	2	3/12
Zn Pass 3	100/0	10/-45	200	200	80	20 %	2	3/12
Ag Pass 1	100/0	10/-45	50	50	20	100 g/t	2	5/12
Ag Pass 2	100/0	10/-45	100	100	40	100 g/t	2	3/12
Ag Pass 3	100/0	10/-45	200	200	80	100 g/t	2	3/12
Au Pass 1	100/0	10/-45	50	50	20	8 g/t	2	5/12
Au Pass 2	100/0	10/-45	100	100	40	8 g/t	2	3/12
Au Pass 3	100/0	10/-45	200	200	80	8 g/t	2	3/12

Note:

- the search distances for the Esso/Sumac model were increased twofold to account for the decreased number of composites available in these zones.
- the ranges used for values falling above the restriction threshold were 12 x 12 x 5 m in the X, Y, and Z axes.

9.1.12 MINERAL RESOURCE CLASSIFICATION

The mineral resource has been classified with respect to the mean of distance to the samples used to interpolate a particular block, and the kriging variance relative to the drill hole spacing in strike and dip. An average distance of 30 m and relative kriging variance of 0.6 (based on examination of kriging variance relative to pierce points in the plane of the deposit) was used to identify Measured material. The average distance was increased to 70 m for Indicated material and an average distance greater than 70 m identified Inferred material. The Main Zone classification relative to the drill holes is illustrated in Figure 9.7.

Figure 9.7 Main Zone Resource Classification – Looking Southeast

Brown = Measured, Orange = Indicated, Green = Inferred

9.1.13 MINERAL RESOURCE SUMMARY

The mineral resources estimated for the Main, Esso, and Sumac Zones are tabulated in Table 9.4 at a cutoff of 0.75% copper equivalent.

Table 9.4 Resource Summary

	Tonnage	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	CuEq (%)
Main Deposit						
Measured	2,937,507	0.39	27.98	1.83	2.65	3.06
Indicated	12,716,725	0.31	25.69	1.60	2.04	2.56
Measured plus Indicated	15,654,232	0.32	26.12	1.65	2.15	2.65
Inferred	811,103	0.33	24.17	0.95	1.92	1.86
Sumac Deposit						
Inferred	10,614,652	0.14	13.96	0.94	1.45	1.60
Esso Deposit						
Indicated	2,040,445	0.49	37.70	2.24	3.96	4.05
Inferred	442,506	0.53	38.09	2.47	4.15	4.37
All Deposits						
Measured and Indicated	17,694,678	0.34	27.45	1.71	2.36	2.81
Inferred	11,868,260	0.17	15.55	1.00	1.58	1.72

9.1.14 *MODEL VALIDATION*

A number of methods were used to validate the grade modelling:

- visual comparison of block and composite grades on plan and section
- global statistical comparison of block and declustered (nearest neighbour) composite grades
- local comparison of block and declustered (nearest neighbour) composite grades.

9.1.15 *VISUAL COMPARISON*

Few discrepancies between block and composite grades were observed.

9.1.16 *GLOBAL STATISTICAL COMPARISON*

The mean of the block grades in the Main and Esso/Sumac Zones lie within the acceptable limit for global bias of +/-5% of the mean of the declustered (nearest neighbour) composites.

9.1.17 *LOCAL COMPARISON OF BLOCK AND DECLUSTERED COMPOSITE GRADES*

Swath plots were generated to check for local estimation bias. Average kriged block values on a series of parallel slices are plotted against the average declustered (nearest neighbour) values in the three primary model directions (row, column, and level), as illustrated in Figure 9.8 and Figure 9.9.

Kriged blocks generally honour the distribution of declustered (nearest neighbour) sample grades, indicating that there is no local bias. Deviation notes generally correspond to areas where the number of samples is low.

Figure 9.8 Main Zone Swath Plots Cu

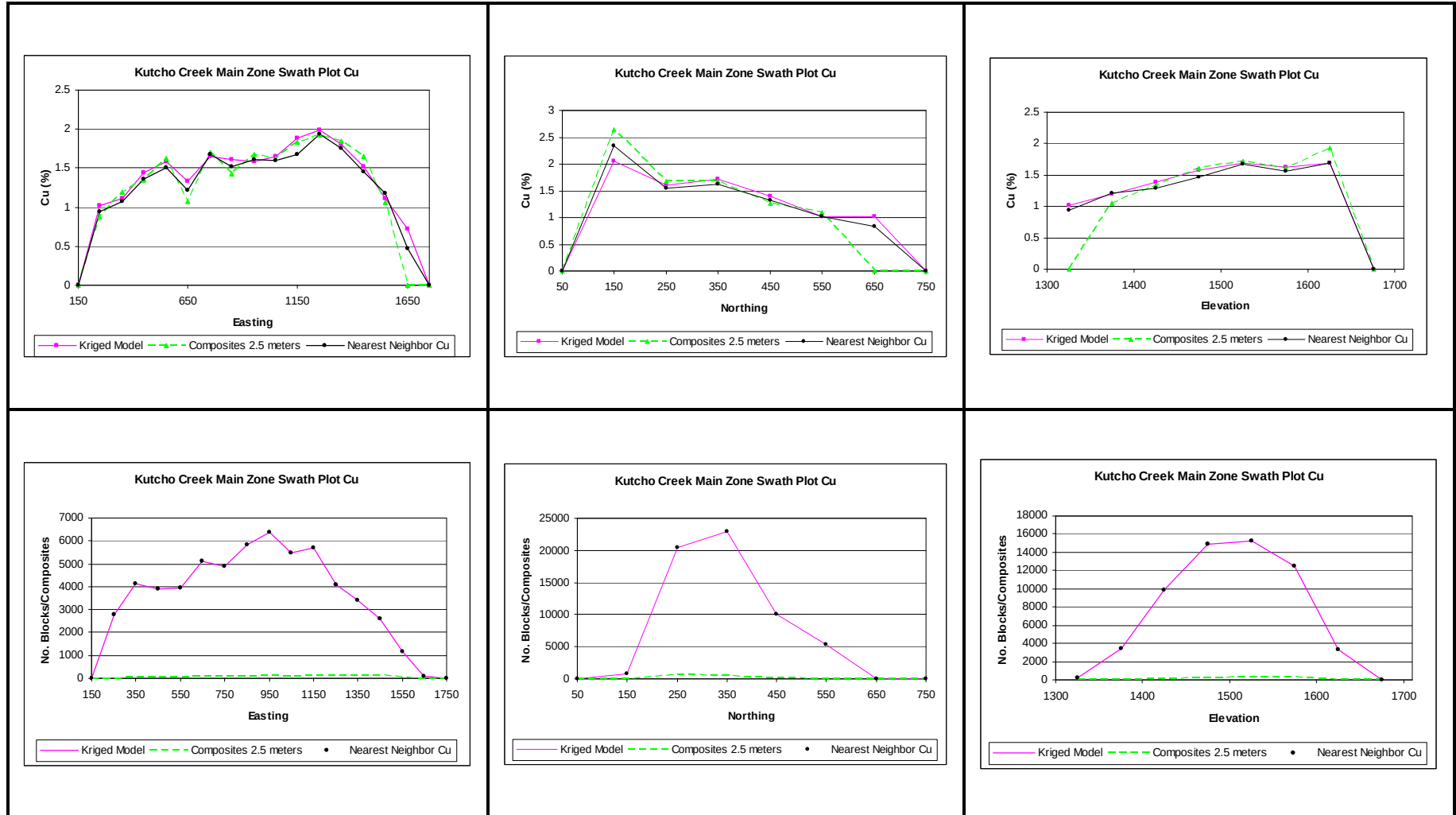
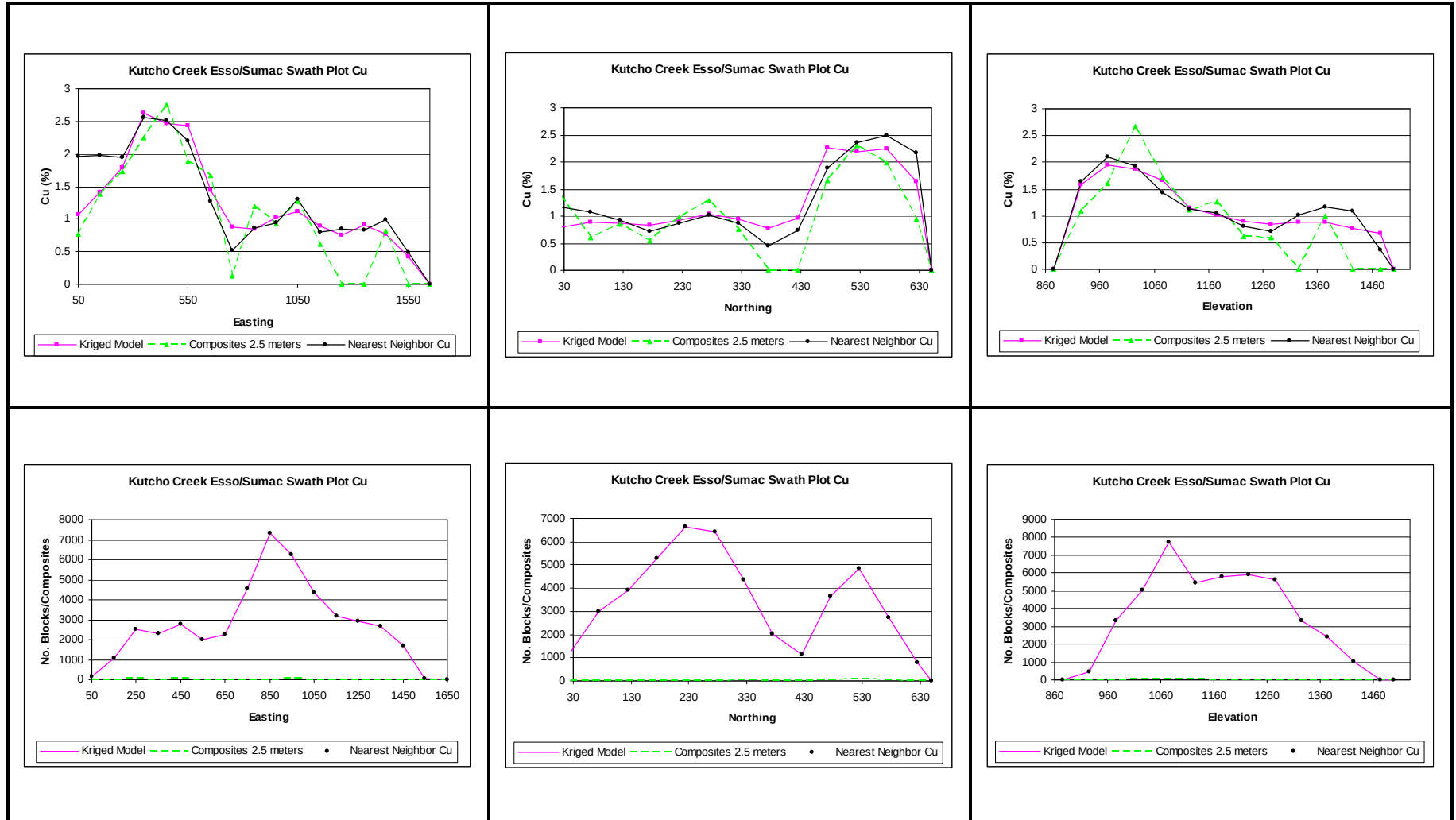


Figure 9.9 Ezzo/Sumac Zone Swath Plots Cu



9.1.18 *MINERALOGICAL AND METALLURGICAL CORRELATION – MAIN DEPOSIT AND ESSO DEPOSIT*

Mineralogically, the Main orebody and Esso orebody are essentially identical, and can be expected to respond in the same way quantitatively and qualitatively to mineral separation processes.

10.0 OTHER RELEVANT DATA AND INFORMATION

No other data or information is relevant for the review of the Kutcho Project.

11.0 INTERPRETATION AND CONCLUSIONS

The Kutcho Project, comprising the Main, Sumac, and Ezzo deposits, is a volcanogenic massive sulphide deposit containing copper, zinc, silver, and gold.

Wardrop estimate that the three deposits contain 17.7 Mt of mineral resource in the Indicated and Measured categories grading 1.71% copper, 2.36% zinc, 27.45 g/t silver, and 0.34 g/t gold. The deposit also contains 11.9 Mt resource in the Inferred category grading 1.0% copper, 1.58% zinc, 15.55 g/t silver, and 0.17 g/t gold.

12.0 RECOMMENDATIONS

Based on knowledge of similar projects and of prevailing market conditions for metal concentrates that could be produced from the Kutcho Project, Wardrop recommends that feasibility studies be conducted to convert the mineral resource to a mineral reserve and to evaluate the economic viability of the project.

13.0 REFERENCES

- Bridge, D., 1982. 1981 Progress Report on the Kutcho Creek Property, an unpublished report for Esso Minerals Canada Ltd.
- Bridge, D., 1983. 1982 Progress Report on the Kutcho Creek Property, an unpublished report for Esso Minerals Canada Ltd.
- Bridge, D., 1984. 1983 Progress Report on the Kutcho Creek Property, an unpublished report for Esso Minerals Canada Ltd.
- Didur, B., 1979. Kutcho Creek Ore Reserves. Unpublished report prepared for Esso Minerals Canada Ltd.
- Didur, B., 1981. Diamond Drill Results and Ore Reserve Estimates. Unpublished text of presentation given in Tokyo Japan on behalf of Esso Minerals Canada Ltd.
- Gabrielse, H., 1978. Geology of NTS Map Sheet 104I (Cry Lake); Geological Survey of Canada, Open File, # 610.
- Gasparini, C., 1979. Study of Cu, Zn, and Ag distribution in Five Samples from the Kutcho Creek; In: Summary of 1978 Metallurgical testwork for Kutcho Creek, by H.E. Neal.
- Holbek, P.M., 1985. 1984 Exploration Report on the Kutcho Creek Project; an unpublished report for Esso Minerals Canada.
- Holbek, P.M., 1989. 1988 Geochemical and Geophysical Report on the Kutcho South Area – Kutcho 89A and 89B Claim Groups. British Columbia Assessment Report.
- Holbek, P.M., 1990. 1990 Diamond Drilling Report on the Kutcho Creek Property. British Columbia Assessment Report.
- Holbek, P.M., and Heberlein, D., 1986. 1985 Exploration Report on the Kutcho Property; and unpublished report for Esso Minerals Canada.
- Holbek, P.M., and McPherson, M.D., 1990. The Kutcho Creek Property: A summary of exploration status and proposed future work; unpublished report for Homestake Canada Limited.
- Holbek, P.M., and Champigny, N., 1992. Geological Reserve Estimate for the Kutcho Lens, Kutcho Creek Volcanogenic Massive Sulphide Deposits,

Northwestern British Columbia; unpublished report for Homestake Canada Limited, American Reserve Mining Corp., and Sumitomo Metal Mining Canada Ltd.

Pearson, D.E., and Pantaleyev, A., 1975. Cupiferous Iron Sulphide Deposits, Kutcho Creek Map Area (104I/1W); Geological Field Work, British Columbia Ministry of Mines and Petroleum Resources, pp. 86-93.

Smith, J.B., 1991. Report on the Potential for Underground Mining – Main Zone; Kutcho Creek Property. A study by Laxey Mining Services for American Reserve Mining Corp. and Homestake Canada Ltd.

Stanton, R.L., 1991. Understanding Volcanic Massive Sulphides – Past, Present, and Future. In: Historical Perspectives of Genetic Concepts and Case Histories of Famous Discoveries, Papers Arising from SEG Symposia, Oct 30 and 31, 1988 Denver, Colorado; Economic Geology Monograph 8, Skinner, B.J., Ed.

Sumitomo Exploration Dept., 1984. ORC1: Explanatory note on the ore reserve calculation of the Kutcho main ore body, B.C., Canada; unpublished report for Sumitomo Metal Mining Canada Ltd.

Thorstad, L.E., 1983. The Upper Triassic Kutcho Formation, Cassiar Mountains, North Central British Columbia; unpublished M.Sc. Thesis, The University of British Columbia.

Thorstad, L.E., and Gabrielse, H., 1986. The Upper Triassic Kutcho Formation, Cassiar Mountains, North Central British Columbia; Geological Survey of Canada, Paper 86-16, 53p.

Wright Engineers Limited, 1985. Pre-feasibility Study of the Kutcho Creek Project for Esso Minerals Canada Ltd. and Sumac Mines Ltd.

14.0 CERTIFICATE OF QUALIFIED PERSONS

I, Timothy Carew, of Reno, Nevada do hereby certify that as the author of this **TECHNICAL REPORT ON THE KUTCHO PROPERTY, BRITISH COLUMBIA, CANADA**, I hereby make the following statements:

- I am a Geological Consultant doing business as Reserva International LLC with a business address at PO Box 19848, Reno, NV 89511, USA.
- I graduated with a B.Sc. Honours degree in Geology from the University of Rhodesia in 1971. I have also earned a B.Sc. Special Honours degree in Geology (University of Rhodesia, 1976) and a Master's degree in Mineral Production Management (Royal School of Mines, 1981).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License # 19706).
- 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 purpose of NI 43-101.
- I have practiced my profession in mineral exploration and exploitation continuously since graduation. I have over 35 years experience in mineral exploration and have 25 years experience in preparing mineral resource estimates using block-modelling software.
- I am responsible for the preparation of Sections 4, 5, 6, 7, and 9.1 of this technical report titled "Technical Report on the Kutcho Property, British Columbia, Canada". In addition, I visited the Property on 11 June 2007.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 9th day of August, 2007 at Reno, NV

"Original Document, Revision 00 signed
and sealed by Timothy Carew, P.Ge."

Timothy Carew, P.Ge.

Principal

Reserva International LLC

APPENDIX A

2006 DIAMOND DRILLING PROGRAM

QA/QC STUDY

KUTCHO PROJECT
2006 DIAMOND DRILLING PROGRAM
QA/QC STUDY

Western Keltic Mines Inc.

Anja Weiss
Rob G. Wilson P.Geol.
Peter M. Holbek P.Geol.

March 23, 2007

Table of Contents

	Page
1. Introduction	1
2. Procedures	1
2.1 Core Sampling	1
2.2 Assaying.....	1
2.3 Specific Gravity	2
2.4 Blanks & Standards.....	2
3. WKM Internal Standard Sample Characterization	2
4.0 Analytical Accuracy: 2006 results for Standards.....	6
5. Quality Control.....	9
5.1 Results for 2006 Blanks.....	9
5.2 Analytical Variation with Time: Combined 2005 and 2006 WKM Standards and ALS Chemex Internal Standards.....	11
6. Validation of SML Historical Analyses	13
7.0 Study of Variation in Core Sampling.....	21
8. Check Analyses on Duplicate Pulps.....	27
9.0 Conclusions	31

List of Tables

Table 3.1: Cu (ppm) values by different Labs and Methods	3
Table 3.2: Zn (ppm) values by different Labs and Methods.....	4
Table 3.3: Ag (ppm) values by different Labs and Methods.....	5

List of Charts

Chart 3.1: Lab Comparison for Cu.....	3
Chart 3.2: Lab Comparison for Zn	4
Chart 3.3: 2005 Lab Comparison for Ag (Not tested in 2005).....	5
Chart 4.1: Results for Cu Standard	6
Chart 4.2: Results for Zn Standards.....	7
Chart 4.3: Results for Western Keltic Ag Standards	7
Chart 4.4: Results for Canadian Resource Labs Ag Standards	8
Chart 4.5: Results for Au Standards.....	8
Chart 4.6: Results for S Standards.....	9
Chart 5.1.1: Blank results for Cu and Zn (Data located in Table 5.1.1 to 5.1.4)	10
Chart 5.2.1: Results for Cu Standard	11
Chart 5.2.2: Results for Zn Standards.....	12
Chart 5.2.3: Results for Ag Standards	12

Chart 5.2.4: Results for Au Standards.....	13
Chart 5.2.5: Results for S Standards.....	13
Chart 6.1: Cu comparison.....	14
Chart 6.2: Zn comparison.....	15
Chart 6.3: Ag comparison.....	16
Chart 6.4: Au comparison.....	16
Chart 6.5: S% comparison.....	17
Chart 6.6: Cu standard results for historical core intersection resample.....	18
Chart 6.7: Zn standard results for historical core intersection resample.....	18
Chart 6.8: Ag standard results for historical core intersection resample.....	19
Chart 6.9: Au standard results for historical core intersection resample.....	19
Chart 6.10: S standard results for historical core intersection resample.....	20
Chart 6.11: Cu and Zn blank results for historical core intersection resample.....	20
Chart 7.1: Cu comparison.....	22
Chart 7.2: Zn comparison.....	22
Chart 7.3: Ag comparison.....	23
Chart 7.4: Au comparison.....	23
Chart 7.5: S comparison.....	24
Chart 7.6: Cu and Zn comparison for the original and resample blank results versus the previous samples.....	25
Chart 7.7: Cu values of the original and re-sampled standards.....	25
Chart 7.8: Zn values of the original and re-sampled standards.....	26
Chart 7.9: Ag values of the original and re-sampled standards.....	26
Chart 7.10: Au values of the original and re-sampled standards.....	27
Chart 8.1: Comparison of Cu% for Chemex and EcoTech.....	28
Chart 8.2: Comparison of Zn% for Chemex and EcoTech.....	28
Chart 8.3: Comparison of Ag g/t for Chemex and EcoTech.....	29
Chart 8.4: Comparison of Au g/t for Chemex and EcoTech.....	29
Chart 8.5: Comparison of S% for Chemex and Ecotech.....	30

Appendices

Appendix I Data Tables

1. Introduction

A multi-phased quality assurance – quality control program (QA/QC) was implemented during the 2006 diamond drill program at Kutcho Creek. The QA/QC program involved the use of blank samples as well as standards supplied by Western Keltic (internal standard samples) and Canadian Resource Laboratories (Certified Reference Material). Verification of Western Keltic's internal standard samples was completed and an instrumentation drift study was accomplished by comparing Western Keltic's internal standards and ALS Chemex internal standards for sample batches submitted during 2005 and 2006. Re-sampling of historical mineralized core drilled by Sumitomo and check sampling of 5 holes from 2006 was carried out. As well, 59 samples analyzed 2006 by ALS Chemex were sent to umpire lab Eco Tech Laboratories in Vancouver for comparative analysis.

2. Procedures

2.1 Core Sampling

Drill core was transported from the drill by truck or helicopter to the core logging facilities situated adjacent to the Kutcho Airstrip, approximately 8 km west of the drill area. Core was geologically logged and marked for splitting with approximately 70% of the deposit intersections also selected for metallurgical study. In the later case, the core was sawn in half and the metallurgical samples, consisting of half the core, collected by a geologist and sealed in a labeled, nitrogen filled bag. The half core was sawn into quarters and the geologist collected the quarter core sample, measured the specific gravity, placed the sample in a plastic bag and heat-sealed the bag closed. The process was the same for core not being used for metallurgical samples, except that half core samples were collected and not packed in nitrogen. Sealed sample bags were then placed into labeled rice bags for shipment to the laboratory. Sample shipment bags were transported to Dease Lake (or Smithers) by aircraft and picked up at the aircraft hanger by the trucking company for transport to the laboratory.

2.2 Assaying

All routine analysis of drill core was carried out by ALS Chemex Laboratories (Chemex), located in North Vancouver. In 2006, a total of 872 core samples (including 50 blanks and 74 standards) were analyzed by Chemex using ICP methods for 33 elements following an aqua-regia digestion. Copper, zinc, or silver values above the ICP detection limits (10,000 ppm for Cu and Zn, and 100 ppm for Ag) were assayed by atomic absorption methods following an aqua-regia digestion. All samples were analyzed for gold by fire assay on 30g sub-samples and sulphur was analyzed by Leco furnace.

2.3 Specific Gravity

All sampled core except the first three 2006 holes were measured for Specific Gravity (SG) unless the rock was so highly altered as to render a measurement meaningless. SG's were measured in the field by weighing the sample using an Ohaus scale in air and in water, with the SG calculated using the formula (wt in air / (wt in air – wt in water)). Sample weights were accurate to the nearest tenth of a gram.

2.4 Blanks & Standards

Blanks and Standards were inserted into the sample stream at a rate of 1 per 10 to 20 samples and numbered consecutively with the core samples. Field blanks consisted of non-mineralized limestone and rhyolite crush. The Western Keltic internal field standards were prepared in 2004 by collecting 30 kgs of mineralized material from the dumps adjacent to the adit. The material was visually hand sorted into low, medium and high grade material and shipped to Pioneer Laboratories in Richmond B.C. for grinding and homogenization with aliquots placed in laboratory supplied pulp bags. The WKM standards were assayed by 3 independent laboratories ensure that the samples were fully homogenous, and to determine the grades for the standards. Additionally, for 2006, standards at three copper and zinc grades from Canadian Resource Laboratories were also used.

3. *WKM Internal Standard Sample Characterization*

Western Keltic's internal standards were created in 2005 by collecting mineralized material from the bulk sample mined underground from Kutcho's Main deposit. The samples were crushed and pulverized before being well mixed to ensure homogeneity. Sample aliquots were analyzed by several laboratories to ensure consistency of grades. Standards at three different grades were prepared.

In 2005 the standards were sent to Pioneer Laboratories, Assayers Canada, and Acme Laboratories. In 2006, two each of the three Western Keltic Standards were sent to Assayers Canada, to International Plasma Laboratories, as well as to SGS Lakefield for additional comparative analysis. Those results were compared to the average ALS Chemex assay results for the standards used in the 2006 Drill Program. Charts 3.1 to 3.3 show the comparison for Cu, Zn, and Ag analyses, respectively. Tables 3.1 to 3.3 contain the data for the charts. Note that in 2005, standard samples were not assayed for Ag, Au or S.

The charts and tables below demonstrate that the average of the Chemex assayed WKM internal standards fall within the range of the standards as analyzed by the Reference Labs. The standards analyzed by Assayers Canada'06 however generally appear to be lower and those assayed by IPL using multi-acid digestion ICP are higher for Cu, Zn and Ag; because these values are systematically lower for all elements this is interpreted as a 'Lab' difference and not as an indication of variability of the standard.

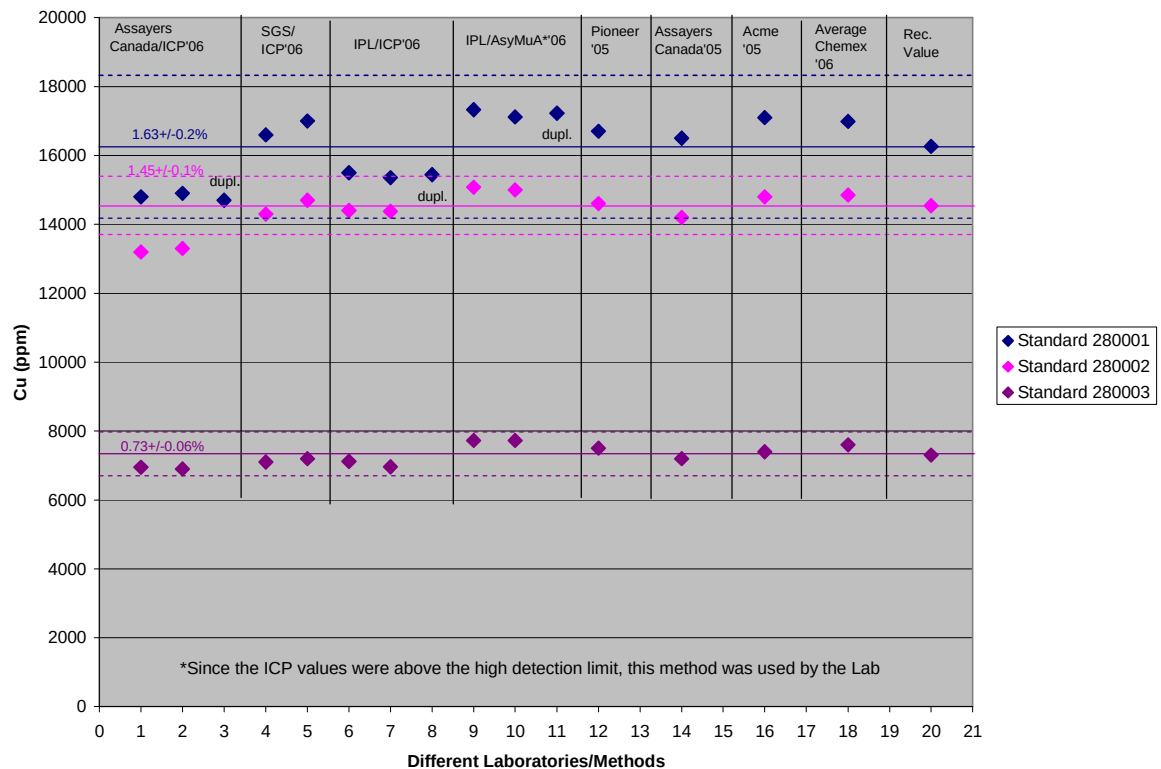


Chart 3.1: Lab Comparison for Cu

Table 3.1: Cu (ppm) values by different Labs and Methods

WKM Internal Standard Id	Assayers Canada	IPL	IPL/Asy*	SGS	Pioneer	Assayers Canada	Acme	Average Chemex	Average	STDEV
	ICP'06	ICP'06	MuA'06	ICP'06	ICP'05	ICP'05	ICP'05	ICP'06		
280001a	14800	15497	17328	16600	16700	16500	17100	16989	16439	862
280001b	14900	15359	17113	17000				16989	16272	1057
280001 dupl.	14700	15441	17226					16989	16089	1218
280002a	13300	14400	15080	14300	14600	14200	14800	14850	14441	549
280002b	14300	14375	14999	14700				14850	14645	301
280003a	6950	7116	7726	7100	7500	7200	7400	7602	7324	274
280003b	6900	6960	7726	7200				7602	7278	373

*AsyMuA: Assay Multi Acid Digestion

Recommended values are as follows:

WKM SRM1 Cu 1.63% +/-0.2
WKM SRM2 Cu 1.45% +/-0.1
WKM SRM 3 Cu 0.73% +/-0.06

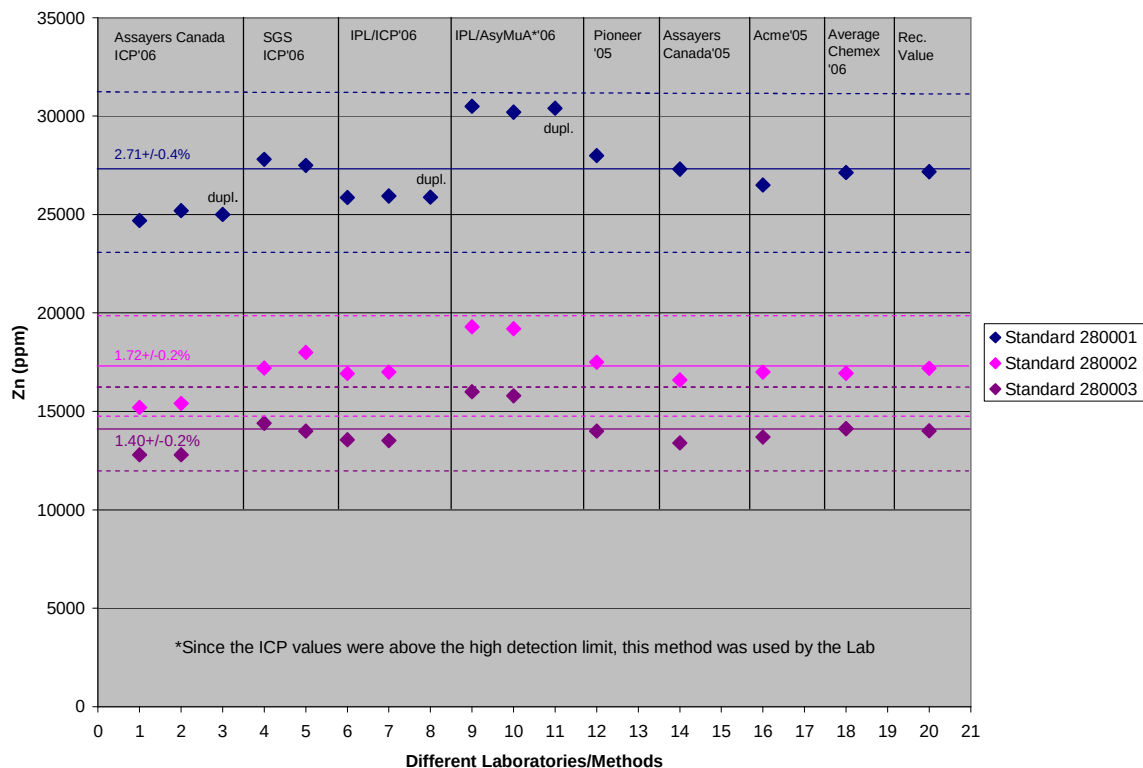


Chart 3.2: Lab Comparison for Zn

Table 3.2: Zn (ppm) values by different Labs and Methods

	Assayers Canada	IPL	IPL/MuA*	SGS	Pioneer	Assayers Canada	Acme	Average Chemex		
WKM Internal Standard Id	ICP'06	ICP'06	ICP'06	ICP'06	ICP'05	ICP'05	ICP'05	ICP'06	Average	STDEV
280001a	24700	25869	30500	27800	28000	27300	26500	27133	27225	1708
280001b	25200	25936	30200	27500				27133	27194	1917
280001 dupl.	25000	25878	30400					27133	27103	2366
280002a	15200	16924	19300	17200	17500	16600	17000	16933	17082	1130
280002b	15400	16997	19200	18000				16933	17306	1409
280003a	12800	13562	16000	14400	14000	13400	13700	14130	13999	946
280003b	12800	13517	15800	14000				14130	14049	1109

*MuAICP: Multi Acid Digestion ICP

Recommended values are as follows:

WKM SRM1 Zn 2.72% +/-0.4
WKM SRM2 Zn 1.71% +/-0.2
WKM SRM 3 Zn 1.40% +/-0.2

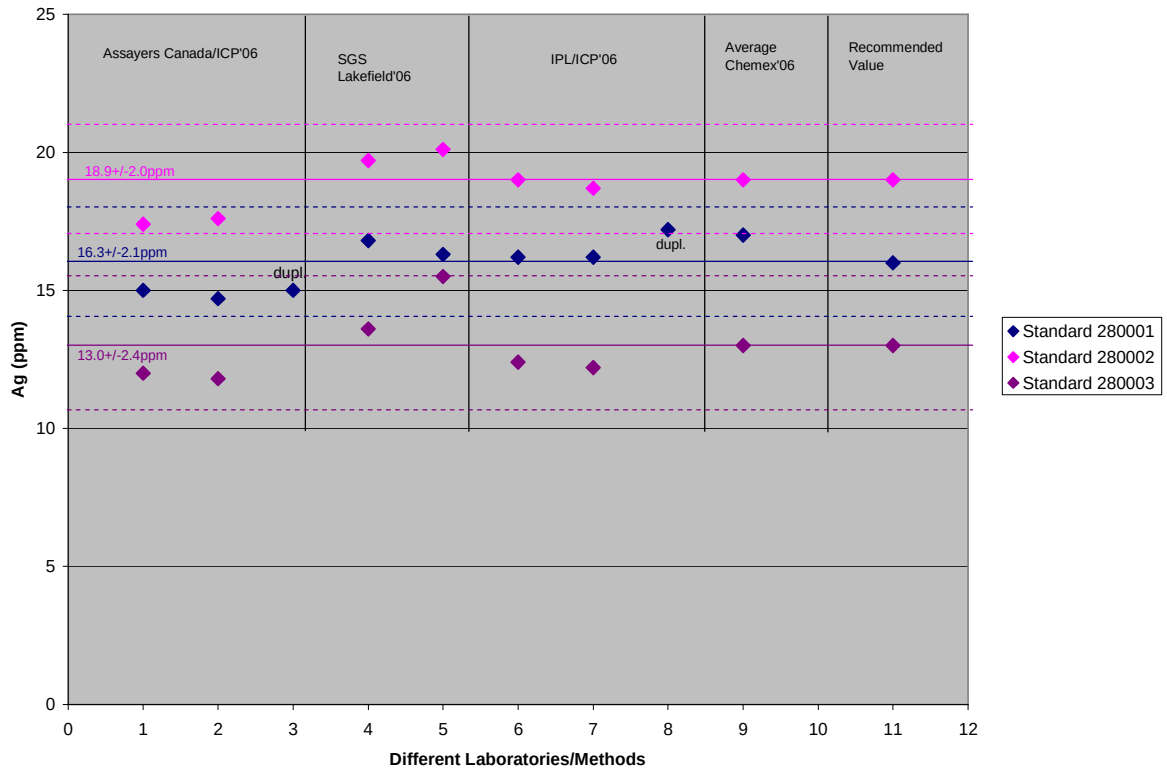


Chart 3.3: 2005 Lab Comparison for Ag (Not tested in 2005)

Table 3.3: Ag (ppm) values by different Labs and Methods

WKM Internal Standard Id	Assayers Canada	IPL	SGS	Average Chemex	Average	STDEV
	ICP'06	ICP'06	ICP'06	ICP'06		
280001a	15	16	17	17	16	1
280001b	15	16	16	17	16	1
280001 dupl.	15	17		17	16	1
280002a	17	19	20	19	19	1
280002b	18	19	20	19	19	1
280003a	12	12	14	13	13	1
280003b	12	12	16	13	13	2

Recommended values are as follows:

WKM SRM1 Ag 16 g/t +/-2g/t

WKM SRM2 Ag 19g/t +/-2g/t

WKM SRM 3 Ag 13g/t +/-2g/t

4.0 Analytical Accuracy: 2006 results for Standards

For the 2006 Diamond Drilling QA/QC Program Western Keltic used their internal standards as well as Certified Reference Materials (three different standards) from Canadian Resources Laboratories. Individual reference standard specification notes for the three CDN Resource Laboratories may be found within the appendix and results for all the standards are found on Tables 4.1 to 4.6 (appendix). Note that CDN High and CDN Low samples are reference samples CDN-HLHC and CDN-HLLC. Two standards (one Western Keltic and one Canadian Resources Laboratories) were inserted every 10 to 20 samples. The results for Cu standards can be seen in Chart 4.1, for Zn in Chart 4.2, for Ag in Chart 4.3 and 4.4, for Au in Chart 4.5 and for S in Chart 4.6. The Cu, Zn, Ag, Au and S values are plotted against sample submission batches. Data tables are attached in the Appendix (Tables 4.1 to 4.6).

Standards generally plot within expected ranges indicating acceptable laboratory control on instrumentation drift. Exceptions to this include a Zn outlier in Chart 4.2, within sample submission batch 1, for the standard 280003. The cause for this outlier is uncertain however, it is most likely standard 280002 that was mislabeled as 280003. In Charts 4.3 and 4.4 the Ag values have a wider range than the Cu and Zn standards but still fall within recommended values (at two standard deviations). The wider range of silver analyses may be due to iron interferences with the ICP analysis, or to the natural Ag variability in the standard. The silver range of the certified Cdn High (HLHC) $\pm 8.6\text{g/t}$ Ag, for the Cdn Low (HLLC) $\pm 6.7\text{g/t}$ Ag and for the FCM3 it is $\pm 3.3\text{g/t}$. Examination of the 'standards' plots, indicate that the values for the standards demonstrate reasonable precision (reproducibility) and accuracy by the analytical laboratory.

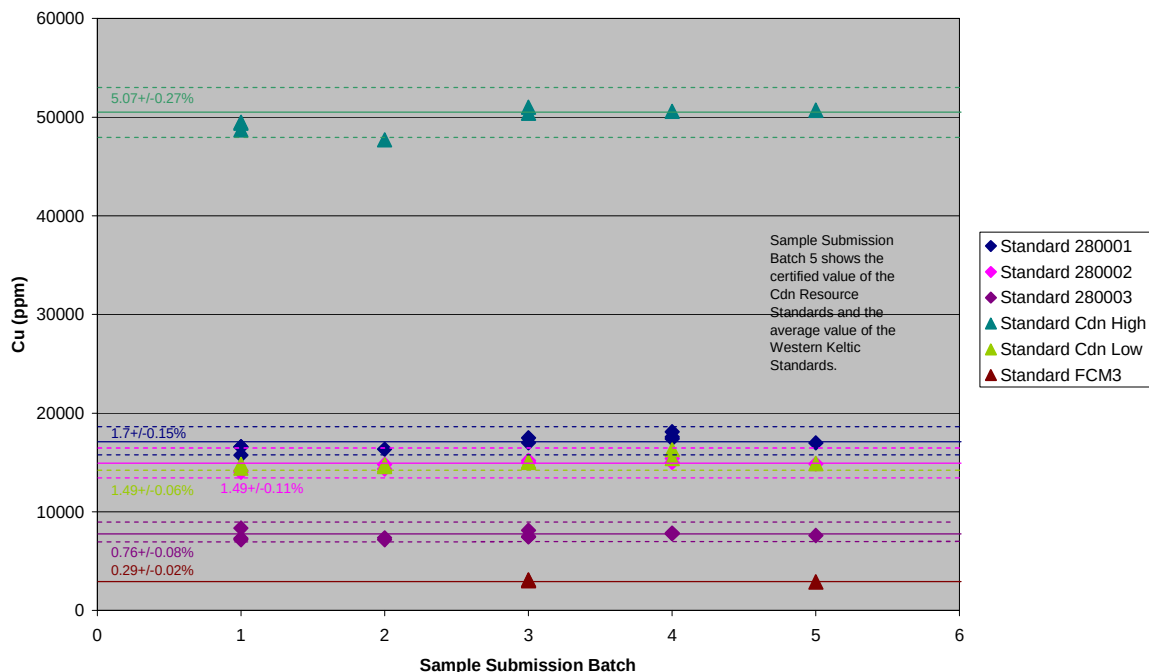


Chart 4.1: Results for Cu Standard

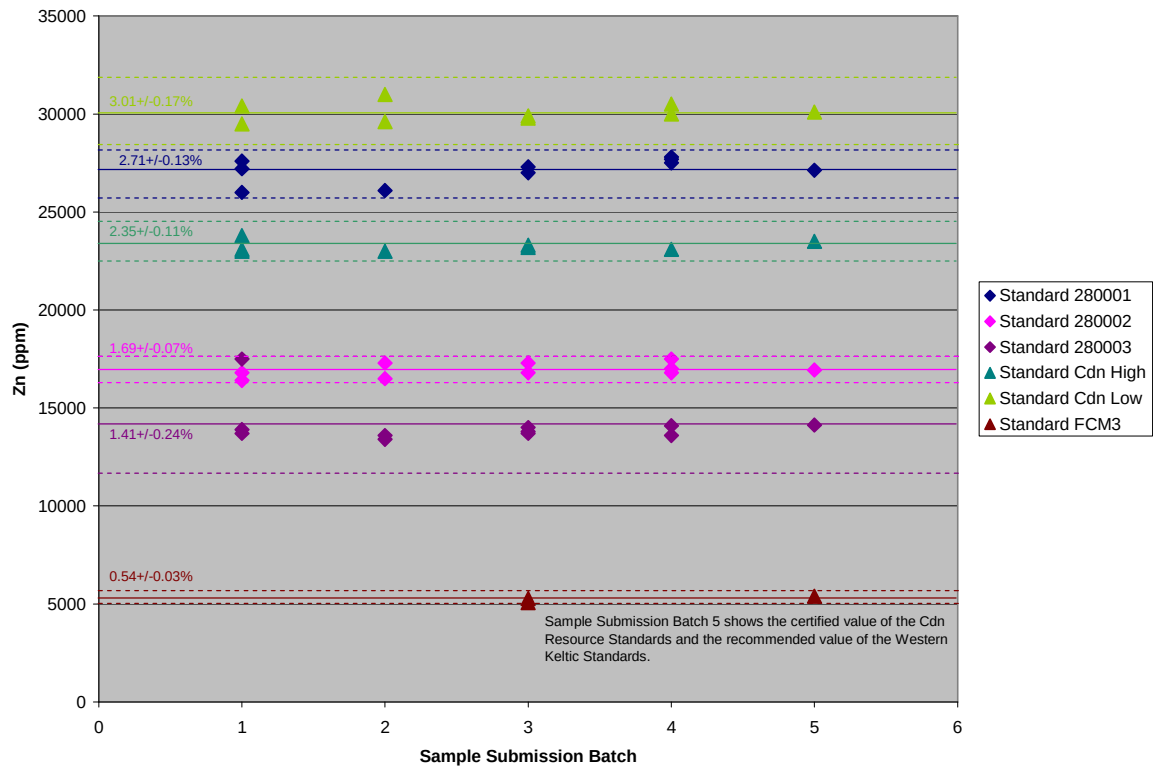


Chart 4.2: Results for Zn Standards

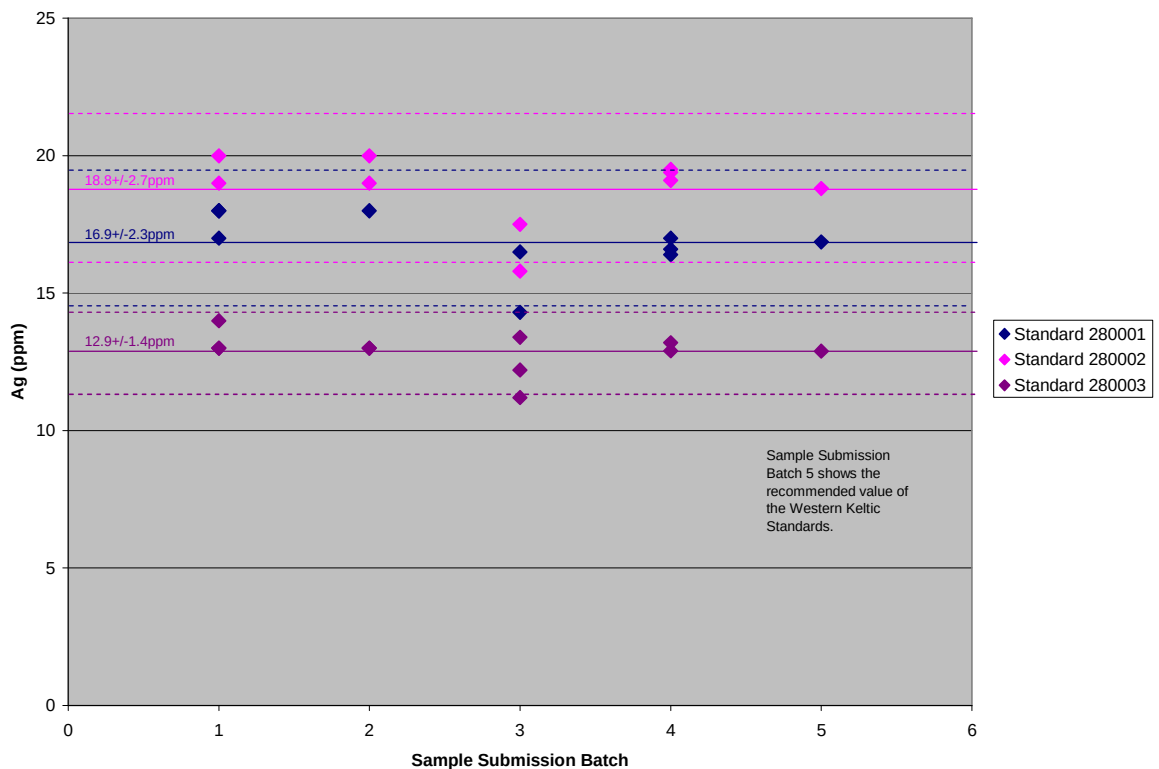


Chart 4.3: Results for Western Keltic Ag Standards

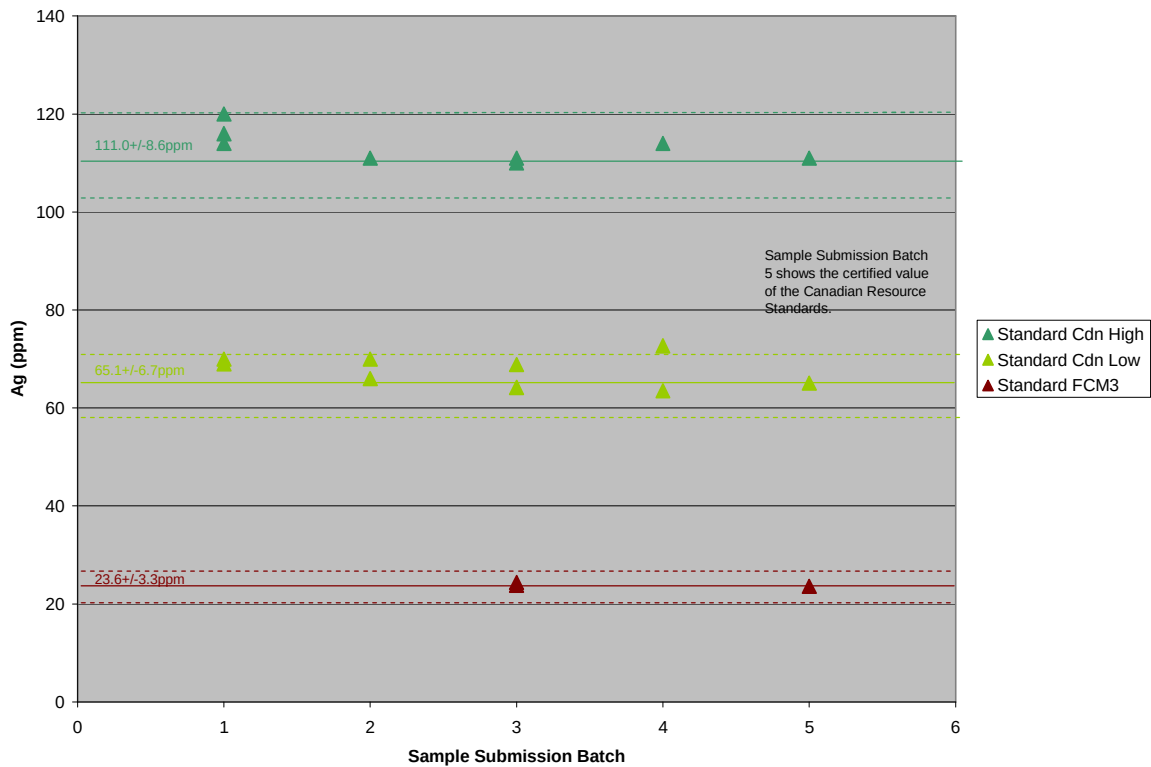


Chart 4.4: Results for Canadian Resource Labs Ag Standards

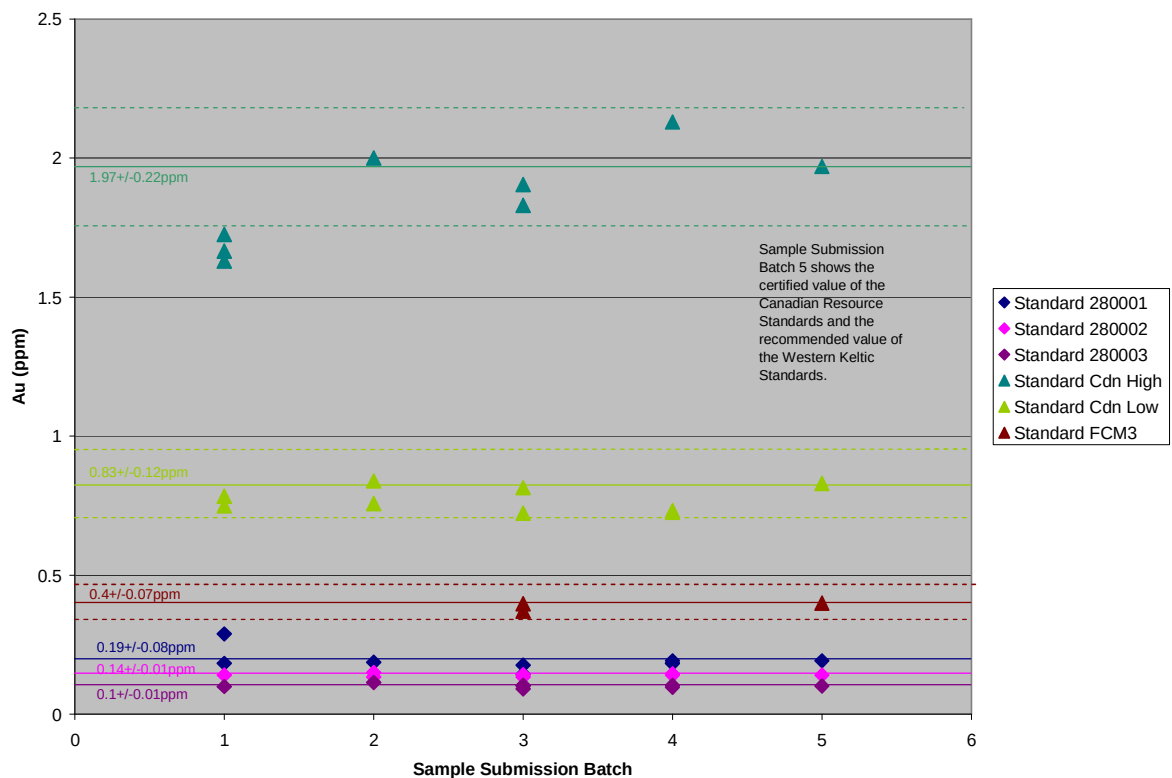


Chart 4.5: Results for Au Standards

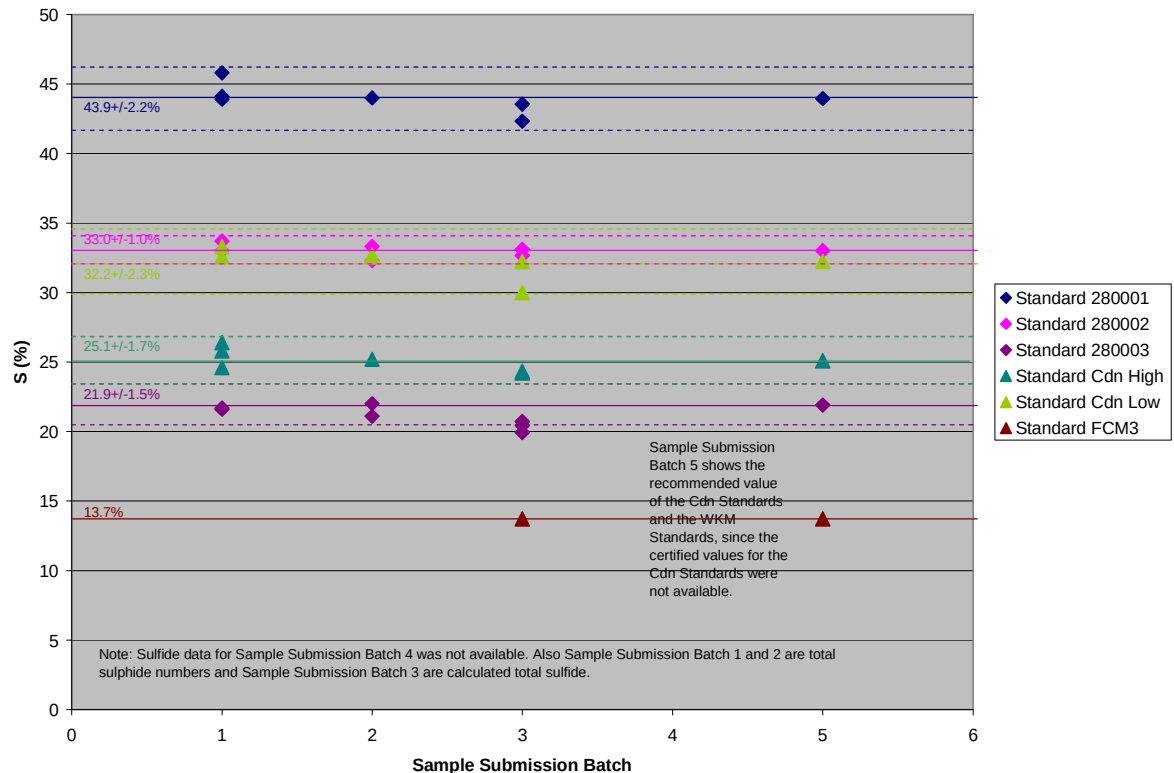


Chart 4.6: Results for S Standards

5. Quality Control

5.1 Results for 2006 Blanks

Blanks were inserted in the drill sample stream, at a rate of approximately 1 per 10 to 20 samples from the Kutcho Creek Project in 2006. These samples help to monitor the presence of unacceptably high levels of background concentrations caused by cross-over contamination (dust, boil-overs, etc); incomplete lab-ware cleaning; or, in circumstances where below-detection results are infrequent, mis-ordering of samples.

Blanks are samples of material essentially devoid of the element undergoing analysis. This material is then subjected to the same sample preparation steps undergone by a project sample. Although the mineralogy of the blank should resemble that of the material being routinely analyzed (*e.g.* barren drill cuttings) (S. Long, 2003), this is not possible for massive sulphide mineralization. Blanks used at Kutcho during the 2006 drill program consisted of crushed black limestone and pink rhyolite obtained from a Landscaping supply company. The results for the Limestone and Rhyolite blanks can be seen in Chart 5.1. The charts were created by plotting the Cu and Zn values of the sample prior to the blank against the blank sample value for the same element. Data tables are attached in the Appendix (Tables 5.1.1 to 5.1.4).

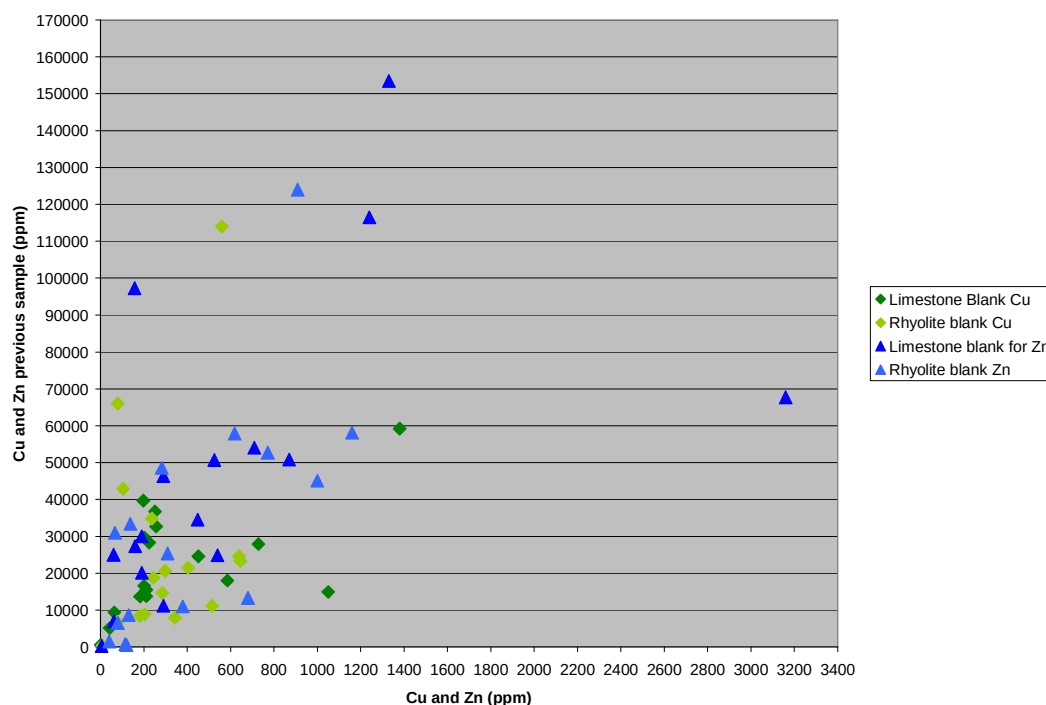


Chart 5.1.1: Blank results for Cu and Zn (Data located in Table 5.1.1 to 5.1.4)

A trend is noted in Chart 5.1: the higher the Cu and Zn value of the previous sample the higher the blank sample's Cu and Zn value. This trend is strongly suggestive of crossover contamination, likely due to insufficient cleaning between laboratory crushing and grinding runs. The soft matrix of massive sulphide mineralization does lend itself to "sticking to" the grinding equipment and compressed air cleaning may not be sufficient to remove all particles. Chemex Labs notes that it has an allowance of 0.5% for grinding and 0.5% for other areas of the analytical process, or a total of 1%. A significant number of the samples (60%) exceed the 1% threshold with a few (20%) even exceeding the 2% threshold.

The contamination, however, is minimal relative to the high grade nature of the zones being sampled. The mean of the blanks for copper is 0.031% Cu and the median 0.023% Cu, for a set of previous samples with a copper mean of 2.398% Cu and median 1.870% Cu. For zinc, the mean value of the blanks is 0.054% Zn and median 0.030% Zn, for a previous Zn sample set with a 3.666% Zn mean and 2.780% Zn median value. The cross sample contamination is less than the expected natural variability experienced through multiple aliquot analysis of the same sample. This can be seen by examining the CDN Resource Laboratories Ltd ore reference high grade copper standard CDN-HLHC specification sheet (Appendix). The within-lab variability of 12 individual labs assaying 10 sample aliquots ranges between 0.03% to 0.36% Cu and 0.03% to 0.17% Zn. The effect of the cross sample contamination on a subsequent high grade sample is within analytical variability and would have no effect on ore boundaries or average grades. This is true even on the sample following the last "high grade" sample as the contamination would not cause that "shoulder sample" to be included in the resource calculation.

It is concluded that while lab procedures may result in minor cross-contamination the cost and productivity loss incurred from running barren material (quartz gravel) through the grinding circuit after every sample can not be justified.

5.2 Analytical Variation with Time: Combined 2005 and 2006 WKM Standards and ALS Chemex Internal Standards

Monitoring laboratory instrumentation drift, which can affect consistency of results over a field season, is a standard part of any QA/QC study. Drift is a change in laboratory results over time for a given element or elements, and is usually tracked by resubmitting a known standard with each sample batch submitted. The drift can be caused by minor changes in laboratory procedures, or instrumentation calibration. Laboratories regularly track drift as part of internal QA/QC, but an external check by laboratory clients is a useful exercise. To that end, a plot of WKM and Chemex Internal Standards is shown in Charts 5.2.1 to 5.2.5, with data tables for the standards presented in the Appendix (Table 5.2.1 to Table 5.2.10).

The charts below contain plots of Cu, Zn, Ag, Au and S standards per sample submission batch for the years 2005 and 2006. While the track for 2005 is fairly stable, 2006 is more variable, with a rising drift for Cu and descending drift for Ag, especially for the WKM standards. The high iron nature of the WKM standards may cause interference for some elements. There is only one outlier in chart 5.2.2 within sample submission batch 7 for Western Keltic standard 280003 and the cause of this anomaly is not immediately clear however, it is most likely standard 280002 that was mislabeled as 280003.

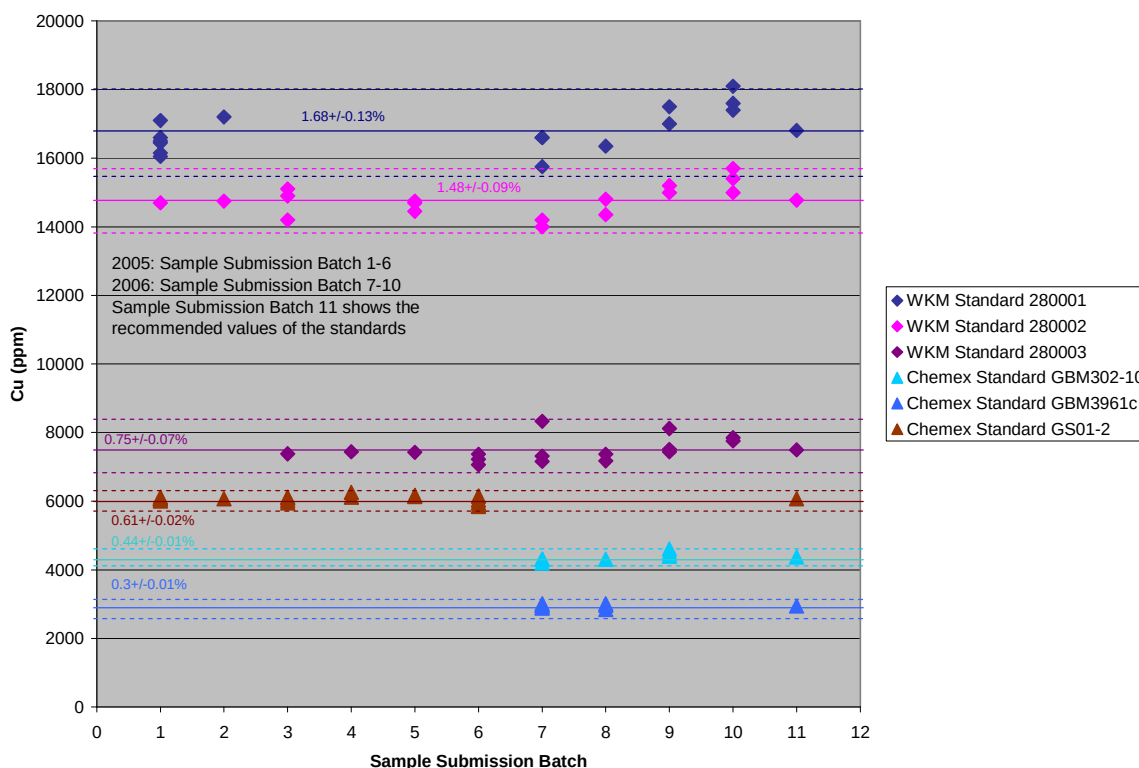


Chart 5.2.1: Results for Cu Standard

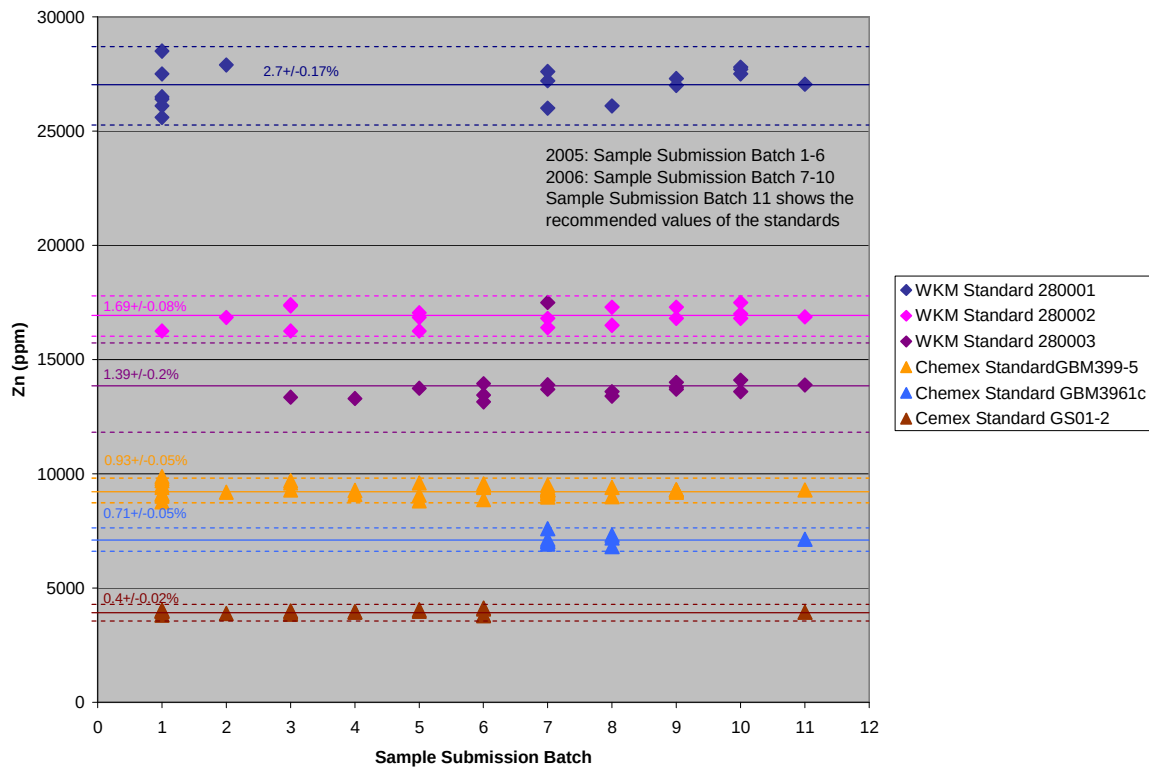


Chart 5.2.2: Results for Zn Standards

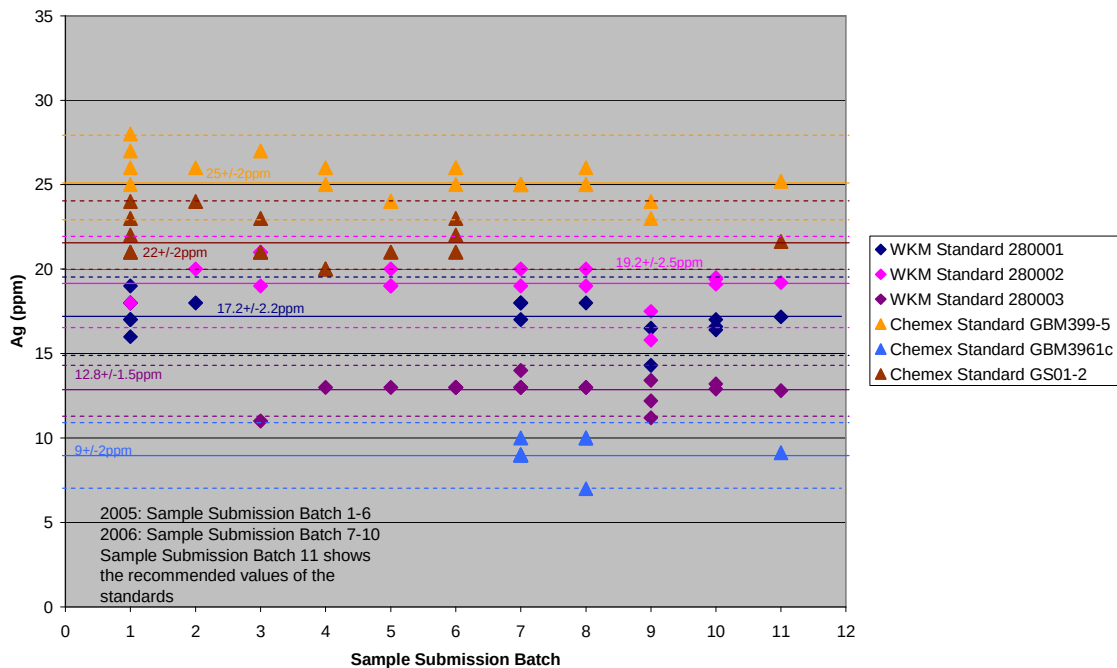


Chart 5.2.3: Results for Ag Standards

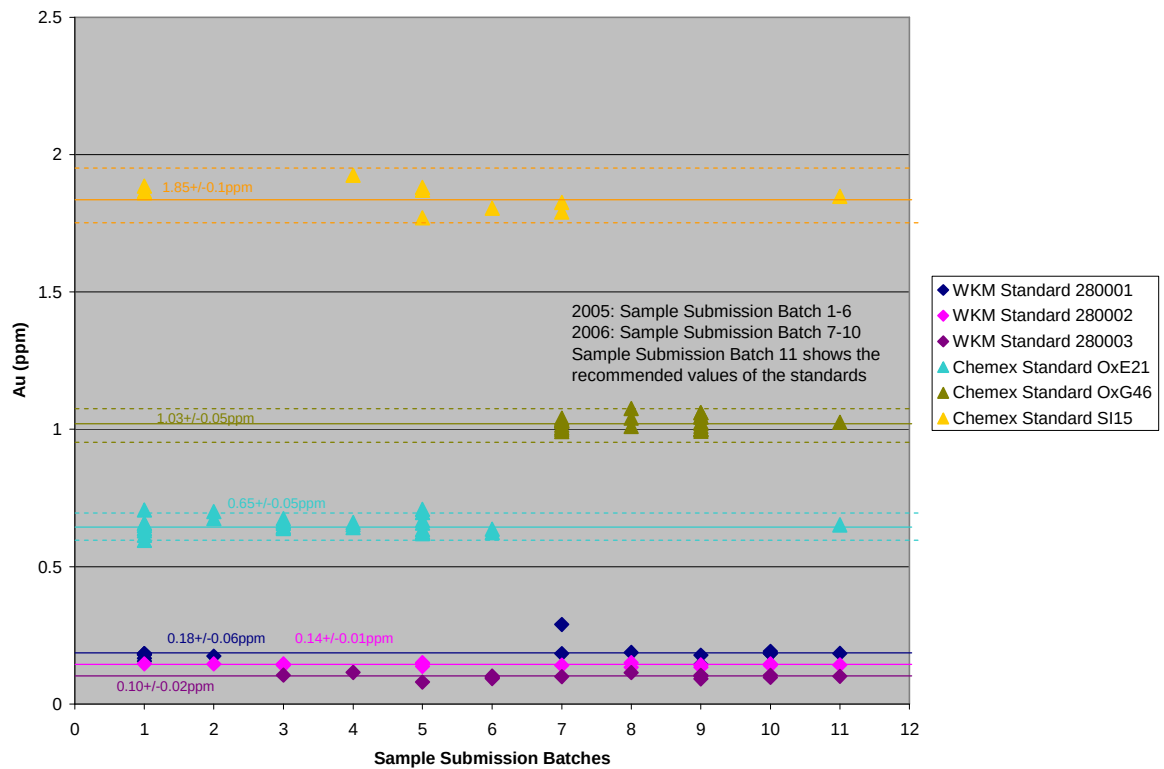


Chart 5.2.4: Results for Au Standards

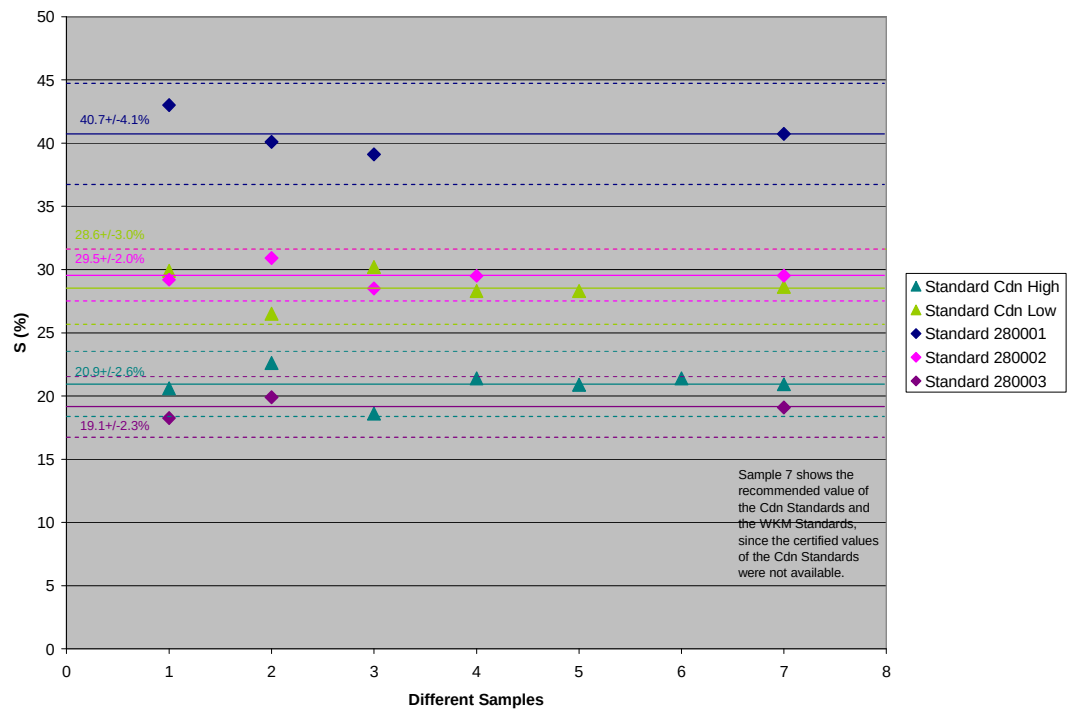


Chart 5.2.5: Results for S Standards

6. Validation of SML Historical Analyses

In order to verify historical drill results for the SML drill-holes (where original certificates are not available) WKM conducted re-sampling of mineralized core drilled by Sumitomo during the period 1974 to 1982. There are 129 SML drill-holes in the project area of which approximately 105 are within the Main deposit. SML core had been previously re-sampled and new analyses were very similar to historical analyses, however no CRM or standards were included during that investigation so as per AMEC recommendations a second re-sampling program was undertaken. A total of 137 samples were collected from 18 drill holes spanning the range from K001 to K094, in order to provide corroboration for the historical data. SML core was sawn and all of the remaining half-core was collected for re-analysis. The core was not quartered due to relatively poor core and core-box condition after 30 years in the elements.

The historical data compared to the new assays is presented in Charts 6.1 to 6.5 for Cu, Zn, Ag, Au and S respectively. See the appendix (Table 6-1) for the supporting data table.

Slope and intercept have been calculated using the “Reduction to Major Axis” method, which does not introduce underlying assumptions about which data set is correct nor does it assume any dependence or independence between the sample pairs as does a linear regression. This method also allows for differences in population variance between the two data sets.

The formula used is: $y=mx+b$, with x and y being known numbers (the mean values of the data set: $x=1.55$, $y=1.58$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEV_y=1.607331$, $STDEV_x=1.71899$, $m=0.9350$) and $b=y-mx=0.1285$.

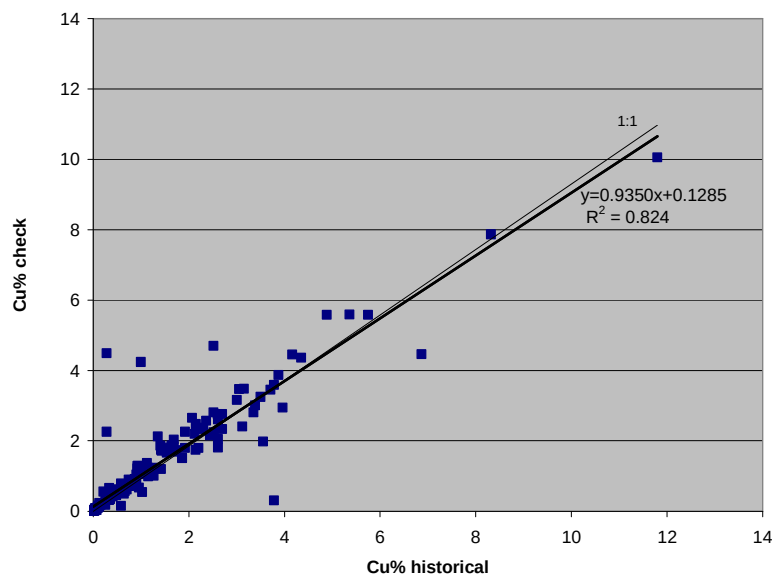


Chart 6.1: Cu comparison

The chart above and the charts below show that the two data sets correlate well for Cu and Zn. The reproducibility of historical assays is limited by several factors. Split core shifting in the boxes through handling, and footage markers not always being legible led to difficulties in collecting identical sample intervals. As well, improved lab techniques may yield different results especially where lower detection limits are now available. Given these challenges, however, the correspondence between data sets is remarkably good. The few outliers in chart 6.1, 6.2 and 6.3 can be explained by either normal analytical variability or the noted challenges in obtaining ‘true’ sample duplicates.

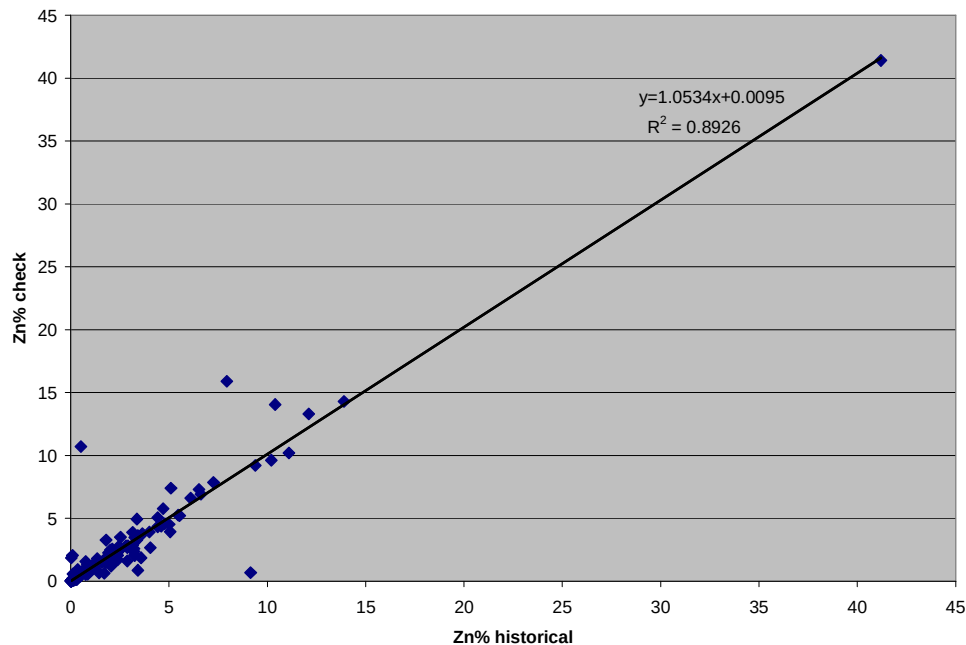


Chart 6.2: Zn comparison

The formula used for Zn is: $y = mx + b$, with x and y being known numbers (the mean values of the data set: $x = 2.48$, $y = 2.65$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEV_y = 4.544127$, $STDEV_x = 4.31382$, $m = 1.0534$) and $b = y - mx = 0.0095$.

With only a few exceptions, the agreement for Zn results between historic and new data sets is very good. The most dramatic variances are from drill-hole KT001, where both the near 10% - sub 1% Zn points could, in fact, be the same sample interval, but were incorrectly designated during re-sampling. The other two outliers cannot easily be explained, but may be due to incomplete sample intervals available for re-sampling leading to a higher grade bias induced by a few high grade Zn core pieces within the remaining sampled core.

The chart for Ag (below) contains more outliers and a wider spread in the correlation cluster. At least two outliers are related to the KT001 sampling error noted for Zn. Others could be explained by an Ag ‘nugget’ effect where silver rich core pieces were or were not included in the historic and re-sampled cores. The overall correlation between historic and re-sampled core is reasonable with a slight increase in overall grade due to higher quality analytical methods and equipment available today.

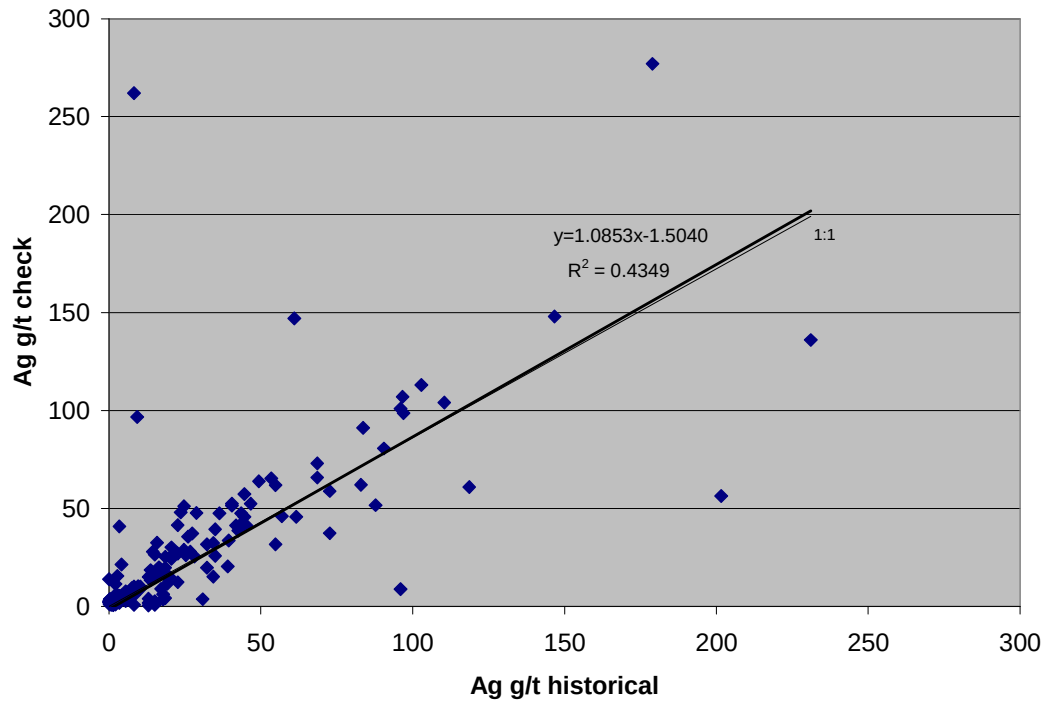


Chart 6.3: Ag comparison

The formula used for Ag is: $y = mx + b$, with x and y being known numbers (the mean values of the data set: $x = 31.38$, $y = 32.55$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEVy = 42.71559$, $STDEVx = 39.36$, $m = 1.0853$) and $b = y - mx = -1.5040$.

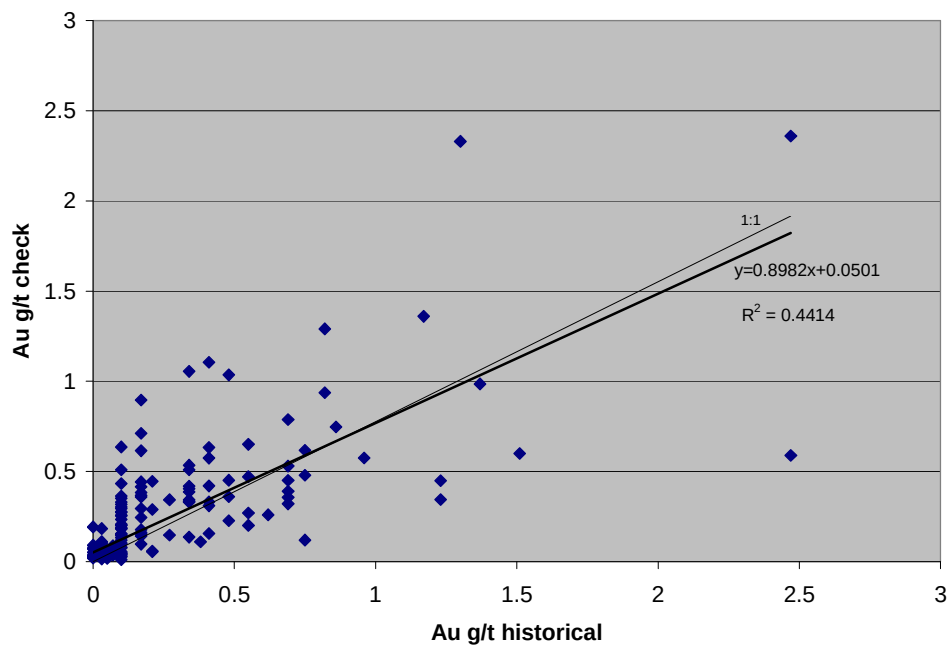


Chart 6.4: Au comparison

The formula used for Au is: $y=mx+b$, with x and y being known numbers (the mean values of the data set: $x=0.31$, $y=0.33$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEVy= 0.369982$, $STDEVx= 0.41189$, $m= 0.8982$) and $b=y-mx= 0.0501$.

Correlation of gold results is severely limited by Au detection limits, and historic reporting of gold assays in oz/t to three decimal places. Combined with a predictable nugget effect exacerbated by the small historic core volumes, only a moderate correlation is measured, however, average grades would be approximately equal with slightly higher values for the modern analysis.

The chart below shows that the two data sets correlate well for the S% values. Given the mineralized interval's massive pyrite nature a good correlation would be expected and significant deviations would demonstrate errors in sampling over a non-mineralized interval.

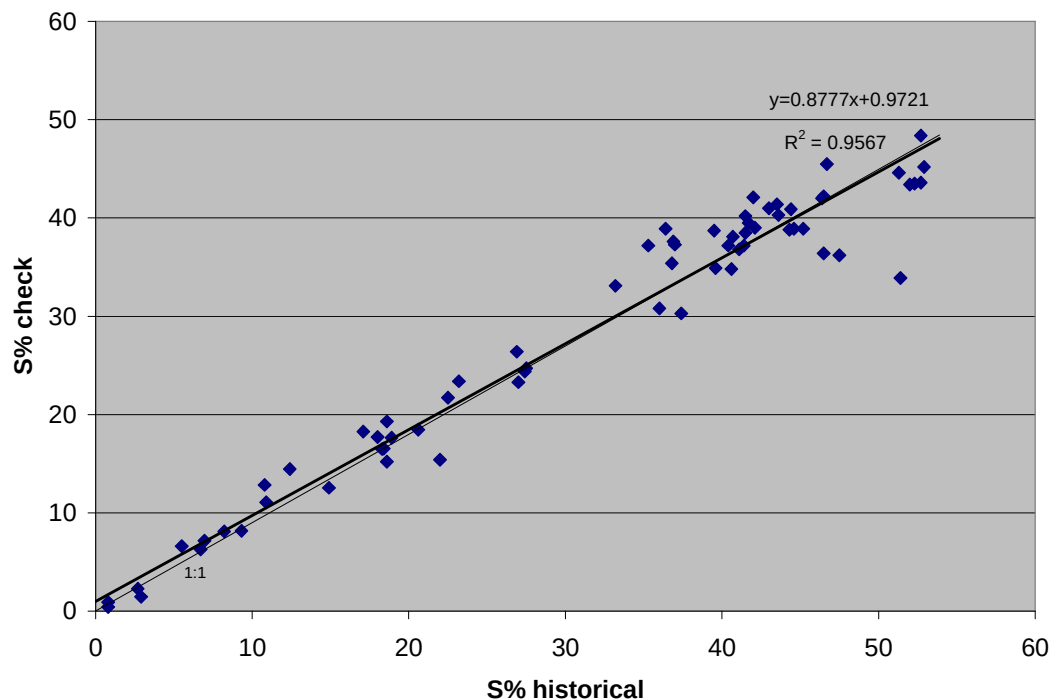


Chart 6.5: S% comparison

The formula used for S is: $y=mx+b$, with x and y being known numbers (the mean values of the data set: $x=31.44$, $y=28.57$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEVy= 13.73373$, $STDEVx= 15.6477$, $m= 0.8777$) and $b=y-mx= 0.9721$.

A separate standard and blank QA/QC study has been done for the historical core intersection resample. Since all the standards were part of one sample submission batch, they are plotted as different samples. The results are shown in Chart 6.6, 6.7, 6.8, 6.9 and 6.10. The data tables are attached in the appendix (Table 6.2 and 6.3).

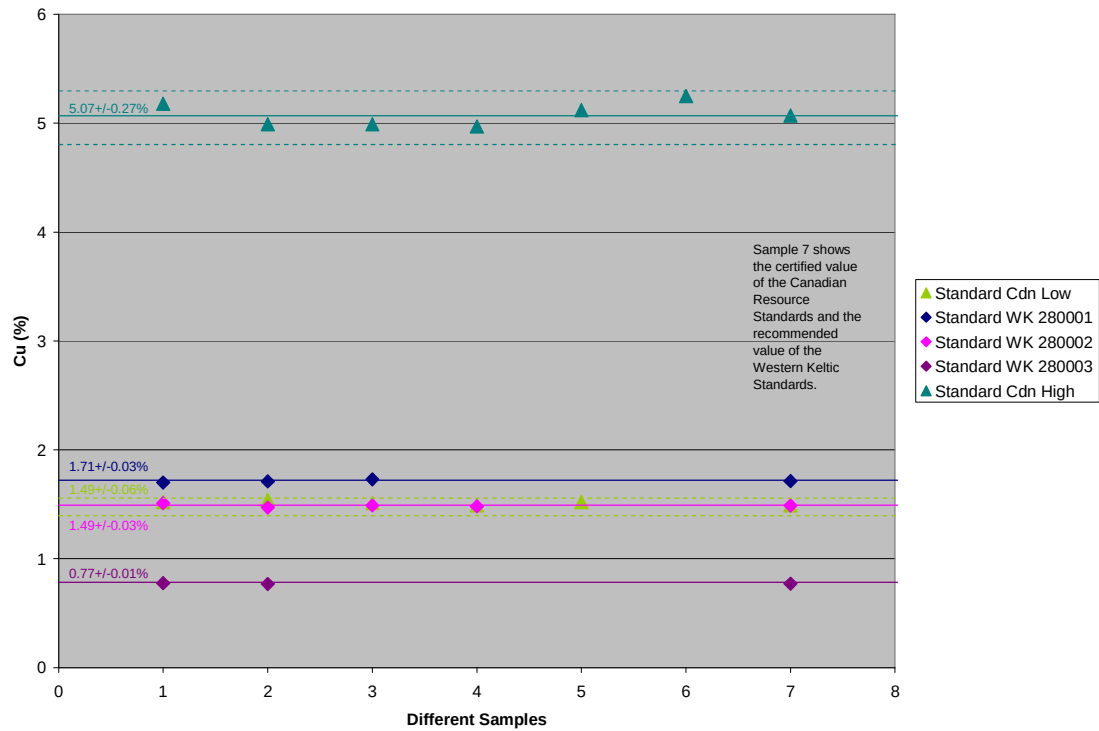


Chart 6.6: Cu standard results for historical core intersection resample

The values for the standards shown in the charts above and below are within recommended ranges and confirm analytical accuracy.

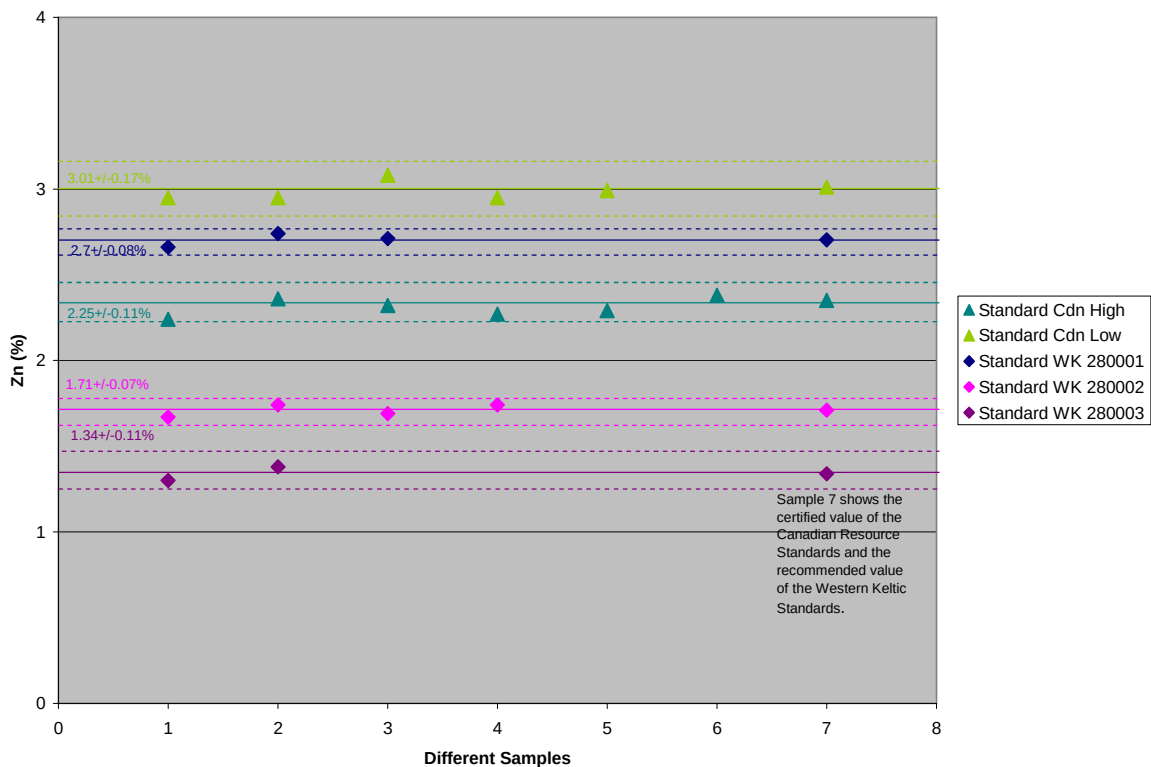


Chart 6.7: Zn standard results for historical core intersection resample

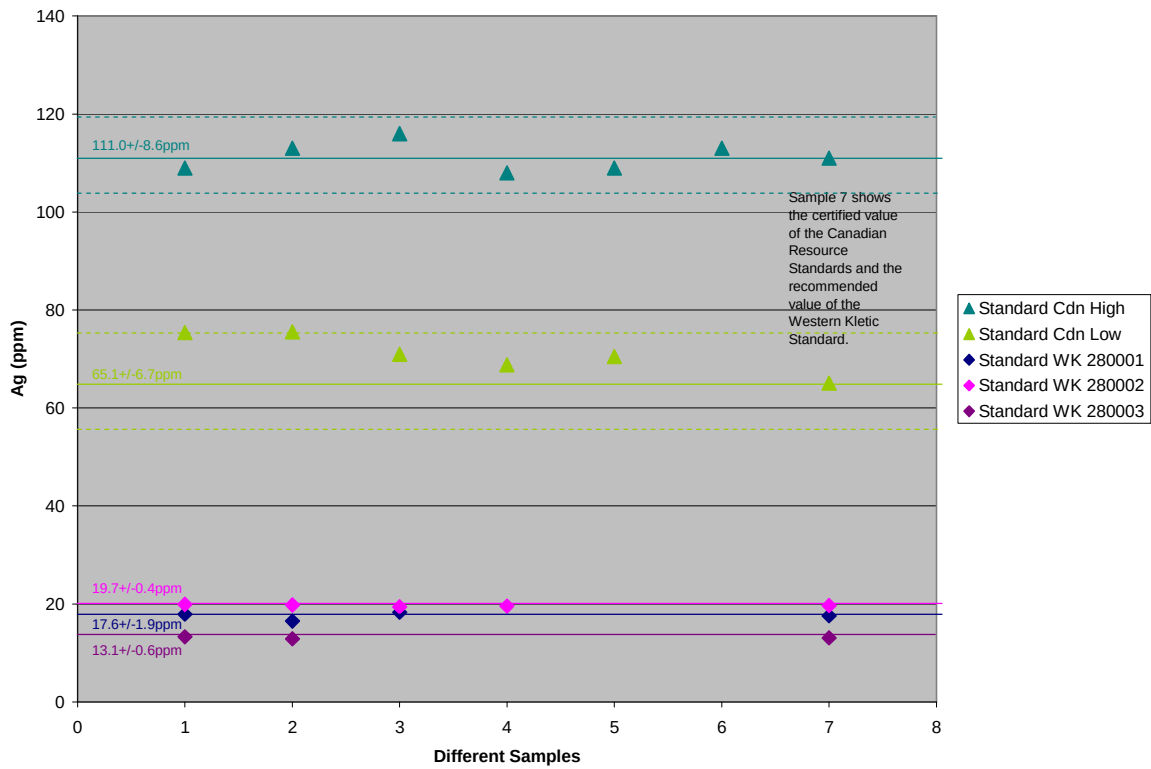


Chart 6.8: Ag standard results for historical core intersection resample

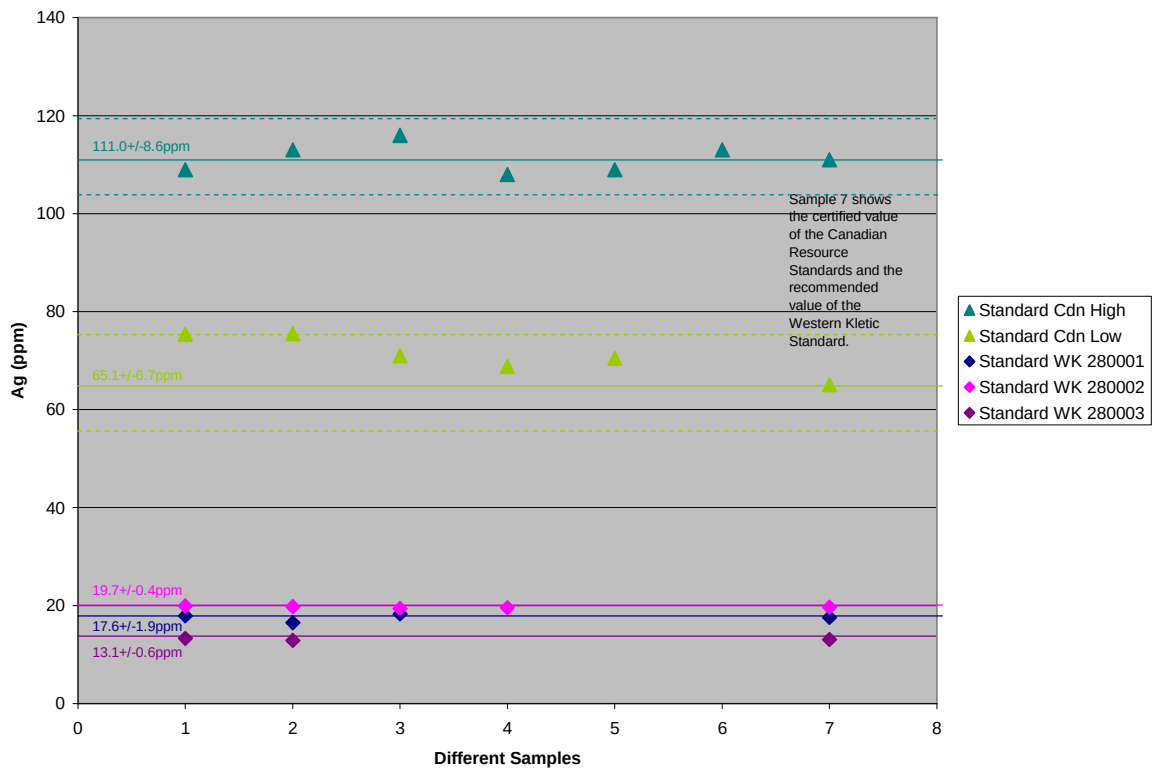


Chart 6.9: Au standard results for historical core intersection resample

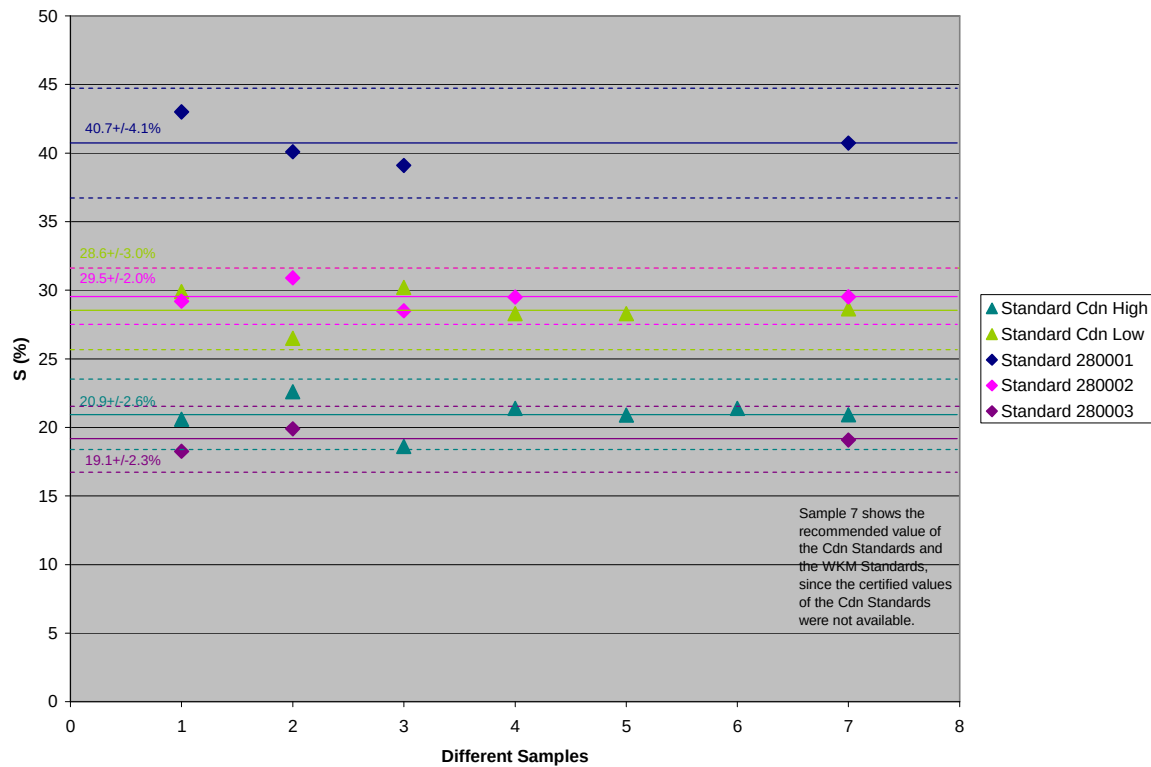


Chart 6.10: S standard results for historical core intersection resample

Blanks for historical core re-sampling consisted of crushed quartz. The Cu and Zn values in the blank samples are plotted against the Cu, and Zn values of the samples preceding the blanks. The results for the blanks can be seen in Chart 6.11 below. Data tables are attached in the Appendix (Table 6.4 and 6.5).

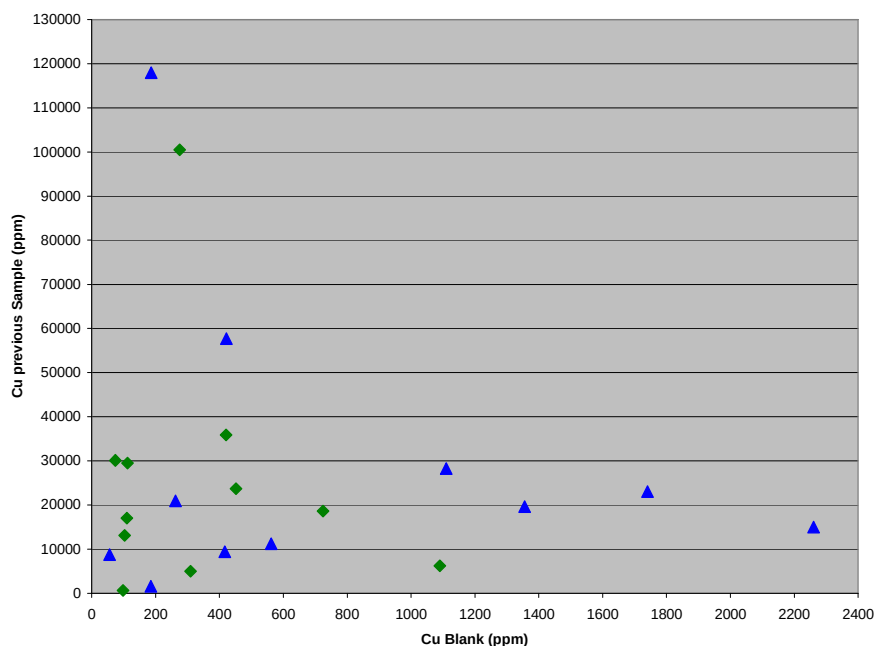


Chart 6.11: Cu and Zn blank results for historical core intersection resample

As with blanks submitted during the 2006 drill sampling program, cross contamination is evident, but the absolute value of that contamination is again within the range of analytical variability for most ore-grade samples. Unequal chart axis ranges exaggerate the cross-contamination effect, and illustrates that Zn is more difficult to keep from affecting the following sample. The results are suggestive of incomplete lab-ware cleaning between sample preparations.

7.0 Study of Variation in Core Sampling

During the 2006 drill program, a large sample shipment was temporarily lost which led to an unusual double sampling of certain drill-hole cores. This resulted in an opportunity to compare and evaluate the assays from two sampling procedures on the same cores. One sample shipment containing samples from holes WK06-74 to 76 and WK06-78 and 79 was delivered to the wrong lab. Due to an overload of samples arriving at the labs the error was not caught until much later. Fearing that samples had been lost during transit, WKM implemented a program to resample the core from the five drill holes. Unbeknownst to WKM the original samples eventually arrived at the lab and were processed as usual, allowing this comparative analysis.

The process of re-sampling varied depending upon whether the initial assay sampling consisted of half core or quartered core, the latter being from holes where metallurgical samples had been collected. The re-sampling consisted of a) retrieving the unprocessed metallurgical samples for holes WK06-75, 78 and 79 and submitting those samples for assaying (half core), and b) retrieving core boxes for holes WK06-74 and 76, transporting them to Smithers where the remaining half-core was quartered, sampled and shipped to the lab by CJL Enterprises. Blanks and standards were inserted into the sample streams at the equivalent breaks. The replacement samples were given equivalent sample numbers as the boxes were already labeled. When the duplication of sample was discovered, the lab added a letter 'A' to the sample number for the originally lost samples.

Assay results from the original and replacement samples were subjected to QA/QC study. As per the regular standards, the "Reduction to Major Axis" method was used for the current analysis, shown in Charts 7.1 to 7.5. The data table is attached in the appendix (Table 7.1). The formula used for Cu is: $y = mx + b$, with x and y being known numbers (the mean values of the data set: $x = 1.29$, $y = 1.25$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEV_y = 1.36$, $STDEV_x = 1.40$, $m = 0.9734$) and $b = y - mx = 0.0787$.

As illustrated in the charts below, the two data sets are nearly identical indicating: that the mineralization is relatively homogeneous throughout the core and a quarter core assay is, in most cases just as valid as a half core assay, and that analytical precision is good. The few Cu outliers may be related to bornite or chalcocite clusters along fractures which would have 'close-range' variability.

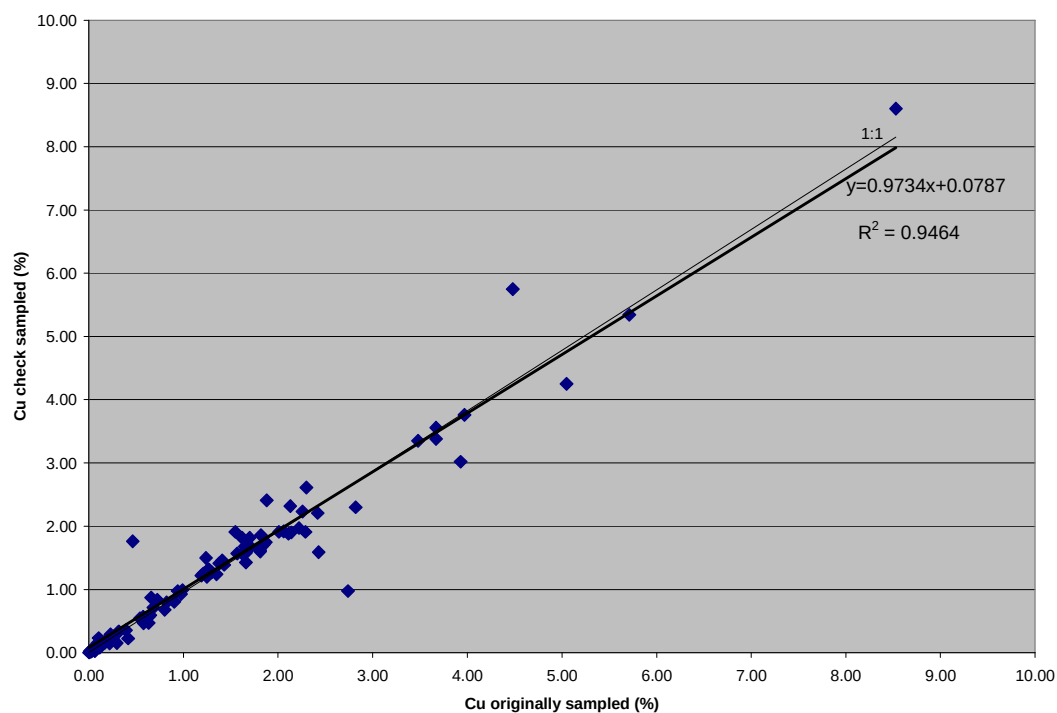


Chart 7.1: Cu comparison

The formula used for Zn is: $y = mx + b$, with x and y being known numbers (the mean values of the data set: $x = 2.06$, $y = 2.03$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEVy = 2.79$, $STDEVx = 2.90$, $m = 0.9628$) and $b = y - mx = 0.0492$.

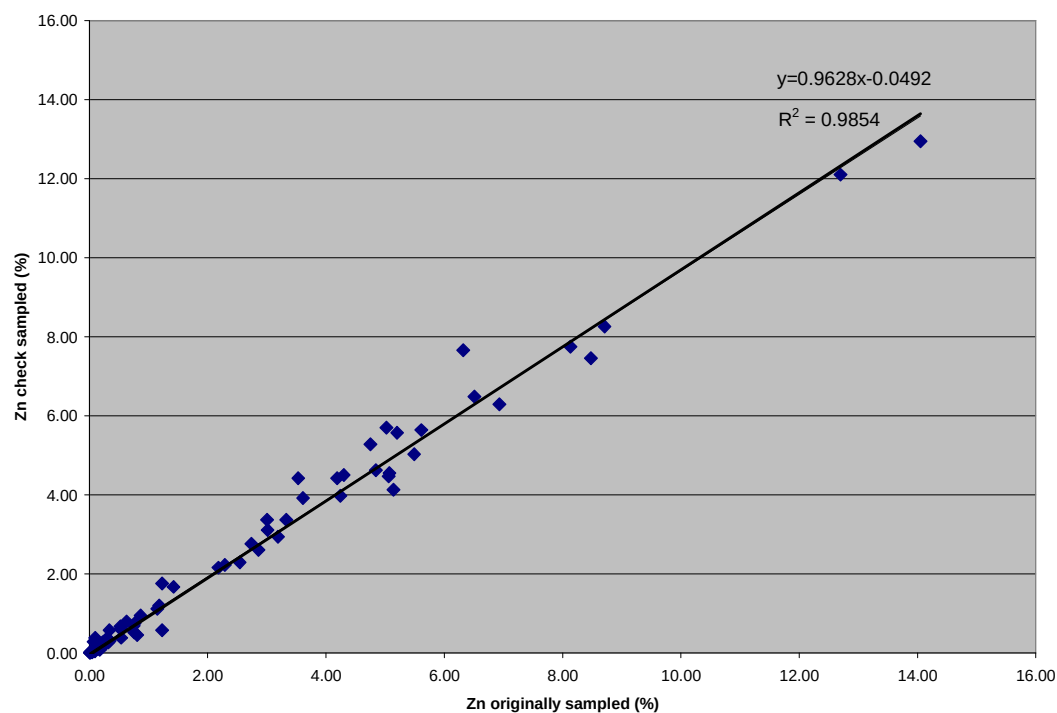


Chart 7.2: Zn comparison

The formula used for Ag is: $y=mx+b$, with x and y being known numbers (the mean values of the data set: $x=18.72$, $y=18.17$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEVy= 24.26$, $STDEVx= 23.81$, $m= 1.0192$) and $b=y-mx= -0.9154$.

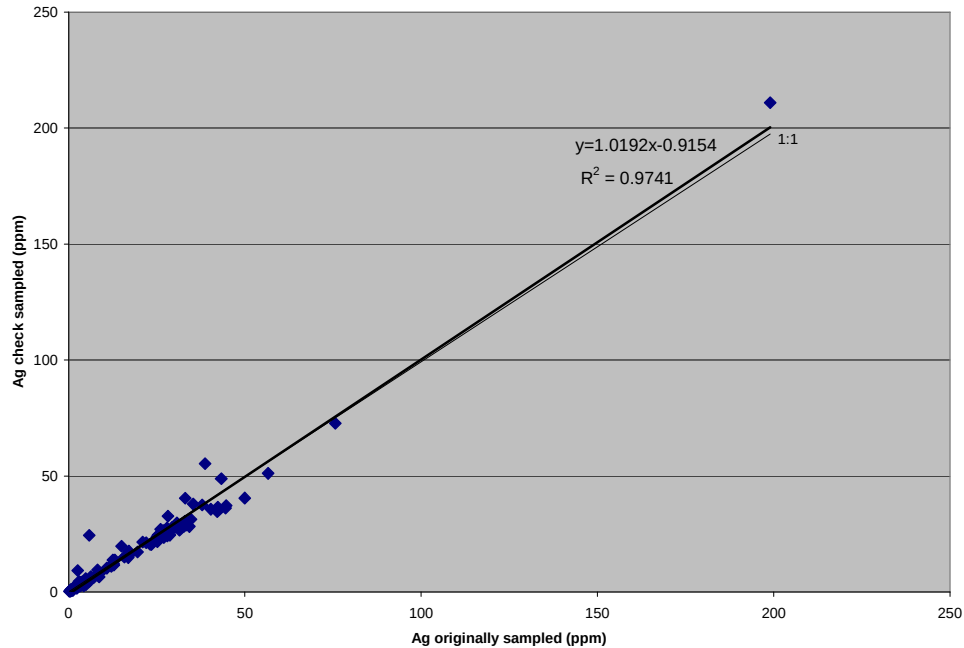


Chart 7.3: Ag comparison

The formula used for Au is: $y=mx+b$, with x and y being known numbers (the mean values of the data set: $x=0.23$, $y=0.25$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEVy= 0.29$, $STDEVx= 0.26$, $m= 1.0886$) and $b=y-mx= -0.0090$.

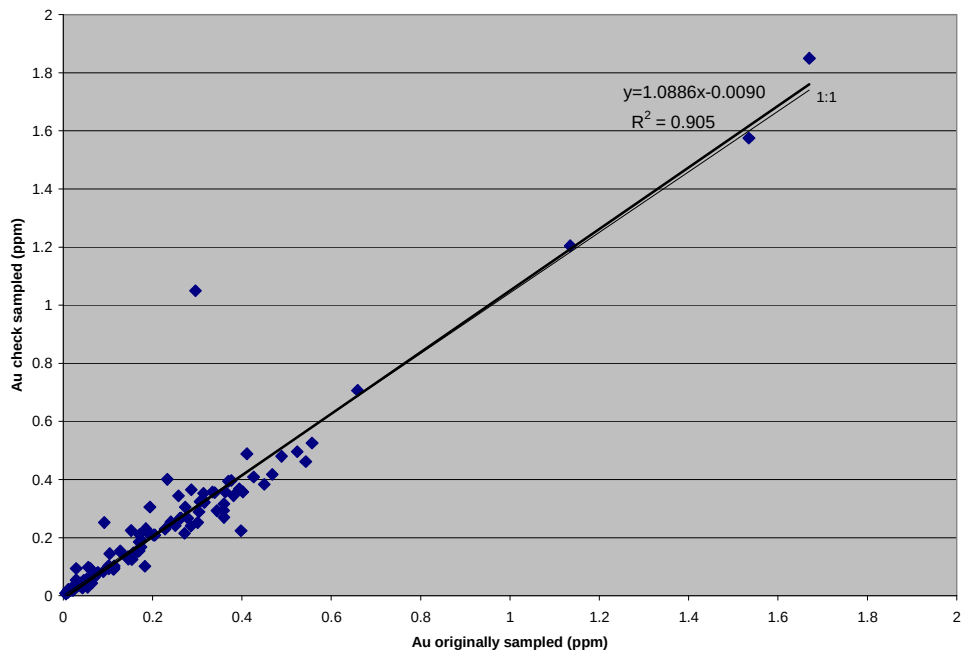


Chart 7.4: Au comparison

The formula used for S is: $y=mx+b$, with x and y being known numbers (the mean values of the data set: $x=4.71$, $y=4.95$), m derived out of the standard deviation of the y data set divided by the standard deviation of the x data set ($STDEV_y= 2.92$, $STDEV_x= 2.77$, $m= 1.0548$) and $b=y-mx= -0.0174$.

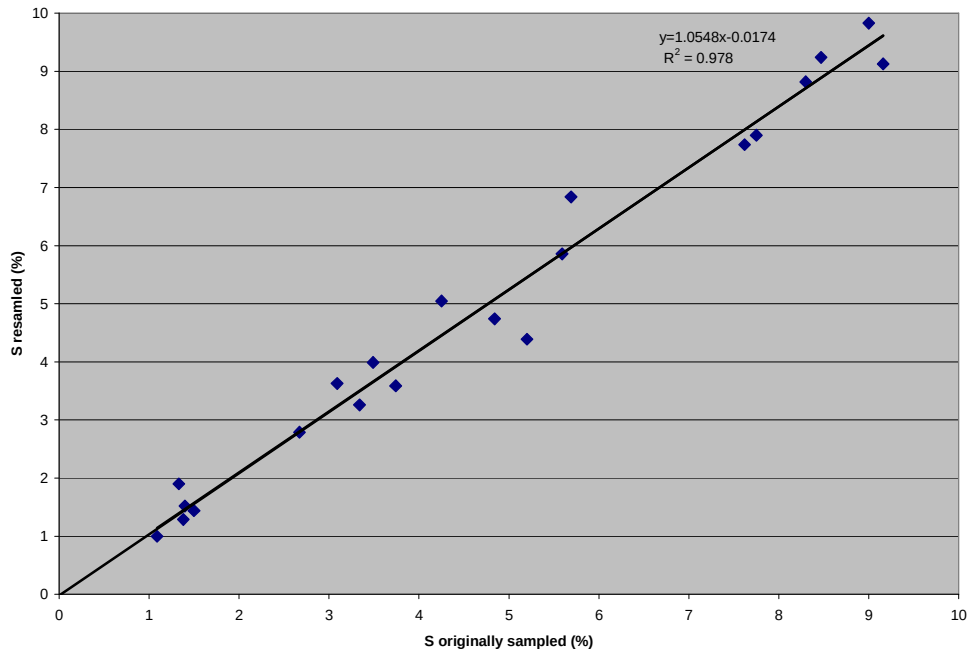


Chart 7.5: S comparison

A comparison of the blanks and standards for the two batches (the original and the resample) was completed as an independent check on cross-contamination. Results for the blanks can be seen in Chart 7.6. Data tables are attached in the Appendix (Table 7.2 and 7.3). The blanks for the original batch consisted of crushed limestone and rhyolite and the blanks for the resample consisted of patio bricks and limestone. The chart 7.6 shows the same trends that have been identified and discussed previously, however, there is a significantly lower level of samples that exceed the 1% threshold which may be a function of better cleaning of the grinding equipment or could be due to a change of hardness of the blank material (patio bricks as opposed to rhyolite)

The results for the standards are shown in Chart 7.7 and 7.8. Data tables are attached in the Appendix (Table 7.4). The variation between the two sets of standards may reflect that the standards were obtained from the 'master bag' at different times and would be more likely to show internal sample variation. The results still fall within the recommended ranges for the standards.

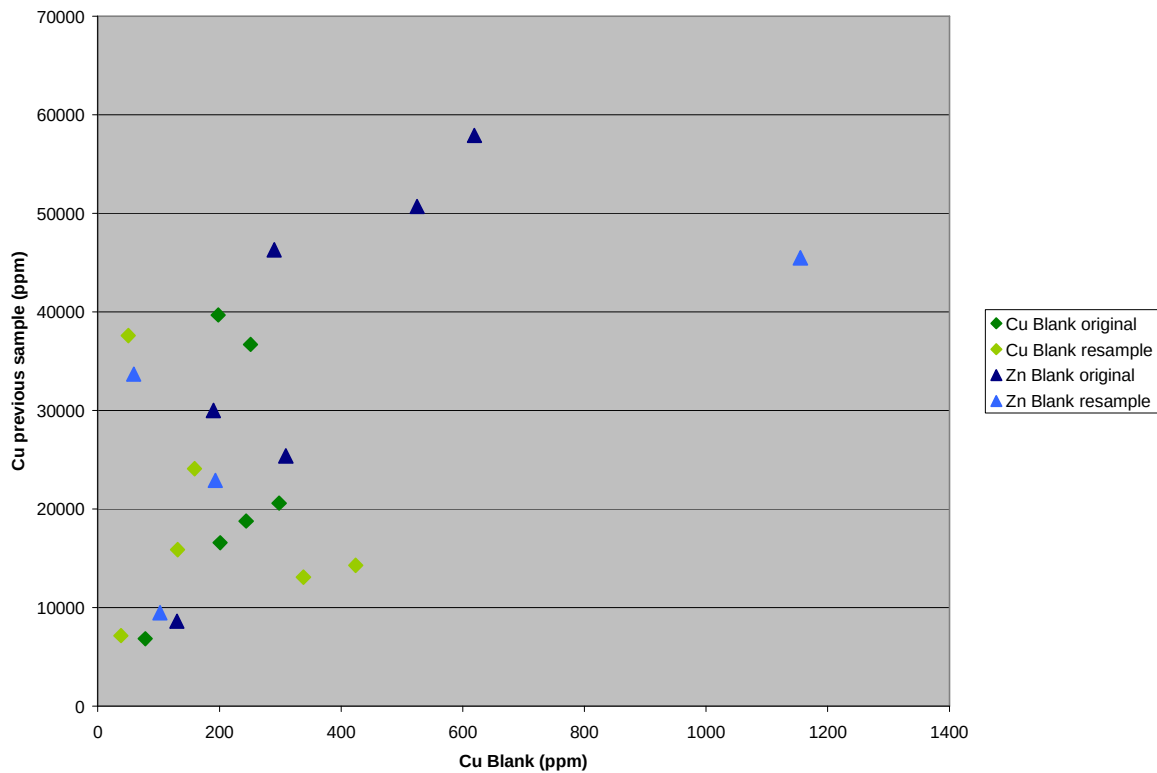


Chart 7.6: Cu and Zn comparison for the original and resample blank results versus the previous samples

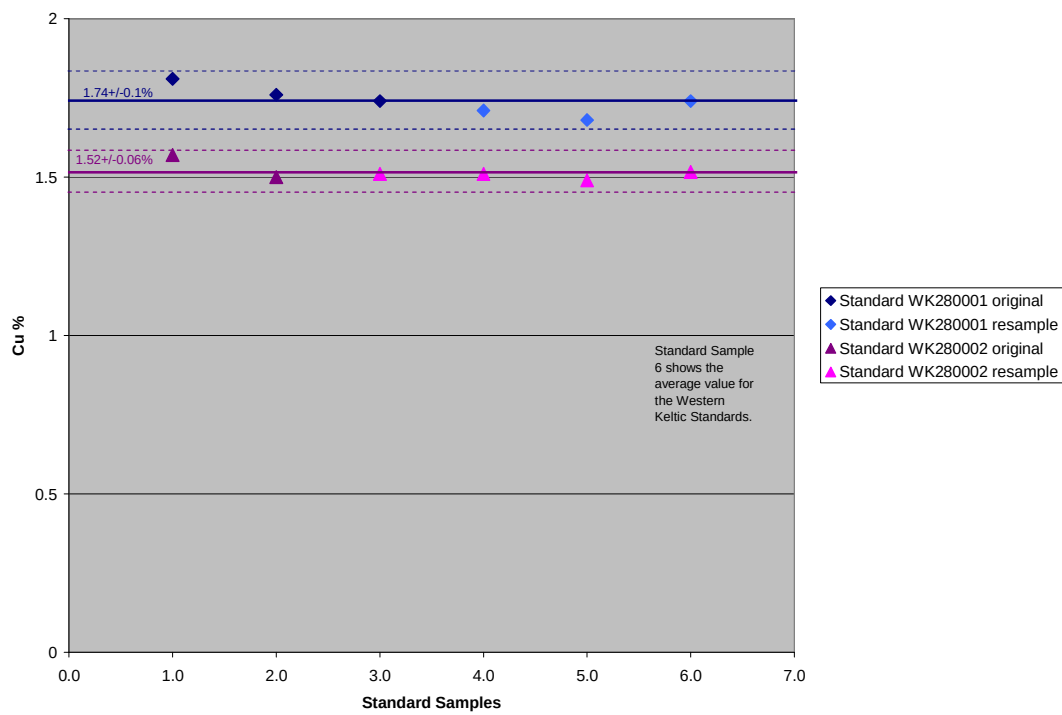


Chart 7.7: Cu values of the original and re-sampled standards

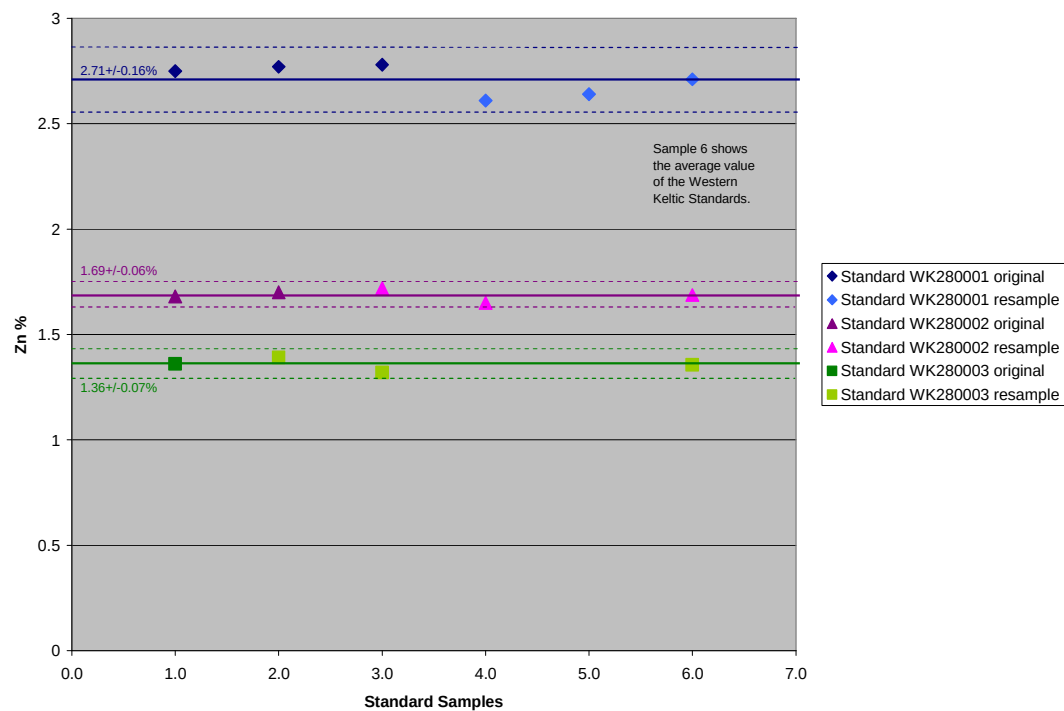


Chart 7.8: Zn values of the original and re-sampled standards

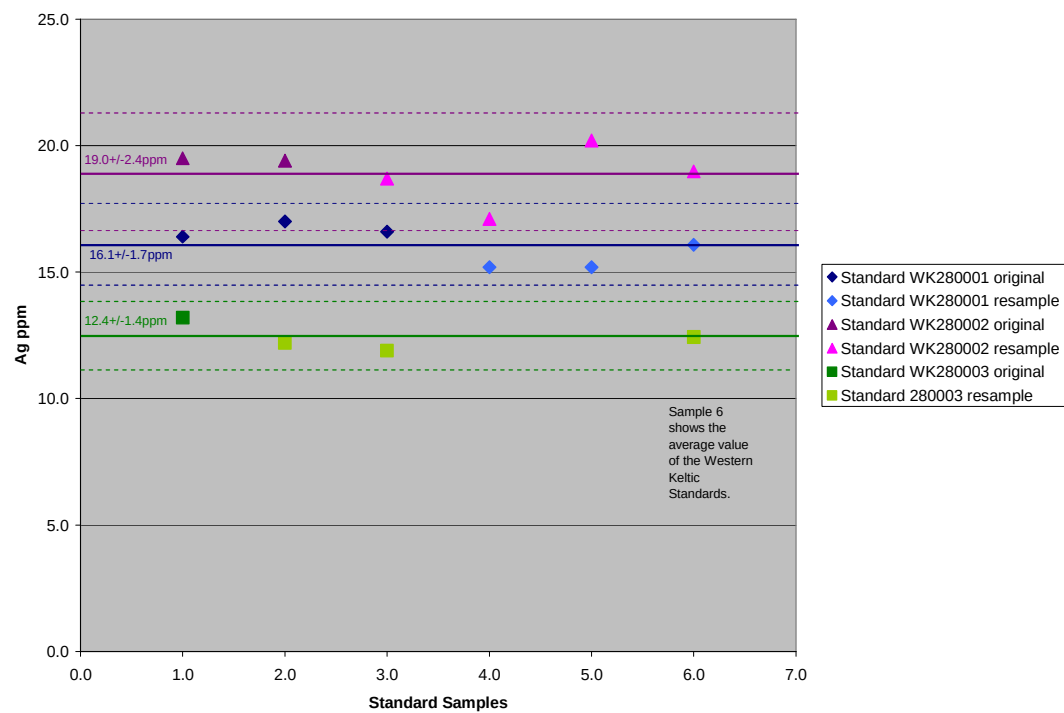


Chart 7.9: Ag values of the original and re-sampled standards

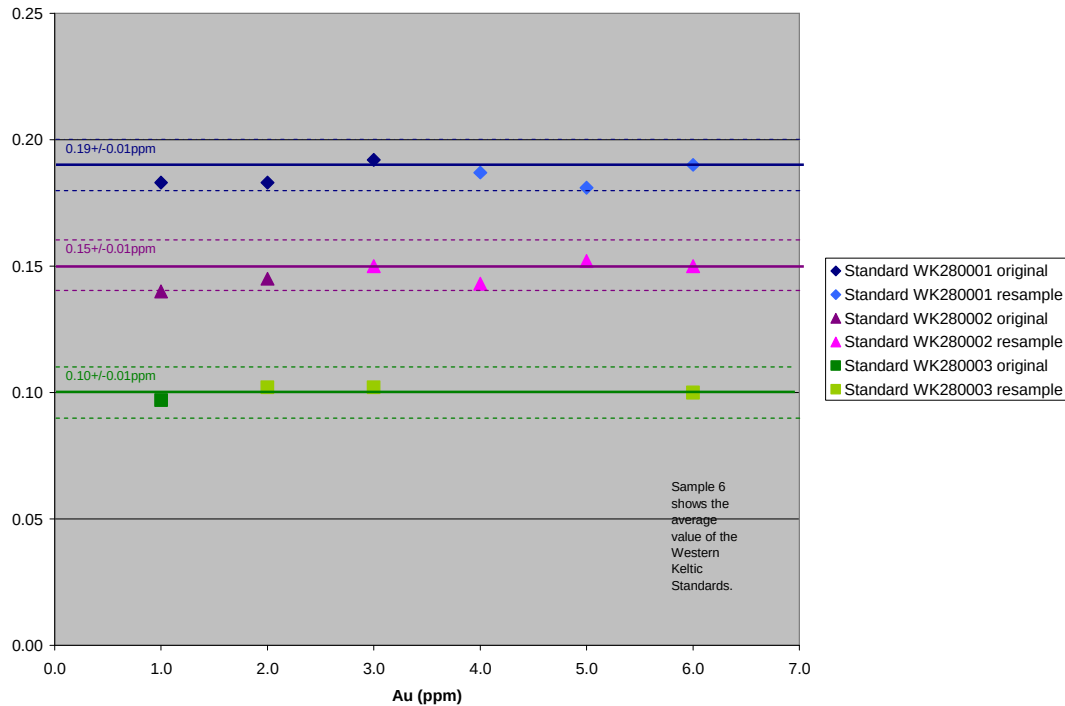


Chart 7.10: Au values of the original and re-sampled standards

8. Check Analyses on Duplicate Pulps

A total of 59 pulps including some blanks and standards were pulled from Chemex Labs and sent to EcoTech Laboratories in Kamloops for re-analysis. Samples were collected from every 2006 drill hole and every analytical shipment as well as spanning a range of copper and zinc values. Results are plotted at X-Y scatter plots in Figures 8.1 to 8.5. The scatter plots show that there is very little difference between the two laboratories, in fact these plots display such a high level of reproducibility that statistical analysis hardly seems necessary. For some of the standards there was insufficient material for analysis, this is because 30 g of material is used for gold analysis and a slightly lesser volume for sulphur analyses during the first round of work; there would be enough material for ICP analysis but unfortunately EcoTech loaded up the fire assay charges first before realizing there was insufficient sample. As the standards start with at least 100g of material there is a suggestion that they may be undergoing re-analysis at the original lab.

The “Reduction to Major Axis” method was used for the current analysis, shown in Charts 8.1 to 8.5. The data table is attached in the Appendix (Table 8.1), as well as QC Data from Eco Tech (Table 8.2 for Repeats and Table 8.3 for Standards).

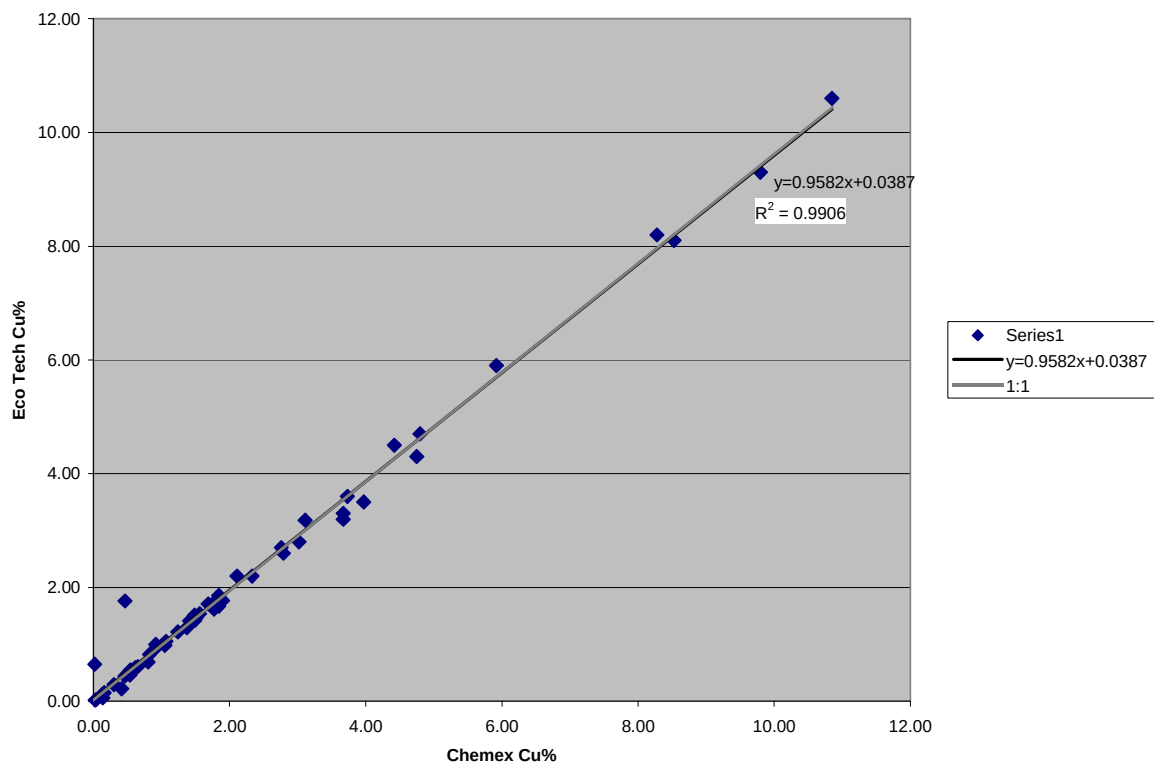


Chart 8.1: Comparison of Cu% for Chemex and EcoTech

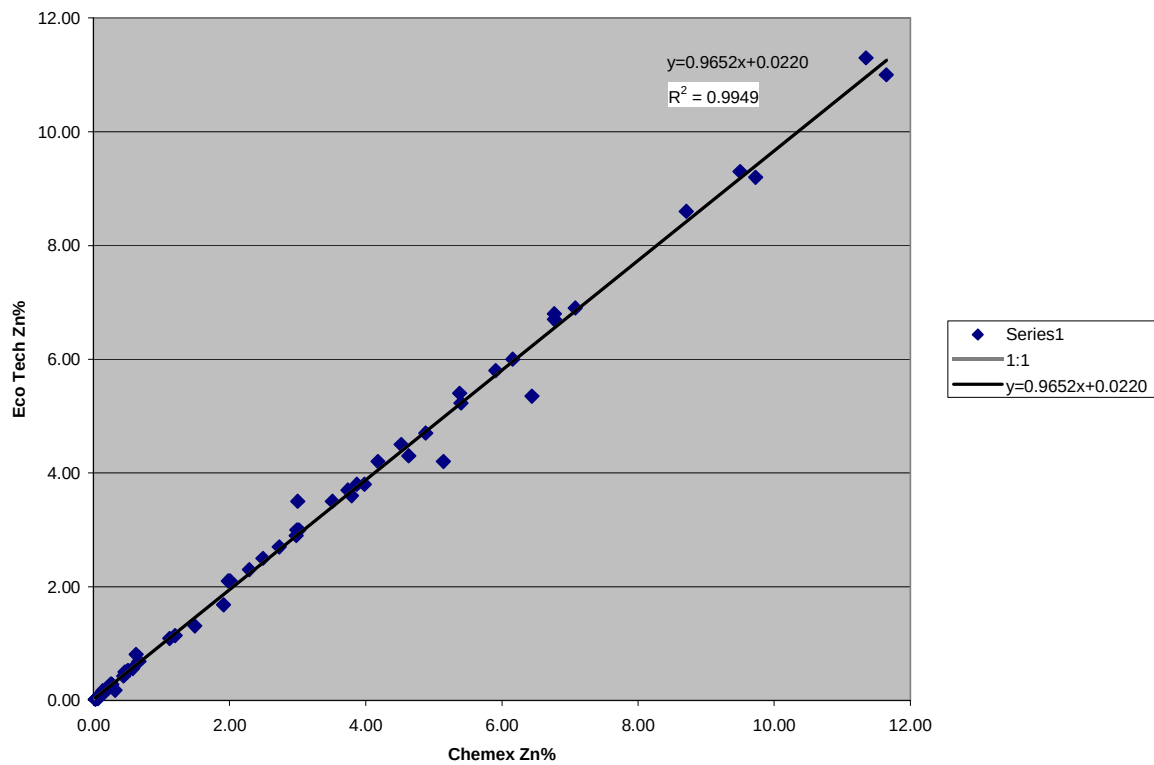


Chart 8.2: Comparison of Zn% for Chemex and EcoTech

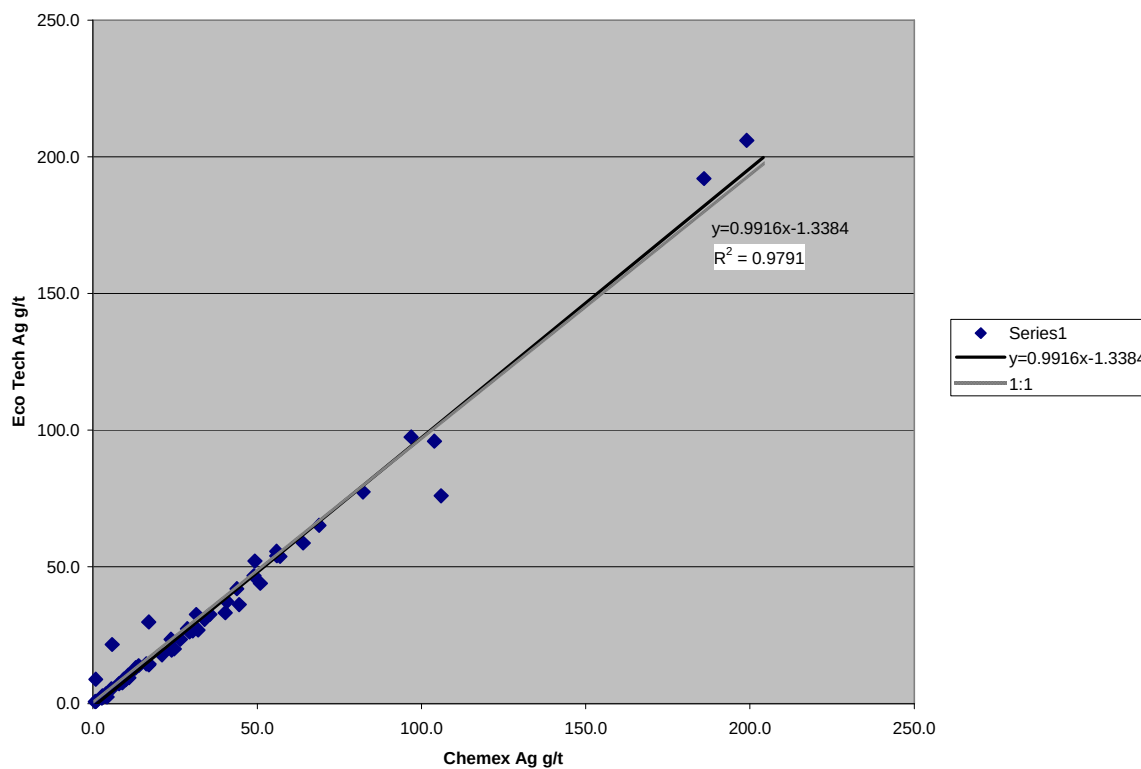


Chart 8.3: Comparison of Ag g/t for Chemex and EcoTech

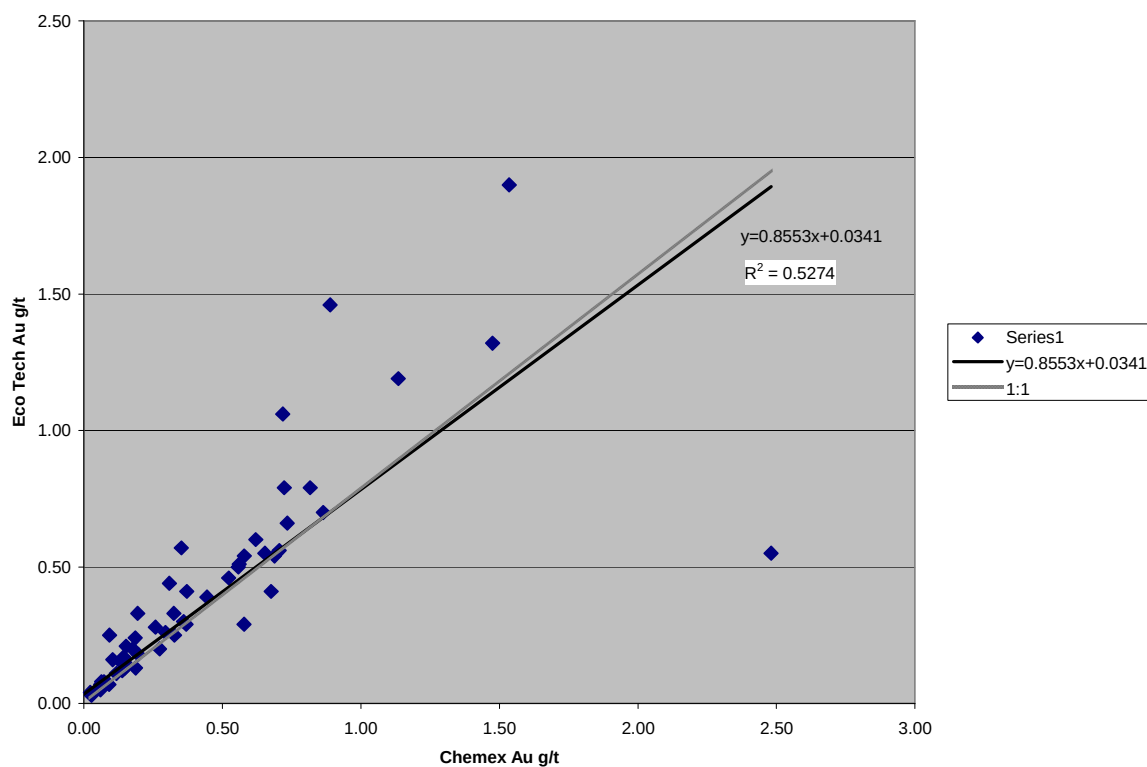


Chart 8.4: Comparison of Au g/t for Chemex and EcoTech
(note: the scale is slightly different for the axes)

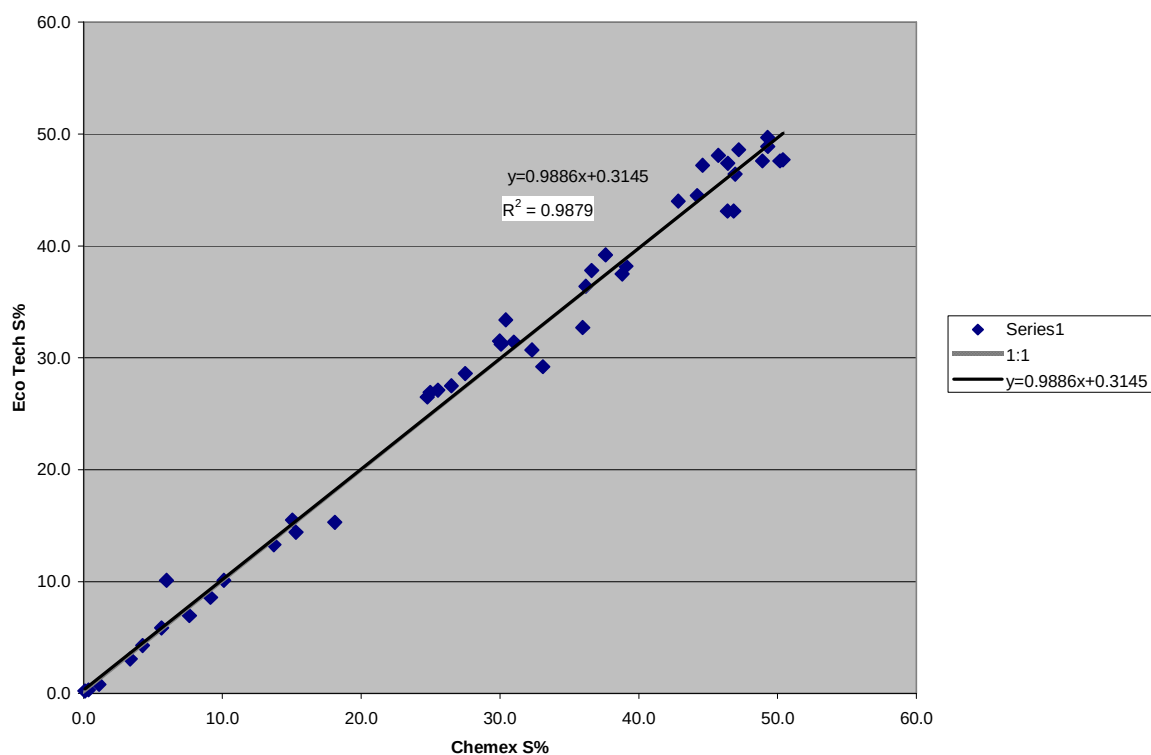


Chart 8.5: Comparison of S% for Chemex and Ecotech

Note that the Chemex analyses were for sulphide sulphur to which a correction factor of 11.1% was applied. The correction factor was determined by using the measured SG values and solving for S in the SML SG formula.

9. EMC Validation – Twin Hole Results

original Hole ID	WK										Hole ID	From (m)	To (m)	Width (m)	SG	Cu %	Zn %	Ag ppm	Au ppm	S %
	From (m)	To (m)	Width (m)	SG	Cu %	Zn %	Ag ppm	Au ppm	S %											
E002	49.6	50.9	1.34	2.95	1.12	2.15	24	0.270	n/a											
E002	50.9	52.1	1.22	2.95	0.16	3.65	7.89	0.690	n/a	WK0674	51.8	52.4	0.6	2.75	0.01	0.02	0.2	0.007	1.5	
E002	52.1	52.9	0.76	2.81	0.00	0	0	0.000	n/a	WK0674	52.4	52.9	0.5	3.26	1.19	1.58	12.6	0.164	>10.0	
E002	52.9	54.1	1.22	2.95	0.16	0.27	5.14	0.030	n/a	WK0674	52.9	54.1	1.2	2.71	0.01	0.04	0.4	0.005	1.4	
E002	54.1	56.5	2.37	2.81	0.00	0	0	0.000	n/a	WK0674	54.1	55.3	1.2	2.72	0.03	0.06	1.1	0.006	1.38	
										WK0674	55.3	56.5	1.2	3.28	2.82	2.50	33	0.157	>10.0	
E002	56.5	58.9	2.47	4.37	2.48	4.18	41.83	0.340	n/a	WK0674	56.5	57.9	1.4	4.69	2.06	5.79	42.3	0.543	>10.0	
E002	58.9	62.1	3.11	4.37	1.37	7.7	25.71	0.240	n/a	WK0674	57.9	59.3	1.4	4.63	1.83	7.01	32.7	0.45	>10.0	
										WK0674	59.3	60.6	1.3	4.63	1.26	9.82	22	0.306	>10.0	
										WK0674	60.6	62.0	1.4	4.62	1.25	7.56	21	0.334	>10.0	
E002	62.1	65.7	3.63	4.37	1.63	6.6	31.2	0.340	n/a	WK0674	62.0	64.0	2	4.60	1.41	9.73	29.4	0.36	>10.0	
										WK0674	64.0	65.3	1.3	4.59	1.66	6.97	26.7	0.316	>10.0	
E002	65.7	68.4	2.74	3.84	0.74	1.3	10.29	0.140	n/a	WK0674	65.3	66.7	1.4	4.59	1.62	5.72	37.9	0.468	>10.0	
										WK0674	66.7	67.4	0.7	2.74	0.05	0.15	0.8	0.01	3.09	
										WK0674	67.4	68.9	1.5	3.97	2.13	1.18	28.2	0.273	>10.0	
E002	68.4	70.9	2.44	3.14	3.04	0.07	47.66	0.450	n/a	WK0674	68.9	70.1	1.2	3.03	0.12	0.07	0.7	0.043	>10.0	
E002	70.9	73.6	2.71	3.14	0.09	0.04	2.4	0.070	n/a	WK0674	70.1	71.8	1.7	3.14	0.07	0.04	1.2	0.017	>10.0	
										WK0674	71.8	73.4	1.6	2.92	0.06	0.77	2.5	0.014	9.3	
E002	73.6	76.1	2.56	3.14	1.23	5.65	21.26	0.140	n/a	WK0674	73.4	75.0	1.6	3.21	0.90	8.41	6.3	0.078	>10.0	
										WK0674	75.0	75.5	0.5	2.81	0.54	0.63	13.1	0.104	4.25	
										WK0674	75.5	76.2	0.7	2.78	0.02	0.06	0.9	0.061	2.67	
										WK0676	55.0	56.4	1.4	2.68	0.01	0.03	<0.2	<0.005	0.52	
										WK0676	56.4	57.1	0.7	4.51	0.72	0.52	17.1	0.171	>10.0	
										WK0676	57.1	57.9	0.8	2.71	0.66	0.18	2.6	0.061	6.54	
										WK0676	57.9	59.0	1.1	2.78	0.11	0.12	3	0.053	5.69	
										WK0676	59.0	60.8	1.8	2.92	1.24	0.53	15	0.17	>10.0	
										WK0676	60.8	62.5	1.7	2.85	0.23	0.10	3.5	0.046	5.59	
										WK0676	62.5	64.0	1.5	2.81	0.05	0.06	0.9	0.022	3.74	

original Hole ID	WK										Hole ID	From (m)	To (m)	Width (m)	SG	Cu %	Zn %	Ag ppm	Au ppm	S %
	From (m)	To (m)	Width (m)	SG	Cu %	Zn %	Ag ppm	Au ppm	S %											
E054	64.2	65.3	1.1	2.8	0.02	0.18	2.1	0.020	n/a	WK0676	64.0	65.6	1.6	2.78	1.70	0.23	4.9	0.046	3.49	
E054	65.3	66.4	1.1	4.55	1.99	2.18	71.7	0.740	n/a	WK0676	65.6	67.0	1.4	3.86	3.67	4.63	44.5	0.557	>10.0	
E054	66.4	69.5	3.1	2.79	0.32	0.96	11	0.120	n/a	WK0676	67.0	67.5	0.5	2.73	0.46	0.14	5.9	0.092	5.97	
										WK0676	67.5	68.7	1.2	3.59	1.82	3.65	25.2	0.359	>10.0	
										WK0676	68.7	69.5	0.8	4.01	2.43	2.79	38.7	0.287	>10.0	
E054	69.5	72.5	3	2.79	0.42	0.41	6.2	0.090	n/a	WK0676	69.5	70.8	1.3	2.87	0.09	0.20	1.3	0.02	1.09	
										WK0676	70.8	72.4	1.6	3.83	1.55	1.19	33.1	0.301	>10.0	
E054	72.5	73.4	0.9	2.7	0.03	0.24	2.7	0.020	n/a	WK0676	72.4	73.6	1.2	2.84	0.13	0.14	1.9	0.04	8.3	
E054	73.4	76.0	2.6	3.89	1.39	3.55	33.3	0.400	n/a	WK0676	73.6	74.9	1.3	2.85	0.31	0.17	3.8	0.077	7.75	
										WK0676	74.9	76.4	1.5	2.87	0.42	0.03	4.3	0.064	7.62	
E054	76.0	77.4	1.4	3.45	4.95	2.6	65.1	1.030	n/a	WK0676	76.4	77.8	1.4	3.00	1.83	0.30	23.1	0.411	>10.0	
E054	77.4	79.0	1.6	4.77	2.80	3.14	59	0.740	n/a	WK0676	77.8	79.2	1.4	2.85	0.58	0.02	5.1	0.146	9.16	
E054	79.0	81.1	2.1	4.33	1.79	1.78	32.6	0.440	n/a											
E054	81.1	82.3	1.2	3.01	0.41	0.68	10.6	0.120	n/a											
E054	82.3	84.7	2.4	3.01	1.43	0.21	19.5	0.300	n/a											
E054	84.7	87.8	3.1	3.01	0.26	0.24	3.1	0.050	n/a											
E054	87.8	88.8	1	3.01	0.44	0.1	6.5	0.100	n/a											
										WK0678	81.0	83.0	2	2.70	0.02	0.07	0.6	0.011	1.33	
										WK0678	83.0	83.5	0.5	3.15	5.05	5.20	5.9	0.286	>10.0	
										WK0678	83.5	84.0	0.5	3.94	1.88	2.54	43.3	0.314	>10.0	
										WK0678	84.0	85.3	1.3	3.52	1.43	6.32	50	0.659	>10.0	
										WK0678	85.3	87.1	1.8	2.82	0.30	1.23	5	0.113	5.2	
										WK0678	87.1	87.8	0.7	3.80	2.29	12.70	32.8	0.426	>10.0	
										WK0678	87.8	89.0	1.2	3.32	0.39	0.32	8.8	0.145	>10.0	
										WK0678	89.0	90.3	1.3	3.85	0.25	0.13	4.9	0.128	>10.0	
										WK0678	90.3	91.3	1	3.82	0.82	0.14	15.8	0.205	>10.0	
										WK0678	91.3	92.2	0.9	3.83	1.81	1.18	23.4	0.261	>10.0	
										WK0678	92.2	93.5	1.3	4.58	0.63	2.29	4.2	0.13	>10.0	

[illegible]

10. Conclusions

References

Long, Scott D., 2003, Assay quality assurance-quality control program for drilling projects at the pre-feasibility to feasibility report level, 57p.

APPENDIX I

Data Tables

Table 4.1: Standard QC 2006 – Standard WK 280001

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
280001	WK06-67	186247	15750	26000	18	0.18	44.1	1	12/10/2006
280001	WK06-68	186282	16600	27200	18		45.8	1	12/10/2006
280001	WK06-70	186323	16600	27600	17	0.29	43.9	1	12/10/2006
280001	WK06-71	186374	16350	26100	18	0.19	44	2	20/10/2006
280001	WK06-81	186709	17000	27000	16.5	0.14	42.3	3	1/11/2006
280001	WK06-83	186739	17500	27300	14.3	0.18	43.6	3	1/11/2006
280001	WK06-74	186530	17600	27700	17	0.18		4	19/12/2006
280001	WK06-76	186584	17400	27800	16.6	0.19		4	19/12/2006
280001	WK06-78	186622	18100	27500	16.4	0.18		4	19/12/2006
Recommended value of 280001:			16989	27133	16.9	0.19	43.9	5	
Two Standard Deviations:			1464	1327	2.3	0.08	2.2		

Table 4.2: Standard QC 2006 – Standard WK 280002

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
280002	WK06-67	186257	14200	16800	20	0.14	33.7	1	12/10/2006
280002	WK06-69	186297	14000	16400	19		33.0	1	12/10/2006
280002	WK06-71	186339	14800	17300	20	0.15	33.3	2	20/10/2006
280002	WK06-72	186390	14350	16500	19	0.13	32.3	2	20/10/2006
280002	WK06-87	186834	15000	17300	17.5	0.14	33.1	3	1/11/2006
280002	WK06-84	186777	15200	16800	15.8	0.13	32.7	3	1/11/2006
280002	WK06-75	186552	15000	17000	19.4	0.15		4	19/12/2006
280002	WK06-77	186600	15400	17500	19.1	0.14		4	19/12/2006
280002	WK06-79	186643	15700	16800	19.5	0.14		4	19/12/2006
Recommended value of 280002:			14850	16933	18.8	0.14	33.0	5	
Two Standard Deviations:			1138	748	2.7	0.01	1.0		

Table 4.3: Standard QC 2006 – Standard WK 280003

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
280003	WK06-66	186227	7310	13900	13	0.10	21.6	1	12/10/2006
280003	WK06-68	186266	7160	13700	13		21.7	1	12/10/2006
280003	WK06-69	186308	8330	17500	14			1	12/10/2006
280003	WK06-71	186359	7170	13400	13	0.11	21.1	2	20/10/2006
280003	WK06-73	186510	7370	13600	13		22	2	20/10/2006
280003	WK06-80	186669	7510	13800	12.2	0.10	20.4	3	1/11/2006
280003	WK06-82	186727	8120	14000	13.4	0.10	19.9	3	1/11/2006
280003	WK06-85	186787	7440	13700	11.2	0.09	20.7	3	1/11/2006
280003	WK06-75	186568	7850	13600	13.2	0.10		4	19/12/2006
280003	WK06-77	186611	7760	14100	12.9	0.10		4	19/12/2006
Recommended value of 280003:			7602	14130	12.9	0.10	21.9	5	
Two Standard Deviations:			800	2404	1.5	0.01	1.5		

Table 4.4: Standard QC 2006 – Standard Canadian High (CDN-HLHC)

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
Std Cdn High	WK06-65	186211	49500	23000	116	1.73	25.8	1	12/10/2006
Std Cdn High	WK06-67	186246	49400	23800	120	1.67	26.4	1	12/10/2006
Std Cdn High	WK06-69	186309	48700	23100	114	1.63	24.6	1	12/10/2006
Std Cdn High	WK06-71	186373	47700	23000	111	2.00	25.2	2	20/10/2006
Std Cdn High	WK06-86	186820	50400	23200	111	1.83	24.2	3	1/11/2006
Std Cdn High	WK06-84	186762	51000	23300	110	1.91	24.3	3	1/11/2006
Std Cdn High	WK06-75	186553	50600	23100	114	2.13		4	19/12/2006
Certified value of Std Cdn High (HLHC):			50700	23500	111	1.97	n/a	5	
Two Standard Deviations:			2700	1100	8.6	0.22			

Table 4.5: Standard QC 2006 – Standard Canadian Low (CDN-HLLC)

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
Std Cdn Low	WK06-66	186226	14450	29500	69	0.78	32.6	1	12/10/2006
Std Cdn Low	WK06-68	186281	14800	30400	70	0.75	33.3	1	12/10/2006
Std Cdn Low	WK06-71	186340	14750	31000	70	0.84	32.6	2	20/10/2006
Std Cdn Low	WK06-73	186509	14600	29600	66	0.76	32.7	2	20/10/2006
Std Cdn Low	WK06-86	186811	15000	29800	64.2	0.81	32.2	3	1/11/2006
Std Cdn Low	WK06-82	186728	15100	29900	68.9	0.72	30.0	3	1/11/2006
Std Cdn Low	WK06-76	186585	15400	30500	63.5	0.73		4	19/12/2006
Std Cdn Low	WK06-79	186644	16300	30000	72.7	0.73		4	19/12/2006
Certified value of Std Cdn Low (HLLC):			14900	30100	65.1	0.83	n/a	5	
Two Standard Deviations:			600	1700	6.7	0.12			

Table 4.6: Standard QC 2006 – Standard FCM3

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
Std FCM3	WK06-80	186681	3030	5080	23.8	0.37	13.7	3	1/11/2006
Std FCM3	WK06-81	186702	3100	5290	24.4	0.40		3	1/11/2006
Certified value of Std FCM3:			2900	5400	23.6	0.4	n/a	5	
Two Standard Deviations:			200	320	3.3	0.07			

Table 5.1.1: Blank QC 2006 – Limestone Cu blanks

Sample ID	Cu ppm	Cu previous Sample ppm
186213	452	24600
186243	225	28300
186278	2	553
186306	1380	59200
186336	64	9350
186371	212	13800
186506	1050	14900
186831	212	15500
186676	202	29700
186683	183	13700
186716	729	27900
186756	585	18000
186784	257	32700
186549	201	16600
186581	251	36700
186609	42	5140
186639	198	39700

Table 5.1.2: Blank QC 2006 – Limestone Zu

Sample ID	Zn ppm	Zn previous Sample ppm
186213	870	50800
186243	1330	153500
186278	5	290
186306	3160	67700
186336	160	27300
186371	290	11200
186506	540	24900
186831	61	25000
186676	157	97300
186683	709	54000
186716	191	20100
186756	1240	116500
186784	448	34500
186549	525	50700
186581	290	46300
186609	62	7000
186639	190	30000

Table 5.1.3: Blank QC 2006 – Rhyolite Cu blanks

Sample ID	Cu ppm	Cu previous Sample ppm
186224	639	24600
186255	515	11150
186264	9	428
186295	343	7940
186320	645	23300
186357	182	8440
186386	285	14650
186803	560	114000
186665	80	66000
186699	105	42900
186736	404	21500
186774	203	8850
186527	298	20600
186566	78	6840
186598	237	34800
186618	244	18800

Table 5.1.4: Blank QC 2006 – Rhyolite Zn

Sample ID	Zn ppm	Zn previous Sample ppm
186224	1160	58100
186255	120	500
186264	40	1500
186295	1000	45100
186320	380	11050
186357	910	124000
186386	680	13300
186803	80	6510
186665	67	31000
186699	138	33400
186736	772	52700
186774	114	706
186527	619	57900
186566	130	8630
186598	283	48600
186618	309	25400

Table 5.2.1: WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined – Standard WK 280001

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
280001	WK05-39	280426	16450	28500	19	0.18	44.2	1	29/7/2005
280001	WK05-39	280465	16500	26400	18	0.18	44.9	1	29/7/2005
280001	WK05-41	280483	16600	25600	18	0.18	42.3	1	29/7/2005
280001	WK05-42	280354	17100	27500	17	0.18	43.8	1	29/7/2005
280001	WK05-45	280383	16150	26500	17	0.16	43.3	1	29/7/2005
280001	WK05-45	280405	16050	26100	16	0.17	42.2	1	29/7/2005
280001	WK05-54B1	280345	17200	27900	18	0.17	44.8	2	12/8/2005
Recommended value for 2005&2006:			16809	27044	17.2	0.18	43.8	11	
Two Standard Deviations:			1274	1660	2.2	0.06	2.1		

Table 5.2.2: WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined – Standard WK 280002

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
280002	WK05-44	280370	14700	16250	18	0.15	33.1	1	29/7/2005
280002	WK05-47	280448	15100	17400	21	0.14	33.5	3	12/8/2005
280002	WK05-49	280560	14200	16250	19	0.15	31	3	12/8/2005
280002	WK05-52	280321	14900	17350	21	0.14	32.9	3	12/8/2005
280002	WK05-39	280631	14750	16850	20	0.15	32.2	2	12/8/2005
280002	WK05-57	280700	14450	16250	19	0.15	32.4	5	2/9/2005
280002	WK05-58	280763	14700	17050	20	0.15	32.7	5	2/9/2005
280002	WK05-59	280801	14750	16850	19	0.14	32.7	5	2/9/2005
Recommended value for 2005&2006:			14776	16862	19.2	0.14	31.9	11	
Two Standard Deviations:			896	844	2.5	0.01	1.3		

Table 5.2.3: WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined – Standard WK 280003

Standard Type	Hole ID	Sample ID	Cu ppm	Zn ppm	Ag ppm	Au ppm	S %	Sample Submission Batch	Date Submitted
280003	WK05-50	280582	7380	13350	11	0.11	20.1	3	12/8/2005
280003	WK05-54B2	280666	7440	13300	13	0.12	20.3	4	25/8/2005
280003	WK05-59	280780	7420	13750	13	0.08	21.1	5	2/9/2005
280003	WK05-62	280841	7370	13950	13	0.09	20.8	6	26/9/2005
280003	WK05-63	280890	7060	13150	13	0.10	20.2	6	26/9/2005
280003	WK05-64	280862	7220	13450	13	0.10	20.4	6	26/9/2005
Recommended value for 2005&2006 combined:			7494	13891	12.8	0.10	20.8	11	
Two Standard Deviations:			704	1999	1.5	0.02	1.3		

Table 5.2.4: WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined – Standard Chemex GBM399-5

Standard Type	Cu ppm	Zn ppm	Ag ppm	S %	Date Submitted	Sample Submission Batch
GBM399-5	28300	9130	23	2.4	29.7.2005	1
GBM399-5	29100	9100	26	2.3	29.7.2005	1
GBM399-5	28600	9890	28	2.5	29.7.2005	1
GBM399-5	28000	8780	27	2.4	29.7.2005	1
GBM399-5	28200	9040	25	2.4	29.7.2005	1
GBM399-5	29900		24		29.7.2005	1
GBM399-5	29500	9400			29.7.2005	1
GBM399-5		9700			29.7.2005	1
GBM399-5		9800			29.7.2005	1
GBM399-5		9500			29.7.2005	1
GBM399-5	29600	9200	26	2.4	12.8.2005	2
GBM399-5	29800	9300	27	2.4	12.8.2005	3
GBM399-5		9700			12.8.2005	3
GBM399-5		9600			12.8.2005	3
GBM399-5	28800	9100	26	2.4	25.8.2005	4
GBM399-5	29600	9270	25	2.4	25.8.2005	4
GBM399-5	29000	9070	24	2.4	2.9.2005	5
GBM399-5	28100	8820	24	2.5	2.9.2005	5
GBM399-5		9600			2.9.2005	5
GBM399-5	29200	9560	26	2.4	26.9.2005	6
GBM399-5		9400		2.3	26.9.2005	6
GBM399-5	28000	8880	25	2.6	26.9.2005	6
GBM399-5	28300	9440	26		26.9.2005	6

**Table 5.2.4: WK Internal Standards and ALS Chemex Internal Standards for 2005/2006
combined – Standard Chemex GBM399-5 (continued...)**

Standard Type	Cu ppm	Zn ppm	Ag ppm	S %	Date Submitted	Sample Submission Batch
GBM399-5	28200	8980	25	2.4	12.10.2006	7
GBM399-5	28500	9140	25	2.5	12.10.2006	7
GBM399-5	28300	9110	25	2.5	12.10.2006	7
GBM399-5	28800	9300	25	2.4	12.10.2006	7
GBM399-5	28500	9380	25	2.4	12.10.2006	7
GBM399-5	29000	9500			12.10.2006	7
GBM399-5	29000				12.10.2006	7
GBM399-5		9300			12.10.2006	7
GBM399-5		9200			12.10.2006	7
GBM399-5		9300			12.10.2006	7
GBM399-5	28800	9300			12.10.2006	7
GBM399-5	29100				12.10.2006	7
GBM399-5	28600				12.10.2006	7
GBM399-5	28800				12.10.2006	7
GBM399-5	27900		25	2.4	20.10.2006	8
GBM399-5	29300		25	2.5	20.10.2006	8
GBM399-5	28500		26	2.5	20.10.2006	8
GBM399-5		9400			20.10.2006	8
GBM399-5		9000			20.10.2006	8
GBM399-5	29000				20.10.2006	8
GBM399-5	30000	9300	23		1.11.2006	9
GBM399-5	30300	9300			1.11.2006	9
GBM399-5	29500	9300			1.11.2006	9
GBM399-5	29300	9200	24		1.11.2006	9
Recommended value:	n/a	9297	25.2	2.4		10
Two Standard Deviations:		519	2.4	0.1		

Table 5.2.5 WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined – Standard Chemex GBM302-10

Standard Type	Cu ppm	Zn ppm	Ag ppm	Date Submitted	Sample Submission Batch
GBM302-10	4300			12.10.2006	7
GBM302-10	4200			12.10.2006	7
GBM302-10	4300	142000		12.10.2006	7
GBM302-10	4300			12.10.2006	7
GBM302-10	4300			20.10.2006	8
GBM302-10	4400	146000		1.11.2006	9
GBM302-10	4400	144500	77	1.11.2006	9
GBM302-10	4600	146500		1.11.2006	9
GBM302-10	4500	146500	78	1.11.2006	9
Recommended value:	4367	n/a	n/a		10
Two Standard Deviations:	245				

Table 5.2.6 WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined – Standard Chemex GBM3961c

Standard Type	Cu ppm	Zn ppm	Ag ppm	S %	Date Submitted	Sample Submission Batch
GBM3961c	2890	7140	9	3.7	12.10.2006	7
GBM3961c	2890	6930	9	3.9	12.10.2006	7
GBM3961c	2970	7030	9	4.0	12.10.2006	7
GBM3961c	2950	7060	9	3.8	12.10.2006	7
GBM3961c	3010	7600	10	3.9	12.10.2006	7
GBM3961c	3010	7200	7	4.0	20.10.2006	8
GBM3961c	2840	6820	10	3.6	20.10.2006	8
GBM3961c	2970	7320	10	3.9	20.10.2006	8
Recommended value:	2941	7138	9	3.8		10
Two Standard Deviations:	123	486	2	0.2		

**Table 5.2.7 WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined –
Standard Chemex GS01-2**

Standard Type	Cu ppm	Zn ppm	Ag ppm	S %	Date Submitted	Sample Submission Batch
GS01-2	6120	4040	21	1.2	29.7.2005	1
GS01-2	6070	3990	22	1.2	29.7.2005	1
GS01-2	6070	4000	23	1.2	29.7.2005	1
GS01-2	6060	3830	21	1.2	29.7.2005	1
GS01-2	6010	3820	24	1.2	29.7.2005	1
GS01-2	6080	3990	21	1.2	29.7.2005	1
GS01-2	6070	3900	24	1.2	12.8.2005	2
GS01-2	5950	3880	21	1.1	12.8.2005	3
GS01-2	6110	3870	21	1.2	12.8.2005	3
GS01-2	6100	4000	23	1.2	12.8.2005	3
GS01-2	6020	3960	21	1.2	12.8.2005	3
GS01-2	6110	3940	20	1.2	25.8.2005	4
GS01-2	6260	3970	20	1.2	25.8.2005	4
GS01-2	6170	4050	21	1.3	2.9.2005	5
GS01-2	6130	3990	21	1.2	2.9.2005	5
GS01-2	5850	3820	21	1.1	26.9.2005	6
GS01-2	5950	3800	21	1.2	26.9.2005	6
GS01-2	6030	4090	22	1.2	26.9.2005	6
GS01-2	6150	4130	23	1.2	26.9.2005	6
Recommended value:	6069	3951	22	1.2		10
Two Standard Deviations:	181	191	2	0.1		

**Table 5.2.8 WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined –
Standard Chemex OxE21**

Standard Type	Au ppm	Date Submitted	Sample Submission Batch
OxE21	0.66	29.7.2005	1
OxE21	0.60	29.7.2005	1
OxE21	0.62	29.7.2005	1
OxE21	0.66	29.7.2005	1
OxE21	0.63	29.7.2005	1
OxE21	0.64	29.7.2005	1
OxE21	0.65	29.7.2005	1
OxE21	0.71	29.7.2005	1
OxE21	0.68	12.8.2005	2
OxE21	0.70	12.8.2005	2
OxE21	0.65	12.8.2005	3
OxE21	0.64	12.8.2005	3
OxE21	0.68	12.8.2005	3
OxE21	0.67	12.8.2005	3
OxE21	0.66	12.8.2005	3
OxE21	0.64	12.8.2005	3
OxE21	0.66	12.8.2005	3
OxE21	0.65	25.8.2005	4
OxE21	0.66	25.8.2005	4
OxE21	0.64	25.8.2005	4
OxE21	0.67	2.9.2005	5
OxE21	0.66	2.9.2005	5
OxE21	0.62	2.9.2005	5
OxE21	0.63	2.9.2005	5
OxE21	0.63	2.9.2005	5
OxE21	0.64	2.9.2005	5
OxE21	0.71	2.9.2005	5
OxE21	0.70	2.9.2005	5
OxE21	0.64	28.9.2005	6
OxE21	0.62	28.9.2005	6
Recommended value:	0.65		11
Two Standard Deviations:	0.05		

**Table 5.2.9 WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined –
Standard Chemex OxG46**

Standard Type	Au ppm	Date Submitted	Sample Submission Batch
OxG46	1.04	12.10.2006	7
OxG46	1.02	12.10.2006	7
OxG46	1.03	12.10.2006	7
OxG46	1.01	12.10.2006	7
OxG46	1.00	12.10.2006	7
OxG46	1.03	12.10.2006	7
OxG46	1.04	12.10.2006	7
OxG46	0.99	12.10.2006	7
OxG46	1.03	12.10.2006	7
OxG46	1.04	20.10.2006	8
OxG46	1.01	20.10.2006	8
OxG46	1.08	20.10.2006	8
OxG46	1.05	1.11.2006	9
OxG46	1.00	1.11.2006	9
OxG46	1.02	1.11.2006	9
OxG46	1.05	1.11.2006	9
OxG46	1.01	1.11.2006	9
OxG46	1.06	1.11.2006	9
OxG46	1.03	1.11.2006	9
OxG46	1.00	1.11.2006	9
OxG46	1.05	1.11.2006	9
OxG46	1.00	1.11.2006	9
OxG46	0.99	1.11.2006	9
OxG46	1.06	1.11.2006	9
Recommended value:	1.03		11
Two Standard Deviations:	0.05		

Table 5.2.10 WK Internal Standards and ALS Chemex Internal Standards for 2005/2006 combined – Standard Chemex

Standard	Au	Date	Sample Submission
Type	ppm	Submitted	Batch
SI15	1.86	29.7.2005	1
SI15	1.89	29.7.2005	1
SI15	1.87	29.7.2005	1
SI15	1.93	25.8.2005	4
SI15	1.87	2.9.2005	5
SI15	1.77	2.9.2005	5
SI15	1.88	2.9.2005	5
SI15	1.81	28.9.2005	6
SI15	1.83	12.10.2006	7
SI15	1.79	12.10.2006	7
Recommended value:	1.85		11
Two Standard Deviations:	0.10		

Table 6.1 Data for Historical SML Core Intersection Resample (...)

Hole ID	Sample ID	resampled							historic						
		From (m)	To (m)	Cu %	Zn %	Ag ppm	Au ppm	S %	From (m)	To (m)	Cu %	Zn %	Ag ppm	Au ppm	S %
KT049	186402	109.73	110.49	0.51	2.37	24.3	0.118	12.55	109.72	110.49	0.33	3.21	20.57	0.1	14.9
KT049	186403	110.49	111.01	0.21	3.27	2.7	0.049	40.2	110.49	111.00	0.24	3.36	5.49	0.07	41.5
KT049	186404	111.01	111.86	1.01	0.23	8.9	0.103	16.5	111.00	111.86	1.26	0.18	17.14	0.1	18.3
KT049	186405	111.86	112.24	0.56	1.8	3.9	0.048	14.45	111.86	112.77	0.21	1.35	13.03	0.1	12.4
KT049	186406	112.24	112.78	2.95	5.77	46	0.343	16.55	112.77	113.38	3.96	4.71	56.91	0.27	18.4
KT049	186408	113.39	114.00	7.87	0.66	104	0.711	11.05	113.38	113.99	8.32	0.51	110.4	0.17	10.9
KT049	186409	114.00	115.21	0.57	1.06	5.6	0.097	17.7	113.99	114.90	0.66	0.98	4.8	0.1	18
KT049	186410	115.21	115.52	0.091	1.08	3.7	0.018	2.28	114.90	115.51	0.04	0.96	0.34	0.05	2.7
KT049	186411	115.52	115.82	0.44	5.27	2	0.034	17.65	115.51	115.82	0.41	5.47	0.34	0.05	18.9
KT049	186412	115.82	116.43	0.02	0.02	3.2	0.036	40.9	115.82	116.43	0.03	0.03	0.17	0.05	44.4
KT049	186414	116.43	117.50	0.01	0.05	1	0.028	38.5	116.43	117.50	0.01	0.02	0.17	0.05	41.5
KT075	186415	55.69	57.30	3.47	9.2	101	0.573	41.4	55.68	57.30	3.05	9.39	96	0.41	43.5
KT075	186416	57.30	58.22	4.36	0.67	107	1.035	45.5	57.30	58.21	4.35	1.44	96.69	0.48	46.7
KT075	186417	58.22	59.77	2.48	7.3	63.9	0.449	38.8	58.21	59.74	2.15	6.52	49.37	1.23	44.3
KT075	186418	59.74	60.96	2.37	2.83	45.8	0.529	38.9	59.74	60.96	2.3	2.82	44.57	0.69	44.6
KT075	186420	60.96	62.73	2.37	14.3	91.1	0.599	37.2	60.96	62.72	2.15	13.9	83.66	1.51	41.4
KT075	186421	62.73	64.01	1.15	0.72	26.5	0.382	26.4	62.72	64.00	0.96	0.9	15.09	0.17	26.9
KT075	186423	64.01	65.53	1.74	0.13	28.9	0.534	39	64.00	65.53	1.42	0.31	24.69	0.34	42.1
KT075	186424	65.53	66.45	4.24	2.06	41.4	0.896	34.9	65.53	66.44	1	0.09	22.63	0.17	39.6
KT075	186425	66.45	69.49	0.37	0.11	4.1	0.055	15.2	66.44	69.49	0.28	0.19	5.49	0.1	18.6
KT094	186426	110.79	111.47	1.29	41.4	21.4	0.245	24.4	110.79	111.46	0.93	41.2	4.11	0.17	27.4
KT094	186427	111.47	112.17	5.58	2.05	51.1	1.105	15.4	111.46	112.16	5.75	3.26	24.69	0.41	22
KT094	186428	112.17	113.63	0.55	0.64	15.2	0.119	1.46	112.16	113.62	1.02	1.71	34.29	0.75	2.9
KT094	186429	113.63	114.85	10.05	2.09	262	1.055	8.17	113.62	114.84	11.8	3.26	8.23	0.34	9.31
KT094	186432	114.85	117.50	0.79	0.57	18.5	0.207	8.12	114.84	117.50	0.58	0.4	13.71	0.1	8.21
KT094	186433	132.74	133.87	1.73	1.59	15.6	0.295	30.3	132.74	133.86	1.58	1.34	18.51	0.1	37.4
KT094	186434	133.87	135.18	2.16	1.47	32.5	0.366	30.8	133.86	135.17	2.44	1.79	15.77	0.17	36
KT094	186435	135.18	136.73	2.65	0.13	47.6	0.445	38.9	135.17	136.73	2.07	0.1	28.8	0.21	45.2

Table 6.1 Data for Historical SML Core Intersection Resample (continued...)

Hole ID	Sample ID	From (m)	To (m)	resampled					From (m)	To (m)	historic				
				Cu %	Zn %	Ag ppm	Au ppm	S %			Cu %	Zn %	Ag ppm	Au ppm	S %
KT094	186436	136.73	138.04	1.51	0.78	20.4	0.289	43.5	136.73	138.04	1.86	0.56	39.09	0.21	52.3
KT094	186437	138.04	139.29	0.34	0.06	3.6	0.141	33.9	138.04	139.29	0.35	0.07	30.86	0.17	51.4
KT094	186438	139.29	140.33	0.07	0.05	1.8	0.102	48.4	139.29	140.32	0.05	0.05	3.43	0.03	52.7
KT094	186439	140.33	142.34	0.11	0.9	1.9	0.058	36.4	140.32	142.34	0.1	1	1.71	0.21	46.5
KT094	186440	142.34	143.87	0.42	0.46	3.6	0.074	43.4	142.34	143.86	0.38	0.5	1.71	0	52
KT094	186441	143.87	145.39	0.92	0.42	5.4	0.074	43.6	143.86	145.38	0.87	0.37	3.43	0.03	52.7
KT094	186442	145.39	146.91	0.2	0.43	5.6	0.11	45.2	145.38	146.91	0.16	0.44	4.8	0.03	52.9
KT094	186443	146.91	148.44	0.23	0.06	0.8	0.087	44.6	146.91	148.43	0.13	0.1	8.23	0.07	51.3
KT094	186444	148.44	150.88	0.02	0.02	0.6	0.054	36.2	148.43	150.87	0.02	0.03	1.37	0.21	47.5
KT082	186451	49.86	50.38	1.29	0.03	5.5	0.509	7.16	49.86	50.38	0.93	0.03	4.11	0.34	6.95
KT082	186452	50.38	51.81	5.58	3.78	113	2.36	24.7	50.38	51.81	4.88	3.65	102.86	2.47	27.5
KT082	186453	51.81	53.03	1.86	1.12	31.7	0.632	40.3	51.81	53.03	1.4	1.18	54.86	0.41	43.6
KT082	186455	53.03	54.25	1.37	2.57	37.3	0.451	42.2	53.03	54.25	1.12	2.12	72.69	0.48	46.5
KT082	186456	54.25	55.16	2.81	2.44	57.3	0.937	38.7	54.25	55.16	2.52	2.48	44.57	0.82	39.5
KT082	186457	55.16	56.69	1.23	1.29	6.2	0.478	37.6	55.16	56.69	1.15	1.18	6.86	0.75	36.9
KT082	186458	56.69	57.91	1.04	0.11	4.1	0.355	37.2	56.69	57.91	0.9	0.14	3.43	0.69	35.3
KT082	186460	57.92	59.92	1.81	0.08	6.8	0.39	42.1	57.91	59.92	1.54	0.06	8.23	0.69	42
KT082	186461	59.92	60.44	2.26	0.86	9	0.273	37.3	59.92	60.44	1.92	1.02	10.29	0.1	37
KT082	186462	60.44	61.87	0.16	0.14	3.6	0.09	18.25	60.44	61.87	0.1	0.21	0.34	0	17.1
KT046	186463	86.25	86.71	0.72	0.56	6.1	0.08	6.3	86.25	86.71	0.83	0.87	17.83	0.1	6.7
KT046	186464	86.71	87.93	0.05	0.12	0.7	0.011	0.92	86.71	87.93	0.04	0.09	15.09	0.1	0.8
KT046	186465	87.93	89.55	3.01	0.87	25.8	0.343	19.3	87.93	89.55	3.39	3.42	34.97	0.34	18.6
KT046	186467	89.55	90.83	0.13	0.22	1.8	0.026	0.43	89.55	90.83	0.13	0.24	13.03	0.1	0.8
KT046	186469	90.83	92.75	0.62	1.34	11.5	0.201	6.6	90.83	92.75	0.45	0.85	19.2	0.1	5.5
KT046	186470	92.75	93.08	3.16	2.26	41.3	0.442	12.85	92.75	93.08	3	1.92	45.26	0.17	10.8
KT046	186471	93.08	94.18	0.89	4.39	19.8	0.404	33.1	93.08	94.18	0.79	4.6	32.23	0.34	33.2
KT046	186472	94.18	95.79	0.08	0.02	3.5	0.046	37.2	94.18	95.79	0.06	0.04	17.83	0.1	40.4
KT046	186473	95.79	96.31	0.46	0.55	12.3	0.182	42	95.79	96.31	0.48	0.76	22.63	0.1	46.4

Table 6.1 Data for Historical SML Core Intersection Resample (continued...)

Hole ID	Sample ID	From (m)	To (m)	resampled					From (m)	To (m)	historic				
				Cu %	Zn %	Ag ppm	Au ppm	S %			Cu %	Zn %	Ag ppm	Au ppm	S %
KT046	186474	96.31	96.77	0.99	0.07	4.1	0.182	23.3	96.31	96.77	1.15	0.12	18.51	0.1	27
KT046	186475	96.77	97.84	2.03	10.2	45.8	1.29	35.4	96.77	97.84	1.69	11.1	61.71	0.82	36.8
KT046	186476	97.84	99.06	3.45	7.4	136	0.985	38.9	97.84	99.06	3.71	5.1	231.09	1.37	36.4
KT046	186477	99.06	100.58	2.05	14.05	80.6	0.788	38.1	99.06	100.58	2.61	10.4	90.51	0.69	40.7
KT046	186478	100.58	102.10	3.59	1.5	61.9	0.414	34.8	100.58	102.10	3.78	1.58	54.86	0.17	40.6
KT046	186480	102.10	103.32	3.25	1.21	27.8	0.363	36.8	102.10	103.32	3.5	2.06	26.74	0.1	41.1
KT046	186481	103.32	103.93	0.18	0.23	1.8	0.07	23.4	103.32	103.93	0.15	0.11	0.34	0	23.2
KT046	186482	103.93	105.67	0.63	0.41	7.7	0.154	39.5	103.93	105.67	0.58	0.41	5.49	0.1	41.7
KT046	186484	105.67	107.80	0.1	0.11	0.8	0.029	18.45	105.67	107.80	0.07	0.17	0.69	0	20.6
KT046	186485	107.80	108.53	0.58	0.42	6.5	0.132	41	107.80	108.53	0.58	0.5	6.86	0.1	43
KT046	186486	108.53	110.15	1.1	0.23	20	0.432	21.7	108.53	110.15	1.04	0.18	16.46	0.1	22.5
							mean							mean	
							y	1913.94						x	2106.47
								28.5663							31.4399
KT001	C401001	76.20	78.46	3.48	7.85	73	0.47	37.3	75.89	78.45	3.15	7.26	68.57	0.55	
KT001	C401002	78.46	83.21	1.24	0.45	25.36	0.65	18.32	78.45	83.21	1.28	0.35	28.11	0.55	
KT001	C401005	83.21	84.67	4.49	10.7	96.6	0.351	35.9	83.21	84.67	0.28	0.52	9.26	0.1	
KT001	C401006	84.67	86.26	0.5	0.33	14.6	0.177	37.2	84.67	86.25	0.64	0.41	14.74	0.17	
KT001	C401007	86.26	87.78	0.31	0.69	8.8	0.109	34.7	86.25	87.78	3.78	9.15	96	0.38	
KT008	C401009	115.37	115.70	2.13	2.67	47.5	0.635	38.1	115.36	115.70	1.35	4.05	36.34	0.1	
KT008	C401010	115.70	116.92	4.46	1.57	65.7	0.45	22.2	115.70	116.92	6.87	0.77	68.57	0.69	
KT008	C401011	116.92	117.96	2.26	1.87	40.8	0.183	22	116.92	117.95	0.28	0.03	3.43	0.03	
KT008	C401012	117.96	120.09	1.8	1.72	33.6	0.2	22	117.95	120.09	2.2	2.39	39.43	0.1	
KT008	C401013	120.09	121.10	1.72	3.27	47.9	0.33	32.8	120.09	121.09	1.43	1.8	23.66	0.1	
KT008	C401014	127.10	128.32	2.2	3.89	47.5	0.302	38	127.10	128.32	2.12	3.15	43.54	0.1	
KT008	C401015	128.32	129.54	2.41	4.53	52.4	0.746	29.3	128.32	129.54	3.12	5	46.63	0.86	
KT008	C401016	129.54	130.52	0.18	0.15	3.4	0.051	28	129.54	130.51	0.25	0.23	6.17	0.05	
KT008	C401017	145.91	146.40	0.77	5.44	32	0.748	25							

Table 6.1 Data for Historical SML Core Intersection Resample (continued...)

Hole ID	Sample ID	From (m)	To (m)	resampled					S %	historic				
				Cu %	Zn %	Ag ppm	Au ppm	From (m)		To (m)	Cu %	Zn %	Ag ppm	Au ppm
KT010	C401018	154.99	155.45	0.04	0.39	2	0.032	5.13	154.99	155.44	0.02	0.35	1.71	0
KT010	C401019	155.45	156.03	0.59	5.06	16.3	0.257	29.6	155.44	156.02	0.61	4.43	15.09	0.1
KT010	C401020	156.03	156.33	0.03	0.09	0.5	0.029	3.65	156.02	156.33	0.01	0.05	13.03	0.1
KT010	C401021	156.33	156.51	0.62	15.9	15.1	0.253	26	156.33	156.63	0.7	7.94	13.03	0.1
KT010	C401022	156.51	156.91	0.67	0.08	15.1	0.144	3.26	156.63	156.91	0.96	0.07	19.89	0.1
KT010	C401023	156.91	157.43	0.44	5.23	13.9	0.191	26.1	156.91	157.42	0.41	5.54	0	0
KT010	C401024	157.43	158.56	1.7	0.94	35.6	0.317	43.9	157.42	158.86	1.54	0.36	26.06	0.1
KT014	C401027	151.58	152.16	5.59	4.58	147	2.33	38.3	151.57	152.15	5.36	4.79	61.03	1.3
KT014	C401028	152.16	153.50	1.2	2.88	60.9	0.588	17.95	152.15	153.49	1.42	3.15	118.63	2.47
KT014	C401029	153.50	155.60	0.72	0.72	37.2	0.32	8.76	153.49	155.60	0.74	0.28	27.43	0.69
KT014	C401030	155.60	156.67	2.34	1.52	19.8	0.2	43	155.60	156.66	2.7	1.35	18.51	0.55
KT014	C401031	156.67	158.19	1.11	1	26.9	0.259	45.5	156.66	158.19	1.2	0.82	22.63	0.62
KT014	C401032	158.19	159.72	1.21	2.05	30.1	0.334	44.7	158.19	159.71	0.92	2.39	20.57	0.34
KT014	C401033	159.72	162.15	1.69	1.91	41.2	0.359	45.5	159.71	162.15	1.69	1.92	44.57	0.48
KT014	C401034	162.15	163.68	0.62	2.3	13.3	0.155	45	162.15	163.67	0.52	2.39	13.71	0.41
KT014	C401037	163.68	165.20	0.87	2.68	26.1	0.227	43	163.67	165.20	0.9	2.87	25.37	0.48
KT014	C401038	165.20	166.73	1.84	2.8	41.2	0.269	43.5	165.20	166.72	1.69	2.46	41.83	0.55
KT014	C401039	166.73	167.94	1.09	0.52	25.3	0.136	31.4	166.72	167.94	0.96	0.33	18.51	0.34
KT024	C401040	281.09	282.06	2.81	1.06	98.6	0.419	33.3	281.08	282.21	3.35	1.16	97.03	0.41
KT024	C401041	282.06	283.43	0.04	0.02	1.6	0.028	20.3	282.21	283.43	0.07	0.02	0.69	0
KT024	C401042	300.44	302.42	0.03	0.01	2	0.019	34.1	300.44	302.42	0	0	0	0
KT025	C401044	81.11	82.72	1.81	3.64	32.3	0.309	40.1	81.10	82.72	2.61	3.41	34.29	0.41
KT025	C401045	82.72	84.73	2.61	3.92	27.8	0.33	40.3	82.72	84.73	2.61	4	14.4	0.34
KT025	C401050	84.73	86.78	1.06	6.61	11.3	0.16	36.4	84.73	86.77	0.96	6.1	2.06	0.17
KT027	C401046	169.77	170.93	2.57	9.62	58.8	0.574	13.15	169.92	170.93	2.36	10.2	72.69	0.96
KT027	C401047	170.93	171.82	0.35	1.46	5.6	0.068	6.05	170.93	171.81	0.25	1.22	8.23	0.07
KT027	C401048	171.82	173.52	1.8	3.94	51.7	0.096	5.66	171.81	173.52	1.92	5.06	87.77	0.17
KT027	C401049	173.52	174.32	2.76	2.31	17.4	0.151	13.45	173.52	174.31	2.7	2.12	17.14	0.1

Table 6.1 Data for Historical SML Core Intersection Resample (continued...)

Hole ID	Sample ID	From (m)	To (m)	resampled					S %	From (m)	To (m)	historic					
				Cu %	Zn %	Ag ppm	Au ppm	Cu %				Zn %	Ag ppm	Au ppm			
KT033	C401051	195.07	196.29	4.45	6.91	65.3	0.418	41.7		195.07	196.29	4.16	6.62	53.49	0.34		
KT035	C401055	147.22	147.89	0.1	0.09	2.6	0.039	8.27		147.21	147.88	0.09	0.04	15.09	0.1		
KT035	C401052	147.89	149.08	1.81	2.6	277	0.617	17.75		147.88	149.07	1.64	3.21	178.97	0.75		
KT035	C401053	149.08	151.12	4.7	4.94	148	1.36	21.7		149.07	151.11	2.52	3.36	146.74	1.17		
KT035	C401054	151.12	152.16	0.03	0.07	2.6	0.028	9.37		151.11	152.15	0.02	0.04	0	0		
KT038	C401056	277.06	278.28	1.87	1.39	6.4	0.037	37.9		277.06	278.28	1.64	1.2	2.4	0.1		
KT038	C401057	278.28	279.81	0.9	2.82	9.2	0.076	46.1		278.28	279.80	0.74	2.92	7.54	0.1		
KT038	C401058	279.81	281.18	0.7	2	10	0.153	45		279.80	281.17	0.63	1.88	8.23	0.17		
KT038	C401059	281.18	281.79	1.66	0.44	7.1	0.035	7.56		281.17	281.78	1.54	0.41	6.17	0.07		
KT038	C401060	281.79	282.85	0.06	0.16	1	0.015	2.13		281.78	282.85	0.05	0.1	2.06	0.03		
KT039	C401062	151.42	152.06	2.25	0.46	62.1	0.331	3.93		151.42	152.06	2.52	0.28	82.97	0.41		
KT039	C401063	152.06	152.95	0.66	0.21	15.4	0.032	1.87		152.06	152.94	0.33	0.32	2.74	0.03		
KT039	C401064	152.95	153.50	1.74	0.58	41.6	0.509	8.68		152.94	153.49	2.14	0.12	43.2	0.1		
KT039	C401065	153.50	153.95	1.52	13.3	38.7	0.233	15.3		153.49	153.95	1.86	12.1	42.51	0.1		
KT039	C401066	153.95	154.87	0.1	3.5	2.9	0.051	4.43		153.95	154.86	0.1	2.53	0.69	0		
KT039	C401067	154.87	155.75	0.15	1.61	2.4	0.039	3.26		154.86	156.75	0.58	2.87	2.74	0.03		
KT039	C401069	162.67	163.04	1.98	0.08	56.3	0.344	4.85		162.67	163.03	3.55	0.12	201.6	1.23		
KT039	C401070	163.04	163.98	2.34	0.29	51.5	0.357	17.65		163.03	163.98	2.3	0.3	40.46	0.17		
KT039	C401071	163.98	165.51	0.48	1.57	5	0.08	5.69		163.98	165.50	0.48	1.63	4.8	0.03		
KT039	C401072	165.51	167.03	0.2	2.48	1.6	0.035	3.21		165.50	167.03	0.17	2.02	0.34	0		
KT039	C401073	167.03	168.43	0.32	0.87	3.8	0.09	7.72		167.03	168.43	0.33	0.72	3.43	0.03		
KT039	C401074	168.43	170.51	0.73	0.3	10.3	0.187	37.4		168.43	170.50	0.63	0.36	9.6	0.1		
KT039	C401075	170.51	170.99	0.02	0.04	1.2	0.03	13.75		170.50	170.99	0.02	0.02	2.06	0.03		
KT040	C401076	157.70	158.86	1.73	3.49	31.7	0.294	41.2		157.70	158.86	1.74	3.26	32.23	0.17		
KT040	C401077	158.86	160.05	1.05	4.34	26.9	0.276	42.3		158.86	160.05	1.04	4.43	21.26	0.1		
KT040	C401078	160.05	161.12	0.5	1.96	10.1	0.146	34		160.05	161.11	0.47	1.95	10.29	0.27		
KT040	C401080	161.12	162.40	2.76	2.28	39.4	0.386	31		161.11	162.39	2.6	2.29	34.97	0.34		
KT040	C401081	162.40	163.68	3.87	1.87	52.3	0.615	26.7		162.39	163.67	3.87	3.58	40.46	0.17		
				216.73	359.62	4459.06	45.28							211.40	337.68	4267.25	42.45
mean				y	1.58	2.62	32.55	0.33	x					1.55	2.48	31.38	0.31

Table 6.2 WKM Standards analyses within SML re-sampled core

Standard Type	Hole-ID location	Sample ID	Cu %	Zn %	Ag ppm	Au ppm	S %
WK280001	KT049	186413b	1.70	2.66	18	0.18	43
WK280001	KT082	186459b	1.71	2.74	17	0.20	40.1
WK280001	KT040	C401082	1.73	2.71	18	0.19	39.1
Rec. Value			1.63+.1	2.72+.2	16+2	0.19+.02	40.73+4.1
WK280002	KT075	186422b	1.51	1.67	20	0.12	29.2
WK280002	KT046	186468b	1.47	1.74	20		30.9
WK280002	KT014	C401036	1.49	1.69	19	0.15	28.5
WK280002	KT009	KT0906	1.48	1.74	20	0.16	29.5
Rec. Value			1.45+.10	1.71+.10	19+2	0.14+.04	29.52+2.02
WK280003	KT094	186431b	0.78	1.30	13		18.3
WK280003	KT046	186483b	0.77	1.38	13	0.11	19.9
Rec. Value			0.73+.05	1.40+.10	13+2	0.11	19.07+2.33

Table 6.3 Canadian Resource SRM analyses within SML re-sampled core

Standard Type	Hole-ID Location	Sample ID	Cu %	Zn %	Ag ppm	Au ppm	S %
Cdn High	KT075	186422a	5.18	2.24	109	2.12	20.6
Cdn High	KT082	186459a	4.99	2.36	113	1.92	22.6
Cdn High	KT046	186483a	4.99	2.32	116	1.99	18.6
Cdn High	KT010	C401026	4.97	2.27	108	1.745	21.4
Cdn High	KT039	C401068	5.12	2.29	109	1.89	20.9
Cdn High	KT040	C401083	5.25	2.38	113	2.04	21.4
Certified Value			5.07+.27	2.35+.11	111+8.6	1.97+0.22	n/a
Cdn Low	KT049	186413a	1.52	2.95	75	0.789	29.9
Cdn Low	KT094	186431a	1.54	2.95	76	0.804	26.5
Cdn Low	KT046	186468a	1.51	3.08	71	0.702	30.2
Cdn Low	KT001	C401008	1.49	2.95	69	0.722	28.3
Cdn Low	KT024	C401043	1.52	2.99	71	0.813	28.3
Certified Value			1.49+.06	3.01+.17	65+7	0.83+.12	n/a

Table 6.4 Values for Zn blanks and previous samples

Sample ID	Cu ppm	Cu previous ppm
186407	112	29500
186419	452	23700
186430	275	100500
186454	724	18600
186466	74	30100
186479	420	35900
C401025	110	17000
C401035	1090	6200
C401061	98	592
C401079	309	4990
KT09-5	103	13100

Table 6.5 Values for Zn blanks and previous samples

Sample ID	Zn ppm	Zn previous ppm
186407	422	57700
186419	1110	28300
186430	262	20900
186454	562	11200
186466	56	8710
186479	2260	15000
C401025	417	9440
C401035	1740	23000
C401061	185	1600
C401079	1355	19600

Table 7.1 Data for Study of Variation in Core Sampling 2006 (...)

Hole ID	Sample ID	From (m)	To (m)	resampled					originally sampled				
				Cu %	Zn %	Ag ppm	Au ppm	S %	Cu %	Zn %	Ag ppm	Au ppm	S %
WK06-74	186521	51.8	52.4	0.00	0.02	0.3	0.009	1.44	0.01	0.02	0.2	0.007	1.5
WK06-74	186522	52.4	52.9	1.22		13.8	0.148		1.19		12.6	0.164	
WK06-74	186523	52.9	54.1	0.02	0.04	0.4	0.009	1.52	0.01	0.04	0.4	0.005	1.4
WK06-74	186524	54.1	55.3	0.02	0.05	0.7	0.007	1.29	0.03	0.06	1.1	0.006	1.38
WK06-74	186525	55.3	56.5	2.30		30.5	0.148		2.82		33	0.157	
WK06-74	186526	56.5	57.9	1.92		36.5	0.462		2.06		42.3	0.543	
WK06-74	186528	57.9	59.3	1.69		28.3	0.384		1.83		32.7	0.45	
WK06-74	186529	59.3	60.6	1.31		21.2	0.324		1.26		22	0.306	
WK06-74	186531	60.6	62.0	1.20		21.5	0.357		1.25		21	0.334	
WK06-74	186532	62.0	64.0	1.46		27.8	0.316		1.41		29.4	0.36	
WK06-74	186533	64.0	65.3	1.58		24.1	0.322		1.66		26.7	0.316	
WK06-74	186534	65.3	66.7	1.82		37.5	0.418		1.62		37.9	0.468	
WK06-74	186535	66.7	67.4	0.07	0.25	1.1	0.017	3.63	0.05	0.15	0.8	0.01	3.09
WK06-74	186536	67.4	68.9	2.32		32.7	0.305		2.13		28.2	0.273	
WK06-74	186537	68.9	70.1	0.11	0.08	0.7	0.027		0.12	0.07	0.7	0.043	
WK06-74	186538	70.1	71.8	0.03	0.01	0.7	0.018		0.07	0.04	1.2	0.017	
WK06-74	186539	71.8	73.4	0.07	0.76	2.1	0.018		0.06	0.77	2.5	0.014	
WK06-74	186540	73.4	75.0	0.80		6.6	0.079		0.90		6.3	0.078	
WK06-74	186541	75.0	75.5	0.54	0.79	13.8	0.145	5.05	0.54	0.63	13.1	0.104	4.25
WK06-74	186542	75.5	76.2	0.01	0.04	0.8	0.061	2.79	0.02	0.06	0.9	0.061	2.67
WK06-75	186543	53.6	54.7	0.99	1.12	16.3	0.254		0.99	1.15	17	0.241	
WK06-75	186544	54.7	55.4	2.61	2.76	27.5	0.368		2.30	2.74	27.9	0.394	
WK06-75	186545	55.4	57.6	0.15	0.22	2.9	0.04		0.22	0.27	3.7	0.042	
WK06-75	186546	57.6	58.6	1.68	3.11	37.9	1.05		1.65	3.01	35.4	0.296	
WK06-75	186547	58.6	59.7	1.88	5.28	34.6	0.289		2.11	4.75	42.2	0.304	
WK06-75	186548	59.7	61.1	1.43	4.55	24.2	0.401		1.66	5.07	28	0.233	
WK06-75	186550	61.1	62.1	1.67	6.49	24.5	0.293		1.71	6.51	25.4	0.344	
WK06-75	186551	62.1	63.0	1.75	4.42	28.2	0.48		1.87	3.53	34.3	0.489	

Table 7.1 Data for Study of Variation in Core Sampling 2006 (...)

Hole ID	Sample ID	From (m)	To (m)	resampled					originally sampled				
				Cu %	Zn %	Ag ppm	Au ppm	S %	Cu %	Zn %	Ag ppm	Au ppm	S %
WK06-75	186554	63.0	64.3	2.23	3.37	26.7	0.355		2.26	3.33	28.2	0.339	
WK06-75	186555	64.3	65.3	1.57	4.42	27.5	0.397		1.62	4.19	29.7	0.376	
WK06-75	186556	65.3	66.6	1.24	4.50	17.2	0.267		1.35	4.30	19.6	0.279	
WK06-75	186557	66.6	67.8	0.97	4.47	14.8	0.242		0.94	5.06	16.9	0.251	
WK06-75	186558	67.8	69.2	0.97	3.98	23.5	0.358		2.74	4.24	27	0.402	
WK06-75	186559	69.2	70.4	0.93	0.20	6.1	0.093		0.98	0.17	7.1	0.102	
WK06-75	186560	70.4	72.0	1.32	0.12	11.6	0.225		1.27	0.12	12.8	0.152	
WK06-75	186561	72.0	73.3	0.08	0.04	1.7	0.034		0.12	0.10	2.4	0.045	
WK06-75	186562	73.3	74.8	0.04	0.02	2.3	0.03		0.05	0.02	2.4	0.055	
WK06-75	186563	74.8	76.3	0.03	0.69	1.6	0.02		0.03	0.75	1.7	0.015	
WK06-75	186564	76.3	77.0	0.34	1.76	1.7	0.094		0.32	1.23	2.2	0.029	
WK06-75	186565	77.0	78.5	0.72	0.95	12.3	0.231		0.68	0.86	12.7	0.185	
WK06-75	186567	78.5	79.2	1.41	0.32	26.5	0.224		1.38	0.27	26.8	0.398	
WK06-75	186569	79.2	80.6	1.60	0.31	28.9	0.496		1.81	0.34	33.4	0.524	
WK06-75	186570	80.6	81.9	0.59	0.14	4.8	0.155		0.65	0.12	4.6	0.17	
WK06-75	186571	81.9	82.5	0.07	0.01	0.9	0.04	9.83	0.05	0.01	0.6	0.028	9
WK06-76	186573	56.4	57.1	0.84	0.67	17.6	0.211		0.72	0.52	17.1	0.171	
WK06-76	186574	57.1	57.9	0.87	0.18	9.2	0.093		0.66	0.18	2.6	0.061	
WK06-76	186575	57.9	59.0	0.23	0.11	4.5	0.058	6.84	0.11	0.12	3	0.053	5.69
WK06-76	186576	59.0	60.8	1.50	0.61	19.7	0.186		1.24	0.53	15	0.17	
WK06-76	186577	60.8	62.5	0.24	0.10	4.1	0.041	5.86	0.23	0.10	3.5	0.046	5.59
WK06-76	186578	62.5	64.0	0.06	0.07	0.8	0.018	3.59	0.05	0.06	0.9	0.022	3.74
WK06-76	186579	64.0	65.6	1.82	0.27	5.8	0.053	3.99	1.70	0.23	4.9	0.046	3.49
WK06-76	186580	65.6	67.0	3.38		36.1	0.526		3.67		44.5	0.557	
WK06-76	186582	67.0	67.5	1.76	0.18	24.4	0.252		0.46	0.14	5.9	0.092	
WK06-76	186583	67.5	68.7	1.86		21.7	0.294		1.82		25.2	0.359	
WK06-76	186586	68.7	69.5	1.59		55.3	0.365		2.43		38.7	0.287	
WK06-76	186587	69.5	70.8	0.15	0.22	1.4	0.025	1	0.09	0.20	1.3	0.02	1.09

Table 7.1 Data for Study of Variation in Core Sampling 2006 (...)

Hole ID	Sample ID	From (m)	To (m)	resampled					originally sampled				
				Cu %	Zn %	Ag ppm	Au ppm	S %	Cu %	Zn %	Ag ppm	Au ppm	S %
WK06-76	186588	70.8	72.4	1.91		40.4	0.252		1.55		33.1	0.301	
WK06-76	186589	72.4	73.6	0.16	0.22	2.5	0.044	8.82	0.13	0.14	1.9	0.04	8.3
WK06-76	186590	73.6	74.9	0.32	0.08	4.2	0.079	7.9	0.31	0.17	3.8	0.077	7.75
WK06-76	186591	74.9	76.4	0.23	0.03	2.7	0.044	7.74	0.42	0.03	4.3	0.064	7.62
WK06-76	186592	76.4	77.8	1.68	0.38	20.5	0.489		1.83	0.30	23.1	0.411	
WK06-76	186593	77.8	79.2	0.47	0.03	4.1	0.127	9.13	0.58	0.02	5.1	0.146	9.16
WK06-78	186615	81.0	83.0	0.03	0.28	0.4	0.012	1.9	0.02	0.07	0.6	0.011	1.33
WK06-78	186616	83.0	83.5	4.25	5.57	4.6	0.241		5.05	5.20	5.9	0.286	
WK06-78	186617	83.5	84.0	2.41	2.29	48.8	0.353		1.88	2.54	43.3	0.314	
WK06-78	186619	84.0	85.3	1.39	7.66	40.5	0.707		1.43	6.32	50	0.659	
WK06-78	186620	85.3	87.1	0.15	0.58	3.2	0.091	4.39	0.30	1.23	5	0.113	5.2
WK06-78	186621	87.1	87.8	1.91	12.10	28.8	0.409		2.29	12.70	32.8	0.426	
WK06-78	186623	87.8	89.0	0.36	0.27	8	0.134		0.39	0.32	8.8	0.145	
WK06-78	186624	89.0	90.3	0.23	0.13	4.7	0.155		0.25	0.13	4.9	0.128	
WK06-78	186625	90.3	91.3	0.80	0.16	15	0.209		0.82	0.14	15.8	0.205	
WK06-78	186626	91.3	92.2	1.62	1.20	20.4	0.266		1.81	1.18	23.4	0.261	
WK06-78	186627	92.2	93.5	0.57	2.23	4.5	0.148		0.63	2.29	4.2	0.13	
WK06-78	186628	93.5	94.5	1.27	2.16	15.7	0.102		1.30	2.18	16.3	0.183	
WK06-78	186629	94.5	96.1	1.25	2.94	15.8	0.209		1.21	3.19	17.2	0.202	
WK06-78	186630	96.1	97.4	1.90	5.03	23.8	0.268		2.14	5.49	25.1	0.264	
WK06-78	186631	97.4	98.7	3.56	8.26	35.7	0.344		3.67	8.71	40.3	0.258	
WK06-78	186632	98.7	99.5	0.68	0.26	10.3	0.19		0.80	0.26	10.8	0.176	
WK06-78	186633	99.5	100.6	0.25	0.52	2.1	0.083		0.24	0.76	2.1	0.09	
WK06-78	186634	100.6	101.7	0.19	0.06	1.5	0.064	9.24	0.18	0.08	1.5	0.07	8.47
WK06-79	186635	78.6	79.3	0.04	0.11	2.7	0.037	3.26	0.04	0.14	2.9	0.04	3.34
WK06-79	186636	79.3	79.9	5.75	7.46	26.6	0.215		4.48	8.48	31.5	0.272	
WK06-79	186637	79.9	81.0	0.04	0.15	3.4	0.104	4.74	0.04	0.20	3.5	0.102	4.84
WK06-79	186638	81.0	82.3	3.76	3.37	27.8	0.305		3.97	3.00	30.4	0.194	

Table 7.1 Data for Study of Variation in Core Sampling 2006 (...)

Hole ID	Sample ID	From (m)	To (m)	resampled					originally sampled				
				Cu %	Zn %	Ag ppm	Au ppm	S %	Cu %	Zn %	Ag ppm	Au ppm	S %
WK06-79	186640	82.3	83.7	8.60	4.13	211	1.575		8.53	5.14	199	1.535	
WK06-79	186641	83.7	84.8	0.29	0.11	2.4	0.022		0.23	0.12	2.1	0.012	
WK06-79	186642	84.8	86.0	0.15	1.67	9.4	0.215		0.17	1.42	8.2	0.191	
WK06-79	186645	86.0	87.4	0.06	0.17	1.9	0.125		0.07	0.22	2.2	0.154	
WK06-79	186646	87.4	88.4	3.35	0.38	51.1	1.205		3.48	0.54	56.6	1.135	
WK06-79	186647	88.4	89.6	1.71	7.75	21.8	0.23		1.85	8.13	24	0.228	
WK06-79	186648	89.6	90.7	3.02	3.92	37.3	0.345		3.93	3.61	44.7	0.381	
WK06-79	186649	90.7	92.1	1.91	2.61	29.8	0.394		2.01	2.86	30.7	0.37	
WK06-79	186650	92.1	93.1	5.34	5.70	72.7	1.85		5.71	5.02	75.7	1.67	
WK06-79	186651	93.1	94.6	1.57	12.95	27	0.332		1.57	14.05	26.1	0.313	
WK06-79	186652	94.6	95.1	0.47	5.64	6.5	0.103		0.63	5.61	8.6	0.114	
WK06-79	186653	95.1	95.9	0.13	0.57	2.4	0.041		0.12	0.34	1.9	0.028	
WK06-79	186654	95.9	97.5	2.21	6.29	31.3	0.358		2.42	6.93	34.7	0.363	
WK06-79	186655	97.5	99.4	1.97	4.62	24.5	0.27		2.22	4.84	28.7	0.36	
WK06-79	186656	99.4	99.9	0.57	0.38	11	0.168		0.57	0.10	12.1	0.174	
WK06-79	186657	99.9	100.6	0.22	0.46	2.6	0.098		0.25	0.81	3.3	0.056	
WK06-79	186658	100.6	101.8	0.14	0.18	1.6	0.055		0.13	0.12	1.6	0.029	
WK06-79	186659	101.8	102.8	0.26	0.03	1.8	0.095		0.27	0.04	2.3	0.114	
WK06-79	186660	102.8	104.1	0.06	0.01	0.4	0.037		0.08	0.01	0.6	0.038	
mean y (resampled) vs. x (orig. sampled)				1.25	2.03	18.17	0.25	4.95	1.29	2.06	18.72	0.23	4.71

Table 7.2 Original and resample Cu blanks and previous samples

original Sample ID	Blank Type	Cu ppm	Cu previous	resample Sample ID	Cu ppm	Cu previous
186549	Blank L	201	16600	186530	338	13100
186581	Blank L	251	36700	186639	50	37600
186639	Blank L	198	39700	186549	424	14300
186527	Blank P	298	20600	186566	38	7160
186566	Blank P	78	6840	186584	131	15900
186618	Blank P	244	18800	186618	159	24100

Table 7.3 Original and resample Zn blanks and previous samples

original Sample ID	Blank Type	Zn ppm	Zn previous	resample Sample ID	Zn ppm	Zn previous
186549	Blank L	525	50700	186530		
186581	Blank L	290	46300	186639	59	33700
186639	Blank L	190	30000	186549	1155	45500
186527	Blank P	619	57900	186566	102	9500
186566	Blank P	130	8630	186584		
186618	Blank P	309	25400	186618	193	22900

Table 7.4 Analyses for WKM Standards included in Drill-core 'duplicate' samples.

original								resampled							
Standard	Insertion	Sample	Cu	Zn	Ag	Au	S	Standard	Insertion	Sample	Cu	Zn	Ag	Au	S
Type	Point	ID	%	%	ppm	ppm	%	Type	Point	ID	%	%	ppm	ppm	%
WK280001	WK0678	186622	1.81	2.75	16	0.18	>10.0	WK280001	WK0678	186622	1.71	2.61	15	0.19	n/a
WK280001	WK0674	186530	1.76	2.77	17	0.18	>10.0	WK280001	WK0679	186644	1.68	2.64	15	0.18	n/a
WK280001	WK0676	186584	1.74	2.78	17	0.19	>10.0	WK280002	WK0675	186552	1.51	1.72	19	0.15	n/a
WK280002	WK0679	186643	1.57	1.68	20	0.14	>10.0	WK280002	WK0679	186643	1.51	1.65	17	0.14	n/a
WK280002	WK0675	186552	1.50	1.70	19	0.15	>10.0	WK280002	WK0676	186585	1.49		20	0.15	n/a
WK280003	WK0675	186568		1.36	13	0.10	>10.0	WK280003	WK0675	186553	0.77	1.39	12	0.10	n/a
								WK280003	WK0675	186568	0.78	1.32	12	0.10	n/a
Average value for 280001 combined:			1.74	2.71	16	0.19									
Two Standard Deviations:			0.10	0.16	1.7	0.01									
Average value for 280002 combined:			1.52	1.69	19	0.15									
Two Standard Deviations:			0.06	0.06	2.4	0.01									
Average value for 280003 combined:			0.77	1.36	12	0.10									
Two Standard Deviations:			0.02	0.07	1.4	0.01									

Table 8.1 Check Analyses on Duplicate Pulp by Eco Tech...

Tag #	y Eco Tech					x Chemex				
	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)
186208	1.86	3.70	44.0	0.55	47.6	1.84	3.74	51.0	2.48	50.2
186216	1.54	11.3	54.0	0.33	30.7	1.56	11.35	56.0	0.33	32.3
186233	4.50	1.09	58.7	0.56	27.5	4.42	1.12	64.0	0.71	26.5
186234	0.15	0.14	2.9	0.04	15.5	0.16	0.16	3.0	0.02	15.1
186249	0.59	1.31	14.1	0.51	49.7	0.61	1.49	17.0	0.56	49.3
186264					0.2					0.1
186269	1.00	0.28	29.8	0.41	10.1	0.92	0.27	17.0	0.68	10.1
186273	0.02	0.03	0.8	0.08	28.6	0.03	0.03	1.0	0.07	27.5
186280	3.18	1.14	96.0	1.19	31.4	3.11	1.20	104.0	1.14	31.0
186298	1.51	2.30	17.8	0.46		1.49	2.29	21.0	0.52	
186305	5.90	6.80	76.0	1.32	15.3	5.92	6.77	106.0	1.48	18.1
186306	0.06	0.18			0.8	0.14	0.32			1.1
186311	0.82	0.43	7.3	0.07	8.6	0.83	0.44	8.0	0.09	9.1
186325	4.70	6.70	53.8	0.66	47.4	4.80	6.77	57.0	0.73	46.4
186330	2.20	3.60	2.3	0.03	5.9	2.11	3.79	3.0	0.03	5.6
186331	0.55	5.40	13.9	0.24	14.4	0.55	5.38	14.0	0.19	15.3
186342	0.65	0.03	8.9	0.07		0.02	0.06	0.9	0.06	
186335	0.96	2.70	30.8	0.79	37.8	0.94	2.73	34.0	0.82	36.6
186354	1.22	2.90	9.5	0.20	47.2	1.24	2.98	11.0	0.27	44.6
186358	1.71	6.00	46.7	0.29	48.6	1.69	6.16	49.0	0.37	47.2
186366	0.54	0.56	8.7	0.18	37.5	0.56	0.58	10.0	0.20	38.8
186387	0.61	6.90	7.6	0.12	47.6	0.65	7.08	9.0	0.14	48.9
186388	1.05	3.80	7.9	0.13	48.9	1.07	3.87	9.0	0.15	49.3
186394	0.11	0.06	3.6	0.05	44.5	0.12	0.07	4.0	0.06	44.2
186503	8.20	3.50	19.6	0.29	39.2	8.28	3.51	24.0	0.58	37.6
186505	1.42	2.50	26.9	0.25	46.4	1.49	2.49	32.0	0.33	47.0
186532	1.41	9.20	26.2	0.30		1.41	9.73	29.4	0.36	
186541	0.53	0.81	13.1	0.16	4.3	0.54	0.63	13.1	0.10	4.3
186580	3.20	4.30	36.2	0.50		3.67	4.63	44.5	0.56	
186582	1.76	0.17	21.6	0.25	10.1	0.46	0.14	5.9	0.09	6.0
186591	0.22	0.02	2.4	0.08	7.0	0.42	0.03	4.3	0.06	7.6

Table 8.1 Check Analyses on Duplicate Pulp by Eco Tech (continued...)

Tag #	y Eco Tech					x Chemex				
	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)
186605	1.77	1.68	32.6	0.44		1.90	1.91	35.6	0.31	
186607	0.44	0.69	5.4	0.11		0.46	0.67	5.6	0.12	
186631	3.30	8.60	33.2	0.28		3.67	8.71	40.3	0.26	
186632	0.69	0.29	10.5	0.20		0.80	0.26	10.8	0.18	
186638	3.50	3.50	26.6	0.33		3.97	3.00	30.4	0.19	
186640	8.10	4.20	206	1.90		8.53	5.14	199.0	1.54	
186667	0.46	4.50	2.0	0.15	29.2	0.54	4.52	2.7	0.14	33.1
186672	3.60	0.51	77.4	1.06	38.2	3.73	0.50	82.2	0.72	39.1
186681	0.29	0.53	23.5	0.41	13.3	0.30	0.51	23.8	0.37	13.7
186682	1.29	5.23	14.4	0.15	31.2	1.37	5.40	17.1	0.15	30.1
186684	1.67	0.11	14.6	0.21	36.4	1.84	0.10	16.3	0.15	36.2
186690	2.20	4.20	55.7	0.57	32.7	2.33	4.18	55.9	0.35	36.0
186715	2.60	2.10	42.0	0.55	33.4	2.79	2.01	43.8	0.65	30.4
186724	0.98	3.80	23.5	0.13	26.5	1.05	3.98	26.6	0.19	24.8
186728	1.47	3.00	65.2	0.79	31.5	1.51	2.99	68.9	0.72	30.0
186736	0.03	0.07	0.7		0.3	0.04	0.08	0.7		0.4
186738	1.62	9.30	52.1	0.54	48.1	1.77	9.50	49.3	0.58	45.7
186740	0.54	0.50	8.9	0.17	3.1	0.57	0.46	9.4	0.14	3.4
186755	1.77	11.0	32.6	0.26	44.0	1.80	11.65	31.5	0.30	42.8
186761	4.30	5.80	97.5	0.54	27.1	4.75	5.91	96.9	0.69	25.5
186766	9.30	2.10	192	1.46	26.9	9.80	1.98	186.0	0.89	25.0
186806	10.60	3.00	20.0	0.70	43.1	10.85	3.02	24.8	0.86	46.4
186807	2.80	4.70	27.4	0.60	47.7	3.02	4.88	28.8	0.62	50.4
186808	2.70	5.35	37.3	0.39	43.1	2.76	6.44	40.8	0.44	46.8
mean value	2.19	3.20	34.8	0.42	29.1	2.24	3.29	36.4	0.46	29.1
	Cu		Zn		Ag		Au		S	
	STDEV x	2.4822	STDEV x	3.1030	STDEV x	40.9939	STDEV x	0.4509	STDEV x	16.4412
	STDEV y	2.3785	STDEV y	2.9952	STDEV y	40.6503	STDEV y	0.3857	STDEV y	16.2535
	m		m		m		m		m	
	(STDy/STDx)	0.9582	(STDy/STDx)	0.9652	(STDy/STDx)	0.9916	(STDy/STDx)	0.8553	(STDy/STDx)	0.9886
	mx	2.1500	mx	3.1745	mx	36.0969	mx	0.3899	mx	28.7633
	b (y-mx)	0.0387	b (y-mx)	0.0220	b (y-mx)	-1.3384	b (y-mx)	0.0341	b (y-mx)	0.3145
	y=0.9582x+0.0387		y=0.9652x+0.0220		y=0.9916x-1.3384		y=0.8553x+0.0341		y=0.9886x+0.3145	

Table 8.2 QC Data from Eco Tech Labs - Repeats

Western Keltic Mines Inc AK7-0218

QC DATA:

Repeat:	Tag #	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)
1	186208	1.84	3.60	42.0	0.59	56.4
3	186233					28.4
6	186249				0.49	
8	186269				0.44	
10	186280	3.11	1.14	96.2	1.10	30.2
12	186305				1.37	14.2
16	186325				0.67	
19	186342	0.67	0.03	8.5	0.07	26.0
20	186335				0.76	
21	186354				0.26	44.7
23	186366					37.9
27	186503				0.33	
29	186532					47.4
31	186580				0.53	
34	186605				0.35	
40	186640				1.80	
43	186672				1.04	
45	186682	1.26	5.29	14.2	0.15	28.7
47	186690				0.49	
48	186715				0.55	33.0
52	186738				0.56	
55	186761				0.58	
56	186766				1.45	
57	186806				0.61	
58	186807				0.58	

Table 8.3 QC Data from Eco Tech Labs – Standards

Western Keltic Mines Inc AK7-0218

QC DATA:

Standard:	Tag #	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)
S125					1.80	
S125					1.81	
S125					1.77	
S125					1.80	
Cu120		1.52		34.2		
Cu120		1.54		34.2		
Pb106		0.63	0.84	58.3		
Pb106		0.63	0.85	58.4		
KZK-1						0.803
KZK-1						0.789

APPENDIX B

MEMORANDUM ON SPECIFIC GRAVITY DETERMINATIONS AND ESTIMATIONS FOR THE KUTCHO PROJECT

Memorandum on Specific Gravity Determinations and Estimations for the Kutcho Project

For

Western Keltic Mines Inc.
600-888 Dunsmuir Street,
Vancouver, B.C. V6C 3K4

By

Peter Holbek, M.Sc., P.Geo.
Viking GeoScience
&
Rob Wilson, P.Geo.
Western Keltic Mines Inc.

March, 2007

Introduction

The Kutcho project consists of three volcanogenic massive sulphide deposits located in north central British Columbia. Exploration of the area began in the late 1960's and the discovery drill holes were completed in 1984. Discovery and deposit delineation was carried out more or less simultaneously by Esso Minerals Canada (EMC) and Sumac Mines Ltd., the Canadian subsidiary of Sumitomo Metal Mining Canada Ltd. (SML). EMC and SML carried out exploration and delineation drilling from 1984 through to 1989, when EMC sold its claims to Homestake Canada Ltd., which was subsequently bought by Barrick in 1998. Western Keltic Mines Inc., (WKM) purchased both sets of claims from Barrick and SML in 2004 and carried out diamond drilling programs annually from 2004 to 2006. Each company dealt with the Specific Gravity determinations of the mineralization in different ways. This paper attempts to describe and evaluate the different approaches such that the best values can be used going forward.

Historical Work

In the initial stages of drilling on the Kutcho project, neither Esso Minerals Canada, (EMC) nor Sumac Mines Limited, (SML) conducted specific gravity (SG) or density work on assay samples. By the time SML got to drill-hole number 40 they realized that SG was both variable and would have a significant impact on tonnage determination of the resource estimates. SML had Chemex Labs (now ALS Chemex) of North Vancouver perform density measurements on holes KT040 through KT049. Thereafter, SML conducted SG determinations in the field using a "weigh in water, weigh in air" set up, or in some occasions had the laboratory carry out SG determinations on un-crushed core. Additionally, in 1979 (?), SML went back to the old core and carried out specific gravity measurements, in the field, on the main or 'significant grade' parts of the intersections in most, but not all of the holes drilled prior to hole 40. As well, SML also carried out sulphur analyses on all core samples from drill-hole 41 and on. There are, therefore, 7 SML drill-holes (KT11-13, 15, 17, 18 and 30) that do not have measured SG or sulphur analyses from which to calculate SG; these holes, as well as the 'edges' of other intersections have estimated SG values (designated in the 'SGe' column of the database).

EMC was given the Chemex SG measurements from drill-holes KT040-49 and subsequently began to investigate drill-hole densities. Initially, EMC conducted a variety of calculations based on estimated pyrite content and base metal grades to determine the average SG on full drill-hole intersections, as this would be used in manual, polygonal or sectional resource estimates (undated, (but probable 1979) document located in EMC's engineering files). The exact formula varied depending if the resource was being estimated in Vancouver, Calgary or Houston (Exxon Research). EMC did not measure SG's until drill-holes E083 and 84 (metallurgical laboratory measurements) and did not have sulphur content analyzed until this time. Consequently, there are 36 drill holes in the Main deposit that have estimated, as opposed to calculated or measured SG values (E001-E014, E031-40, E044-54, E062, E067 and E068).

Methodologies for SG Estimation

The SG estimation method for SML holes is reasonably well documented in Appendix III of the 1985 WEL authored prefeasibility study. This method essentially uses geology for the estimate. SML subdivided the sulphide zone into five ore types (massive ore - >70 sulphide; semi-massive ore – sulphide between 40 and 70%; banded ore – sulphide between 10 and 40%, low sulphide or ‘intercalated’ ore – sulphide between 5 and 10%; and quartz-carbonate ore – sulphide <10%. Further the deposit was subdivided into three stratigraphic zones (A, B and C) which correspond to the hanging wall zone – A; the central part – B, and the lower, low-grade massive footwall zone - C. SML subdivided the 96 holes in the database with measured SG and estimated the average SG’s for each of the ore-types within each of the zones and derived the following matrix:

	Massive Ore	Semi-mass	Banded	Intercalated	Qz-carb
Zone A	4.12	3.56	2.95	2.81	2.90
Zone B	4.37	3.84	3.14	2.83	3.11
Zone C	4.31	3.79	3.18	2.87	N/A

The above values were then assigned to holes without SG data on the basis of ore type and position within the deposit. This estimation is a reasonable approach given the nature of the SML drill logs/information available. In comparison to drill-holes with measured SG’s, the estimated SG’s tend to be slightly lower for the high sulphide samples, and slightly higher for the low sulphide material.

The WEL prefeasibility study reports that the above estimation method was also used for the EMC drill-holes lacking both SG measurements and sulphur analyses. However, examination of the data base (originally obtained from WEL in 1986) indicates that while the above estimation method was used for most of the EMC holes it was not used for all. At least two methods of estimation are apparent within the database. There are 51 EMC holes within the database in or near the Main deposit. Twelve holes have sulphur analyses (E083-84, and E098-118) and four (6?) of these holes also have lab measured SG’s (E098-101; lab certs available; E083-84 possible lab measurements but no lab certs and questionable data). Twenty-two drill holes (E001-14, E031-34, E036, and E049-53) had SG’s estimated on the basis of lithology according to the method described in Appendix III of the Wright Engineers pre-Feasibility study. The remaining 17 holes contain estimated SG’s that are presumed to be estimated by EMC, based on visual pyrite estimations recorded in drill logs and base metal grades.

The SG estimation methodology used by EMC is not clearly documented. There are two ‘historical’ memos documenting various attempts by EMC to estimate SG’s. Both memos that list actual data have SG values for the total sulphide intersection (as this would be used in polygonal or sectional style resource estimation). Working backwards from the data, it is inferred that the same or similar methods were used to estimate SG on

an assay interval basis. A key item in the EMC memos is a graph based on the data from KT041-49, where measured SG is plotted against % sulphide minerals, which produces a straight line relating SG against % sulphides. To get total sulphide minerals, EMC appears to have used the visually estimated pyrite content and other sulphide phases, and may have used a formula to convert Cu and Zn assays to sulphide percentages where these values are high and only pyrite was visually estimated. Examination of the drill logs indicates that some logs record an estimate of the pyrite and the other sulphide phases separately whereas other logs just estimate the total % of sulphide minerals. The use of copper and zinc assays would be more important for the Esso deposit where the base metal contents are commonly double or triple the values in the Main deposit and the pyrite content of the Esso deposit is 30 to 50% what it is in the Main deposit. Within the Main deposit there are only a small percentage of cases where an estimate of SG based on pyrite content would be significantly different than an estimated based on pyrite + other sulphide phases.

A number of drill logs have been examined and an estimate of the pyrite or total sulphide content has been made (Table 1). The repetition of many SG values in the EMC database suggests that a chart was likely drawn up with total sulphide values in 5 or 10% increments and the corresponding SG value based on the above described graph. Thus 0-5% Sx equated to an SG of 2.7, 5-10% = 2.95, 10-20% = 3.01, 20-30%=3.18, 30-40%=3.23, 40-50%=3.45, 50-60%=3.84, 60-70%= 4.11, 70-80%=4.33, 80-90=4.55 >90%=4.77. This type of system appears to match the data fairly well as indicated in Table 2, however in some cases there are SG values with more varied numbers (E058 for example – which is actually a drill-hole in the Sumac deposit) and these possibly represent the use of the Cu and Zn assays. Use of the Cu and Zn assays seems not to make a significant difference to the SG estimate (at least for the pyrite rich Main and Sumac deposits). There is a very good correlation between visual sulphide estimate and the SG value assigned. Because visual estimates of sulphide mineralization are not done on the same intervals as the assay samples there is some leeway in how the sulphide percentages are interpreted for the assay intervals. There are at least two cases where the assay interval has been divided within the database to compensate for a dramatic change in the visual estimation of sulphide concentration that occurs within the assay interval.

It should also be noted that samples on the edge of the ore zones that did not have SG data from EMC at the time of the pre-feasibility study were assigned SG's on the basis of numerous tests that SML performed on hanging wall and footwall samples. It is inferred that this is why low grade samples within or adjacent to the ore zone, in some EMC drill-holes have a slightly different values (eg: 2.79) than those predicted from sulphide content.

Evaluation of SG Estimation

Basic statistics for the SG data within the data base is shown below:

Table 1 STATISTICS							
Database : ktres06b							
Table : Assay							
Parameter	SG	SGm	SGe	SGc	SGe*	SGc*	SGe
-----					Main	Main	Esso
Number of samples	4395	2864	715	2922	295	140	127
Minimum	2.60	2.60	2.60	2.72			
Maximum	5.40	5.40	4.86	4.98			
Mean	3.464	3.549	3.286	3.451	3.43	3.43	3.14
Variance	0.433	0.465	0.323	0.390			
Standard deviation	0.658	0.681	0.568	0.624			
Skewness	0.636	0.383	1.300	0.604			
Curtosis	1.957	1.670	3.374	1.957			
Coeff. of variation	0.189	0.191	0.172	0.180			

*EMC data Main deposit (E001-E069)

(SG = best choice of all methods, SGm= measured SG, SGe= estimated SG and SGc= calculated SG)

Of the 715 estimated SG values in the database about 295 are in the EMC data and 293 in the SML data of the Main deposit with the remainder being in the Esso deposit (which has a lower overall SG). As can be seen from the above table the mean values for the estimated SG's are in line with the mean values for the other data. Comparison of drill-hole histograms between holes with measured and estimated SG's are included as figures 4 through 9 (back of report). The drill-holes in these figures are proximal to one another (twins in some cases) and demonstrate the general correspondence between the estimated data and measured data (note that the holes are plotted as vertical histograms when in actuality the holes are inclined at various angles). Although the interval thicknesses are not the same between the drill-hole pairs the overall trend in SG corresponds well between the holes.

Having examined many of the drill logs it would appear that the EMC methodology would give a superior SG estimation over the SML method and the methodology is a much better fit for the EMC data and style of drill log.

While it is recognized that estimated SG is not going to match measured SG perfectly, basic statistics and comparison between measured and estimated values within nearby drill-holes, indicate, that on a global basis, the estimated SG is not going to distort the tonnage values within the resource estimations.

Table 2. % Sx is authors estimate of sulphide content from descriptions

*SGe estimated SG – SML method for E001, EMC method for rest.

Hole_ID	From	To	Cu%	Zn%	%SX (int. from logs by PH)	SGe* Database	SGe EMC method by PH
E001	35.05	35.66	0.11	0.79		2.95	
E001	35.66	43.76	0	0		2.81	
E001	43.76	44.34	0.36	2.25		2.95	
E001	44.34	60.35	0	0		2.81	
E001	60.35	63.42	0.4	0.1		2.95	
E001	63.42	66.14	0.65	0.15	80	4.37	4.55
E001	66.14	69.18	0.43	0.4	75	4.37	4.33
E001	69.18	72.23	0.03	0.05	60	3.84	4.11
E001	72.23	75.28	4	0.25	50	3.84	3.84
E001	75.28	78.33	0.08	0.05	30	3.18	3.18
E001	78.33	81.38	0.02	0	25	3.18	3.18
E001	81.38	83.57	0.02	0	35	3.18	3.23
E035	98.8	100.8	2.45	3.32	7	3.01	2.95
E035	100.8	102.8	6.05	2.05	10,31*	3.23	3.23
E035	102.8	103.4	2.57	4.7	5	2.79	2.95
E035	103.4	105.8	2.57	4.7	90	4.77	4.77
E035	105.8	107.5	4.6	12.4	10,38*	3.23	3.23
E035	107.5	109.8	2.15	0.62	25	3.23	3.18
E035	109.8	113.5	1.27	1.95	10	3.01	3.01
E035	113.5	115.2	1.76	7.6	95	4.77	4.77
E035	115.2	116.4	1.76	7.6	95	4.77	4.77
E035	116.4	118.1	3.6	2.05	60	4.11	4.11
E035	118.1	120.4	1.08	0.22	60	4.11	4.11
E035	120.4	121.1	1.08	0.22	5	2.79	2.95
E035	121.1	124.1	0.93	0.09	5	2.79	2.95
E037	111.7	115.7	0.81	0.14	2	2.79	2.7
E037	115.7	117.1	6.55	4.15	80	4.55	4.55
E037	117.1	119.6	3.2	5.72	95	4.77	4.77
E037	119.6	122.6	1.68	2.63	95	4.77	4.77
E037	122.6	125.6	1.68	3.34	95	4.77	4.77
E038	129.4	130.2	0.72	0.45	40	3.45	3.45
E038	130.2	131.7	4.11	5.4	70+	4.33	4.33
E038	131.7	134.1	7.52	1.29	5	2.79	2.95
E038	134.1	137.5	0.89	0.42	5	2.79	2.95
E038	137.5	141.4	0	0	1	2.6	2.7
E038	141.4	142.3	2.09	0.12	50	3.67	3.45- 3.84
E038	142.3	147.1	0	0	15	2.6	3.01
E038	147.1	149.1	0.04	0.11	75	4.33	4.33
E038	149.1	164.4	0	0	20	2.6	3.01
E038	164.4	165.5	0.07	0.48	75+	4.33	4.33
E038	165.5	167	0.07	0.48	75+	4.33	4.33

E038	167	170.1	0.09	0.25	75+	4.33	4.33
E039	129.6	132.1	0.24	0.02	?	3.67	?
E040	129.2	130.36	0.21	0.11		2.95	
E040	130.36	137.61	0	0		2.81	
E040	137.61	141.42	0	0		2.83	
E043	67.9	71.9	1.37	5.25		3.96	
E044	125	127	1.38	2.01	60	3.45	4.11
E044	127	130.2	0.03	0.13	low	2.8	
E044	130.2	131.8	0.07	0.2	10	2.79	2.95
E044	131.8	132.7	7.05	2.09	70	4.33	4.33
E044	132.7	134.7	4.45	0.84	15	3.01	3.01
E044	134.7	137.6	1.02	3.82	15?	3.01	3.01
E044	137.6	140.3	1.87	6.48	75	4.11	4.33
E044	140.3	142.6	1.1	1.12	50	3.89	3.84
E044	142.6	145.2	2.5	3.04	85	4.55	4.77
E044	145.2	147.8	2.42	3.32	85	4.55	“
E044	147.8	148.1	2.25	0.85	85	4.55	“
E044	148.1	150.9	2.25	0.85	85	4.55	“
E044	150.9	153.3	1.11	0.41	85	4.55	“
E044	153.3	155.1	0.71	0.46	80	4.5	
E044	155.1	156.3	0.71	0.46	55	3.67	3.84
E044	156.3	158.9	0.1	0.48	12	3.01	3.01
E044	158.9	161.9	0.22	0.3	30	3.23	3.18
E047	112.65	114.9	0.72	0.21	10	2.95	3.01
E047	114.9	117.95	0.23	0.13	5	2.95	2.95
E047	117.95	121	0.34	0.11	15	3.14	3.01
E047	121	124.05	0.32	0.08	15	3.14	3.01
E048	18.9	21.3	0.03	0.27		2.95	
E048	21.3	23.5	0.23	0.77	40	3.05	3.45
E048	23.5	26.2	0.02	0.69	20	2.9	3.18
E048	26.2	28.2	2.38	7.35	90	4**	4.77
E048	28.2	31.4	2.05	2.15	90	3.6**	4.77
E048	31.4	34.4	0.88	0.18	50	3.84	3.84
E053	144.47	147.82	0.41	0.27	5	2.95	2.95
E053	147.82	148.19	2.65	0.14	18	2.95	3.01
E053	148.19	151.05	0.63	0.23	5	2.95	2.95
E053	151.05	170.07	0	0		2.95	2.7
E053	170.07	185.01	0	0		3.14	2.7
E054	64.2	65.3	0.02	0.18	1	2.8	2.7
E054	65.3	66.4	1.99	2.18	85	4.55	4.55
E054	66.4	69.5	0.32	0.96	9	2.79	2.95
E054	69.5	72.5	0.42	0.41	5	2.79	2.70
E054	72.5	73.4	0.03	0.24	1	2.7	2.70
E054	73.4	76	1.39	3.55	60	3.89	4.11
E054	76	77.4	4.95	2.6	37	3.45	3.23
E054	77.4	79	2.8	3.14	95	4.77	4.77
E054	79	81.1	1.79	1.78	65	4.33	4.11
E054	81.1	82.3	0.41	0.68	15	3.01	3.01
E054	82.3	84.7	1.43	0.21	15	3.01	“
E054	84.7	87.8	0.26	0.24	15	3.01	“
E054	87.8	88.8	0.44	0.1	15	3.01	“

E056	65.37	66.14	0.29	2.31		2.77	
E057	86.6	87.9	0.4	0.8		2.79	
E057	87.9	112.5	0	0		2.8	
E057	112.5	114	3.54	6.94		3.01	
E058	457.35	458.11	2.59	1.25	40	3.46	3.45
E058	458.11	459.18	0.29	3.01	97	4.79	4.77
E058	459.18	460	0.04	0.5	6	2.77	2.95
E058	460	462.68	0.45	2.14	90-95	4.78	4.77
E058	462.68	464.78	0.78	2.04	90-95	4.79	4.77
E058	464.78	465.73	3.68	3.33	90-95	4.81	"
E058	465.73	468.78	1.33	3.31	95plus	4.83	"
E058	468.78	472.13	1.45	2.17	95plus	4.85	"
E058	472.13	474.87	0.58	2.47	95plus	4.86	"
E058	474.87	477.31	0.73	3.14	95plus	4.84	"
E058	477.31	478.81	1.32	3.36	95plus	4.84	"

* pyrite%/total sulphide %

** significant differences here may be to the interpretation of the amount of sulphide mineralization present, a lower estimated SG due to presence of 15% sphalerite may also be a contributing factor but does not account for both samples.

Calculated Specific Gravity

Following SML's investigation into specific gravity measurements, sulphur analyses were carried out on most assay samples beyond drill-hole KT040. EMC did not start doing sulphur analyses until drill-hole E083. Drill holes with the Main zone that do not have measured SG but do have sulphur analysis have had SG values calculated on the basis of the sulphur analysis. SML calculated a formula of $SG = 0.00027S\%^2 + 0.025S\% + 2.72$ based on regression analysis of more than 60 drill holes with measured SG and S analyses. This formula has been used for all Main zone drill holes. EMC used a different formula of $SG = 0.043S\% + 0.01Cu\% + 0.002Zn\% + 2.65$ for Esso deposit drill holes. The appropriateness of these formulas have been checked against WKM's measured SG values and S analyses. A comparison of SG versus total S% (Fig.1, below) shows a good correlation. The correlation is not quite a straight line as the trend flattens at both ends as it reaches a minimum SG and a maximum S content; this is where the SML formula tries to compensate by having the S% squared value included. A comparison of the measured SG versus the calculated SG using the SML formula (SGc, Fig. 2) and the EMC formula (SGcE, Fig. 3) show that the primary difference is in the area of very low sulphur values, where the calculated values tend to be slightly lower than the measured values, otherwise there is very little difference between the two formulas, at least on the basis of the graphs. Given the overall minor component of calculated SG values within the database the above variations are deemed insignificant.

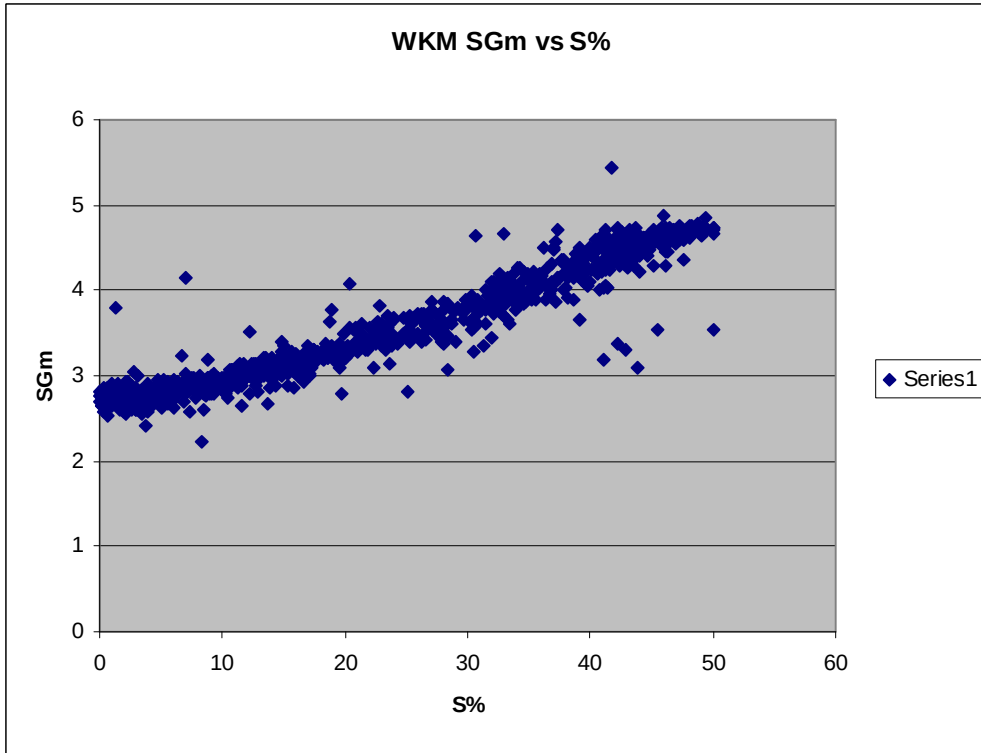


Figure 1. Field measured SG values vs. S% from all WKM data.

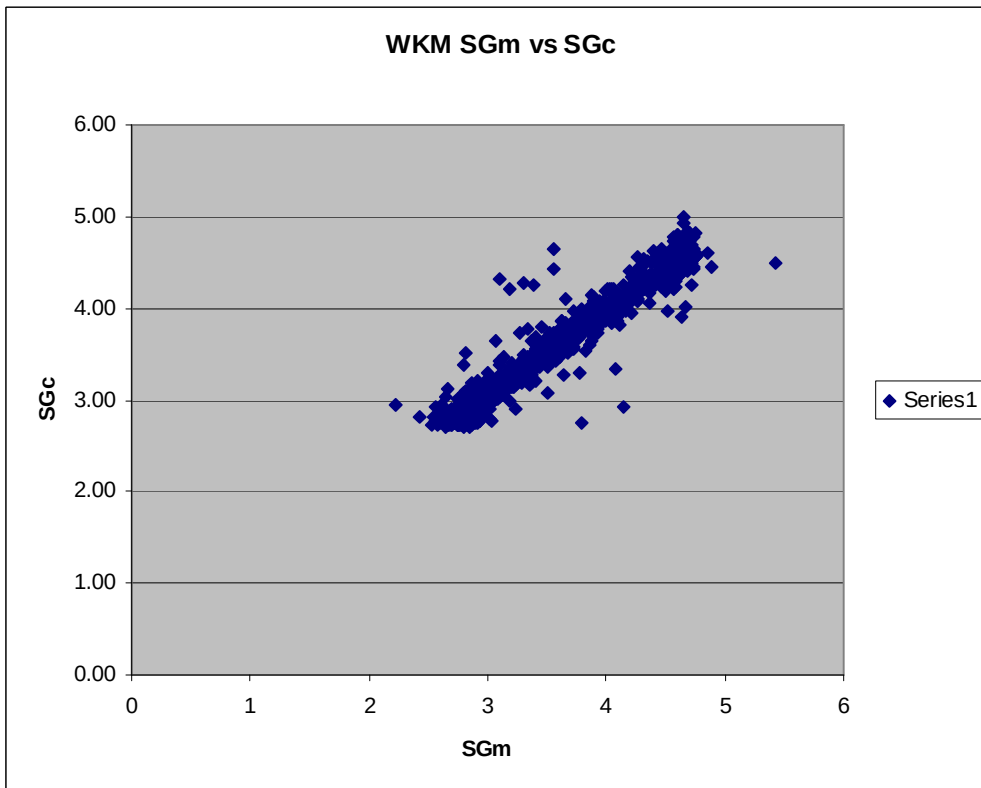


Figure 2. Plot of calculated SG (SML formula) vs. measured SG; all WKM data.

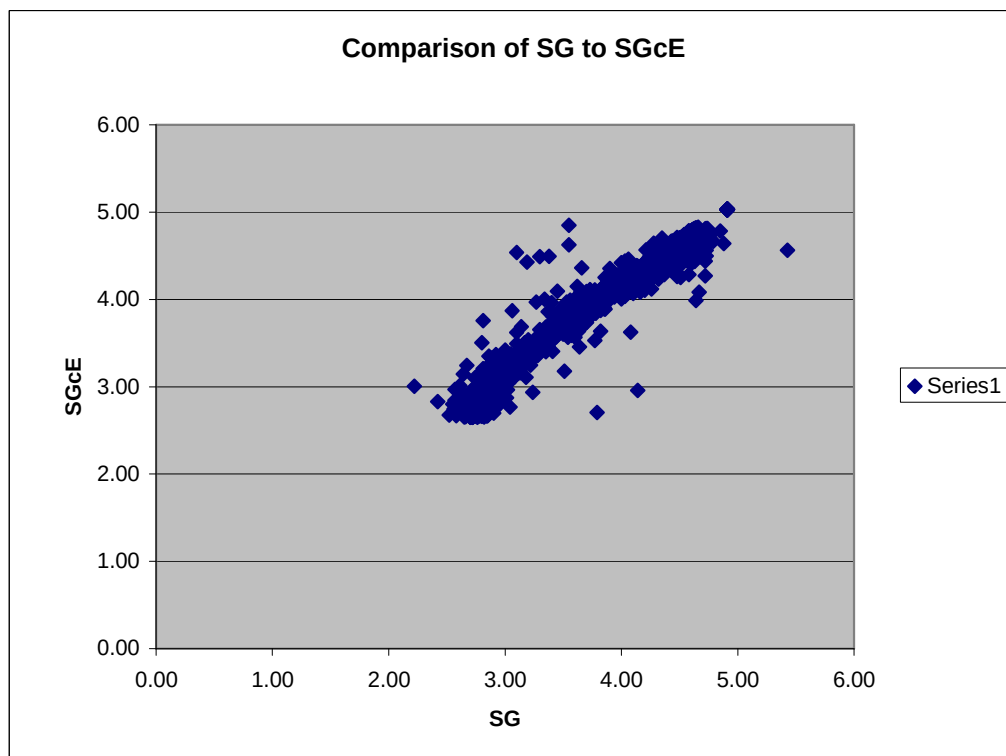


Figure 3. Plot of calculated SG (EMC formula) vs. measured SG, WKM data.

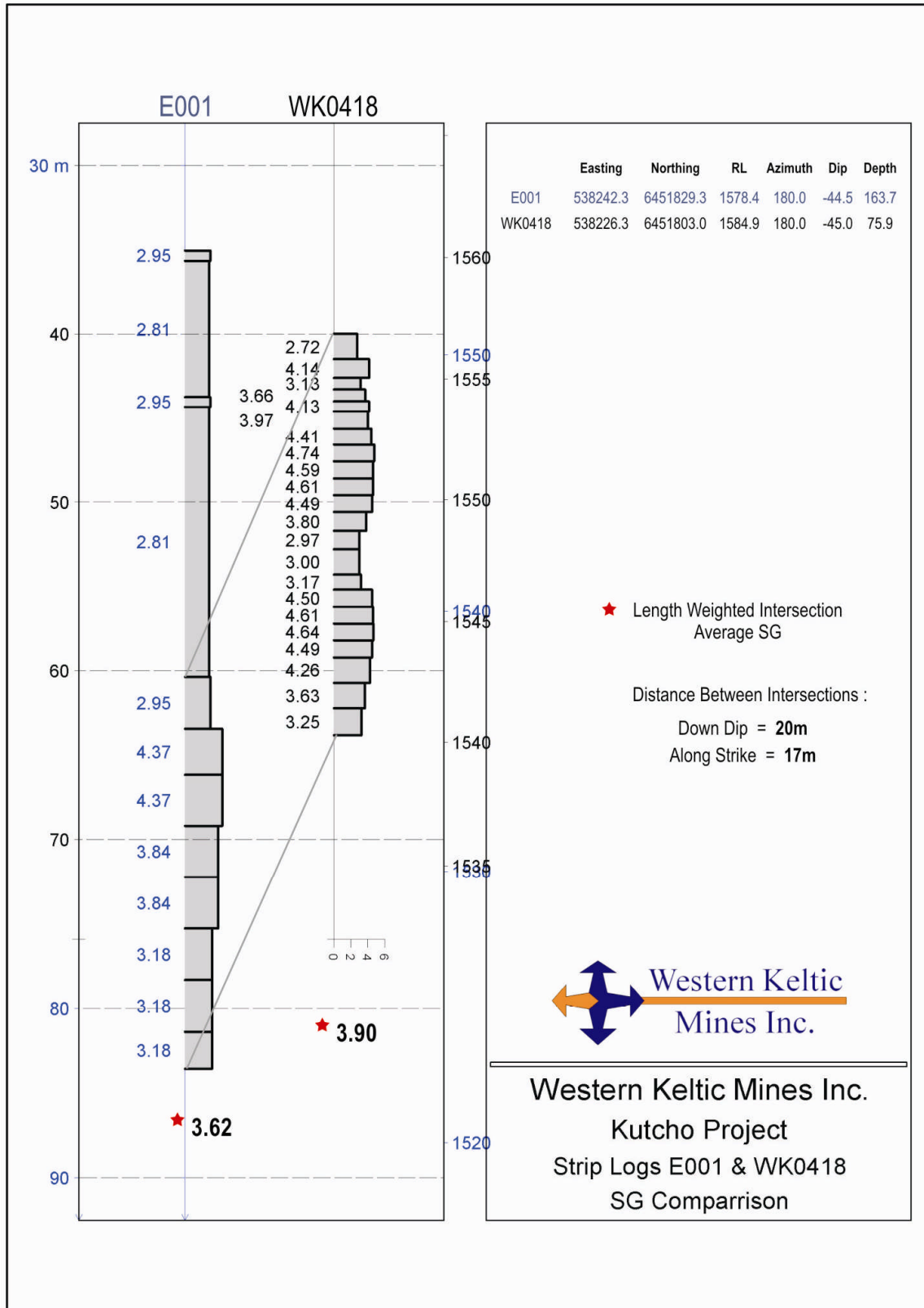


Figure 4.

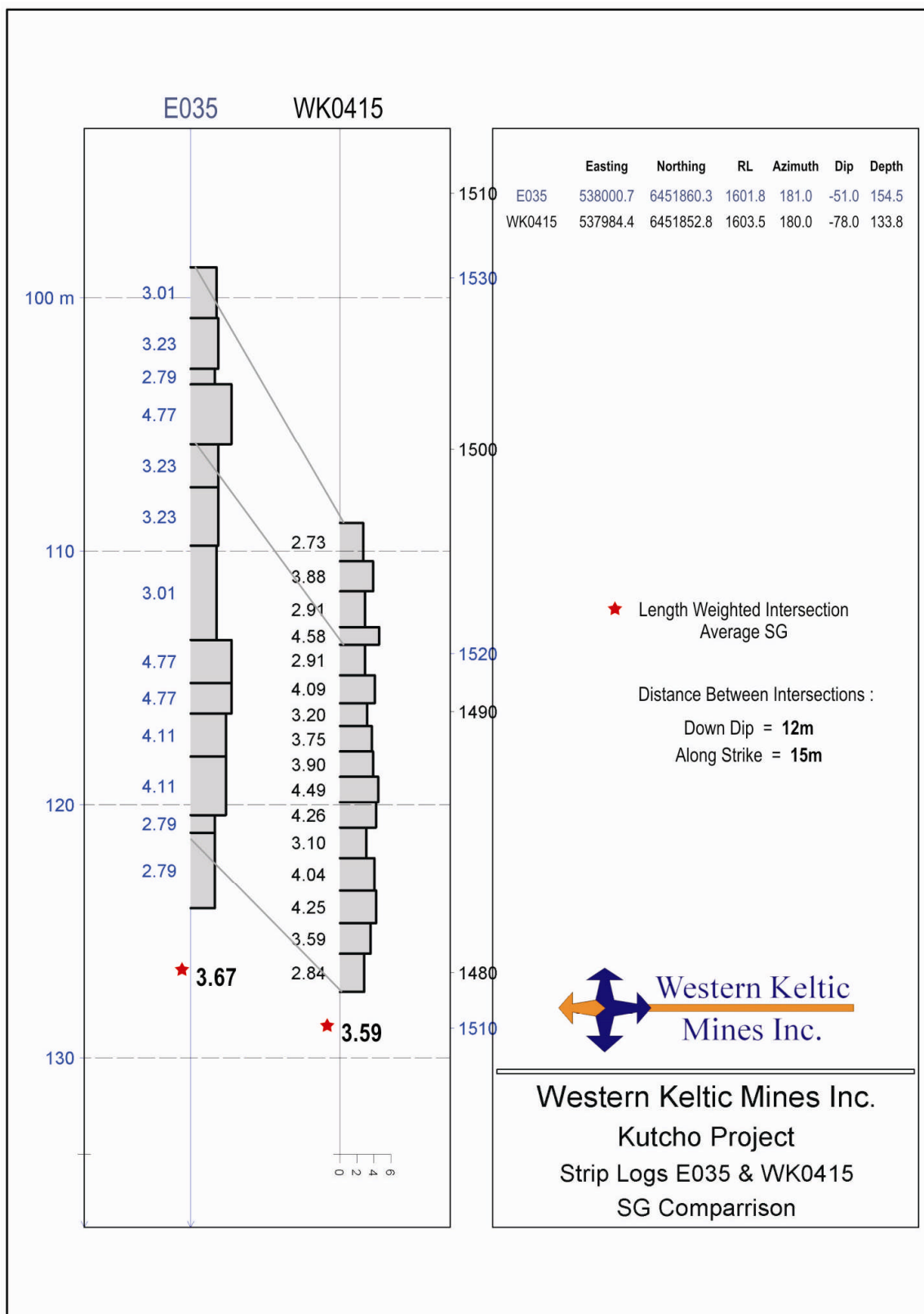
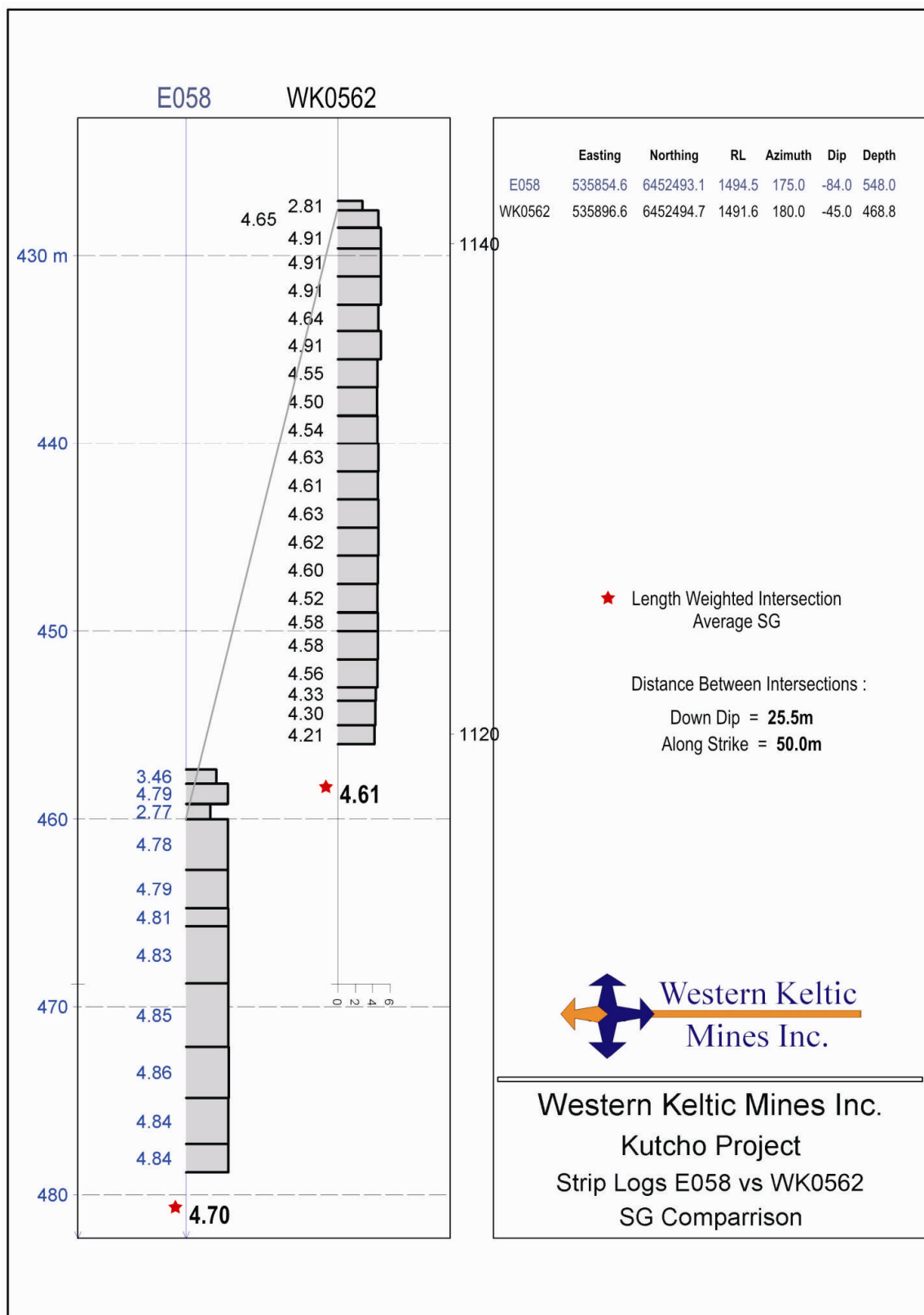


Figure 5.
Figure 6.



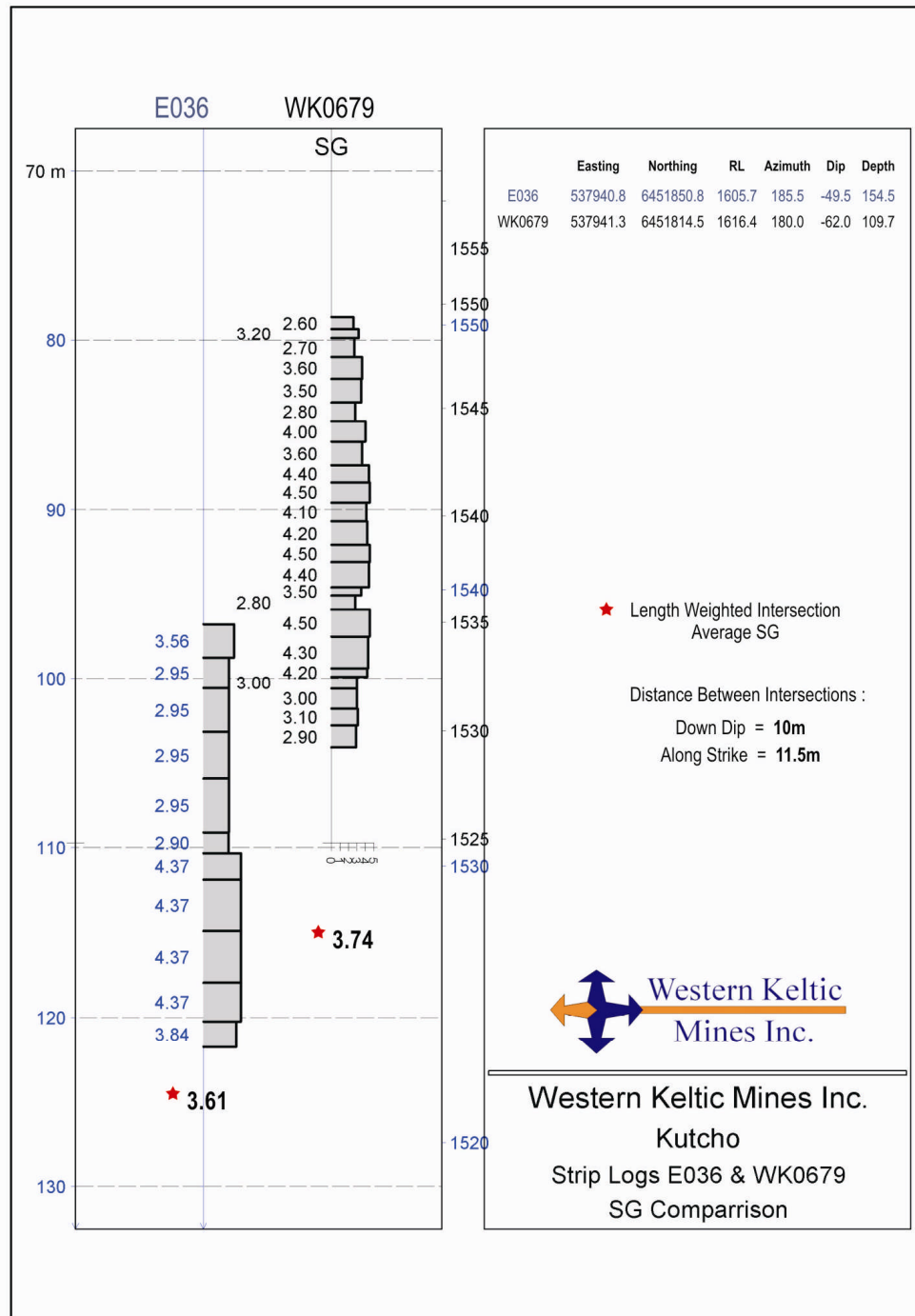


Figure 7.

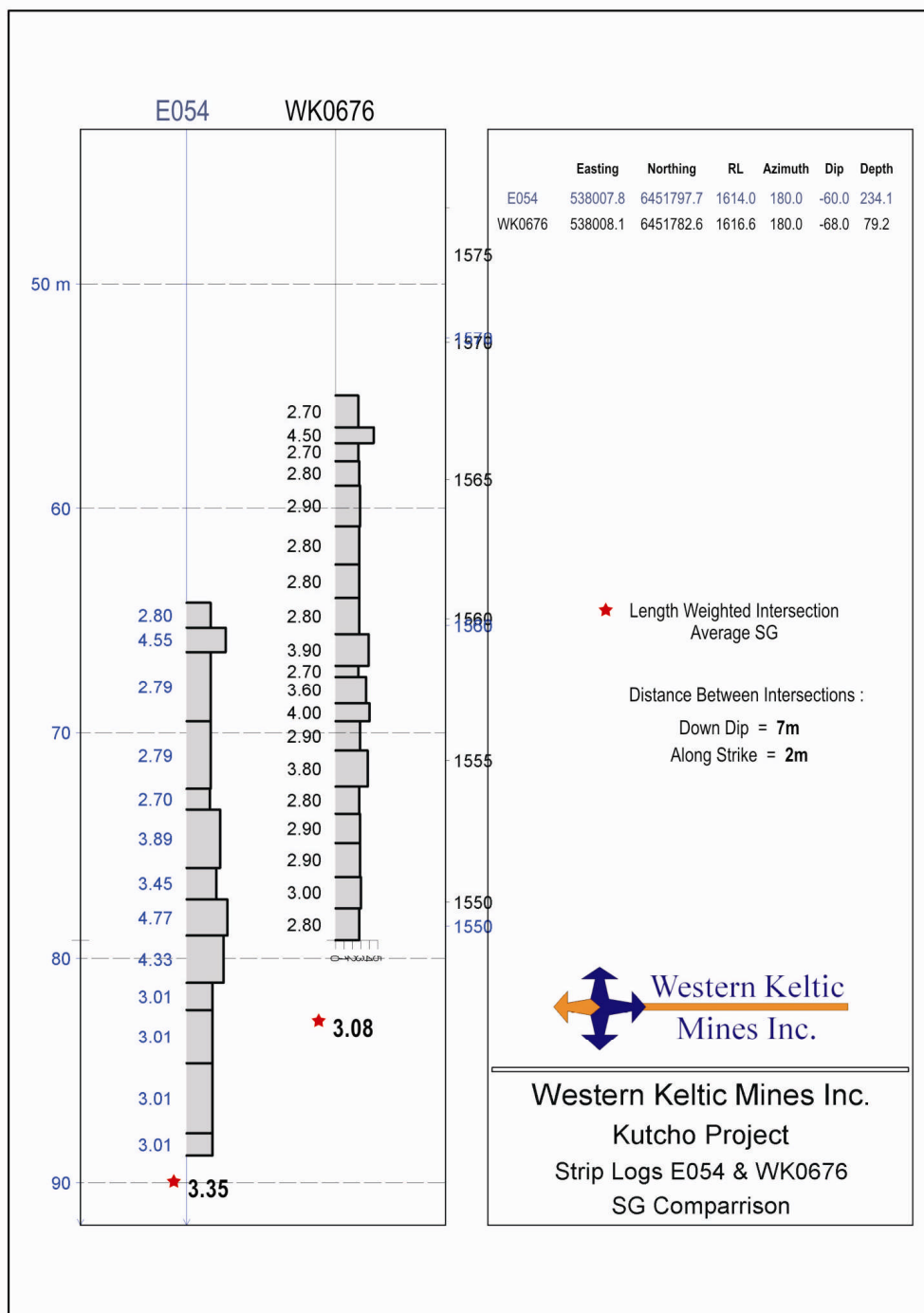


Figure
8.

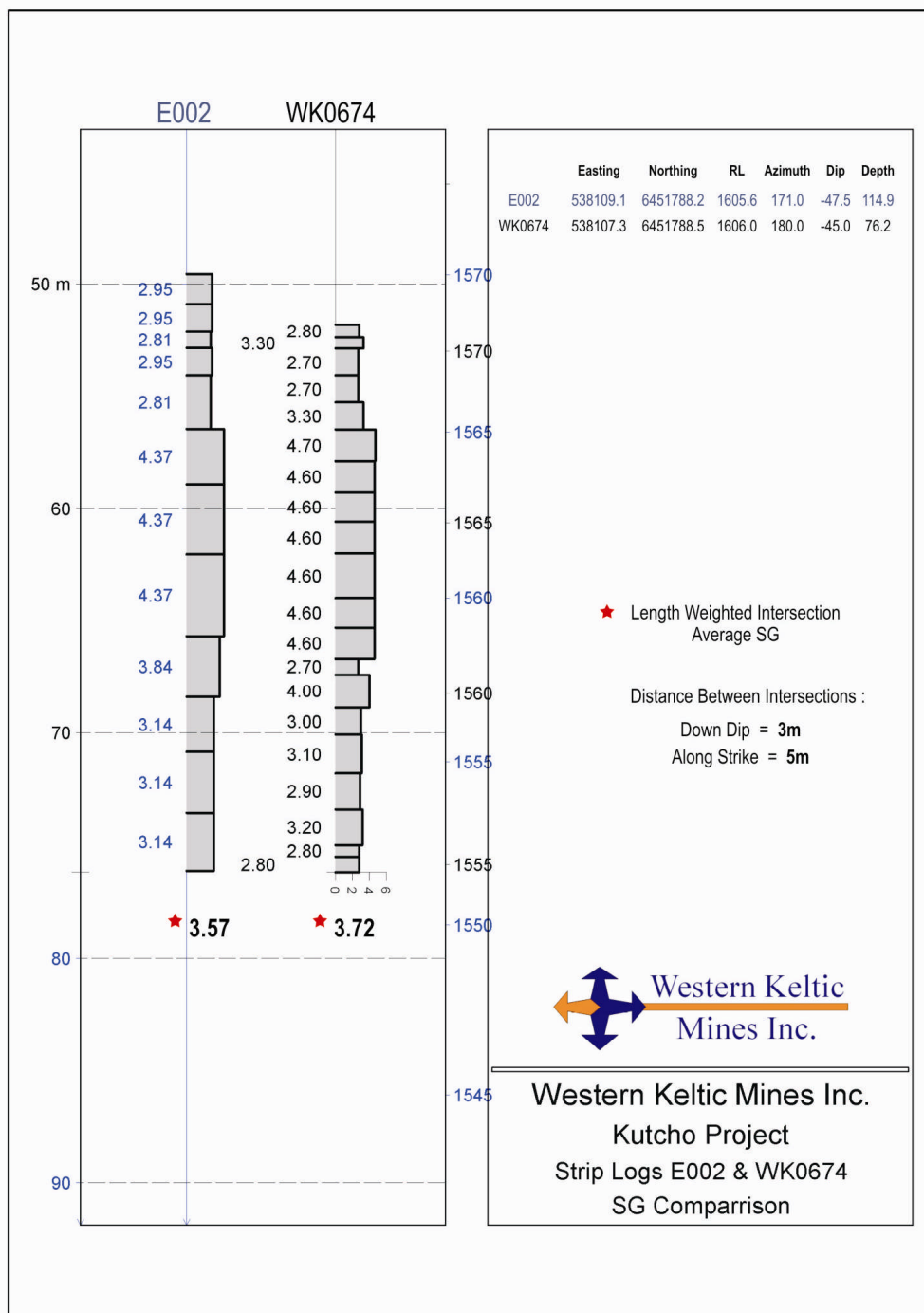
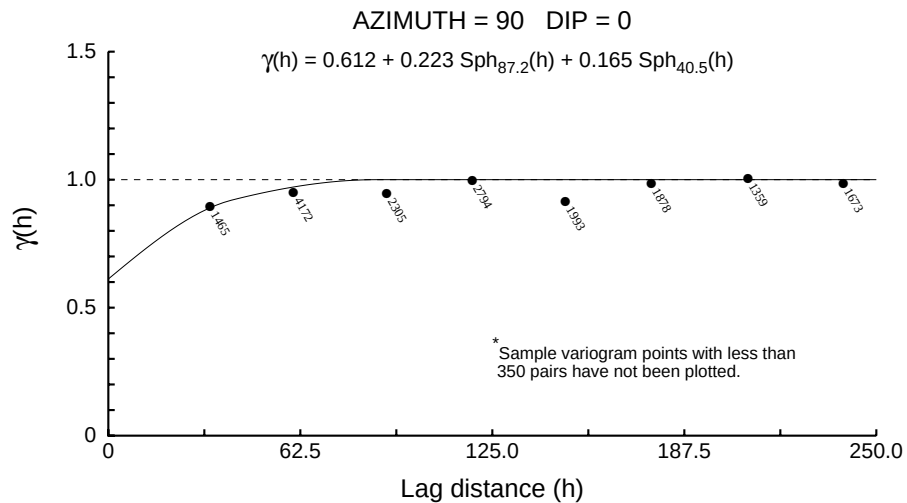
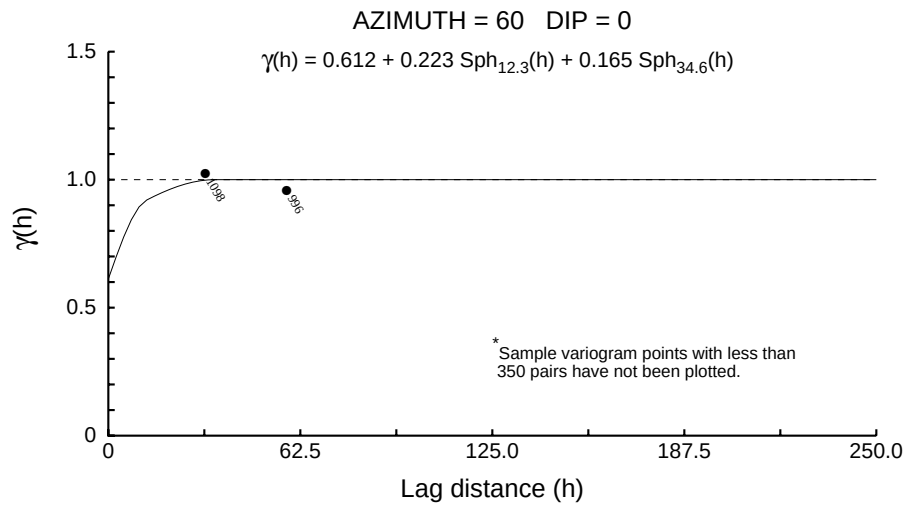
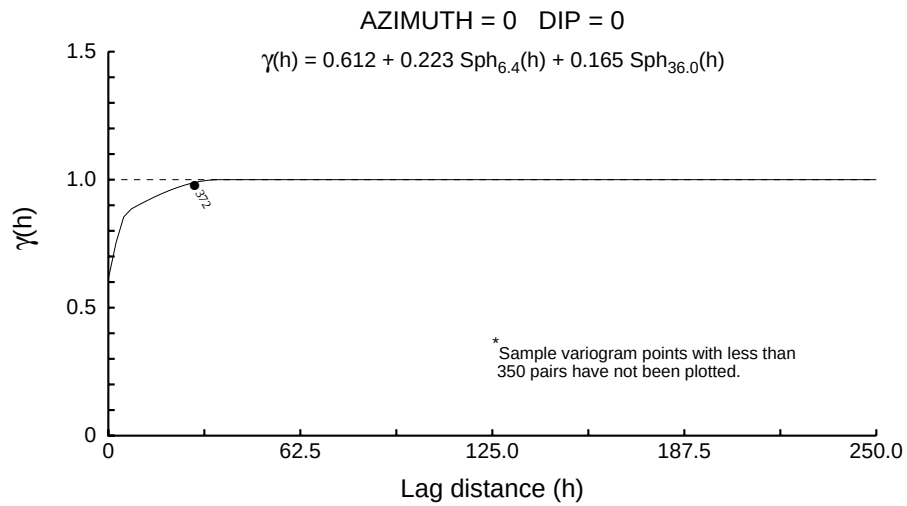


Figure 9.

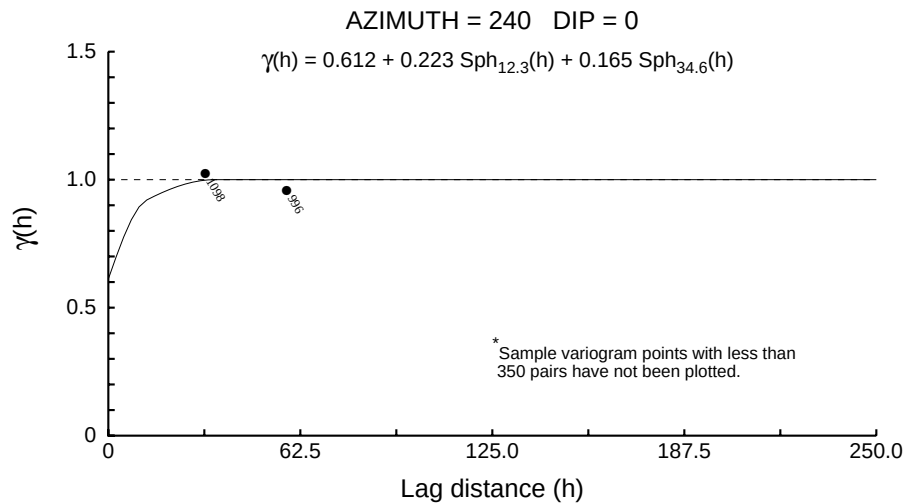
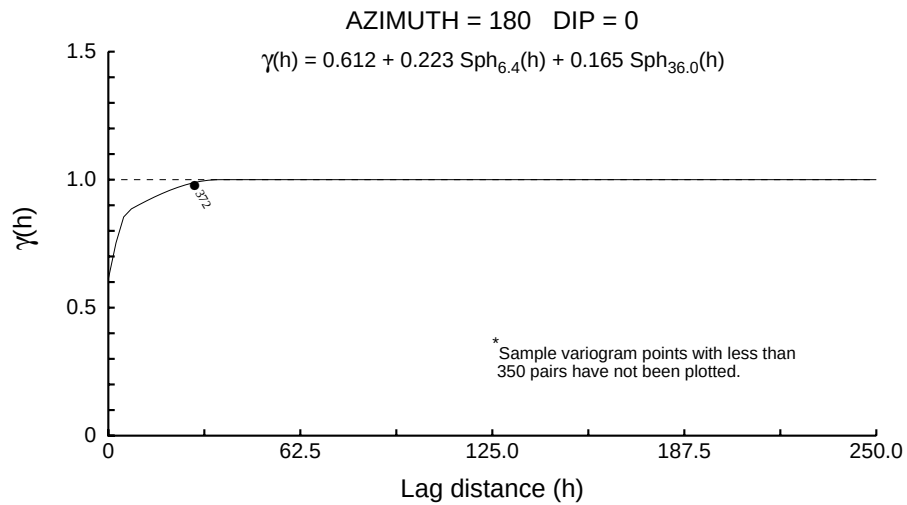
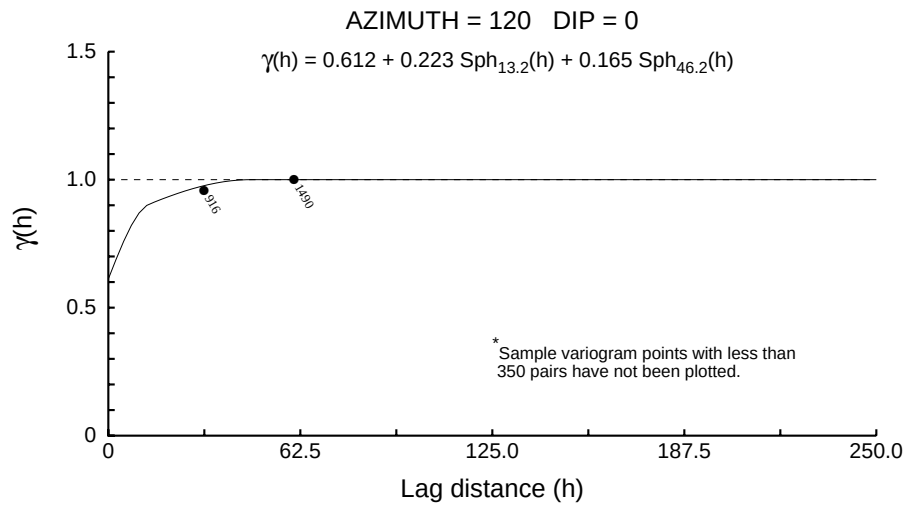
APPENDIX C

V A R I O G R A M S

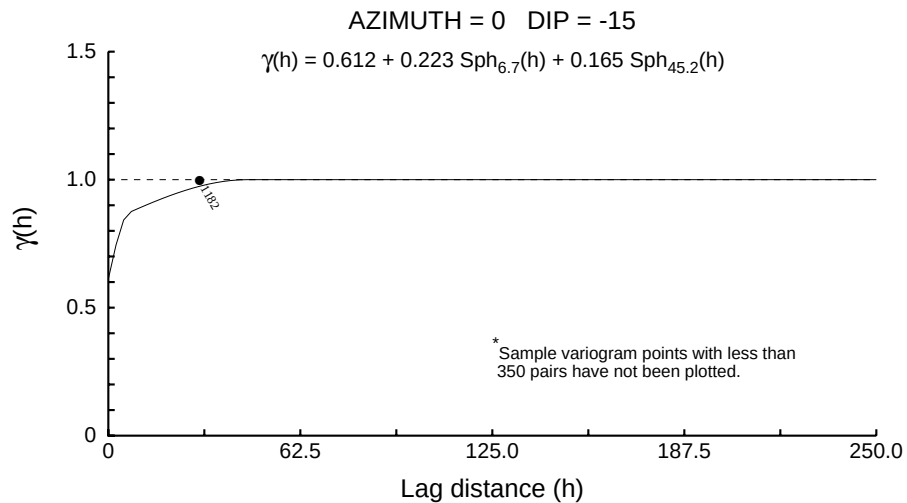
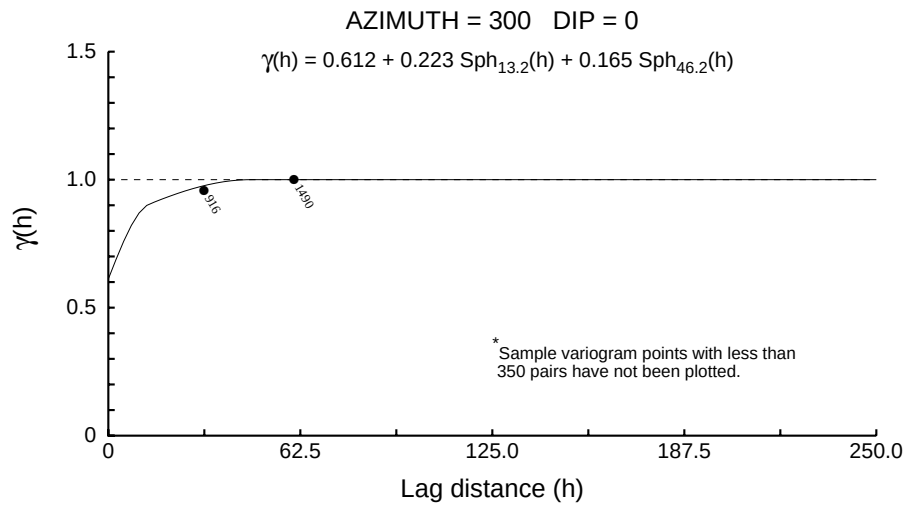
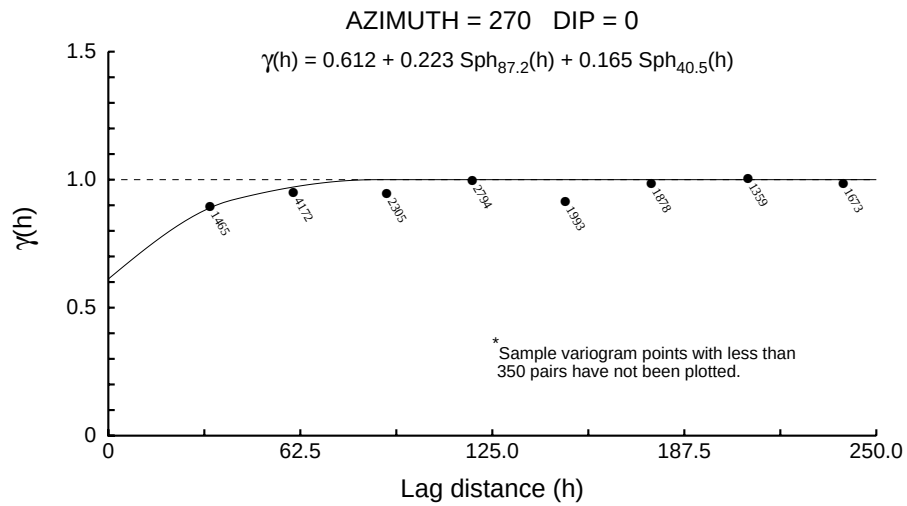
Main Ag 2.5m Comps



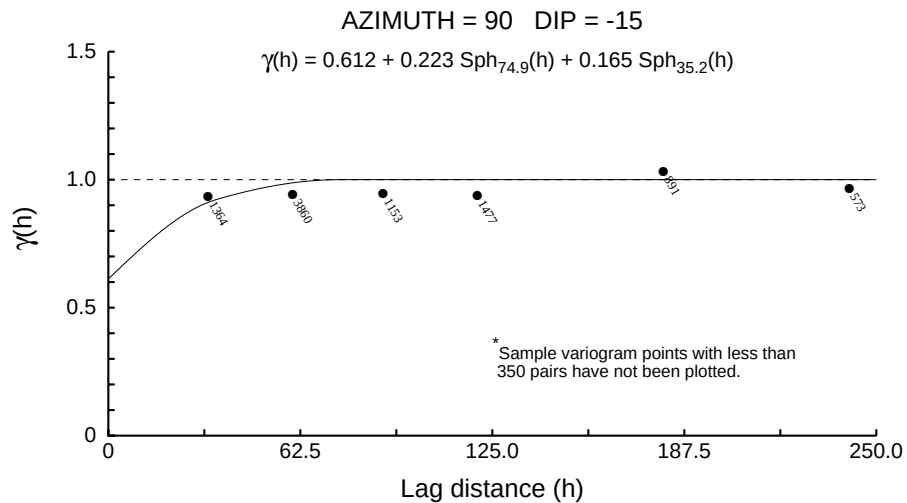
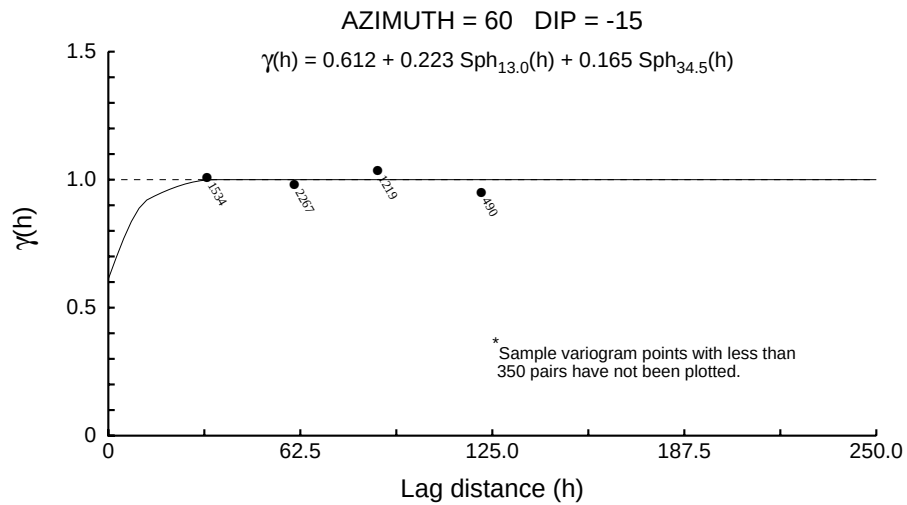
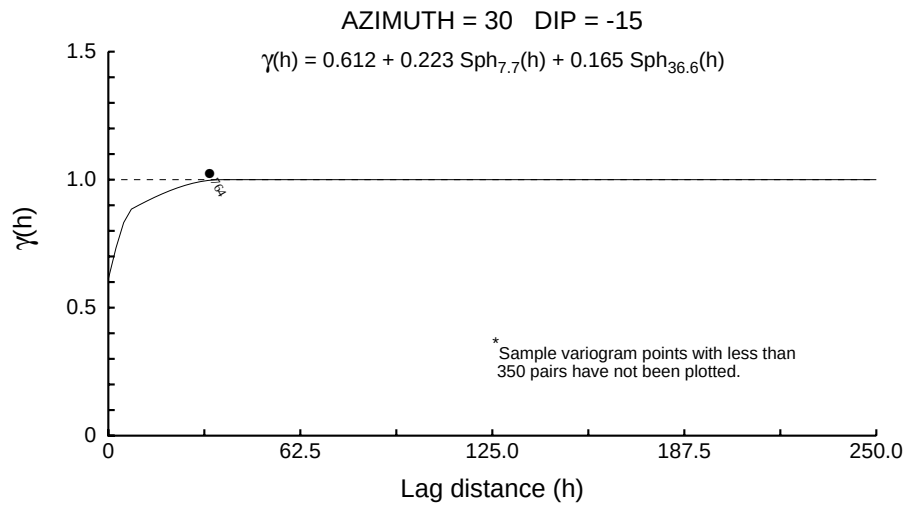
Main Ag 2.5m Comps



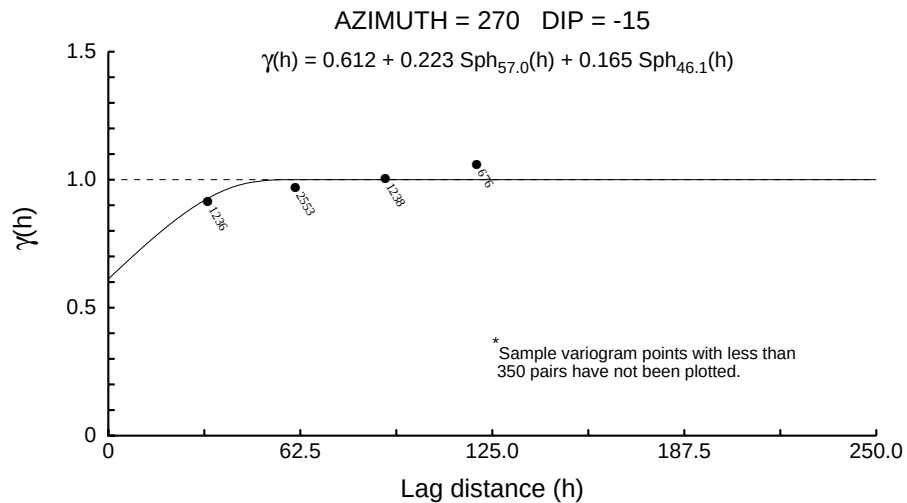
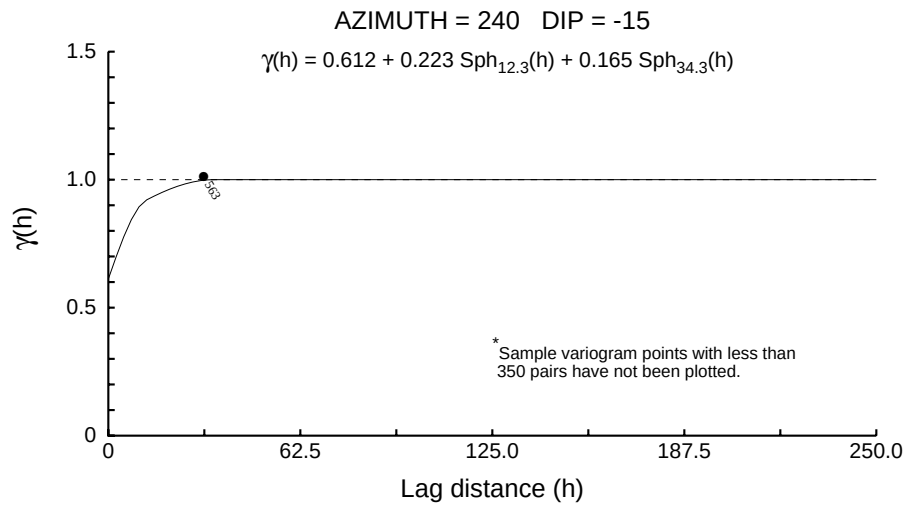
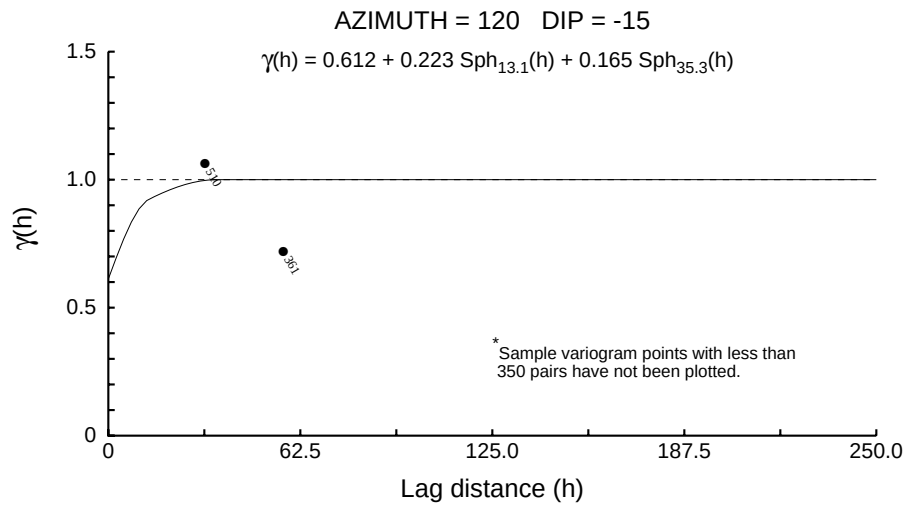
Main Ag 2.5m Comps



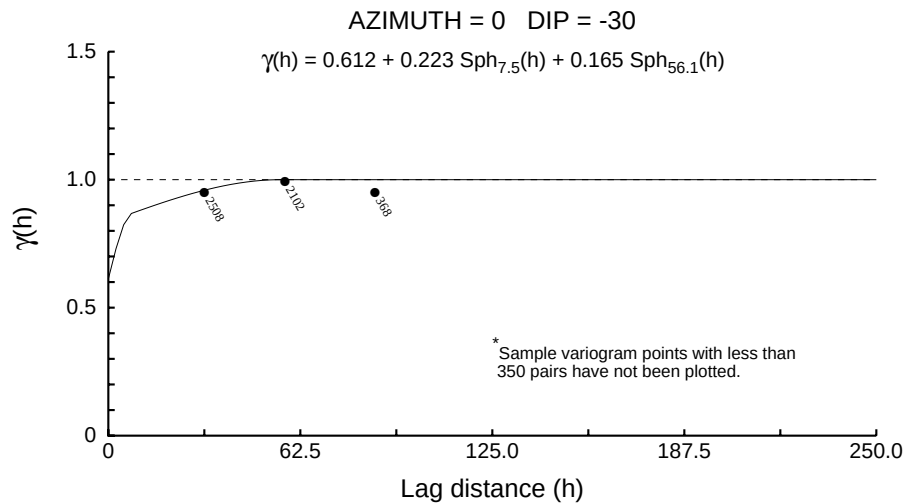
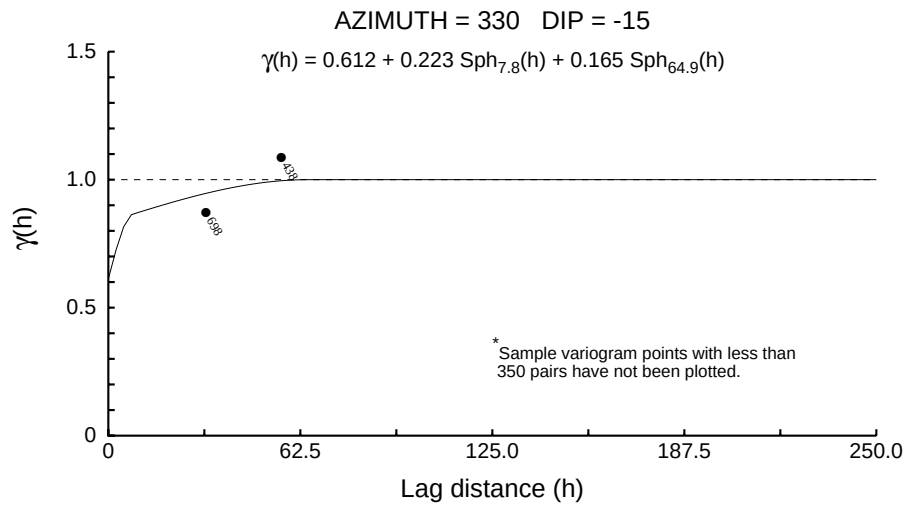
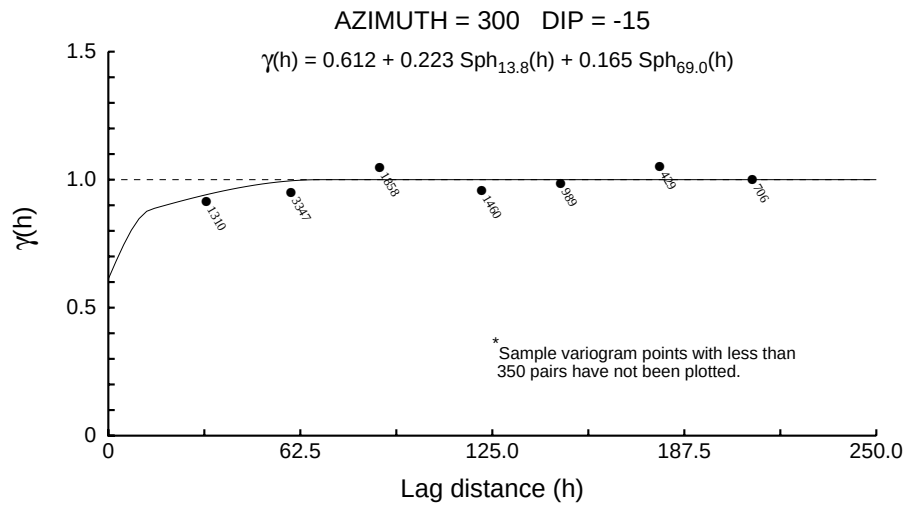
Main Ag 2.5m Comps



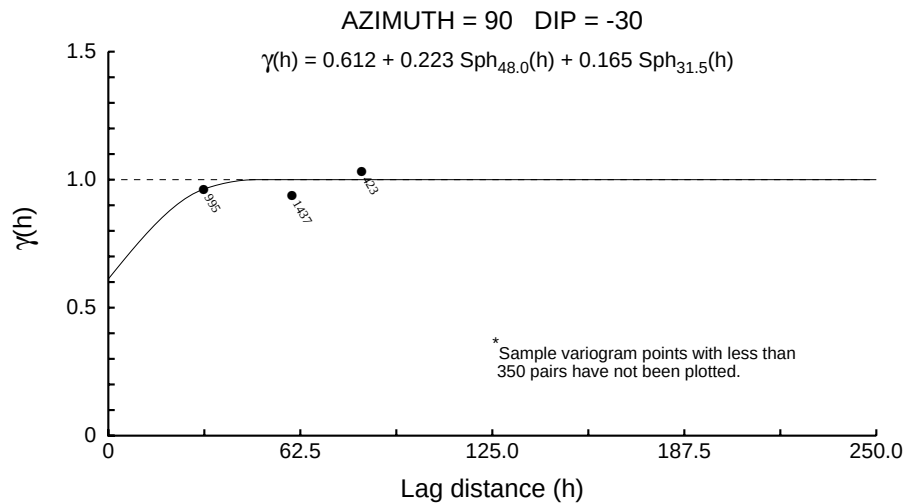
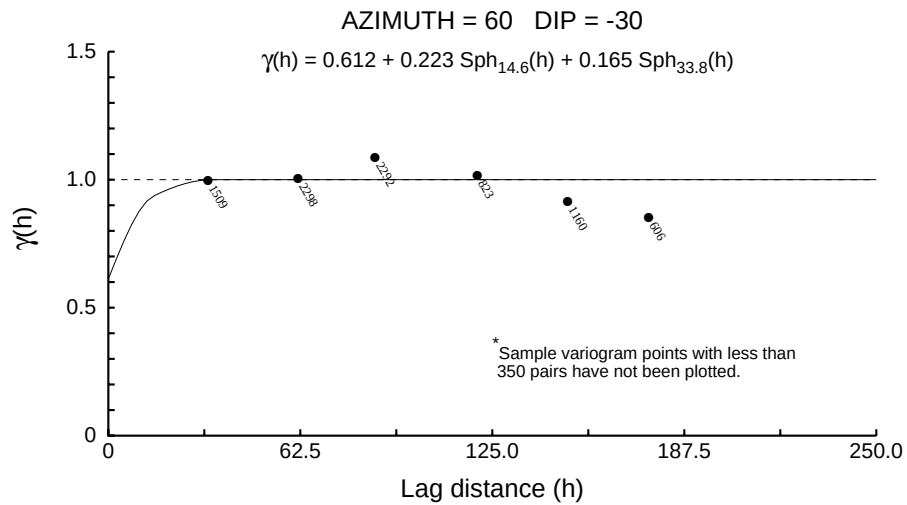
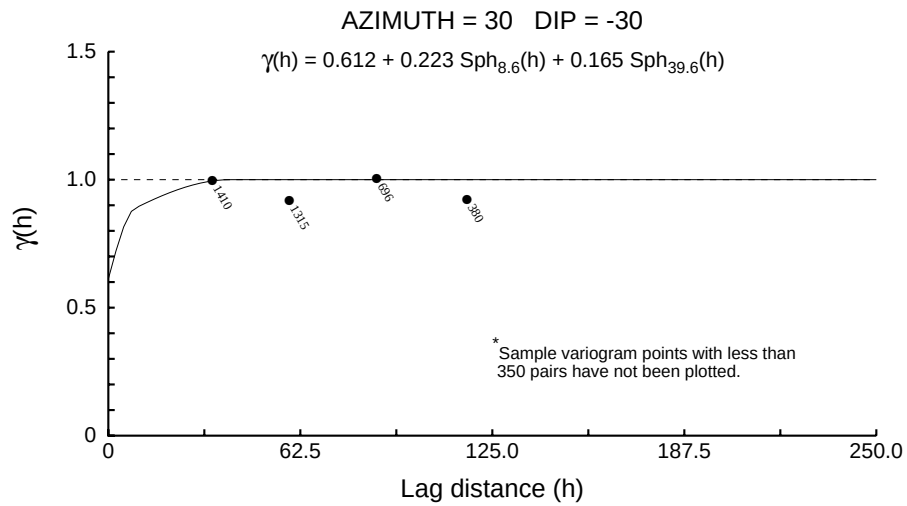
Main Ag 2.5m Comps



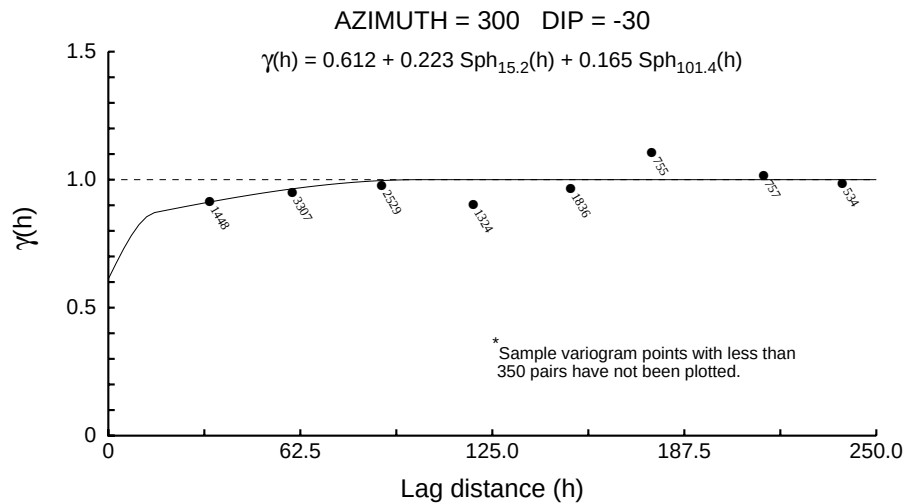
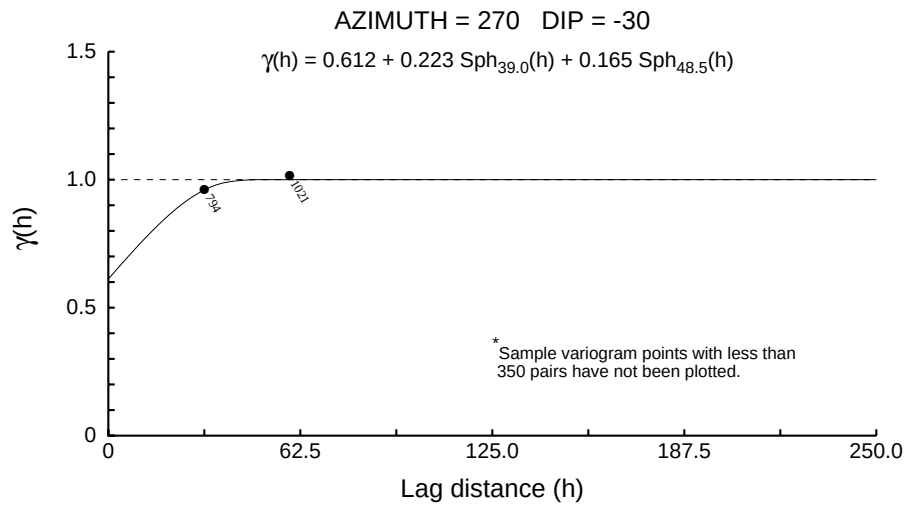
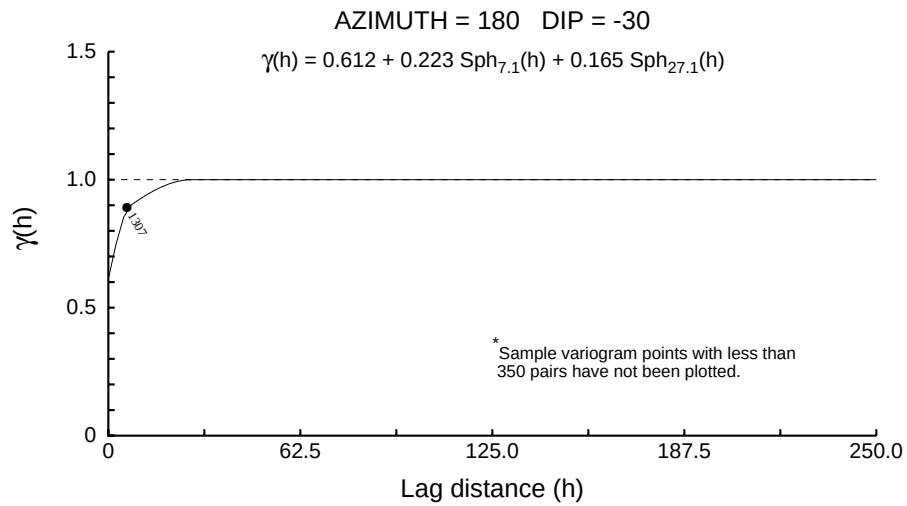
Main Ag 2.5m Comps



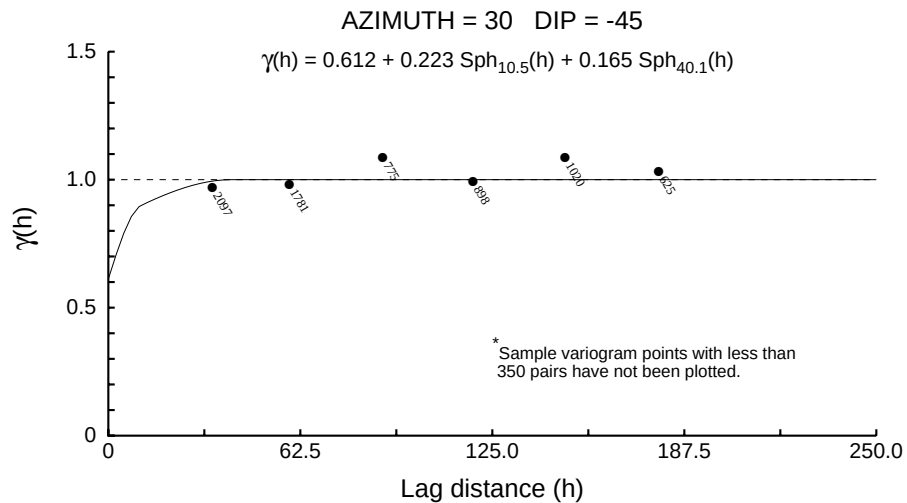
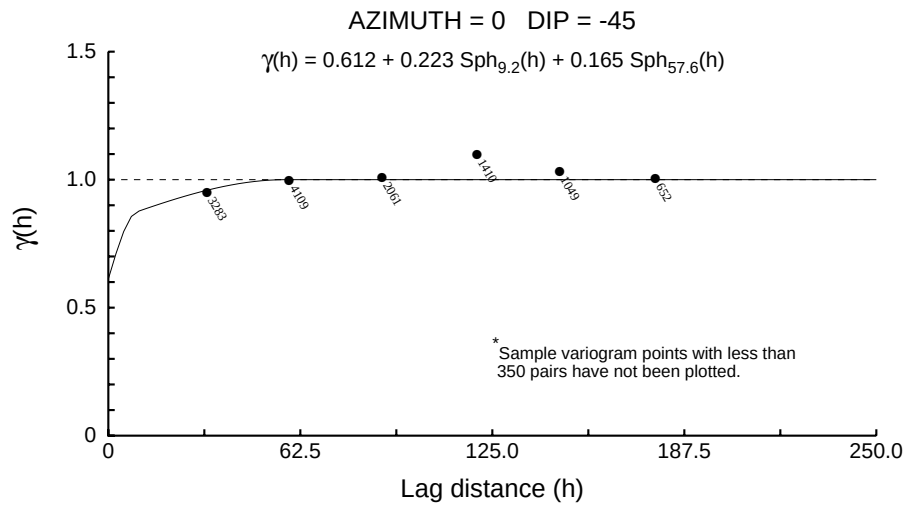
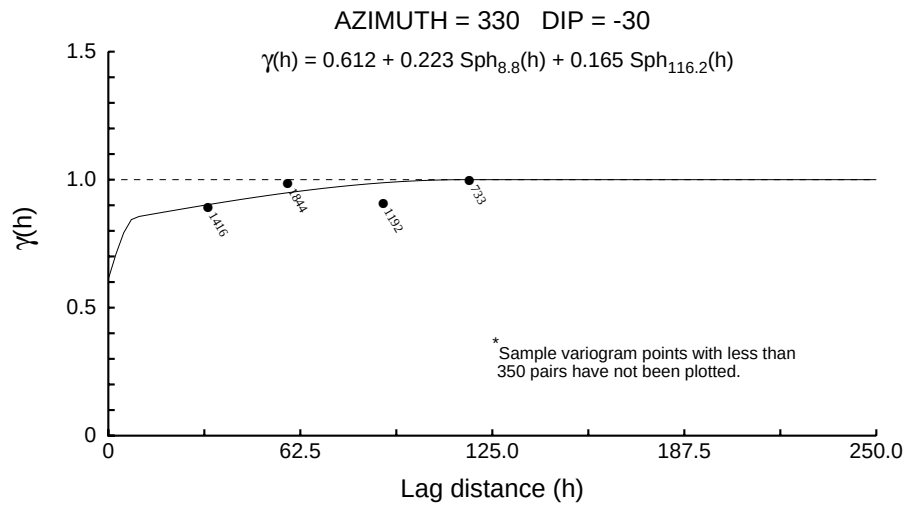
Main Ag 2.5m Comps



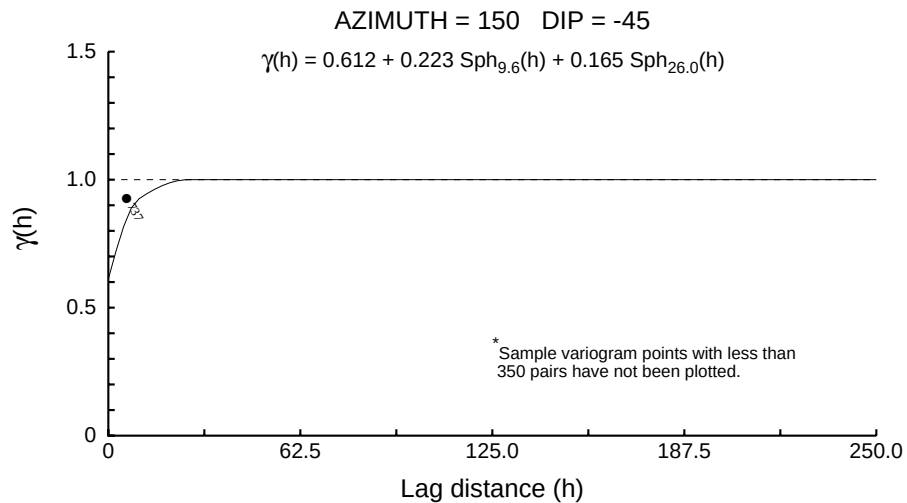
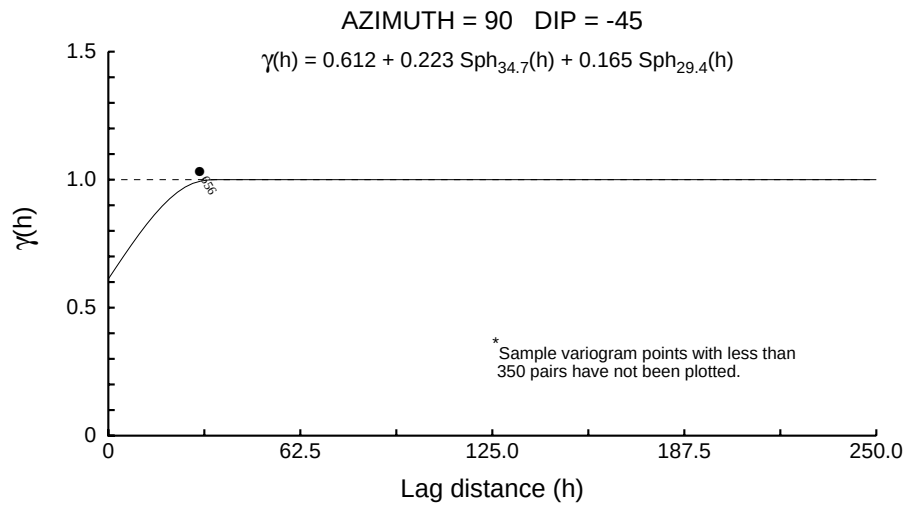
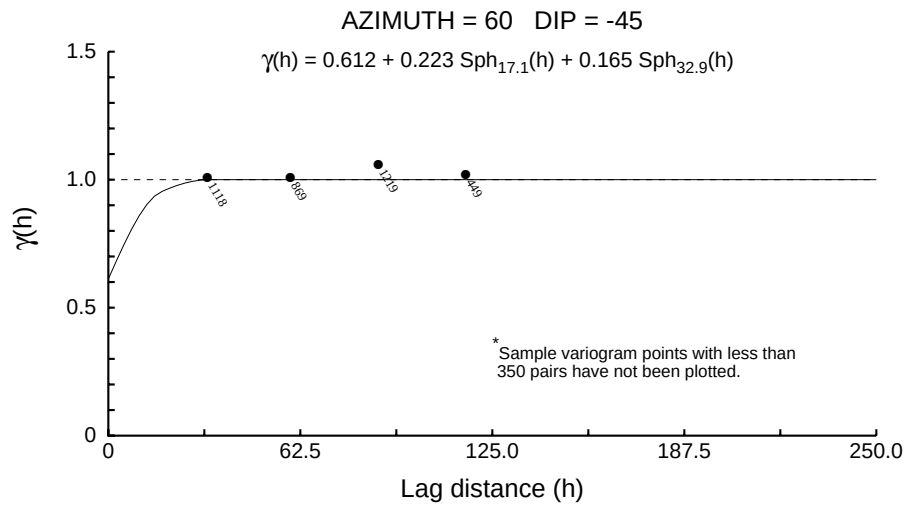
Main Ag 2.5m Comps



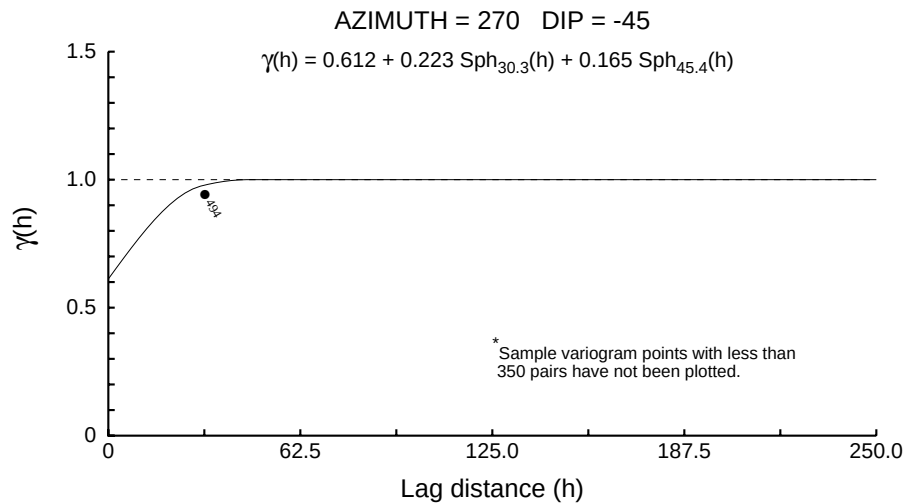
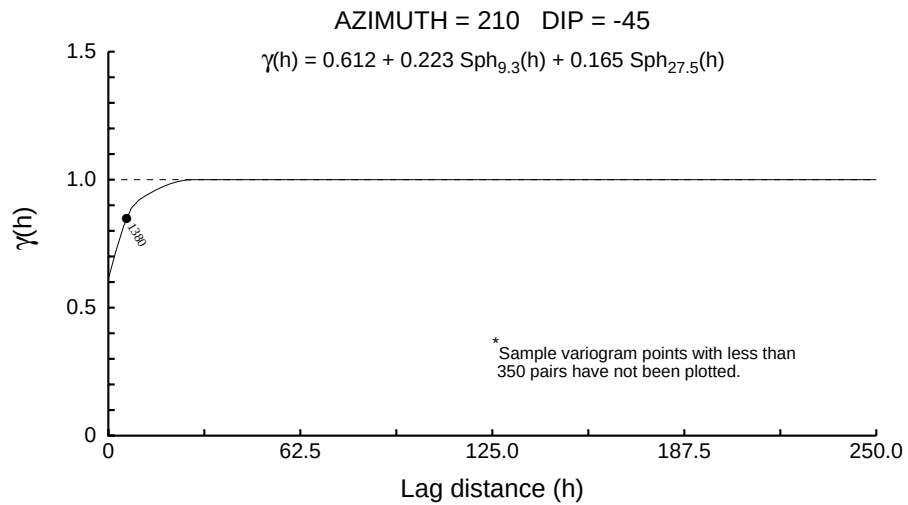
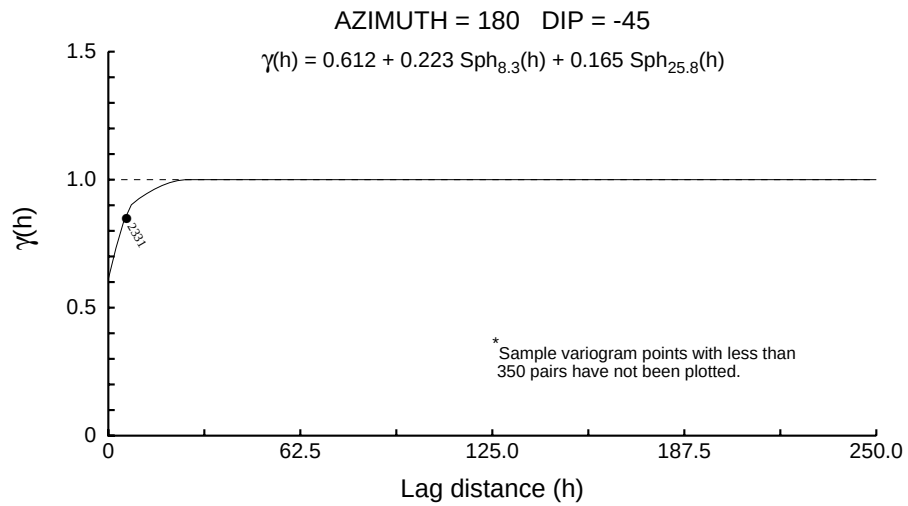
Main Ag 2.5m Comps



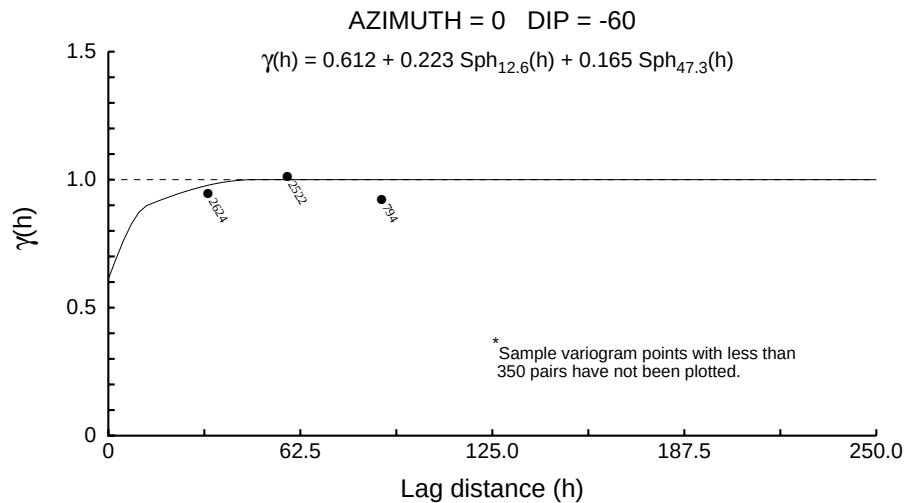
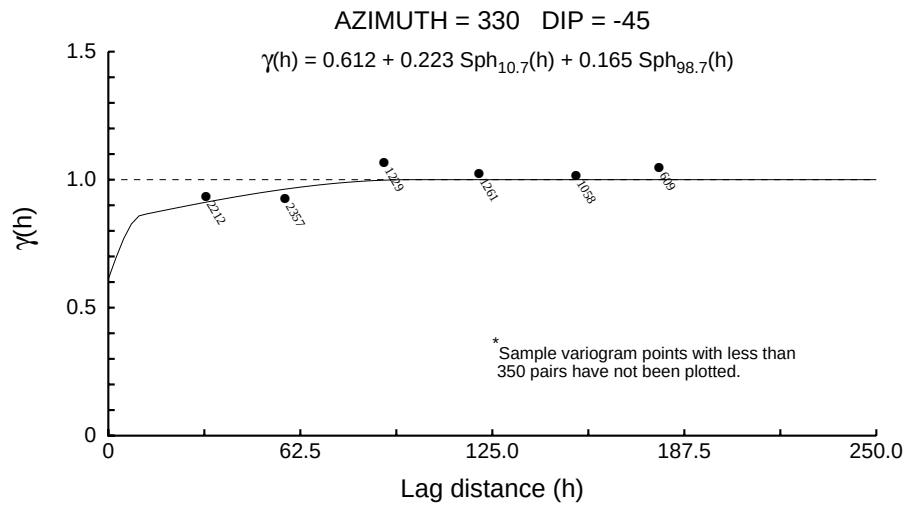
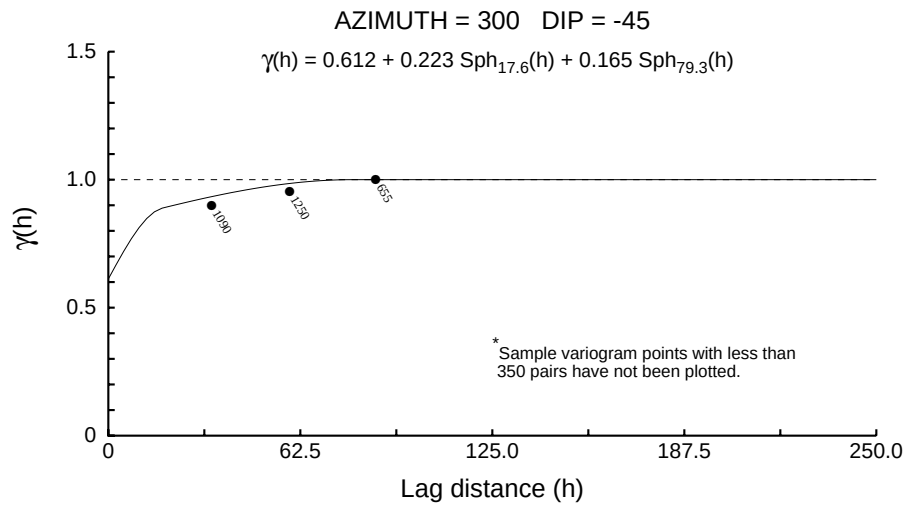
Main Ag 2.5m Comps



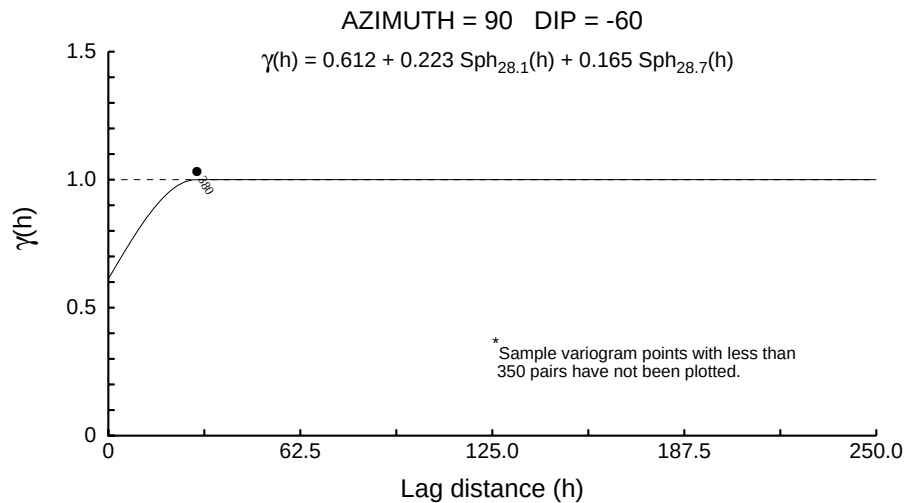
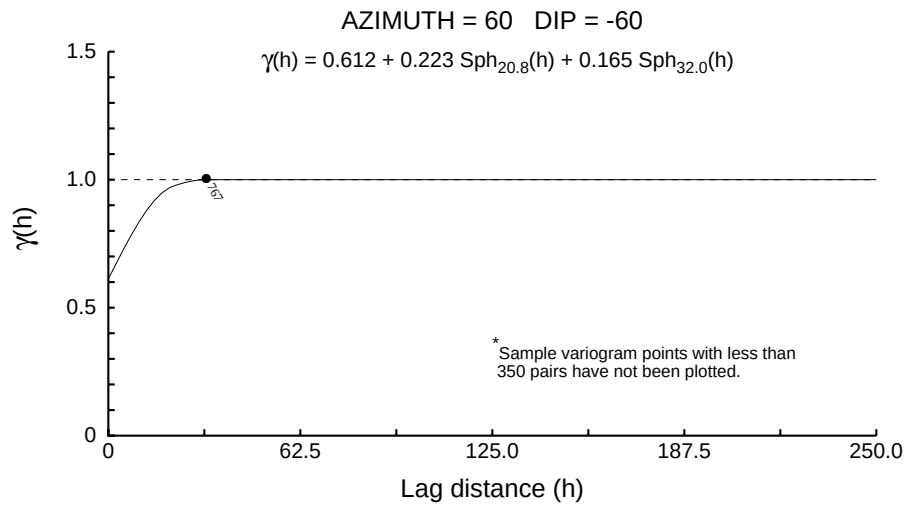
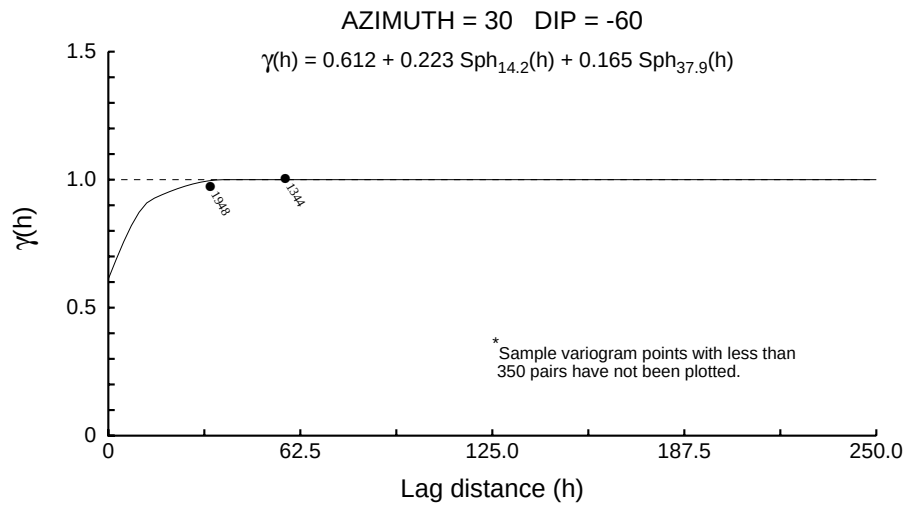
Main Ag 2.5m Comps



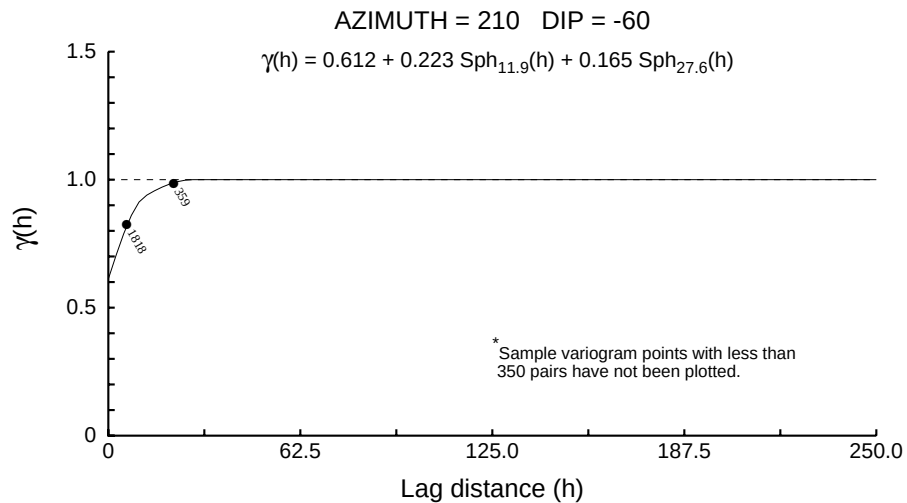
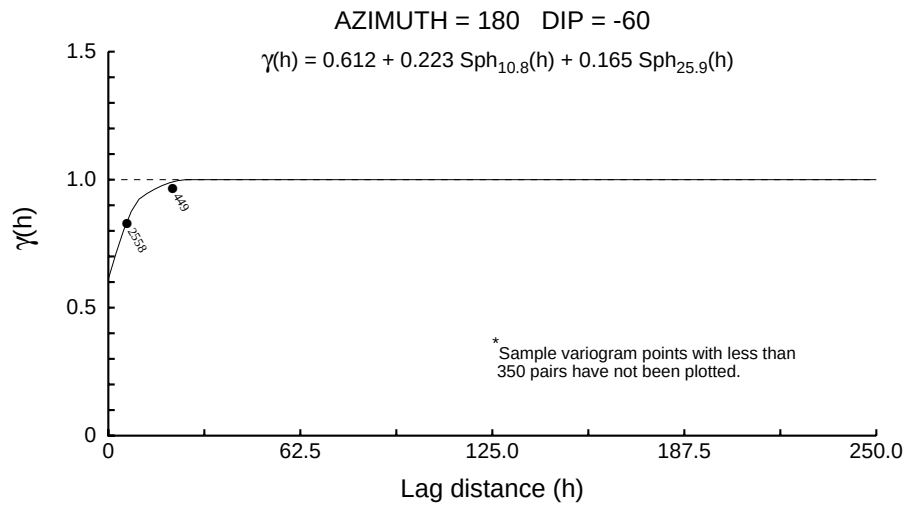
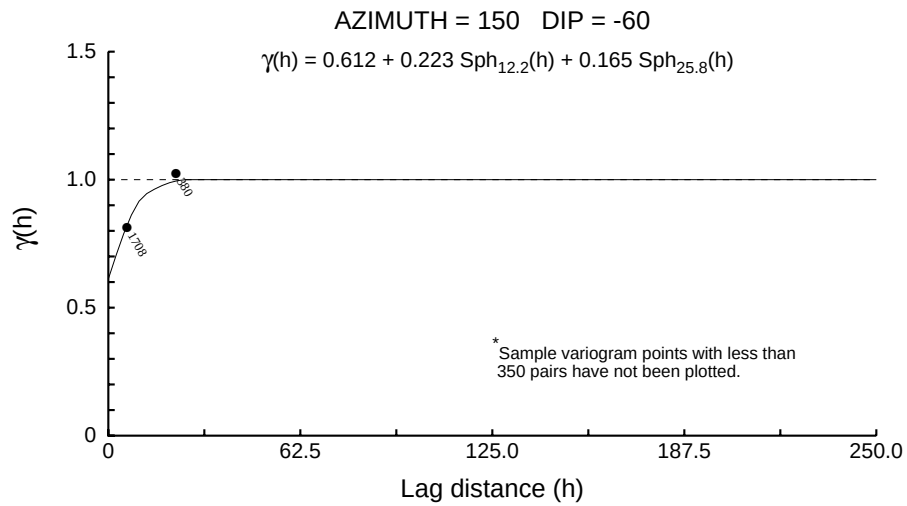
Main Ag 2.5m Comps



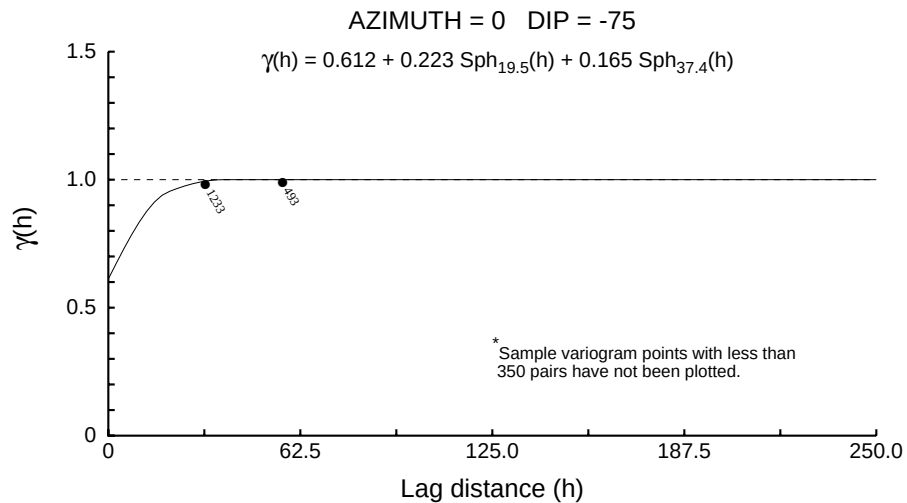
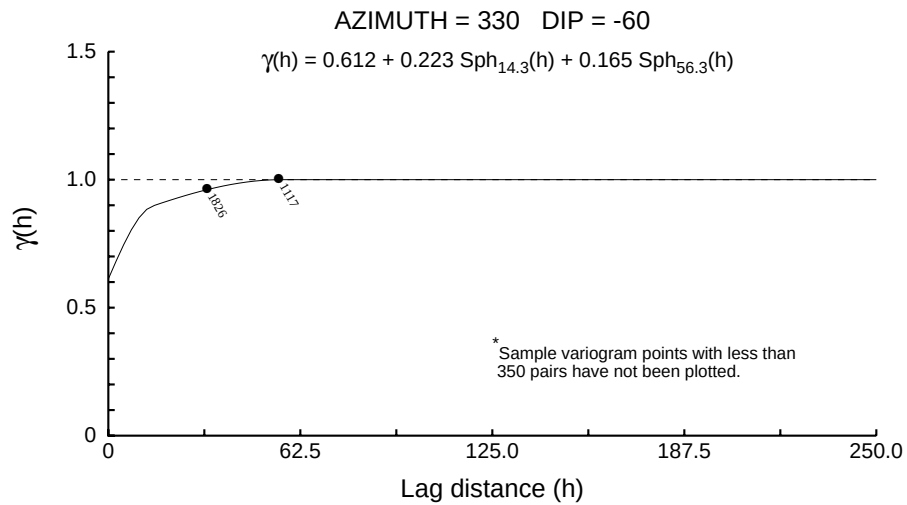
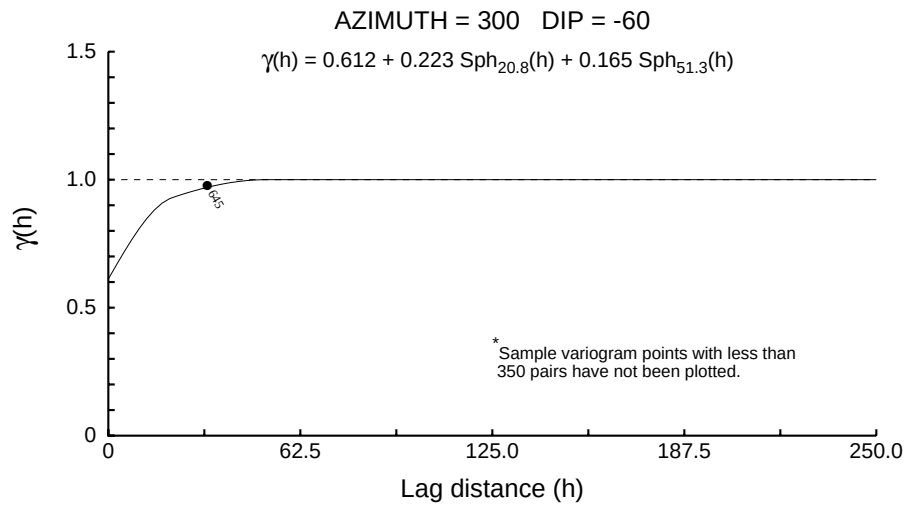
Main Ag 2.5m Comps



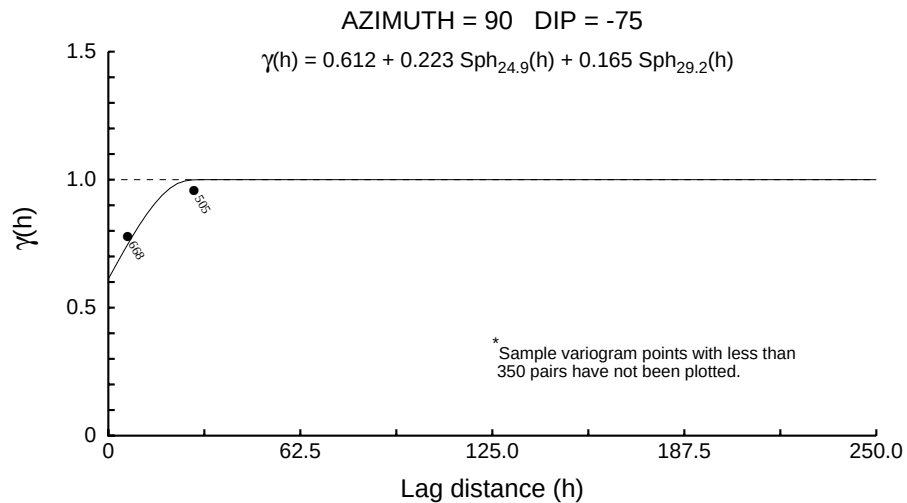
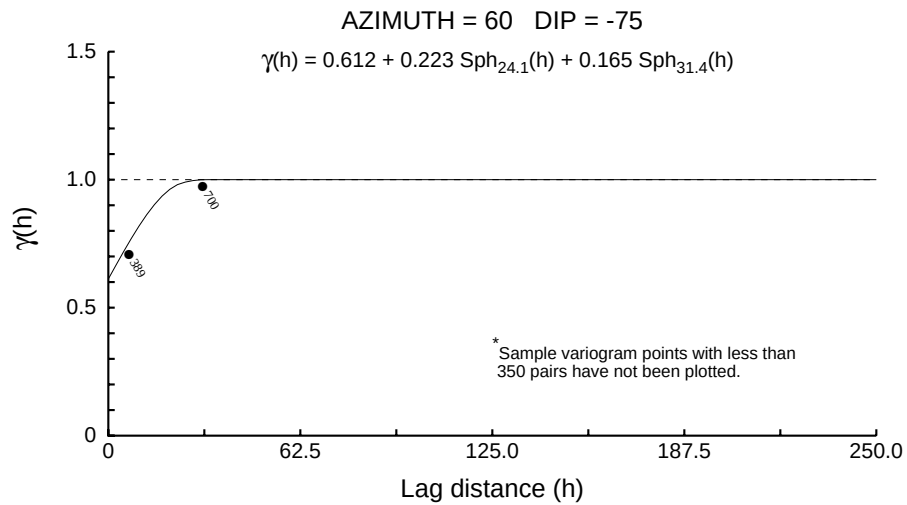
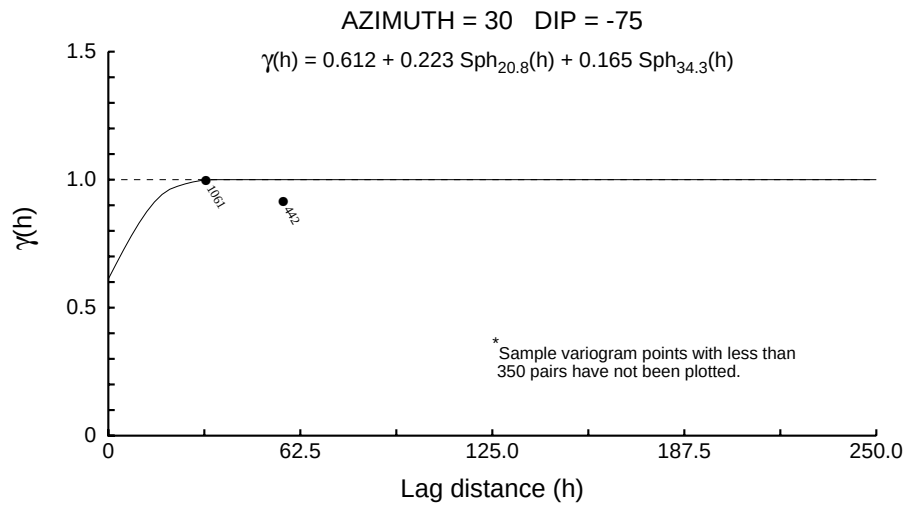
Main Ag 2.5m Comps



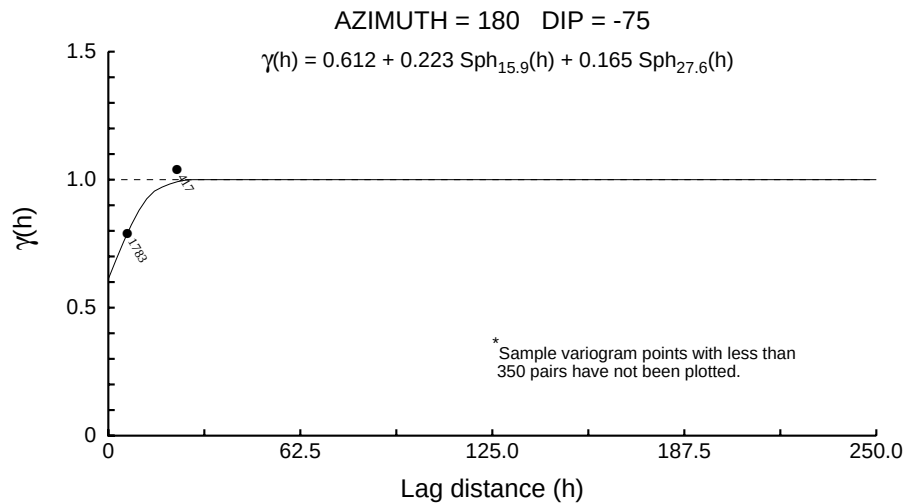
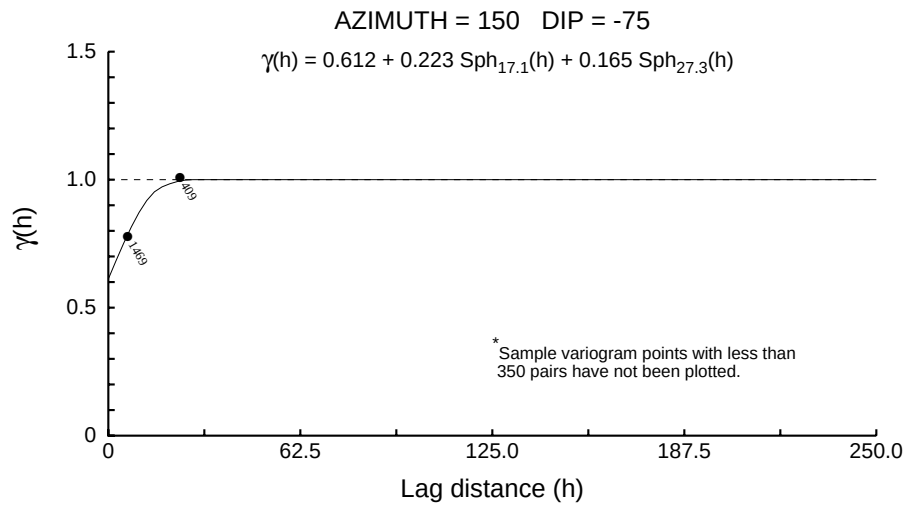
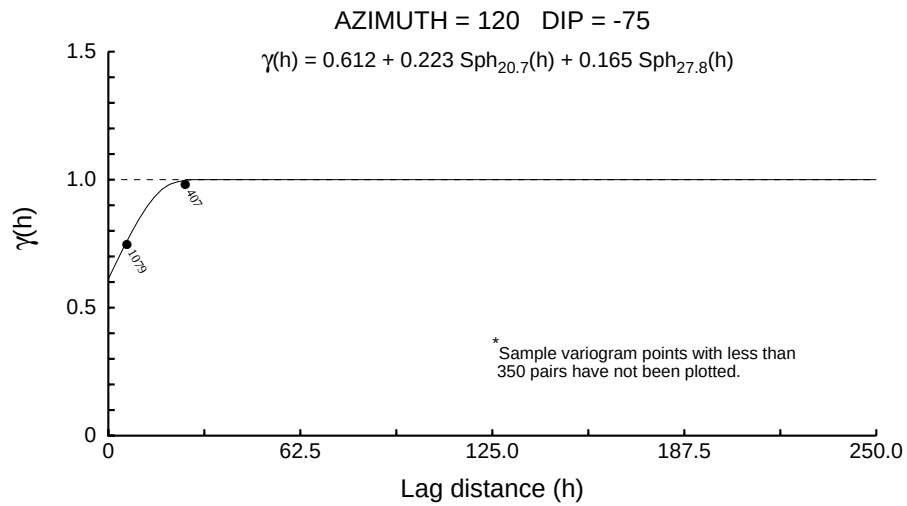
Main Ag 2.5m Comps



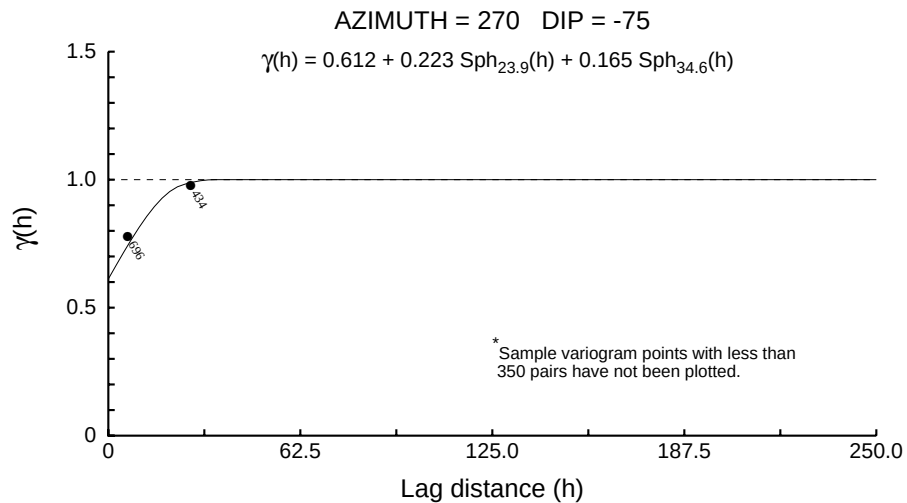
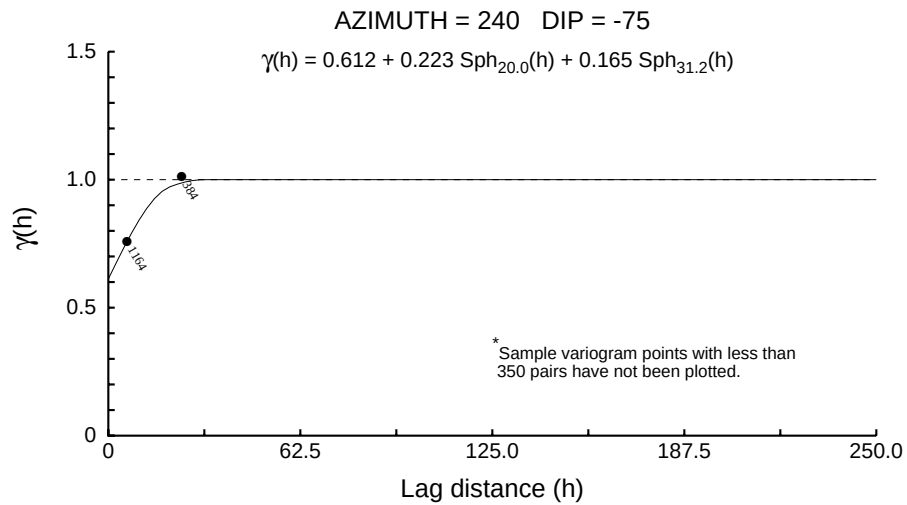
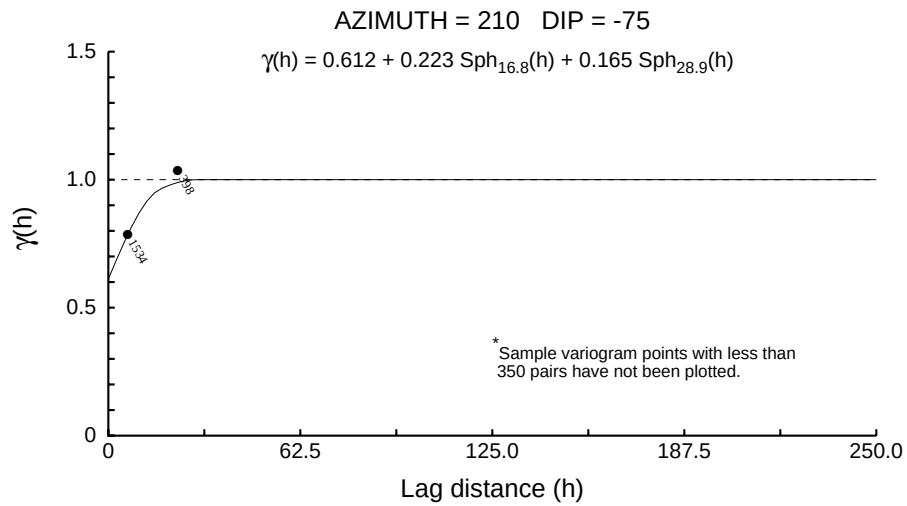
Main Ag 2.5m Comps



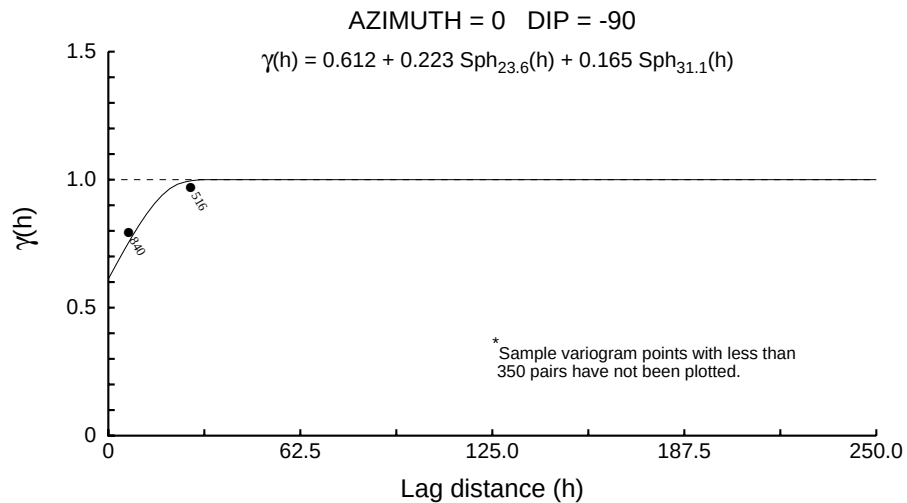
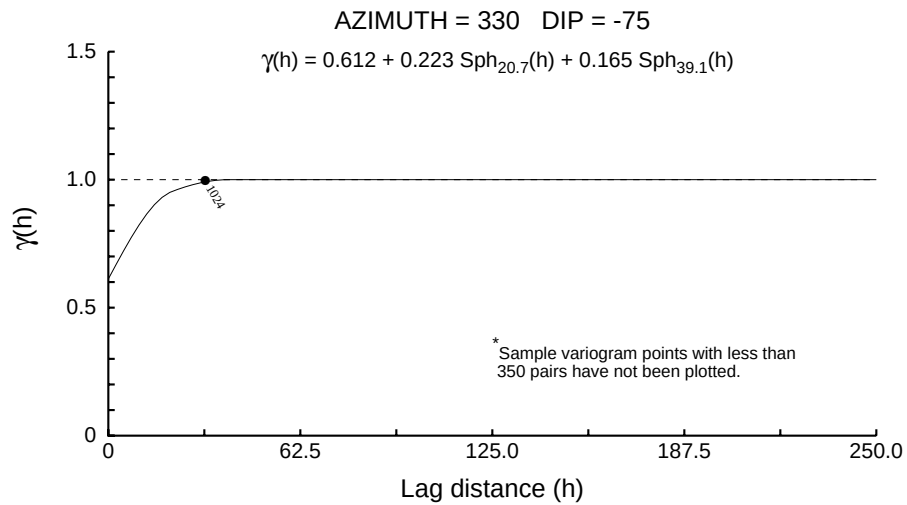
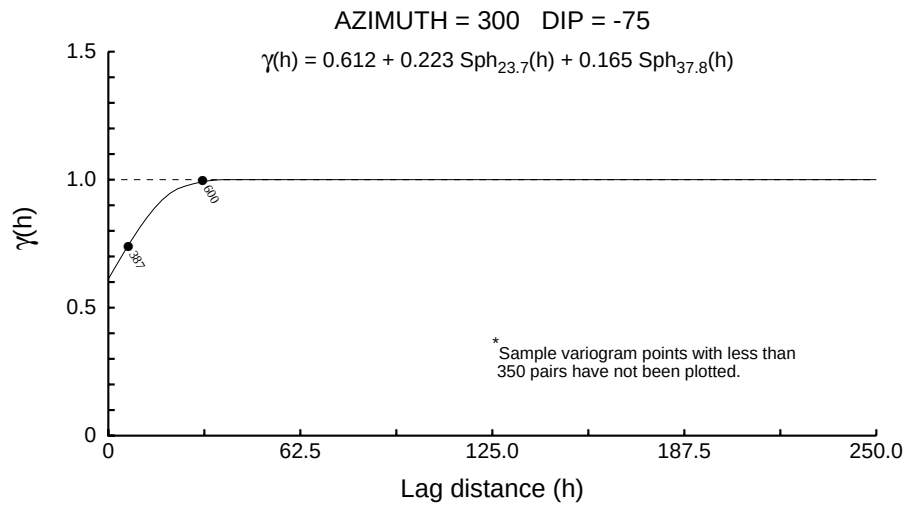
Main Ag 2.5m Comps



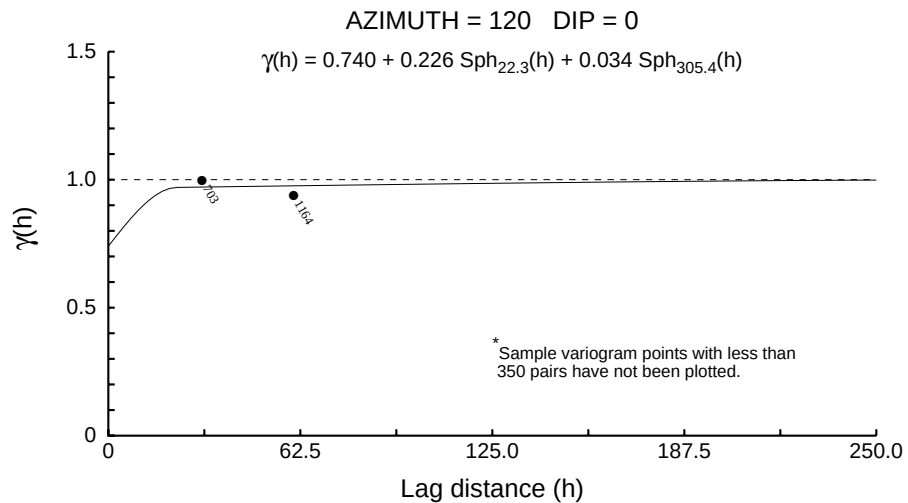
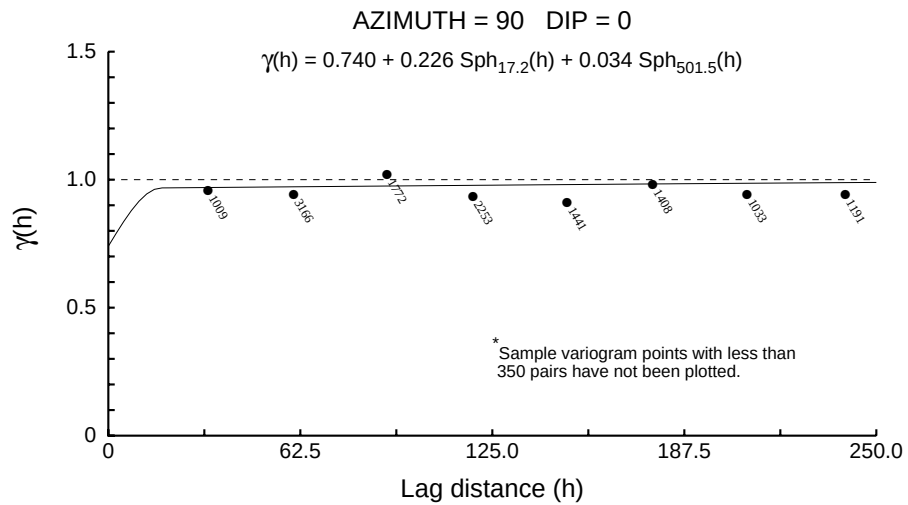
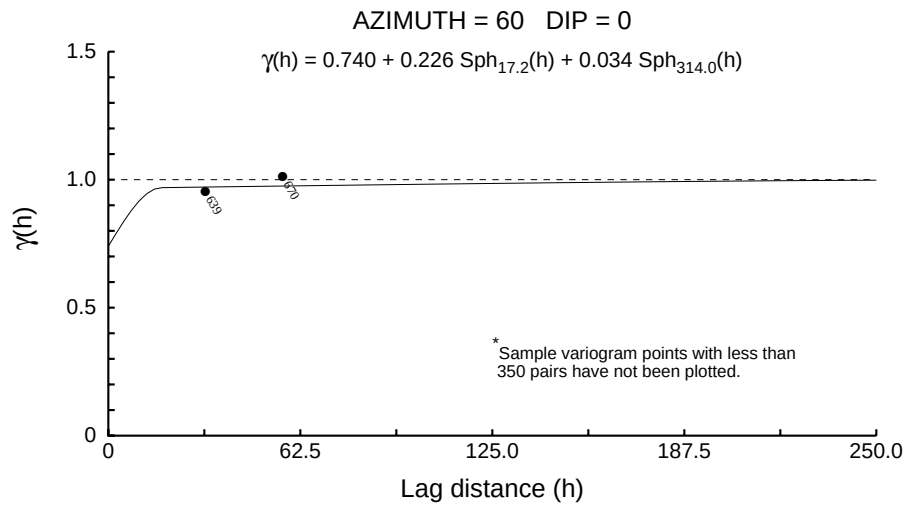
Main Ag 2.5m Comps



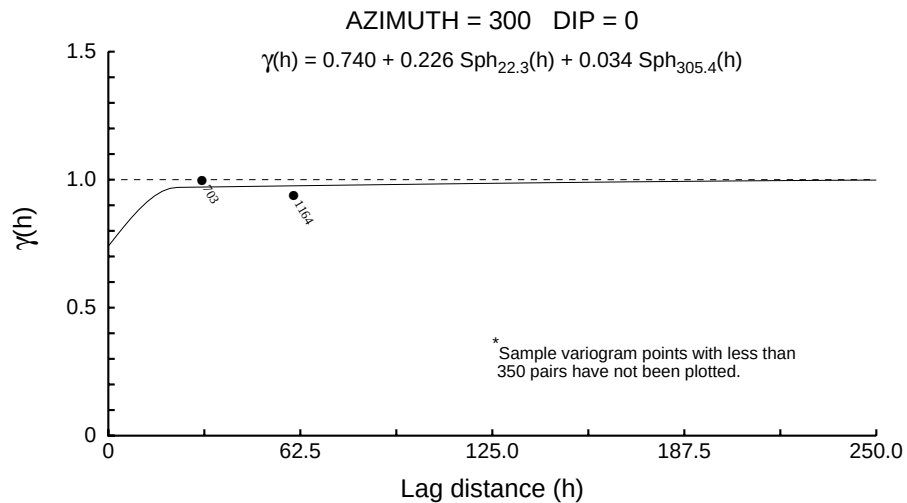
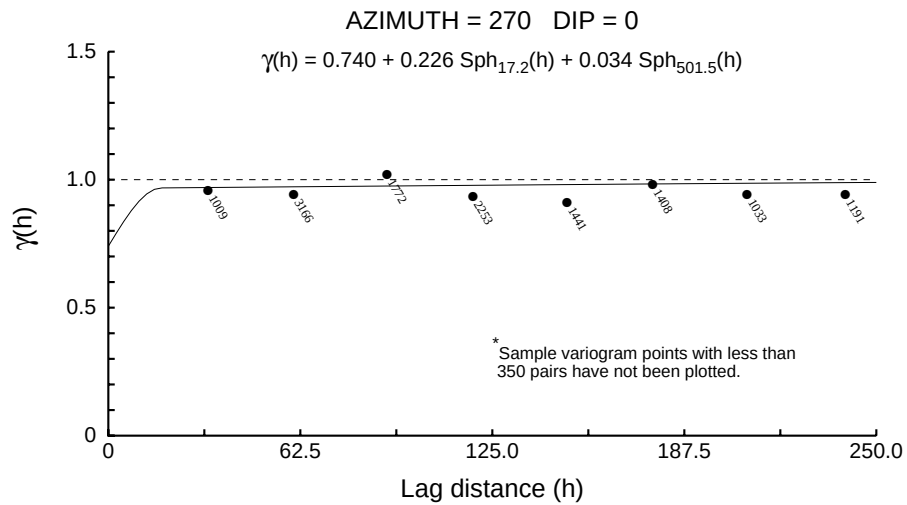
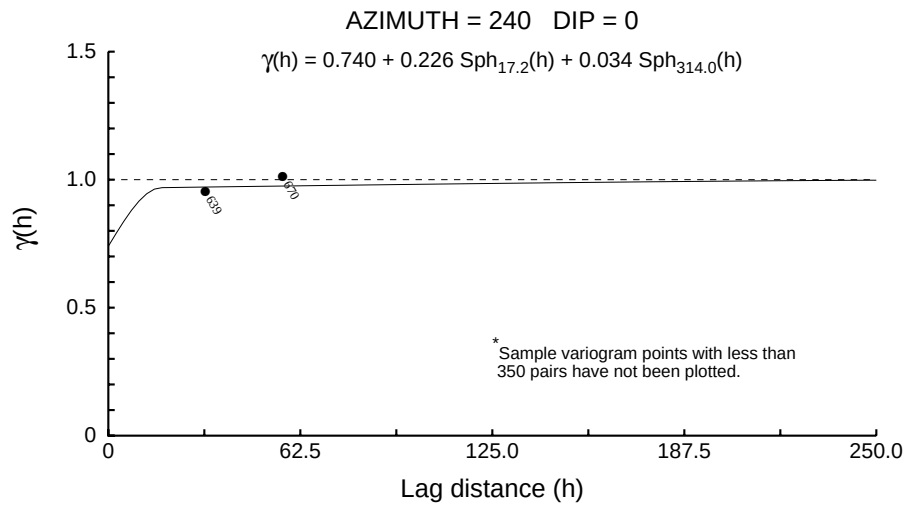
Main Ag 2.5m Comps



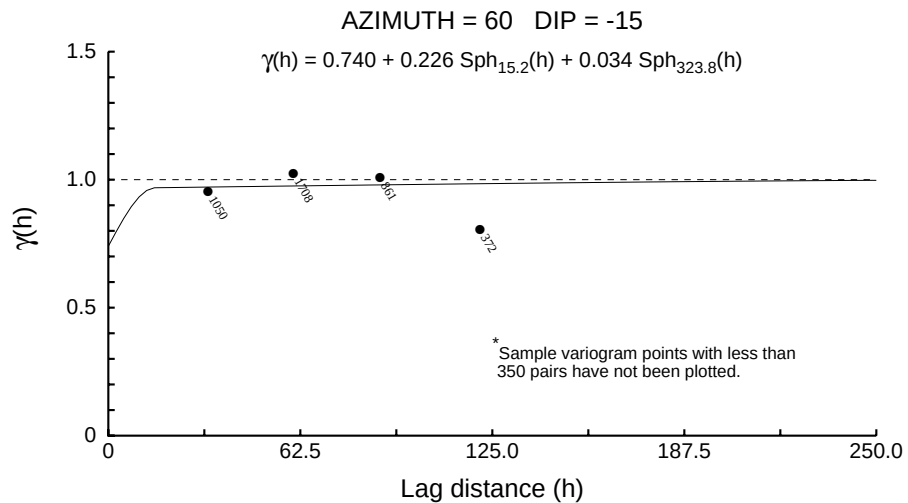
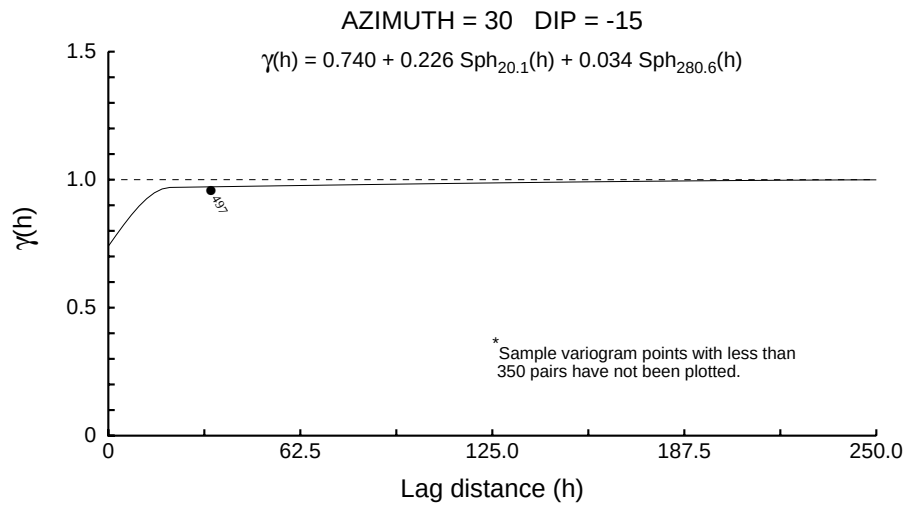
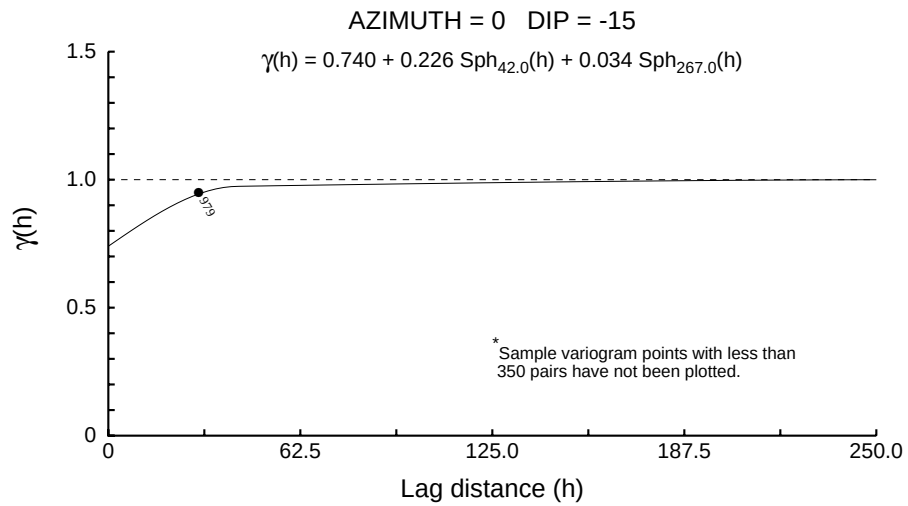
Main Au 2.5m Comps



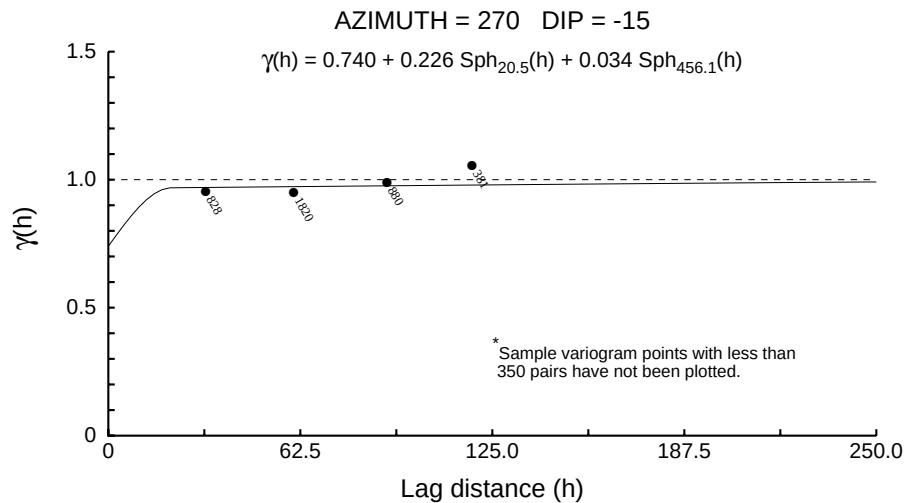
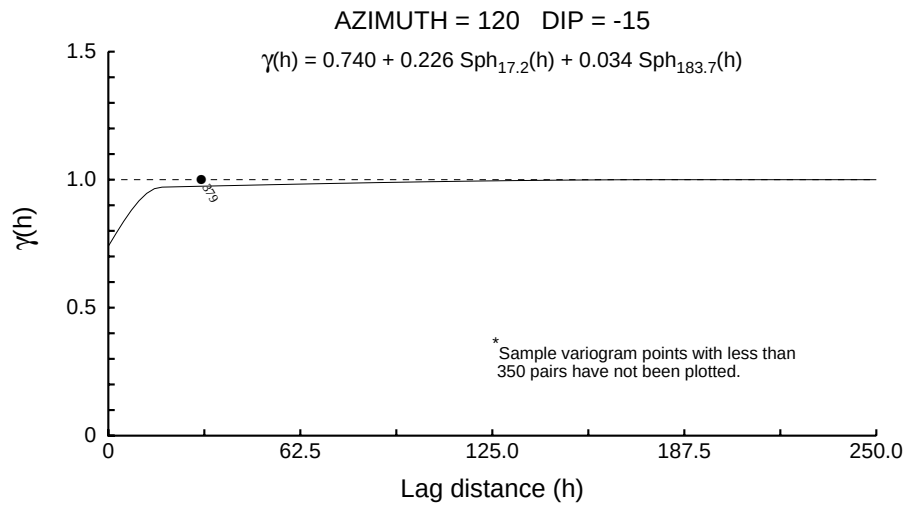
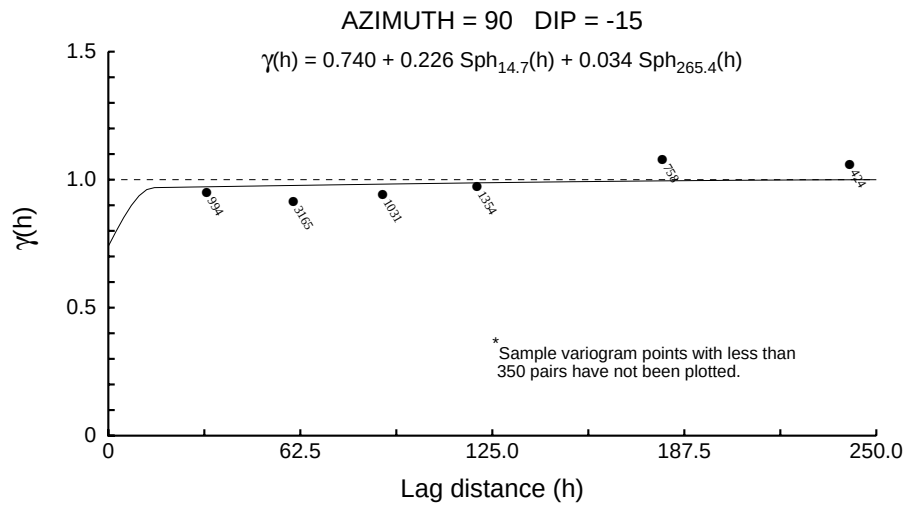
Main Au 2.5m Comps



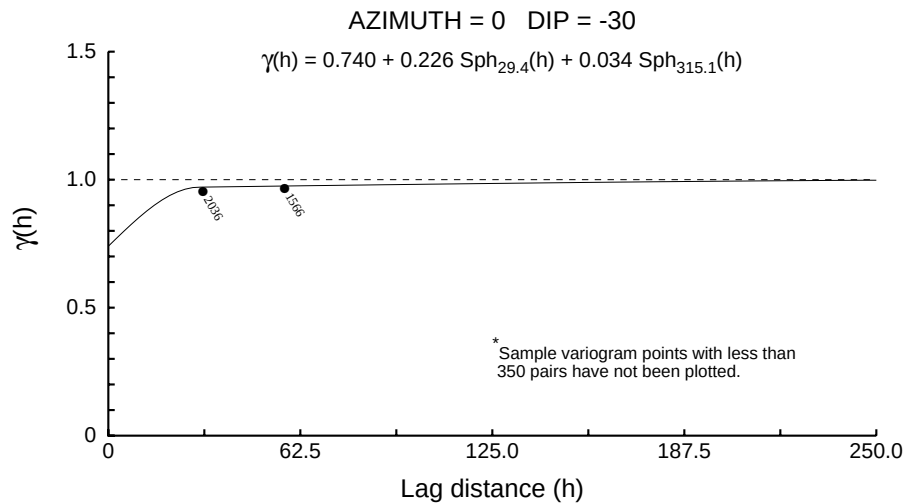
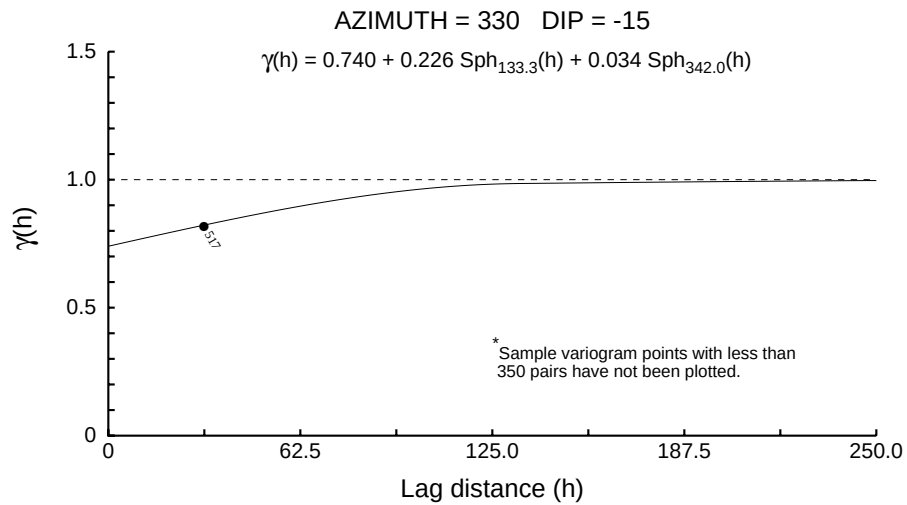
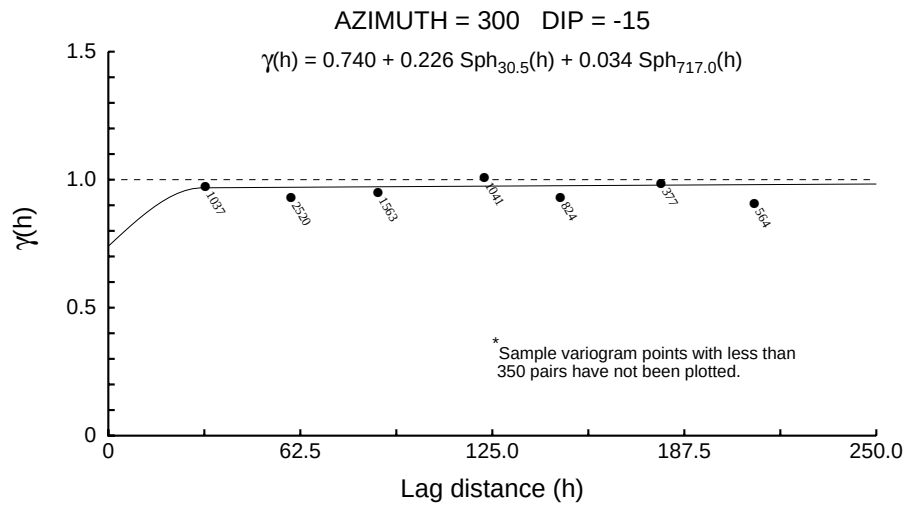
Main Au 2.5m Comps



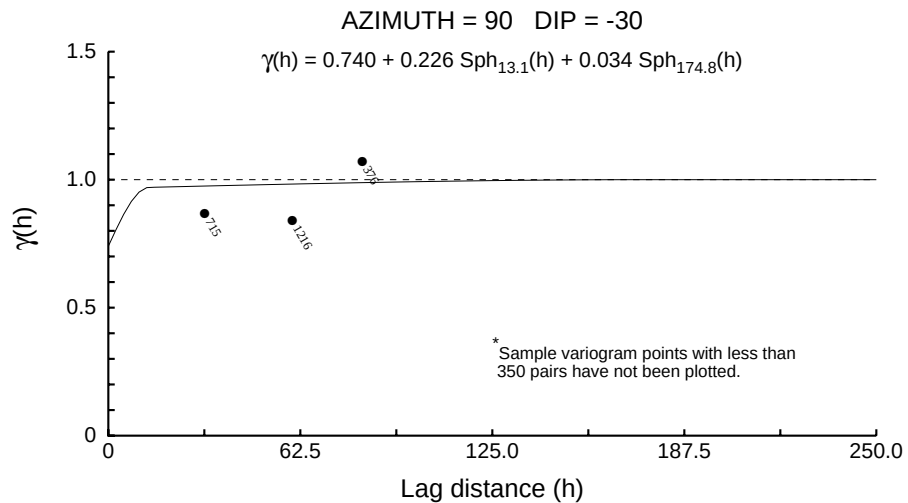
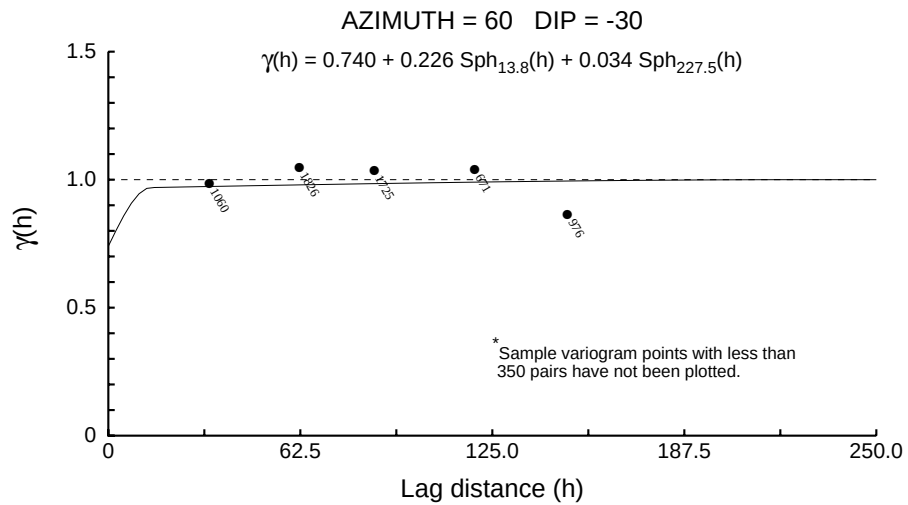
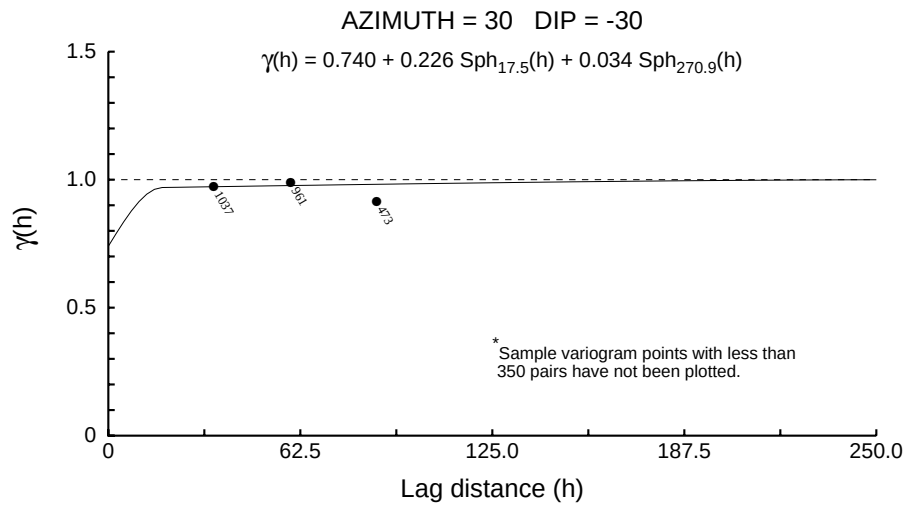
Main Au 2.5m Comps



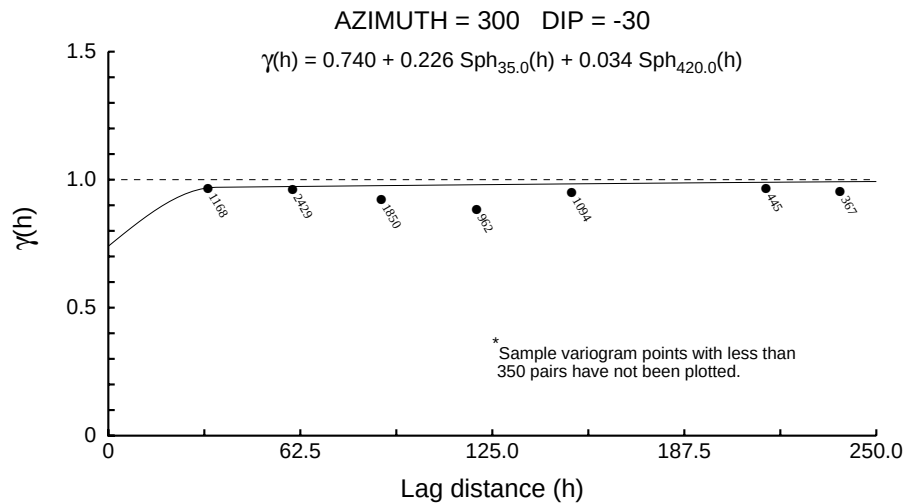
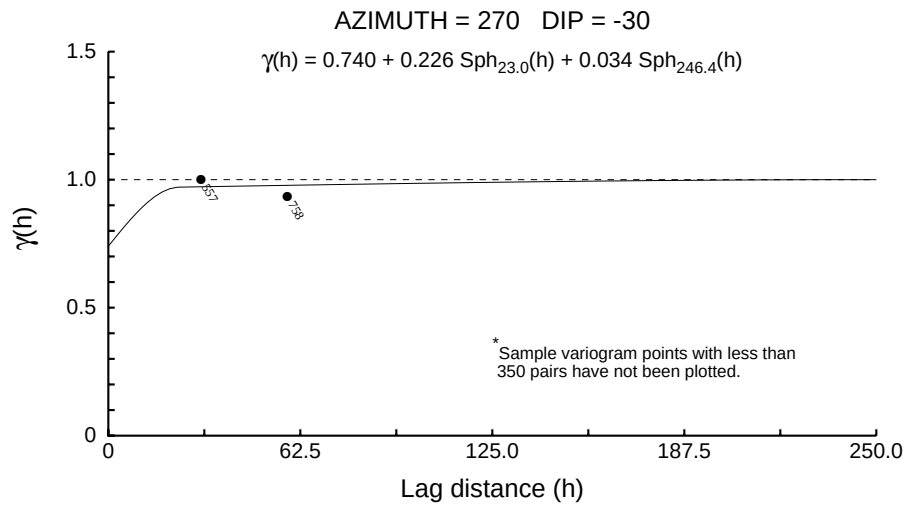
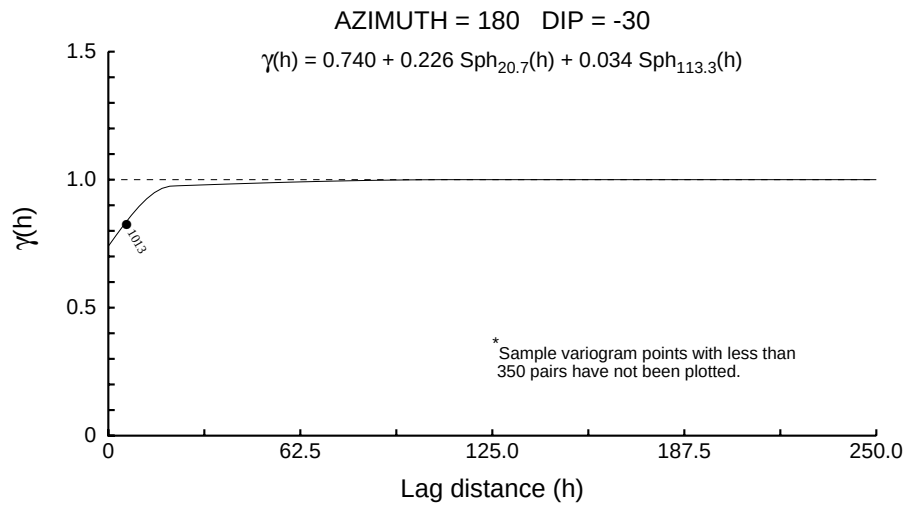
Main Au 2.5m Comps



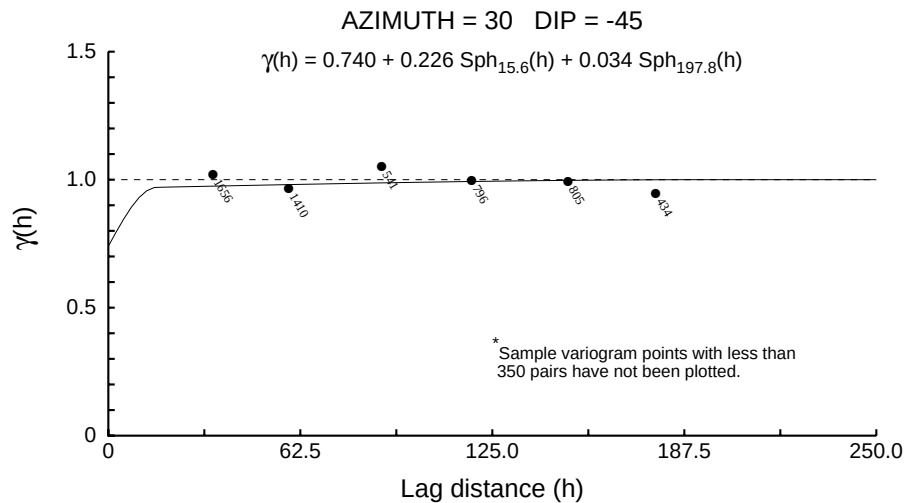
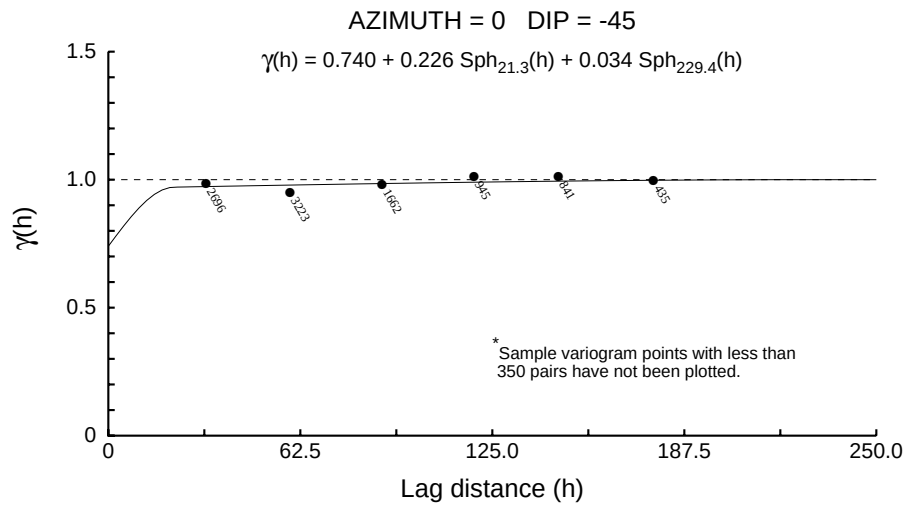
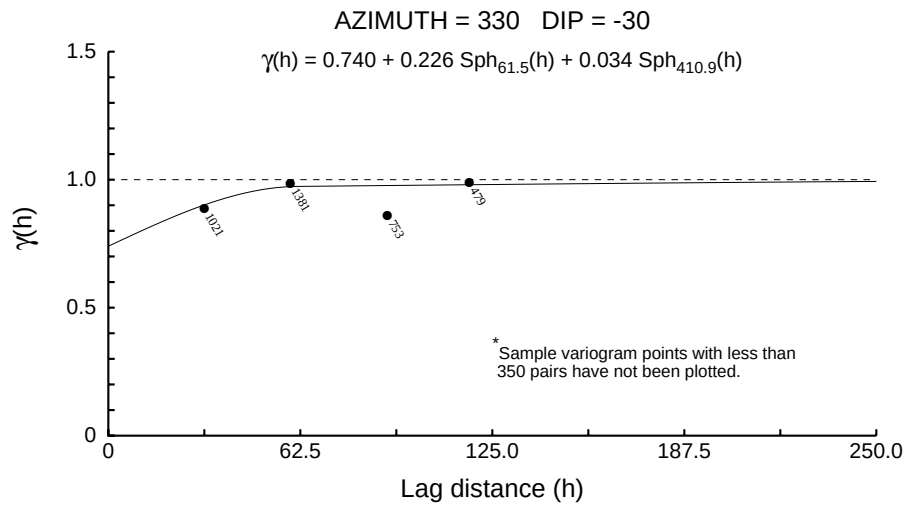
Main Au 2.5m Comps



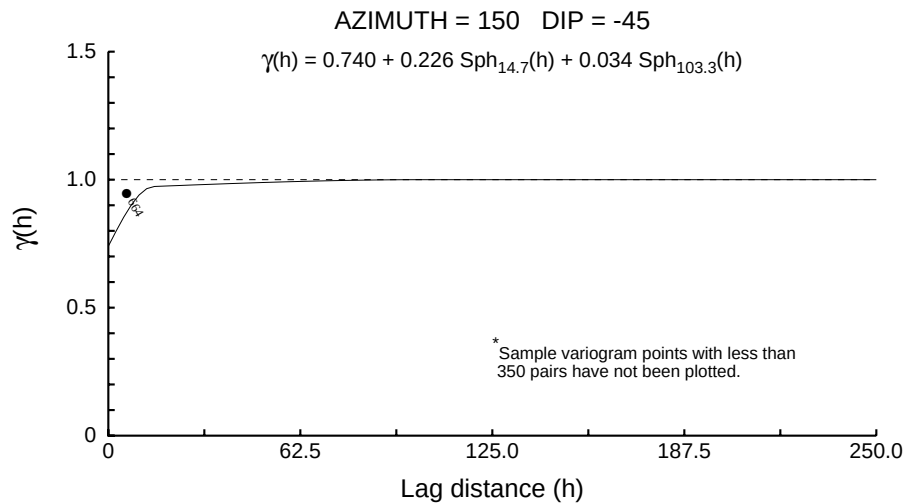
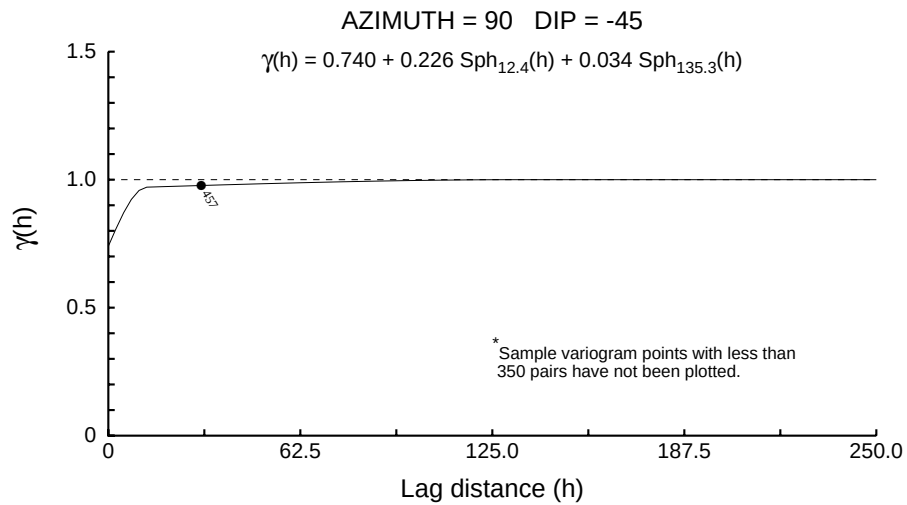
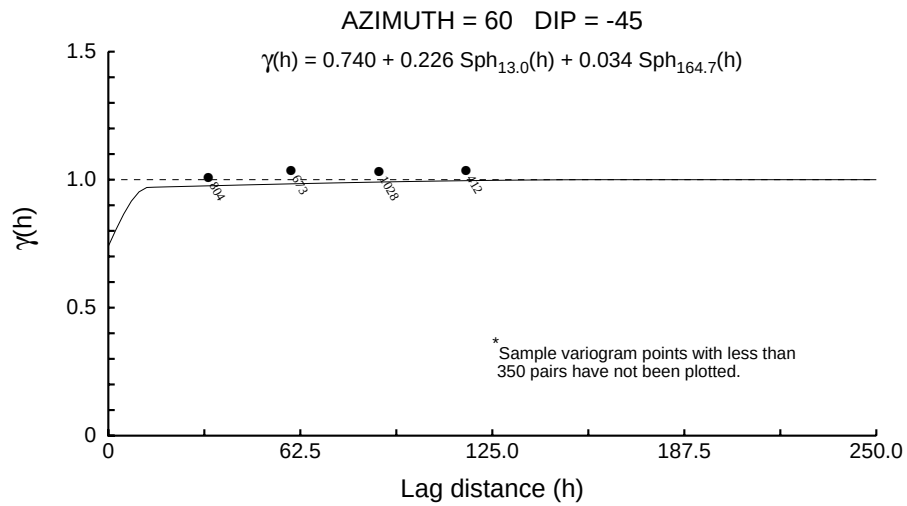
Main Au 2.5m Comps



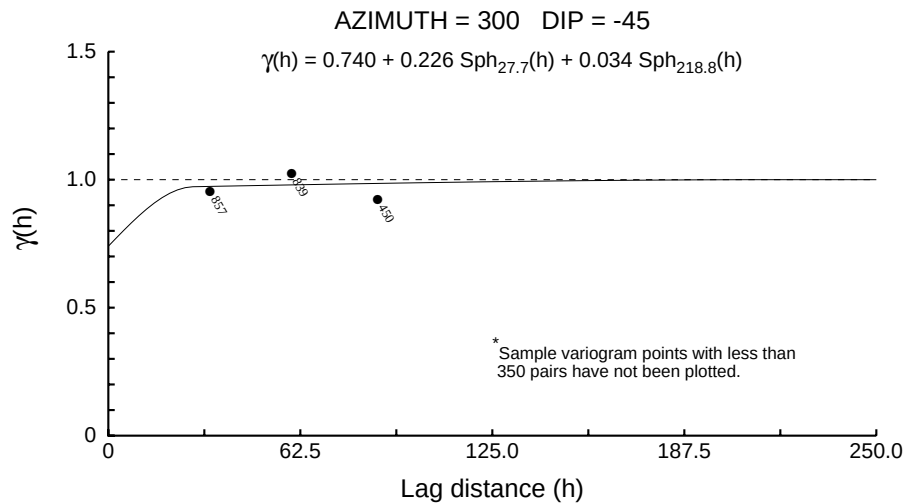
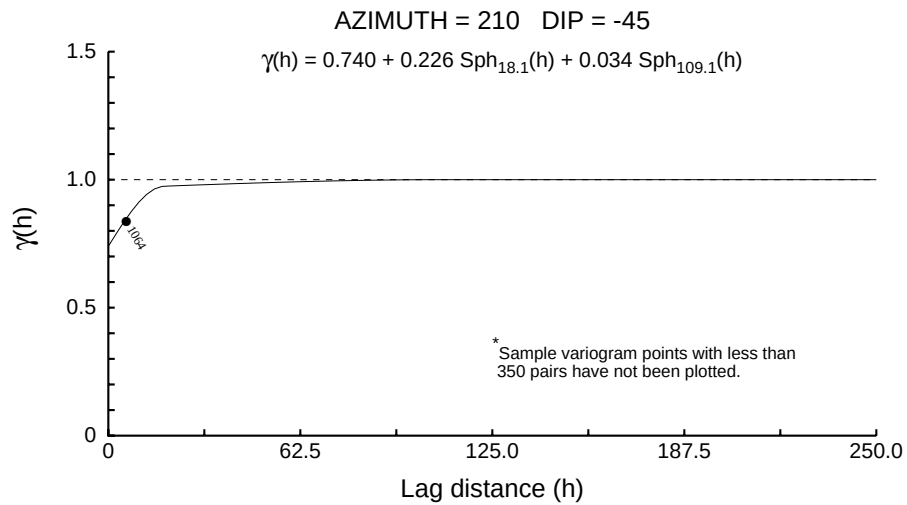
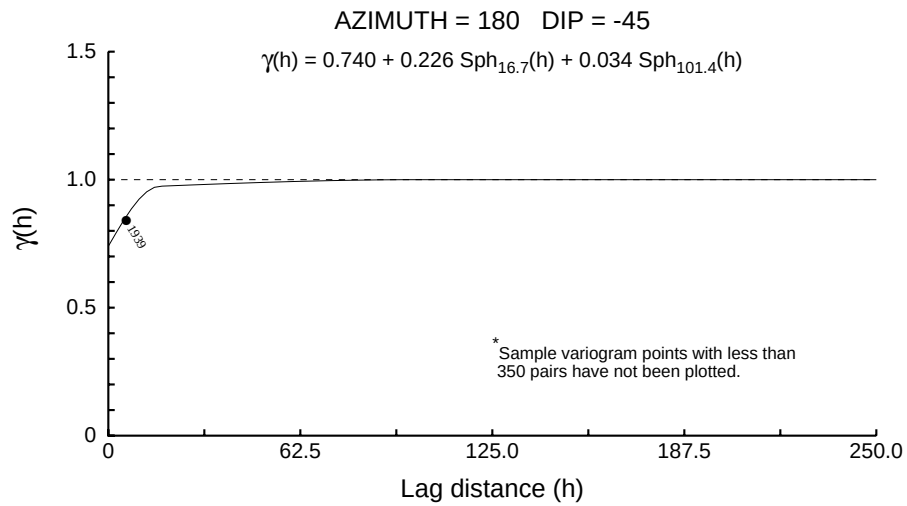
Main Au 2.5m Comps



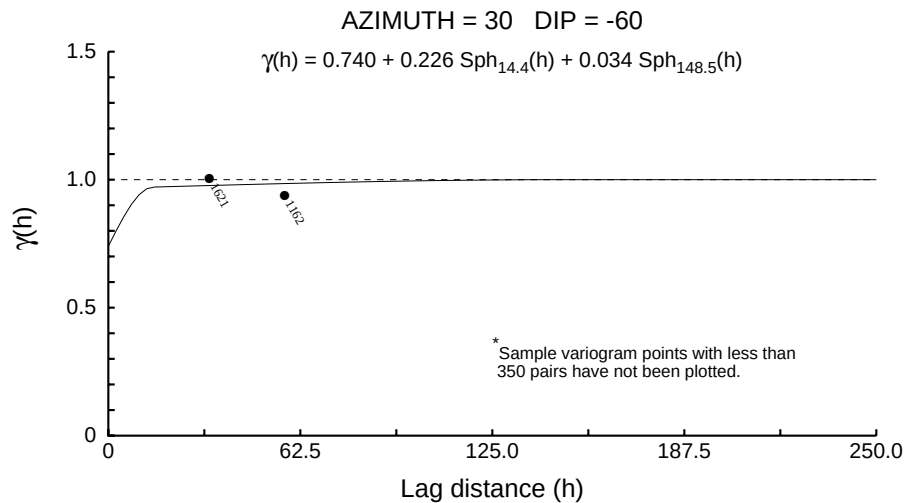
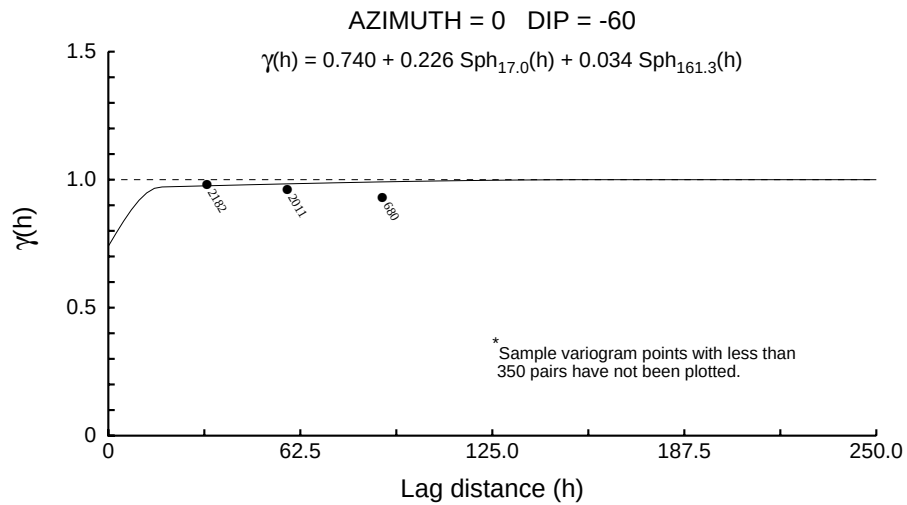
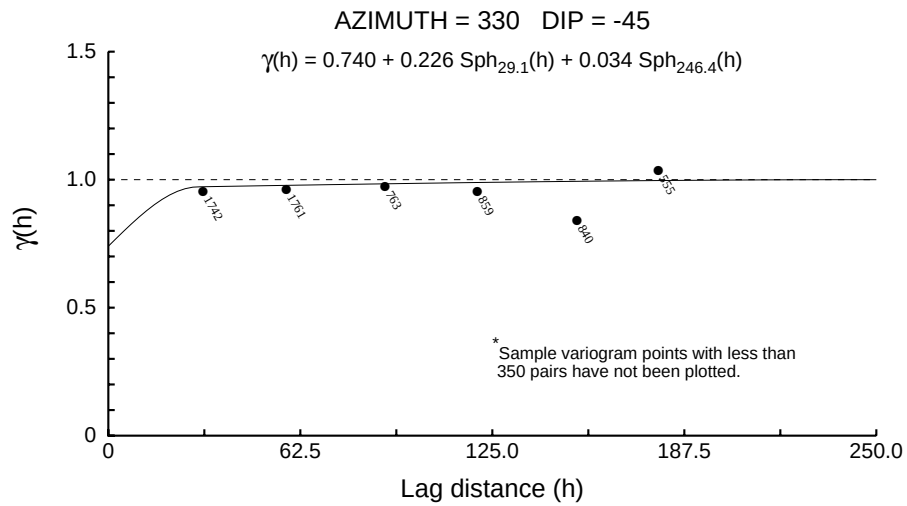
Main Au 2.5m Comps



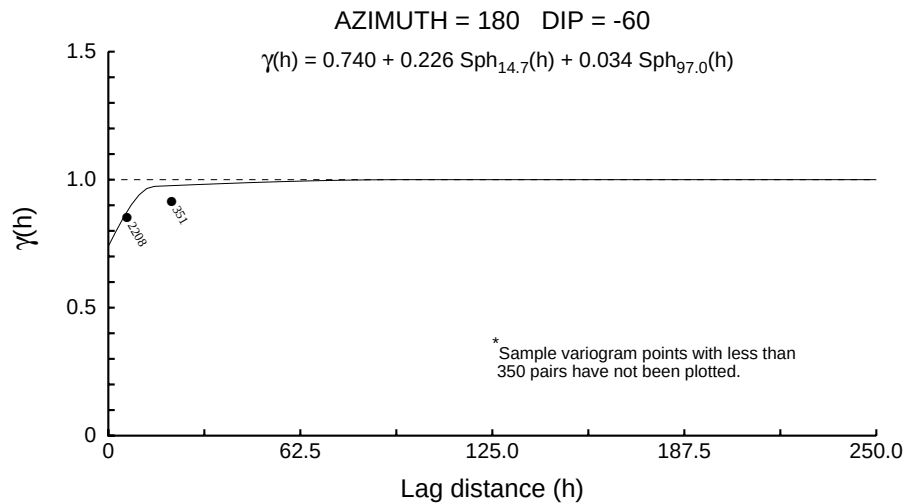
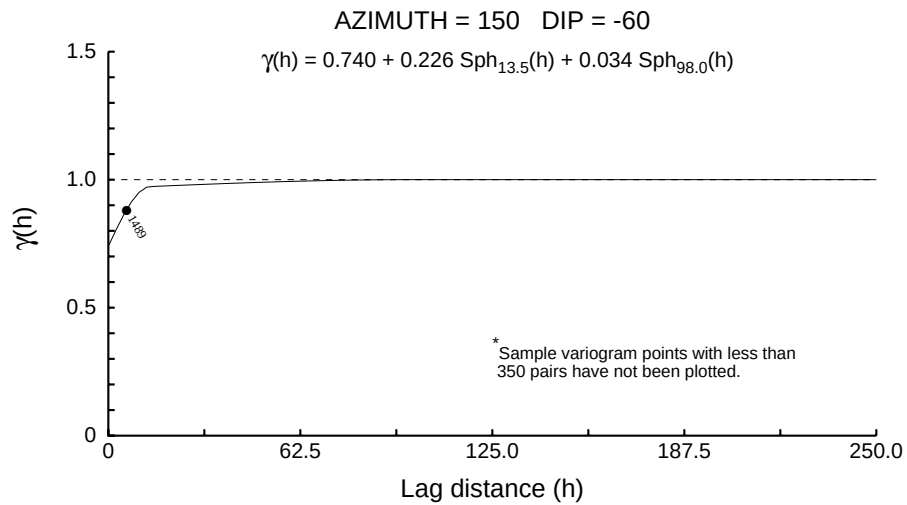
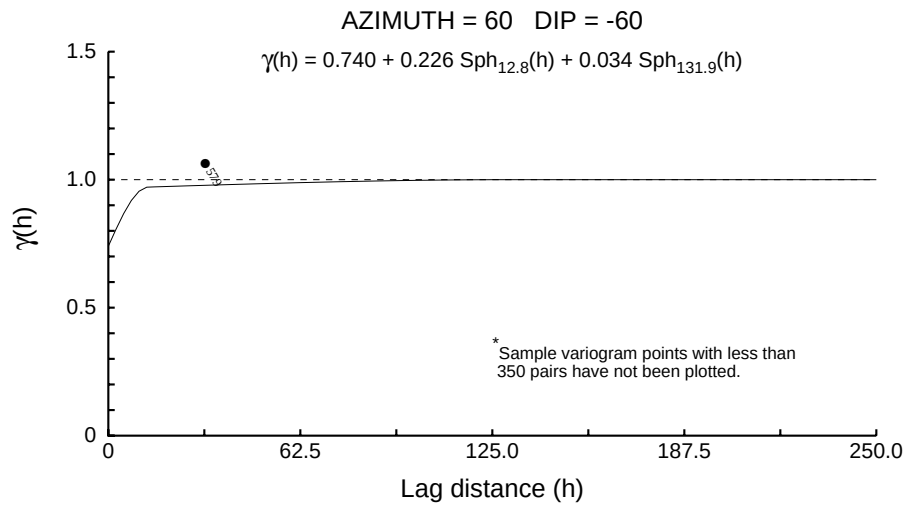
Main Au 2.5m Comps



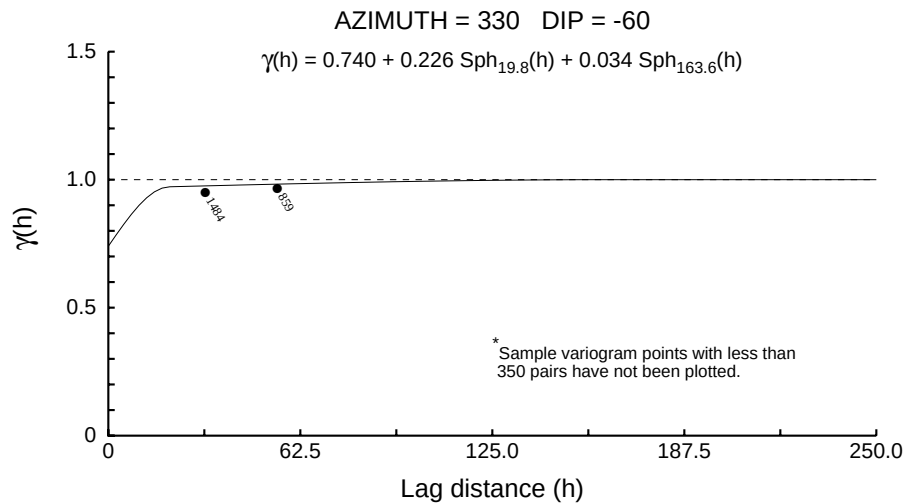
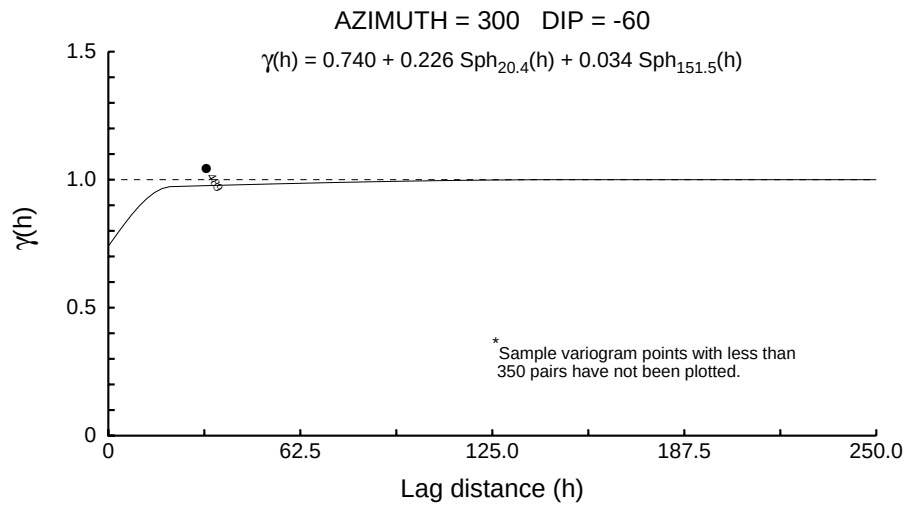
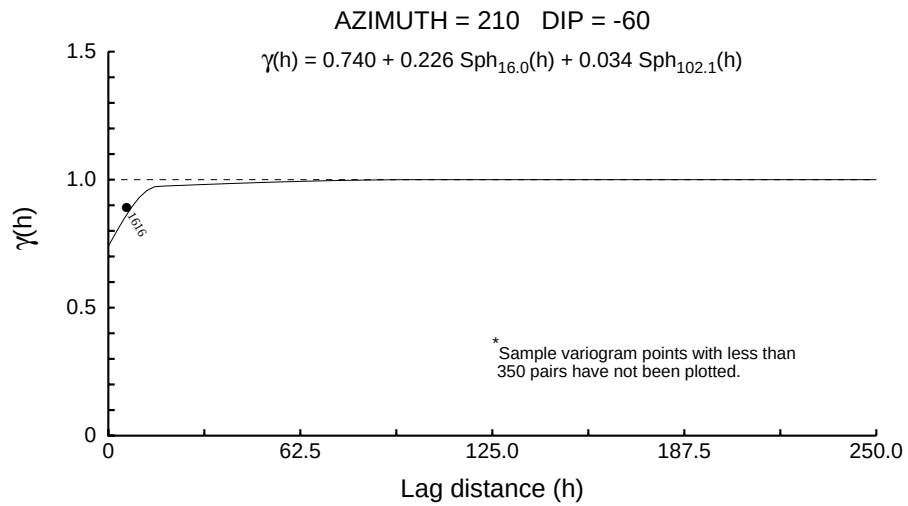
Main Au 2.5m Comps



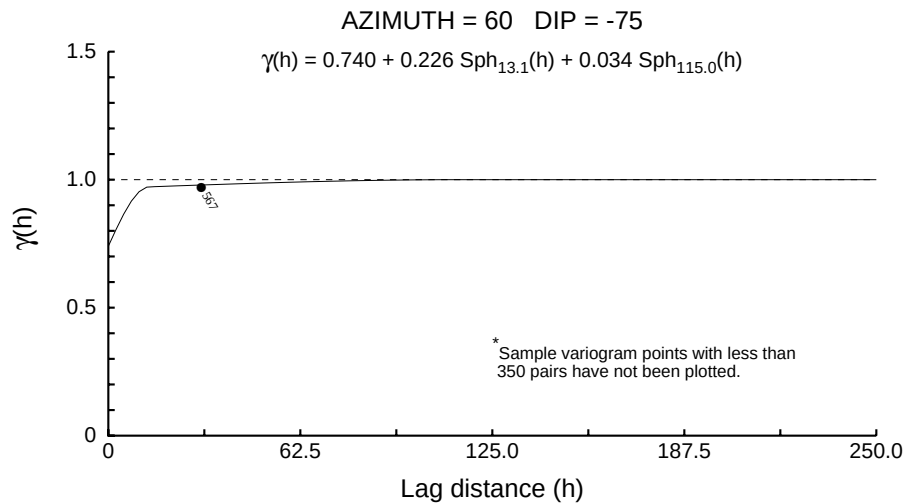
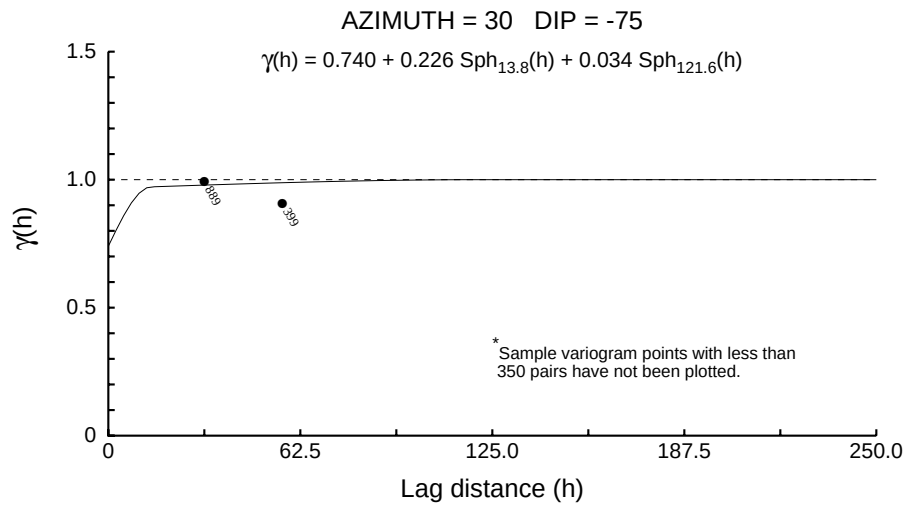
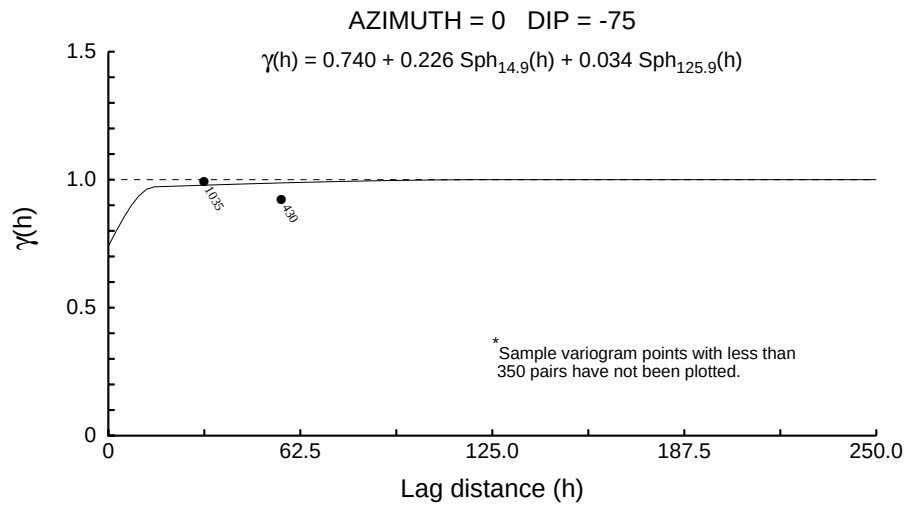
Main Au 2.5m Comps



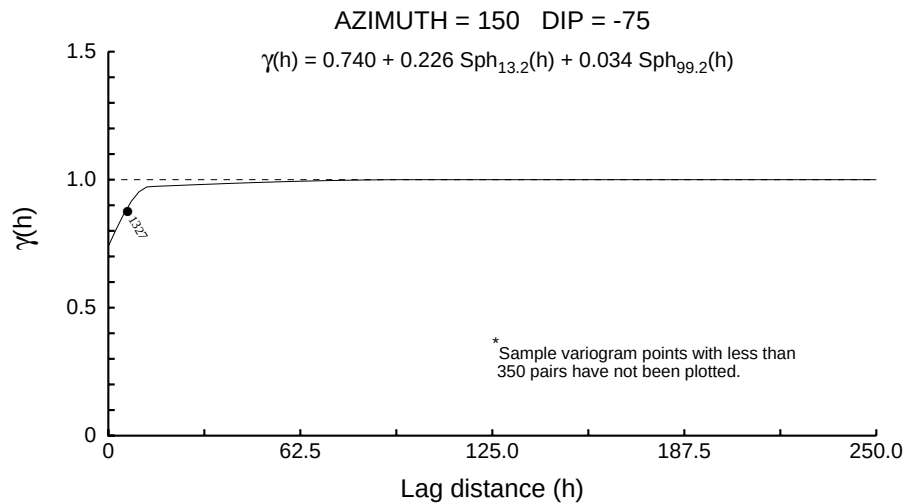
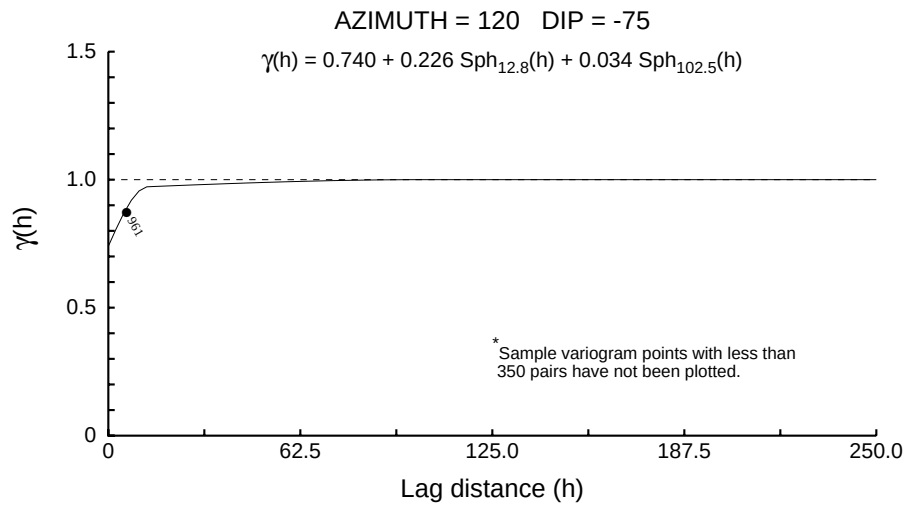
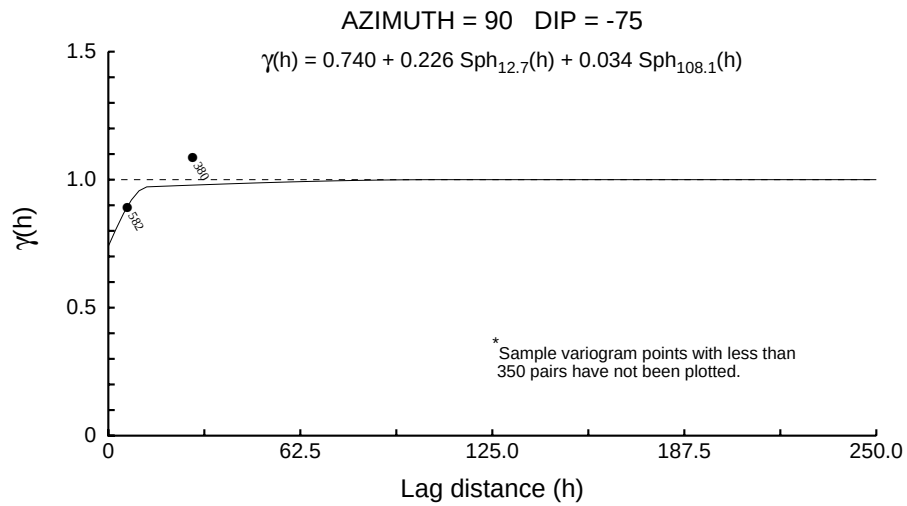
Main Au 2.5m Comps



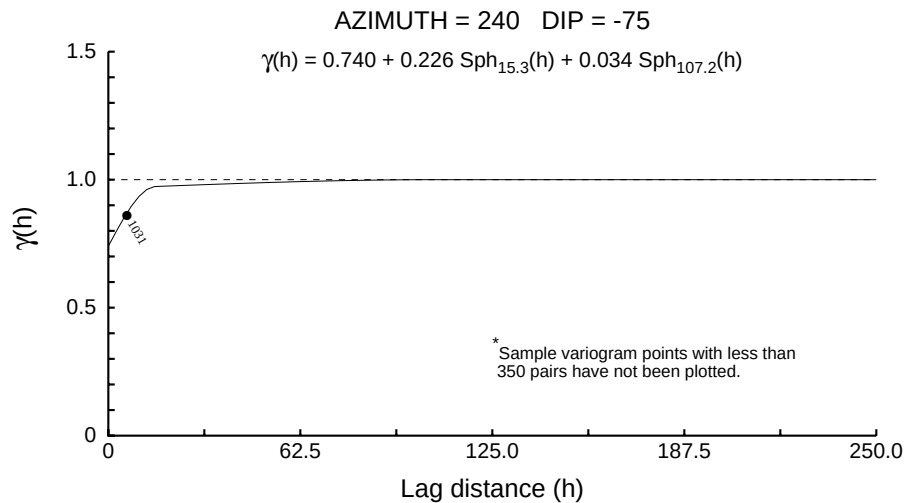
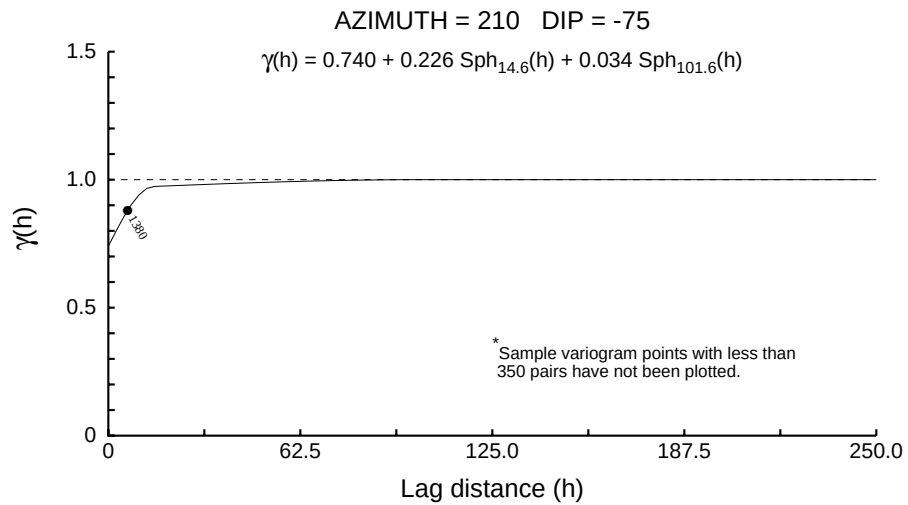
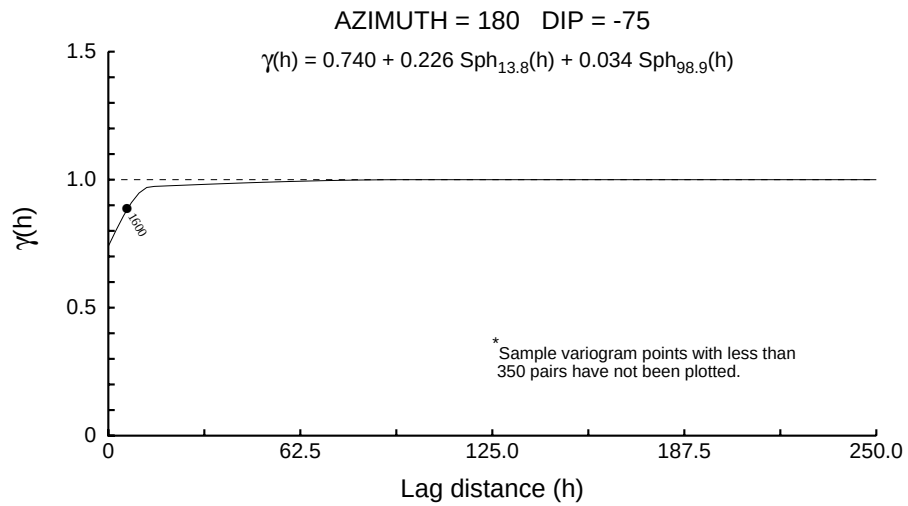
Main Au 2.5m Comps



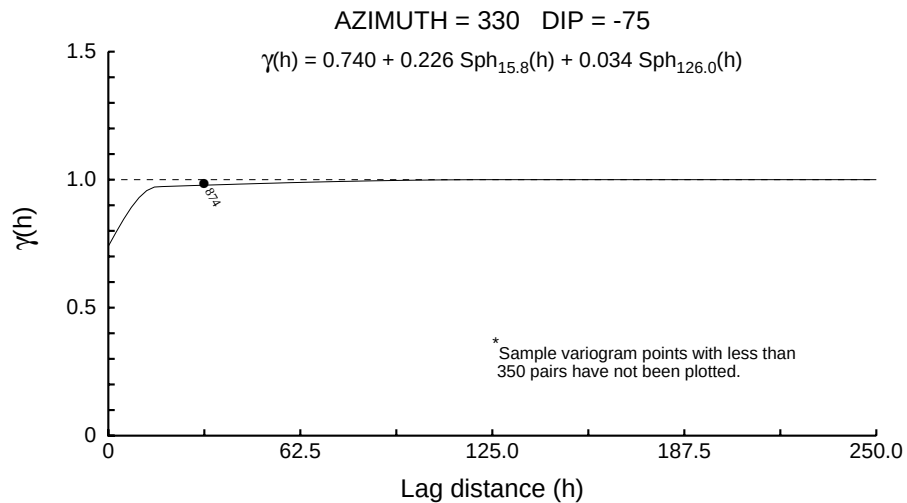
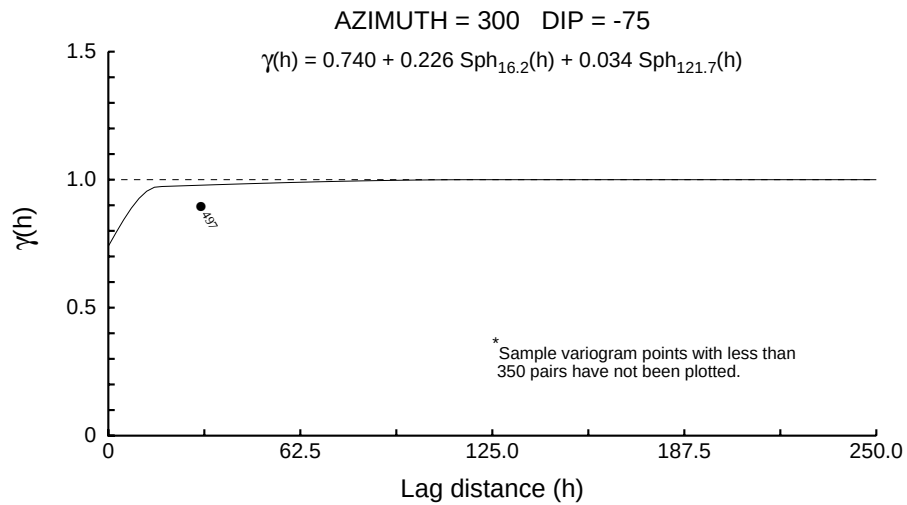
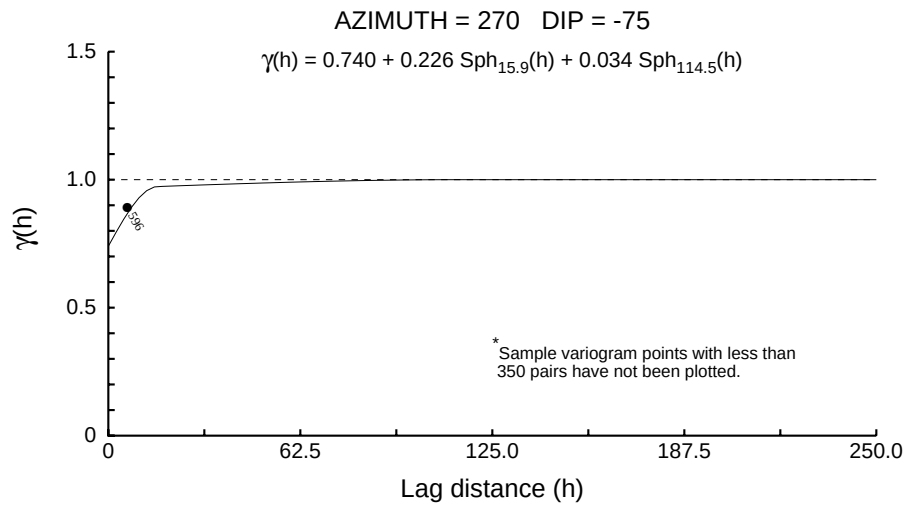
Main Au 2.5m Comps



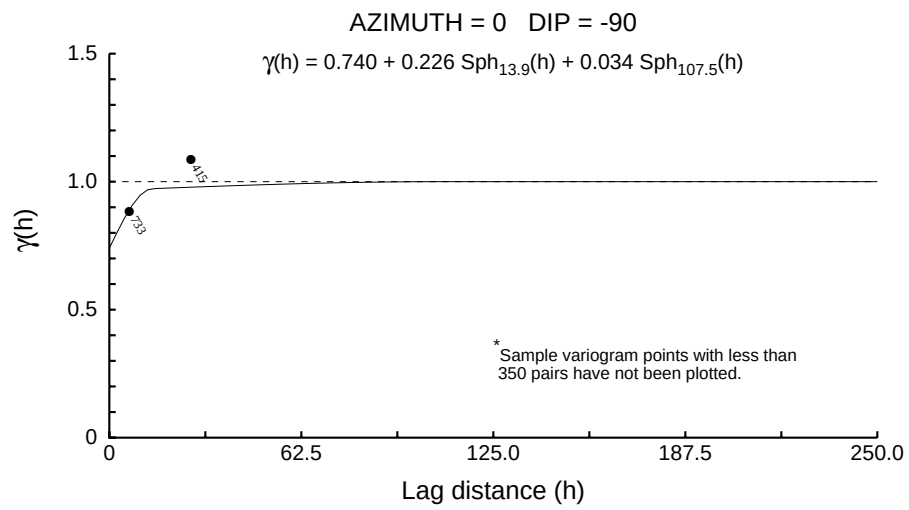
Main Au 2.5m Comps



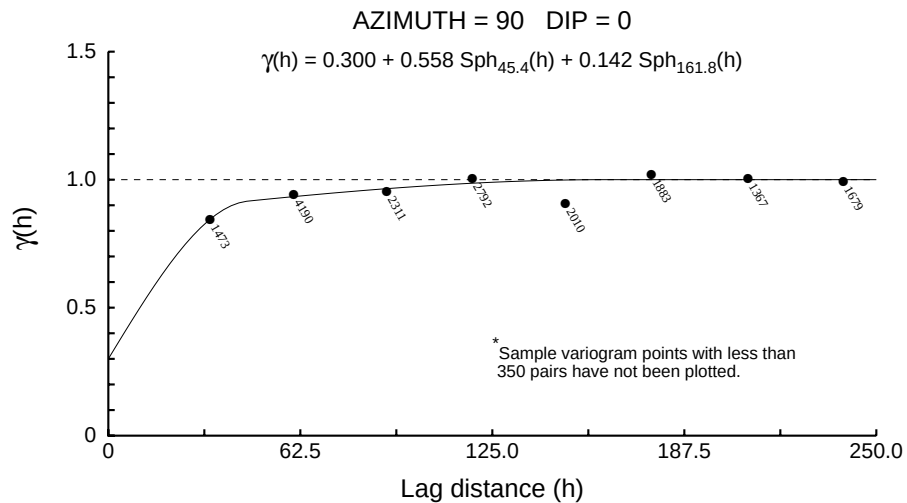
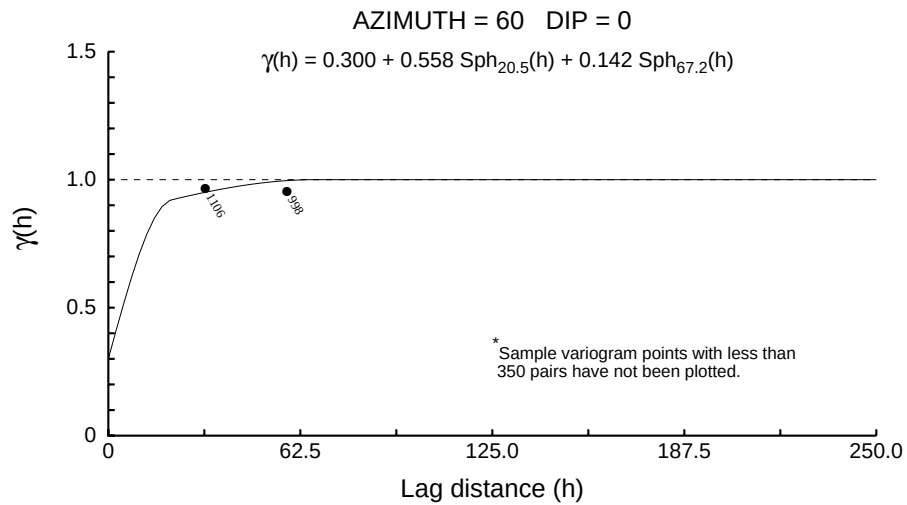
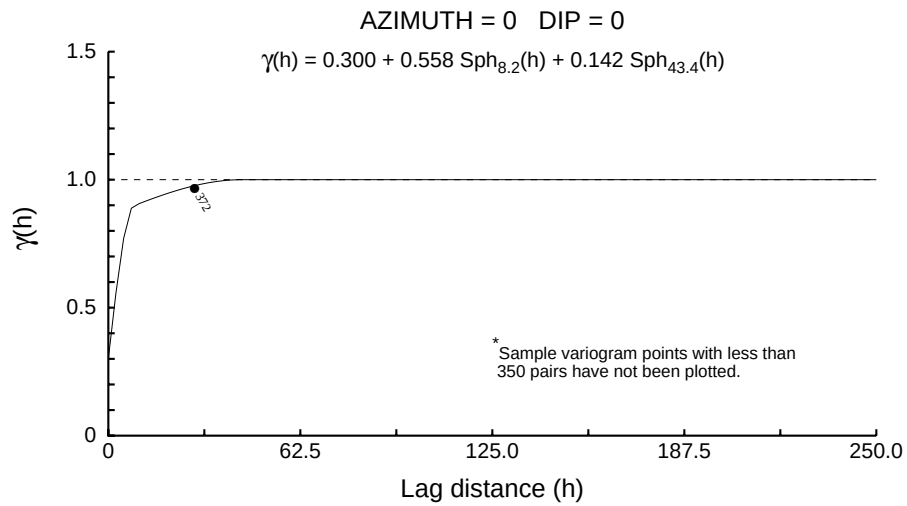
Main Au 2.5m Comps



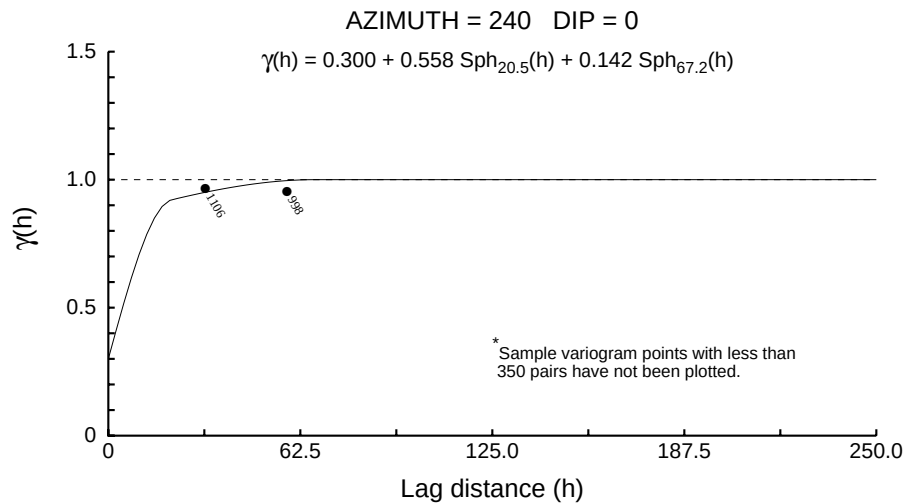
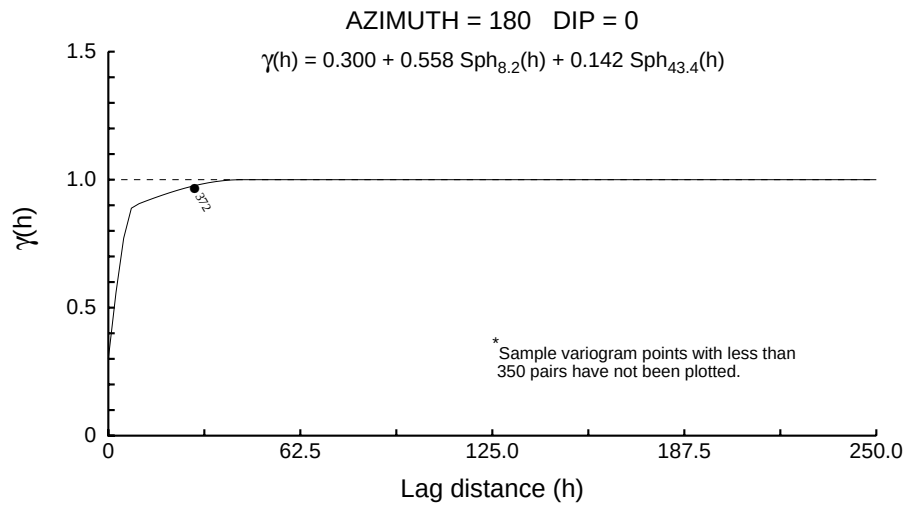
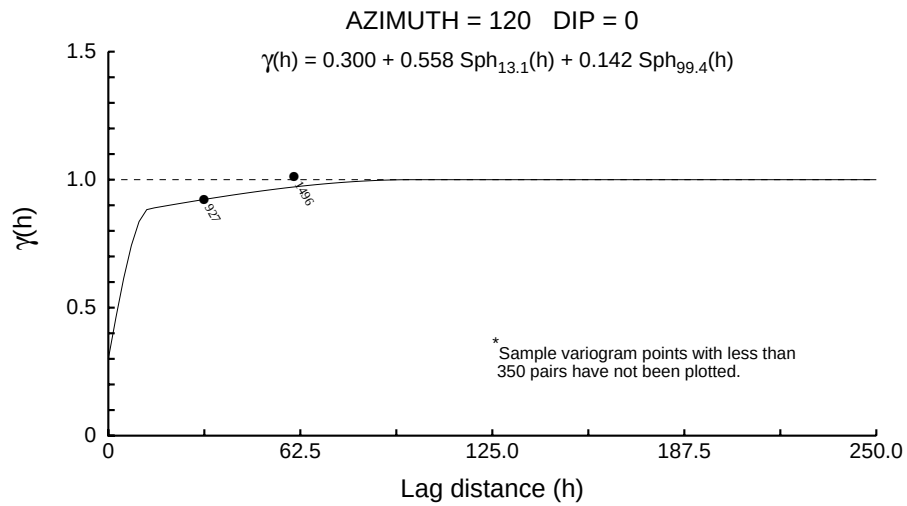
Main Au 2.5m Comps



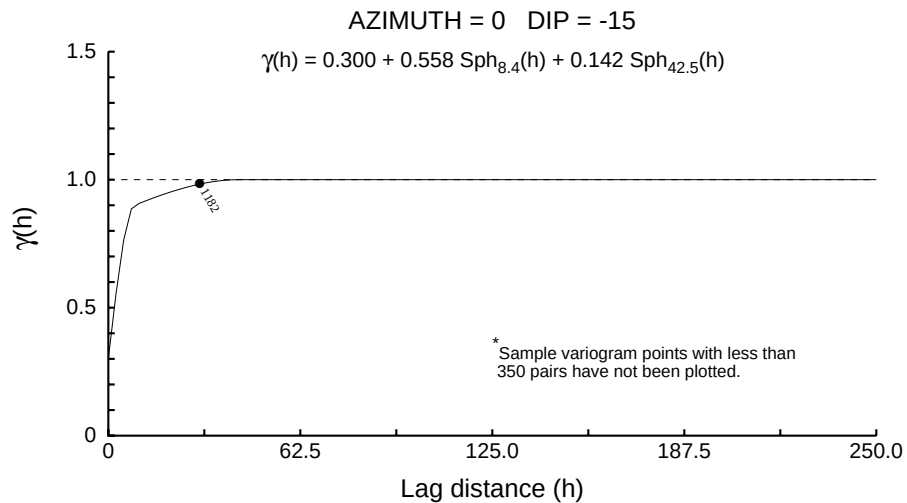
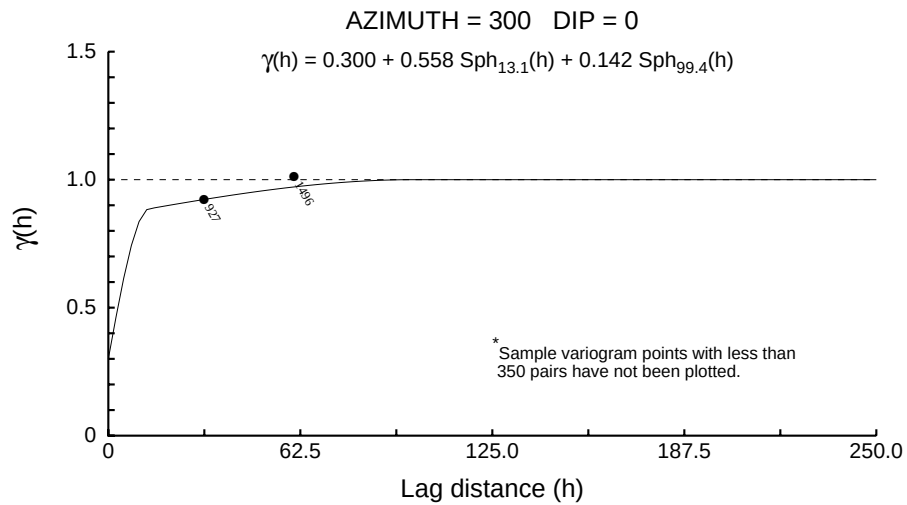
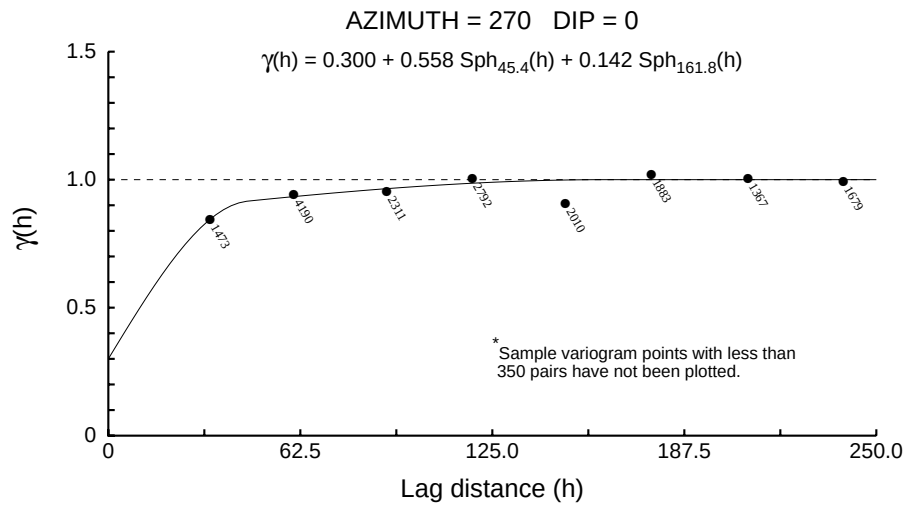
Main Cu 2.5m Comps



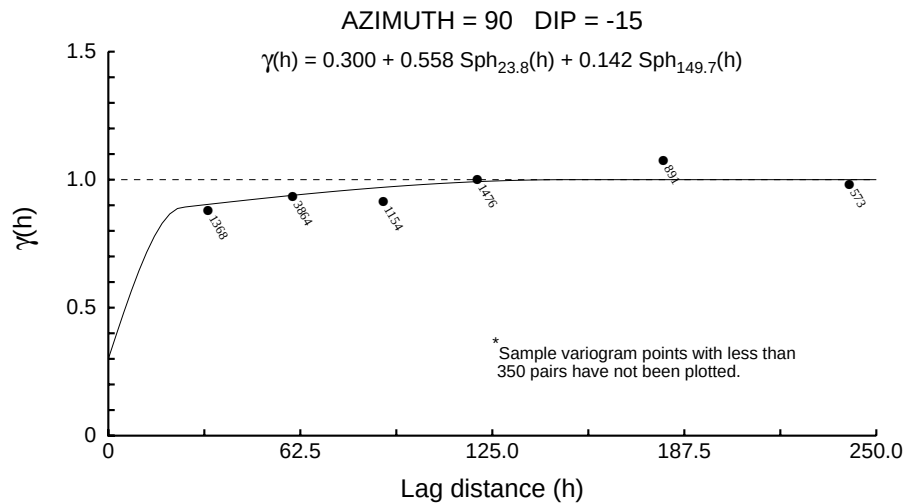
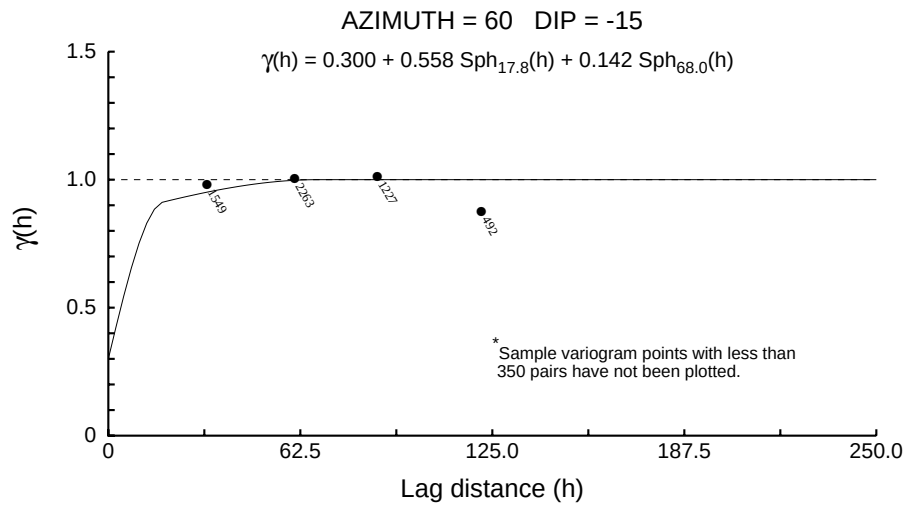
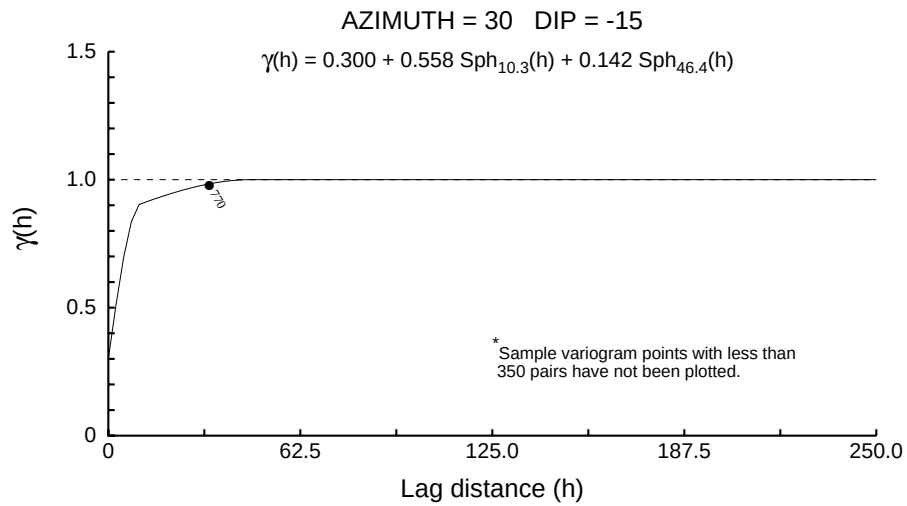
Main Cu 2.5m Comps



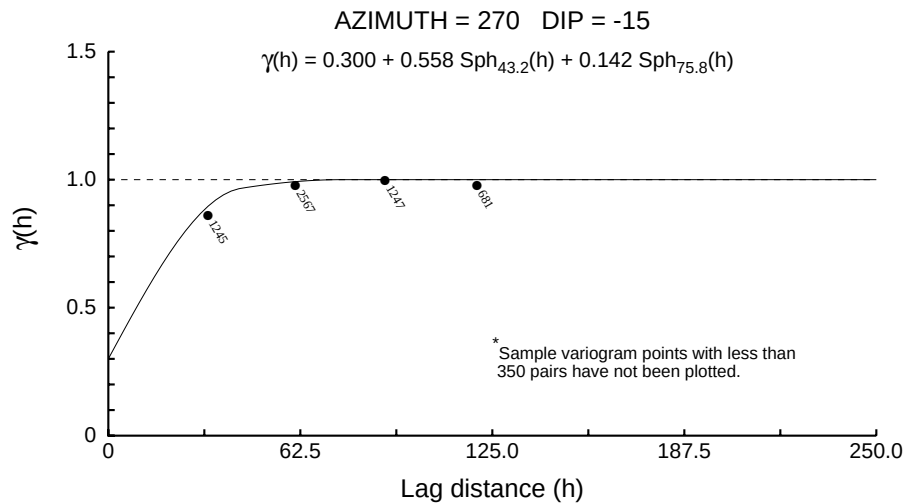
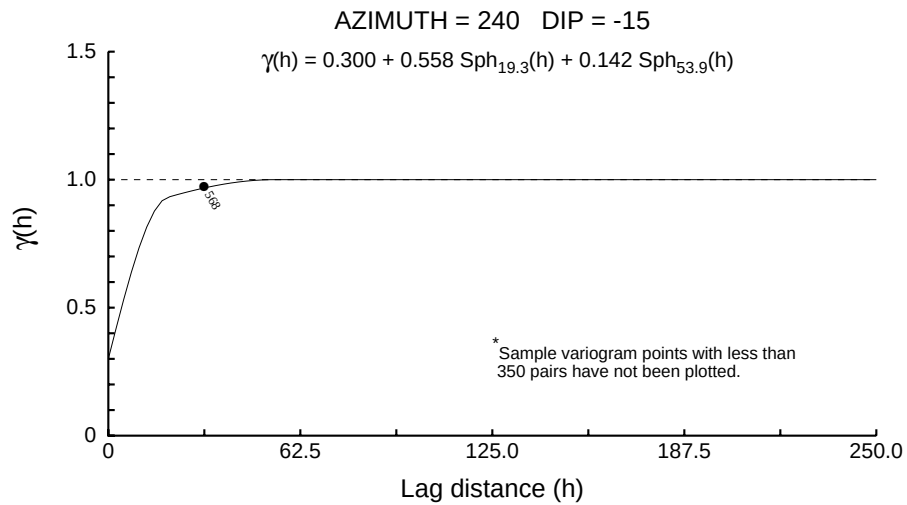
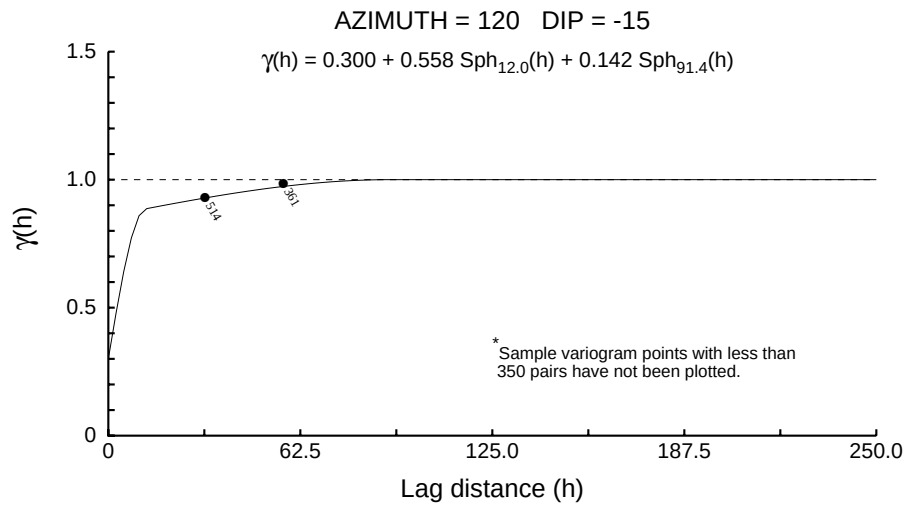
Main Cu 2.5m Comps



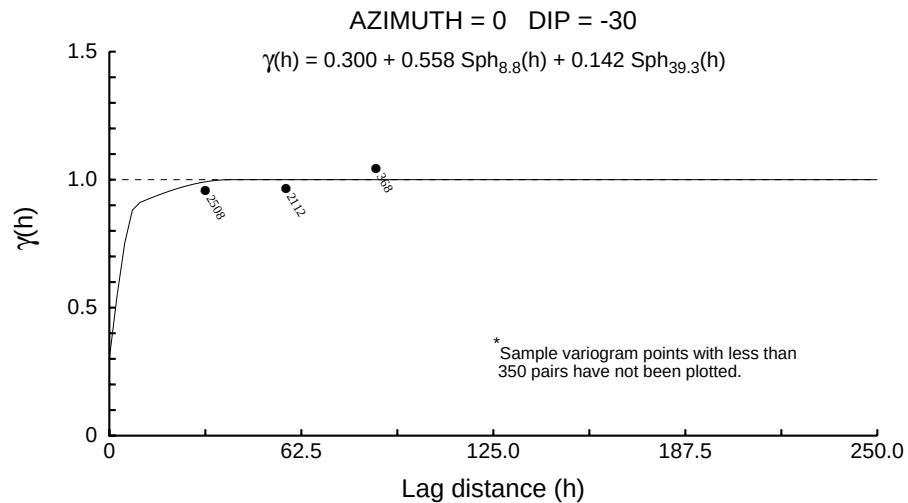
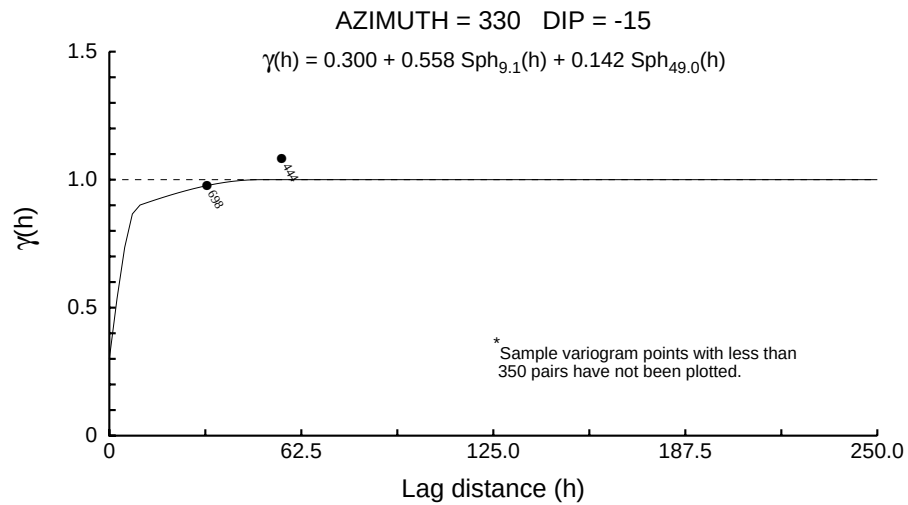
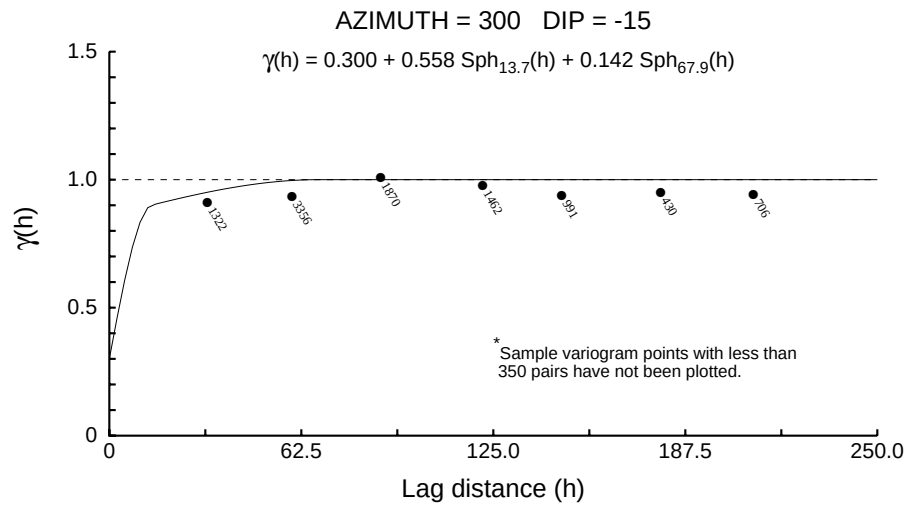
Main Cu 2.5m Comps



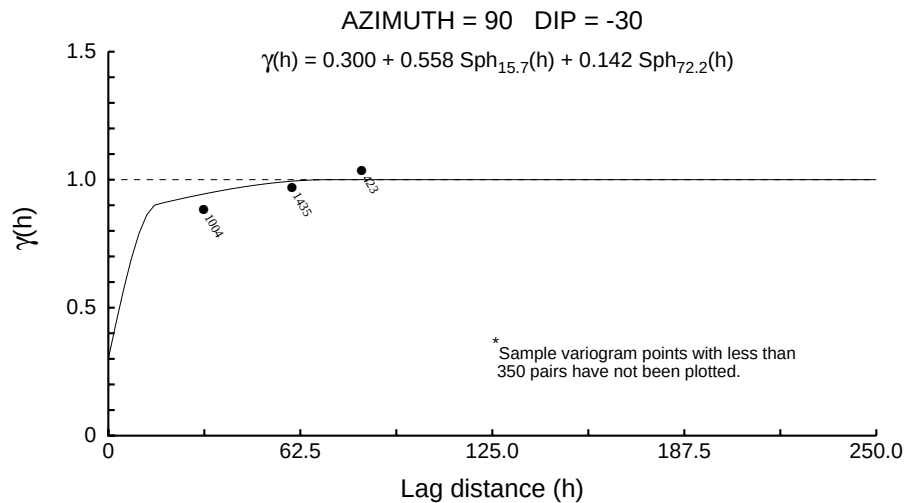
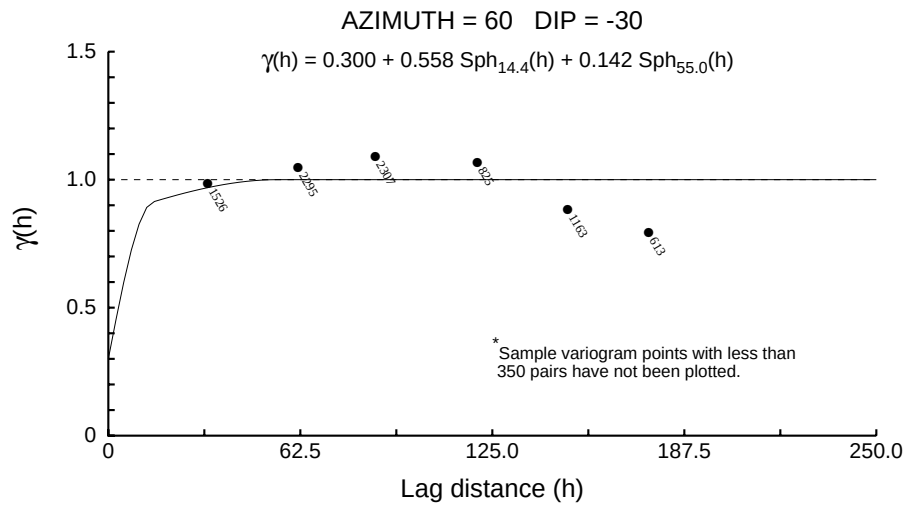
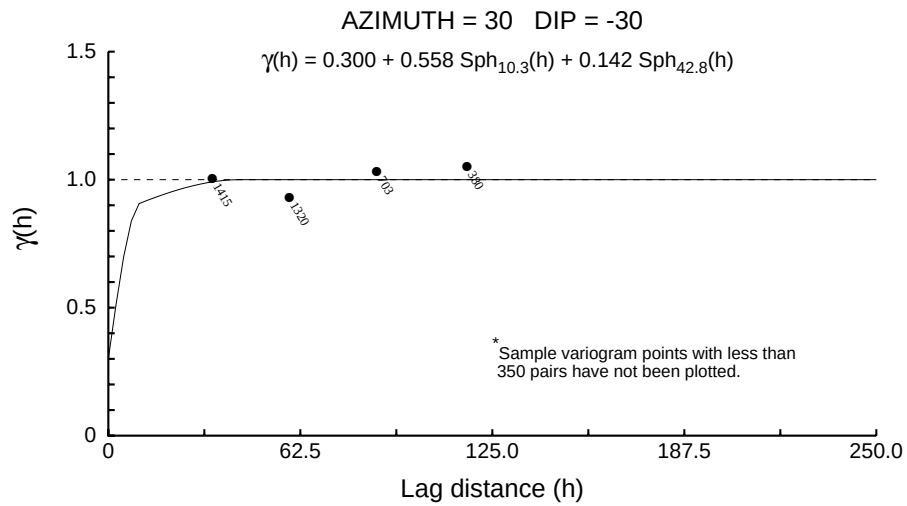
Main Cu 2.5m Comps



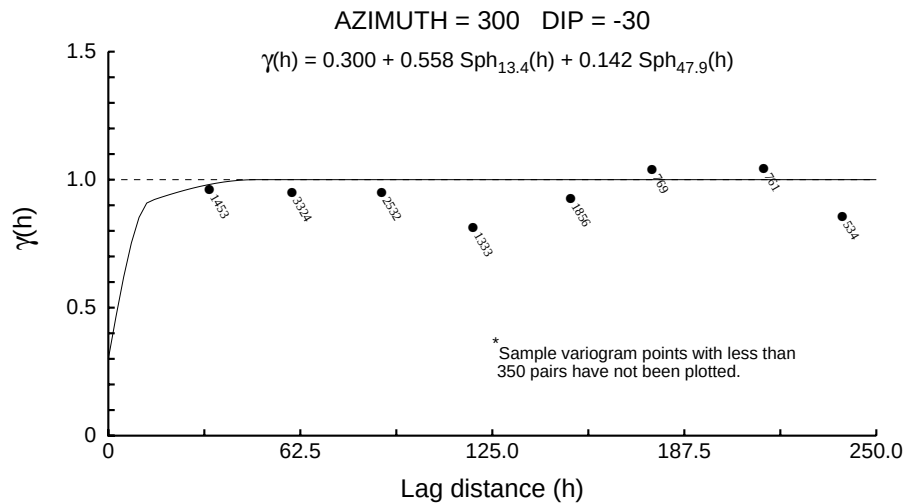
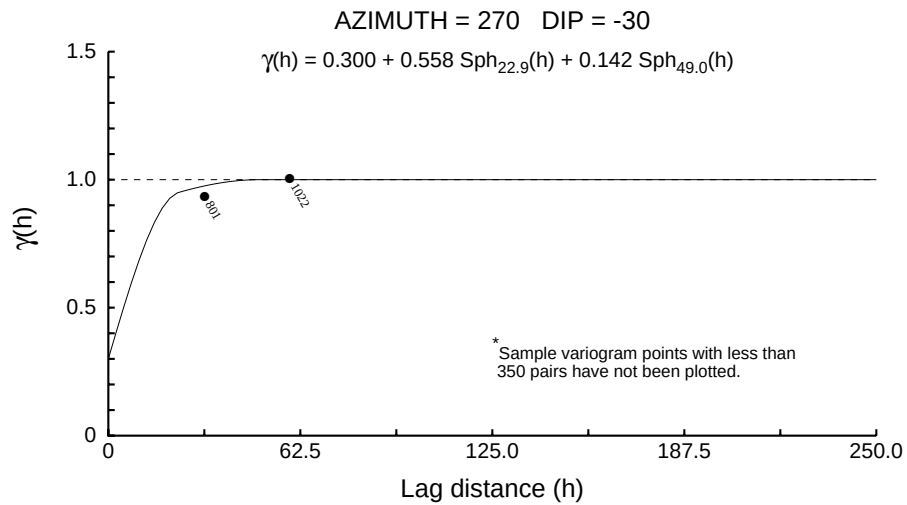
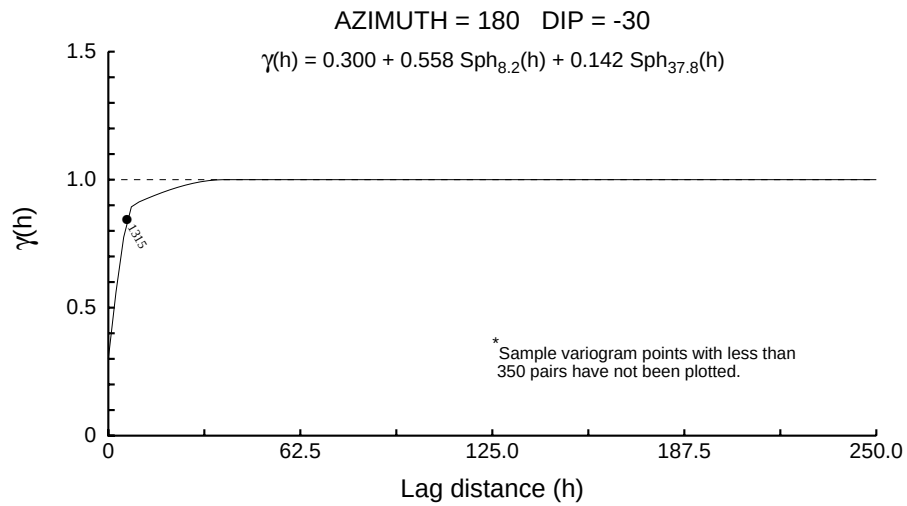
Main Cu 2.5m Comps



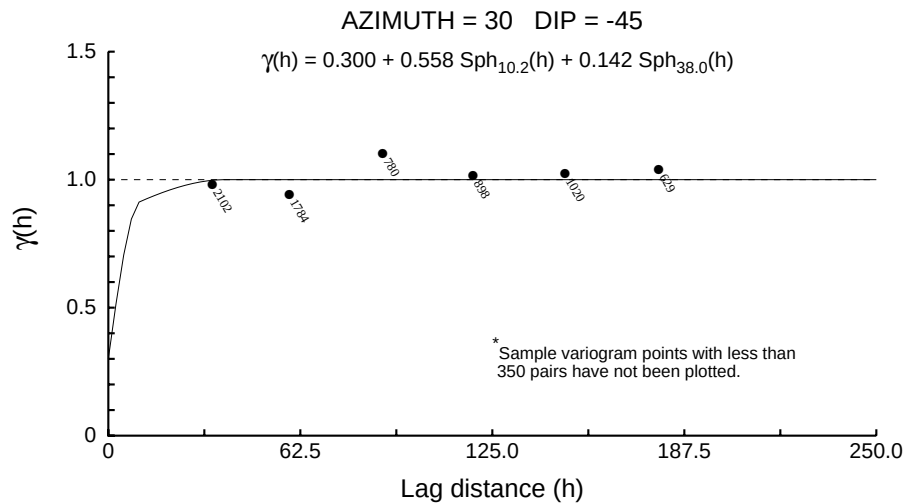
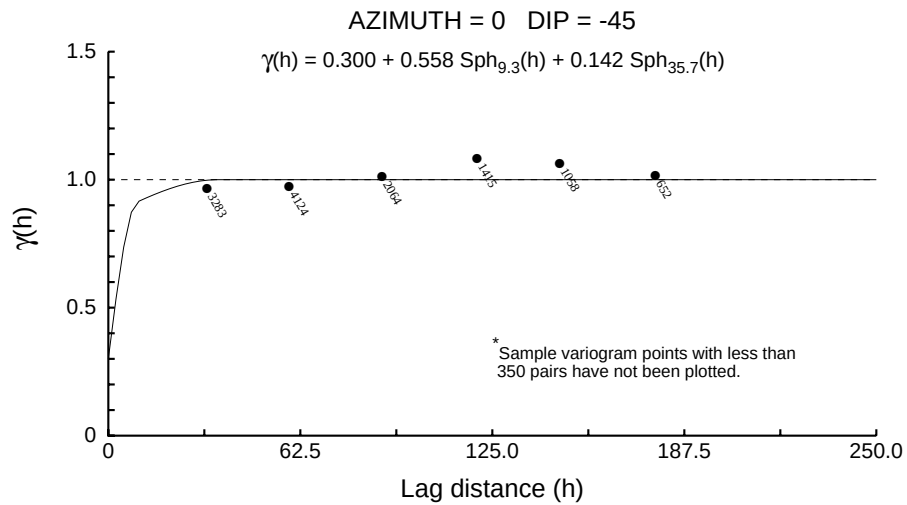
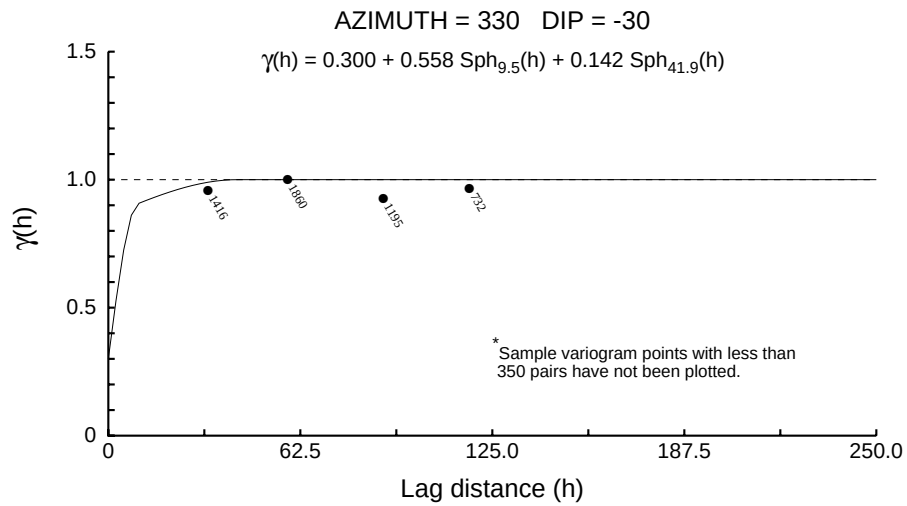
Main Cu 2.5m Comps



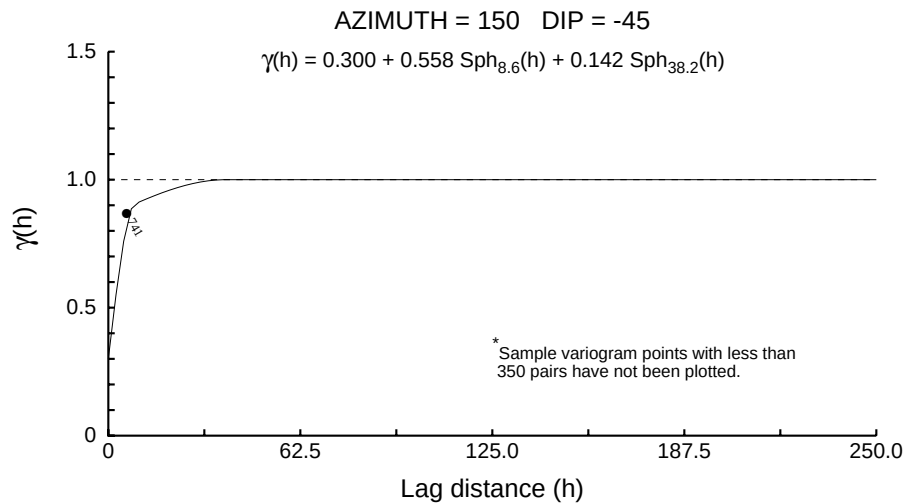
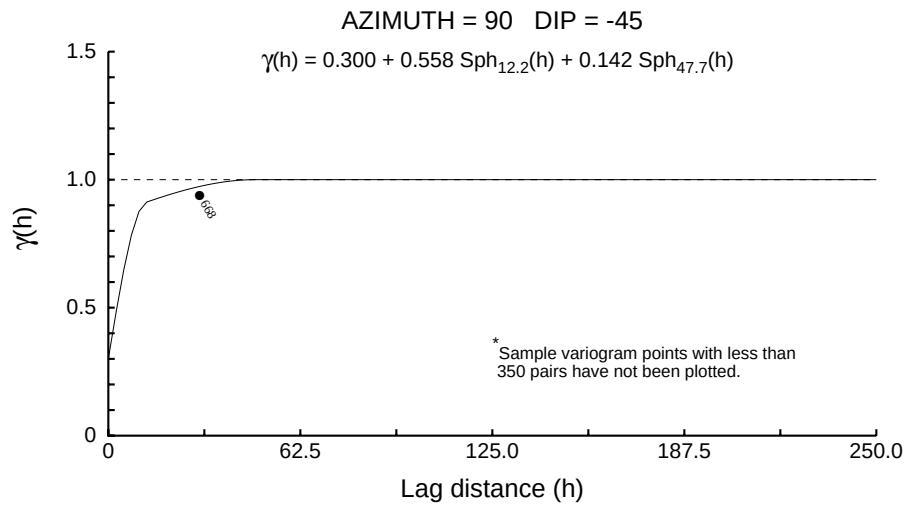
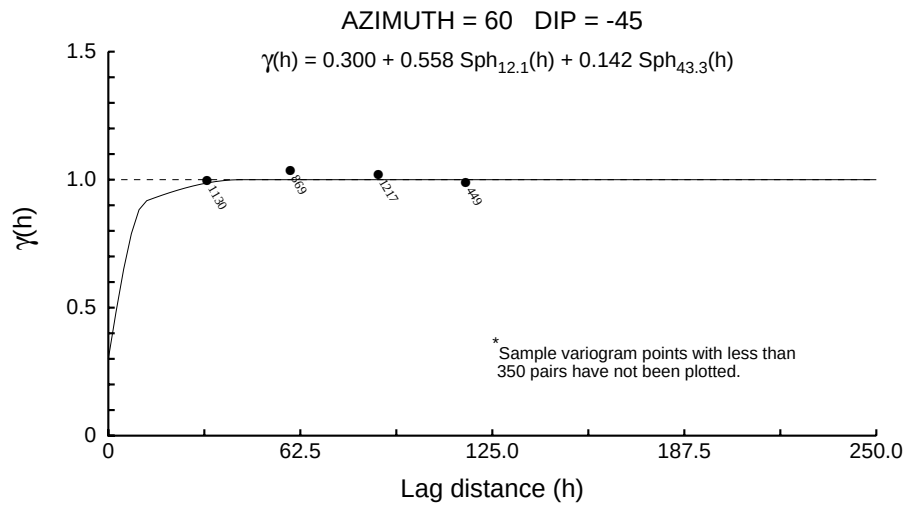
Main Cu 2.5m Comps



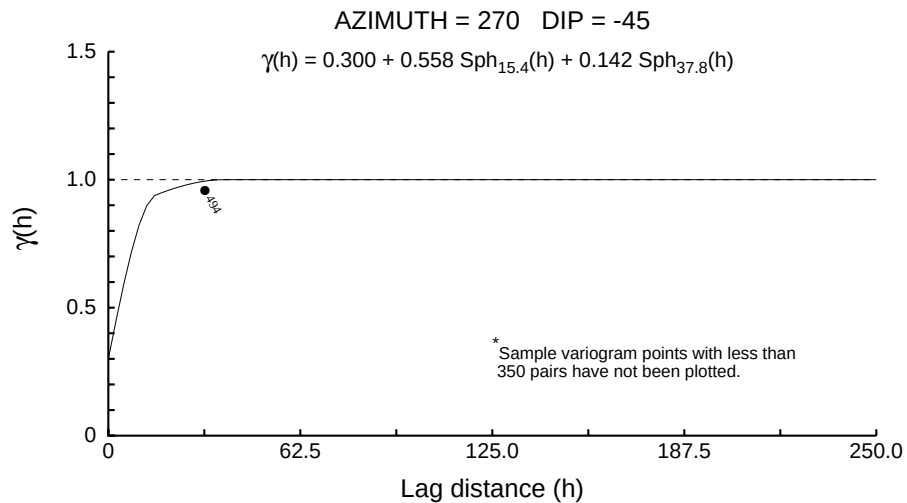
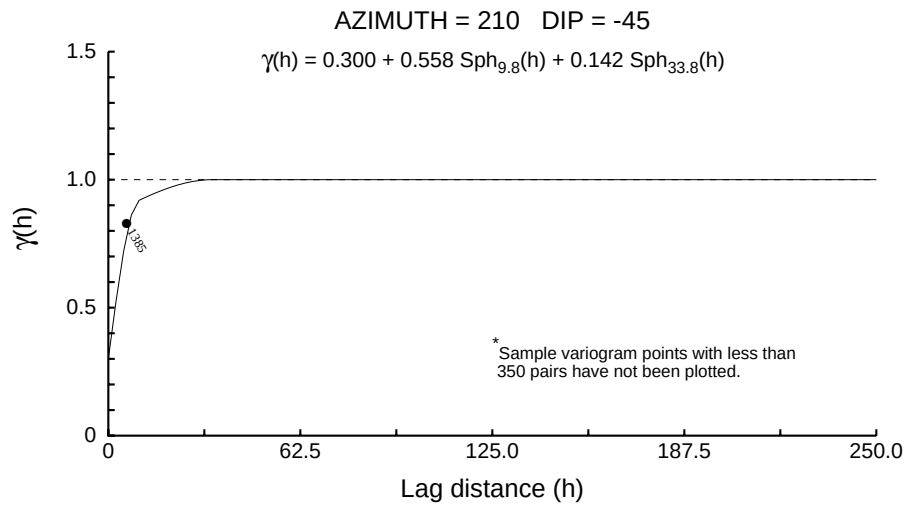
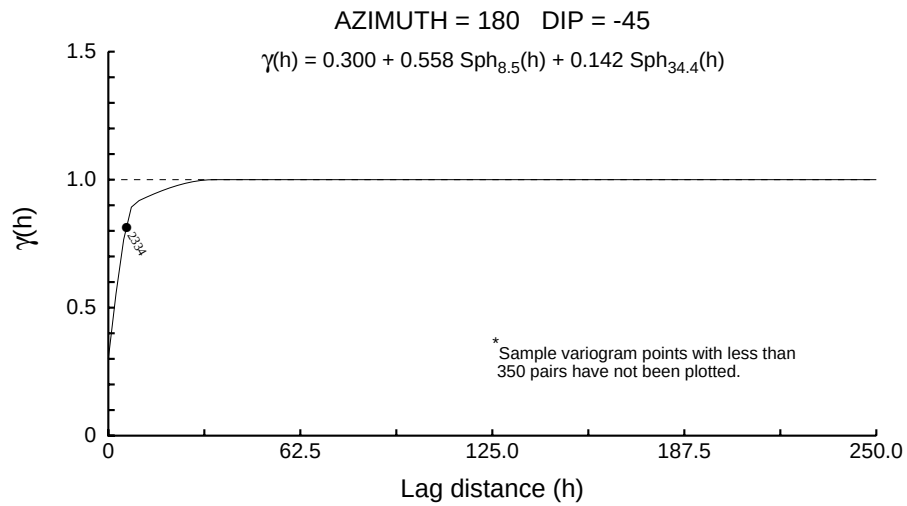
Main Cu 2.5m Comps



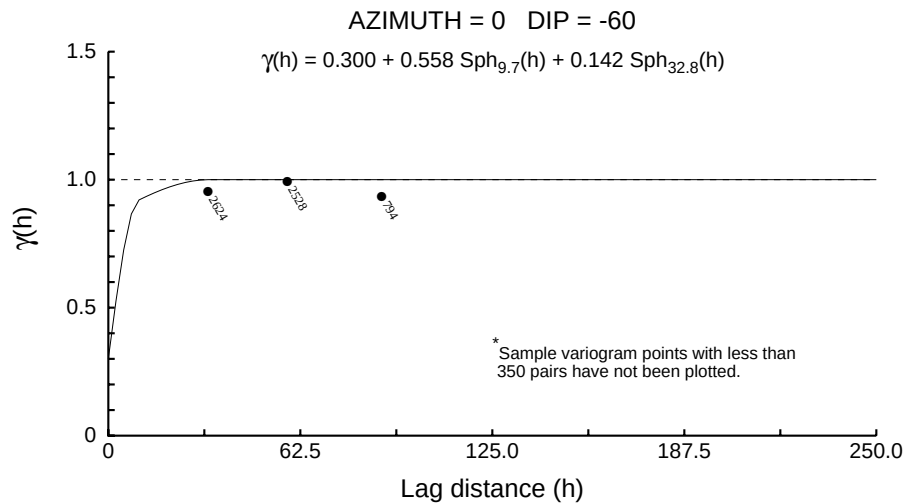
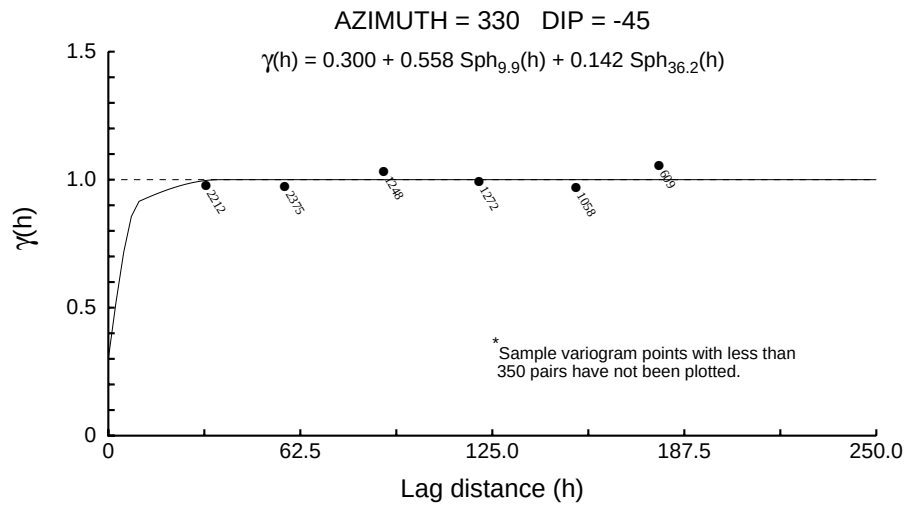
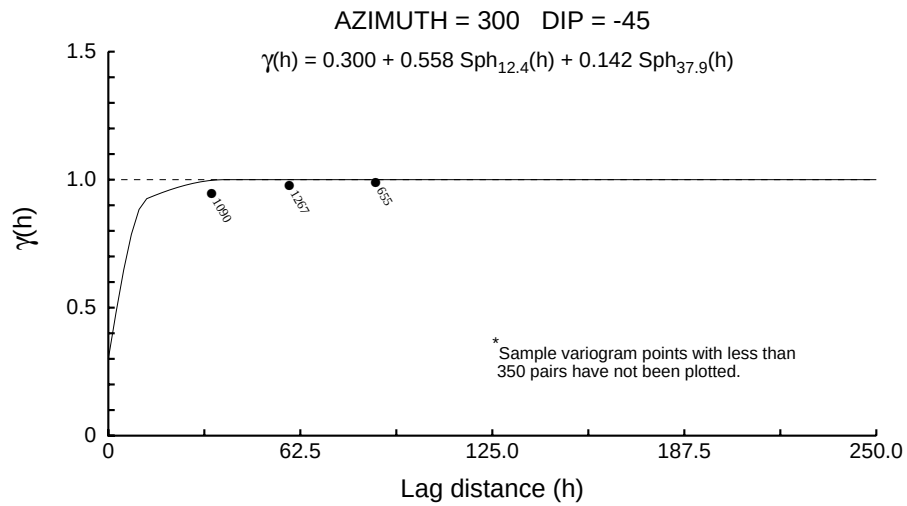
Main Cu 2.5m Comps



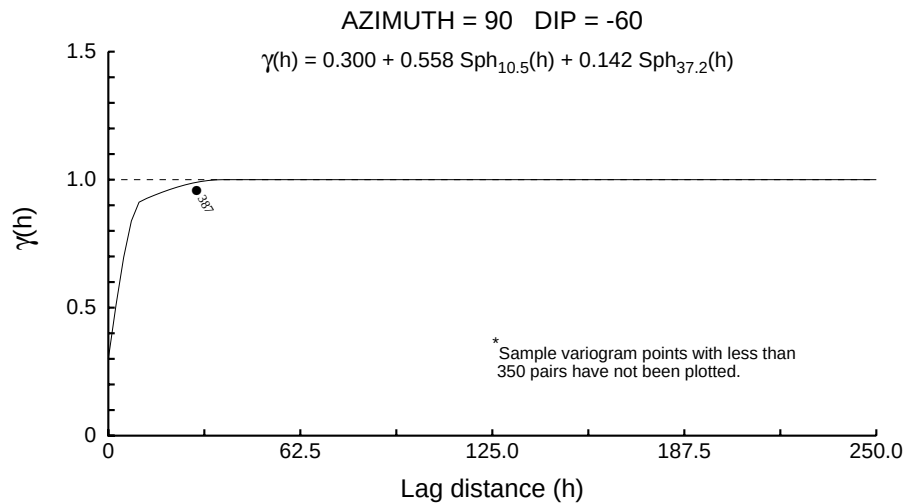
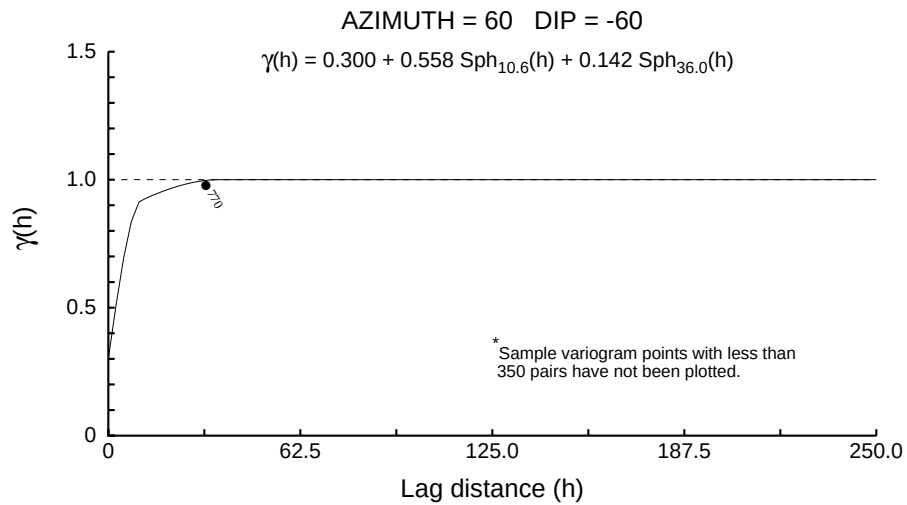
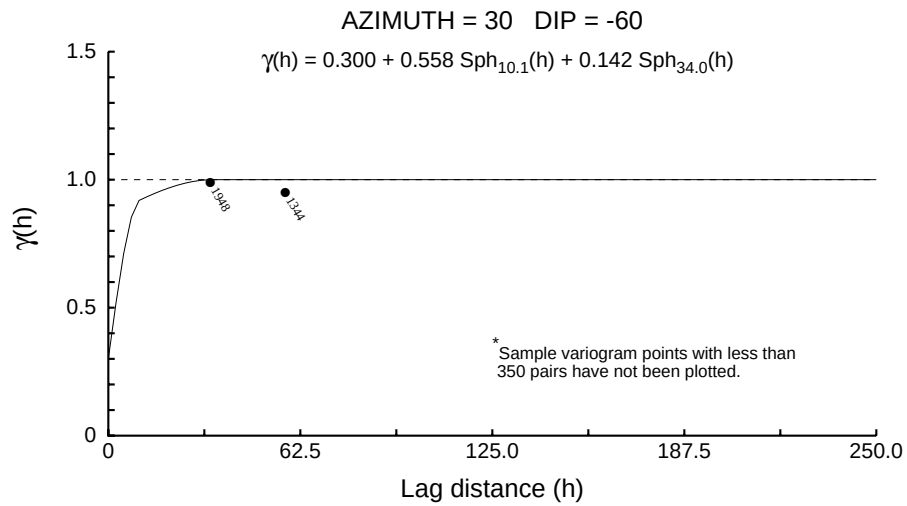
Main Cu 2.5m Comps



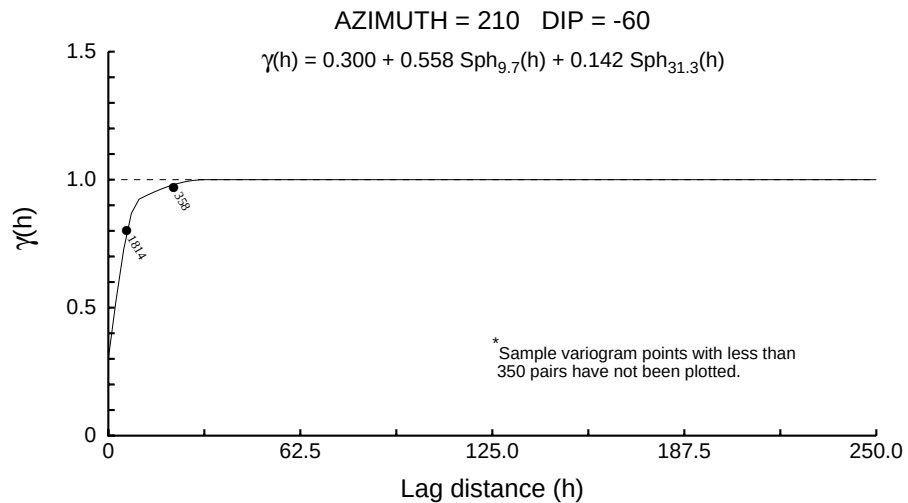
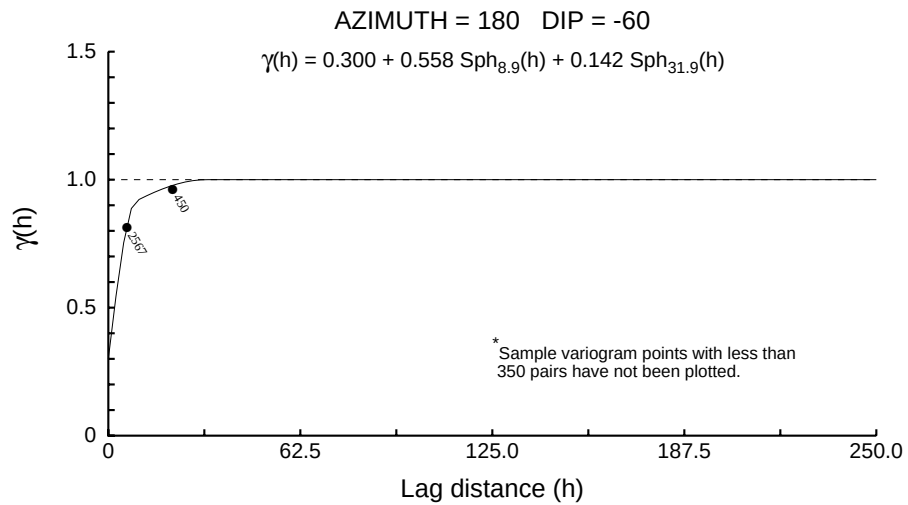
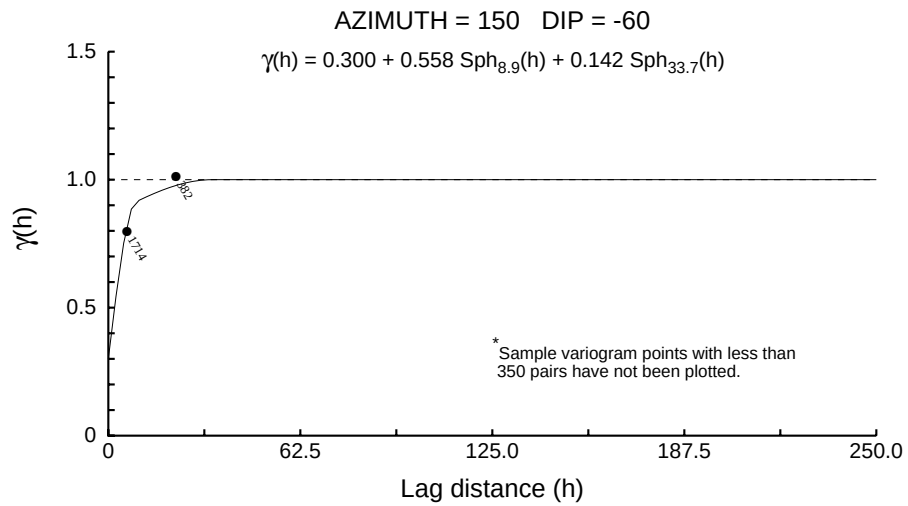
Main Cu 2.5m Comps



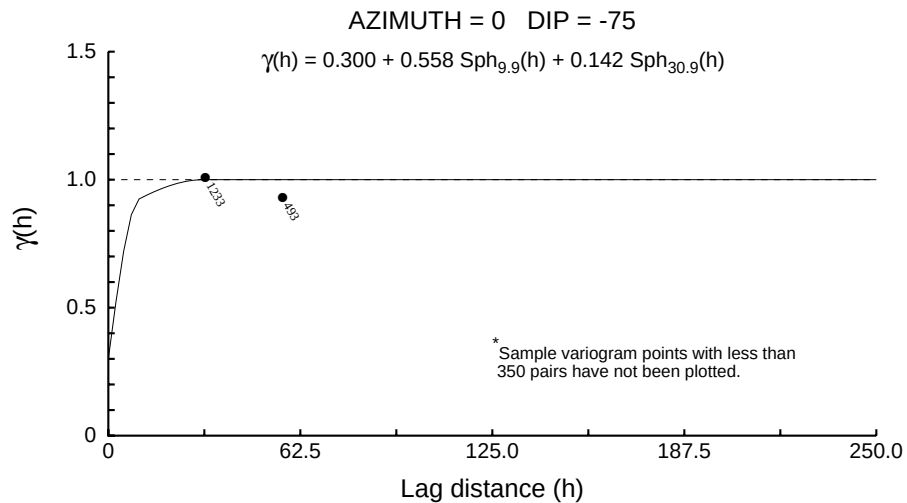
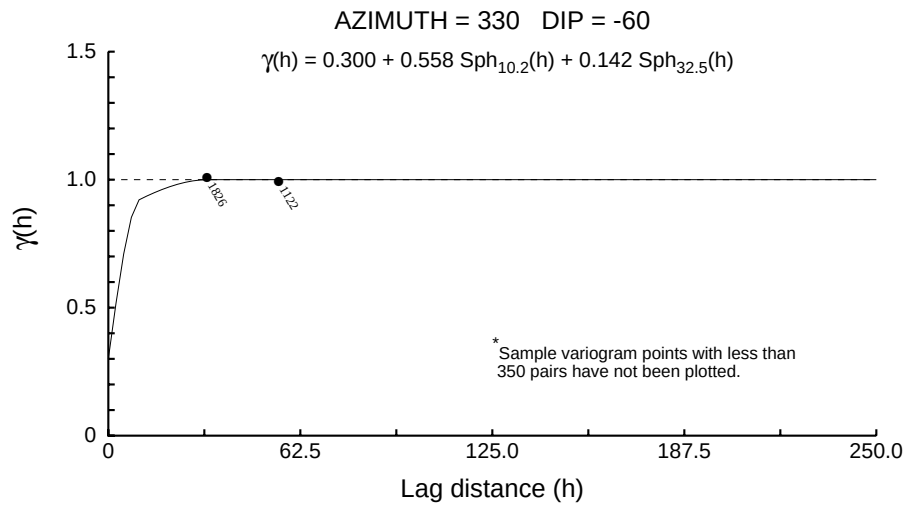
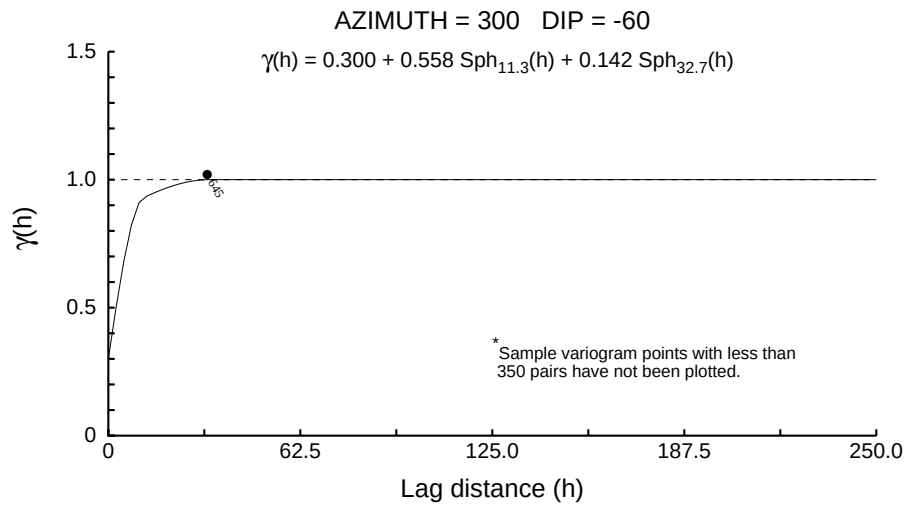
Main Cu 2.5m Comps



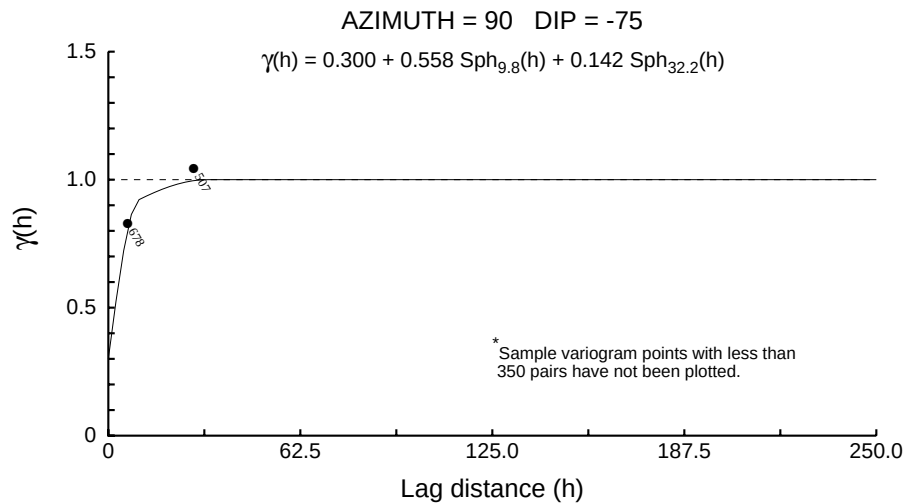
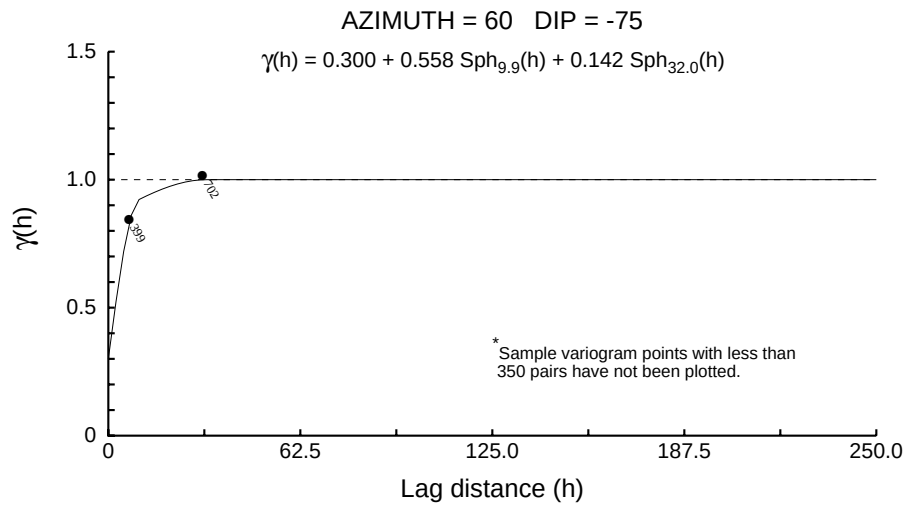
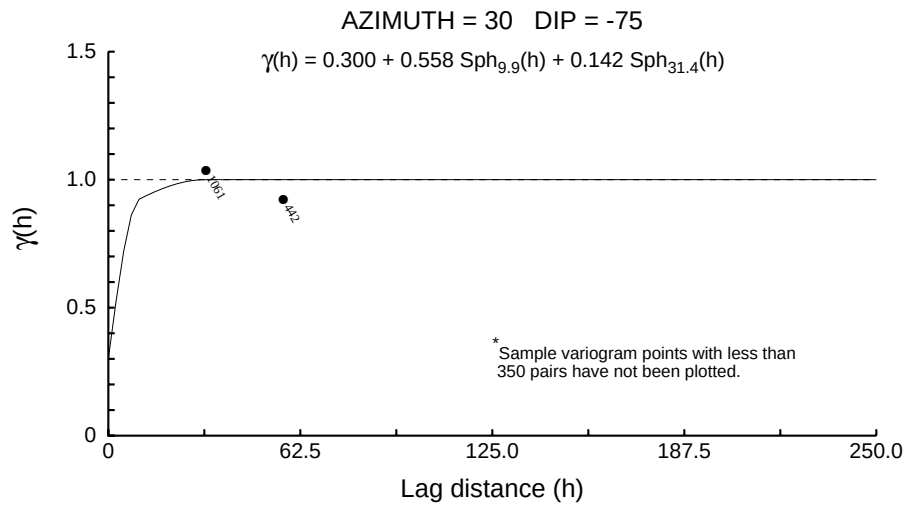
Main Cu 2.5m Comps



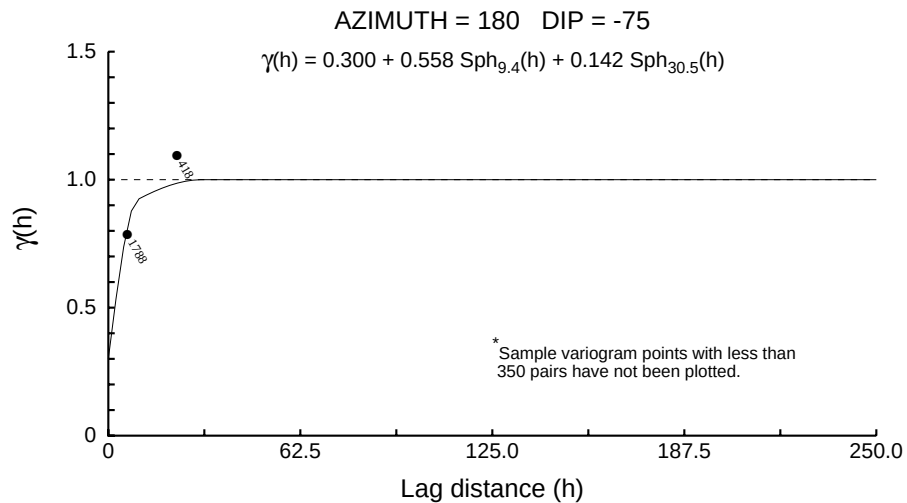
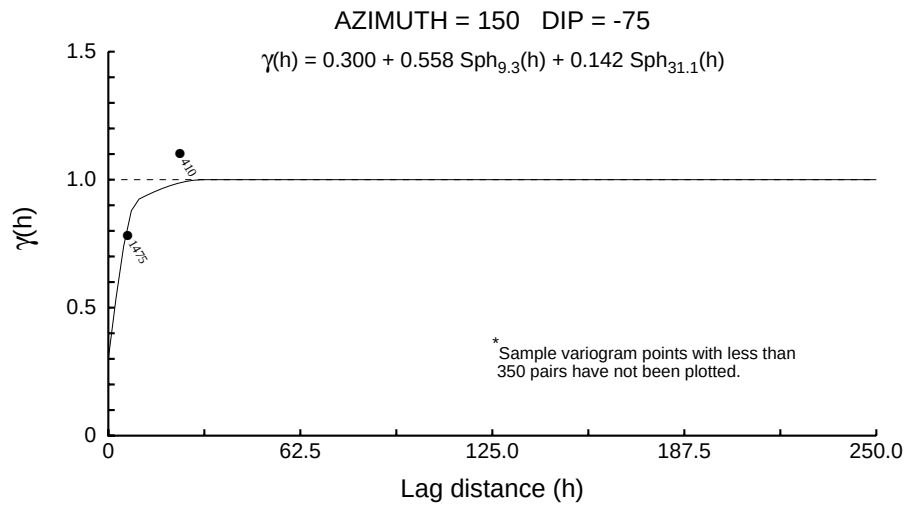
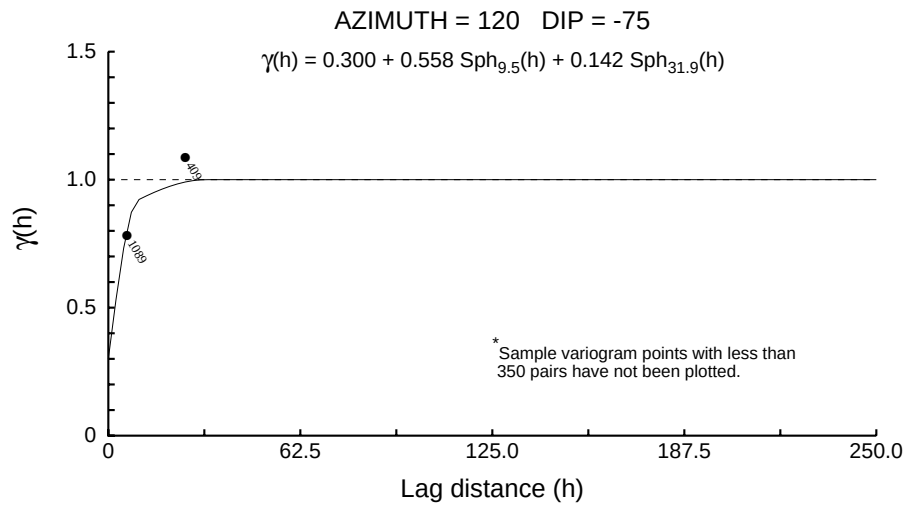
Main Cu 2.5m Comps



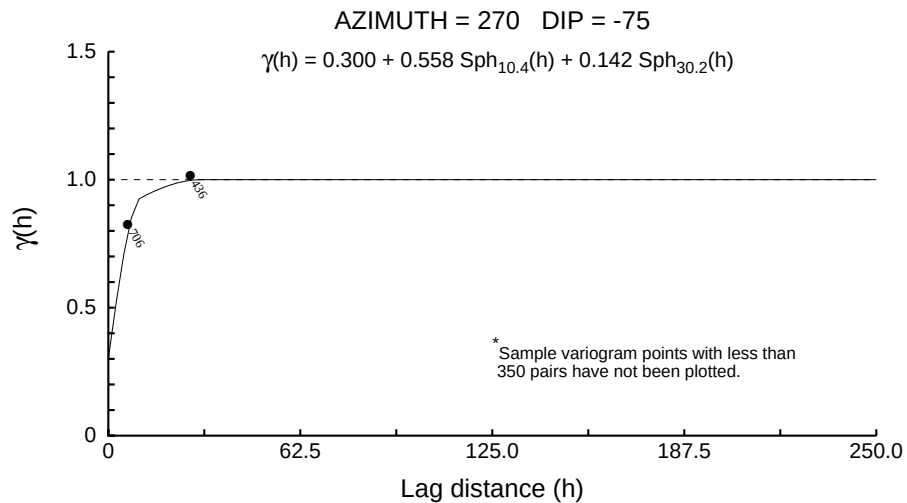
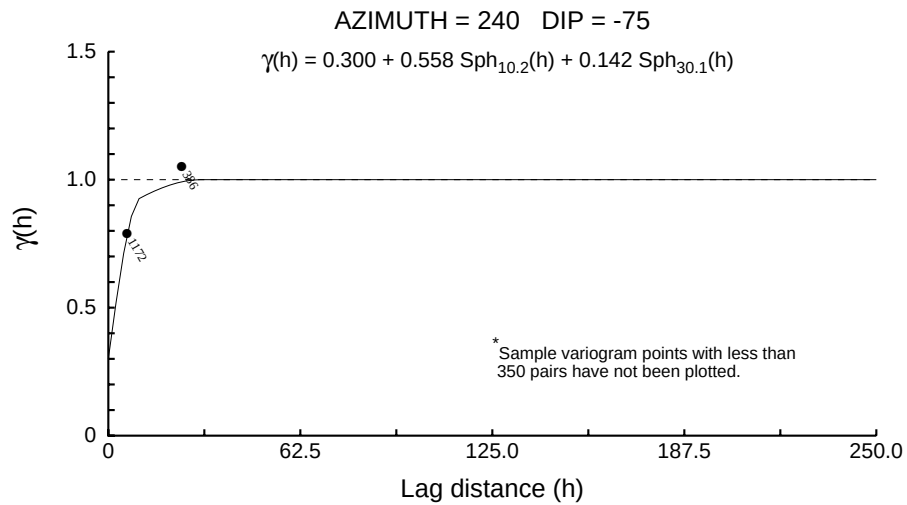
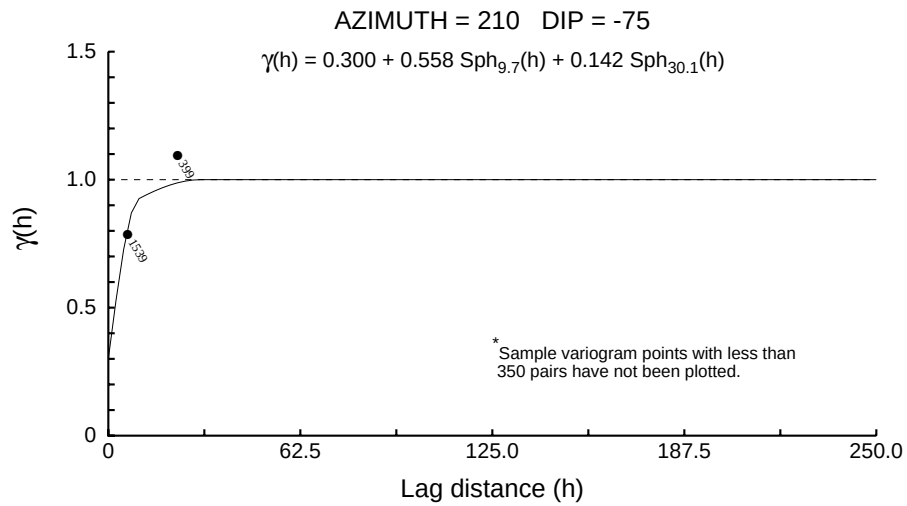
Main Cu 2.5m Comps



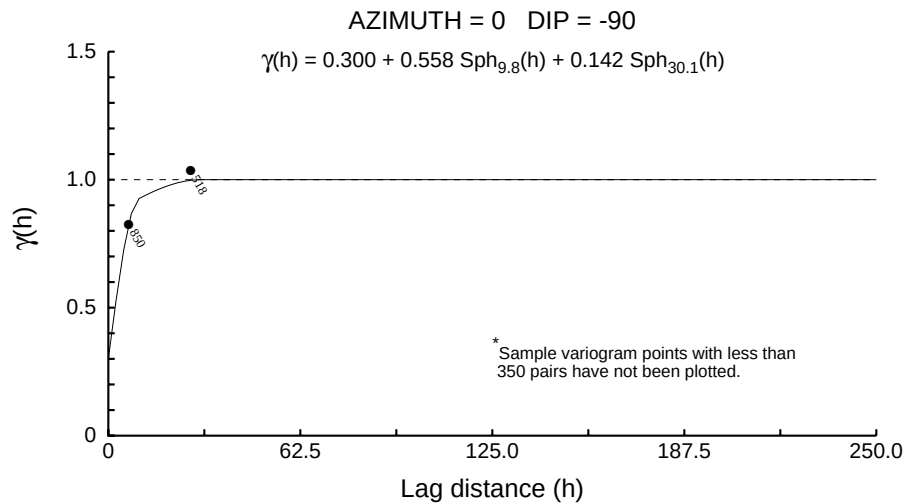
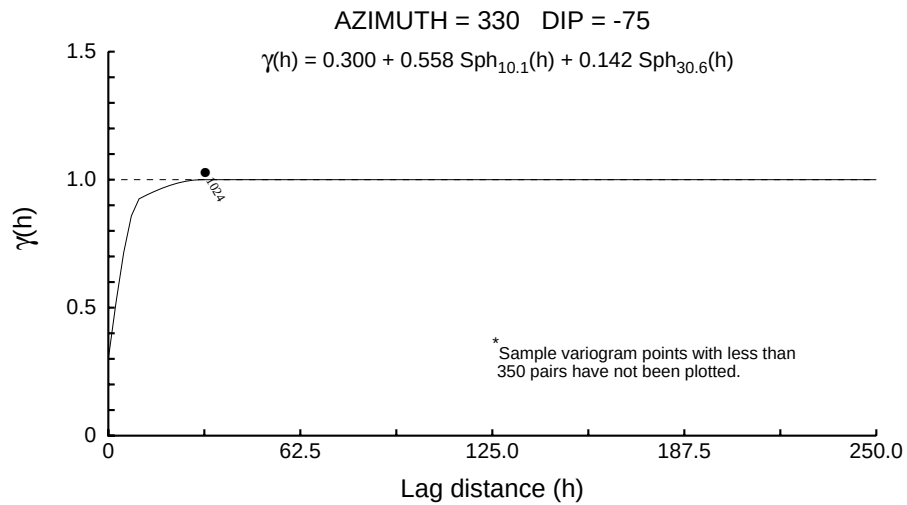
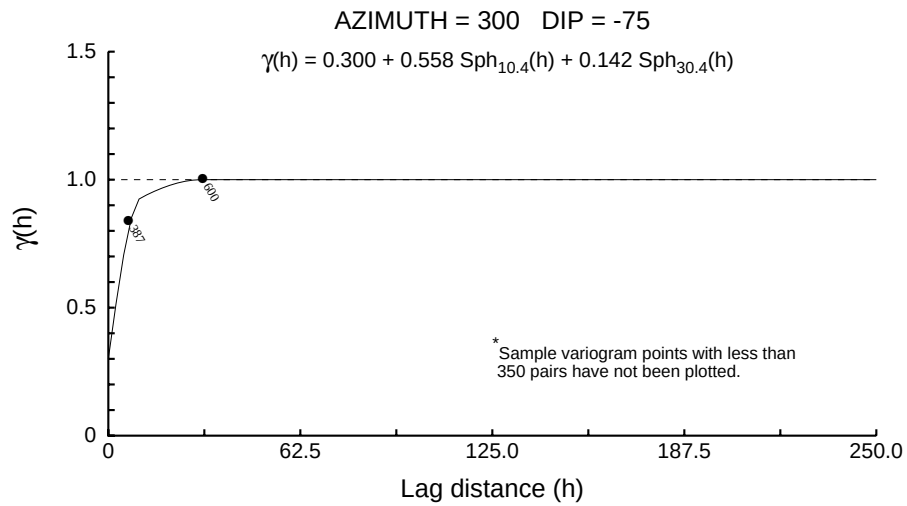
Main Cu 2.5m Comps



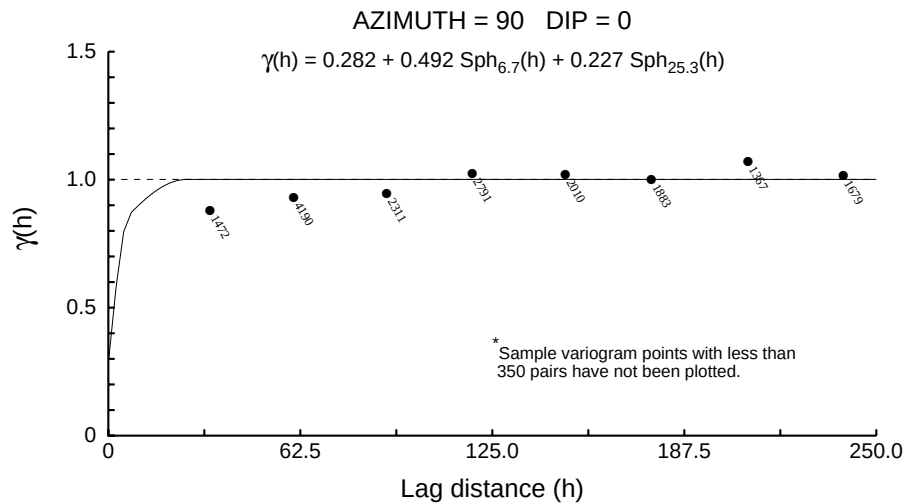
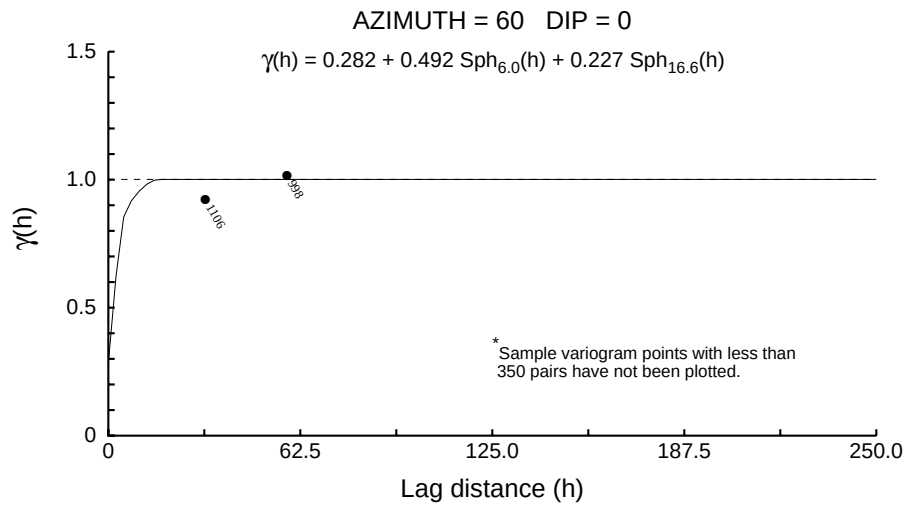
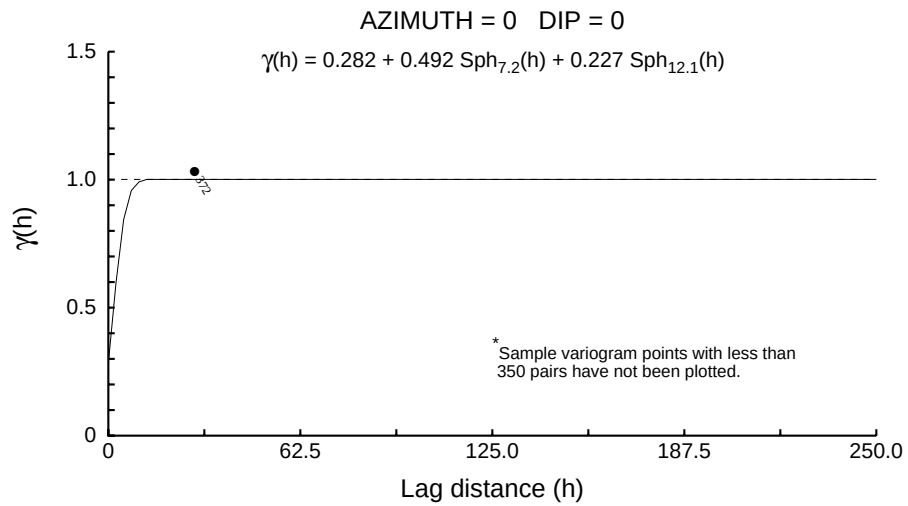
Main Cu 2.5m Comps



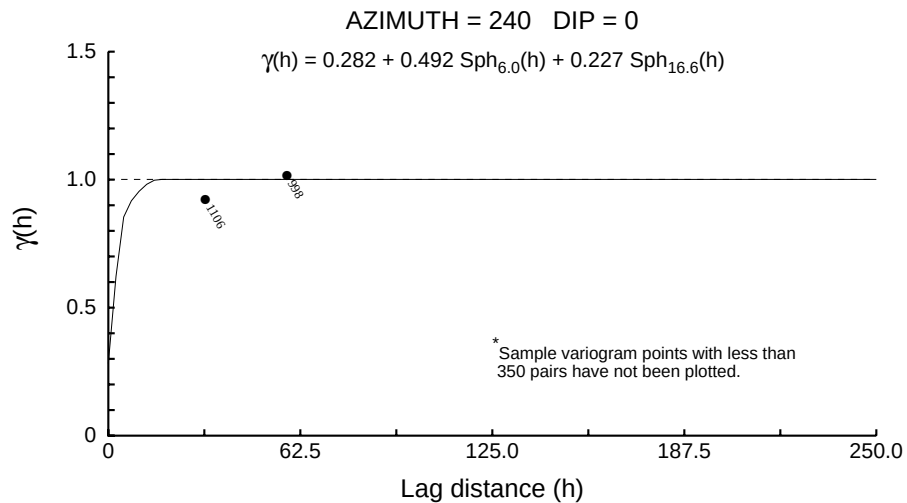
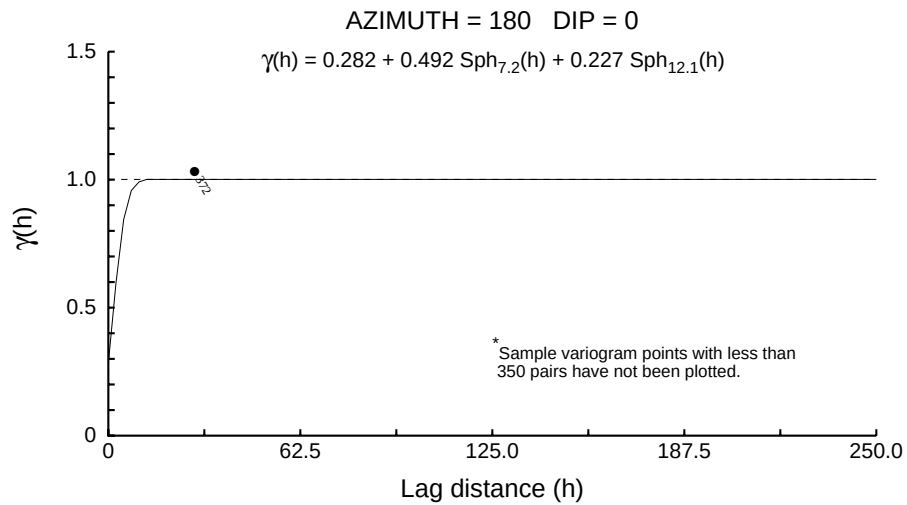
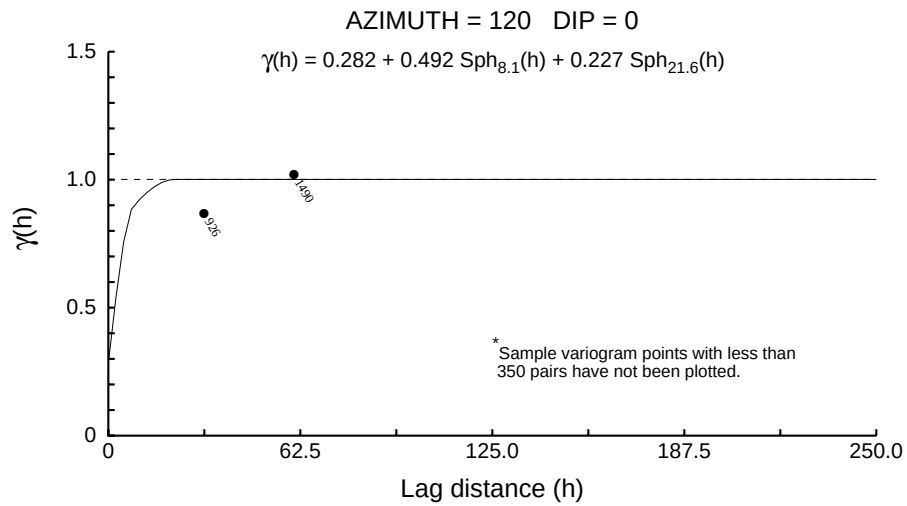
Main Cu 2.5m Comps



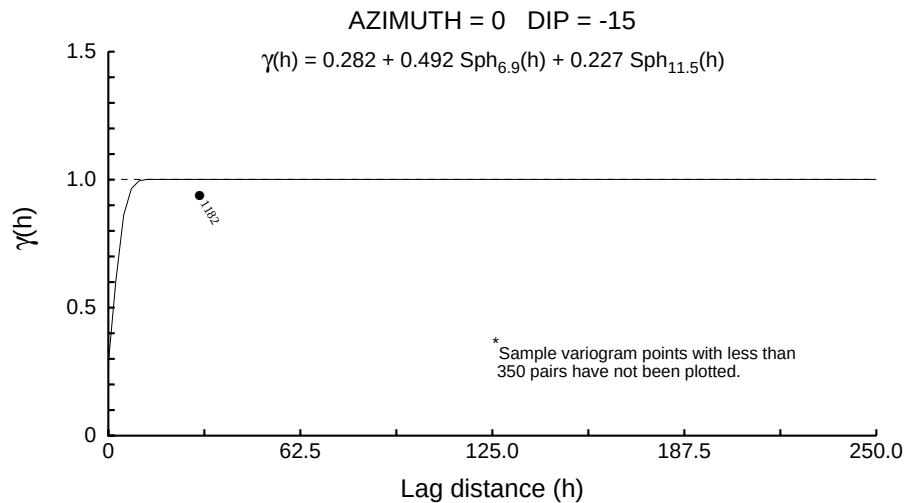
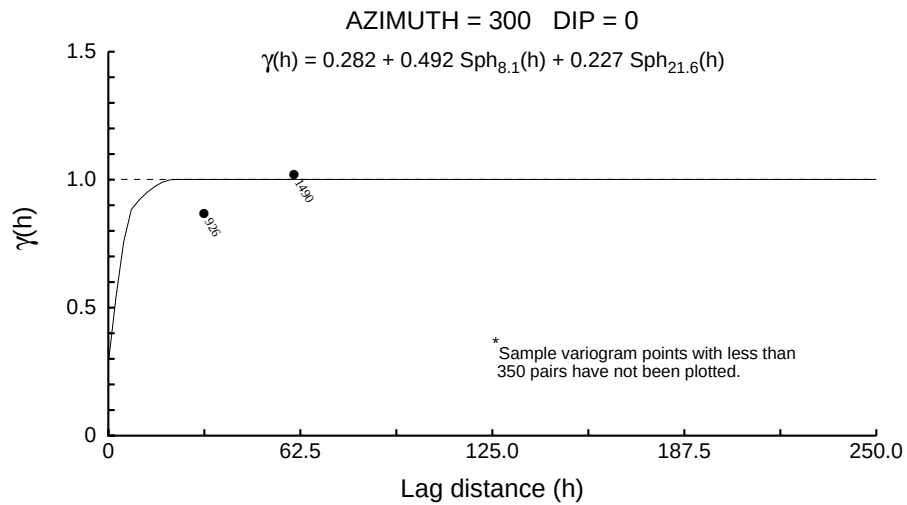
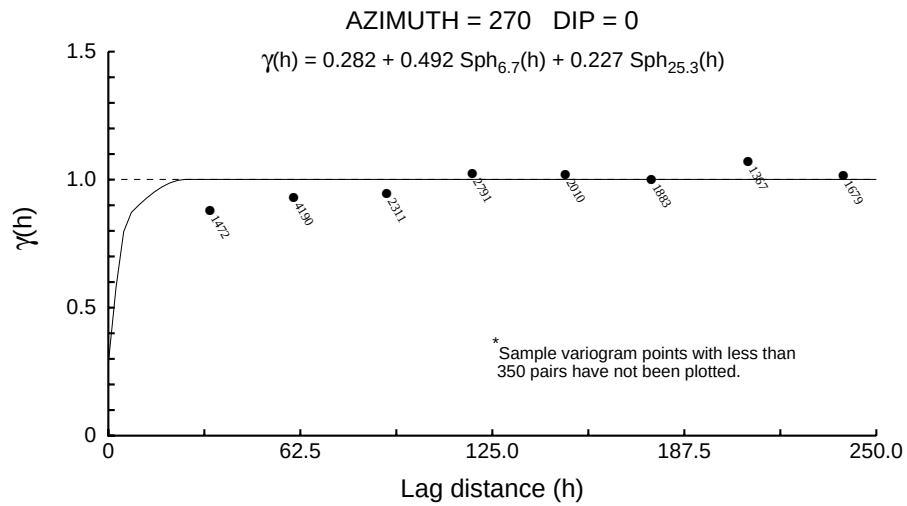
Main Zn 2.5m Comps



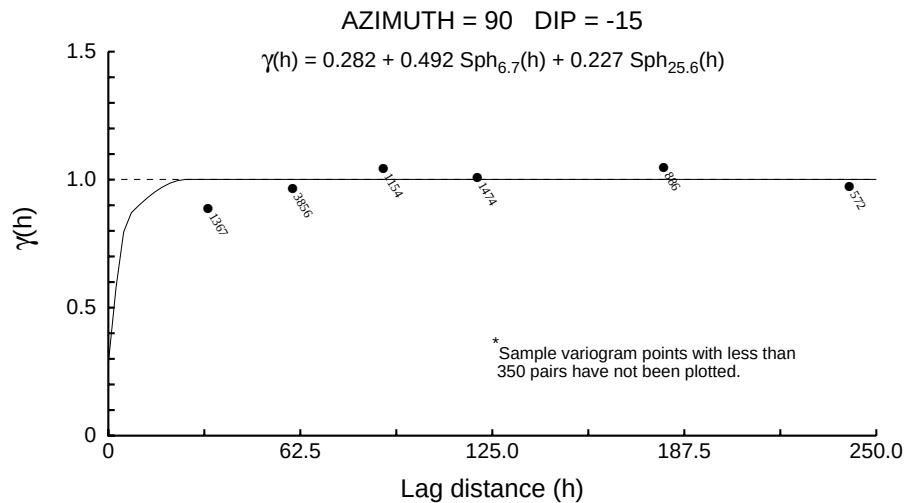
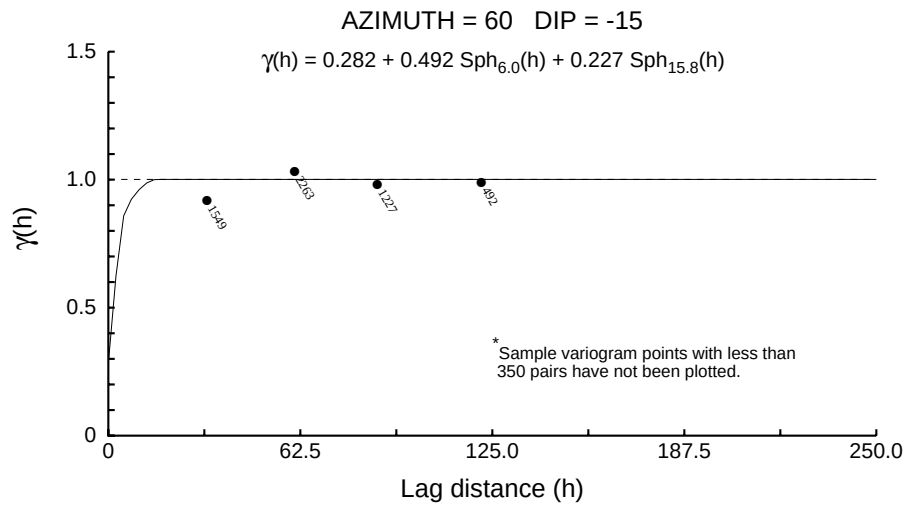
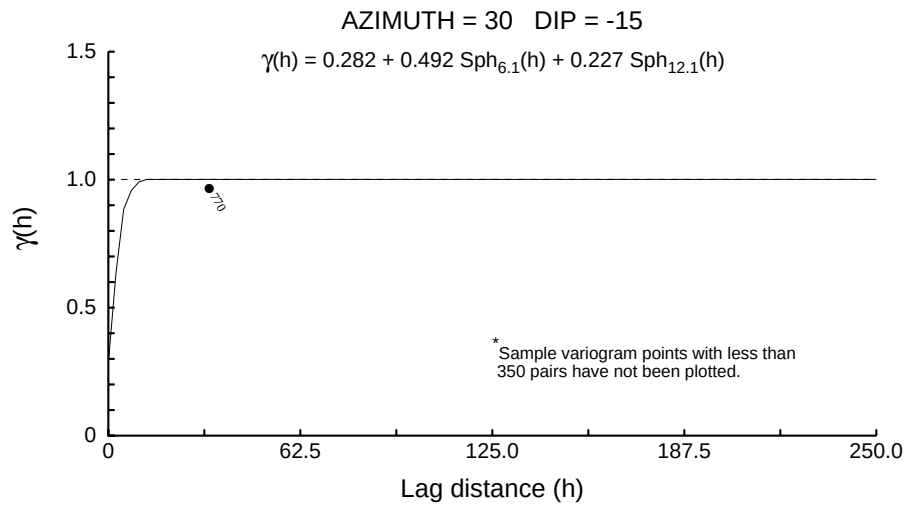
Main Zn 2.5m Comps



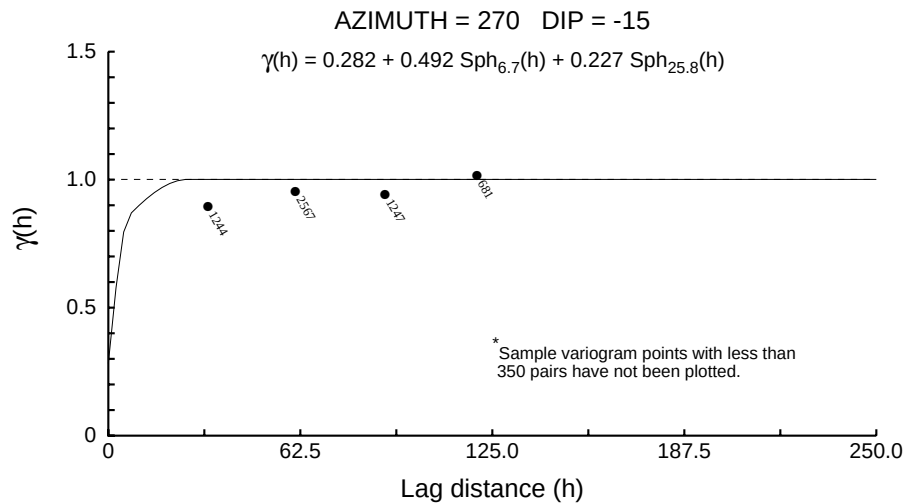
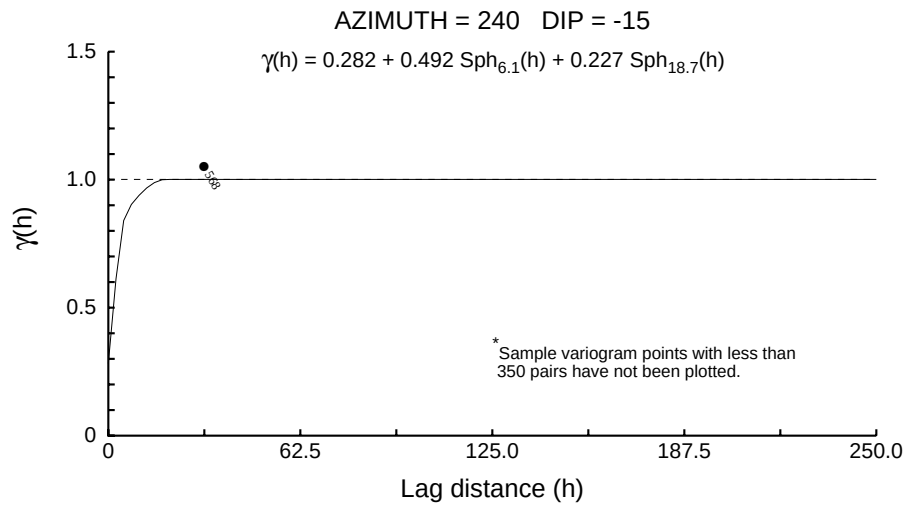
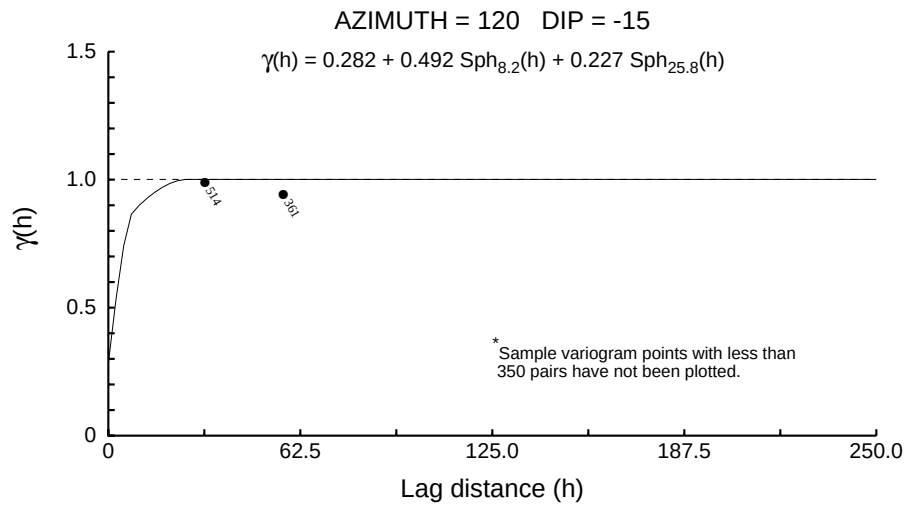
Main Zn 2.5m Comps



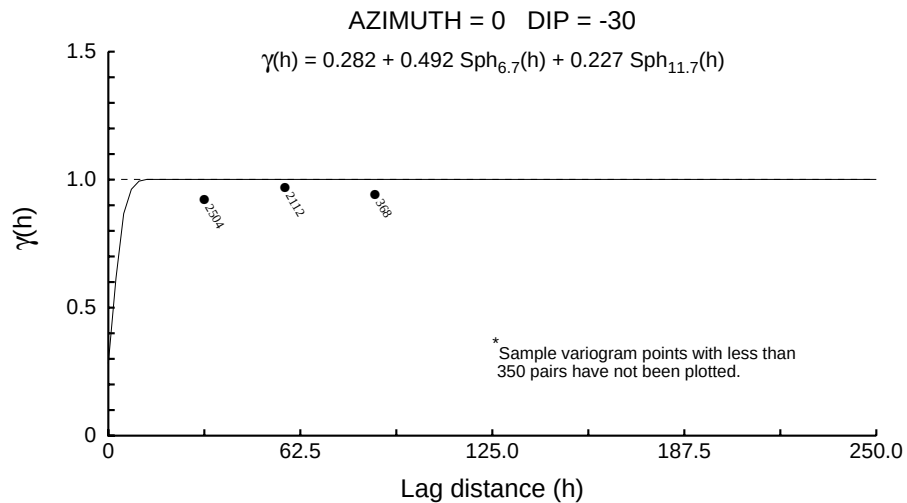
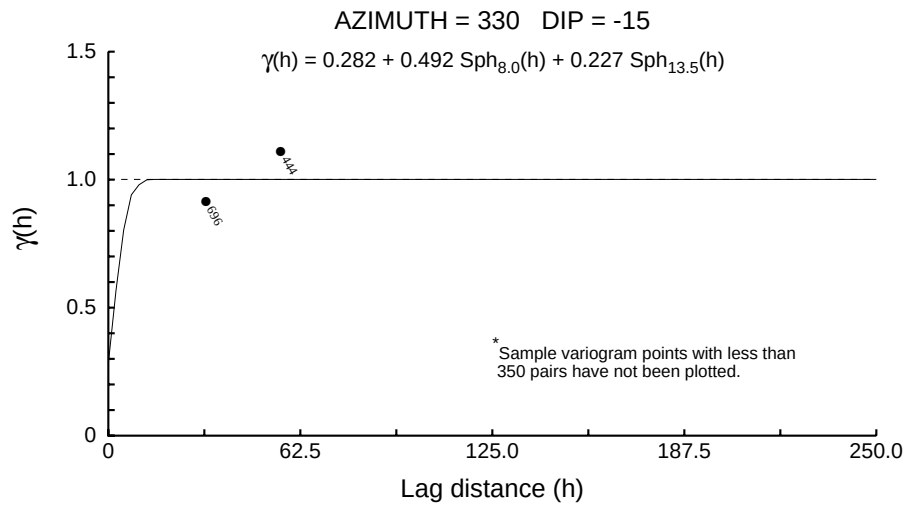
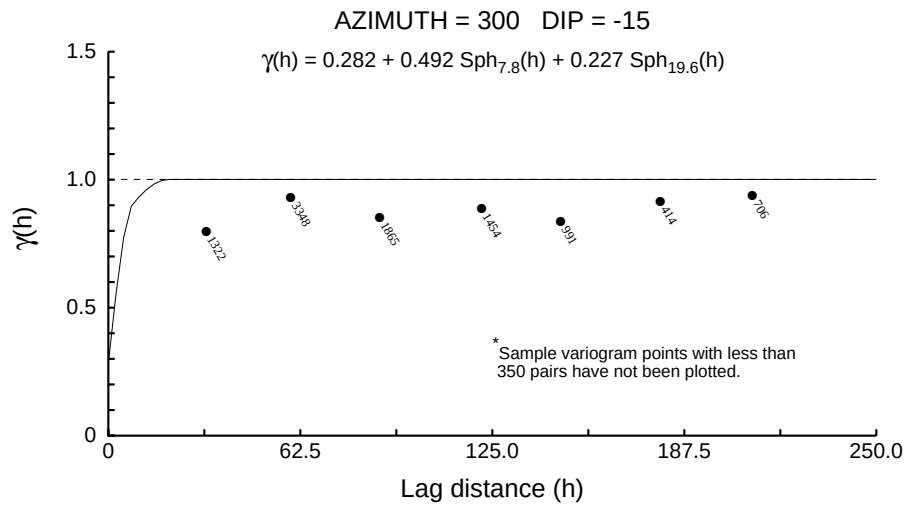
Main Zn 2.5m Comps



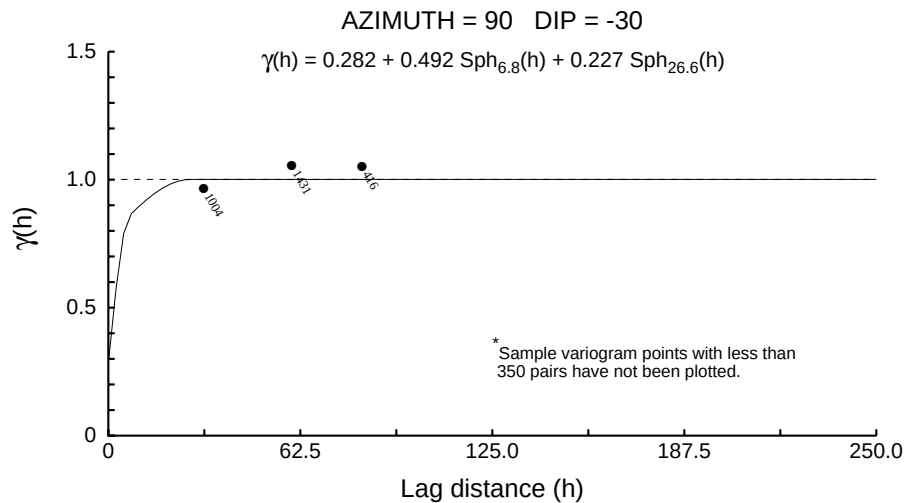
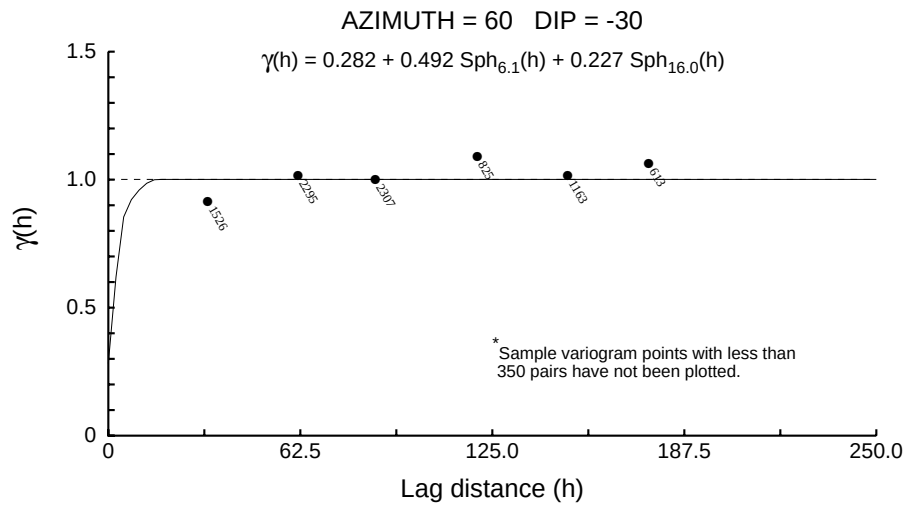
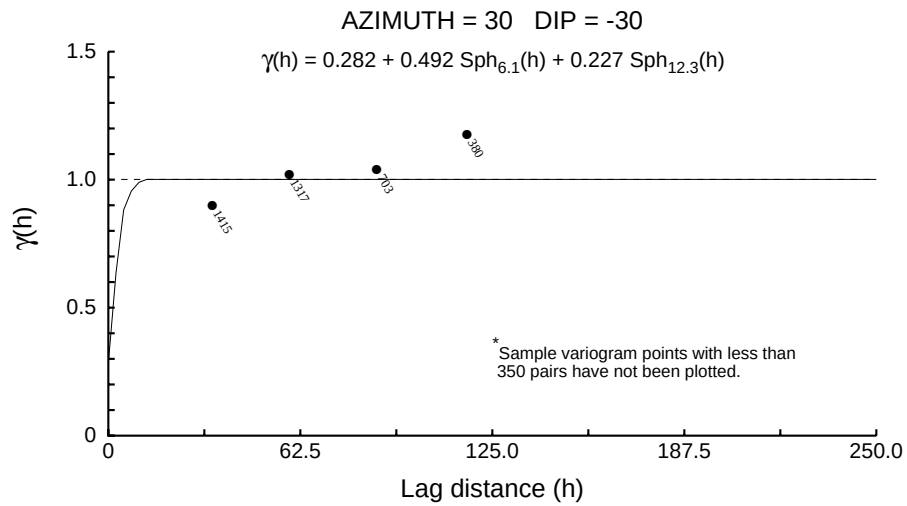
Main Zn 2.5m Comps



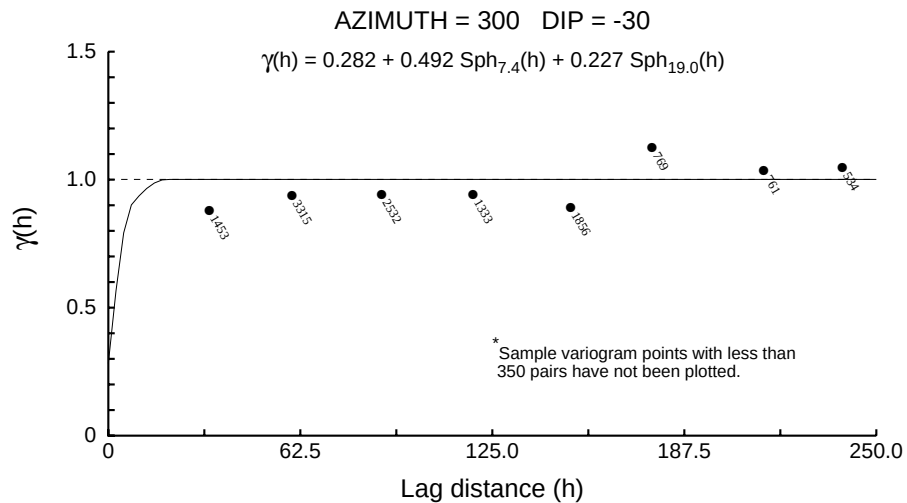
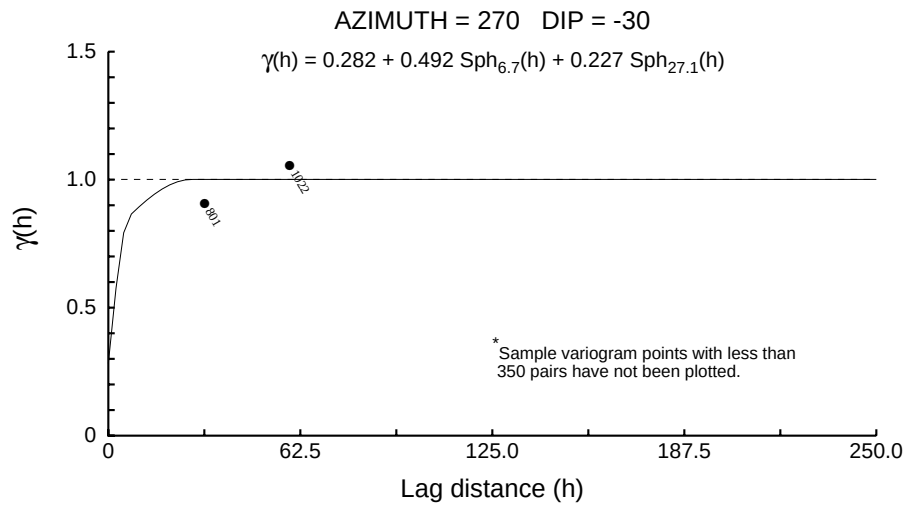
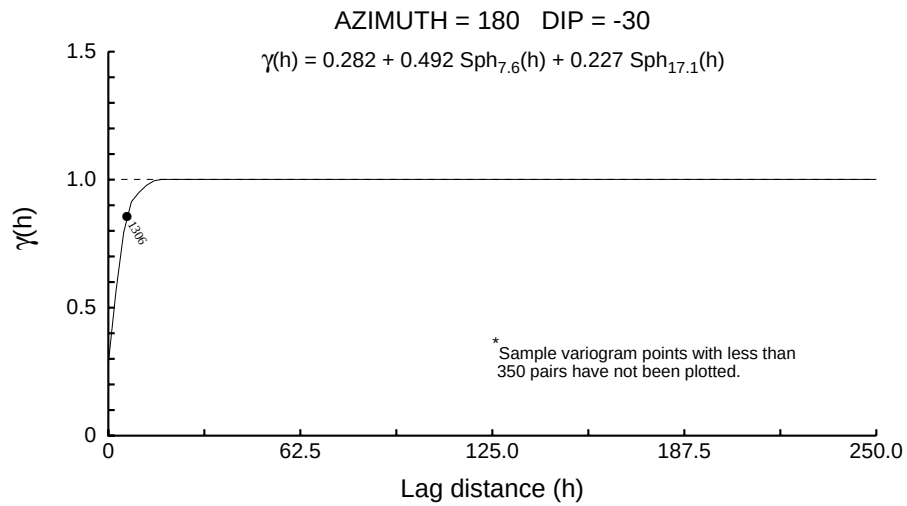
Main Zn 2.5m Comps



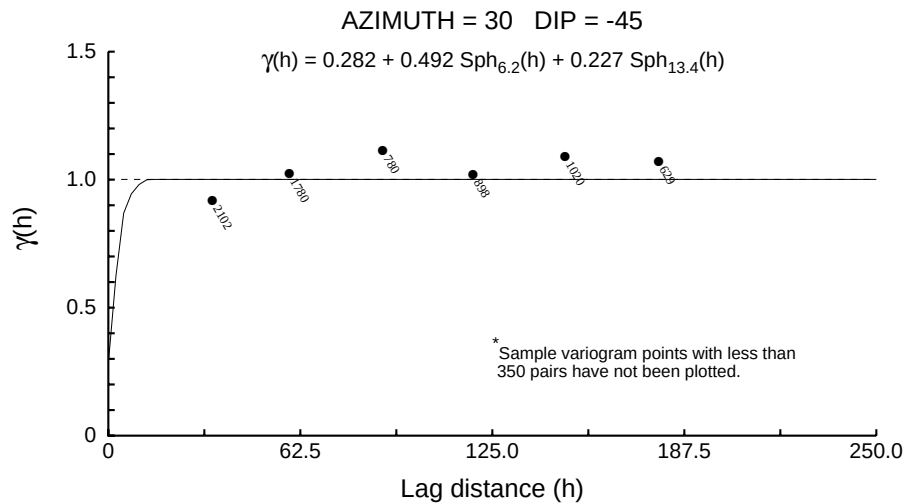
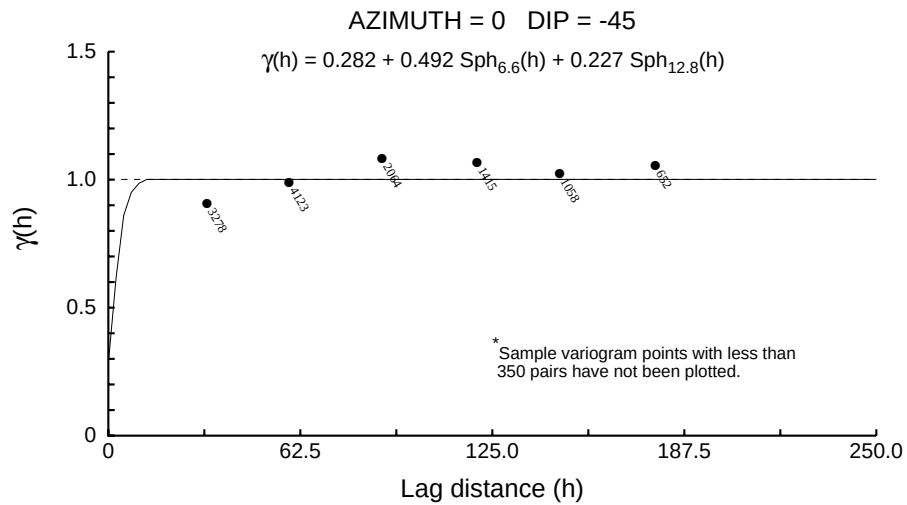
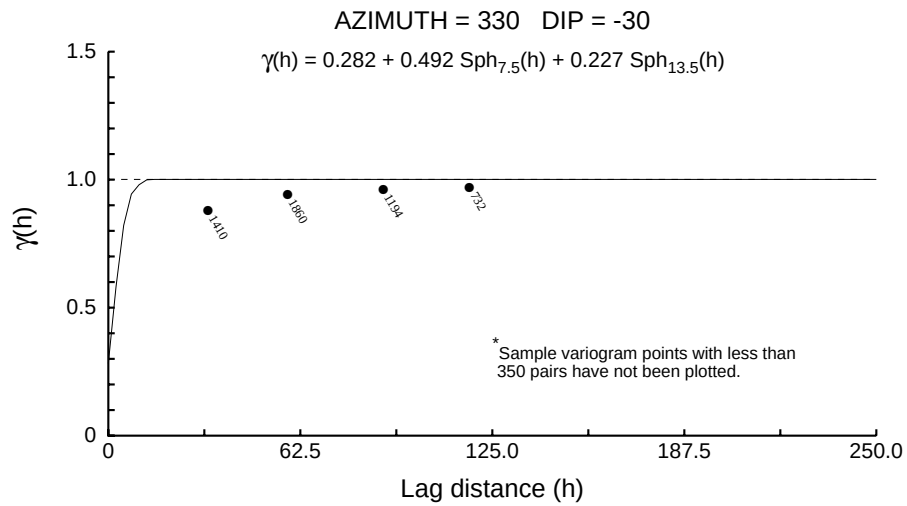
Main Zn 2.5m Comps



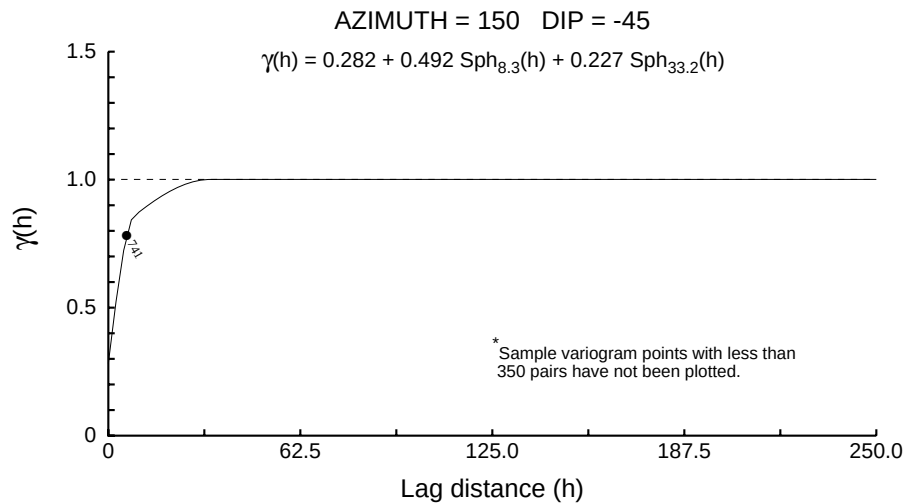
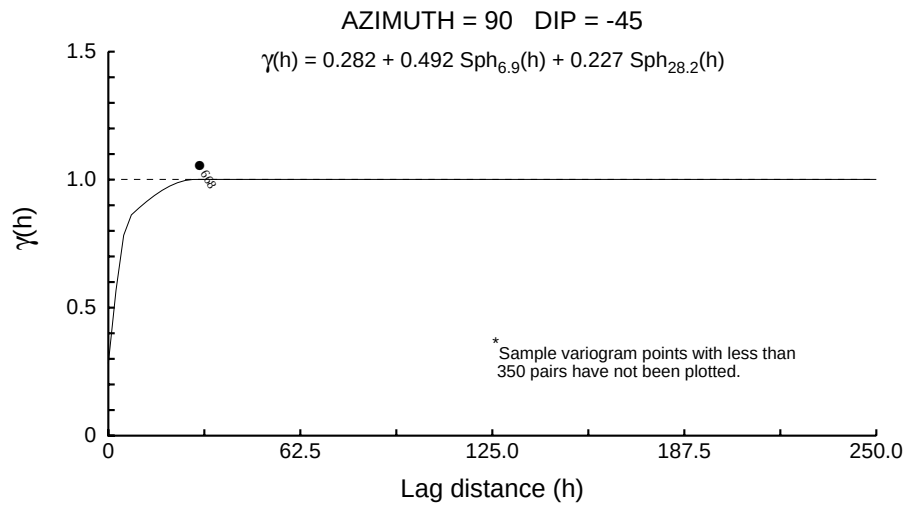
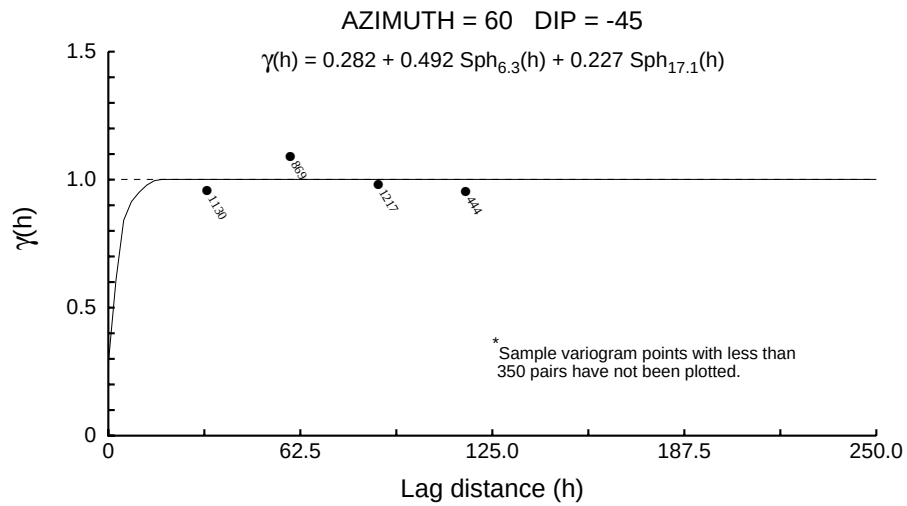
Main Zn 2.5m Comps



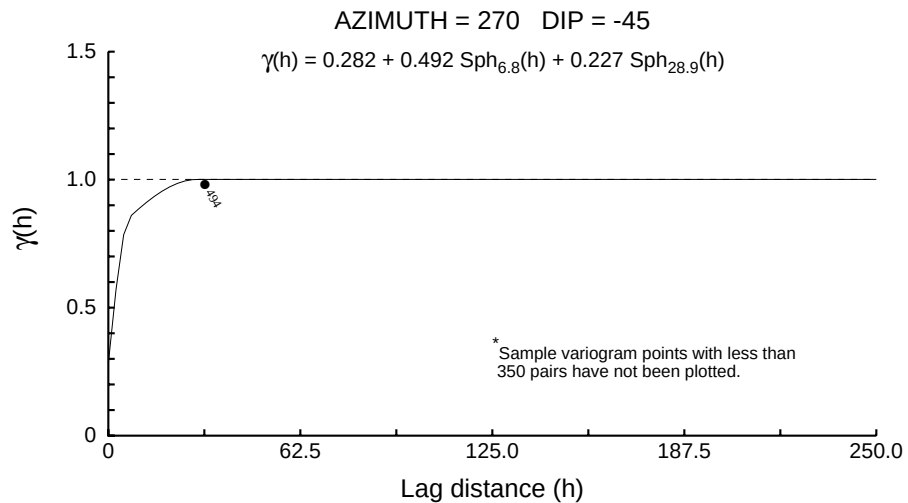
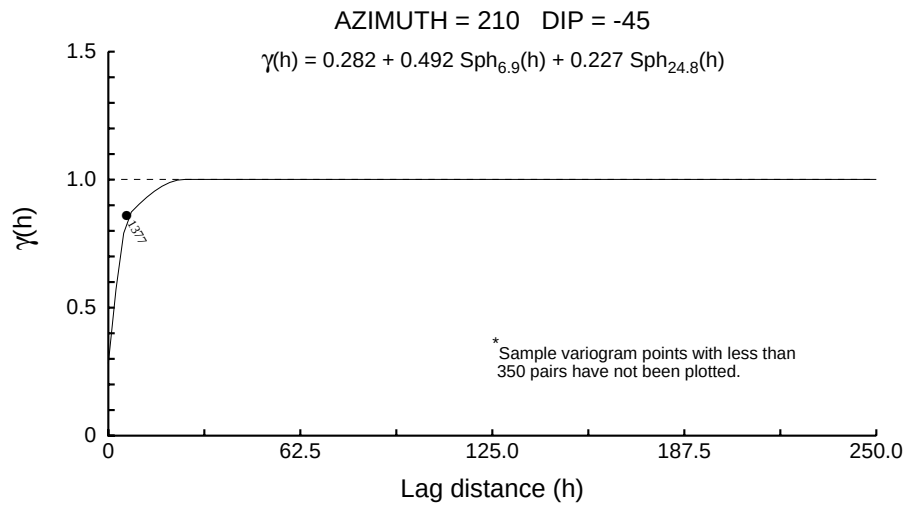
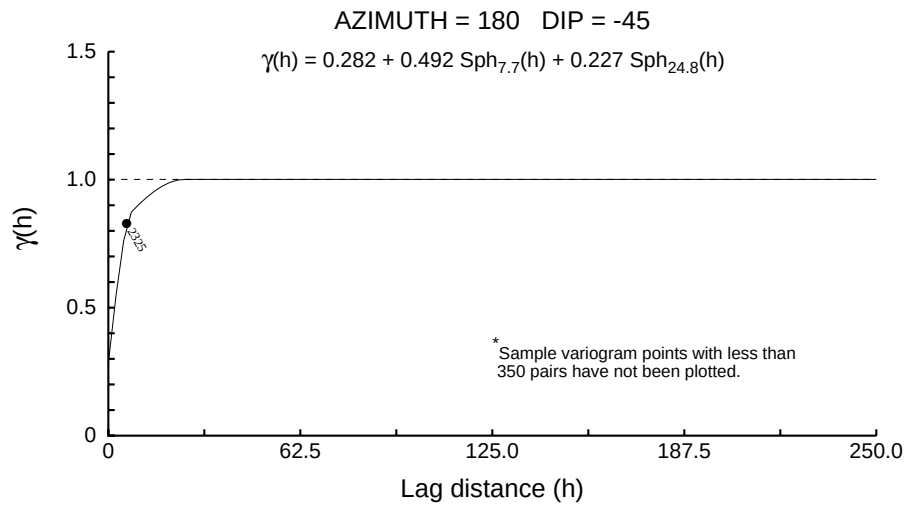
Main Zn 2.5m Comps



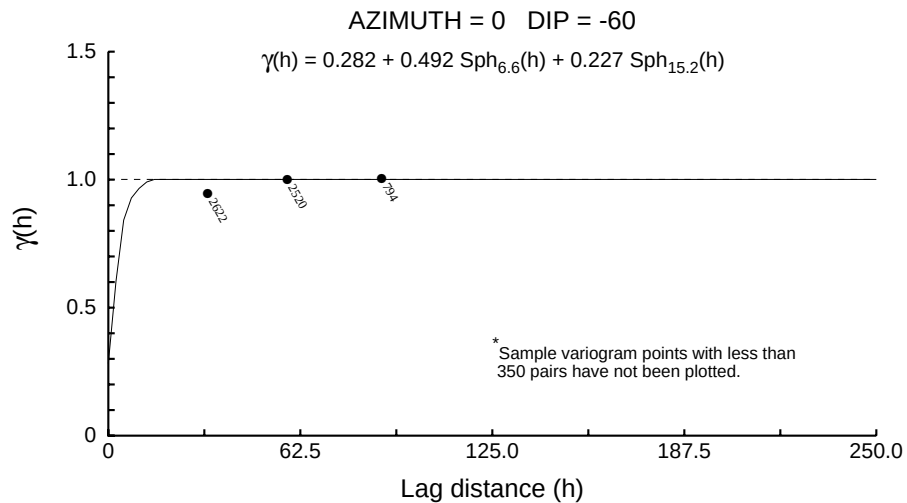
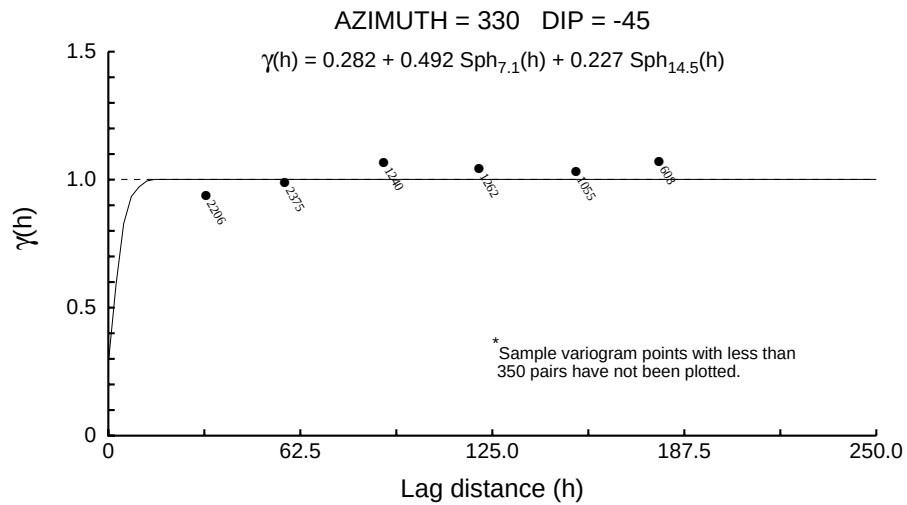
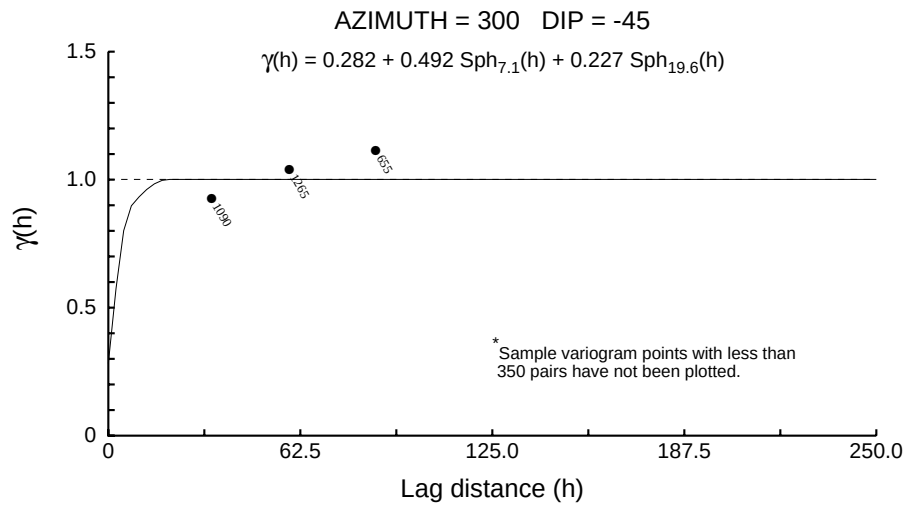
Main Zn 2.5m Comps



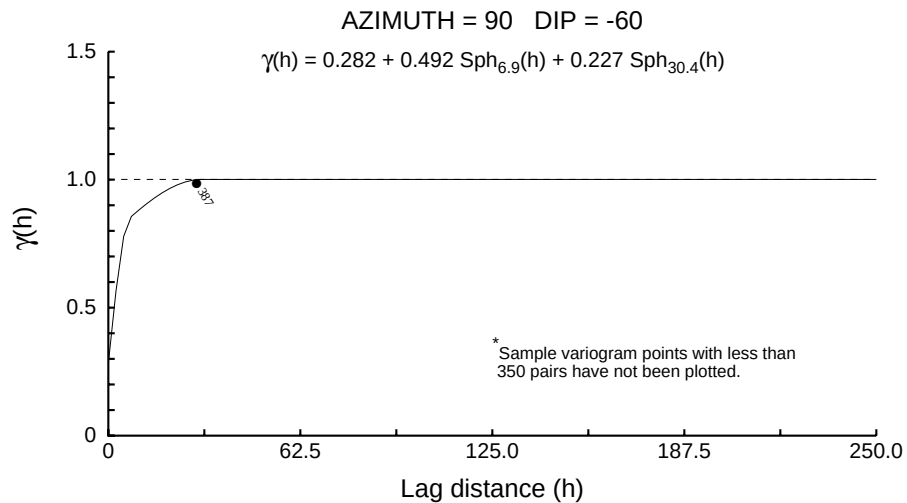
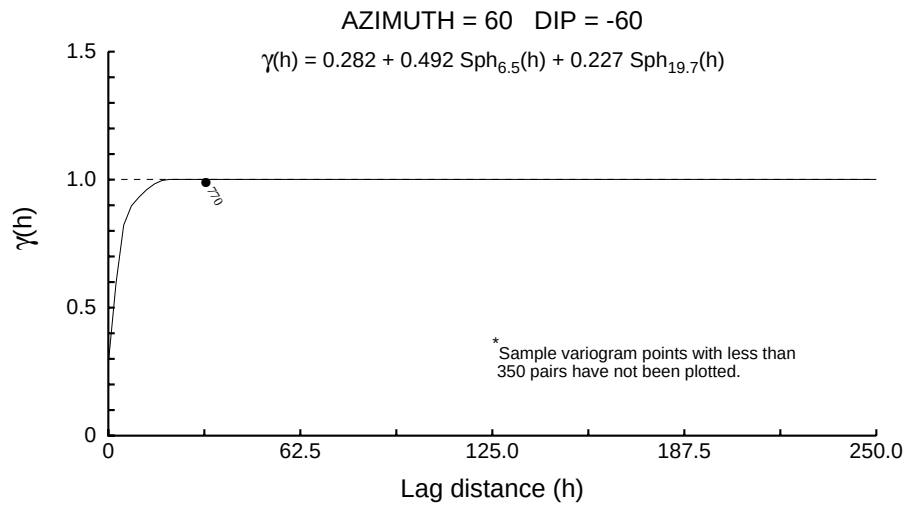
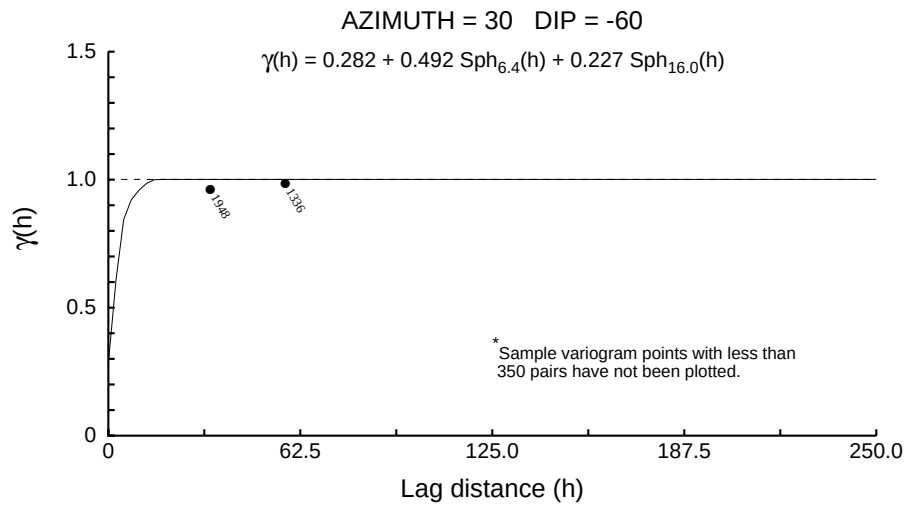
Main Zn 2.5m Comps



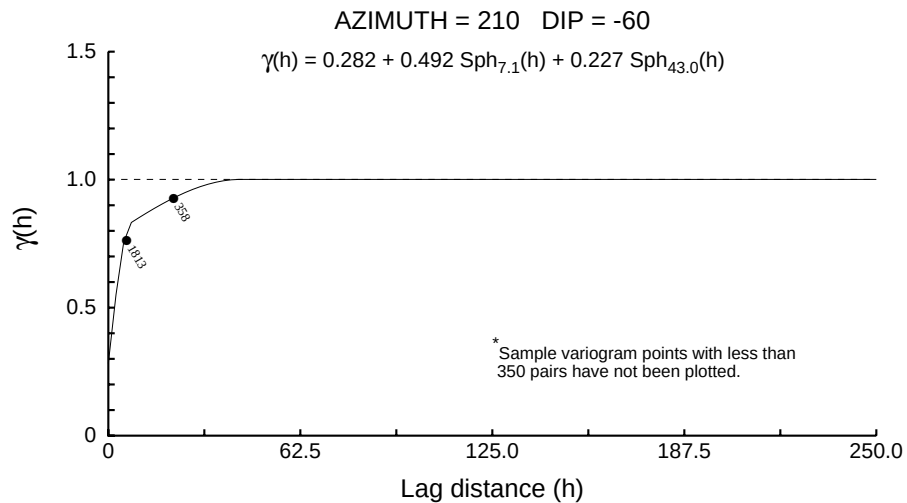
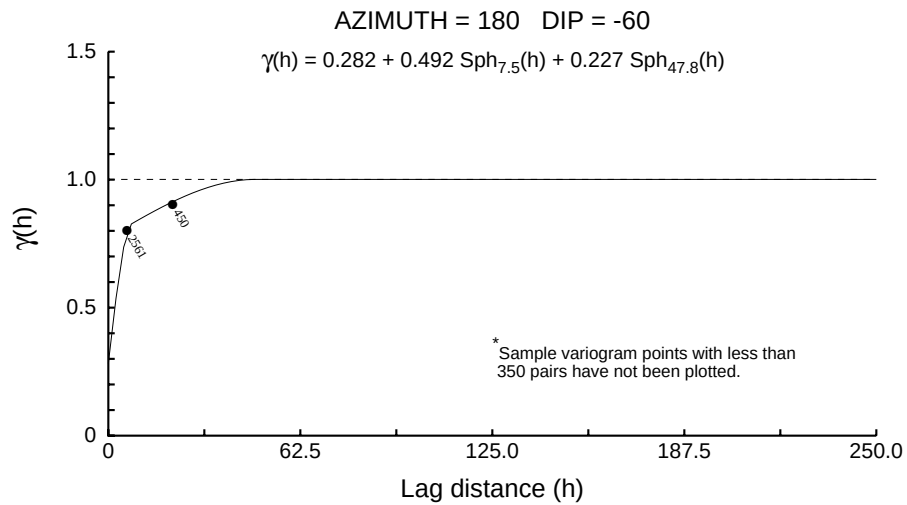
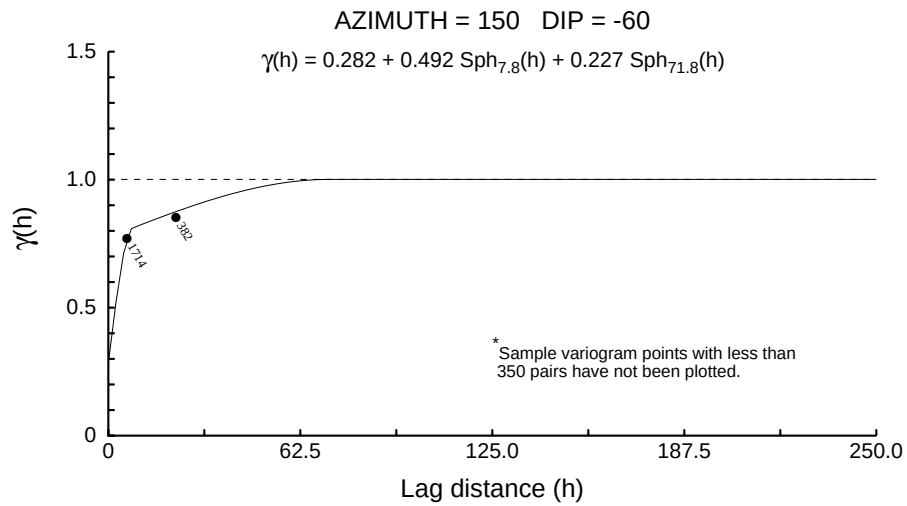
Main Zn 2.5m Comps



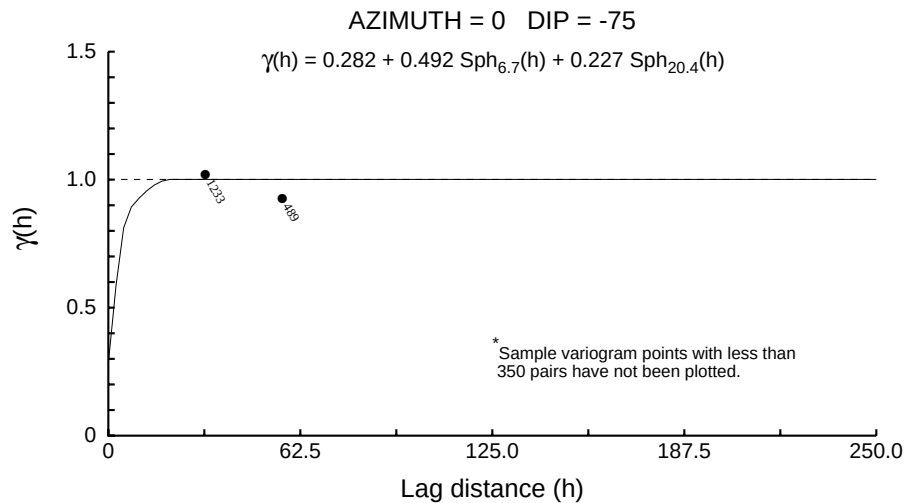
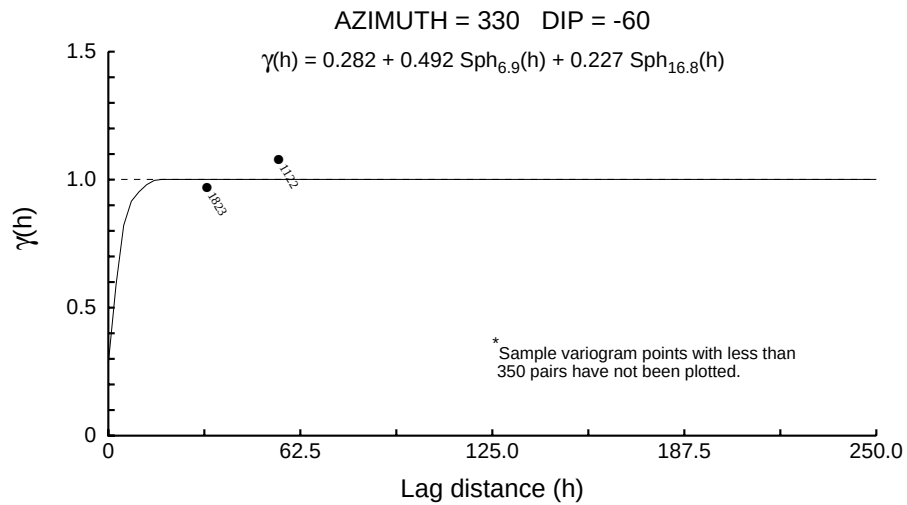
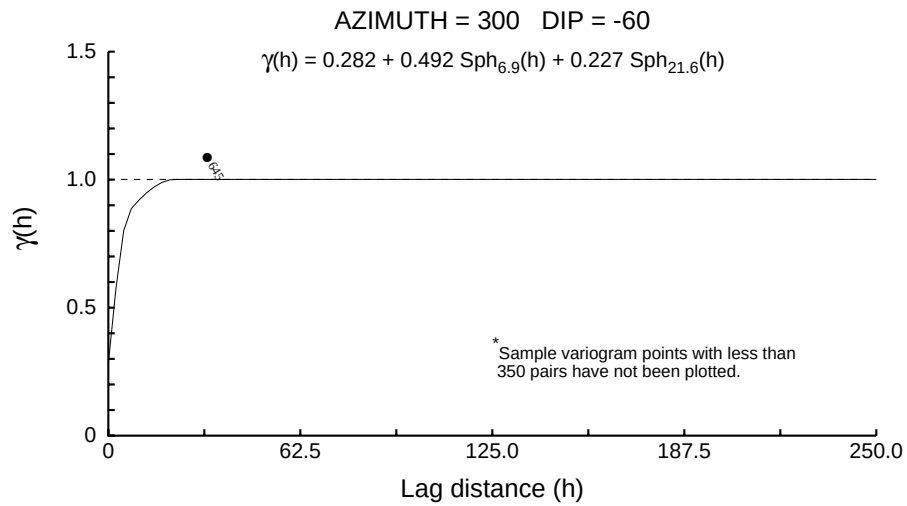
Main Zn 2.5m Comps



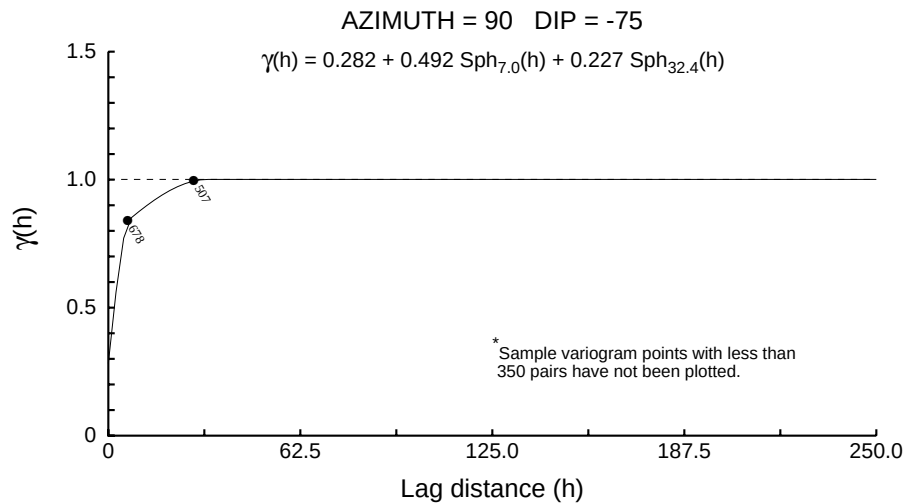
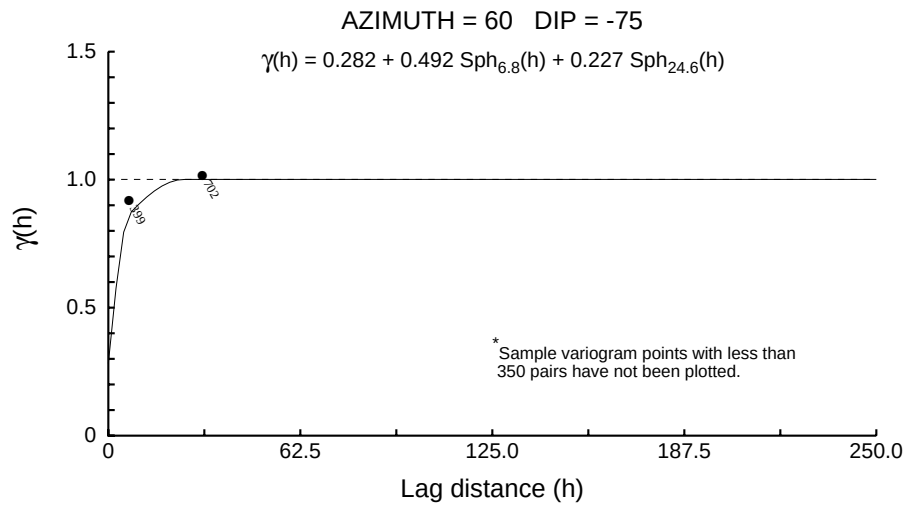
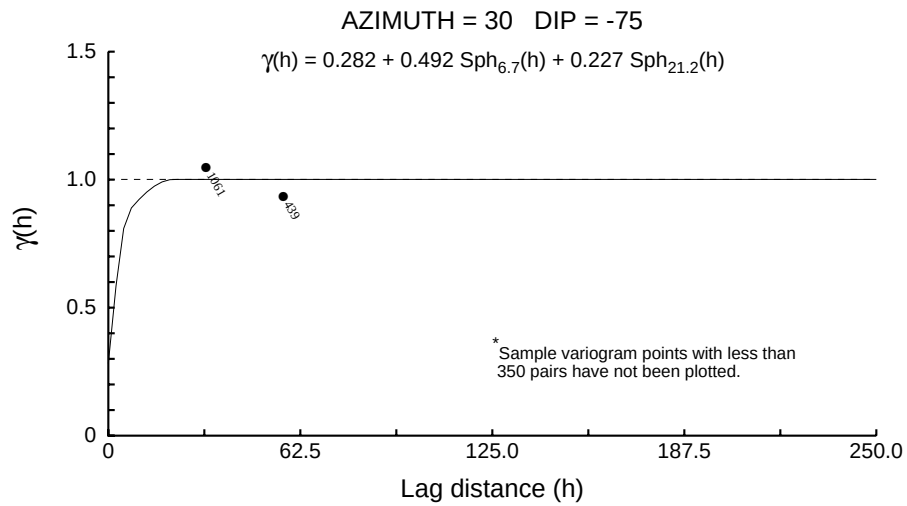
Main Zn 2.5m Comps



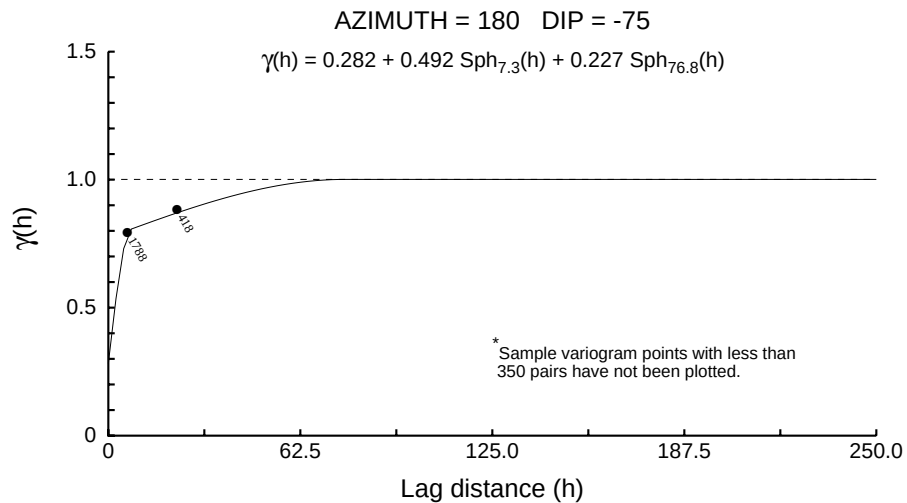
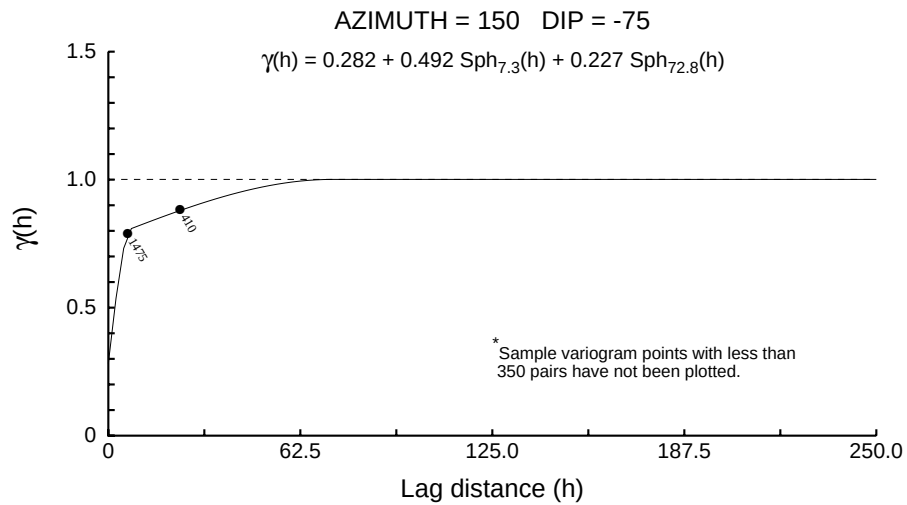
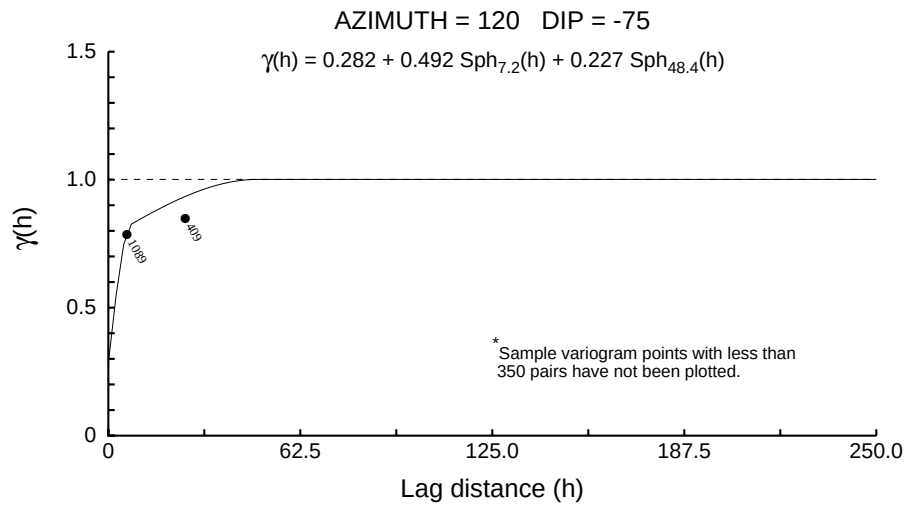
Main Zn 2.5m Comps



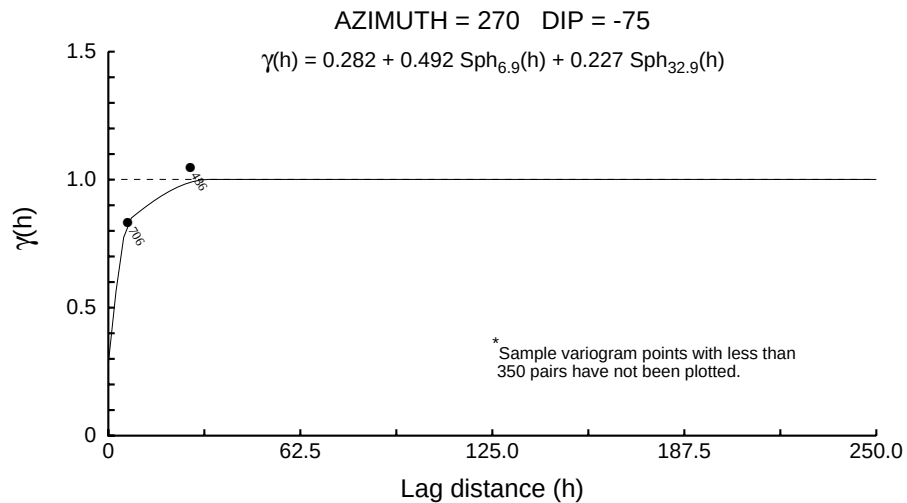
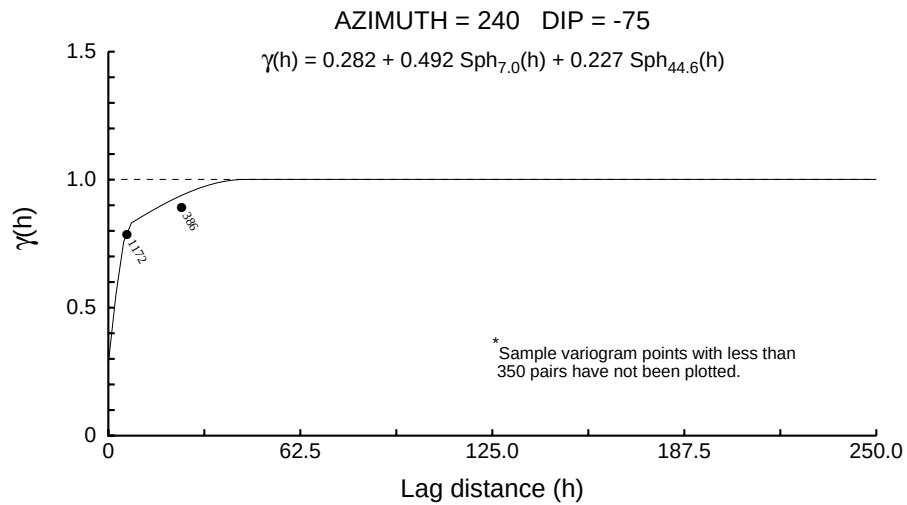
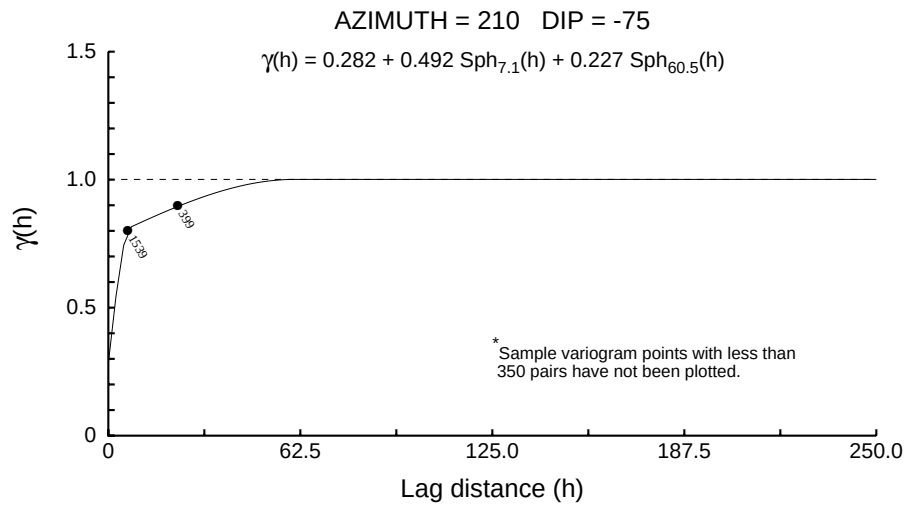
Main Zn 2.5m Comps



Main Zn 2.5m Comps



Main Zn 2.5m Comps



Main Zn 2.5m Comps

