Report to:

Tintina Resources Inc.

Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana

Document No. 1391880100-REP-R0004-01

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TINTINA RESOURCES INC.

UPDATED TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE BLACK BUTTE COPPER PROJECT, MONTANA

EFFECTIVE DATE: JULY 12, 2013

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GLOSSARY

UNITS OF MEASURE

above mean sea level .... amsl
acre .... ac
ampere .... A
annum (year) .... a
billion .... B
billion tonnes .... Bt
billion years ago .... Ga
British thermal unit .... BTU
centimetre .... cm
cubic centimetre .... cm^3
cubic feet per minute .... cfm
cubic feet per second .... ft^3/s
cubic foot .... ft^3
cubic inch .... in^3
cubic metre .... m^3
cubic yard .... yd^3

Coefficients of Variation	CVs
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBa
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	US$
dollar (Canadian)	Cdn$
dry metric ton	dmt
foot	ft
gallon	gal
gallons per minute (US)	gpm
gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	"
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV

kilovolt-ampere	kVA
kilovolts	kV
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne (metric ton)	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
metric ton (tonne)	t
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
ounce	oz
pascal	Pa
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"

second (time)	s
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
thousand tonnes	kt
three dimensional	3D
three dimensional model	3DM
tonne (1,000 kg)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt
year (annum)	a

Abbreviations and Acronyms

acid rock drainage	ARD
Advancement of Cost Engineering	AACE
ammonium nitrate/fuel oil	ANFO
area of interest	AOI
atomic absorption spectrophotometer	AAS
BHP Billiton Limited	BHP
Black Butte Copper Project	the Project
Black Butte Fault	BBF
Black Butte Property	the Property
Bond ball mill work index	BWI
Canadian Institute of Mining	CIM
Canyon Diablo Troilite standard	CDT
capital cost estimate	CAPEX
closed-circuit television	CCTV
cobalt	Co
Coefficient of Variation	CV
Cominco American Inc.	CAI
copper	Cu
cumulative net cash flow	CNCF

Department of Environmental Quality	DEQ
distributed control system	DCS
effective grinding length	EGL
Engineering Procurement and Construction Management	EPCM
environmental assessment	EA
Environmental Impact Statement	EIS
Environmental Protection Agency	EPA
Fish, Wildlife & Parks	FWP
fresh air raise	FAR
general and administrative	G&A
Global Acid Rock Drainage	GARD
global positioning system	GPS
grade times thickness	Grd-Thk
Great Falls Tectonic Zone	GFTZ
Hangingwall Dolomite	HWD
Hard Rock Impact Act	HRIA
heating, ventilation, and air conditioning	HVAC
high-density polyethylene	HDPE
incremental percentage	Inc %
inductively coupled plasma	ICP
inductively coupled plasma-atomic emission spectroscopy	ICP-AES
inductively coupled plasma-mass spectrometry	ICP-MS
Inspectorate Exploration & Mining Services Ltd	Inspectorate
internal rate of return	IRR
International Organization for Standardization	ISO
inverse distance	ID
Knight Piésold Ltd.	Knight Piésold
land application discharge	LAD
life-of-mine	LOM
load-haul-dump	LHD
London Metal Exchange	LME
Lower Sulphide Zone	LSZ
Lower Zone	LZ
Major Facility Siting Act	MFSA
maximum contaminant level	MCL
Metal Mines Reclamation Act	MMRA
methyl isobutyl carbinol	MIBC
Middle Sulphide Zone	MSZ
Middle Zone	MZ
Mine Safety and Health Administration	MSHA
Montana Environmental Policy Act	MEPA

Montana Groundwater Pollution Control System permit	MGWPCS permit
Montana Pollutant Discharge Elimination System	MPDES
National Instrument 43-101	NI 43-101
nearest neighbour	NN
net cash flow	NCF
net present value	NPV
net smelter return	NSR
non-potentially acid generating	non-PAG
North American Datum	NAD
operating cost estimate	OPEX
operator interface stations	OIS
pebble crushing circuit	SABC
potentially acid generating	PAG
preliminary economic assessment	PEA
qualified person	QP
quality assurance/quality control	QA/QC
Quantitative Evaluation of Materials by Scanning Electron Microscopy	QEMSCAN
reduction to pole	RTP
Resource Modeling Incorporated	RMI
return air raise	RAR
return air	RA
rock quality designation	RQD
run-of-mine	ROM
semi-autogenous grinding	SAG
silver	Sg
Small Miner Exclusion Statement	SMES
sodium isopropyl xanthate	SIPX
standard industrial classification codes	SIC
standard reference material	SRM
Stantec Consulting Ltd.	Stantec
State Historic Preservation Office	SHPO
sulphur	S
sulphur isotope composition	$\delta^{34}S$
tailings management facility	TMF
Tintina Resources Inc.	Tintina
Universal Transverse Mercator	UTM
Upper Sulphide Zone	USZ
Upper Zone	UZ
US Forest Services	USFS
US Geological Survey	USGS
Utah International Inc.	UII

variable frequency drive .... VFD
Volcano Valley Fault .... VVF
World Geodetic System .... WGS

1.0 SUMMARY

1.1 INTRODUCTION

Tintina Resources Inc. (Tintina) retained Tetra Tech to prepare an updated National Instrument 43-101 (NI 43-101) Preliminary Economic Assessment (PEA) for the Black Butte Copper Project (the Project) located in Meagher County, Montana, US. This report is an update to the previous PEA, dated August 30, 2012, and incorporates the results of recent diamond drilling on the Johnny Lee deposit as well as a revised mining sequence based on the updated resource estimate.

The Black Butte Property (the Property) is situated on private ranch lands, approximately 17 miles north of the town of White Sulphur Springs (Figure 1.1). This area contains all currently known deposits, including the high-grade copper with cobalt-silver Johnny Lee deposit. This report is specific to the Johnny Lee deposit and the Lowry deposit has not been included as part of the overall analysis.

The Project will involve an underground mine operation that will mine and process up to 3,300 t/d of mineralized material. The current resource base considered for this updated PEA consists of 11.57 Mt of Measured and Indicated mineral resources and 1.46 Mt of Inferred mineral resources support an 11-year life-of-mine (LOM). These resources are from the Upper and Lower Johnny Lee zone.

Table 1.1 outlines general information for the Project.

All dollar figures presented in this updated PEA are stated in US dollars, unless otherwise specified. The long term consensus metal prices with an effective date of April 26, 2013, and an exchange rate of Cdn$1.00 to US$1.00 have been used, unless otherwise specified.

Figure 1.1 General Location Map

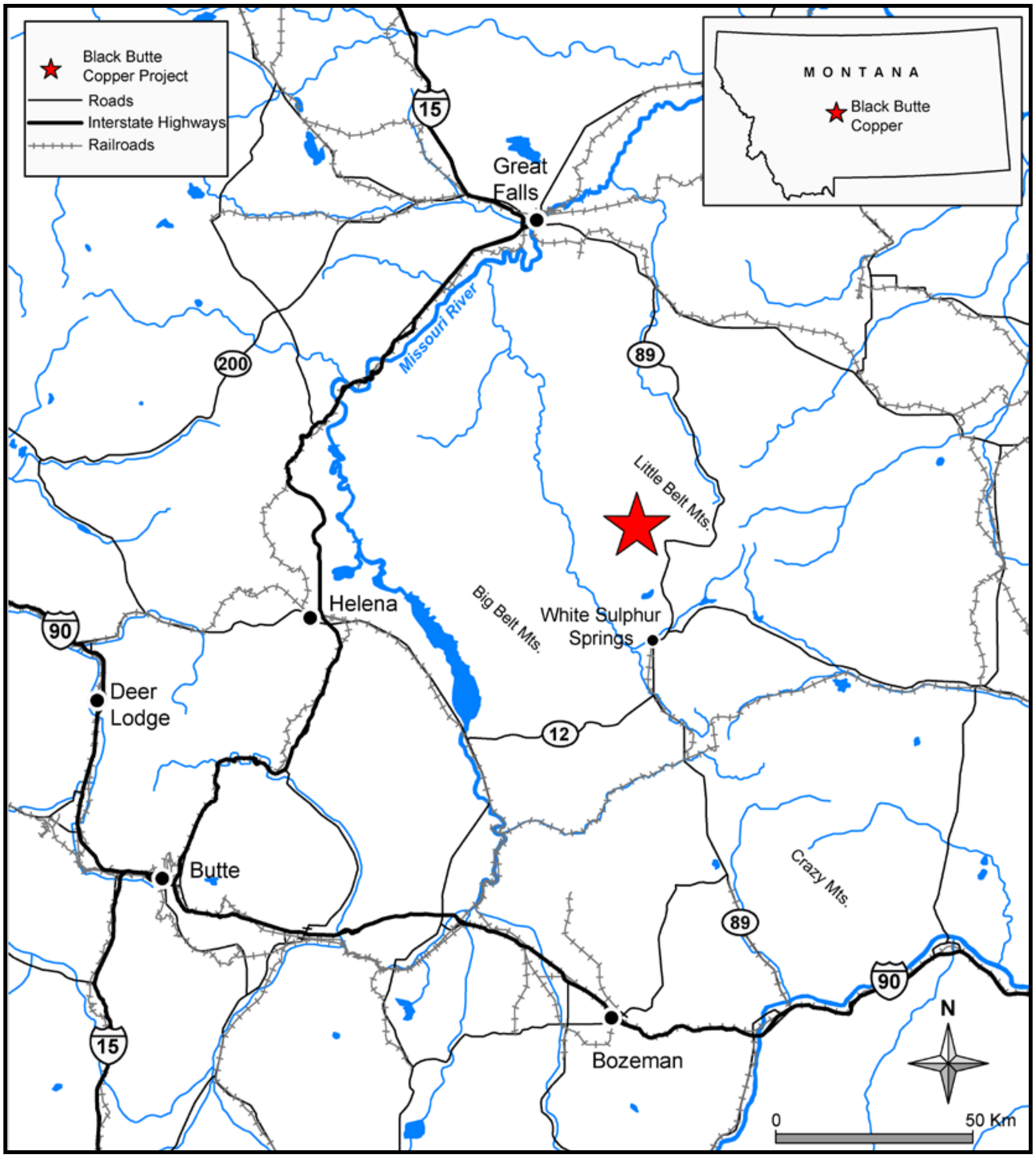


Table 1.1 General Project Information

Description	Unit	Amount
Estimated Mineral Resources (Measured and Indicated)	Mt	11.57
Estimated Mineral Resources (Inferred)	Mt	1.46
LOM	years	11
Milling Rate (Nominal)	t/d	3,300
Total Project Initial Capital Cost	US$ million	217.8
Average Overall Operating Cost	US$/t milled	66.48
Copper Price	US$/lb	3.05
Pre-tax Net Present Value (NPV) at 8% Discount Rate	US$ million	218
Pre-tax Internal Rate of Return (IRR)	%	30.5
Pre-tax Payback Period	Years	3.6
Post-tax NPV at 8% Discount Rate	US$ million	110
Post-tax IRR	%	20.2
Post-tax Payback Period	Years	4.7

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

Tetra Tech prepared this updated PEA for Tintina, which incorporates work by the following independent consultants:

- Resource Modeling Incorporated (RMI): Property Description and Location, Accessibility, Climate, Local Resources, Infrastructure, Physiography, History, Geological Setting and Mineralization, Deposit Types, Exploration, Drilling, Sample Preparation, Analysis and Security, Data Verification, Mineral Resource Estimates
- Arthur H. Winckers & Associates Inc.: Mineral Processing and Metallurgical Testing
- AMEC E&C Services, Inc. (AMEC): Mineral Reserve Estimates, Mining Methods
- Tetra Tech: Recovery Methods, Roads, Buildings, Power, Capital Cost Estimate, Economic Analysis
- Knight Piésold Ltd. (Knight Piésold): Tailings Management, Water Management, Waste Dumps, Instrumentation.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Property is located in Meagher County, Montana, US, approximately 17 miles north of the town of White Sulphur Springs. The Property is accessed by 1.5 miles of well-maintained county graveled road which branches off from US Highway 89, an all-weather state-maintained highway. US Highway 89 connects the Property area with White Sulphur Springs, Montana, which has a population of approximately 984 residents. Elevations in the resource area range from 5,600 to 6,000 ft above sea level and the topography is gently rolling hills and valleys. Timber cover consists of primarily Douglas fir on north-facing slopes, grass and mountain sagebrush cover on valley floors and draws, and open to partly timbered ridge tops. Timber covers approximately 10% of the resource area.

The Property consists of three tracts of fee-simple lands totalling 7,684.28 ac and 239 unpatented lode mining claims on US Forest Service lands totalling approximately 4,541 ac. For Tract 1 (named for purposes of this description only), the Bar Z Ranch controls 100% surface and three members of the Hanson family share equal interest in 100% of the mineral interest. For Tract 2, Mrs. Rose I. Holmstrom, a local rancher, controls 100% of both surface and mineral interests. For Tract 3, Steve Buckingham, a local rancher, controls 100% of the surface rights and shares equal interest in the mineral interests with two siblings. The 56 mining claims (named the SB claims) were staked by Tintina Alaska Exploration Inc., a wholly owned subsidiary of Tintina Resources, Inc., in November 2010 for a total of approximately 1,064 ac. An additional 183 BSP claims were staked by Tintina Alaska Exploration Inc. in the spring of 2011 and total approximately 3,477 ac.

Tract 1, for which Tintina has one surface lease and three mineral leases, consists of 2,594.28 ac of surface 100% owned by Bar Z Ranch, and includes the surface over 2,555 mineral acres covered by three mining leases with the Hanson family, each of whom own one third of the mineral interest. The additional 39.28 ac covered by the surface lease consists of two patented mining claims, the Copper Hill (Mineral Survey #10311) and Rio Tinto (Mineral Survey #10304). The mineral rights for these two claims are owned by another party with whom Tintina has no agreement. The surface lease requires lease payments of $50,000 on signing (May 2, 2010) and on each of the first four anniversary dates. Payments on the fifth anniversary date and each anniversary date thereafter are $75,000.

Each of the three mining leases for Tract 1 requires advance minimum royalty payments of $16,150 on signing (May 2, 2010) and on the first and second anniversaries, $32,300 on the third anniversary, $48,450 on the fourth anniversary $64,600 on each anniversary thereafter through the term of the lease.

The term is for 30 years and is renewable for subsequent periods of 10 years each. The combined mineral interest has a net smelter return (NSR) of 5%, with an option to buy this down to a 2% NSR for $5,000,000, thereby reducing each mineral lessors royalty to 0.6666% NSR in return for a payment of $1,666,666. Exercising the buy down option eliminates further advance minimum royalty payments.

Tract 2 consists of 2,120 ac and is subject to a single mining lease with Mrs. Rose I. Holmstrom for 100% surface and 100% mineral interest. The agreement requires advance minimum royalty payments of $40,195 on signing (May 2, 2010) and on the first and second anniversaries, $80,411 on the third anniversary, $120,607 on the fourth anniversary, and $160,802 on each anniversary thereafter through the term of the lease. The term is for 30 years and is renewable for subsequent periods of 10 years each. The agreement has a 5% NSR with an option to buy this down to a 2% NSR for $5,000,000.

Tract 3 consists of 2,970 ac and is subject to a single mining lease with Mr. Steve Buckingham, 100% surface owner and one-third mineral owner, and his two siblings, Kathy Johnston and Marilyn Bodell, each one-third mineral owners. The agreement requires advance minimum royalty payments of $5,000 on signing, $15,000 on or before six months after signing, $20,000 on or before the first and second anniversaries, $25,000 on or before the third through fifth anniversaries, $30,000 on or before the sixth through eighth anniversaries, $35,000 on or before the ninth through eleventh anniversaries, $40,000 on or before the twelfth through fourteenth anniversaries, and $50,000 per annum through the remainder of the lease term or until commercial production. The agreement has a term of 30 years and a 5% NSR, which can be bought down to 2% NSR for a payment of $5,000,000.

The mineral owners warrant that there are no prior or underlying agreements encumbering the above described surface and mineral interest. All agreements stipulate underground mining only.

1.3 HISTORY

According to Weed (1899) local hay ranchers, located claims on copper-stained quartzite at the Virginia Mine and by 1894 had a 70 ft shaft with a 30 ft drift, but not much copper mineralization was exposed. Presumably, the workings were too shallow to penetrate below surface oxidation and encounter any sulphide. This location is approximately 500 m west of the present resource area. In 1910, John Lee sunk a shallow shaft nearby on similar material (pers. comm., Hanson family).

During the first half of the 20th century, interest focused on extensive gossans developed on Iron Butte area and between Butte Creek and Sheep Creek (Goodspeed 1945; Roby 1950). This work resulted in surveying and patenting of a number of patented claims, both inside and adjacent to the Tintina lease area. Work focused on the iron potential, and while prospectors dug a few prospect pits and drove a few small adits, no workings penetrated the redox boundary into sulphide-bearing rock.

Cominco American Inc. (CAI) carried out the first modern exploration work on the property. Exxon Minerals obtained a lease on a portion of the Property in 1981 and joint ventured it to CAI in late 1984. CAI joint ventured the entire Property to Utah International Inc. (UII) in 1985, and UII was subsequently taken over by BHP Billiton Limited (BHP). UII/BHP operated the joint venture through early 1988 and earned a 50% interest in the Project, at which time operatorship reverted back to CAI. Within the next two years, CAI purchased BHP’s interest in the property and regained 100% control with no retained royalties or back-in rights. CAI dropped the leases in the mid-1990s and retained no royalites or rights. Approximately 66 diamond core holes were completed in the two lease areas by CAI and the CAI/BHP joint venture.

1.4 GEOLOGICAL SETTING AND MINERALIZATION

1.4.1 GEOLOGICAL SETTING

The copper-cobalt deposits of Black Butte occur in middle Proterozoic sediments of the Belt Supergroup which are extensively exposed in an eastward protrusion of the Rocky Mountain chain called the Helena salient in central Montana (Zieg and Leitch 1993). During formation of the Belt Basin, a deep water middle Proterozoic calcareous shale facies (Newland Formation) deposited in an embayment, known as the Helena embayment, which extended in trough-like fashion east into the craton through central Montana (Godlewski and Zieg 1984). The northern boundary of the deeper water portion of the Helena embayment lay along the southern flank of the Little Belt Mountains north of White Sulphur Springs, Montana. During the Cretaceous Laramide orogeny, renewed faulting along the ancestral northern margin of the Helena embayment formed the Volcano Valley thrust fault (Winston 1986). The bedded massive sulphides of the Black Butte are concentrated along the northern margin of the Helena embayment along the Volcano Valley Fault (VVF) zone.

The Newland Shale hosts the Black Butte massive sulphides, and consists of a lower shale-dominated part which measures approximately 760 m thick and an upper carbonate-dominated part which measures approximately 350 m thick. The shale was deposited as microturbidites in a sub-wavebase depositional setting. Debris flow conglomerates punctuate the section along the northern margin of the embayment. Though in places the lower Newland shale shows ubiquitous bedded pyrite throughout, more typically sulphides are concentrated in several discrete stratigraphic horizons of greater lateral extent.

1.4.2 MINERALIZATION

Sulphides in the Johnny Lee deposit are concentrated in two copper rich zones, the Upper Zone (UZ) and the Lower Zone (LZ). In the Johnny Lee UZ, copper is concentrated in lenses up to 28 m thick within the lower part of a bedded pyrite zone, which can reach over 100 m thick. The Johnny Lee UZ is capped by barite-rich sulphides. The Johnny Lee UZ consists of a lens of fine grained bedded sulphides and contains up to three chalcopyrite-bearing horizons. Pyrite occurs as laminations and beds of very fine grained pyrite and marcasite with disseminated and lenticular masses of chalcopyrite and minor bornite, tennantite, cobaltite, and siegenite. Gangue material includes barite, dolomite, calcite, and fine-grained quartz. Microscopic textures and species of sulphide minerals, primarily from copper-enriched horizons, have been well described by Himes and Petersen (1990) and by Graham et al. (2012).

The Johnny Lee LZ reaches over 17 m thick and consists of bedded and replacement pyrite with high concentrations of replacement chalcopyrite in silicified shale and conglomerate. Overall, sulphide grain sizes are much coarser, vein like and replacement textures dominate the fabric of the mineralized zones, and the zone is strongly silicified. Pyrite in the Johnny Lee LZ includes the fine grained varieties with marcasite, but coarser grained secondary pyrite overprinting earlier dolomite alteration dominates much of the zone. Chalcopyrite has replaced secondary pyrite and dolomite. Some occurrences of siegenite occur with some of the fine grained bedded pyrite occurrences.

1.5 METALLURGY

Tintina contracted Arthur H. Winckers, P.Eng. of Arthur H Winckers & Associates Inc. to conduct various metallurgical tests to determine the flotation response of composite samples representing the typical sulphide mineralization of the Johnny Lee Upper and Lower Zones. The objective of the preliminary metallurgical program was to develop effective flotation conditions for the recovery of copper and to identify potential amenability problems. The test work was conducted at the metallurgical division of Inspectorate Exploration and Mining Services Ltd, and the analytical work was conducted by Inspectorate's analytical division which has an International Organization for Standardization (ISO) 9001 accreditation and uses standard quality assurance/quality control (QA/QC) procedures. Supporting mineralogy studies were performed by G&T Metallurgical and SGS.

The samples selected for the test work are believed to be typical but not necessarily representative of the massive sulphide mineralization in the UZ and LZ of the Johnny Lee deposit. The test work completed to-date is appropriate for a PEA level of study but more test work on a much larger suite of samples taken from the across the mineralization in each zone is required for a feasibility-level study.

The investigations indicated the following results:

- The Johnny Lee UZ copper-cobalt mineralization is very fine grained and complex requiring a primary grind level of 80% passing 38 μm and a rougher concentrate regrind of 80% passing 8 μm for effective liberation and recovery of copper minerals to a marketable concentrate.
- The Johnny Lee LZ copper mineralization is much coarser grained and could be processed at a coarser grind but as the mineralized material from both zones will be comingled the process conditions of the locked cycle test on the LZ composite were kept the same as those used for the UZ composite; the LZ composite responded very well to these conditions.

The recovery of copper to concentrate from the UZ mineralization was estimated based on the locked cycle test results on the Master Composite which graded 2.24% copper; the annual copper recovery was then calculated to reflect the higher annual mine production plan head grades which have an LOM average of 2.6% copper. The average annual copper recovery for the head grade works out to 83.6% compared to the locked cycle test copper recovery of 82.2%.

The locked cycle test on the LZ composite with a head grade of 4% copper produced a concentrate grading 27% copper at a copper recovery of 96.6%. The LOM average grade mineralization of 4.9% copper is estimated to yield a copper recovery of 97%.

The concentrate produced in the locked cycle tests contained very low levels of potentially deleterious elements; this provides a preliminary indication that the risk with regard to the effect of deleterious elements on the Project economics is relatively low.

The cobalt and silver recoveries to concentrate for both zones were very low due to the complex very fine grain mineralogy of these elements which appear to be mostly associated with pyrite; the potential for economic recovery of these elements is considered to be very low.

1.6 JOHNNY LEE UZ MINERAL RESOURCES

Tintina contracted Mike Lechner, P.Geo., from RMI to review all applicable geologic and analytical data for the Johnny Lee UZ with the goal of estimating potential mineral resources. To that extent, Mr. Lechner used available drillhole data and various geologic information to construct a three-dimensional block model. Wireframes representing two copper-rich horizons were constructed by Tintina's technical staff and reviewed by Mr. Lechner. Those wireframes were used to constrain the estimate of block copper and cobalt grades.

A bulk density value of 3.99 based on 181 massive sulphide diamond core sample determinations was used to tabulate tonnage. A cut-off grade of 1.6% copper was used to estimate a Measured and Indicated mineral resource of 9,179,000 t with an average grade of 2.83% copper, 0.12% cobalt, 0.008 g/t gold, and 15.7 g/t silver. In addition to the Measured and Indicated resources, there is an estimated Inferred resource containing 1,255,000 t at an average grade of 2.52% copper, 0.10% cobalt, and 15.2 g/t silver using a copper cut-off grade of 1.6%. The cut-off grade was established by using a copper price of US$2.75/lb, a copper recovery of 81%, mining costs of US$59/t, processing costs of US$16.00/t, and general and administrative (G&A) costs of US$5.00/t. The current undiluted Johnny Lee UZ Measured and Indicated mineral resources are tabulated in Table 1.2. Undiluted Johnny Lee UZ Inferred mineral resources are tabulated in Table 1.3.

Table 1.2 Undiluted Johnny Lee UZ Measured and Indicated Resources

Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Measured	2,659	2.99	0.12	0.007	16.3	175	6.9	0.6	1,393
Indicated	6,520	2.77	0.13	0.009	15.5	398	18.0	1.9	3,249
Measured & Indicated	**9,179**	**2.83**	**0.12**	**0.008**	**15.7**	**573**	**24.9**	**2.5**	**4,642**

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

Table 1.3 Undiluted Johnny Lee UZ Inferred Resources

Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Inferred	1,255	2.52	0.10	0.008	15.2	70	2.8	0.3	613

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

1.7 JOHNNY LEE LZ MINERAL RESOURCES

Tintina personnel constructed a three-dimensional wireframe which represents a single copper-rich horizon within the Johnny Lee LZ. Mr. Lechner reviewed and confirmed the LZ wireframe, performed various statistical studies, and estimated resources for the LZ.

A bulk density value of 3.49 based on 53 LZ massive sulphide core sample determinations was used to tabulate tonnage. A cut-off grade of 1.5% copper was used to define an Indicated mineral resource of 2,387,000 t with an average grade of 6.40% copper, 0.03% cobalt, 0.304 g/t gold, and 4.5 g/t silver. The cut-off grade was established by using a copper price of US$2.75/lb, a copper recovery of 84%, mining costs of US$50/t, processing costs of US$16.00/t, G&A costs of US$5.00/t, and refining costs of US$5.53/t. Johnny Lee undiluted LZ Indicated resources are summarized in Table 1.4. The estimate of undiluted LZ Inferred mineral resources is tabulated in Table 1.5.

Table 1.4 Undiluted Johnny Lee LZ Indicated Resources

Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Indicated	2,387	6.40	0.03	0.304	4.5	337	1.7	23.3	345

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

Table 1.5 Undiluted Johnny Lee LZ Inferred Resources

Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Inferred	205	5.33	0.03	0.207	4.1	24	0.1	1.4	27

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

1.8 LOWRY MINERAL RESOURCES

The Lowry deposit resource is within the Project area, but is not included in the economic analysis of this updated PEA.

Tintina contracted Mike Lechner, P.Geo. to review all applicable geologic and analytical data for the Lowry Middle Zone (MZ) with the goal of estimating potential mineral resources. To that extent, Mr. Lechner used available drillhole data and various geologic information to construct a three-dimensional block model. Wireframes representing a copper-rich horizon were constructed by Tintina's technical staff and reviewed by Mr. Lechner. Those wireframes were used to constrain the estimate of block grades.

A bulk density value of 3.18 g/cm³ based on 117 diamond core sample determinations was used to tabulate tonnage. A cut-off grade of 1.6% copper was used to define estimated Indicated mineral resources of 4,099,000 t with an average grade of 2.94% copper, 0.10% cobalt, 0.006 g/t gold, and 15.1 g/t silver. Using the same cut-off grade, there is an estimated undiluted Inferred resource of 801,000 t with an average grade of 2.58% copper, 0.10% cobalt, 0.008 g/t gold, and 14.1 g/t silver. The cut-off grade was established by using a copper price of US$2.75/lb, a copper recovery of 81%, mining costs of US$59/t, processing costs of US$16.00/t, and G&A costs of US$5.00/t. The estimate of undiluted Lowry MZ Indicated mineral resources is tabulated in Table 1.6. The estimate of undiluted Lowry MZ Inferred mineral resources is tabulated in Table 1.7.

Table 1.6 Undiluted Lowry MZ Indicated Resources

Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Indicated	4,099	2.94	0.10	0.006	15.1	266	9	0.8	1,990

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

Table 1.7 Undiluted Lowry MZ Inferred Resources

Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Inferred	801	2.58	0.10	0.008	14.1	46	2	0.2	363

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

1.9 BLACK BUTTE TOTAL MINERAL RESOURCES

Table 1.8 and Table 1.9 shows undiluted Black Butte Measured and Indicated mineral resources, and Inferred mineral resources, respectively, that are pertinent to this updated PEA. The data shown in Table 1.8 and Table 1.9 do not include resources from the Lowry MZ as it is not included in the economic analysis of this updated PEA.

Table 1.8 Undiluted Black Butte Measured and Indicated Mineral Resources for Updated PEA

Area/Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Johnny Lee UZ Measured & Indicated	9,179	2.83	0.12	0.008	15.7	573	24.9	2.5	4,642
Johnny Lee LZ Indicated	2,387	6.40	0.03	0.304	4.5	337	1.7	23.3	345
Total Johnny Lee Measured & Indicated	**11,566**	**3.57**	**0.10**	**0.069**	**13.4**	**910**	**26.6**	**25.8**	**4,987**

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

Table 1.9 Undiluted Black Butte Inferred Mineral Resources for Updated PEA

Area/Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Johnny Lee UZ Inferred	1,255	2.52	0.10	0.008	15.2	70	2.8	0.3	613
Johnny Lee LZ Inferred	205	5.33	0.03	0.207	4.1	24	0.1	1.4	27
Total Johnny Lee Inferred	**1,460**	**2.91**	**0.09**	**0.036**	**13.6**	**94**	**2.9**	**1.7**	**640**

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

Table 1.10 and Table 1.11 tabulates the total Black Butte Measured+Indicated and Inferred resources, respectively. The data shown in Table 1.10 and Table 1.11 contain Lowry resources, which were not included in the economic analysis in this updated PEA.

Table 1.10 Total Undiluted Black Butte Measured and Indicated Mineral Resources

Area/Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Johnny Lee UZ Measured & Indicated	9,179	2.83	0.12	0.008	15.7	573	24.9	2.5	4,642
Johnny Lee LZ Indicated	2,387	6.40	0.03	0.304	4.5	337	1.7	23.3	345
Lowry MZ Indicated	4,099	2.94	0.10	0.006	15.1	266	9.0	0.8	1,990
Total Black Butte Measured & Indicated	**15,665**	**3.40**	**0.10**	**0.053**	**13.9**	**1,176**	**35.6**	**26.6**	**6,977**

Table 1.11 Total Undiluted Black Butte Inferred Mineral Resources

Area/Resource Category	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Johnny Lee UZ Inferred	1,255	2.52	0.10	0.008	15.2	70	2.8	0.3	613
Johnny Lee LZ Inferred	205	5.33	0.03	0.207	4.1	24	0.1	1.4	27
Lowry MZ Inferred	801	2.58	0.10	0.008	14.1	46	2.0	0.2	363
Total Black Butte Inferred	**2,261**	**2.80**	**0.09**	**0.026**	**13.8**	**140**	**4.9**	**1.9**	**1,003**

The Johnny Lee UZ and Lowry MZ resources were tabulated using a copper cut-off grade of 1.6%. The Johnny Lee LZ resource was tabulated using a 1.5% copper cut-off grade.

1.10 MINING

The Johnny Lee UZ and LZ will be accessed from a single portal, three main ramps, and one decline. There will be six raises that reach the surface, which will provide secondary egress and ventilation circuits. All personnel and materials will be transported through the portal and down the decline to the working areas. All mineralized material and waste will be trucked up the decline to stockpiles located on surface within 100 m of the portal. Paste backfill will be pumped from the paste plant through a pipe that will extend from the plant to the portal and down the decline to the working areas.

The drift-and-fill mining method was selected for the deposit due to the overall thinness and shallow dip of the mineralized zones, and for flexibility in adapting to local variations. Although the ground conditions may allow for larger stoping methods, the geometry of the deposit was the controlling factor in selecting this method.

The mine life is estimated to be 12 years, including 1.5 years of pre-production development and 10.5 years of production. Underground mine development will start at the beginning of Year 1. All initial development will start underground from the pre-existing exploration decline, approximately 1 km from the portal. Initial stope panel development will begin in Q3 of Year 2, and the full production rate of 3,300 t/d will begin in Q3 of Year 3.

Tintina supplied two block models that were used to estimate the mineralized material contained in the mining shapes: one block model for the UZ and one block model for the LZ. The two block models were used to estimate copper grades and rock densities. Table 1.12 summarizes the subset of mineral resources contained in the mine plan by mining area, and accounts for mining dilution and recovery assumptions. A nominal 1.9% copper cut-off grade was used for planning purposes.

Table 1.12 Subset of Mineral Resources in Mine Plan

Area/ Class	In Stope ('000 t)	In Stope Cu Grade (%)	In Stope Cu ('000 lb)	Mining Recovery (%)	Dilution (%)	Dilution Cu Grade (%)	Mineral Resources in Mine Plan*		
							'000 t	Cu Grade (%)	Cu ('000 lb)
Total Johnny Lee									
Measured	2,252	2.91	144,418	98.0	10	1.30	2,452	2.75	148,551
Indicated	7,622	3.50	587,593	98.0	10	1.21	8,299	3.27	598.011
Inferred	1,004	2.85	63,025	98.0	10	1.25	1,093	2.69	64,764

Note: *Dilution and mine losses applied.

1.11 PROCESS

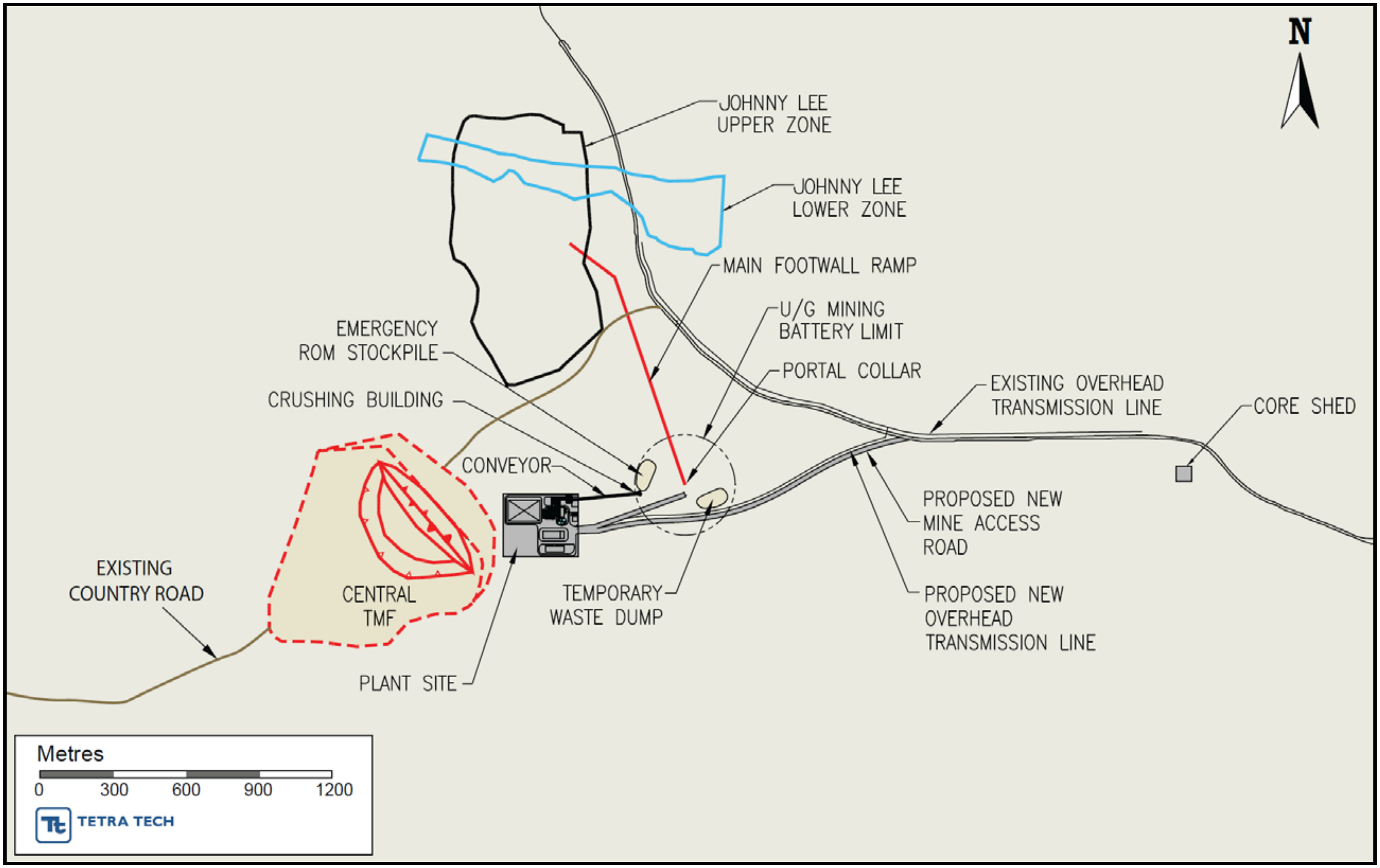
Tetra Tech designed a 3,300 t/d process plant for the Project to process sulphide mineralization containing copper and associated cobalt and silver. The process plant will operate in two 12-hour shifts per day, 365 d/a; the plant will process mineralized material at a nominal annual rate of 1,204,500 t. The crushing plant availability will be 70%, and grinding and flotation plant availability will be 92%.

The mill feed will be crushed by a jaw crusher to 80% passing 125 mm, and then ground to 80% passing 38 µm in a SAG/ball mill/tower mill circuit. SAG mill discharge screen oversize pebbles will be fed to a cone crusher. The ground material will be processed using copper rougher flotation followed by copper rougher concentrate regrinding in stirred mills; the reground copper rougher flotation concentrate will then be upgraded by three stages of cleaner flotation. Copper rougher flotation tailings, together with the copper cleaner scavenger flotation tailings, will be dewatered by thickening prior to being delivered to the backfilling plant or to the TMF. The third cleaner flotation concentrate, which will on average contain approximately 23.5% copper, will be thickened and then pressure-filtered before it is shipped to smelters. The LOM average copper recovery is estimated to be approximately 88.3%.

1.12 INFRASTRUCTURE

The Property is located in Meagher County, Montana, US, about 17 miles north of the town of White Sulphur Springs. Figure 1.2 illustrates the overall Project site layout.

Figure 1.2 General Arrangement Plan



The Property is currently accessed by a 1.5-mile gravel road leading from US Highway 89, an all-weather state-maintained highway. The gravel access road will require minimal upgrading to service the mine.

The process or mill building will house the semi-autogenous grinding (SAG) mill, ball mill and tower mill, rougher floation cells and cleaner floation columns, regrind area, reagent area, concentrate surge tank, concentrate filter press, and laydown areas. There is a mezzanine level above for the control room, offices, and electrical room. The tailings thickener, concentrate thickener, and water services will be located immediately north of the process building. A sprung structure housing the concentrate stockpile and load-out will be located adjacent to the west side of the process plant. An optical fibre backbone is included throughout the plant in order to provide a path for the data requirements for voice, data, and control systems. A fibre backbone for a site ethernet-type system is included to provide data and voice bandwidth.

The administrative building is a single-storey steel structure and will house the mine dry, lockers, shower facilities, first aid and emergency vehicle parking, as well as office areas for management, administrative, engineering, and geology personnel.

The maintenance/truck shop and warehouse (cold/warm) will house a wash bay, repair bays, parts storage areas, welding area, machine shop, electrical room, mechanical room, compressor room, and lube storage room. The facility will also house the cold/warm storage warehouse and support warehouse and maintenance personnel. The facility is designed to support both the mining haul fleet and the process plant fleet.

Fuel storage requirements for the mining equipment, process equipment, and ancillary facilities will be supplied from above-ground diesel fuel tanks located near the truck shop. A dedicated service truck will transport the fuel to the mining equipment and the process plant fleet.

The assay laboratory will be a single-story modular building, complete with all equipment required for metallurgical grade testing and control. The laboratory will also be equipped with heating, ventilation, and air conditioning (HVAC) systems and chemical disposal equipment.

1.12.1 Tailings Management Facility

The tailings management facility (TMF) is designed as a partially excavated impoundment contained by an earthfill embankment and lined with a 100 mil high-density polyethylene (HDPE) liner to minimize seepage from the facility. The TMF will store 5.92 Mt of tailings (50% of total tailings production) over the mine life, with the remainder of the tailings pumped back into the underground mine workings as backfill. The interior slopes of the impoundment will be at a 3H:1V slope to facilitate liner installation. The downstream slope of the final embankment will be constructed at a 2H:1V slope.

The embankment for the impoundment will be constructed in stages during operations in order to limit capital costs and maintain an inherent flexibility to allow for variations in operation and production throughout the life of the mine. The starter impoundment will be built to contain tailings and PAG waste rock for the first four years of the mine life. A cut/fill approach will be utilized in the construction of the impoundment where excavated soils will be used as embankment fill material.

Regional evaporation data indicates a surface water deficit will exist during operations. An external water supply system will be constructed to provide water for the plant systems and the supernatant pond, chiefly supplied from dewatering the mine. It is possible that ongoing dewatering of the mine may result in a water surplus, particularly during the latter stages of the mine life. It is assumed that mine water inflows to the Project components will exceed the consumption and losses; therefore, a water treatment plant and disposal system may be required in the latter years of the LOM.

1.13 ENVIRONMENTAL

Tintina proposes to conduct advanced exploration of the Johnny Lee deposit through development of an exploration adit decline in 2013, with subsequent permitting of full scale mining operations by 2015. The Project is located in Meagher County, Montana, 17 miles north of White Sulphur Springs, in the headwaters of the Sheep Creek drainage. The site ranges in elevation from 5,600 to 6,800 ft atop Black Butte. Timber-covered hills surround grass and mountain sagebrush-covered valleys, which are used predominantly for agricultural and recreational purposes. Annual precipitation averages 13″ to 16″.

Permits to mine this privately owned land will be issued by the Montana Department of Environmental Quality (DEQ), following submittal of complete and compliant operating plans, environmental baseline studies, and reclamation plans, and completion of an Environmental Impact Statement (EIS) in compliance with the *Montana Environmental Policy Act* (MEPA). In addition to a mine operating permit, the Project will likely require permitting of power lines under the *Major Facility Siting Act*, a Surface Water Discharge Permit, verification of water rights, an air quality permit, permits to modify streams in compliance with the *Montana Streambed Preservation Act* (Section 310) and the *US Clean Water Act* (Section 404) for wetlands, a *Montana Hard Rock Impact Act* permit to manage socioeconomic impacts, and approval from the Montana State Historic Preservation Office (SHPO). The environmental baseline review will consider resources that may be affected by the proposed operations, including surface and groundwater, geology/topography, rock/sediment, soil, wetlands, vegetation, fish and wildlife, and historical/cultural resources. Baseline studies have been initiated for water resources, environmental geochemistry, soils, wetlands, vegetation, wildlife, and cultural resources, with approximately two years of data collection completed. Key environmental issues for the Project will be acid rock drainage and metal mobility risks to surface and groundwater resources from waste rock and tailings, due to elevated sulphide content of ore-bearing portions of the mineralized deposit, and management of water from underground workings. Water treatment facilities are planned.

1.14 CAPITAL AND OPERATING COSTS

1.14.1 CAPITAL COST

An initial capital cost of US$217.8 million is estimated for the Project (Table 1.13). All currencies in this section are expressed in US dollars and an exchange rate of Cdn$1.00=US$1.00 has been used throughout this updated PEA.

This estimate has been prepared in accordance with recommended practices of the Association for the Advancement of Cost Engineering (AACE). It is a Class 5 estimate (International Classification System) with an expected accuracy range of ±40%.

Contributors to the estimate include:

- AMEC: underground mining and backfill
- Knight Piésold: tailings and reclaim, and water management
- Tintina: Owner's costs.

This estimate is prepared with a base date of Q1 2013 and does not include any escalation past this date. The quotations used in this estimate were obtained in Q1 2013 and are budgetary and non-binding.

Budget quotations were obtained for all major equipment. The vendors provided equipment prices, delivery lead times. For non-major equipment, costing is based on in-house data or quotes from recent similar projects.

All equipment and material costs include Free Carrier (FCA) manufacturer plant Inco terms 2000. Other costs such as spares, freight and initial fills will be covered separately in the Indirects section of the estimate.

Table 1.13 Capital Cost Summary

Item	Total Cost ($)
Direct Costs	
Overall Site	2,790,724
Mine Capital	54,406,432
Mine Surface Facilities	12,017,674
Processing	52,218,559
Water Management (Knight Piésold)	11,069,469
Utilities	5,314,571
Buildings	8,242,691
Off-site Infrastructure	4,066,207
Plant Mobile Equipment	2,063,212
Subtotal	**152,199,539**

table continues...

Item	Total Cost ($)
Indirect Costs	26,567,854
Owner's Costs	5,642,746
Contingency	33,342,538
Total Capital Costs	**217,752,677**

Note: Numbers may not add due to rounding.

1.14.2 Operating Cost

On site operating costs are estimated to be US$66.48/t milled including mining, processing, G&A, and plant services. A total of 11,844,000 t mineralization from the underground mine will be processed during the LOM based on the proposed mining schedule. On average, the annual nominal process rate is approximately 1,204,500 t/a or 3,300 t/d for 365 d/a. The unit cost is estimated based on the LOM average mill feed rate. Table 1.14 summarizes the operating costs.

Table 1.14 Operating Cost Summary

Area	Unit Operating Cost (US$/t milled) at LOM
Mining*	45.83
Processing	15.83
Tailings Management	0.25
G&A	2.97
Plant Services	1.60
Total	**66.48**

Note: *Including backfill cost and mining electrical energy cost.

1.15 Economic Analysis

Tetra Tech performed a base case, 100% equity, pre-tax economic analysis of the Project, based on the following:

- price of copper – US$3.05/lb
- total LOM production of 11,844,000 t of mineralized material
- average grade of 3.11% copper and average process recovery of 88.3%
- total of 716,014,000 lb of copper recovered over the 11-year LOM and 65,092,000 lb of copper recovered per year
- LOM payable copper value of US$2,081,979,000 with an on-site operating cost estimate of US$787,370,000 and a total LOM capital cost estimate of US$346,007,000.

The resulting pre-tax discounted cash flow NPV at 8% is $217,926,000, the IRR is 30.5%, and the payback period is 3.6 years.

The resulting post-tax discounted cash flow NPV is 8% at $109,967,000, the IRR is 20.2%, and the payback period is 4.7 years.

In addition to the possible impact on overall economics that could result from variations in process recovery or mineralized material grades, sensitivity analyses show that the Project economics are particularly sensitive to changes in copper price with lesser influence from operating and capital costs. It is apparent that the copper price would have a very significant impact on profitability of the Project.

This updated PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

1.16 Project Development

It is estimated that the Project will take approximately 15 to 18 months of construction activities to complete.

1.17 Opportunities and Recommendations

Based on the work carried out in this updated PEA and the resultant economic evaluation, Tetra Tech recommends that Tintina continue investigating the Property and proceed to the next phase of study to further assess the economic viability of the Project.

Detailed opportunities and recommendations are provided in Section 26.0 of this technical report, along with the associated costs.

2.0 INTRODUCTION

Tetra Tech was commissioned by Tintina to complete a technical report on the Project. Tetra Tech has prepared this report in accordance with guidelines provided in NI 43-101 Standards of Disclosure for Mineral Projects.

The objectives of the report are to:

- prepare a technical report on the Project in accordance with NI 43-101
- summarize land tenures, exploration history, and drilling
- generate a resource estimate on the Johnny Lee deposit
- provide a PEA of the Project based on an economic evaluation for processing at a maximum rate of 3,300 t/d and to assist Tintina with the Project development
- provide recommendations and budget for additional work on the Property.

A summary of qualified persons (QPs) responsible for each section of this report is provided in Table 2.1. The following QPs completed a site visit of the Property:

- Ken Brouwer, P.Eng. completed a site visit on February 1, 2011.
- Lisa Kirk, P.G. completed a site visit on October 27, 2011.
- Michael Lechner, P.Geo. completed a site visit on September 20, 2011.

Table 2.1 Summary of QPs

Report Section		Company	QP
1.0	Summary	All	Sign-off by Section
2.0	Introduction	Tetra Tech	Andrea Cade, P.Geo.
3.0	Reliance on Other Experts	Tetra Tech	Andrea Cade, P.Geo.
4.0	Property Description and Location	RMI	Michael J. Lechner, P.Geo.
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	RMI	Michael J. Lechner, P.Geo.
6.0	History	RMI	Michael J. Lechner, P.Geo.
7.0	Geological Setting and Mineralization	RMI	Michael J. Lechner, P.Geo.
8.0	Deposit Types	RMI	Michael J. Lechner, P.Geo.
9.0	Exploration	RMI	Michael J. Lechner, P.Geo.
10.0	Drilling	RMI	Michael J. Lechner, P.Geo.
11.0	Sample Preparation, Analyses, and Security	RMI	Michael J. Lechner, P.Geo.
12.0	Data Verification	RMI	Michael J. Lechner, P.Geo.

table continues...

Report Section		Company	QP
13.0	Mineral Processing and Metallurgical Testing	Arthur H. Winckers & Associates	Arthur H. Winckers, P.Eng.
14.0	Mineral Resource Estimates	RMI	Michael J. Lechner, P.Geo.
15.0	Mineral Reserve Estimates	AMEC	Srikant Annavarapu, RM SME
16.0	Mining Methods	AMEC	Srikant Annavarapu, RM SME
17.0	Recovery Methods	Tetra Tech	John Huang, Ph.D., P.Eng.
18.0	Infrastructure	-	-
	18.1 Introduction	Tetra Tech	John Huang, Ph.D., P.Eng.
	18.2 Roads	Tetra Tech	John Huang, Ph.D., P.Eng.
	18.3 Buildings	Tetra Tech	John Huang, Ph.D., P.Eng.
	18.4 Tailings Management Facility	Knight Piésold	Ken Brouwer, P.Eng.
	18.5 Seepage Management	Knight Piésold	Ken Brouwer, P.Eng.
	18.6 Instrumentation Installation and Monitoring	Knight Piésold	Ken Brouwer, P.Eng.
	18.7 Tailings Delivery System	Knight Piésold	Ken Brouwer, P.Eng.
	18.8 Reclaim Water System	Knight Piésold	Ken Brouwer, P.Eng.
	18.9 Waste Rock Storage Area	Knight Piésold	Ken Brouwer, P.Eng.
	18.10 Additional Water Management Facilities	Knight Piésold	Ken Brouwer, P.Eng.
	18.11 Power Distribution, Energy Efficiency, and Utilization	Tetra Tech	John Huang, Ph.D., P.Eng.
	18.12 Underground Mine Related Infrastructure Projections	AMEC	Srikant Annavarapu, RM SME
	18.13 Proposed Paste Backfill Plant	AMEC	Srikant Annavarapu, RM SME
19.0	Market Studies and Contracts	Tetra Tech	Andrea Cade, P.Geo.
20.0	Environmental Studies, Permitting, and Social or Community Impact	Enviromin	Lisa Kirk, P.G.
21.0	Capital and Operating Costs	-	-
	21.1 Capital Cost Estimate	Tetra Tech	Harvey Wayne Stoyko, P.Eng.
	21.2 Mining Costs – Basis of Estimate	AMEC	Srikant Annavarapu, RM SME
	21.3 Underground Mine Capital Cost Estimate	AMEC	Srikant Annavarapu, RM SME
	21.4 Paste Backfill Plant – Capital Costs	AMEC	Srikant Annavarapu, RM SME
	21.5 Operating Cost Estimate	-	-
	21.5.1 Summary	Tetra Tech	John Huang, Ph.D., P.Eng.
	21.5.2 Mine Operating Costs	AMEC	Srikant Annavarapu, RM SME
	21.5.3 Paste Backfill Plant – Operating Costs	AMEC	Srikant Annavarapu, RM SME
	21.5.4 Processing Operating Costs	Tetra Tech	John Huang, Ph.D., P.Eng.
	21.5.5 General and Administrative Costs and Surface Services Costs	Tetra Tech	John Huang, Ph.D., P.Eng.
	21.5.6 Tailings Management Cost	Knight Piésold	Ken Brouwer, P.Eng.
22.0	Economic Analysis	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
23.0	Adjacent Properties	RMI	Michael J. Lechner, P.Geo.

table continues...

Report Section		Company	QP
24.0	Other Relevant Data and Information	Tetra Tech	Andrea Cade, P.Geo.
25.0	Interpretation and Conclusions	All	Sign-off by Section
26.0	Recommendations	All	Sign-off by Section
27.0	References	All	Sign-off by Section
28.0	Certificates of Qualified Persons	All	Sign-off by Section

2.1 INFORMATION AND DATA SOURCES

A complete of references is provided in Section 27.0.

3.0 RELIANCE ON OTHER EXPERTS

Tetra Tech has reviewed and analyzed data and reports provided by Tintina Resources, together with publicly available data, and has drawn its own conclusions augmented by direct field examination.

Tetra Tech has not independently verified the legal status or ownership of the mineral properties or underlying lease option agreements. Tetra Tech is relying on Tintina regarding statements about the validity of the Property position.

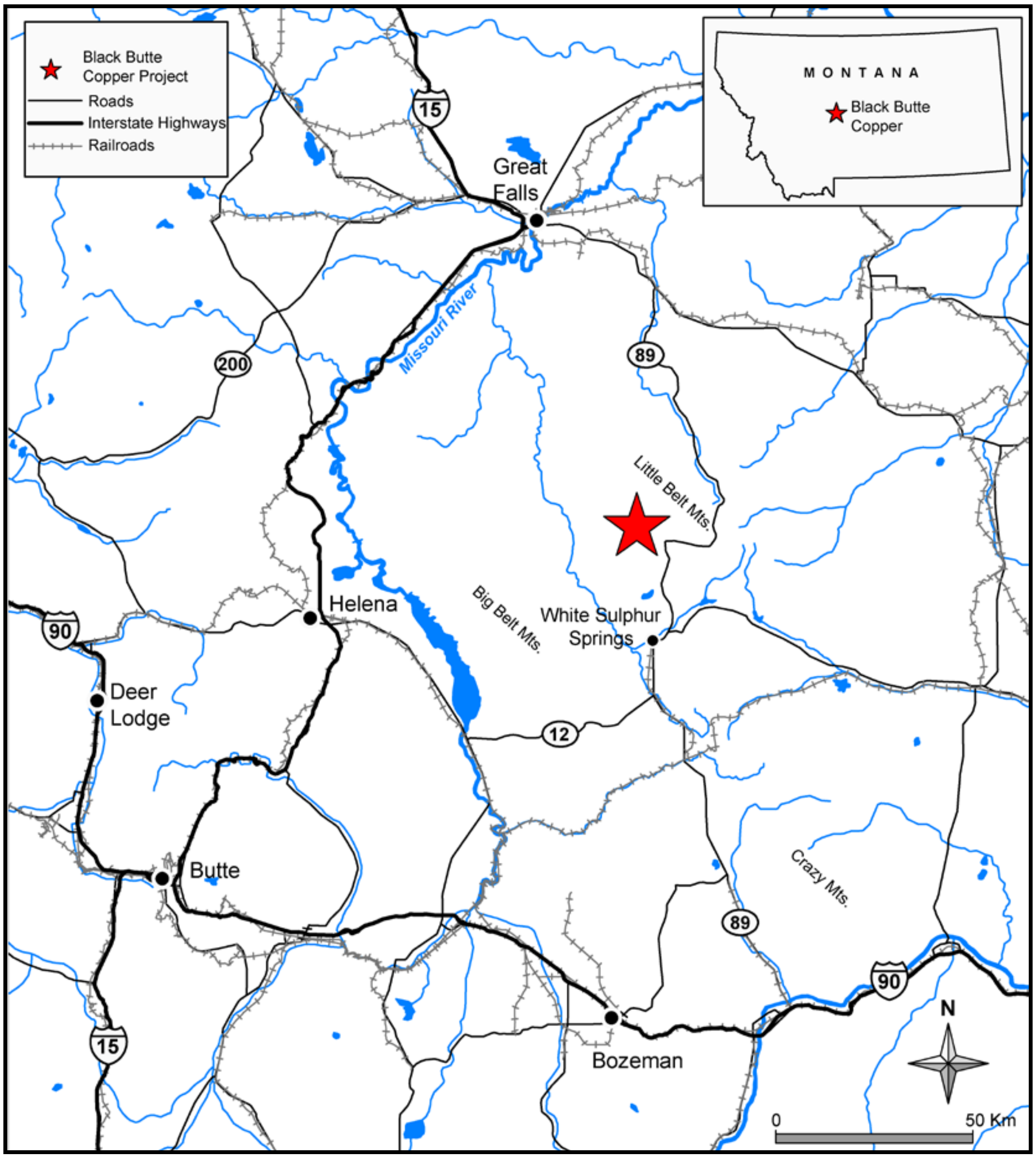
The QPs who prepared this report relied upon information provided by the following experts who are not QPs:

- Mr. Don ("Fess") Foster, an exploration/mining permit specialist has been relied on for advice on matters relating to general permitting trends in Section 4.0.
- Mr. Allan R. Kirk, Principal Geologist, Geomin Resources, Inc., has been relied on for review of permitting requirements and land position, general environmental information in Section 20.0.
- Mr. William Thompson, Principal Hydrogeologist, Hydrometrics, has been relied on for hydrogeology and water quality data in Section 20.0.
- Mr. Shane Matolyak, Environmental Scientist, Tetra Tech, has been relied on for soil surveys/land application baseline characterization in Section 20.0.
- Dr. Joe Elliott, independent Wildlife Biologist has been relied on for matters relating to wildlife biology in Section 20.0.
- PricewaterhouseCoopers LLP (PwC), has been relied on for advice concerning tax matters relevant to the technical report. The reliance is based on a letter to Tintina entitled "Assistance with the calculation and review of the income and mineral tax portions of the economic analysis prepared by Tetra Tech Wardrop ("Tetra Tech") in connection with the Preliminary Economic Assessment Report (the "Report") on Tintina Resources Inc.'s ("Tintina") Black Butte Project (the "Project")" and dated July 10, 2013. Sabry Abdel Hafez, Ph.D., P.Eng. has relied entirely on this letter for disclosure contained in Section 22.0. Sabry Abdel Hafez, Ph.D., P.Eng. believes that it is reasonable to rely on PwC, based on the assumption that PwC staff have the necessary education, professional designations, and relevant experience in tax matters relevant to the technical report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located in Meagher County, Montana, approximately 17 miles north of the town of White Sulphur Springs, as shown in Figure 4.1.

Figure 4.1 General Location Map



Tintina's land holdings are located in sections 23, 24, 25, 26, 28, 32, 33, 34, 35, and 36, Township 12 North, Range 6 East, sections 19, 29, 30, 31, and 32, Township 12 North, Range 7 East, sections 1, 2, 3, 4, 6, 7, 10, 11, 12, and 13 Township 11 North, Range 6 East, sections 5, 6, 7, 8, and 18 Township 11 North, Range 7 East, and section 1 and 12, Township 11 North, Range 5 East.

The Property consists of three tracts of fee-simple lands totalling 7,684.28 ac and 239 unpatented lode mining claims on US Forest Service lands totalling approximately 4,541 ac. For Tract 1 (named for purposes of this description only), the Bar Z Ranch controls 100% surface and three members of the Hanson family share equal interest in 100% of the mineral interest. For Tract 2, Mrs. Rose I. Holmstrom, a local rancher, controls 100% of both surface and mineral interests. For Tract 3, Steve Buckingham, a local rancher, controls 100% of the surface rights and shares equal interest in the mineral interests with two siblings. The 56 mining claims (named the SB claims) were staked by Tintina Alaska Exploration Inc., a wholly owned subsidiary of Tintina Resources, Inc., in November 2010 for a total of approximately 1,064 ac. An additional 183 BSP claims were staked by Tintina Alaska Exploration Inc. in the spring of 2011 and total approximately 3,477 ac.

Tract 1, for which Tintina has one surface lease and three mineral leases, consists of 2,594.28 ac of surface 100% owned by Bar Z Ranch, and includes the surface over 2,555 mineral acres covered by three mining leases with the Hanson family, each of whom own one third of the mineral interest. The additional 39.28 ac covered by the surface lease consists of two patented mining claims, the Copper Hill (Mineral Survey #10311) and Rio Tinto (Mineral Survey #10304). The mineral rights for these two claims are owned by another party with whom Tintina has no agreement. The surface lease requires lease payments of $50,000 on signing (May 2, 2010) and on each of the first four anniversary dates. Payments on the fifth anniversary date and each anniversary date thereafter are $75,000.

Each of the three mining leases for Tract 1 requires advance minimum royalty payments of $16,150 on signing (May 2, 2010) and on the first and second anniversaries, $32,300 on the third anniversary, $48,450 on the fourth anniversary $64,600 on each anniversary thereafter through the term of the lease.

The term is for 30 years and is renewable for subsequent periods of 10 years each. The combined mineral interest has a net smelter return (NSR) of 5%, with an option to buy this down to a 2% NSR for $5,000,000, thereby reducing each mineral lessors royalty to 0.6666% NSR in return for a payment of $1,666,666. Exercising the buy down option eliminates further advance minimum royalty payments.

Tract 2 consists of 2,120 ac and is subject to a single mining lease with Mrs. Rose I. Holmstrom for 100% surface and 100% mineral interest. The agreement requires advance minimum royalty payments of $40,195 on signing (May 2, 2010) and on the first and second anniversaries, $80,411 on the third anniversary, $120,607 on the fourth anniversary, and $160,802 on each anniversary thereafter through the term of the lease. The term is for 30 years and is renewable for subsequent periods of 10 years each. The agreement has a 5% NSR with an option to buy this down to a 2% NSR for $5,000,000.

Tract 3 consists of 2,970 ac and is subject to a single mining lease with Mr. Steve Buckingham, 100% surface owner and one-third mineral owner, and his two siblings, Kathy Johnston and Marilyn Bodell, each one-third mineral owners. The agreement requires advance minimum royalty payments of $5,000 on signing, $15,000 on or before six months after signing, $20,000 on or before the first and second anniversaries, $25,000 on or before the third through fifth anniversaries, $30,000 on or before the sixth through eighth anniversaries, $35,000 on or before the ninth through eleventh anniversaries, $40,000 on or before the twelfth through fourteenth anniversaries, and $50,000 per annum through the remainder of the lease term or until commercial production. The agreement has a term of 30 years and a 5% NSR, which can be bought down to 2% NSR for a payment of $5,000,000.

The mineral owners warrant that there are no prior or underlying agreements encumbering the above described surface and mineral interest. All agreements stipulate underground mining only.

Property boundaries for Tracts 1, 2, and 3 are based on the government-surveyed meridian, section, township, and range system marked at section corners and some one-quarter section corners with permanent brass cap markers. Ranch owners generally align fences along property boundaries based on these survey markers.

There are no accessible mine workings on the Property, only shallow prospect pits and a caved 70 ft deep shaft (Section 6.0). The location of mineralized zones and resources is shown in Figure 4.2. None of these zones have been developed.

Table 4.1 lists the SB unpatented claims and Table 4.2 lists the BSP unpatented claims.

There are no recognized significant environmental liabilities on the Property. Sheep Creek supports livestock and irrigation, as well as fisheries, and mine development on the Property must protect in-stream flow and water quality. Permitting for exploration and development drilling is granted by the Montana DEQ, and the necessary permits for such drilling have been granted to Tintina.

Table 4.1 SB Unpatented Claim List

Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document	BLM Serial Number
SB-1	S 32, T12N, R6E	137906	MMC-223234	SB-29	S 32, T12N, R6E	137934	MMC-223262
SB-2	S 32, T12N, R6E	137907	MMC-223235	SB-30	S 32, T12N, R6E	137935	MMC-223263
SB-3	S 32, T12N, R6E	137908	MMC-223236	SB-31	S 32, T12N, R6E	137936	MMC-223264
SB-4	S 32, T12N, R6E	137909	MMC-223237	SB-32	S 32, T12N, R6E	137937	MMC-223265
SB-5	S 32, T12N, R6E	137910	MMC-223238	SB-33	S 5, T12N, R6E; S 32, T12N, R6E	137938	MMC-223266
SB-6	S 32, T12N, R6E	137911	MMC-223239	SB-34	S 5, T12N, R6E; S 32, T12N, R6E	137939	MMC-223267
SB-7	S 32, T12N, R6E	137912	MMC-223240	SB-35	S 5, T12N, R6E; S 32, T12N, R6E	137940	MMC-223268
SB-8	S 32, T12N, R6E	137913	MMC-223241	SB-36	S 5, T12N, R6E; S 32, T12N, R6E	137941	MMC-223269
SB-9	S 32, T12N, R6E	137914	MMC-223242	SB-37	S 28, T12N, R6E	137942	MMC-223270
SB-10	S 32, T12N, R6E	137915	MMC-223243	SB-38	S 28, T12N, R6E	137943	MMC-223271
SB-11	S 32, T12N, R6E	137916	MMC-223244	SB-39	S 28, T12N, R6E	137944	MMC-223272
SB-12	S 32, T12N, R6E	137917	MMC-223245	SB-40	S 28, T12N, R6E	137945	MMC-223273
SB-13	S 32, T12N, R6E	137918	MMC-223246	SB-41	S 28, T12N, R6E	137946	MMC-223274
SB-14	S 32, T12N, R6E	137919	MMC-223247	SB-42	S 28, T12N, R6E	137947	MMC-223275
SB-15	S 32, T12N, R6E	137920	MMC-223248	SB-43	S 34, T12N, R6E	137948	MMC-223276
SB-16	S 32, T12N, R6E	137921	MMC-223249	SB-44	S 34, T12N, R6E	137949	MMC-223277
SB-17	S 32, T12N, R6E	137922	MMC-223250	SB-45	S 34, T12N, R6E	137950	MMC-223278
SB-18	S 32, T12N, R6E	137923	MMC-223251	SB-46	S 34, T12N, R6E	137951	MMC-223279
SB-19	S 32, T12N, R6E	137924	MMC-223252	SB-47	S 34, T12N, R6E	137952	MMC-223280
SB-20	S 32, T12N, R6E	137925	MMC-223253	SB-48	S 34, T12N, R6E	137953	MMC-223281
SB-21	S 32, T12N, R6E	137926	MMC-223254	SB-49	S 34, T12N, R6E	137954	MMC-223282
SB-22	S 32, T12N, R6E	137927	MMC-223255	SB-50	S 34, T12N, R6E	137955	MMC-223283
SB-23	S 32, T12N, R6E	137928	MMC-223256	SB-51	S 34, T12N, R6E; S 3, T11N, R6E	137956	MMC-223284

table continues…

Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document	BLM Serial Number
SB-24	S 32, T12N, R6E	137929	MMC-223257	SB-52	S 34, T12N, R6E; S 3, T11N, R6E	137957	MMC-223285
SB-25	S 32, T12N, R6E	137930	MMC-223258	SB-53	S 34, T12N, R6E; S 3, T11N, R6E	137958	MMC-223286
SB-26	S 32, T12N, R6E	137931	MMC-223259	SB-54	S 34, T12N, R6E; S 3, T11N, R6E	137959	MMC-223287
SB-27	S 32, T12N, R6E	137932	MMC-223260	SB-55	S 34, T12N, R6E	137960	MMC-223288
SB-28	S 32, T12N, R6E	137933	MMC-223261	SB-56	S 28, T12N, R6E	137961	MMC-223289

Table 4.2 BSP Unpatented Claim List

Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
BSP-1	S 4&5, T11N, R6E; S 32&33, T12N, R6E	138254	MMC-223580	BSP-51	S 3, T11N, R6E	138304	MMC-223630
BSP-2	S 4&5, T11N, R6E	138255	MMC-223581	BSP-52	S 3&10, T11N, R6E	138305	MMC-223631
BSP-3	S 4, T11N, R6E; S 33, T12N, R6E	138256	MMC-223582	BSP-53	S 3, T11N, R6E	138306	MMC-223632
BSP-4	S 4, T11N, R6E	138257	MMC-223583	BSP-54	S 3&10, T11N, R6E	138307	MMC-223633
BSP-5	S 4, T11N, R6E; S 33, T12N, R6E	138258	MMC-223584	BSP-55	S 2&3, T11N, R6E	138308	MMC-223634
BSP-6	S 4, T11N, R6E	138259	MMC-223585	BSP-56	S 2, T11N, R6E	138309	MMC-223635
BSP-7	S 4, T11N, R6E	138260	MMC-223586	BSP-57	S 2, T11N, R6E	138310	MMC-223636
BSP-8	S 4, T11N, R6E	138261	MMC-223587	BSP-58	S 2, T11N, R6E	138311	MMC-223637
BSP-9	S 4, T11N, R6E	138262	MMC-223588	BSP-59	S 2, T11N, R6E	138312	MMC-223638
BSP-10	S 4, T11N, R6E	138263	MMC-223589	BSP-60	S 2, T11N, R6E	138313	MMC-223639
BSP-11	S 4, T11N, R6E	138264	MMC-223590	BSP-61	S 2, T11N, R6E	138314	MMC-223640
BSP-12	S 4, T11N, R6E	138265	MMC-223591	BSP-62	S 2, T11N, R6E	138315	MMC-223641
BSP-13	S 4, T11N, R6E	138266	MMC-223592	BSP-63	S 2, T11N, R6E	138316	MMC-223642

table continues...

Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
BSP-14	S 4, T11N, R6E	138267	MMC-223593	BSP-64	S 10&11, T11N, R6E	138317	MMC-223643
BSP-15	S 4, T11N, R6E	138268	MMC-223594	BSP-65	S 2,3,10&11, T11N, R6E	138318	MMC-223644
BSP-16	S 4, T11N, R6E	138269	MMC-223595	BSP-66	S 11, T11N, R6E	138319	MMC-223645
BSP-17	S 4, T11N, R6E	138270	MMC-223596	BSP-67	S 2&11, T11N, R6E	138320	MMC-223646
BSP-18	S 4, T11N, R6E	138271	MMC-223597	BSP-68	S 11, T11N, R6E	138321	MMC-223647
BSP-19	S 3, T11N, R6E	138272	MMC-223598	BSP-69	S 2&11, T11N, R6E	138322	MMC-223648
BSP-20	S 3, T11N, R6E	138273	MMC-223599	BSP-70	S 11, T11N, R6E	138323	MMC-223649
BSP-21	S 3, T11N, R6E	138274	MMC-223600	BSP-71	S 2&11, T11N, R6E	138324	MMC-223650
BSP-22	S 3, T11N, R6E	138275	MMC-223601	BSP-72	S 11, T11N, R6E	138325	MMC-223651
BSP-23	S 3, T11N, R6E	138276	MMC-223602	BSP-73	S 2&11, T11N, R6E	138326	MMC-223652
BSP-24	S 3, T11N, R6E	138277	MMC-223603	BSP-74	S 11, T11N, R6E	138327	MMC-223653
BSP-25	S 3, T11N, R6E	138278	MMC-223604	BSP-75	S 2&11, T11N, R6E	138328	MMC-223654
BSP-26	S 3, T11N, R6E	138279	MMC-223605	BSP-76	S 11, T11N, R6E	138329	MMC-223655
BSP-27	S 3, T11N, R6E	138280	MMC-223606	BSP-77	S 2&11, T11N, R6E	138330	MMC-223656
BSP-28	S 3, T11N, R6E	138281	MMC-223607	BSP-78	S 11, T11N, R6E	138331	MMC-223657
BSP-29	S 3, T11N, R6E	138282	MMC-223608	BSP-79	S 2&11, T11N, R6E	138332	MMC-223658
BSP-30	S 3, T11N, R6E	138283	MMC-223609	BSP-80	S 11, T11N, R6E	138333	MMC-223659
BSP-31	S 3, T11N, R6E	138284	MMC-223610	BSP-81	S 2&11, T11N, R6E	138334	MMC-223660
BSP-32	S 3, T11N, R6E	138285	MMC-223611	BSP-82	S 11&12, T11N, R6E	138335	MMC-223661
BSP-33	S 3, T11N, R6E	138286	MMC-223612	BSP-83	S 1,2,11&12, T11N, R6E	138336	MMC-223662
BSP-34	S 3, T11N, R6E	138287	MMC-223613	BSP-84	S 12, T11N, R6E	138337	MMC-223663
BSP-35	S 3, T11N, R6E	138288	MMC-223614	BSP-85	S 1&12, T11N, R6E	138338	MMC-223664
BSP-36	S 3, T11N, R6E	138289	MMC-223615	BSP-86	S 12, T11N, R6E	138339	MMC-223665
BSP-37	S 3, T11N, R6E	138290	MMC-223616	BSP-87	S 1&12, T11N, R6E	138340	MMC-223666
BSP-38	S 3, 9&10, T11N, R6E	138291	MMC-223617	BSP-88	S 12, T11N, R6E	138341	MMC-223667

table continues...

Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
BSP-39	S 3, T11N, R6E	138292	MMC-223618	BSP-89	S 1&12, T11N, R6E	138342	MMC-223668
BSP-40	S 3&10, T11N, R6E	138293	MMC-223619	BSP-90	S 12, T11N, R6E	138343	MMC-223669
BSP-41	S 3, T11N, R6E	138294	MMC-223620	BSP-91	S 1&12, T11N, R6E	138344	MMC-223670
BSP-42	S 3, T11N, R6E	138295	MMC-223621	BSP-92	S 12, T11N, R6E	138345	MMC-223671
BSP-43	S 3, T11N, R6E	138296	MMC-223622	BSP-93	S 1&12, T11N, R6E	138346	MMC-223672
BSP-44	S 3&10, T11N, R6E	138297	MMC-223623	BSP-94	S 12, T11N, R6E	138347	MMC-223673
BSP-45	S 3, T11N, R6E	138298	MMC-223624	BSP-95	S 1&12, T11N, R6E	138348	MMC-223674
BSP-46	S 3&10, T11N, R6E	138299	MMC-223625	BSP-96	S 12, T11N, R6E	138349	MMC-223675
BSP-47	S 3, T11N, R6E	138300	MMC-223626	BSP-97	S 1&12, T11N, R6E	138350	MMC-223676
BSP-48	S 3&10, T11N, R6E	138301	MMC-223627	BSP-98	S 12, T11N, R6E	138351	MMC-223677
BSP-49	S 3, T11N, R6E	138302	MMC-223628	BSP-99	S 1&12, T11N, R6E	138352	MMC-223678
BSP-50	S 3&10, T11N, R6E	138303	MMC-223629	BSP-100	S 7, T11N, R7E	138353	MMC-223679
BSP-101	S 6&7, T11N, R7E	138354	MMC-223680	BSP-151	S 7, T11N, R7E	138404	MMC-223730
BSP-102	S 7, T11N, R7E	138355	MMC-223681	BSP-152	S 7,8,17&18, T11N, R7E	138405	MMC-223731
BSP-103	S 6&7, T11N, R7E	138356	MMC-223682	BSP-153	S 7&8, T11N, R7E	138406	MMC-223732
BSP-104	S 7, T11N, R7E	138357	MMC-223683	BSP-154	S 6, T11N, R7E	138407	MMC-223733
BSP-105	S 6&7, T11N, R7E	138358	MMC-223684	BSP-155	S 6, T11N, R7E	138408	MMC-223734
BSP-106	S 7, T11N, R7E	138359	MMC-223685	BSP-156	S 6, T11N, R7E	138409	MMC-223735
BSP-107	S 6&7, T11N, R7E	138360	MMC-223686	BSP-157	S 6, T11N, R7E	138410	MMC-223736
BSP-108	S 7, T11N, R7E	138361	MMC-223687	BSP-158	S 6, T11N, R7E	138411	MMC-223737
BSP-109	S 6&7, T11N, R7E	138362	MMC-223688	BSP-159	S 6, T11N, R7E	138412	MMC-223738
BSP-110	S 7, T11N, R7E	138363	MMC-223689	BSP-160	S 6, T11N, R7E	138413	MMC-223739
BSP-111	S 6&7, T11N, R7E	138364	MMC-223690	BSP-161	S 6, T11N, R7E	138414	MMC-223740
BSP-112	S 7, T11N, R7E	138365	MMC-223691	BSP-162	S 6, T11N, R7E	138415	MMC-223741
BSP-113	S 6&7, T11N, R7E	138366	MMC-223692	BSP-163	S 6, T11N, R7E	138416	MMC-223742

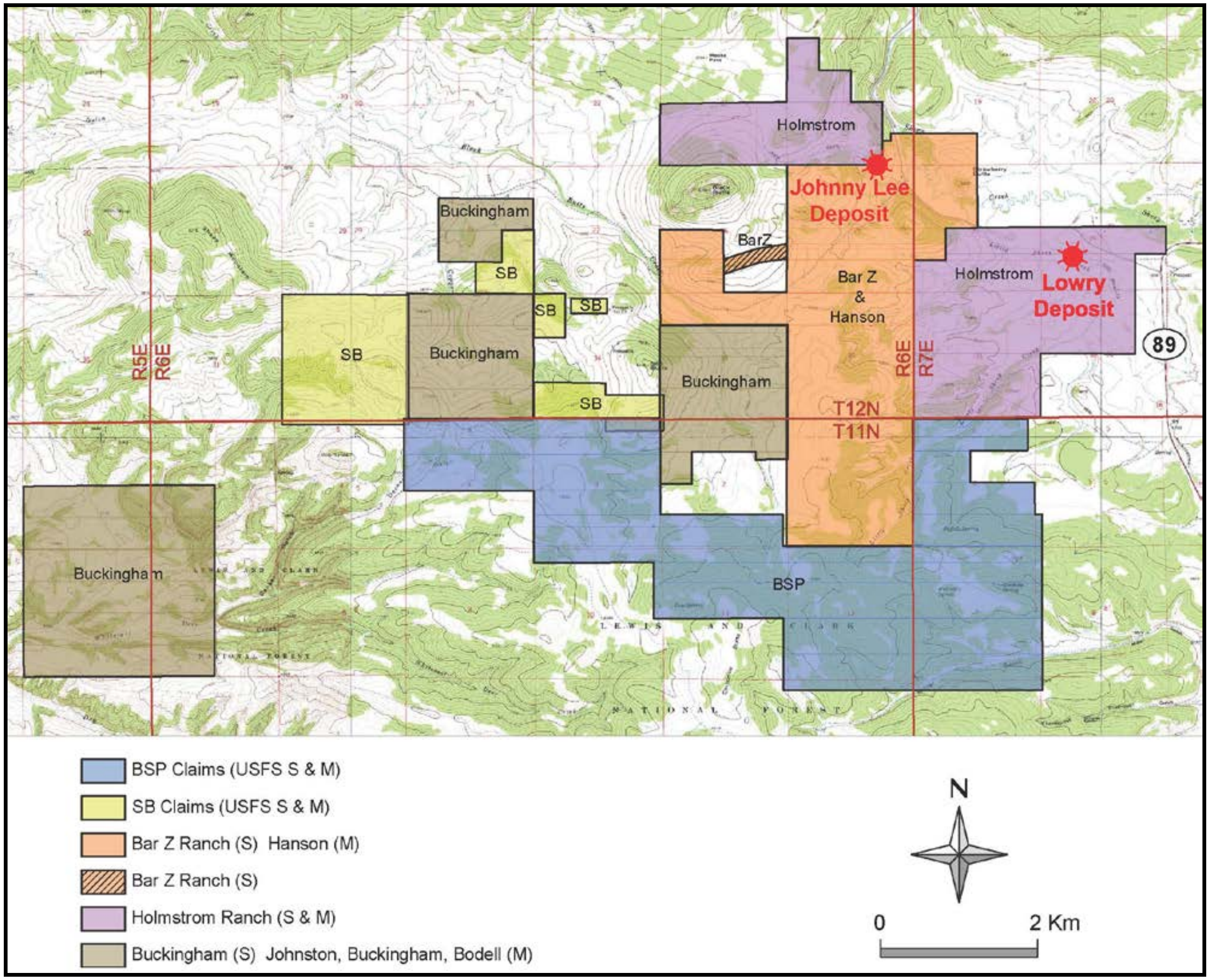
table continues...

Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
BSP-114	S 7, T11N, R7E	138367	MMC-223693	BSP-164	S 6, T11N, R7E	138417	MMC-223743
BSP-115	S 6&7, T11N, R7E	138368	MMC-223694	BSP-165	S 6, T11N, R7E	138418	MMC-223744
BSP-116	S 7&8, T11N, R7E	138369	MMC-223695	BSP-166	S 6, T11N, R7E	138419	MMC-223745
BSP-117	S 5,6,7&8, T11N, R7E	138370	MMC-223696	BSP-167	S 6, T11N, R7E	138420	MMC-223746
BSP-118	S 11,12,13&14, T11N, R6E	138371	MMC-223697	BSP-168	S 6, T11N, R7E	138421	MMC-223747
BSP-119	S 11&12, T11N, R6E	138372	MMC-223698	BSP-169	S 6, T11N, R7E	138422	MMC-223748
BSP-120	S 12&13, T11N, R6E	138373	MMC-223699	BSP-170	S 5&6, T11N, R7E	138423	MMC-223749
BSP-121	S 12, T11N, R6E	138374	MMC-223700	BSP-171	S 6, T11N, R7E	138424	MMC-223750
BSP-122	S 12&13, T11N, R6E	138375	MMC-223701	BSP-172	S 1&6, T11N, R7E	138425	MMC-223751
BSP-123	S 12, T11N, R6E	138376	MMC-223702	BSP-173	S 1&6, T11N, R7E	138426	MMC-223752
BSP-124	S 12&13, T11N, R6E	138377	MMC-223703	BSP-174	S 6, T11N, R7E	138427	MMC-223753
BSP-125	S 12, T11N, R6E	138378	MMC-223704	BSP-175	S 6, T11N, R7E	138428	MMC-223754
BSP-126	S 12&13, T11N, R6E	138379	MMC-223705	BSP-176	S 6, T11N, R7E	138429	MMC-223755
BSP-127	S 12, T11N, R6E	138380	MMC-223706	BSP-177	S 6, T11N, R7E	138430	MMC-223756
BSP-128	S 12&13, T11N, R6E	138381	MMC-223707	BSP-178	S 6, T11N, R7E	138431	MMC-223757
BSP-129	S 12, T11N, R6E	138382	MMC-223708	BSP-179	S 6, T11N, R7E	138432	MMC-223758
BSP-130	S 12&13, T11N, R6E	138383	MMC-223709	BSP-180	S 6, T11N, R7E	138433	MMC-223759
BSP-131	S 12, T11N, R6E	138384	MMC-223710	BSP-181	S 6, T11N, R7E	138434	MMC-223760
BSP-132	S 12&13, T11N, R6E	138385	MMC-223711	BSP-182	S 6, T11N, R7E	138435	MMC-223761
BSP-133	S 12, T11N, R6E	138386	MMC-223712	BSP-183	S 6, T11N, R7E	138436	MMC-223762
BSP-134	S 12&13, T11N, R6E	138387	MMC-223713				
BSP-135	S 7, T11N, R7E	138388	MMC-223714				
BSP-136	S 7&18, T11N, R7E	138389	MMC-223715				
BSP-137	S 7, T11N, R7E	138390	MMC-223716				
BSP-138	S 7&18, T11N, R7E	138391	MMC-223717				

table continues...

Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
BSP-139	S 7, T11N, R7E	138392	MMC-223718				
BSP-140	S 7&18, T11N, R7E	138393	MMC-223719				
BSP-141	S 7, T11N, R7E	138394	MMC-223720				
BSP-142	S 7&18, T11N, R7E	138395	MMC-223721				
BSP-143	S 7, T11N, R7E	138396	MMC-223722				
BSP-144	S 7&18, T11N, R7E	138397	MMC-223723				
BSP-145	S 7, T11N, R7E	138398	MMC-223724				
BSP-146	S 7&18, T11N, R7E	138399	MMC-223725				
BSP-147	S 7, T11N, R7E	138400	MMC-223726				
BSP-148	S 7&18, T11N, R7E	138401	MMC-223727				
BSP-149	S 7, T11N, R7E	138402	MMC-223728				
BSP-150	S 7&18, T11N, R7E	138403	MMC-223729				

Figure 4.2 Tintina Land Position



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Elevations in the resource area range from 1,700 to 1,850 masl and the topography consists of gently rolling hills and valleys. Timber cover consists of primarily Douglas fir on north-facing slopes, grass and mountain sagebrush cover on valley floors and draws, and open to partly timbered ridge tops. Timber covers approximately 10% of the resource area.

The Property can be accessed by 1.5 miles of gravelled maintained fair-weather county road that branches from US Highway 89, an all-weather state-maintained highway. US Highway 89 connects the Property area with White Sulphur Springs, Montana, which lies 17 miles south of the Project and has a population of approximately 984. This is the county seat of Meagher County that includes the Project area and has a population of 1,908. Along US Highway 89 north of the Project area in neighbouring Cascade County are the communities of Belt, which has a population of 633 and lies 80 km from the nearest railhead to the Property, and Great Falls, which has a population of 56,690 and an international airport, and is 132 km from the Property.

Agriculture drives the local economy, and most agricultural operations specialize in cattle ranching with minor grain and hay production. The region has high quality hunting and fishing, and some locals have outfitting businesses for both big game and for fishing, including some that primarily utilize the Sheep Creek drainage. The few small logging operations in the area haul logs to mills outside the valley, often as far as 325 km away. The local sawmill closed 25 years ago after a lifespan of about 30 years.

The climate is typical of uplands in central Montana with moderate summers and cold winters. The average daily minimum and maximum temperatures for White Sulphur Springs (elevation 1,609 masl) are -12 to 0°C in January, -2 to 12°C in April, 8 to 27°C in July; and -1 to 14°C in October. Temperature extremes can reach below -50°C in winter and more than 38°C in summer. The average annual precipitation at White Sulphur Springs is approximately 335 mm. The Property lies between an elevation of 1,700 and 1,850 masl, and is located in the Little Belt Mountains, resulting in cooler temperatures and higher precipitation than those recorded at White Sulphur Springs. In spite of the severe winter conditions, the proximity to the highway and the well-kept branch roads make it possible to carry out drilling programs on the Property throughout the winter months.

Power is available from the local grid, and a 100 kV power line passes across US Highway 89, 16.9 km straight line distance north of the Project (by road) and 21.7 km by highway.

Electrical power lines of a scale appropriate for domestic use service the ranch buildings on the Property.

Water rights for surface water are held by the lessors of the mining leases and this water is available for Tintina's use. Groundwater is abundant, resulting in some artesian flows in the Sheep Creek valley.

The leased property has ample room outside the Sheep Creek valley bottom for a processing plant, mine waste, and tailings, well away from active waterways.

The small population of the local community requires that skilled mining personnel must come from other areas. Because a number of underground mining operations are active within the Montana and Idaho region, some skilled miners will likely be available.

6.0 HISTORY

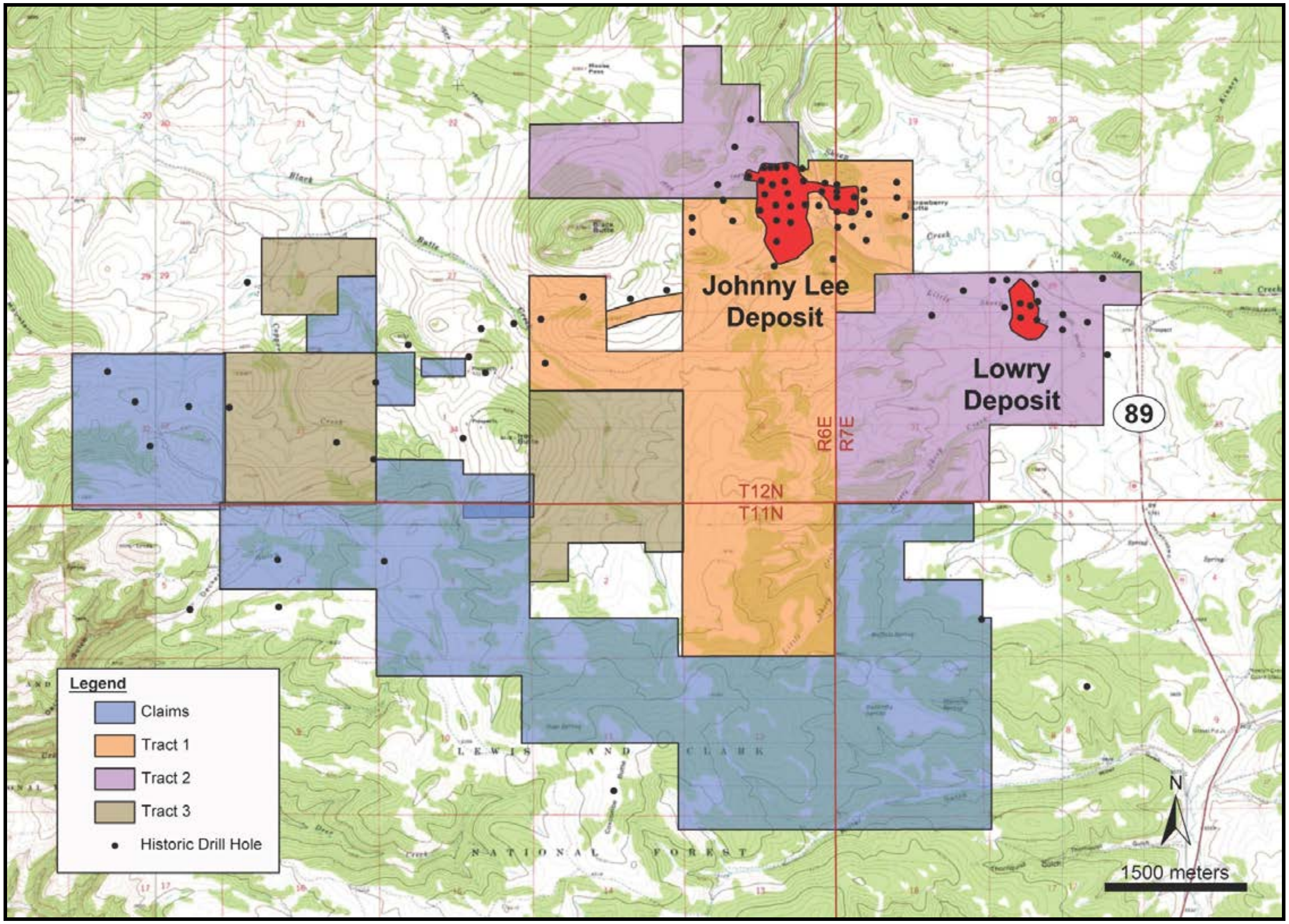
Tract 1 (Hanson/Bar Z lease), Tract 2 (Holmstrom lease), and Tract 3 (Buckingham lease, Section 4.0) are ranch properties that were initially homestead properties and railroad lands, consolidated over time into the fee simple tracts now under lease to Tintina, complete with mineral rights. The same ranching families have controlled each tract throughout all exploration activities on them to date.

Weed (1899) documents the first work on the Property and notes that Messrs. Weir and Tyler, local hay ranchers, located claims on copper-stained quartzite at the Virginia Mine and by 1894 had a 70 ft shaft with a 30 ft drift that exposed only oxidized copper mineralization. This location lies approximately 500 m west of the present resource area and is on the Holmstrom lease. A homesteader, John Lee, settled on the Property (E2SW4 and W2SE4 of section 24, T. 12 N., R. 6 E.) in 1906. In 1910, Mr. Lee sunk a shaft, likely less than 50 ft in depth, on the copper-bearing gossan near the earlier workings and continued to work through at least 1922 (Hanson family, pers. comm.). Based on material on the dump, Mr. Lee's workings only encountered unoxidized copper mineralized material. His cabin and outbuildings lay above the copper resource where Tintina's work is now focused. Tintina has recognized the efforts of Mr. Lee by naming the resources below his workings, the "Johnny Lee Deposit".

During the first half of the twentieth century, interest focused on extensive gossans developed on Iron Butte area between Butte Creek and Sheep Creek (Goodspeed 1945; Roby 1950). This work resulted in the surveying and patenting of a number of patented claims, both inside and adjacent to the Hanson/Bar Z Ranch lease. Work focused on iron potential in the area. The few prospect pits and small adits that were excavated did not penetrate the redox boundary into sulphide-bearing rock.

Cominco American Inc. (CAI) carried out the first modern exploration work on the Property. CAI leased Tract 1 and Tract 3 (Section 4.0) in 1977. Exxon Minerals leased Tract 2 in 1981, and joint ventured it to CAI in late 1984, after which CAI purchased Exxon's remaining interest. CAI joint ventured the entire Property to Utah International Inc. (UII) in 1985, and then UII was subsequently taken over by BHP Billiton Ltd. (BHP). UII/BHP operated the joint venture through early 1988 and earned their 50% interest, at which time operatorship reverted back to CAI. Within the next two years, CAI purchased BHP's interest in the Property and regained 100% control with no retained royalties or back-in rights. CAI dropped the leases in the mid-1990s. No other companies have worked on either tract. CAI drilled their first two holes in 1977 and 1978 on Tract 3, then CAI/BHP drilled a third hole in 1987, and CAI completed a fourth hole on this lease in 1990. Figure 6.1 shows the location of historical drilling with respect to the Johnny Lee and Lowry deposit outlines along the colour-coded Tintina land holdings.

Figure 6.1 Historical Drillhole Locations and Tintina Land Holdings



CAI completed the first hole in the resource area, DDH SC-8, on Tract 1 in 1981. DDH SCC-17, the first drillhole to encounter significant copper, was drilled on Tract 2 in 1985. This was the second hole drilled during the CAI/UII joint venture and it encountered 6.7 m with 2.8% copper and 0.19% cobalt in an Upper Sulphide Zone (USZ) (now called the Johnny Lee UZ) and 4.3 m of 4.1% copper in a Lower Sulphide Zone (LSZ) (now called the Johnny Lee LZ). Following this, an intensive drilling program over the next four years further outlined the two shallowly dipping stratabound massive sulphide zones in what is now called the Johnny Lee deposit. In total, 38 diamond drillholes were completed on Tract 1 by CAI and the CAI/BHP joint venture between 1981 and 1991. Twenty-eight diamond drillholes and one rotary hole were completed on Tract 2 by the same parties between 1985 and 1991.

Within CAI's USZ are UZ #1 and UZ #2. In the north end of the USZ resource, UZ #1 is coincident with the USZ. Further south, UZ #1 lay at or near the base of the much thicker USZ, and UZ #2 lay separated from and above UZ #1 but still within the USZ. UZ #2 has a more limited areal extent than UZ #1. UZ #3 was also encountered and had more limited areal extent than UZ #2. Tintina and RMI have subsequently renamed UZ #1 and UZ #2 mineralized lenses as UCZ 31 and UCZ 32, respectively.

CAI estimated a USZ resource of 4.5 Mt grading 2.5% copper and 0.12 % cobalt (CAI 1996). This resource is not compliant with NI 43-101 standards as NI 43-101 was not in effect at the time the CAI estimate was completed. RMI is not able to comment on the relevance and reliability of the historical estimate due to the fact that many of the key assumptions, parameters, and methods used to prepare the historical estimate were not disclosed and are no longer available. The historical CAI estimate categorized the resource as a "drill indicated possible resource". This is not a category that is recognized by NI 43-101 Standards of Disclosure for Mineral Projects (i.e. Section 1.2 of NI 43-101). RMI is not able to comment on the "resource" category that was mentioned in the historical estimate other than to say that it does not conform to current Canadian Institute of Mining (CIM) definitions. RMI has not done sufficient work to classify the CAI historical estimate as current mineral resources and Tintina is not treating the historical estimate as current resources. RMI is unaware of any other more recent estimates or other data regarding resource estimates.

CAI also completed an estimate of resources for the LSZ prior to the enactment of NI 43-101; therefore, the estimates are not compliant with NI 43-101 standards. RMI is not able to comment on the relevance and reliability of this historical estimate due to the fact that many of the key assumptions, parameters, and methods used to prepare the historical estimate were not disclosed and are no longer available. The historical CAI estimate categorized the resource as a "drill indicated possible resource". This is not a category that is recognized by NI 43-101 Standards of Disclosure for Mineral Projects (i.e. Section 1.2 of NI 43-101). RMI is not able to comment on the "resource" category mentioned in the historical estimate, other than to say that it does not conform to current CIM definitions. RMI has not done sufficient work to classify the CAI historical estimate as current mineral resources and Tintina is not treating the historical estimate as current resources. RMI is unaware of any other more recent estimates or other data regarding resource estimates.

Exploration drilling also located several other bodies of copper mineralization on Tracts 1, 2, and 3. In Tract 1, approximately 750 m west of the Johnny Lee deposit, two holes drilled by CAI in 1989 encountered a copper zone at 453 m deep within the massive sulphide of the USZ. Additional sulphide zones were found at shallower depths at this location, but contained no appreciable copper. Three miles further west, Tract 3 shows extensive copper-mineralized gossans on surface in outcrop and high copper in soil geochemistry, but the few drillholes completed so far were some distance from this and encountered little copper mineralization.

Drilling by CAI on Tract 2 in 1989, 1,500 m east and 600 m south of the east end of the Johnny Lee deposit, also encountered multiple zones of copper-rich massive sulphide now called the Lowry deposit. In this drilling, deepening of an older hole, SCC-39 encountered LSZ intersections with 4.9 m of 4.5% copper and 0.14% cobalt. At 150 m west, DDH SC-75 encountered an LSZ with 7.3 m of 2.7% copper and 0.06% cobalt. Additional drilling by CAI encountered more LSZ intersections further south with depths ranging from 475 to 650 m. Drilling in this area also encountered an USZ with little copper mineralization and, below it, a Middle Sulphide Zone (MSZ) (located between the USZ and LSZ), which included as much as 52.1 m of 2.7% copper and 0.11% cobalt from 391.4 to 443.5 m in DDH SC-80. Drilling also encountered significant mineralization in two stratigraphically higher zones, the 0/I zone (12.2 m at 1.7% copper and 0.1% cobalt in DDH SC-74), and the Ynu II zone (2.7 m at 1.9% copper in DDH SC-82).

CAI carried out no further drilling on Tract 1, Tract 2, or Tract 3 after completion of DDH SC-89 in the spring of 1991. CAI dropped the Property in the mid-1990s and donated all of the drill core and a basic dataset to the University of Montana. CAI did not begin any engineering or baseline environmental work but they did complete preliminary resource calculations (previously mentioned) and completed initial metallurgical testing. These reports are proprietary and are not available to Tintina. There was no further work on Tract 1, Tract 2, or Tract 3 since CAI gave up their lease, and there has been no development activity or mineral production from the Property. There is no recorded production from any of the Black Butte copper occurrences.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 GEOLOGICAL SETTING

The copper-cobalt deposits of the Project occur in middle Proterozoic sediments of the Belt Supergroup that are extensively exposed in an eastward protrusion of the Rocky Mountain chain referred to as the Helena Salient of central Montana (Zieg and Leitch 1993). During formation of the Belt Basin, a deep water middle Proterozoic calcareous shale facies (Newland Formation) was deposited in an embayment, known as the Helena Embayment, which extended in trough-like fashion east into the craton through central Montana (Godlewski and Zieg 1984). The northern boundary of the deeper water portion of the Helena Embayment is approximated by the southern flank of the Little Belt Mountains north of White Sulphur Springs, Montana. This east-west trending structural zone (Figure 7.1) approximates the northern edge of most Newland Formation exposures and has been called the Lewis and Clark lineament by some (Reynolds 1977; Sears and Hendrix 2004) due to nearly continual Cretaceous faulting along the feature, and the Garnet line by others (Winston 1986; Sears and Hendrix 2004) based on an abrupt change in Precambrian erosion levels prior to deposition of the middle Cambrian Flathead sandstone along an east-west lineament. These features both extend easterly from the main Belt basin in the west and follow the south flank of the Little Belt Mountains to mark the approximate boundary between a northern area that preserves only the older shallow water Belt sediments (Neihart and Chamberlain Formations and lowermost Newland Formation of the Lower Belt; Keefer 1972; Godlewski and Zieg 1984; Feeback 1997) and the Helena Embayment on the south in which a portion of the Belt Supergroup section ranging from Neihart Quartzite through lower Piegan Group is preserved (Godlewski and Zieg 1984). The Black Butte area rests on the intersection of this northern embayment margin and the northeast trending Great Falls Tectonic Zone (GFTZ) (O'Neill and Lopez 1985). As outlined by O'Neill (1999), Mueller et al. (2002), and Harms et al. (2004), the 200 km-wide GFTZ is spatially coincident with the Big Sky Orogen of early Proterozoic time. In the Black Butte area, northeast trending faults within and parallel to the GFTZ:

- show some influence on Newland sedimentation and mineralization patterns
- interrupt and influence the pattern of Laramide compressional faulting
- control distribution of some Eocene intrusive rocks in Belt shale and in Paleozoic cover rocks.

Northwest trending faults also focus some Eocene intrusive rocks in both Belt shale and Paleozoic rocks. Klein and Sims (2007) describe the importance of the GFTZ as a structural control for Cretaceous and Eocene crustal metal enrichment.

During the Cretaceous Laramide orogeny, faulting sub-parallel to the ancestral northern margin of the Helena Embayment formed the Volcano Valley thrust fault (Winston 1986), which is a prominent structural feature in the Black Butte area. The bedded massive sulphides of Black Butte are concentrated along the northern margin of the Helena Embayment sub-parallel to the VVF zone. Figure 7.1 is a generalized geologic map showing a portion of the Helena Embayment.

The lowest unit of the Belt Supergroup in the Black Butte area is the Neihart Quartzite, which measures approximately 240 m thick at its type location 20 km northeast of the Property (Weed 1900; Keefer 1972) and is present in exposures and drillholes within the Property area. This unit rests unconformably on early Proterozoic granitic gneiss and amphibolite. Above the Neihart Quartzite is the Chamberlain Shale, a shallow water silty carbonaceous shale measuring approximately 180 m thick on the Property. The Newland Shale hosts the Black Butte massive sulphides, and consists of a lower shale-dominated part, which measures approximately 760 m in thickness and an upper carbonate-dominated part that measures approximately 350 m in thickness. The shale is evenly laminated and was deposited as microturbidites in a sub-wavebase depositional setting. Debris flow conglomerates punctuate the section along the northern margin of the embayment. Though, in places, the lower Newland shale shows ubiquitous bedded pyrite, more typically sulphides are found in several discrete stratigraphic horizons of greater lateral extent. The carbonate-rich upper Newland Formation is further divided into seven units (Zieg 1981; 1986), as follows:

- Unit I, a clean limestone or dolomite
- Unit II, a silty calcareous or dolomitic shale similar to lower Newland shale
- Unit III, a clean, black chert-bearing dolomite
- Unit IV, a silty limestone and silty calcareous shale
- Unit V, a non-calcareous silty shale very similar to Greyson Shale
- Unit VI, a clean thin-bedded limestone
- Unit VII, a silty limestone and silty calcareous shale.

The Greyson Shale, a shallow water silty shale measuring approximately 700 m in thickness, overlies the Newland formation and is overlain by the Spokane Shale, a red argillite measuring at least 300 m in thickness. The latter formation is part of the Ravalli Group portion of the Belt Supergroup (Whipple 1980; Connor, et al. 1984), while the Neihart, Chamberlain, Newland, and Greyson represent the Lower belt portion of the Belt Supergroup in this portion of the Helena Embayment.

The Belt Supergroup rocks in this area have virtually no metamorphic grade (Maxwell and Hower 1967). The Belt stratigraphic section in the Helena Embayment is significantly thinner than in the main Belt Basin where rocks are metamorphosed to greenschist facies. At Black Butte, very delicate sedimentary structures and other early fabrics are very well preserved. Structural modification of the geology began with synsedimentary faulting along the north margin of the Helena Embayment. During Late Proterozoic time, the area hosted limited mafic magmatism, generally reported at 800 Ma (Reynolds

1984). A fault-bounded chlorite and carbonate altered basalt in a Black Butte drillhole produced a potassium/argon date of 769 Ma (±29 Ma) (Himes and Peterson 1990), and locally may reflect this magmatism. No subsequent deformation or magmatism affected the area until Late Cretaceous regional compression, which did not produce intrusives in the Black Butte area. Biotite-hornblende dacite dikes and sills on the Property produce an Eocene date of 50.1 Ma (potassium/argon date obtained by CAI/BHP JV) and whole rock chemistry from numerous dikes and sills encountered in drilling show alkali to sub-alkalic compositions including syenite, monzonite, and dacite. Oligocene basaltic magmatism produced some basalt flows in the Black Butte area (Reynolds and Brandt 2007). Miocene extensional faulting has modified the landscape and drainage patterns to some extent in the Black Butte area. Figure 7.2 is a district scale geologic map showing surficial geology in addition to drillhole locations.

Figure 7.1 Geologic Setting

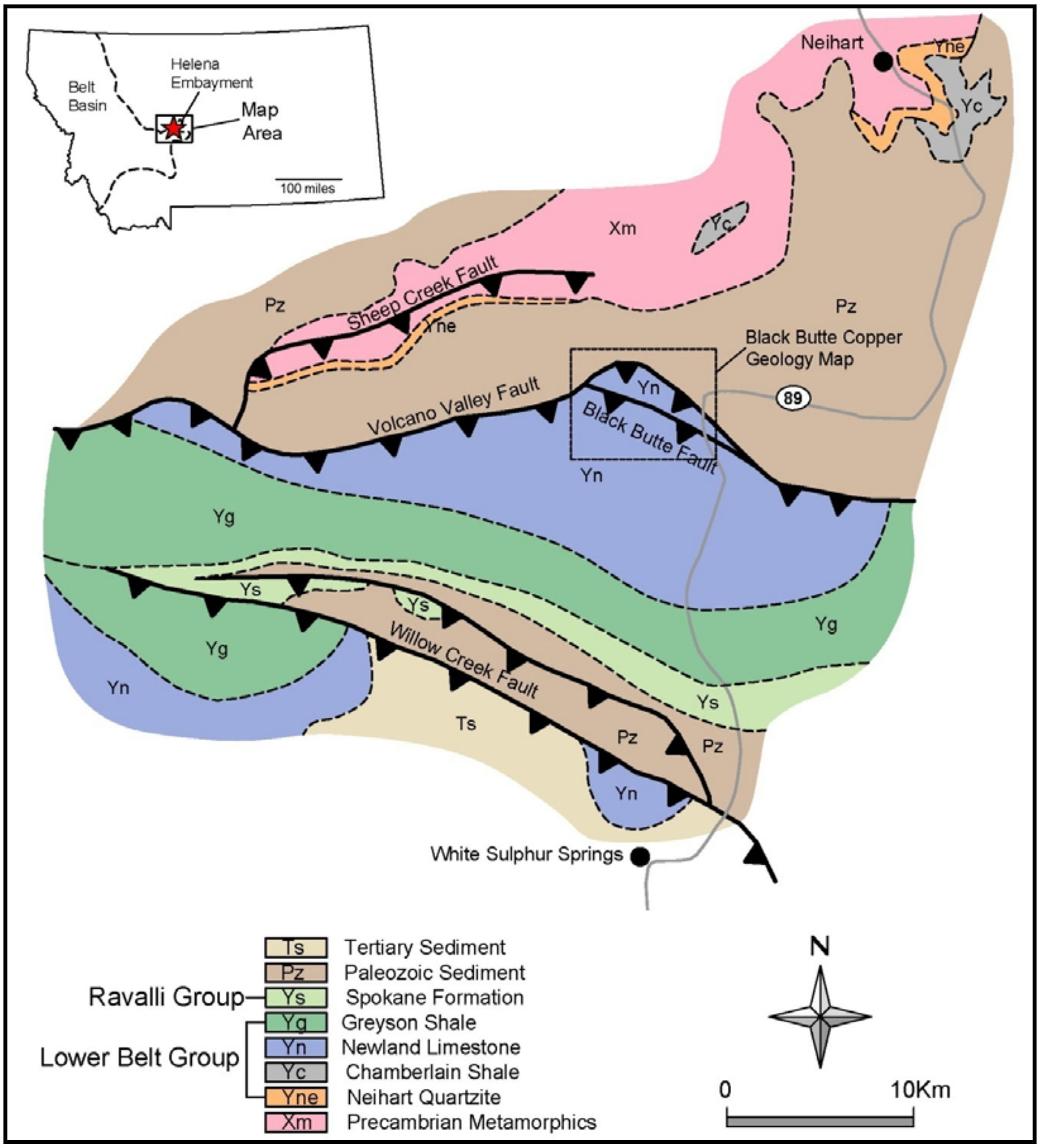
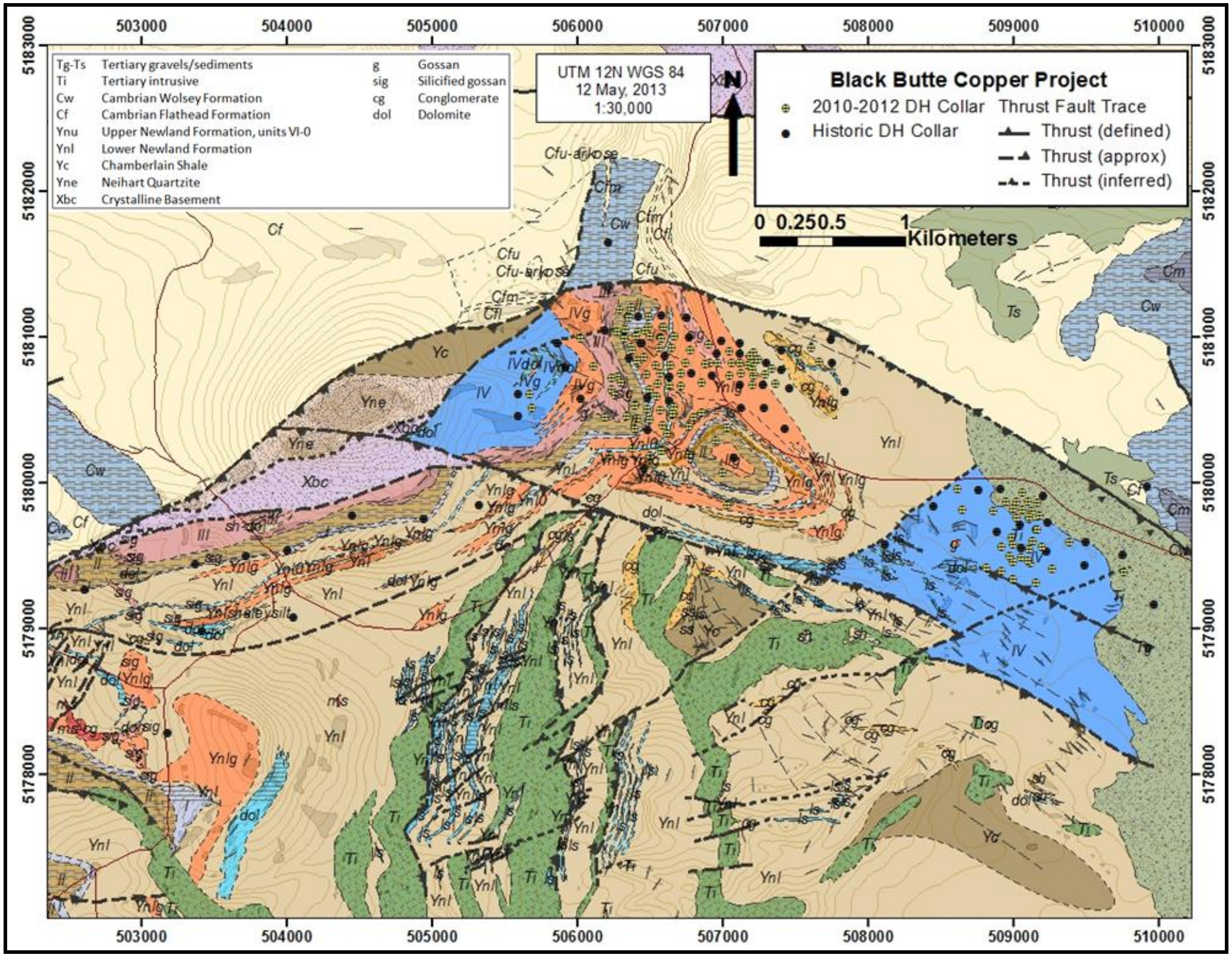


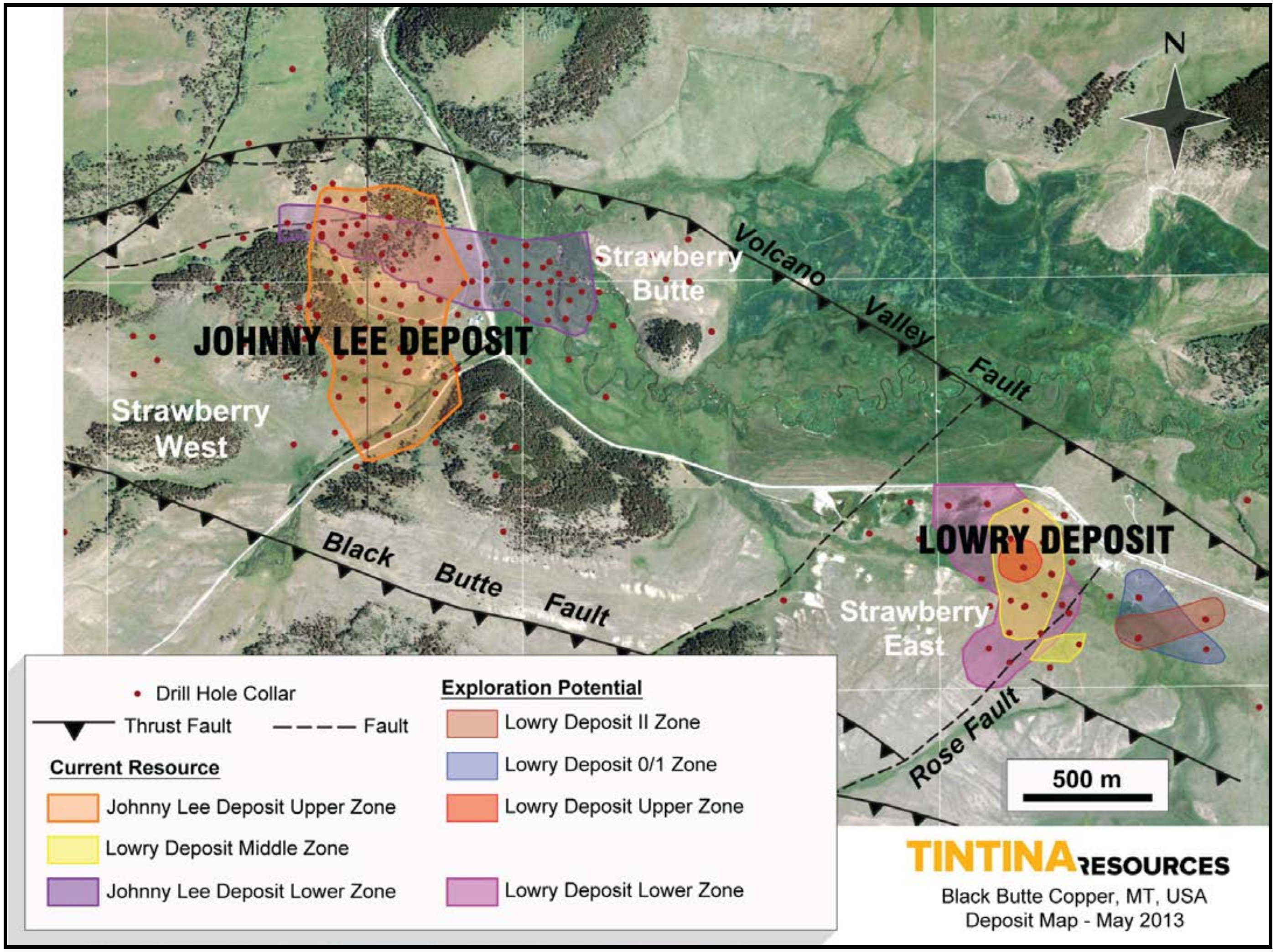
Figure 7.2 District Scale Geologic Map



The VVF is the most prominent structure through the Project area, as it carried Proterozoic sediments and crystalline basement complex over next to Paleozoic sediments to the north side. The VVF regionally cuts through the entire Paleozoic stratigraphic section and is intruded by Eocene dikes and sills, its age is best interpreted as Laramide (late Cretaceous). Drilling at Black Butte shows that the VVF cuts, conceals, and apparently utilized portions of earlier Precambrian normal faults that formed a northern margin to most Newland Formation exposures. The northernmost of these easterly striking structures is known as the Buttress Fault. Allochthonous source-proximal debris is preserved in lower Newland shale along the east-west faults and shows that these Precambrian normal faults, which down-drop Belt stratigraphy on their south sides, approximate the middle Proterozoic northern margin of deeper water sedimentation within the Helena Embayment.

In the Project area, the VVF has ramped up a succession of “stair stepped” fault blocks containing Newland and Chamberlain shale as shown in Figure 7.3. West of Butte Creek, the Copper Creek segment of the VVF shows an orientation of roughly N80°E. At Butte Creek, a N50°E striking structure offsets the VVF with apparent sinistral displacement approximately 1 km. From this point, the Black Butte segment of the VVF strikes east for approximately 2 km and gradually arcs toward the southeast for 7 km at a strike of S45°E toward Newlan Creek. The Newlan Creek segment of the VVF continues with an easterly strike for at least 16 km. The flexures in the VVF at Butte Creek and at Newlan Creek are joined by a S65°E striking northeast verging thrust fault called the Black Butte Fault (BBF) that carries Chamberlain shale over Newland Shale. The area between the Black Butte segment of the VVF and the BBF contains all known copper resource at the Project, and exhibits exposures of Newland formation ranging from the middle part of the lower Newland to Unit IV of the upper Newland. Figure 7.3 is an aerial photograph showing the trace of various structures that have been identified at the Property along with the mineralized zones and potential exploration targets.

Figure 7.3 Structural Setting



In and around the Property area, bedded pyrite in the Newland shale laterally persists across at least 25 km of strike length, and in some places occurs intermittently through over 900 m of Newland Formation stratigraphy. Drilling has shown as much as 25% pyrite across 700 m of stratigraphy, and locally massive pyrite (more than 50%) can extend through as much as 250 m of stratigraphy. The lower Newland shale contains three discrete pyrite zones with important concentrations of copper and cobalt. The lowermost zone, dubbed the Lower Sulphide Zone (LSZ), lies just above the lower contact of the Newland Formation with the underlying Chamberlain Formation. In most areas, debris flow conglomerates occur at this contact. The LSZ is mainly known from drill testing in the Strawberry Butte and Strawberry East areas.

The next stratigraphically higher sulphide zone is the MSZ, a complex and very thick concentration of bedded and replacive pyrite that can occupy over 100 m of lower Newland stratigraphy and can have important concentrations of copper, cobalt, and zinc-lead. Higher in the stratigraphy, from 30 to 60 m below the top of the lower Newland shale, is the USZ. The USZ is widely exposed in the region and drilling along the 25 km of strike length across the district shows variable thicknesses of laminated, thin bedded, and massive thicker-bedded pyrite with variable concentrations of zinc, lead, barium, and copper. These three zones are described in more detail in Section 9.0.

Pyrite zones also occur between the USZ and Upper Newland Unit I (the sub-0 sulphide zone and the 0/I sulphide zone) and in Upper Newland Unit II (the Ynu II sulphide zone) and can locally contain significant copper mineralization. Additional pyrite zones without appreciable copper or zinc concentrations occur in Upper Newland Units III and IV.

Most geologists interpret the genesis of the Black Butte sulphides as having formed at syn-sedimentary hydrothermal vents sites during deposition of the host shale. Sulphides are involved in soft sediment folding, and sulphide accumulations include abundant evidence of vent biota grown over subaqueous hydrothermal hot springs. These include microbial mat fabrics (Rhodes 2011 in-house report) and intricate growths of tubes are interpreted as having formed around algal or bacterial filaments (McGoldrick and Zieg 2004). Secondary pyrite, silicification, dolomitization, barite, and chalcopyrite replaced earlier pyritic “muds” deposited near and adjacent to vent sites. Lead isotope ratios obtained by CAI from USZ galena samples are consistent with a middle Proterozoic age for Black Butte mineralization (Zieg and Leitch 1993). Geologic modelling of the Johnny Lee UZ in 2011 recognized that the lower contact of an overlying debris flow cut through the west margin of the sulphide lens, suggesting a submarine slope failure carried a portion of the deposit away. Unit 0 carbonate persists through this area uninterrupted, so the event must have preceded its deposition.

Figure 7.4 is plan map of a portion of the Johnny Lee deposit showing the surface projection traces of the UZ and LZ. Lines of section for the three geologic cross sections are shown in Figure 7.5 through Figure 7.7.

Figure 7.4 Plan Map Showing Lines of Section

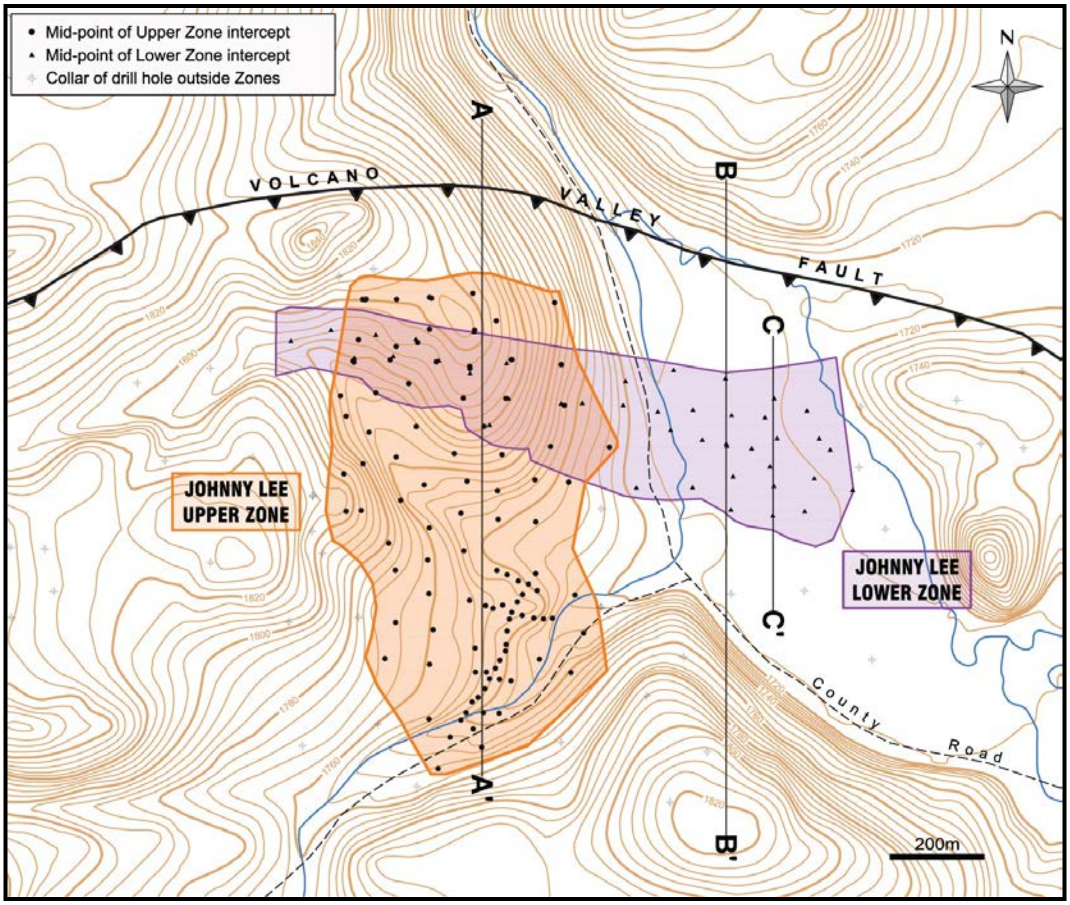


Figure 7.5 UZ Cross Section A-A'

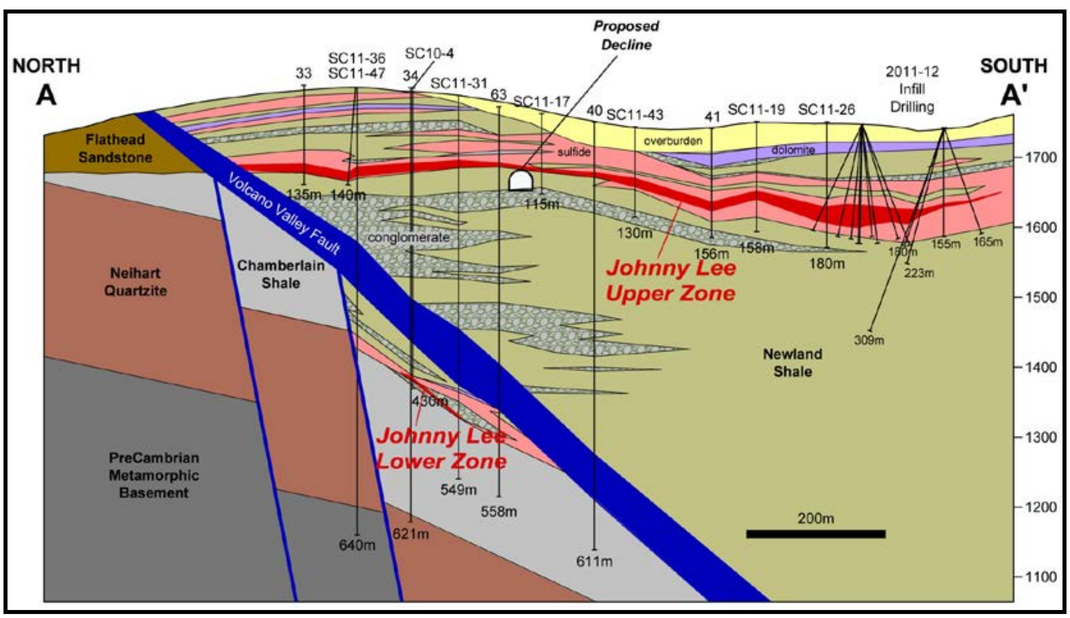


Figure 7.6 LZ Cross Section B-B'

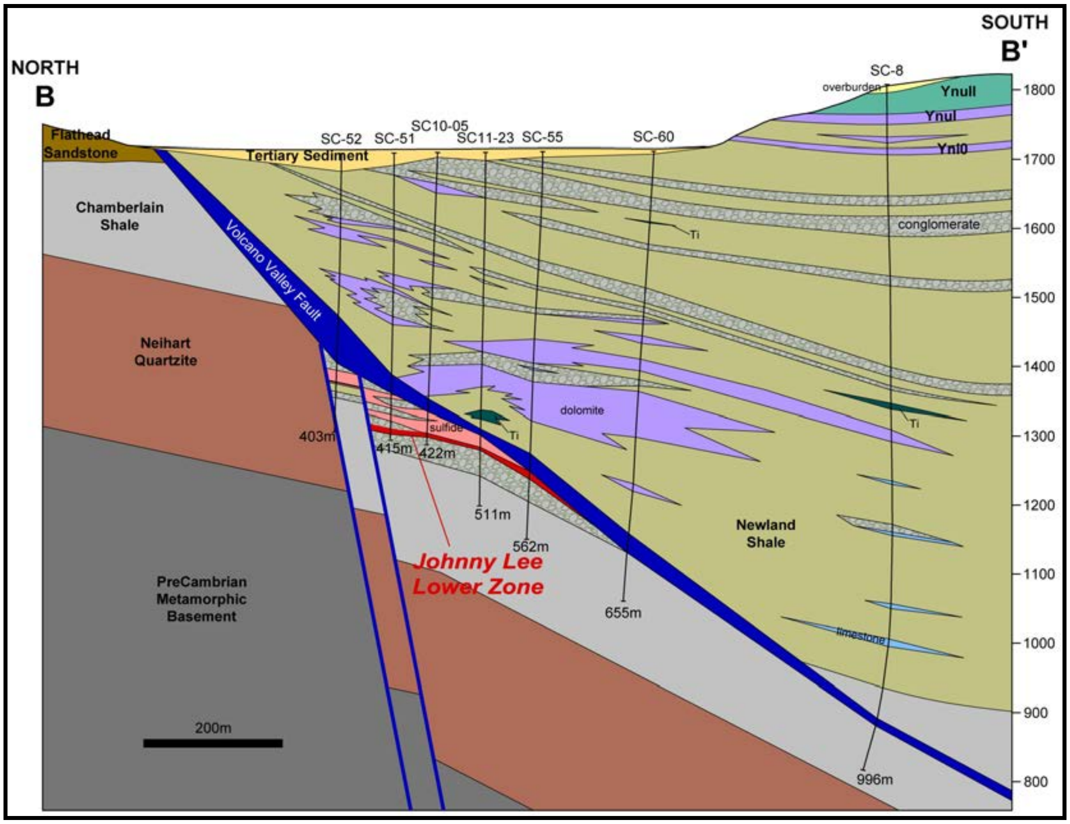
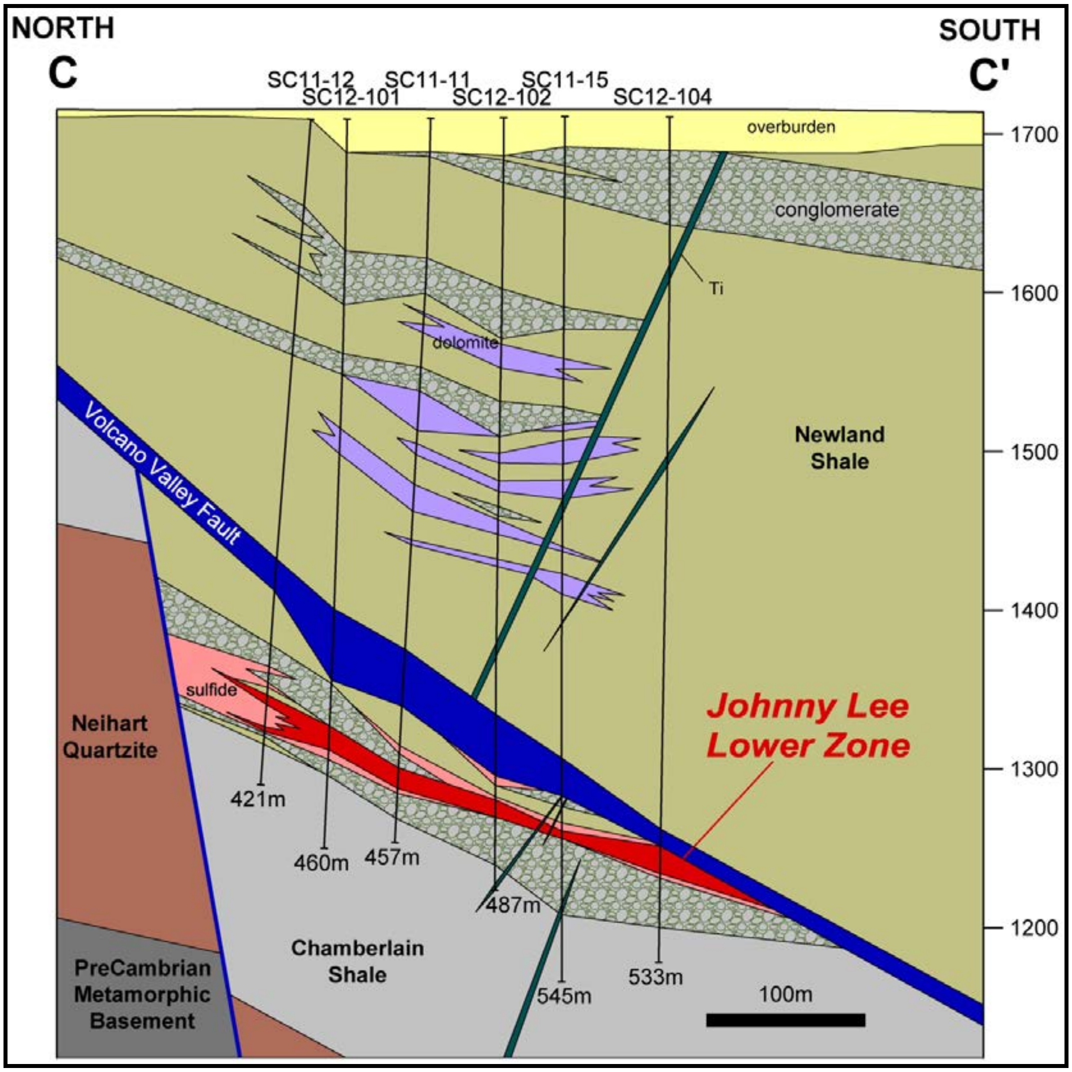


Figure 7.7 LZ Cross Section C-C’



7.2 MINERALIZATION

7.2.1 UPPER SULPHIDE ZONE

The USZ (Zieg, et al. 1991) stratigraphy lies 60 to 90 m below the contact of upper and lower Newland and is hosted in lower Newland calcareous to dolomitic shale. In the Black Butte area north of the BBF, four separate lenses of USZ occur along this stratigraphic horizon and are separated by conglomerate lenses or northeast striking, down to the southeast normal faults. Only the lens of USZ containing the Johnny Lee deposit, called the Johnny Lee UZ, contains enough drillhole information to allow detailed definition of its geometry and compositional character. The mineralogical and textural attributes of the Johnny Lee UZ are typical of the USZ throughout the Black Butte area. The footwall of the Johnny Lee UZ consists of Newland shale with abundant shale-clast conglomerate layers. The footwall to the Johnny Lee UZ in this area contains no alteration or feeder structures. Edges of the sulphide sheet are the VVF on the north, an erosional surface on the east, a conglomeratic lens on the west, and an erosional surface cutting the weakly mineralized sulphide sheet on the south.

The Johnny Lee UZ consists of a lens of fine grained bedded pyrite up to 55 m thick containing up to three chalcopyrite-bearing horizons. Microscopic textures and species of sulphide minerals, primarily from copper-enriched horizons, have been well described by Himes and Petersen (1990) and by Graham et al. (2012). Pyrite bodies occur as laminations and beds of very fine grained pyrite, commonly microcrystals and spheroidal aggregates from 1 to 25+ µm in diameter. Rhodes (2011 in-house report) interprets crinkly thin laminations within pyrite beds as microbial mat textures. Colonies of microbial filaments are preserved as sulphide-replaced intertwined tubular structures now filled with gangue material and additional sulphide. Some coarse euhedral pyrite clearly grew much later than fine grained pyrite. Pyrite and sometimes marcasite aggregates contain rims, patches, and sometimes cores of chalcopyrite and tennantite, commonly with amorphous copper-cobalt-nickel-arsenic-rich material. Pyrite rims contain elevated copper, nickel, arsenic, and cobalt. Chalcopyrite occurs as coarser grained veinlets and clots, in bedding parallel layers and bands, in quartz veinlets, and in barite veins and masses.

In parts of the Johnny Lee UZ, copper zones can contain bornite as well as chalcopyrite. Cobalt minerals, initially recognized by Tintina crews, were identified by Dr. Chris Gammons and his student Josh White at Montana Tech of the University of Montana, as alloclasite inclusions in fine grained pyrite, and siegenite recrystallized during a later stage of mineralization (White 2012, M.Sc. thesis). Separate work by G & L Laboratories during mineralogic identification for metallurgical purposes also identified carrolite. Coarse-grained barite is both intergrown with and crosscuts pyrite. Gangue mineralogy in the Johnny Lee UZ is usually barite but can be dolomite, fine grained quartz, or locally strontium carbonates and sulphates.

Copper-enriched horizons, informally called (in ascending order) UZ #1, UZ #2, and UZ #3, appear stratiform. These copper-rich horizons are collectively known as the Johnny Lee UZ. UZ #1 (3D block model code 31) at the base of the Johnny Lee UZ pyrite sheet typically shows highest grades. UZ #1 extends laterally at least 1,050 m in a north-south direction and 540 m in an east-west direction, ranges from 1 to 29 m thick, and averages 8 m thick. Hanging wall to footwall intersections within UZ #1 reach up to 3.6% copper and 0.54% cobalt although individual assayed intervals often exceed 10% copper. This zone lies approximately 107 m below the surface and outcrops along its eastern margin. Drill results from 2011 show what appears to be a thick mound along the southern end of UZ #1 that approaches 30 m in thickness. UZ #3 (drillhole code 33) appears locally in the northern area but UZ #2 (block model code 32) is absent. Southward, where the USZ thickens to as much as 55 m, UZ # 1 rises from the base of the pyrite sheet, and UZ #2 appears higher within the pyrite sheet. Further south, UZ #1 and UZ #2 coalesce and UZ #3 re-appears. Further to the south, the copper content of each of these horizons decreases to uneconomic concentrations.

UZ #1 and UZ #2 have been modelled as 3D wireframes using a nominal 1% copper cut-off grade. Within the wireframes the UZ #1 lens contains about 12.9 Mt at an average grade of 2.8% copper and the smaller, stratigraphically higher UZ #2 lens contains about 3.2 Mt at an average grade of 2.1% copper. These tonnage/grade quantities were tabulated at a zero cut-off grade.

Zoning patterns show higher concentrations of barium and silver on its southern margin and further south, a drop in copper, cobalt, barium, and silver, and a rise in lead and zinc. Bulk zinc values within copper mineralized areas of the Johnny Lee UZ appear lower than in barren Newland shale, but lead values are approximately ten times shale background. While local silicification occurs within the copper-mineralized stratigraphy, most of the copper-cobalt mineralization occurs within unsilicified bedded pyrite. Better copper grades are associated with barite-chalcopyrite veins and masses of intergrown pyrite, barite, and chalcopyrite that replace and crosscut the bedded sulphide. The upper part of the Johnny Lee UZ shows high concentrations of coarse grained barite. Strontium minerals, apparently both celestite and strontianite, are abundant in some areas of the Johnny Lee UZ and are the subject of ongoing research by Dr. Gammons.

Beyond the conglomerate lens on the west of the Johnny Lee UZ lies an additional mass of USZ with copper-bearing bedded pyrite nearly identical in mineralogical and textural attributes to those of the Johnny Lee UZ. One kilometre southeast of the Johnny Lee UZ is an outcropping lens of USZ that has not been drilled. Approximately 2.5 km to the southeast, the Lowry UZ contains copper-cobalt mineralization but in lower concentrations than those in the Johnny Lee UZ. A typical example of the Johnny Lee UZ copper mineralization is shown in Figure 7.8, which shows fine-grained chalcopyrite mineralization with a copper grade of 1 to 2%.

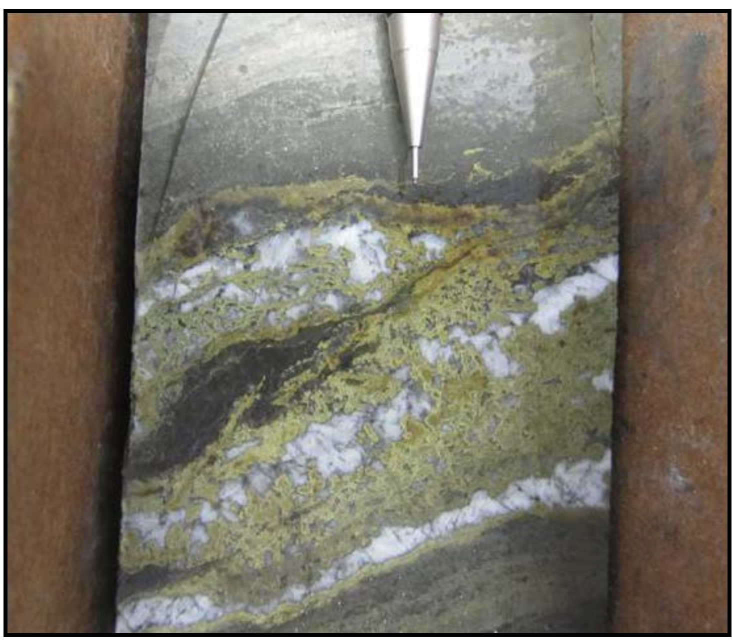
Figure 7.8 Fine-grained Chalcopyrite Mineralization – SC10-002

Higher grade Johnny Lee UZ chalcopyrite mineralization is shown in Figure 7.9, taken from a portion of Tintina drillhole SC10-004. The interval starting at 111.18 m ran about 10.4% copper.

Figure 7.9 Bedded Massive Sulphide Mineralization – SC10-004

Figure 7.10 shows a high-grade interval of chalcopyrite with abundant barite from Tintina core hole SC11-055 which was drilled in 2011. This interval is from the Johnny Lee UZ and runs about 4% copper.

Figure 7.10 Chalcopyrite-Barite Mineralization – SC11-055



7.2.2 MIDDLE SULPHIDE ZONE

East of the Johnny Lee deposit, the MSZ is well-developed in the Lowry deposit area and, where sufficiently copper mineralized, is called the Lowry MZ. Initially called the MSZ for its position between the USZ and LSZ, the Lowry MZ lies within 50 m of the base of the USZ and in places is in contact with it. This complex zone consists of, from bottom to top, barite-rich matrix breccia resting on shale with interbeds of conglomerate; a bedded sulphide zone with abundant biogenic textures; a thick zone of both massive breccia with cobble to boulder sized clasts to interbedded pebble clast breccia and shale with bedded pyrite; and a massive sulphide cap with abundant biogenic textures. The Lowry MZ is typically capped by dolomitic shale and conglomerate but, in places, it is in contact with overlying USZ. Copper-cobalt-silver enrichment occurs throughout the Lowry MZ, but the best mineralization is at the transition from the massive breccia to underlying massive sulphide. The breccia host for the Lowry MZ consists of locally derived lithologies with a wide range of clast sizes, contains vugs filled with dolomite banding, and shows no intervening sedimentary structures that one might expect to find capping a debris flow conglomerate or other transported and re-deposited material. The Lowry MZ breccia appears a result of in situ brecciation by dissolution and collapse of a carbonate-rich protolith. Though now overprinted by intense dolomitization and later silicification, the breccia shows some low aluminium concentrations more typical of Newland carbonate. The upper part of the breccia package shows a strong dolomitization overprint, including breccia vugs filled with banded dolomite, and coarse grained secondary pyrite in the matrix. Below this, strong silicification with chalcopyrite and pyrite overprints the breccia mass, and contains the highest concentrations of copper. Typically, the upper part of the silicified breccia mass is best copper mineralized. The overall sulphide content of the Lowry MZ is quite low compared to the other known mineralization types at Black Butte. The Lowry MZ reaches a maximum thickness of approximately 79 m and forms a lozenge shape with an elongate north-south axis. The Lowry MZ dips gently southwest, though the enclosing strata dip gently southeast. The northeast striking, steeply southeast dipping Rose Fault cuts the Lowry MZ and down-drops the south end of the deposit by approximately 60 m. The Rose Fault and other local faults also host Eocene alkalic and subalkalic intrusive rocks, which are discussed later. Genesis of the Lowry MZ remains poorly understood, but mineralization apparently involved dissolution and collapse of a carbonate-rich part of the lower Newland below the stratigraphic level of the USZ, followed by dolomitization and later silicification with copper mineralization. The proximity of the Lowry MZ and Lowry UZ suggests they may have formed during the same mineralizing event. Zoning patterns show increasing iron content away from a core area of higher copper concentrations and more silicification. In one part of the Lowry deposit, the USZ and the underlying MSZ merge with a resulting true (stratigraphic) thickness of approximately 220 m of more than 50% sulphide. Historic drilling shows as much as 52.1 m grading 2.72% copper and 0.11% cobalt in the Lowry MZ, although the VVF truncates the base of the thickest MSZ occurrences. The Lowry MZ remains incompletely drilled and is open to the south.

Five kilometres west at Butte Creek, the stratigraphy below the USZ includes massive pyrite beds and laminations with local concentrations of chalcopyrite perhaps correlative with MSZ and underlying stratigraphy with bedded pyrite and laminated sphalerite and galena but little copper. Here, a historic hole showed a true (stratigraphic) thickness of

300 m of more than 50% sulphide, again a combination of USZ and MSZ stratigraphy. Work is ongoing to better establish any correlative patterns between the Butte Creek area and the Lowry deposit.

The currently modelled Lowry MZ is intruded by a series of Eocene age dioritic dikes and sills. Historical drilling and wide spaced drilling by Tintina in 2011 encountered both dikes and sills, but geometries and relationships remained poorly understood. Infill drilling by Tintina in 2012 provided sufficient information to model a zone of dikes trending through the modelled MZ wireframe volume. Sills were less commonly encountered and predominantly located near the upper portions of the drillholes, well above the mineralized MSZ, and proximal to the low angle VVF where they likely followed structural zones of weakness. Results from a ground magnetic survey run in late 2012 enable Tintina to correlate intersections of the weakly magnetic diorite in drillholes with magnetic response on surface, aiding in the interpretation. Dike intersection thickness varies from less than one metre to tens of metres thick. In drillholes SC12-156, SC12-165, and SC12-167 multiple dikes were encountered, suggesting a sheeted dike swarm. The dike swarm trends southwest-northeast, striking approximately at N40°E dipping approximately 70° to 75° to the southeast. This dike trend roughly parallels the Rose Fault, located east of the dikes. The Rose Fault zone appears to have been locally intruded by the dikes. In general, the dikes often have very sharp, unaltered contracts with the surrounding host shales. Some of the drillhole intersections show that the dikes are coincident with structural zones and contain a component of clay alteration.

7.2.3 Lower Sulphide Zone

The LSZ lies at or just above the Newland-Chamberlain contact. In the Johnny Lee deposit, the Johnny Lee LZ consists of a stratabound mass of silicification containing coarse-grained pyrite and chalcopyrite that overprints and replaced dolomite, dolomitic shale, black shale, and shale clast conglomerate. The LSZ contains higher copper concentrations and lower cobalt concentrations than the USZ. Chalcopyrite is the only copper mineral identified within the Johnny Lee LZ. Coarse grained chaotic fragmental or crosscutting sulphide textures that may represent sulphide mound construction and collapse dominate the Johnny Lee LZ, through bedded pyrite does occur in and above it. Up to four additional sulphide zones in the hanging wall of the LSZ show more bedded pyrite, some replacive dolomite and pyrite intergrowths, occasional barite, and generally less chalcopyrite. Some silicification occurs in hanging wall sulphide zones.

Alteration associated with Johnny Lee Lower Zone sulphide mineralization includes silicification and dolomitization. The Johnny Lee LZ footwall generally consists of silicified conglomerate, which contains relatively sparse quartz and dolomite veins with chalcopyrite and pyrite. Dolomite alteration, generally as coarse-grained dolomite crystals replacing host rock and crosscutting replacement veins and ribbons, occurs throughout the hanging wall of the Johnny Lee LZ and overprints hanging wall sulphide zones. Dolomite alteration also occurs in the footwall and distal to the LSZ, although distal alteration is weaker and has finer grained dolomite crystals. Associated mineralization commonly contains disseminated chalcopyrite and, in places, sphalerite or galena. In more distal mineralization, barite occurs as pseudomorphs after carbonate.

The LSZ is present in the Lowry deposit LZ, where the footwall is shattered silicified sediment rather than conglomerate as observed in the Johnny Lee LZ. The footwall of the Lowry LZ commonly contains chalcopyrite and pyrite stringers crosscutting this shattered texture. The top of the silicified and chalcopyrite mineralized shattered zone consists of vuggy, coarse massive pyrite, which is chalcopyrite-bearing in places and barren in others. Silicified conglomerate layers lie in the hanging wall of the Lowry LZ and host some chalcopyrite. The Lowry LZ hanging wall also contains zones of silicification, replacive carbonate-sulphide masses, and scattered pyrite beds that contain chalcopyrite mineralization. Distal LSZ mineralization at the Lowry LZ appears similar to distal LSZ mineralization and hanging wall sulphide zones near the Johnny Lee LZ, where it consists of layers of replacive barite or dolomite and pyrite intergrowths, zones of fine grained dolomite, calcite, or barite alteration, and scattered pyrite replacement veins. Distal LSZ mineralization typically lacks silicification. Anhydrite and gypsum veins, so far unique to the Lowry LZ area, cut uppermost Chamberlain shale below the LSZ and its silicified footwall.

A dolomite unit called the Hangingwall Dolomite (HWD) caps the variably altered and mineralized shale and conglomerate that contains the LSZ. The HWD is thin to medium laminated, contains black chert and “Newlandia”-type pressure solution structures and, in places, contains sparse concentrations of sphalerite and galena. While exposures of this stratigraphy south of the BBF show no carbonate, all drillholes that pierce this stratigraphy north of the VVF encounter HWD.

The highest copper grades in the LSZ are in the Johnny Lee LZ. In holes east and west of the Johnny Lee LZ, LSZ stratigraphy shows only weak copper mineralization, and holes east of the Lowry LZ show only weak mineralization or alteration at LSZ stratigraphy. Only two holes test LSZ stratigraphy in the hanging wall of the VVF and neither show mineralization or alteration. Exposures of LSZ stratigraphy at surface south of the BBF show evidence of weak mineralization.

The Johnny Lee LZ has been modelled as the 3D wireframe using a nominal copper cut-off of 2%. The dimensions of the wireframe are approximately 1,175 m in the east-west direction and approximately 125 m in the north-south direction. The unit is wider on the far east end reaching about 300 m in the north-south direction. The LZ is truncated by structures along its southern and northern boundaries. Within the wireframe, there are approximately 2.7 Mt of material at an average copper grade of 4.4% using a zero cut-off grade.

An example of very high-grade copper mineralization from the Johnny Lee LZ is shown in Figure 7.12. This interval contains about 15% copper.

Another example of very high-grade copper mineralization from the Johnny Lee LZ is shown in Figure 7.13 from a 2011 Tintina core hole (SC11-048). The interval between the two wooden run blocks ran 8.5% copper.

Figure 7.11 Finely Bedded LZ Mineralization – SC10-004

Figure 7.12 High-grade LZ Mineralization – SC11-048

8.0 DEPOSIT TYPES

The Black Butte bedded sulphide accumulations best fit a shale-hosted massive sulphide deposit type model. The host rocks contain no volcanic component and in terms of setting and geometry, the sulphide occurrences are quite similar to typical Proterozoic and Phanerozoic shale-hosted zinc and lead rich deposits. However, the high concentrations of copper, cobalt, and barium are unusual in shale hosted sulphide occurrences. Mt. Isa (Perkins 1984) and Walford Creek (Rohrlach et. al. 1998) in Australia are reasonably analogous deposits (Zieg 1992).

The Black Butte exploration model is a middle Proterozoic synsedimentary subaqeous hydrothermal vent field developed at structural intersections during prolonged synsedimentary extensional faulting along the northern margin of the Helena embayment. At Black Butte, early stage pyrite was deposited in sub-wavebase calcareous muddy sediments over extensive areas. Diagenetic sulphide and sulphate mineralization with silicification and carbonate (dolomite) alteration overprinted syngenetic pyrite beds and host muds. Copper ± cobalt and barium replaced some pyrite zones proximal to hydrothermal vent centres at the seabottom. In more distal areas, zinc, lead, and silver are more concentrated. These systems produced multiple stratabound zones of copper-cobalt mineralization in at least five stratigraphic levels within the host shales.

In all bedded sulphide zones, abundant and early fabrics, in the form of microbial mats within massive pyrite beds and masses of phyritized microbial tubiform structures, show a strong biogenic component to mineralization. Such fabrics dominate the sulphide zones in areas where they are of increased thickness and higher sulphide concentrations, and are also present as clasts in debris flows. Sulphide matrix debris flows suggest mound building and collapse. The microbial textures indicate a significant concentration of early thermophiles around seafloor hydrothermal vent centres.

According to Zieg and Lietch (1994), a synsedimentary origin for the Black Butte bedded pyrite and copper-bearing zones is also supported by evidence from fluid inclusion, sulphur isotope, and lead isotope studies. Fluids trapped in pseudosecondary fluid inclusions (Leitch et al. unpublished data) in hydrothermal barite, dolomite, quartz, sphalerite, and calcite average 15 weight percent sodium chloride equivalent, and range from 7 to 23 weight percent sodium chloride equivalent. Homogenization temperatures average 230°C and range from 94 to 300°C, with no evidence of boiling. Himes and Peterson (1990) showed similar results. Metastable melting at temperatures as low as -85°C, with eutectic temperatures clustered around -38°C and -50°C, suggests the presence of Ca++ and possibly Mg++ in the fluid inclusions. Black Butte stable isotope data (Leitch et al. unpublished data) suggest a two-sulphur source: a dominant seawater sulphate source for sulphur in bedded pyrite, which shows a broad range of $\delta^{34}S$ values from -12.1 to 19.7 $^{0}/_{00}$ CDT, and a deep crustal or magmatic source for copper in chalcopyrite, which shows a narrower range from -5.1 to 7.1 ⁰/₀₀ CDT. Chalcopyrite in veins and veinlets in the lower sulphide zone shows $\delta^{34}S$ values, which cluster even more tightly around 0 ⁰/₀₀. Replacement of pyrite by chalcopyrite results in a much broader range of chalcopyrite $\delta^{34}S$ values, suggesting a sulphur-deficient source fluid associated with copper mineralization. One chalcopyrite-pyrite, sulphur isotopic pair yielded a temperature of 276°C. Analyses of USZ barite show $\delta^{34}S$ values of 13.3 to 16.3 ⁰/₀₀, consistent with the expected value for mid-Proterozoic marine sulphate and with sulphate minerals in the Belt basin. Strauss and Schieber (1990) obtained very similar results for pyrite ($\delta^{34}S$ -14 to 18 ⁰/₀₀) and barite ($\delta^{34}S$ 13.6 to 18.3 ⁰/₀₀).

Lead isotope ratios from USZ galena samples, collected six miles apart, are as follows:

- Pb 206/204: 16.843 16.712
- Pb 207/204 15.624 15.550
- Pb 208/204 36.563 36.477.

These values are consistent with a middle Proterozoic age for Black Butte base metal mineralization previously shown by Strauss and Schieber (1990) in other deposits hosted in Belt rocks.

The role of microbial life forms appears to have been critical for reduction of sulphate and substrate for much of the sulphide precipitation during the initial stages of development of the sulphide zones. The lack of more complex life forms during this time allows preservation of far greater quantities of microbial material than in Phanerozoic examples. There is a clear spatial correlation between greater sulphide concentrations, abundance of microbial textures, and increased metal grades.

The various styles of copper mineralization encountered at Black Butte sort into three basic deposit types:

1. shale hosted bedded sulphide copper deposits, as represented by the Johnny Lee UZ
2. contact related deposits, as represented by the Johnny Lee LZ at the Chamberlain Formation – Newland Formation contact
3. breccia-hosted deposits, as represented by the Lowry MZ deposit.

In the shale-hosted bedded sulphide deposits, chalcopyrite replaced fine grained bedded framboidal pyrite, as observed in the USZ, the 0/1 and Unit II sulphide zones in parts of the Johnny Lee and Lowry deposits, and mineralization encountered at Strawberry West. In the contact-related deposits, coarser-grained chalcopyrite replaced pyrite in silicified sediments, as best represented in the Johnny Lee LZ deposit and the widespread copper mineralization in the LSZ in the Lowry deposit. In the breccia-hosted deposits, copper mineralization is strongly associated with replacement of fragmental textures that appear to have resulted from dissolution processes, so far only observed in the Lowry MZ.

9.0 EXPLORATION

CAI, UII, and BHP conducted a variety of geological, geochemical, and geophysical programs between 1976 and 1993. This historic work included surface mapping, surface soil and rock sampling, various geophysical surveys, and core drilling. Geologic maps, down-hole geochemical data, some surface geochemical data, drill logs, some down-hole surveys, and various compilation maps from these programs were recovered from the University of Montana Belt Research Center, to which CAI donated core and geologic information at the end of their tenure at Black Butte. No geophysical data is presently available. Within the area described as “resource” by CAI, the drillhole spacing was approximately 150 m, and 19 holes penetrated the UZ and sub zones, and 12 holes penetrated the LZ. Only three holes encountered both zones. Sixty-four holes in total were drilled on the present leases and an additional four holes were drilled on unpatented claims currently owned by Tintina.

In 2011, Tintina began a more detailed compilation of all available geologic mapping data. Historic maps included work that was completed by CAI and BHP. Tintina staff compiled a number of previous geologic maps of the district, focusing on areas located adjacent to known copper deposits at Black Butte, as well as areas that had historic drilling.

Digitization of historical mapping showed some small differences between geological interpretations on adjoining maps. Tintina's staff has started field-checking some of these discrepancies and additional field examinations will be required. These minor discrepancies are not material regarding the current estimate of mineral resources.

Tintina also compiled historical soil geochemical data and conducted their own soil sampling program in 2011. While compiling historical geochemical data, Tintina's staff converted the older coordinates associated with those samples to Universal Transverse Mercator (UTM) World Geodetic System (WGS) 84 Zone 12N datum. Previous soil sampling focused on areas where prospective Newland stratigraphy is interpreted to be near the surface.

During 2011, Tintina crews collected 744 soil samples over previously un-sampled and geologically permissive areas. Field crews collected samples with a hand auger, and attempted to collect from the “B” soil horizon. If the B horizon was not present, samples were collected from the “C” horizon. The survey consisted of collecting soil samples from 19 north-south oriented lines spaced 300 m apart with samples collected at 60 m intervals along each line. In areas considered more prospective, the line spacing was reduced from 300 to 150 m, with samples collected every 30 m along the line. Three east-west soil lines spaced 500 m apart with samples collected at a spacing of 60 m, covered ground with north-south striking stratigraphy. The area sampled ranged from

501650 east to 506480 east and 5176000 north to 5180000 north. Field crews located all sample sites with a handheld global positioning system (GPS) unit.

ALS Minerals in Reno, Nevada, screened soil samples to 180 µm (PREP 41), digested them in aqua regia, and analyzed them for 51 elements with inductively coupled plasma-mass spectrometry (ICP-MS) (ME-MS41L) and for gold with inductively coupled plasma-atomic emission spectroscopy (ICP-AES) (Au-ICP21). Results showed anomalous samples consistent with the strike extent of known mineralized areas. Other modestly anomalous results require further field review.

In March 2012, Tintina contracted Aeroquest to complete an airborne magnetics and resistivity survey over the district. Results show that the highly conductive properties of the Johnny Lee UZ and many other conductors are generally consistent with known trends of sulphide mineralization. Total reduction to pole (RTP) magnetic data show the transition from a highly magnetic area to the north the correlates with very thin or absent sequences of Belt rocks, to a weaker magnetic area to the south that correlates with thick sequences of Belt rocks. Some strong northeast and northwest trending linear magnetic features correlate with Eocene intrusives on the map. Other linear magnetic features correlate with the surface trace of gossan exposures. Analysis of the results and comparisons with district geology continue.

Also in 2012, Tintina contracted Minex Exploration of Sandpoint, Idaho, to carry out a more detailed ground based magnetic survey over the area of the Johnny Lee and Lowry deposits. This greatly enhanced the accuracy of ground locations of features identified from the airborne magnetics survey results, especially locations of Eocene intrusive dikes and sills.

The exploration potential of the Black Butte area is substantial. A domain of drill-reachable permissive Newland stratigraphy encompasses a zone over 15 miles in length. While the larger airborne electromagnetic survey covers a sizeable portion of this trend, it does not cover all of it. Within the survey area, the prospective Newland stratigraphy, only a few areas have been tested with a high enough density of work to identify potentially mineralized areas similar in size to the Johnny Lee UZ.

10.0 DRILLING

10.1 TYPE AND EXTENT OF DRILLING

As described in Sections 6.0 and 9.0, there have been several drilling campaigns conducted at the Project by four different companies since the early 1980s. The majority of the diamond drilling data for the Johnny Lee UZ, LZ and Lowry MZ has been collected by Tintina as a result of their 2010, 2011 and 2012 drilling campaigns. Drillhole data that were used for the Johnny Lee UZ are summarized in Table 10.1. Approximately 76% of the UZ drilling was completed by Tintina.

Table 10.1 Summary of Johnny Lee UZ Drilling by Company

Company	No. of Holes	Drilled (m)
CAI	4	1,050.31
UII	10	3,094.94
BHP	5	1,604.17
Tintina	87	17,955.12
Total	**106**	**23,704.54**

Table 10.2 tabulates the drillholes that were used to estimate mineral resources for the Johnny Lee LZ. Some of the holes that are summarized in Table 10.2 are also included in Table 10.1.

Table 10.2 Summary of Johnny Lee LZ Drilling by Company

Company	No. of Holes	Drilled (m)
CAI	9	4,185.20
UII	5	2,314.66
BHP	1	456.59
Tintina	32	14,826.71
Total	**47**	**21,783.16**

Table 10.3 tabulates the drillholes that were used to estimate mineral resources for the Lowry MZ. One of the CAI holes (SC-87W) was wedged off another diamond hole (SC-87). Drilling data collected by CAI, UII, and BHP were used by RMI for estimating mineral resources for the Johnny Lee UZ, LZ, and the Lowry MZ.

Table 10.3 Summary of Lowry MZ Drilling by Company

Company	No. of Holes	Drilled (m)
CAI	4	2,558.79
Tintina	25	14,089.82
Total	**29**	**16,648.61**

Table 10.4 is a complete drillhole collar listing for holes that were used to estimate resources for the Project (Johnny Lee UZ, Johnny Lee LZ, and Lowry MZ). Table 10.4 includes XYZ collar locations, azimuth, and dip of the hole at the collar, total depth, which zone or zones were tested, and which company drilled the hole. Two copper sulphide horizons were modelled for the UZ (31 and 32). One zone was modelled for the MZ (21) and two zones were modelled for the LZ (11 and 12).

Figure 10.1 is a plan map showing the location of drillholes in the area of the currently identified Johnny Lee UZ and LZ units. The holes are colour-coded by the company that drilled them. Figure 10.2 shows the drillhole locations of holes used to estimate resources for the Lowry MZ.

Table 10.4 Black Butte Resource Drillhole Locations

BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	UCZ	LCZ	MCZ	Company
SCC-17	506,325.71	5,181,005.73	1,775.10	0.00	-88.50	407.82	31	11	N/A	UII
SCC-34	506,573.56	5,180,998.83	1,799.61	0.00	-90.00	621.49	31	11	N/A	UII
SC10-003	506,325.98	5,181,009.87	1,775.67	0.00	-90.00	365.88	31	11	N/A	Tintina
SC10-004	506,573.20	5,180,996.13	1,799.27	0.00	-90.00	429.91	31	11	N/A	Tintina
SC12-123	506,463.60	5,181,047.10	1,790.91	342.50	-88.00	398.40	31	11	N/A	Tintina
SCC-36	506,768.58	5,181,002.38	1,752.61	0.00	-90.00	425.20	31	12	N/A	BHP
SC11-032	506,501.18	5,181,007.12	1,790.42	0.00	-90.00	469.39	31	12	N/A	Tintina
SC11-036	506,585.20	5,181,076.58	1,800.77	0.00	-90.00	640.10	31	12	N/A	Tintina
SC11-039	506,330.69	5,181,055.43	1,783.56	0.00	-90.00	369.42	31	12	N/A	Tintina
SC-62	506,418.55	5,181,143.93	1,799.49	0.00	-90.00	139.14	31	N/A	N/A	CAI
SC-64	506,344.48	5,181,140.23	1,801.33	0.00	-90.00	131.80	31	N/A	N/A	CAI
SC-71	506,485.12	5,180,363.67	1,759.17	0.00	-90.00	221.59	31	N/A	N/A	CAI
SCC-19	506,352.54	5,181,141.66	1,801.93	0.00	-90.00	252.07	31	N/A	N/A	UII
SCC-23	506,487.55	5,181,145.18	1,802.31	0.00	-90.00	336.19	31	N/A	N/A	UII
SCC-33	506,580.14	5,181,154.03	1,795.89	0.00	-90.00	134.72	31	N/A	N/A	UII
SCC-35	506,754.46	5,181,134.17	1,758.12	0.00	-90.00	176.17	31	N/A	N/A	UII
SCC-37	506,310.12	5,180,689.80	1,801.40	0.00	-90.00	240.49	31	N/A	N/A	BHP
SCC-40	506,632.62	5,180,733.63	1,750.82	0.00	-90.00	611.43	31	N/A	N/A	BHP
SCC-41	506,631.22	5,180,563.44	1,742.26	155.00	-89.30	156.36	31	N/A	N/A	BHP
SC10-001	506,353.76	5,181,138.65	1,801.55	0.00	-90.00	142.07	31	N/A	N/A	Tintina
SC10-002	506,490.51	5,181,146.29	1,802.41	0.00	-90.00	144.78	31	N/A	N/A	Tintina
SC11-014	506,713.05	5,180,756.83	1,745.64	0.00	-90.00	102.11	31	N/A	N/A	Tintina
SC11-016	506,713.82	5,180,667.04	1,739.19	0.00	-90.00	120.09	31	N/A	N/A	Tintina
SC11-024	506,391.91	5,180,376.13	1,766.53	0.00	-90.00	216.10	31	N/A	N/A	Tintina
SC11-030	506,486.44	5,181,077.50	1,797.06	0.00	-90.00	155.45	31	N/A	N/A	Tintina

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	UCZ	LCZ	MCZ	Company
SC11-033	506,277.78	5,180,732.83	1,803.12	32.40	-76.50	227.08	31	N/A	N/A	Tintina
SC11-034	506,413.27	5,180,448.44	1,763.70	33.40	-88.10	199.95	31	N/A	N/A	Tintina
SC11-043	506,635.88	5,180,675.66	1,743.88	219.00	-87.70	129.54	31	N/A	N/A	Tintina
SC11-044	506,369.22	5,180,941.93	1,770.40	0.00	-90.00	140.21	31	N/A	N/A	Tintina
SC11-047	506,590.46	5,181,075.11	1,800.29	61.80	-64.40	139.90	31	N/A	N/A	Tintina
SC11-052	506,801.90	5,180,456.20	1,726.50	276.40	-59.90	173.43	31	N/A	N/A	Tintina
SC11-053	506,799.90	5,180,457.30	1,726.50	357.50	-58.30	153.47	31	N/A	N/A	Tintina
SC11-054	506,802.70	5,180,453.80	1,726.50	149.20	-73.60	148.74	31	N/A	N/A	Tintina
SC11-055	506,767.20	5,180,390.00	1,731.80	253.10	-64.60	155.45	31	N/A	N/A	Tintina
SC11-056	506,767.00	5,180,389.10	1,731.90	153.50	-62.90	161.54	31	N/A	N/A	Tintina
SC11-060	506,583.30	5,180,228.20	1,742.00	354.90	-64.60	309.37	31	N/A	N/A	Tintina
SC11-061	506,326.10	5,180,997.90	1,774.60	198.30	-66.00	242.93	31	N/A	N/A	Tintina
SC11-062	506,413.10	5,180,842.90	1,762.90	232.00	-58.90	215.49	31	N/A	N/A	Tintina
SC11-063	506,579.80	5,180,226.50	1,742.00	59.60	-58.50	222.50	31	N/A	N/A	Tintina
SC11-065	506,583.10	5,180,225.30	1,742.10	157.00	-68.10	161.50	31	N/A	N/A	Tintina
SC11-066	506,377.40	5,180,915.00	1,768.10	252.10	-62.60	215.49	31	N/A	N/A	Tintina
SC11-067	506,498.50	5,180,185.60	1,746.80	347.60	-62.70	197.80	31	N/A	N/A	Tintina
SC11-068	506,661.80	5,180,475.10	1,744.60	0.00	-90.00	148.74	31	N/A	N/A	Tintina
SC11-069	506,494.50	5,180,189.20	1,746.80	60.70	-70.10	182.88	31	N/A	N/A	Tintina
SC11-070	506,652.20	5,180,471.60	1,745.40	180.40	-61.70	173.13	31	N/A	N/A	Tintina
SC11-071	506,498.10	5,180,190.30	1,746.60	169.60	-68.60	167.64	31	N/A	N/A	Tintina
SC11-072	506,667.30	5,180,527.70	1,743.90	57.30	-62.70	148.70	31	N/A	N/A	Tintina
SC12-118	506,620.60	5,180,343.10	1,748.41	33.40	-83.40	164.60	31	N/A	N/A	Tintina
SC12-119	506,666.90	5,180,527.50	1,743.69	333.10	-74.80	146.30	31	N/A	N/A	Tintina
SC12-120	506,620.70	5,180,343.50	1,747.86	29.10	-75.30	164.60	31	N/A	N/A	Tintina
SC12-121	506,666.50	5,180,527.10	1,743.89	37.80	-80.90	143.30	31	N/A	N/A	Tintina

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	UCZ	LCZ	MCZ	Company
SC12-122	506,620.90	5,180,343.80	1,747.91	29.20	-66.00	167.60	31	N/A	N/A	Tintina
SC12-125	506,666.80	5,180,526.00	1,743.94	75.20	-73.10	146.30	31	N/A	N/A	Tintina
SC12-126	506,620.60	5,180,342.80	1,748.11	203.60	-85.10	170.70	31	N/A	N/A	Tintina
SC12-127	506,667.70	5,180,525.70	1,743.92	95.60	-65.60	152.40	31	N/A	N/A	Tintina
SC12-128	506,620.60	5,180,342.70	1,748.06	203.00	-75.10	172.20	31	N/A	N/A	Tintina
SC12-131	506,620.40	5,180,342.40	1,748.03	211.70	-65.00	185.90	31	N/A	N/A	Tintina
SC12-133	506,620.90	5,180,342.90	1,748.32	124.70	-81.10	163.10	31	N/A	N/A	Tintina
SC12-134	506,661.70	5,180,475.40	1,744.91	32.50	-78.20	152.40	31	N/A	N/A	Tintina
SC12-135	506,661.70	5,180,475.70	1,744.87	33.30	-68.90	152.40	31	N/A	N/A	Tintina
SC12-136	506,621.60	5,180,342.50	1,748.19	109.70	-70.70	161.50	31	N/A	N/A	Tintina
SC12-137	506,662.60	5,180,474.80	1,744.80	106.90	-78.70	152.40	31	N/A	N/A	Tintina
SC12-138	506,620.20	5,180,343.40	1,748.11	293.20	-84.40	172.20	31	N/A	N/A	Tintina
SC12-139	506,662.60	5,180,475.10	1,744.73	104.30	-66.60	155.40	31	N/A	N/A	Tintina
SC12-140	506,619.80	5,180,343.60	1,748.07	275.10	-76.00	176.80	31	N/A	N/A	Tintina
SC12-141	506,662.90	5,180,475.00	1,744.69	101.80	-57.70	167.60	31	N/A	N/A	Tintina
SC12-143	506,660.80	5,180,475.70	1,744.92	188.10	-80.50	155.50	31	N/A	N/A	Tintina
SC12-144	506,579.80	5,180,226.30	1,742.15	306.60	-75.20	173.70	31	N/A	N/A	Tintina
SC12-145	506,660.90	5,180,475.40	1,744.69	190.00	-70.80	161.50	31	N/A	N/A	Tintina
SC12-146	506,579.80	5,180,226.70	1,742.11	346.10	-70.10	175.30	31	N/A	N/A	Tintina
SC12-147	506,660.60	5,180,474.90	1,744.69	287.40	-80.00	154.50	31	N/A	N/A	Tintina
SC12-148	506,660.20	5,180,475.00	1,744.71	283.70	-72.10	160.00	31	N/A	N/A	Tintina
SC12-149	506,580.30	5,180,226.40	1,742.26	33.90	-70.60	179.80	31	N/A	N/A	Tintina
SC12-150	506,659.90	5,180,475.20	1,744.72	284.60	-64.80	164.60	31	N/A	N/A	Tintina
SC12-151	506,580.20	5,180,226.00	1,742.25	64.80	-89.60	155.50	31	N/A	N/A	Tintina
SC-63	506,603.44	5,180,870.46	1,776.16	0.00	-90.00	557.78	31 & 32	N/A	N/A	CAI
SCC-21	506,443.38	5,180,962.17	1,778.90	0.00	-90.00	417.58	31 & 32	N/A	N/A	UII

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	UCZ	LCZ	MCZ	Company
SC11-007	506,775.28	5,180,915.26	1,747.01	0.00	-90.00	475.49	31 & 32	N/A	N/A	Tintina
SC11-029	506,650.40	5,180,930.50	1,780.90	0.00	-90.00	512.06	31 & 32	N/A	N/A	Tintina
SC12-110	506,878.00	5,180,816.90	1,723.55	324.00	-70.00	445.01	31 & 32	N/A	N/A	Tintina
SC12-124	506,403.10	5,181,010.50	1,780.36	21.30	-87.60	413.90	31 & 32	N/A	N/A	Tintina
SC11-031	506,559.60	5,180,930.36	1,789.00	0.00	-90.00	548.64	31 & 32	N/A	N/A	Tintina
SC12-130	506,662.80	5,181,012.60	1,785.28	295.00	-88.40	459.30	31 & 32	N/A	N/A	Tintina
SCC-22	506,359.09	5,180,858.10	1,767.20	0.00	-90.00	499.26	31 & 32	N/A	N/A	UII
SCC-25	506,784.21	5,180,751.82	1,739.33	0.00	-90.00	121.62	31 & 32	N/A	N/A	UII
SCC-30	506,476.46	5,180,746.01	1,756.36	0.00	-90.00	128.02	31 & 32	N/A	N/A	UII
SCC-38	506,483.85	5,180,586.13	1,759.04	0.00	-90.00	170.69	31 & 32	N/A	N/A	BHP
SC11-013	506,745.36	5,180,829.58	1,750.35	0.00	-90.00	94.49	31 & 32	N/A	N/A	Tintina
SC11-017	506,642.23	5,180,810.25	1,762.65	0.00	-90.00	114.91	31 & 32	N/A	N/A	Tintina
SC11-018	506,540.74	5,180,811.75	1,765.96	0.00	-90.00	124.97	31 & 32	N/A	N/A	Tintina
SC11-019	506,572.95	5,180,500.03	1,751.43	0.00	-90.00	157.58	31 & 32	N/A	N/A	Tintina
SC11-020	506,459.30	5,180,869.96	1,767.55	0.00	-90.00	124.97	31 & 32	N/A	N/A	Tintina
SC11-021	506,485.50	5,180,512.81	1,758.41	0.00	-90.00	188.37	31 & 32	N/A	N/A	Tintina
SC11-026	506,584.17	5,180,398.57	1,750.91	0.00	-90.00	180.14	31 & 32	N/A	N/A	Tintina
SC11-027	506,662.83	5,181,012.74	1,785.06	0.00	-90.00	111.86	31 & 32	N/A	N/A	Tintina
SC11-028	506,494.30	5,180,438.79	1,758.26	0.00	-90.00	190.20	31 & 32	N/A	N/A	Tintina
SC11-035	506,416.26	5,181,040.87	1,785.68	0.00	-90.00	156.06	31 & 32	N/A	N/A	Tintina
SC11-038	506,303.16	5,180,695.42	1,801.78	101.50	-77.90	220.07	31 & 32	N/A	N/A	Tintina
SC11-040	506,495.21	5,180,645.54	1,758.16	316.80	-84.20	153.62	31 & 32	N/A	N/A	Tintina
SC11-041	506,554.40	5,180,686.40	1,748.20	0.00	-90.00	129.69	31 & 32	N/A	N/A	Tintina
SC11-042	506,402.77	5,180,538.25	1,767.57	17.30	-80.10	205.13	31 & 32	N/A	N/A	Tintina
SC11-045	506,415.87	5,180,826.12	1,761.80	177.40	-79.20	146.91	31 & 32	N/A	N/A	Tintina
SC11-046	506,548.61	5,180,599.52	1,749.66	63.80	-82.80	149.96	31 & 32	N/A	N/A	Tintina

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	UCZ	LCZ	MCZ	Company
SC11-064	506,474.30	5,180,751.60	1,756.80	231.80	-61.10	185.47	31 & 32	N/A	N/A	Tintina
SC11-074	506,396.70	5,180,532.10	1,768.10	2.00	-59.80	206.04	31 & 32	N/A	N/A	Tintina
SC12-132	506,567.10	5,180,753.50	1,755.02	64.40	-89.00	143.00	31 & 32	N/A	N/A	Tintina
SC-50	506,961.67	5,180,889.97	1,713.72	0.00	-90.00	408.43	N/A	11	N/A	CAI
SC-51	507,120.63	5,180,888.73	1,709.12	0.00	-90.00	415.44	N/A	11	N/A	CAI
SC-52	507,119.39	5,180,964.78	1,708.40	0.00	-90.00	402.64	N/A	11	N/A	CAI
SC-55	507,121.15	5,180,674.14	1,711.25	0.00	-90.00	561.75	N/A	11	N/A	CAI
SC-57	507,274.68	5,180,679.07	1,711.07	0.00	-90.00	496.52	N/A	11	N/A	CAI
SC-58	506,928.31	5,180,739.20	1,721.98	0.00	-90.00	569.67	N/A	11	N/A	CAI
SC-90	507,301.03	5,180,830.94	1,709.83	0.00	-90.00	439.52	N/A	11	N/A	CAI
SC-91	506,994.42	5,180,981.23	1,708.07	0.00	-90.00	333.45	N/A	11	N/A	CAI
SCC-20	506,192.42	5,181,052.08	1,788.52	0.00	-90.00	442.57	N/A	11	N/A	UII
SCC-46	507,120.97	5,180,828.54	1,709.90	0.00	-90.00	456.59	N/A	11	N/A	BHP
SC10-005	507,119.35	5,180,825.79	1,709.93	0.00	-90.00	422.43	N/A	11	N/A	Tintina
SC11-008	507,044.04	5,180,906.06	1,708.72	0.00	-90.00	420.00	N/A	11	N/A	Tintina
SC11-011	507,215.58	5,180,825.55	1,709.87	0.00	-90.00	457.20	N/A	11	N/A	Tintina
SC11-012	507,210.76	5,180,901.49	1,709.26	0.00	-90.00	420.01	N/A	11	N/A	Tintina
SC11-015	507,212.95	5,180,740.61	1,710.97	0.00	-90.00	518.38	N/A	11	N/A	Tintina
SC11-048	506,913.20	5,180,824.70	1,720.19	342.50	-74.40	431.90	N/A	11	N/A	Tintina
SC12-100	507,169.20	5,180,813.50	1,710.26	0.00	-90.00	483.70	N/A	11	N/A	Tintina
SC12-101	507,185.30	5,180,878.50	1,709.37	0.00	-90.00	460.25	N/A	11	N/A	Tintina
SC12-102	507,207.80	5,180,779.40	1,710.46	0.00	-90.00	487.07	N/A	11	N/A	Tintina
SC12-103	507,288.70	5,180,751.60	1,710.24	0.00	-90.00	496.82	N/A	11	N/A	Tintina
SC12-104	507,204.50	5,180,674.30	1,710.75	0.00	-90.00	533.10	N/A	11	N/A	Tintina
SC12-142	506,900.30	5,180,958.00	1,724.49	346.70	-88.00	416.40	N/A	11	N/A	Tintina
SC11-009	507,062.33	5,180,827.03	1,709.32	240.00	-87.00	450.95	N/A	12	N/A	Tintina

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BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	UCZ	LCZ	MCZ	Company
SC11-010	507,045.22	5,180,738.69	1,710.28	0.00	-90.00	545.59	N/A	12	N/A	Tintina
SC11-023	507,128.19	5,180,756.62	1,710.04	0.00	-90.00	511.45	N/A	12	N/A	Tintina
SC12-105	506,966.60	5,180,829.50	1,716.19	27.90	-85.00	438.91	N/A	12	N/A	Tintina
SC12-106	507,247.70	5,180,854.40	1,709.55	23.10	-80.00	428.70	N/A	12	N/A	Tintina
SC12-107	507,365.50	5,180,681.30	1,710.95	8.50	-82.00	499.26	N/A	12	N/A	Tintina
SC12-109	507,336.00	5,180,784.90	1,710.25	332.30	-84.00	450.49	N/A	12	N/A	Tintina
SC12-129	506,275.90	5,181,039.30	1,779.40	353.70	-82.80	426.60	N/A	12	N/A	Tintina
SC-80	509,162.80	5,179,686.50	1,726.20	0.00	-90.00	610.51	N/A	N/A	21	CAI
SC-86	509,053.27	5,179,710.62	1,725.53	0.00	-90.00	615.39	N/A	N/A	21	CAI
SC-87	509,058.62	5,179,561.49	1,729.50	0.00	-90.00	681.84	N/A	N/A	21	CAI
SC-87W	509,058.62	5,179,561.49	1,729.50	0.00	-90.00	651.05	N/A	N/A	21	CAI
SC10-006	509,162.70	5,179,683.30	1,726.20	0.00	-90.00	579.12	N/A	N/A	21	Tintina
SC11-049	509,063.00	5,179,934.90	1,736.30	16.20	-89.10	594.40	N/A	N/A	21	Tintina
SC11-076	509,126.50	5,179,808.00	1,727.70	0.00	-90.00	581.56	N/A	N/A	21	Tintina
SC11-077	508,996.20	5,179,458.00	1,733.50	0.00	-90.00	779.68	N/A	N/A	21	Tintina
SC11-079	509,118.90	5,179,455.80	1,731.30	0.00	-90.00	723.60	N/A	N/A	21	Tintina
SC11-081	509,201.50	5,179,566.10	1,726.30	300.00	-80.00	639.47	N/A	N/A	21	Tintina
SC11-082	509,002.00	5,179,582.40	1,729.20	30.00	-82.00	682.14	N/A	N/A	21	Tintina
SC11-084	509,001.40	5,179,824.70	1,728.00	0.00	-90.00	594.06	N/A	N/A	21	Tintina
SC11-085	509,268.00	5,179,413.10	1,731.80	0.00	-90.00	767.59	N/A	N/A	21	Tintina
SC11-090	509,154.60	5,179,322.20	1,735.20	0.00	-90.00	773.58	N/A	N/A	21	Tintina
SC12-152	509,113.30	5,179,759.40	1,725.61	0.00	-90.00	484.63	N/A	N/A	21	Tintina
SC12-153	509,025.90	5,179,767.00	1,725.21	0.00	-90.00	482.50	N/A	N/A	21	Tintina
SC12-154	509,114.40	5,179,769.40	1,725.69	200.30	-80.90	511.30	N/A	N/A	21	Tintina
SC12-155	509,061.60	5,179,821.60	1,731.63	0.00	-90.00	445.77	N/A	N/A	21	Tintina
SC12-156	509,106.00	5,179,871.70	1,737.23	0.00	-90.00	377.34	N/A	N/A	21	Tintina

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BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	UCZ	LCZ	MCZ	Company
SC12-157	508,988.10	5,179,628.50	1,726.10	241.00	-78.80	469.09	N/A	N/A	21	Tintina
SC12-158	508,972.50	5,179,501.90	1,732.53	0.00	-90.00	605.94	N/A	N/A	21	Tintina
SC12-159	509,021.40	5,179,876.10	1,736.15	0.00	-90.00	398.37	N/A	N/A	21	Tintina
SC12-160	508,992.90	5,179,632.40	1,725.73	43.50	-80.80	520.14	N/A	N/A	21	Tintina
SC12-161	508,952.40	5,179,877.90	1,732.96	0.00	-90.00	547.12	N/A	N/A	21	Tintina
SC12-163	509,052.30	5,179,492.50	1,732.25	0.00	-90.00	673.00	N/A	N/A	21	Tintina
SC12-164	509,111.40	5,179,550.40	1,728.14	0.00	-90.00	575.46	N/A	N/A	21	Tintina
SC12-165	509,168.90	5,179,870.30	1,735.06	0.00	-90.00	367.89	N/A	N/A	21	Tintina
SC12-166	509,134.30	5,179,605.70	1,726.08	315.00	-83.00	501.85	N/A	N/A	21	Tintina
SC12-168	509,180.90	5,179,778.20	1,727.00	0.00	-90.00	414.22	N/A	N/A	21	Tintina

Figure 10.1 Johnny Lee Drillhole Collar Locations

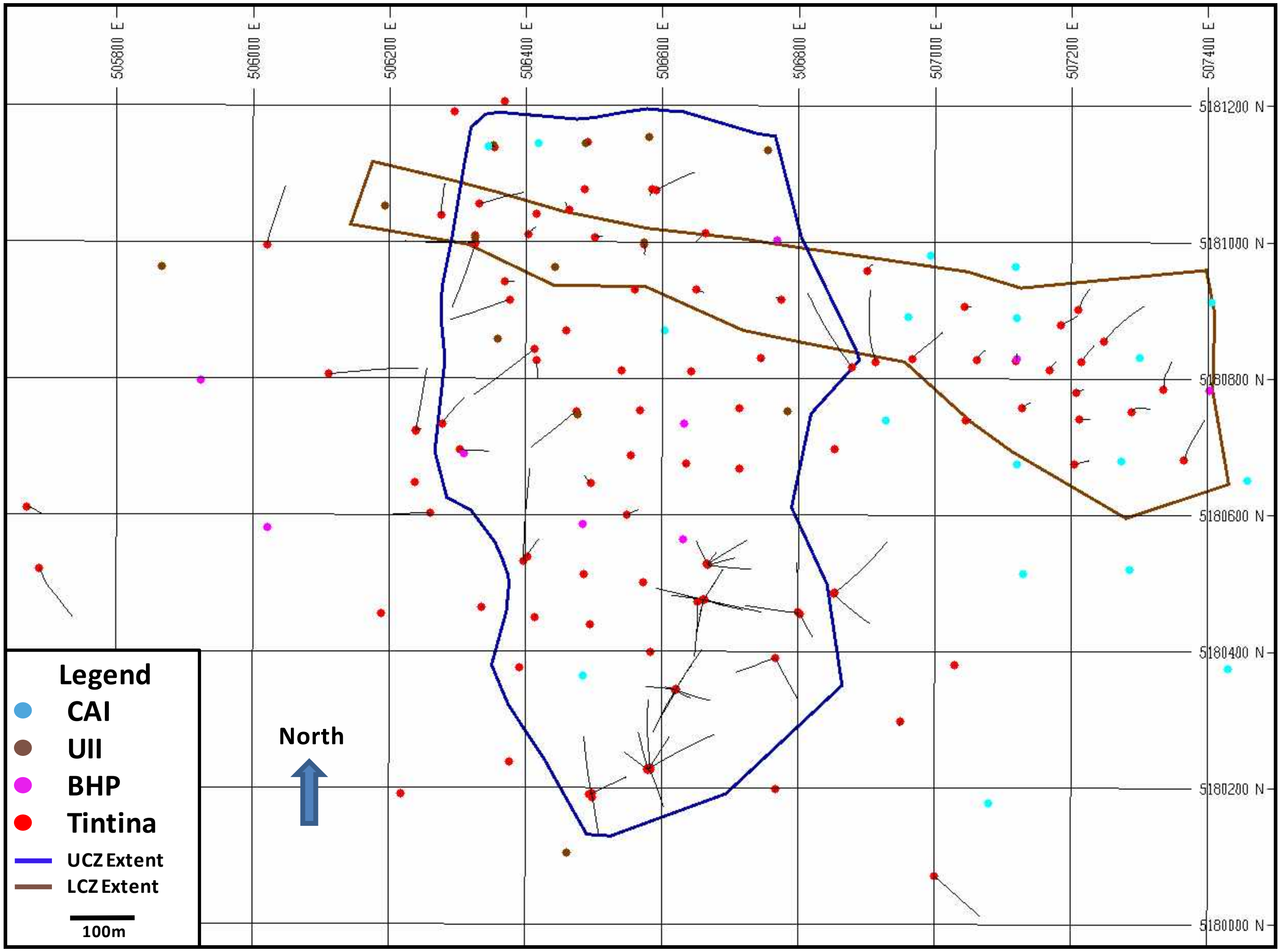
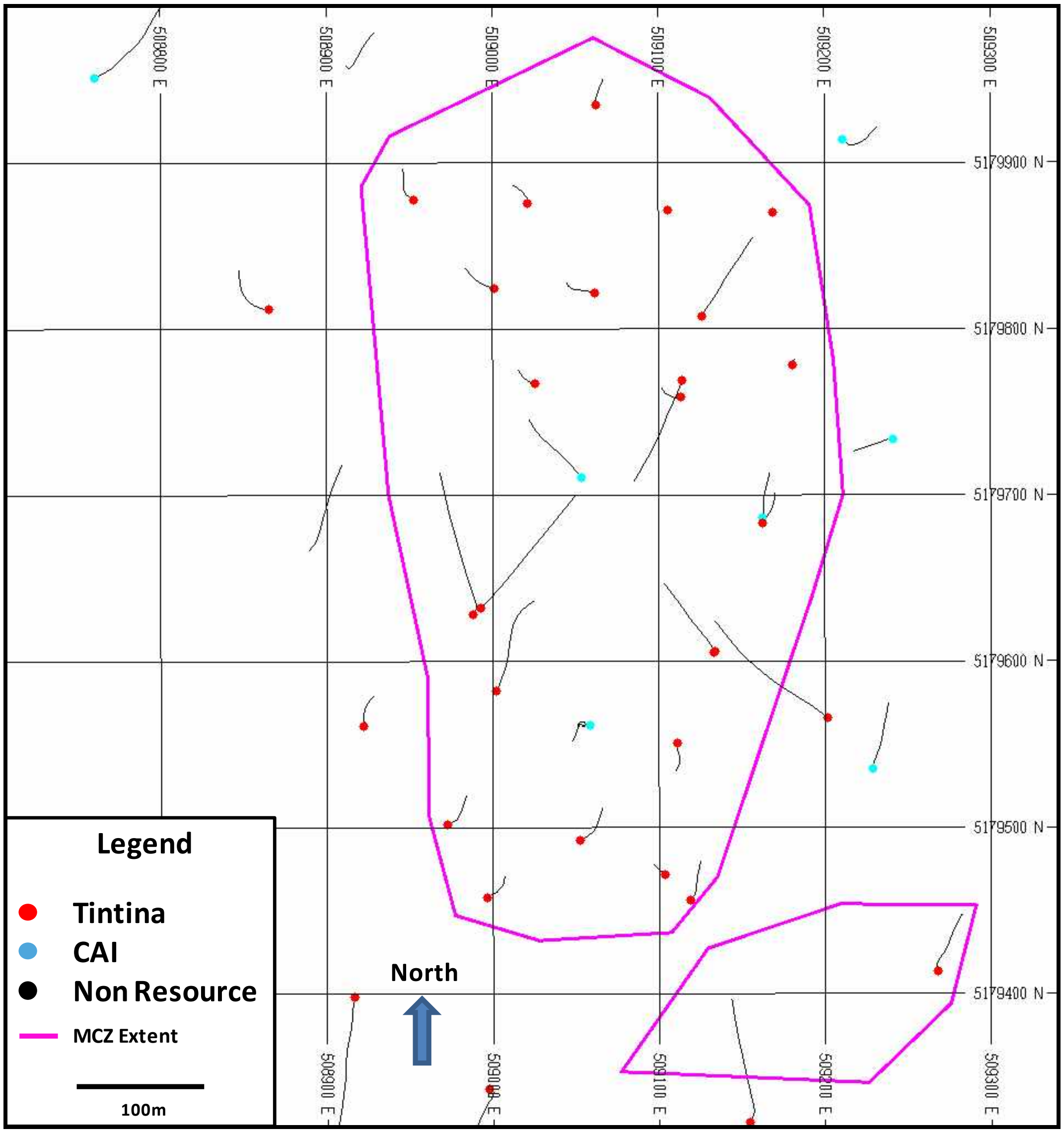


Figure 10.2 Lowry MZ Drillhole Locations



10.2 RELEVANT JOHNNY LEE UZ RESULTS

Samples were collected by Tintina, CAI, UII, and BHP from 106 diamond core holes from the Johnny Lee UZ covering an area measuring approximately 600 m (east-west) by 1,065 m (north-south). Relevant samples from these drilling campaigns are summarized in Table 10.5. The intervals shown in Table 10.5 were used by Tintina's technical staff in developing the shapes for two massive sulphide lenses (UZ #1 and UZ #2 referred to as UZ 31 and UZ 32, respectively).

The intervals shown in Table 10.5 were composited from individual core samples from the hanging wall to footwall contact. Only composited samples above a 2% copper cut-off grade are shown in Table 10.5. Individual copper assays from these composited intervals often exceeded 10% in grade. The raw assay intervals were capped prior to creating 1 m long composites that were used for block grade estimation as described in Section 14.4.

Table 10.5 Relevant Johnny Lee UZ Intervals

BHID	From (m)	To (m)	Length (m)	Cu (%)	Co (%)	Ag (g/t)	Au (g/t)	UCZ	Company
SC10-001	118.26	125.00	6.74	3.44	0.18	12.9	0.004	31	Tintina
SC10-002	131.31	138.77	7.46	2.48	0.36	5.8	0.003	31	Tintina
SC10-003	131.88	139.50	7.62	3.21	0.17	21.6	0.004	31	Tintina
SC10-004	107.90	118.41	10.51	3.03	0.15	10.0	0.006	31	Tintina
SC11-013	50.99	56.82	5.83	3.55	0.17	10.4	0.003	32	Tintina
SC11-014	57.85	74.68	16.83	2.10	0.10	16.5	0.005	31	Tintina
SC11-017	78.74	83.65	4.91	3.56	0.07	9.4	0.006	31	Tintina
SC11-018	72.45	76.76	4.31	2.36	0.15	11.4	0.003	32	Tintina
SC11-018	90.18	94.18	4.00	2.98	0.12	13.1	0.004	31	Tintina
SC11-019	99.50	110.84	11.34	2.10	0.06	11.8	0.004	32	Tintina
SC11-021	150.85	154.23	3.38	2.69	0.03	14.5	0.007	31	Tintina
SC11-027	76.20	83.65	7.45	2.52	0.16	10.3	0.013	32	Tintina
SC11-028	144.07	148.21	4.14	2.78	0.10	22.0	0.009	32	Tintina
SC11-029	63.98	73.00	9.02	2.66	0.15	13.5	0.008	32	Tintina
SC11-029	85.75	88.76	3.01	2.99	0.13	8.6	0.012	31	Tintina
SC11-030	115.50	127.00	11.50	2.06	0.14	8.5	0.022	31	Tintina
SC11-031	93.00	103.60	10.60	2.05	0.11	8.1	0.005	31	Tintina
SC11-034	179.87	183.24	3.37	2.26	0.04	19.8	0.014	31	Tintina
SC11-035	100.88	103.88	3.00	2.30	0.06	14.7	0.003	32	Tintina
SC11-039	124.36	133.70	9.34	2.49	0.31	19.9	0.005	31	Tintina
SC11-041	76.24	86.37	10.13	2.08	0.11	9.2	0.004	32	Tintina
SC11-041	92.91	99.71	6.80	2.45	0.06	11.3	0.003	31	Tintina
SC11-044	113.70	120.00	6.30	3.15	0.16	21.8	0.005	31	Tintina
SC11-045	119.20	122.30	3.10	2.13	0.03	9.2	0.003	31	Tintina
SC11-046	98.47	108.10	9.63	2.65	0.11	9.3	0.008	32	Tintina
SC11-046	111.76	122.86	11.10	2.30	0.18	19.1	0.007	31	Tintina
SC11-054	95.06	98.06	3.00	2.21	0.08	47.8	0.003	31	Tintina
SC11-055	104.50	123.92	19.42	2.19	0.07	14.8	0.004	31	Tintina
SC11-060	121.40	149.18	27.78	2.64	0.06	19.9	0.009	31	Tintina
SC11-061	157.75	165.92	8.17	2.09	0.14	18.2	0.007	31	Tintina
SC11-062	161.49	168.63	7.14	2.92	0.14	43.9	0.005	31	Tintina
SC11-064	107.58	116.00	8.42	4.80	0.12	53.6	0.008	32	Tintina
SC11-066	156.09	161.10	5.01	3.26	0.17	17.2	0.003	31	Tintina
SC11-068	109.75	128.43	18.68	2.91	0.11	14.4	0.003	31	Tintina
SC11-069	122.45	129.14	6.69	2.32	0.07	13.9	0.003	31	Tintina
SC11-072	110.97	132.00	21.03	2.70	0.10	15.7	0.008	31	Tintina

table continues…

BHID	From (m)	To (m)	Length (m)	Cu (%)	Co (%)	Ag (g/t)	Au (g/t)	UCZ	Company
SC12-125	107.70	122.00	14.30	2.18	0.08	12.3	0.004	31	Tintina
SC12-130	75.90	85.30	9.40	3.18	0.12	8.6	0.007	32	Tintina
SC12-130	93.50	106.70	13.20	2.32	0.13	5.8	0.009	31	Tintina
SC12-131	131.00	152.75	21.75	2.21	0.07	18.9	0.006	31	Tintina
SC12-132	70.60	78.10	7.50	3.39	0.17	18.2	0.003	32	Tintina
SC12-132	83.90	92.43	8.53	2.97	0.11	11.0	0.004	31	Tintina
SC12-135	115.31	130.80	15.49	2.54	0.08	11.9	0.005	31	Tintina
SC12-137	113.36	129.47	16.11	3.38	0.09	13.0	0.011	31	Tintina
SC12-139	117.30	135.75	18.45	2.45	0.08	14.7	0.003	31	Tintina
SC12-143	108.22	135.42	27.20	2.70	0.09	12.4	0.003	31	Tintina
SC12-144	115.90	134.00	18.10	2.17	0.05	24.0	0.003	31	Tintina
SC12-145	113.30	139.29	25.99	2.59	0.08	14.1	0.007	31	Tintina
SC12-147	107.05	136.60	29.55	2.52	0.10	13.9	0.003	31	Tintina
SC12-148	111.90	140.40	28.50	2.22	0.09	13.1	0.005	31	Tintina
SC12-150	117.50	133.45	15.95	2.95	0.10	14.5	0.003	31	Tintina
SC12-151	108.70	112.78	4.08	2.02	0.05	16.2	0.007	31	Tintina
SC-64	118.87	123.90	5.03	3.26	0.10	9.4	0.029	31	CAI
SC-71	168.25	171.91	3.66	2.09	0.03	17.9	0.003	31	CAI
SCC-17	130.76	137.46	6.70	2.76	0.19	18.5	0.012	31	UII
SCC-19	115.21	123.75	8.54	3.05	0.18	10.4	0.011	31	UII
SCC-23	132.59	140.51	7.92	3.62	0.54	5.4	0.012	31	UII
SCC-30	92.35	96.62	4.27	2.84	0.17	7.0	0.001	32	UII
SCC-33	109.73	124.36	14.63	2.04	0.15	6.4	0.012	31	UII
SCC-34	108.20	119.79	11.59	2.24	0.13	9.9	0.010	31	UII
SCC-36	31.39	42.37	10.98	2.30	0.10	6.1	0.010	31	BHP

10.3 RELEVANT JOHNNY LEE LZ RESULTS

Samples were collected by Tintina, CAI, UII, and BHP from 33 diamond core holes from the LZ covering an area measuring approximately 1,275 m (northwest-southeast) by 100 to 250 m (northeast-southwest). Relevant samples from these drilling campaigns are summarized in Table 10.6 based on hanging wall to footwall composites.

The intervals shown in Table 10.6 were used by RMI to estimate resources for a single massive sulphide horizon referred to as LZ 11.

Table 10.6 Relevant Johnny Lee LZ Intervals

BHID	From (m)	To (m)	Length (m)	Cu (%)	Co (%)	Ag (g/t)	Au (g/t)	LCZ	Company
SC10-004	414.00	418.05	4.05	10.84	0.08	8.3	0.210	11	Tintina
SC10-005	405.80	412.15	6.35	8.72	0.11	5.0	0.048	11	Tintina
SC11-008	355.22	357.40	2.18	2.45	0.04	6.6	0.697	11	Tintina
SC11-011	409.65	422.70	13.05	3.18	0.02	2.5	0.348	11	Tintina
SC11-015	449.29	456.59	7.30	3.14	0.04	6.1	0.461	11	Tintina
SC11-029	437.00	440.63	3.63	13.97	0.02	7.5	0.232	11	Tintina
SC11-048	356.87	367.60	10.73	5.27	0.06	4.8	0.497	11	Tintina
SC12-100	412.00	424.10	12.10	8.55	0.03	2.6	0.552	11	Tintina
SC12-101	382.95	397.75	14.80	5.60	0.04	2.5	0.274	11	Tintina
SC12-102	429.70	441.35	11.65	3.18	0.10	3.1	0.248	11	Tintina
SC12-103	444.50	447.15	2.65	14.09	0.01	2.7	0.688	11	Tintina
SC12-104	460.10	477.43	17.33	8.32	0.04	7.9	0.281	11	Tintina
SC12-110	405.34	408.04	2.70	2.33	0.02	3.8	0.035	11	Tintina
SC12-123	361.75	363.93	2.18	10.76	0.01	6.2	0.754	11	Tintina
SC12-124	360.32	363.90	3.58	4.40	0.02	3.4	0.366	11	Tintina
SC12-142	340.50	349.54	9.04	3.24	0.02	2.5	0.401	11	Tintina
SC-50	367.89	370.33	2.44	7.75	0.01	3.2	0.395	11	CAI
SC-51	397.61	404.77	7.16	5.80	0.01	1.7	0.190	11	CAI
SC-55	463.60	470.31	6.71	10.12	0.02	12.5	0.429	11	CAI
SC-57	482.50	486.16	3.66	6.47	0.02	6.3	0.307	11	CAI
SC-90	383.26	384.54	1.28	11.64	0.02	10.9	0.095	11	CAI
SCC-17	355.70	358.14	2.44	6.82	0.05	3.0	0.342	11	CAI
SCC-21	394.56	400.66	6.10	4.78	0.04	4.0	0.243	11	UII
SCC-34	413.61	418.49	4.88	7.56	0.15	7.6	0.406	11	UII
SCC-46	400.35	412.76	12.41	5.71	0.03	2.4	0.267	11	BHP

10.4 RELEVANT LOWRY MZ INTERVALS

Samples were collected by Tintina and CAI from 29 diamond core holes from the Lowry MZ covering an area measuring about 650 m (north-south) to approximately 300 m (east-west). Relevant samples from these drilling campaigns are summarized in Table 10.7 based on hanging wall to footwall composites.

The intervals shown in Table 10.7 were used by RMI to estimate resources for a single massive sulphide horizon referred to as MZ 21.

Table 10.7 Relevant Lowry MZ Intervals

BHID	From (m)	To (m)	Length (m)	Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)	MCZ	Company
SC10-006	384.02	430.64	46.62	2.57	0.12	13.3	0.006	21	Tintina
SC11-076	319.36	356.87	37.51	2.38	0.13	13.1	0.004	21	Tintina
SC11-084	324.12	334.19	10.07	2.29	0.18	14.9	0.025	21	Tintina
SC11-085	659.98	668.13	8.15	3.31	0.11	12.6	0.008	21	Tintina
SC12-152	347.50	385.90	38.40	2.66	0.09	12.5	0.010	21	Tintina
SC12-153	348.00	357.00	9.00	2.50	0.09	14.5	0.002	21	Tintina
SC12-164	480.97	519.10	38.13	2.83	0.08	16.7	0.002	21	Tintina
SC12-165	301.70	308.00	6.30	3.07	0.04	9.2	0.005	21	Tintina
SC12-168	348.76	372.42	23.66	5.45	0.11	20.9	0.004	21	Tintina
SC-80	393.50	444.40	50.90	2.80	0.11	11.3	0.005	21	CAI

10.5 2010 TINTINA DRILLING

In 2010, Tintina completed 1,509.65 m of core drilling in five holes to verify historic results obtained by CAI, UII, and BHP (Lechner 2010). The work was carried out on Tintina's behalf by Spring Valley Drilling from Hot Springs, Montana. The contractor came well recommended and successfully completed each of the five holes using a track-mounted core-drilling rig capable of recovering HQ (2.5″) diameter core from the depths required. In holes SC10-001 through SC10-004, HQ-sized core was recovered. In hole SC10-005 drilling problems required reduction to NTW (2.25″ diameter) core. Core recoveries through the sulphide zones were excellent. Downhole surveys were conducted using both Reflex and Devico multi-shot tools and down hole surveys show that the holes are reasonably straight. Because of drilling problems, only two survey stations 70 m apart were collected from the bottom of SC10-005.

10.6 2011 TINTINA DRILLING

In 2011, Tintina contracted Ruen Drilling Inc. (Ruen) from Clark Fork, Idaho. Ruen used two track-mounted Longyear LF-90's drills, a Longyear LF-70, a CS-1000 and two wheel-mounted CS-1500s. The hole collars were surveyed by WWC Engineering from Helena, Montana, the same as in 2010. The 2011 drillholes were surveyed down-the-hole using a Reflex tool. The drills were manned by two crews that each worked 12-hour shifts.

Drill core was delivered to Tintina's core logging facility located in White Sulphur Springs, Montana twice daily during drilling operations by company employees. The core was transferred to logging tables in the core storage warehouse where quick logs were generated by staff/contract geologists. Various geologic information such as core condition, lithologic contacts, bedding orientation, estimate of sulphide content, structure, alteration, mineralization, barite content, vent fauna, presence of microbial mat structures, etc. were recorded into Microsoft Word® documents as daily reports that were sent to corporate headquarters.

After quick logging, the core was transferred to a heated logging facility where more detailed core logging was completed. Drilling depths from wooden run blocks were converted from imperial to metric units. Core recovery and rock quality designation (RQD) measurements were then made and recorded in a Microsoft Access® database. All core boxes were weighed and then photographed. Detailed core logs (lithology, alteration, mineralization, structure, etc.) were electronically scanned and then recorded into a Microsoft Access® database by Tintina geologists or technicians.

10.7 2012 Tintina Drilling

In 2012, Tintina contracted Ruen from Clark Fork, Idaho. Ruen used two track-mounted Longyear LF-90's drills, two Longyear LF-70's, and a Longyear LF-230. The hole collars were surveyed by WWC Engineering from Helena, Montana, the same as in 2010 and 2011. The 2012 drillholes were surveyed down-the-hole using a Reflex tool. The drills were manned by two crews that each worked 12-hour shifts.

Drill core was delivered to Tintina's core logging facility located in White Sulphur Springs, Montana, twice daily during drilling operations by company employees. The core was transferred to logging tables in the core storage warehouse where quick logs were generated by staff/contract geologists. Various geologic information such as core condition, lithologic contacts, bedding orientation, estimate of sulphide content, structure, alteration, mineralization, barite content, vent fauna, presence of microbial mat structures, etc. were recorded into Microsoft Word® documents as daily reports that were sent to corporate headquarters.

After quick logging, the core was transferred to a heated logging facility where more detailed core logging was completed. Drilling depths from wooden run blocks were converted from imperial to metric units. Core recovery and RQD measurements were then made and recorded in a Microsoft Access® database. All core boxes were weighed and then photographed. Detailed core logs (lithology, alteration, mineralization, structure, etc.) were electronically scanned and then recorded into a Microsoft Access® database by Tintina geologists or technicians.

10.8 Sampling and Recovery Factors

In general, core recovery by all four companies was satisfactory to excellent. There are no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results.

10.9 True Thickness

As described in Section 7.0, mineralization is typically stratabound. Banding of the bedded pyrite in the sulphide zones and host shale is assumed to be the true orientation of the mineralized horizon. In all cases, the bedding and sulphide banding lies at approximately 80° to 90° from core axis, and so the intersections represent near true thickness of the mineralization.

10.10 Significantly Higher Grade Intervals

The selection of which drillhole intervals to be included in the Johnny Lee and Lowry wireframes was based on geology (i.e. bedded massive sulphide accumulations with visible copper mineralization) and assay results. RMI notes that, in general, most of the individual assay intervals ranging for 0.25 to 2.0 m contain copper grades in excess of 1%. A certain amount of internal dilution was allowed by including weakly mineralized intervals.

The massive sulphide horizons show appreciable short-range variability in grade but, in most cases, no single high-grade interval was allowed to be smeared out across the horizon so as to "carry" that intercept.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 HISTORIC SAMPLE PREPARATION

CAI and UII/BHP completed historic core sampling on all mineralized or altered sections of drill core. Sample interval lengths were usually 5 ft (1.52 m) or less. In mineralized zones, CAI used geologic boundaries to guide placement of sample interval breaks. UII and BHP sampled on strict 2 ft (0.61 m) sample intervals through mineralized zones. In all cases, a geologist logged and photographed the core, marked sample interval boundaries with flagging with footage written on the flagging, stapled a sample tag in the wax impregnated core box, and a geologic technician split the core as directed by the geologist.

11.2 TINTINA SAMPLE PREPARATION

In Tintina's 2010 to 2012 drilling programs, sawn core samples were collected throughout key mineralized horizons from HQ and NQ diameter drillhole intersections. Because the zones generally have some silicification, core recoveries are quite good throughout the mineralized zones. Each box of core was photographed and logged by the geologist. Sample boundaries were marked on core, sample numbers with beginning and ending meterages were marked on aluminum tags and stapled with sample card stubs onto the sides of core boxes. Samples were collected continuously from the beginning to end of each mineralized zone with interval spacings typically less than 2 m in length and usually broken at geologic boundaries. No gaps were left between subsequent samples in any mineralized zone. Boundaries of mineralized zones were selected on the basis of a visual estimation of chalcopyrite content and a comparison with historic drill logs from the nearby twinned drillhole. After logging the geologic data from the drill core, sampling began approximately 30 ft (9 m) above where core appeared to consistently contain more than approximately 0.5% chalcopyrite, and ended 30 ft (9 m) below the last occurrence of mineralization. Sample cards were filled out with the hole number, date, and beginning and ending of interval, and samples were split with an electric powered tile saw either by the geologist or by a geologic technician under supervision of the geologist. Every effort was made to get an unbiased representative sample of the core. Because core recoveries have been good and core is generally only slightly broken if at all, samples are good quality. The geologist marked the core for sampling, and stapled both the paper sample tags and an aluminum tag showing beginning and ending measurements in the wax-impregnated core box.

11.3 HISTORIC SAMPLE ANALYSES

In the CAI, UII, and BHP core sampling programs, a geologic technician put the split core samples in marked bags and sealed them, then boxed or bagged them and shipped them via UPS to the laboratory of choice. The laboratories that were used were Silver Valley Laboratories in Kellogg, Idaho, Bondar Clegg in Vancouver, BC, and BHP's in-house lab at Sunnydale, California. A QA/QC program was conducted by all companies that involved regular injection of blanks, standards, and duplicates. Companies requested a variety of analytical suites with a minimum of a dozen elements including copper, cobalt, lead, zinc, silver, iron, arsenic, and barium. Trace quantities of most metals were analyzed by atomic absorption, and "over limit" quantities were completed by wet chemical and fire assay methods. Details of procedures are no longer available, but interviews with individuals who were involved in the programs show that sampling was completed to the standards of the time.

11.4 TINTINA SAMPLE ANALYSES

In the 2010 Tintina core sampling program, the geologist, or a technician under the geologists supervision, bagged samples in marked bags, sealed the bags, put the individual sample bags in marked rice bags, sealed these bags, and arranged for an on site FedEx pick-up for international FedEx next day delivery to the ALS Chemex laboratory in Vancouver, BC. Some samples were shipped via bundled pallets by UPS to the ALS Chemex lab in Reno, Nevada. The samples sent to Reno were prepared and the pulps sent to the ALS Chemex laboratory in Vancouver, BC. ALS Chemex is an internationally recognized certified laboratory (ISO 9001:2000). ALS Chemex crushed the core to 70% less than 2 mm, pulverized a 250 g split to 85% less than less than 75 µm, completed a four-acid digestion on a split, and carried out both a 33 element inductively coupled plasma (ICP) analysis (ME-ICP61a) and a mineralized material-grade copper assay routine (Cu-G62).

At least one duplicate, blank, and standard reference material (SRM) was inserted into the sample number series for every 20 samples and included in the shipments. The standard that was used for the 2010 Tintina drilling was purchased from WCM Minerals. This commercial standard (PB 134) has certified copper, lead, zinc, and silver values of 0.58%, 0.91%, 1.72%, and 184 g/t, respectively. For the 2010 drilling program, pieces of Newland Formation Unit VII were collected from accessible outcrops along the highway near the Property and used as blank material.

Tintina used additional certified standards that were purchased from WCM Minerals from their 2011 and 2012 drill programs. Table 11.1 summarizes the expected values from all SRMs used by Tintina. Landscaping marble pieces purchased from a local hardware store were used for blank material for the Tintina 2011 and 2012 drilling campaigns.

Table 11.1 SRM Expected Values

WCM Standard	Copper (%)		Co (%)		Ag (g/t)		Au (g/t)		Lead (%)		Zinc (%)	
	Expected Value	Standard Deviation	Expected Value	Standard Deviation	Expected Value	Standard Deviation	Expected Value	Standard Deviation	Expected Value	Standard Deviation	Expected Value	Standard Deviation
CU 145	3.100	0.090	N/A	N/A	93.0	3.366	N/A	N/A	N/A	N/A	N/A	N/A
CU 182	0.770	0.015	N/A	N/A	33.0	1.197	0.8	0.028	N/A	N/A	N/A	N/A
NI 116	0.780	0.013	0.058	0.002	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
PB 129	0.280	0.012	N/A	N/A	23.0	1.696	N/A	N/A	1.240	0.017	2.000	0.006
PB 131	0.470	0.012	N/A	N/A	262.0	10.826	N/A	N/A	1.040	0.035	1.890	0.059
PB 134	0.580	0.009	N/A	N/A	184.0	5.489	N/A	N/A	0.910	0.030	1.720	0.058

11.5 TINTINA SAMPLE SECURITY

The 2010 Tintina drill core was taken from the drill rig by Tintina personnel to a rental house located near the Project site. After logging, the core samples were placed in sealed bags prior to shipment by FedEx or UPS to ALS Chemex laboratories.

Tintina rented a core logging/processing/storage facility in White Sulphur Springs for their 2011 to 2012 drilling campaigns (Figure 11.1). Drill core was retrieved from the drill rigs twice a day by Tintina contractors and delivered to the core processing facility. The drill core was stored in a secure warehouse with the samples placed in sealed bags. In 2011, Tintina constructed a large core storage facility that is located between the Johnny Lee and Lowry deposits. After the drill core was processed in White Sulphur Springs, it was transferred to the secure new on site core storage building.

In RMI's opinion, the Black Butte core is adequately secure and reasonable measures have been undertaken to ensure the safety and integrity of the samples.

Figure 11.1 White Sulphur Springs Core Logging Facility

11.6 HISTORIC QUALITY ASSURANCE/QUALITY CONTROL

Black Butte drilllhole data that was obtained in the 1980s and 1990s was collected by highly reputable major mining companies of that era (i.e. CAI, UII, and BHP). Tintina has made a concerted effort to obtain assay certificates and QA/QC data from Teck, who acquired Cominco in 2008 and ostensibly has all of the Black Butte data that was collected by CAI, UII and BHP. At the time of this report, Teck has not provided Tintina with the requested data.

11.7 TINTINA QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

For their 2010, 2011 and 2012 drill campaigns, Tintina submitted blanks and certified standards with the sawn drill core samples at a frequency of approximately one QA/QC sample per 20 core samples. Tintina also requested that ALS Chemex prepare duplicate samples from the coarse reject that was left over from the initial sample preparation, which was then submitted to Inspectorate Exploration & Mining Services Ltd. (Inspectorate). A large percentage of representative splits from the original ALS Chemex pulps were submitted to Inspectorate for check assaying purposes. Table 11.2 summarizes the number of QA/QC samples that were submitted by Tintina from their 2010, 2011, and 2012 drilling campaigns that are relative to this report.

Table 11.2 Summary of Submitted QA/QC Samples

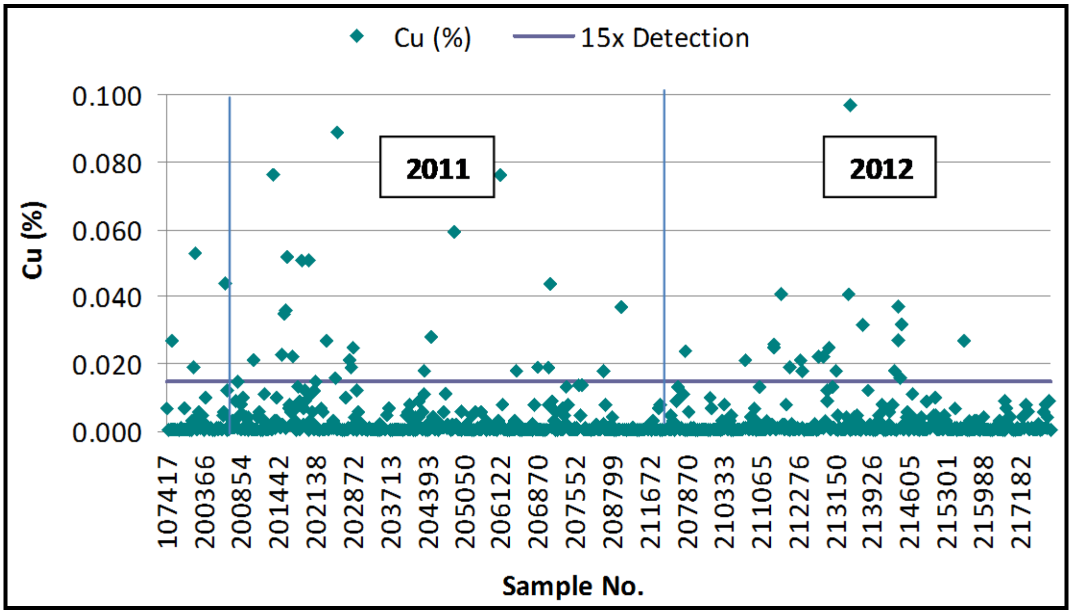
Sample Type	Description	No. of Samples
Blank	Barren outcrop – landscaping marble	810
Standard CU 145	WCM Minerals Certified Standard	128
Standard Cu 182	WCM Minerals Certified Standard	27
Standard NI 116	WCM Minerals Certified Standard	114
Standard PB 129	WCM Minerals Certified Standard	60
Standard PB 131	WCM Minerals Certified Standard	384
Standard PB 134	WCM Minerals Certified Standard	100
HQ-NQ Duplicates	Prepared and assayed by ALS Chemex	787
HQ-NQ Check Assays	Original prepped and assayed by ALS Chemex – pulp split assayed by Inspectorate	70
PQ Check Assays	Original prepped and assayed by Inspectorate – pulp split assayed by ALS Chemex	63

11.8 2010, 2011 AND 2012 TINTINA BLANK PERFORMANCE

In 2010, four blanks were submitted by Tintina for the first four confirmation drillholes. That blank material consisted of locally derived Newland Formation material. Several of the blanks that were assayed for copper returned values higher than 15 times the detection limit. However, those values were significantly lower than 0.1% copper and probably represent trace copper in the Newland Formation. All of the cobalt blanks were less than 10 times the detection limit.

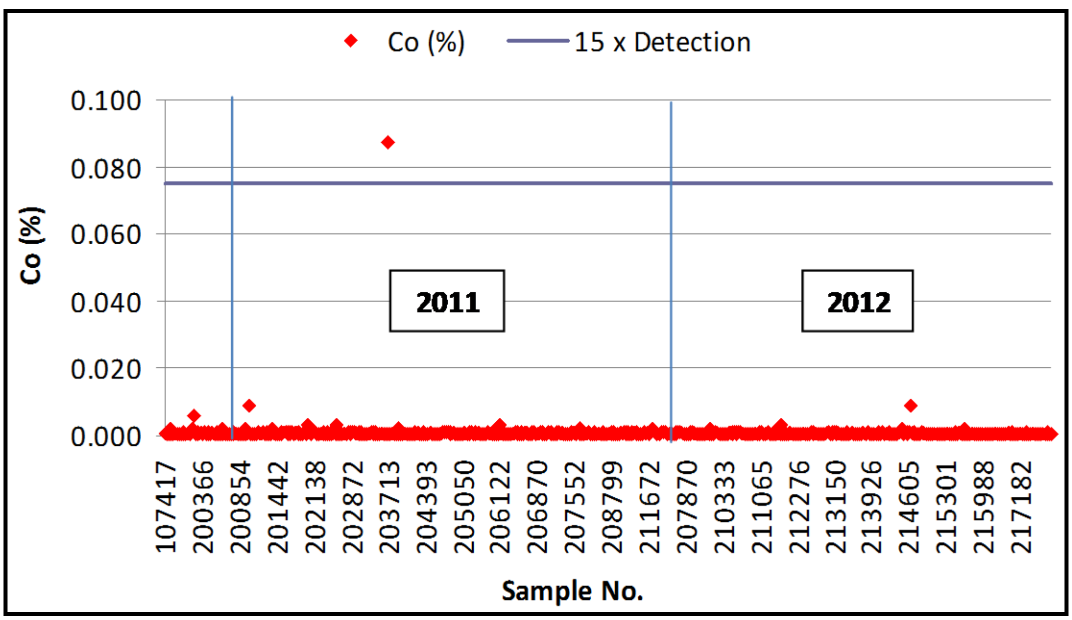
For their 2011 and 2012 drill campaigns, Tintina used commercially available landscape marble pieces, which eliminated the anomalous copper values associated with the Newland Formation blanks used in 2010. Figure 11.2 through Figure 11.5 show values obtained from blank material submitted for the 2010 and 2011 drill campaigns for copper, cobalt, silver, and gold, respectively.

Figure 11.2 Copper Blank Performance



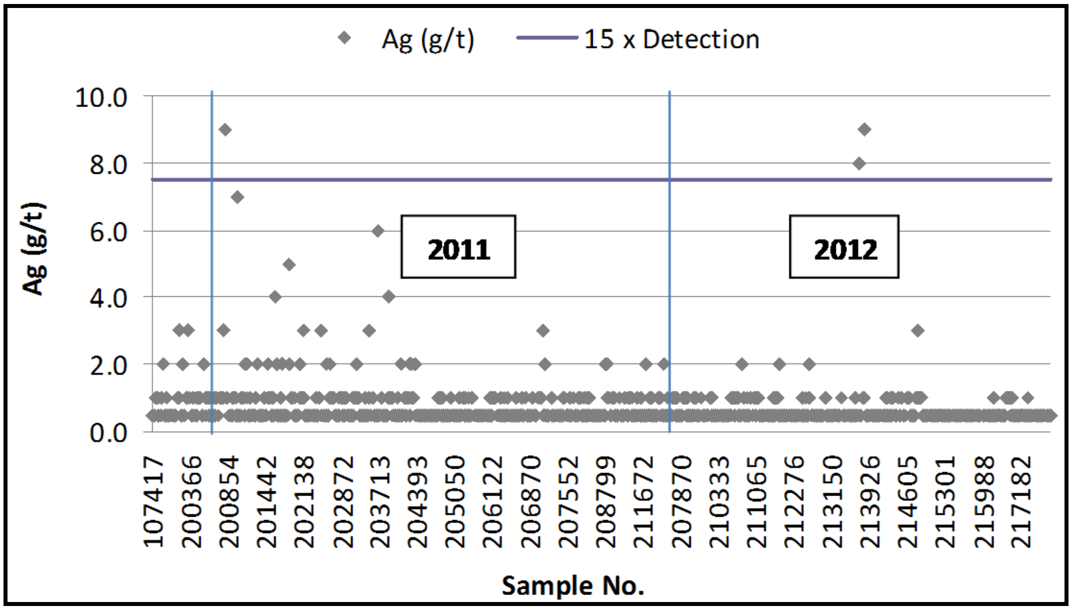
RMI notes that ALS Chemex returned higher copper values for the supposedly barren landscaping marble than would be expected. This noise could also represent contamination of the crushing/grinding equipment. RMI recommends that Tintina pay close attention to the performance of blank material associated with future drilling campaigns.

Figure 11.3 Cobalt Blank Performance



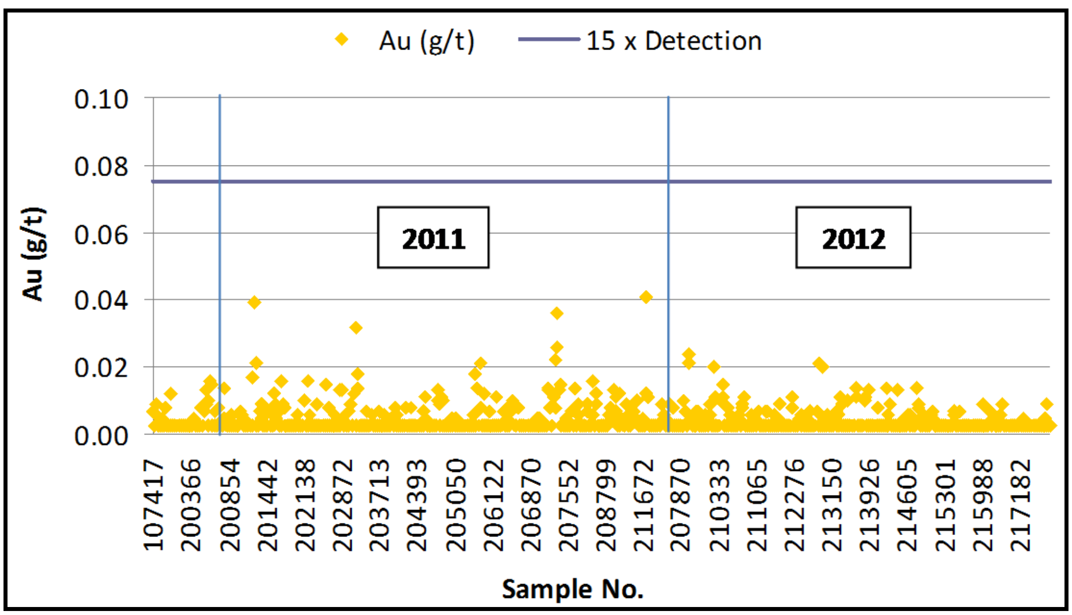
The apparent failure of sample number 203705 is thought to represent a sample numbering error and probably represents a standard reference (i.e. NI 116).

Figure 11.4 Silver Blank Performance



The apparent failure of three samples may be associated with the sample label mix ups that were previously discussed.

Figure 11.5 Gold Blank Performance



11.9 2010, 2011 AND 2012 TINTINA SRM PERFORMANCE

For their 2010, 2011 and 2012 drilling programs, Tintina submitted 813 SRMs along with their HQ-NQ diamond core hole samples that were submitted to ALS Chemex. The certified SRMs were purchased from WCM Sales Ltd. (also known as WCM Minerals) from Burnaby, BC. The insertion rate of the SRMs was approximately one SRM per 20 drillhole samples. Table 11.1 summarizes the expected values for various metals for each of the SRMs along with their associated standard deviations. The expected values and standard deviations were derived by round robin assaying.

In 2010, the commercial standard PB 134, which was purchased from WCM Minerals, was inserted into the sample stream five times for the samples used by the author for estimating resources for the Johnny Lee UZ (Lechner 2010). This same standard along with three other WCM Minerals standards, (PB 129, PB 131, CU 145, CU 182, and NI 116) were used by Tintina for their 2011 and 2012 drill campaign.

Figure 11.6 and Figure 11.7 track the performance of SRM CU 145 that was assayed by ALS Chemex for copper and silver, respectively.

Figure 11.6 SRM CU 145 – Copper Performance

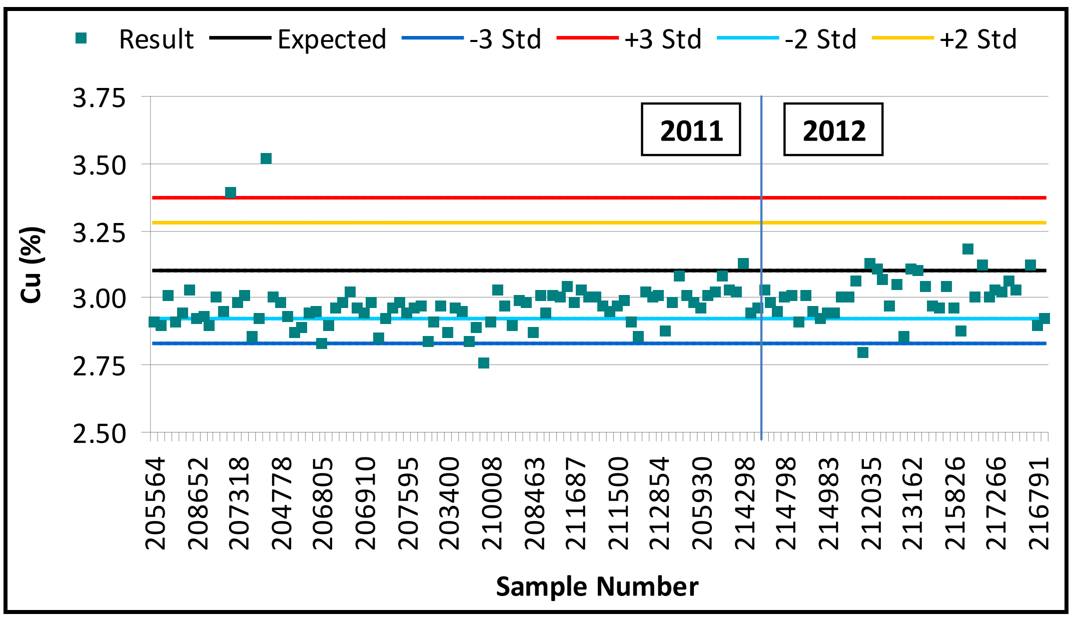


Figure 11.7 SRM CU 145 – Silver Performance

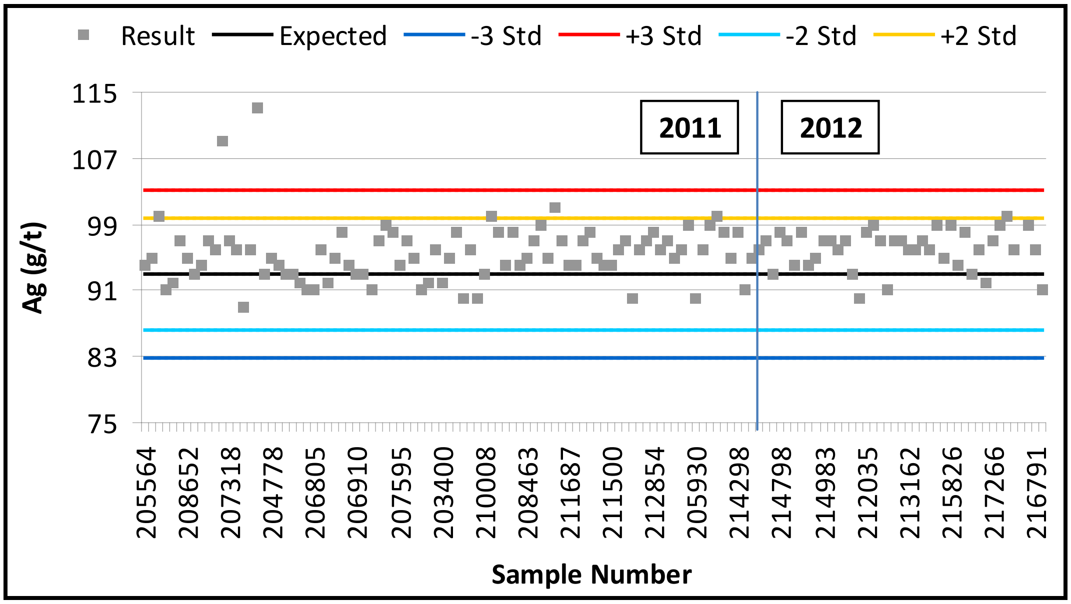


Figure 11.8 and Figure 11.9 track the performance of SRM CU 182 that was assayed by ALS Chemex for copper and silver, respectively.

Figure 11.8 SRM CU 182 – Copper Performance

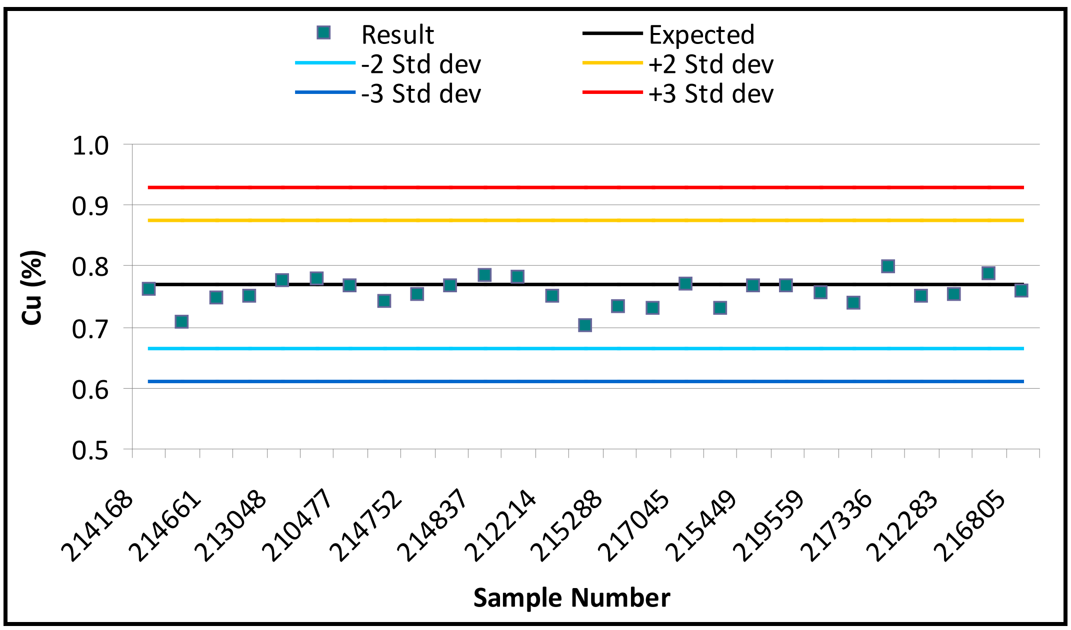


Figure 11.9 SRM CU 182 – Silver Performance

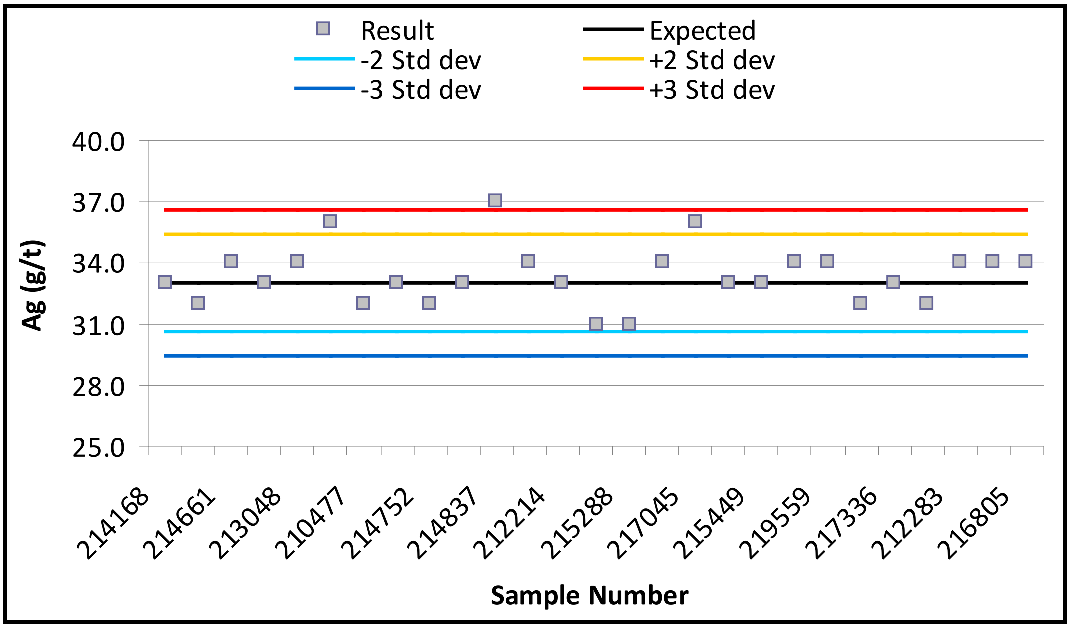


Figure 11.10 and Figure 11.11 track the performance of SRM NI 116 that was assayed by ALS Chemex for copper and cobalt, respectively.

Figure 11.10 SRM NI 118 – Copper Performance

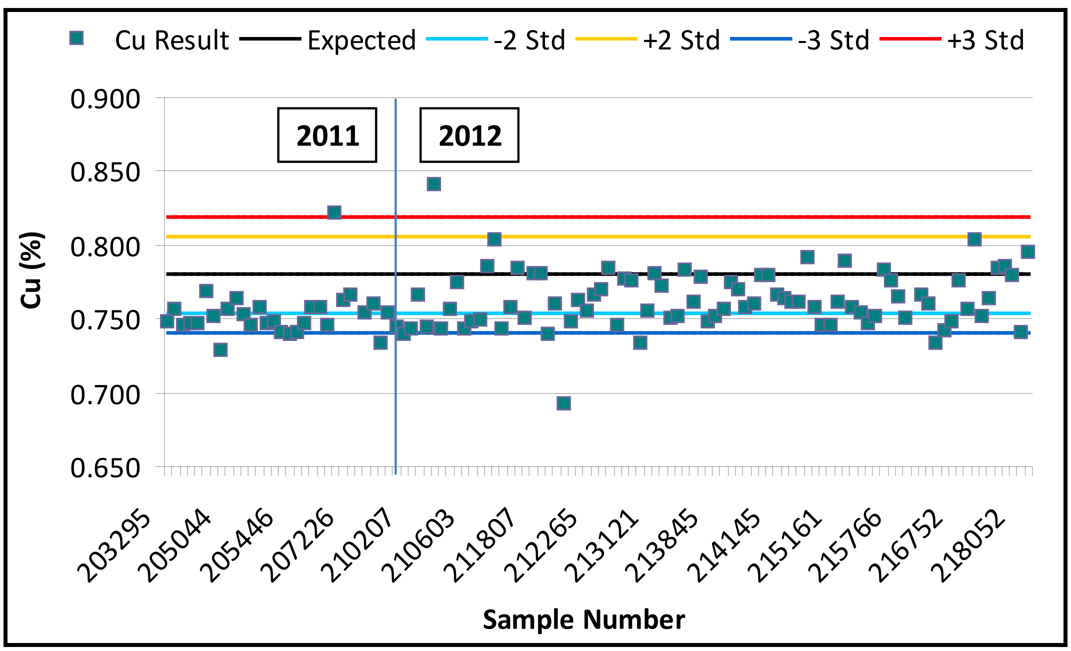


Figure 11.11 SRM NI 118 – Cobalt Performance

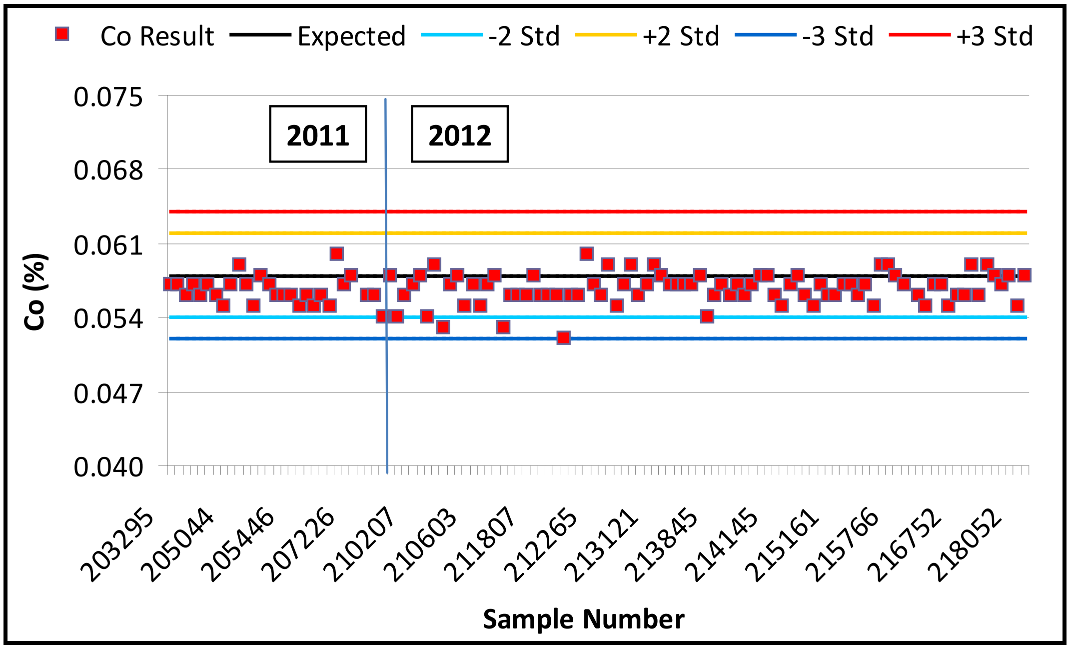


Figure 11.12 to Figure 11.15 track the performance of SRM PB 129 that was assayed by ALS Chemex for copper, silver, lead, and zinc, respectively.

Figure 11.12 SRM PB 129 – Copper Performance

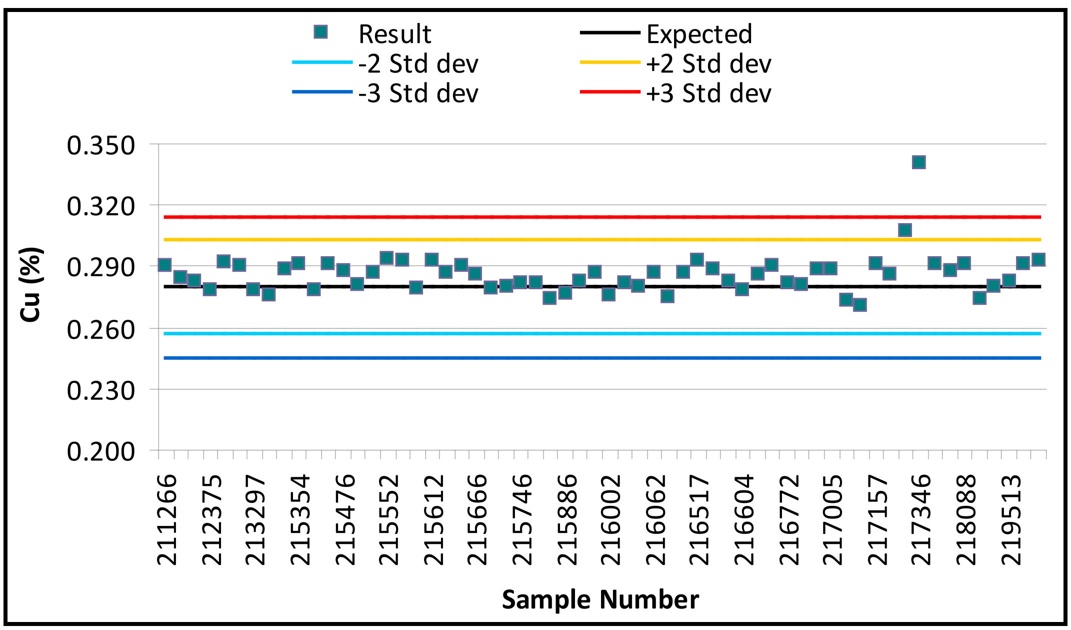


Figure 11.13 SRM PB 129 – Silver Performance

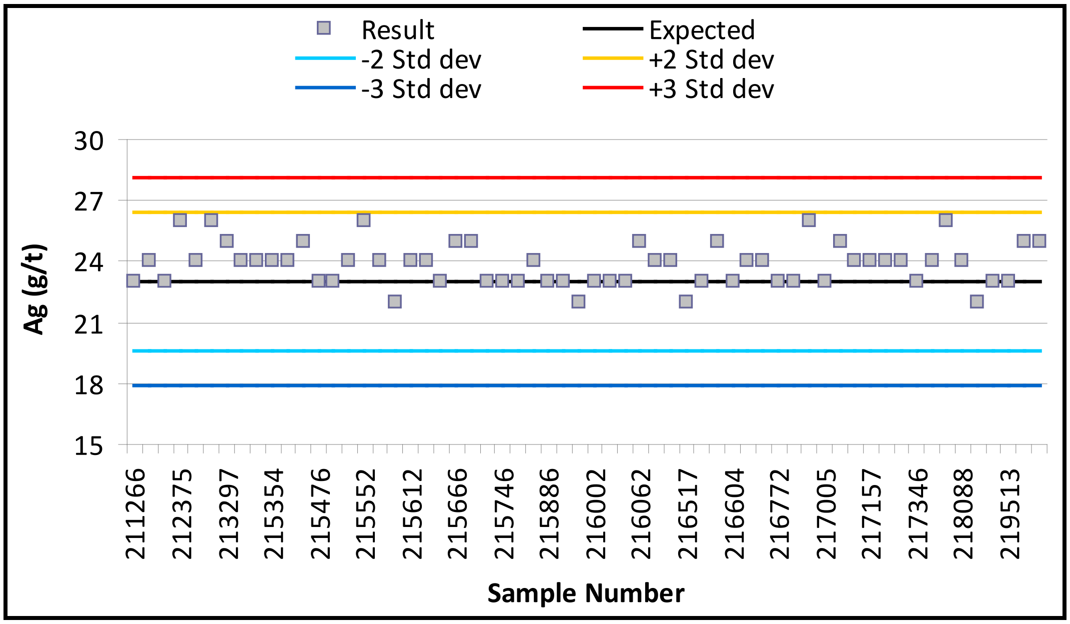


Figure 11.14 SRM PB 129 – Lead Performance

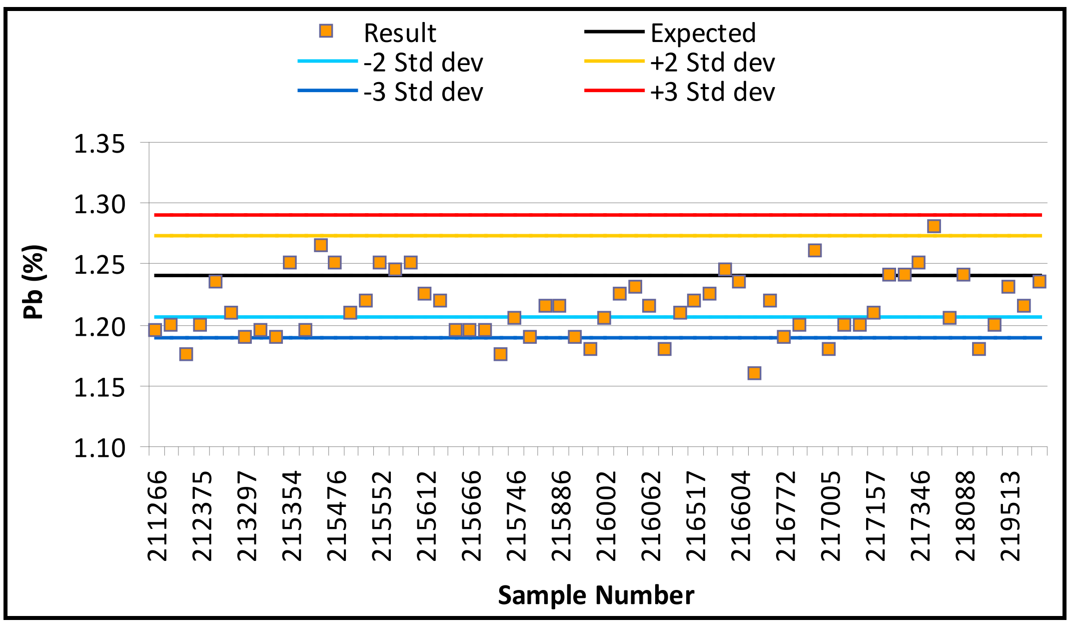


Figure 11.15 SRM PB 129 – Zinc Performance

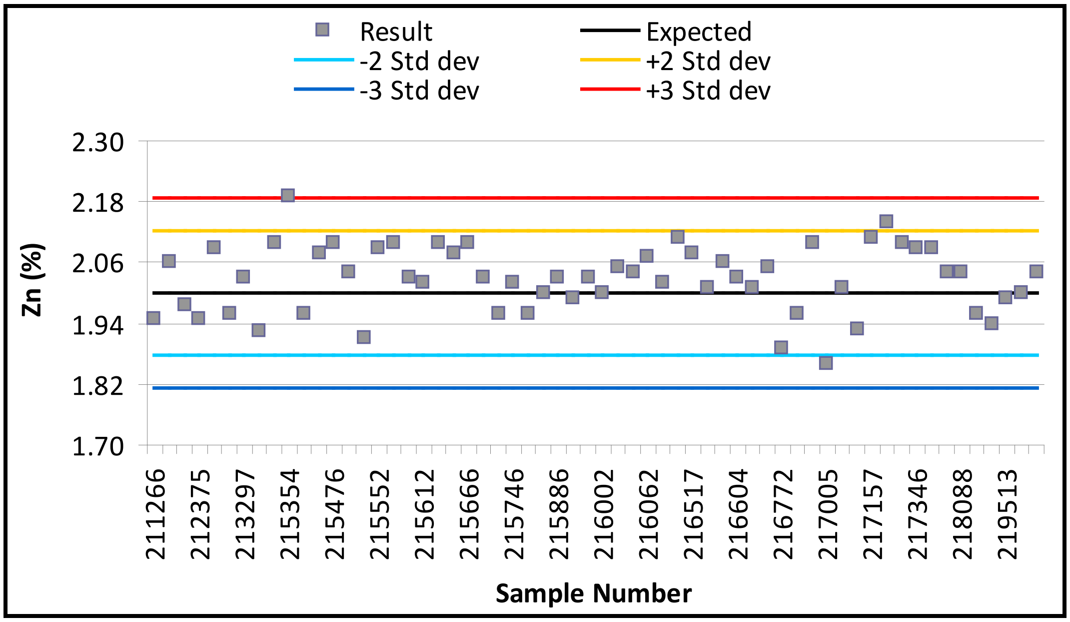


Figure 11.16 to Figure 11.18 track the performance of SRM PB 131 that was assayed by ALS Chemex for copper, lead, and zinc, respectively.

Figure 11.16 SRM PB 131 – Copper Performance

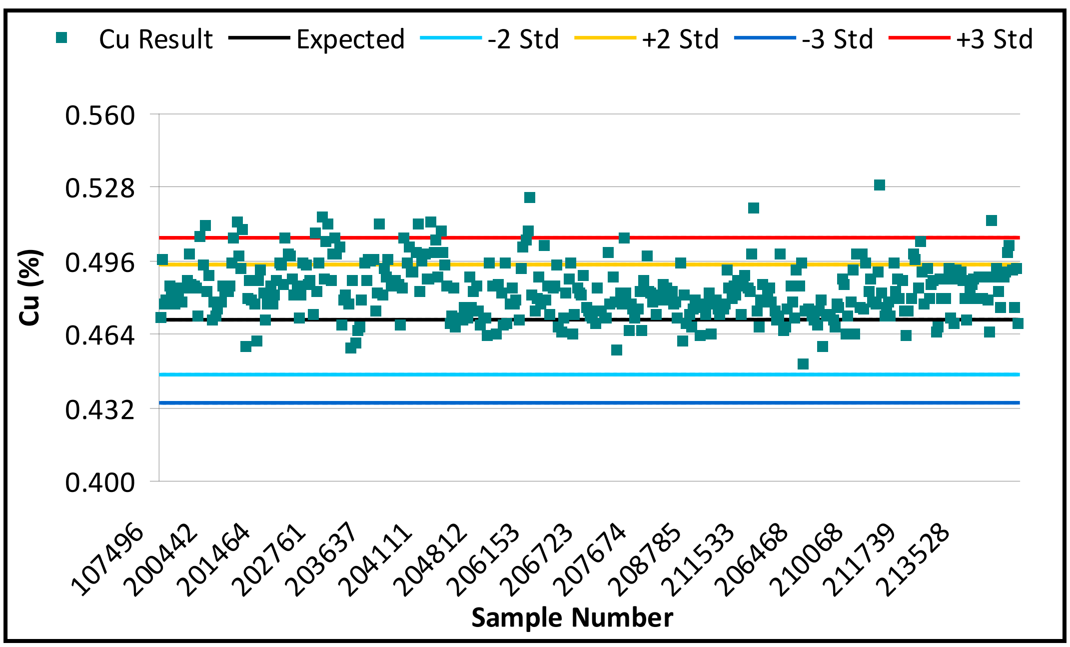


Figure 11.17 SRM PB 131 – Lead Performance

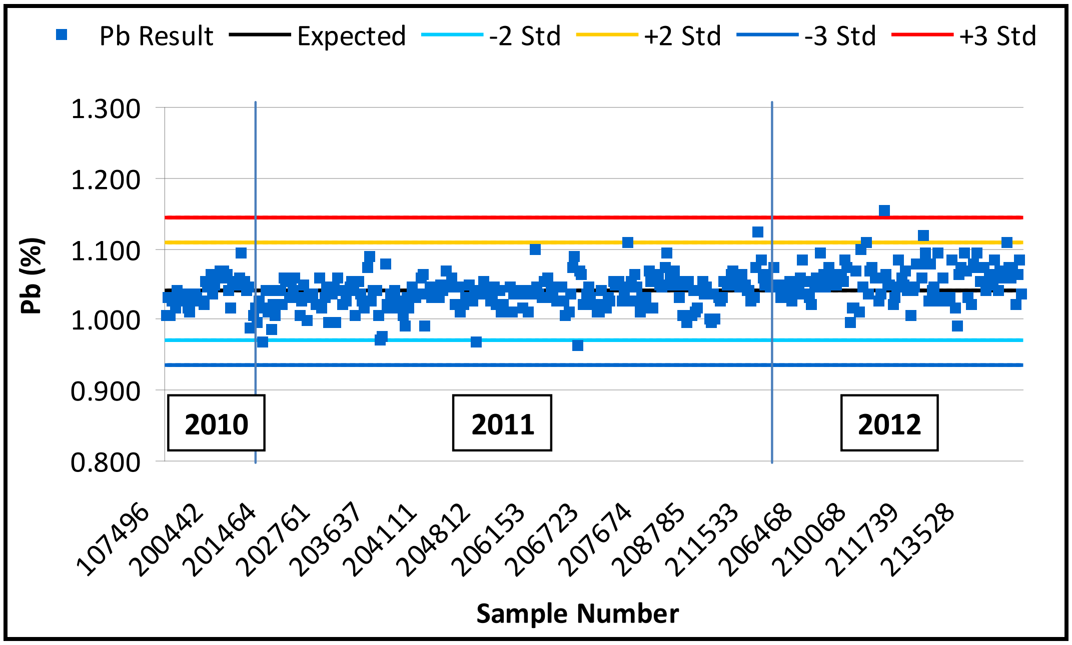


Figure 11.18 SRM PB 131 – Zinc Performance

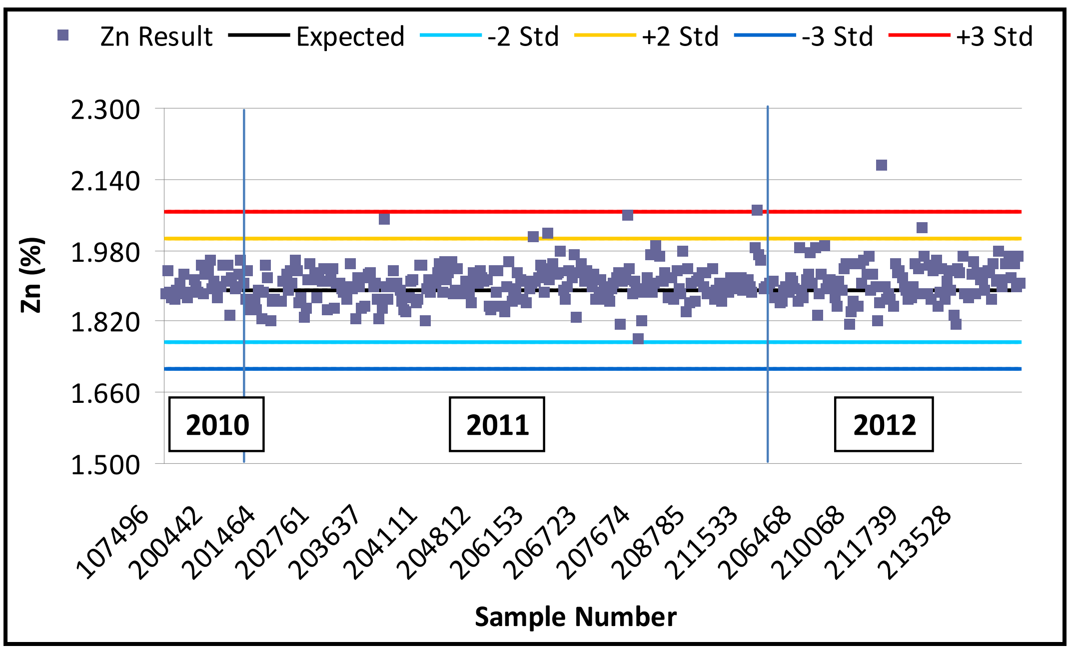


Figure 11.19 to Figure 11.22 track the performance of SRM PB 134 that was assayed by ALS Chemex for copper, silver, lead, and zinc, respectively.

Figure 11.19 SRM PB 134 – Copper Performance

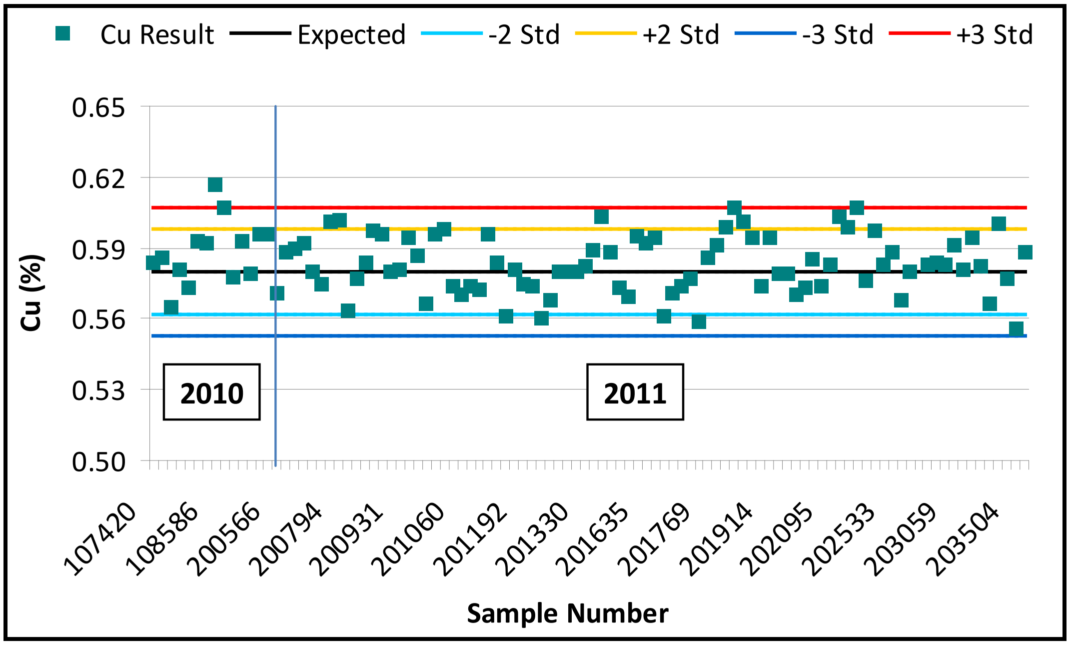


Figure 11.20 SRM PB 134 – Silver Performance

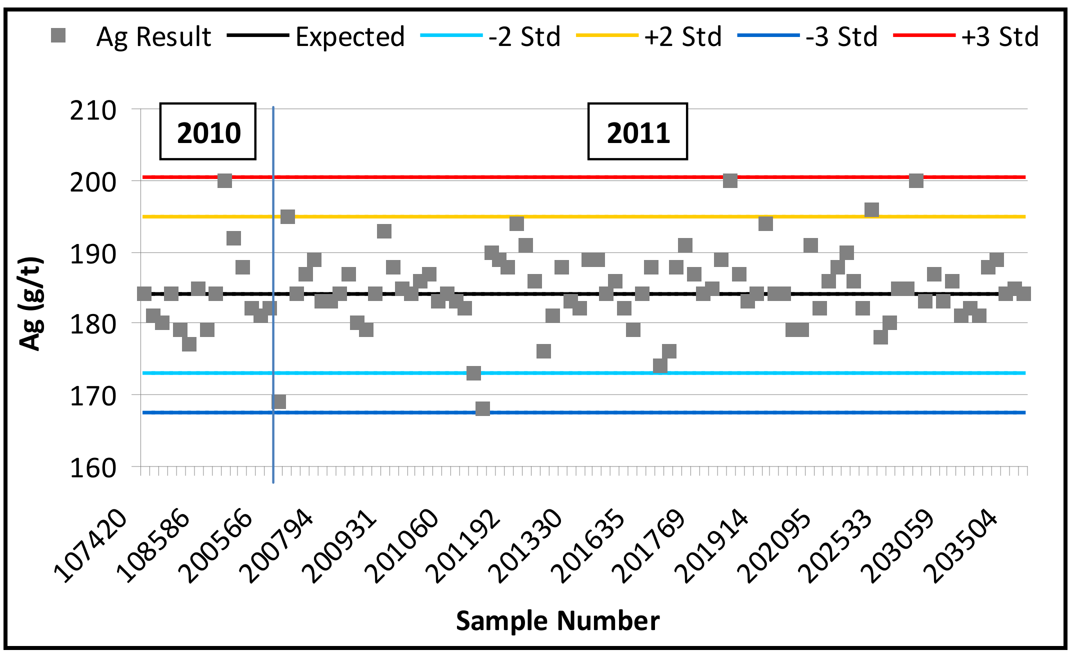


Figure 11.21 SRM PB 134 – Lead Performance

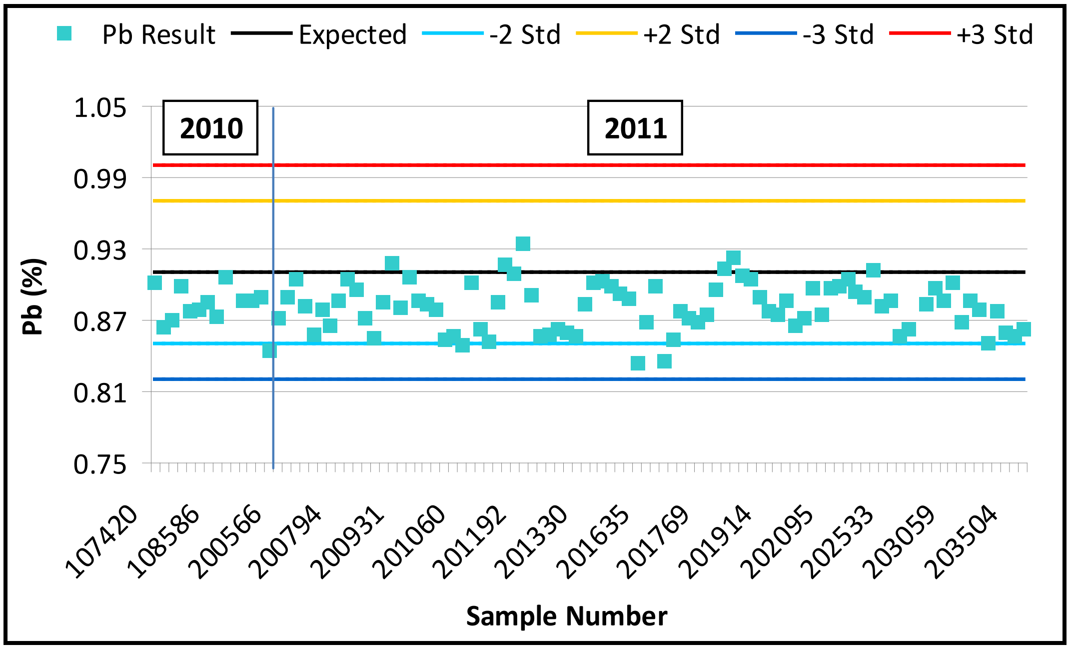
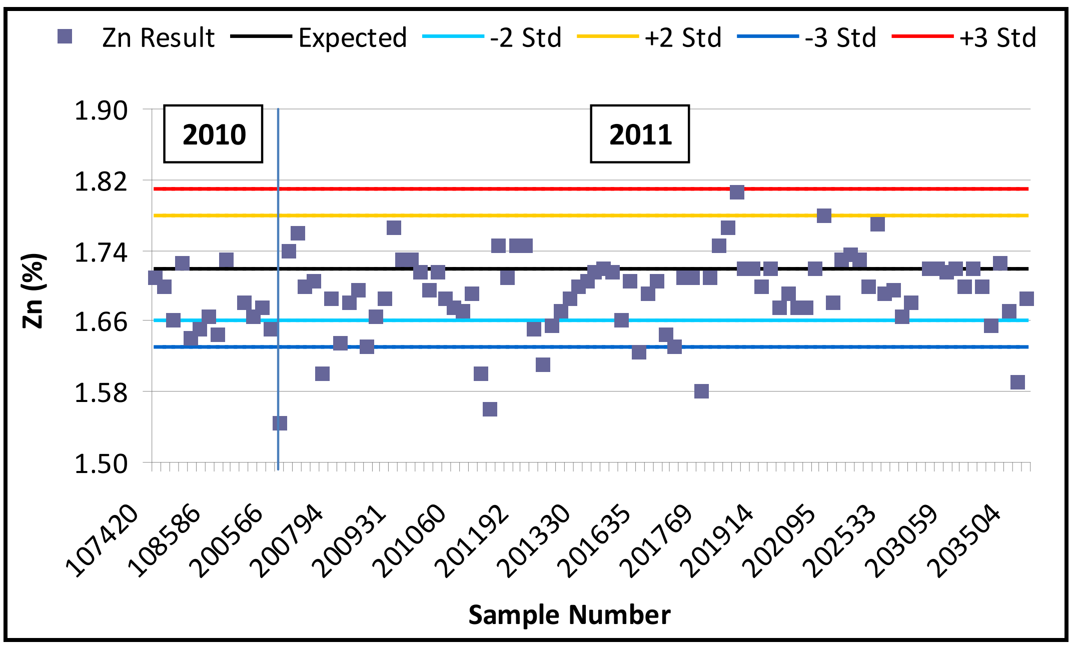


Figure 11.22 SRM PB 134 – Zinc Performance



11.10 2010 TO 2011 TINTINA DUPLICATE SAMPLES

As a part of their QA/QC program, Tintina notified ALS Chemex to create a duplicate pulp from the coarse reject sample at an approximate frequency of one duplicate pulp for every 30 drill core samples. For their 2010 to 2012 drilling campaigns, 803 duplicate pulp samples were prepared and assayed by ALS Chemex (61 in 2010, 388 in 2011 and 354 in 2012). Table 11.3 summarizes basic descriptive statistics for the original and duplicate pulp for those 803 sample pairs.

Table 11.3 Original-Duplicate Sample Comparison

Sample	Minimum	Maximum	Mean	Median	Q1	Q3	SD	CV
Copper Duplicates								
Original Cu (%)	0.001	20.200	0.815	0.028	0.004	0.686	2.87	3.52
Duplicate Cu (%)	0.001	20.500	0.819	0.027	0.003	0.706	2.87	3.51
Difference (%)	0.00	-1.46	-0.51	3.70	16.67	-2.90	-0.27	0.24
Cobalt Duplicates								
Original Co (ppm)	5.000	8540.000	252.825	60.000	10.000	340.000	711.33	2.81
Duplicate Co (ppm)	5.000	8540.000	253.613	60.000	10.000	330.000	714.36	2.82
Difference (%)	0.00	0.00	-0.31	0.00	0.00	3.03	-0.42	-0.11
Silver Duplicates								
Original Ag (g/t)	0.500	115.000	10.686	6.000	1.000	16.000	14.90	1.39
Duplicate Ag (g/t)	0.500	114.000	10.562	6.000	1.000	16.000	14.84	1.40
Difference (%)	0.00	0.88	1.17	0.00	0.00	0.00	0.38	-0.78
Gold Duplicates								
Original Au (g/t)	0.003	1.630	0.0204	0.0025	0.0025	0.0080	0.12	5.87
Duplicate Au (g/t)	0.003	1.625	0.0196	0.0025	0.0025	0.0080	0.12	6.02
Difference (%)	0.00	0.31	4.17	0.00	0.00	0.00	1.48	-2.58

Notes: SD = Standard Deviation; CV = Coefficient of Varation.

The original copper and cobalt pulps tended to assay slightly below the duplicate pulp assays (e.g. the mean original copper pulp grade was 0.5% lower than the duplicate copper pulp). The mean grade statistics are influenced by several high-grade samples. Quantile-quantile (QQ) plots were drawn to compare the original pulp result (X-axis) with the duplicate pulp result (Y-axis). Figure 11.23 through Figure 11.26 shows QQ plots based on the original and duplicate pulp results for copper, cobalt, silver, and gold, respectively.

Figure 11.23 Copper QQ Plot – Original versus Duplicate Pulps

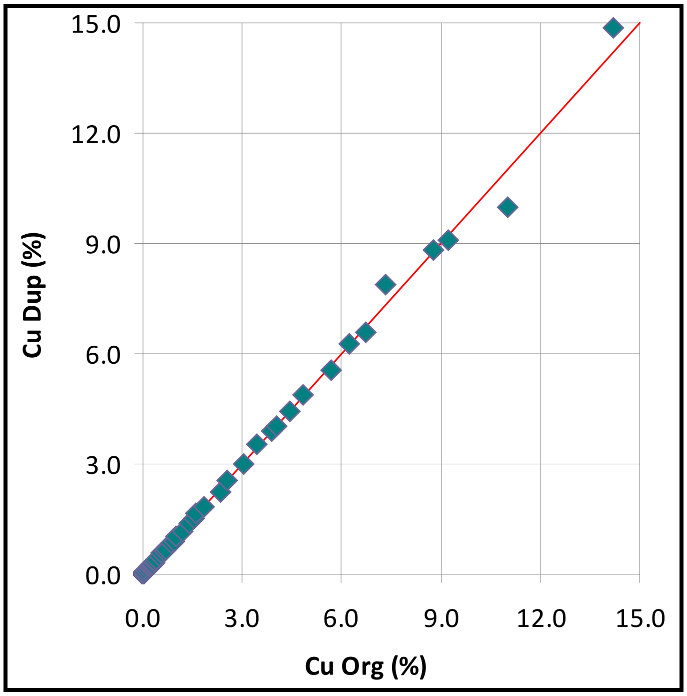


Figure 11.24 Cobalt QQ Plot – Original versus Duplicate Pulps

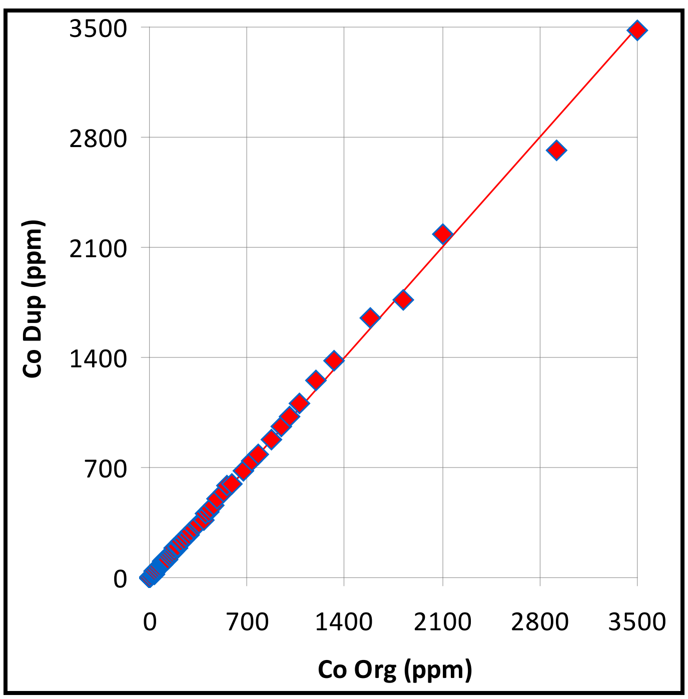


Figure 11.25 Silver QQ Plot – Original versus Duplicate Pulps

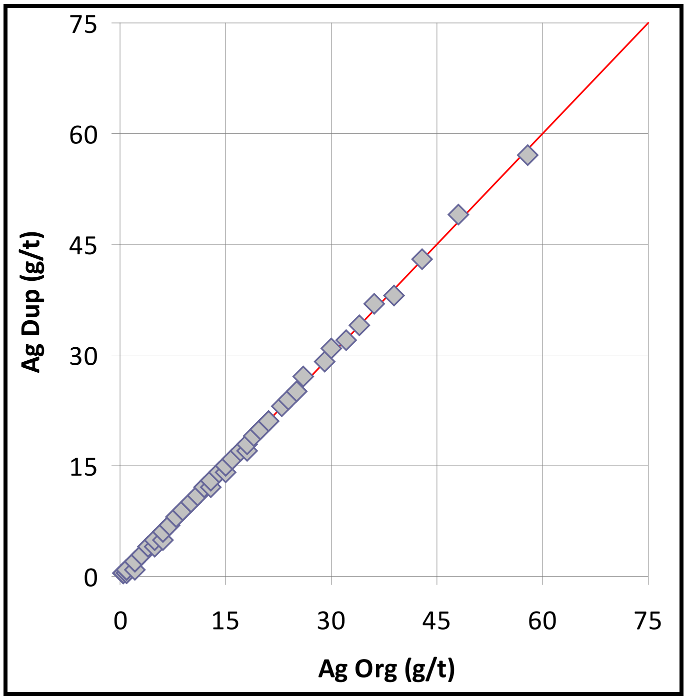
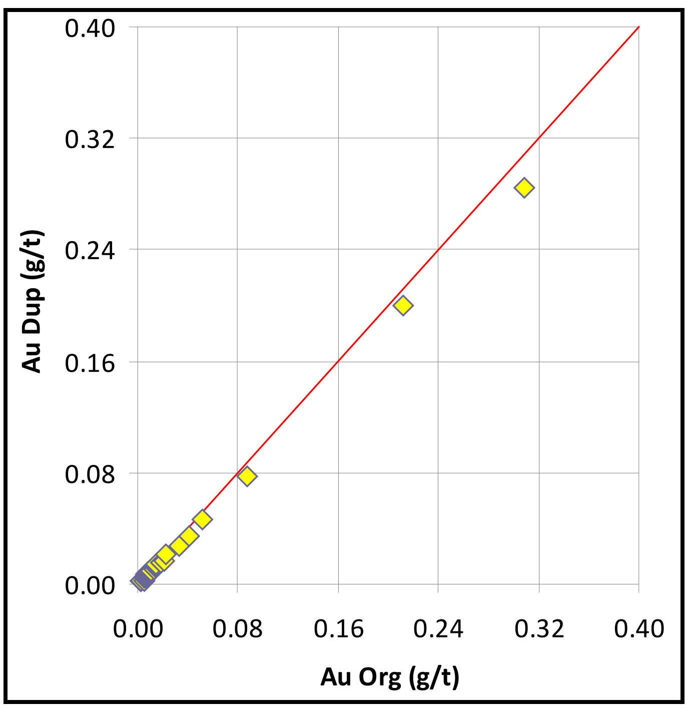


Figure 11.26 Gold QQ Plot – Original versus Duplicate Pulps



11.11 2010 TO 2011 TINTINA CHECK ASSAYS

Two different types of "check assays" were completed for Tintina's 2010 and 2011 drilling campaigns. The first set consists of HQ and NQ core that was prepped and assayed by ALS Chemex. Tintina requested that ALS Chemex create a split from select pulps, which were then submitted to Inspectorate and assayed for a variety of metals. Seventy pulps were re-assayed by Inspectorate.

The second set of check assays were prepared by Inspectorate and re-assayed by ALS Chemex. These samples represented PQ core from metallurgical holes that were submitted to Inspectorate for metallurgical recovery studies. Each of the submitted core intervals were assayed by Inspectorate prior to compositing for metallurgical test work. Tintina requested that Inspectorate split out a representative portion of the initial pulp which was then submitted to ALS Chemex for re-assaying. Sixty-three pulps were re-assayed by ALS Chemex.

Table 11.4 summarizes basic descriptive statistics for the 70 same pulp assays that were originally assayed by ALS Chemex and later assayed by Inspectorate.

Table 11.4 ALS Chemex versus Inspectorate Same Pulp Assay Comparison

Parameter	Cu (%)		Co (ppm)		Ag (ppm)	
	ALS Chemex	Inspectorate	ALS Chemex	Inspectorate	ALS Chemex	Inspectorate
Count	70	70	70	70	70	70
Minimum	0.007	0.008	5	1	4	3
Maximum	8.480	8.470	4,870	4,291	82	91
Mean	3.430	3.169	1065	1001	21	19
Standard Deviation	2.303	2.286	777	740	16	15
Coefficient of Variation	0.67	0.72	0.73	0.74	0.73	0.82
Mean Grade Difference	8%		6%		16%	

The mean ALS Chemex copper, cobalt, and silver grades are 8%, 6%, and 16% higher than Inspectorate. There is no apparent significant bias between the two laboratories as illustrated by XY scatter graphs shown in Figure 11.27 through Figure 11.29 for copper, cobalt, and silver, respectively.

Figure 11.27 and Figure 11.28 suggest that the difference in mean copper and cobalt grades may be influenced by a handful of high-grade samples. Other factors could be associated with how well the initial ALS Chemex pulps were homogenized prior to creating a split for Inspectorate. Differences in acid digestions and acid temperatures between the two laboratories could also add to the mean grade differences. RMI recommends that for future drilling campaigns that Tintina generate more pulps from the coarse reject and submit them to another commercial laboratory. RMI also recommends that Tintina send coarse reject splits to a secondary laboratory so that they can prepare and assay their own independent sample.

Figure 11.27 ALS Chemex versus Inspectorate – Copper

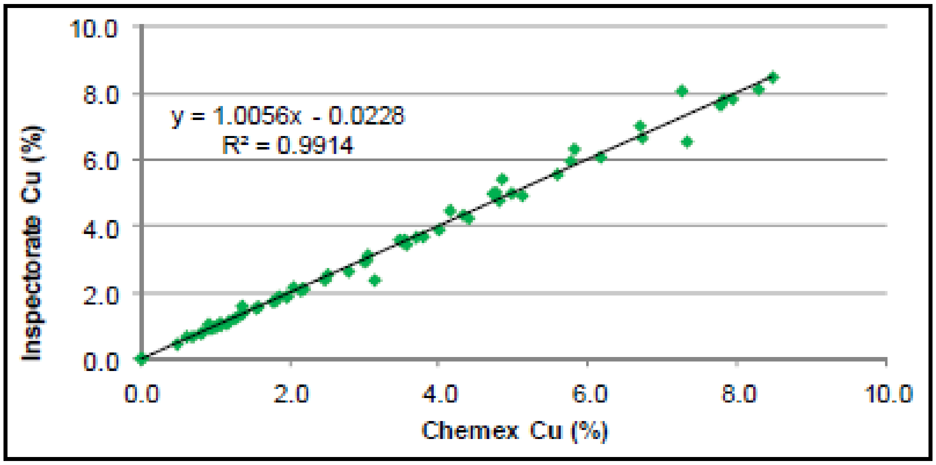


Figure 11.28 ALS Chemex versus Inspectorate – Cobalt

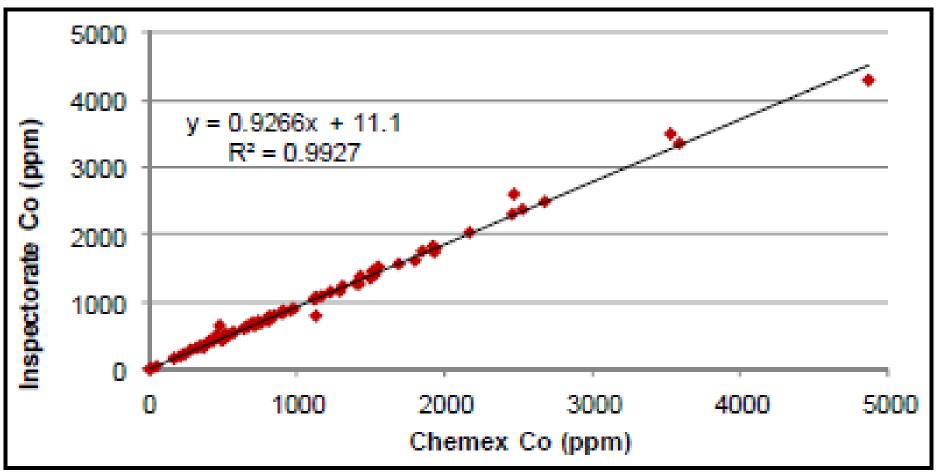
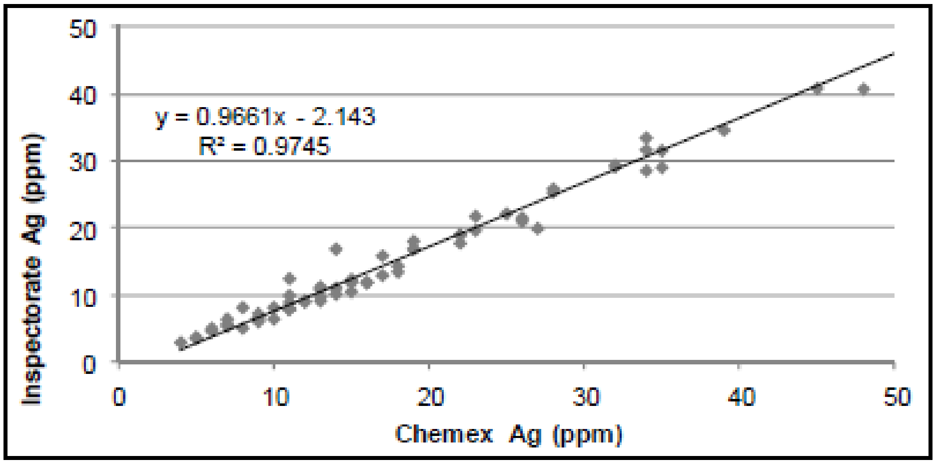


Figure 11.29 ALS Chemex versus Inspectorate – Silver



As previously mentioned, a total of 63 pulps that were prepared by Inspectorate from metallurgical core samples were re-assayed by ALS Chemex. Table 11.5 summarizes basic descriptive statistics for those same pulp assays.

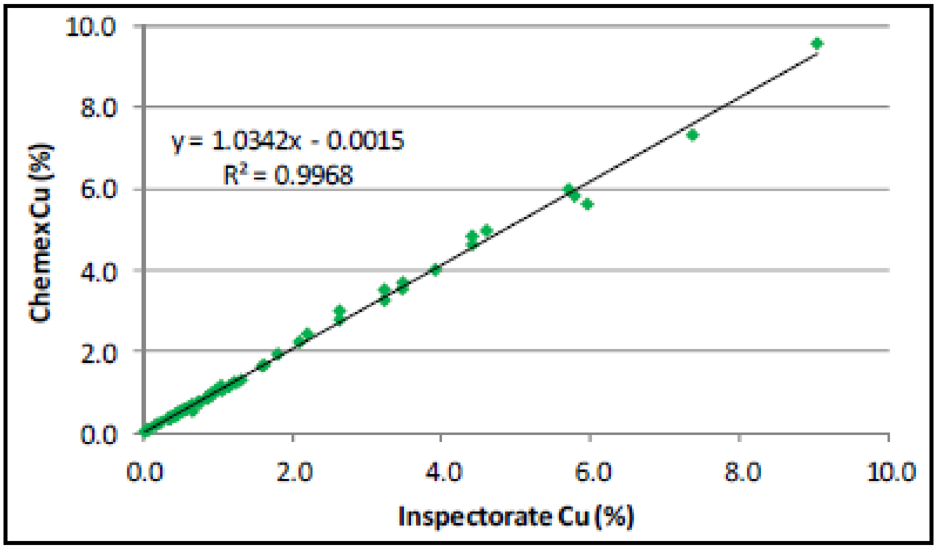
Table 11.5 Inspectorate versus Chemex Same Pulp Assay Comparison

Parameter	Cu (%)		Co (ppm)		Ag (ppm)	
	Inspectorate	ALS Chemex	Inspectorate	ALS Chemex	Inspectorate	ALS Chemex
Count	63	63	63	63	63	63
Minimum	0.024	0.030	40	40	2	2
Maximum	9.040	9.580	7,415	7,120	50	49
Mean	1.405	1.450	1,055	1,025	14	13
Standard Devision	1.930	1.999	1231	1185	8	8
Coefficient of Variation	1.37	1.38	1.17	1.16	0.59	0.63
Mean Grade Difference	-3%		3%		11%	

This comparison show less discrepancy between the two laboratories but the ALS Chemex copper grade is still higher (by 3%) than the Inspectorate grade. The mean Inspectorate cobalt and silver grades are higher than ALS Chemex.

Figure 11.30 is a XY scatter graph that compares the Inspectorate copper grade with ALS Chemex.

Figure 11.30 Inspectorate versus ALS Chemex – Cu



11.12 DISCUSSION

Based on the information available to RMI, CAI, UII, and BHP followed industry accepted procedures for sample preparation, analysis, and security. RMI highly recommends that Tintina continue trying to obtain assay certificates and QA/QC data from Teck and/or BHP for the older drilling data.

RMI believes that there are no material factors that could have affected the accuracy and reliability of the results from the various drilling campaigns. Core recovery tends to be very good except in rare cases of shearing within fault zones.

Based on twin hole comparisons (2010 Tintina data versus older drilling data discussed in Section 12.2) and subsequent QA/QC results from Tintina's 2011 and 2012 drilling campaigns, it is RMI's opinion that the Black Butte data are suitable for estimating mineral resources.

12.0 DATA VERIFICATION

12.1 PRE-TINTINA DATA

Tintina obtained a copy of the electronic drillhole database along with other information (e.g. drillhole collar locations, down-hole surveys, and various maps) from the Belt Research Center located in Missoula, Montana, which is managed by the University of Montana Geology Department. CAI donated this data to the Belt Research Center after they terminated their interest in the Project.

12.2 TINTINA CONFIRMATION DRILLING

For the resource estimates, RMI has stated that Tintina has not been able to verify the drilling data collected prior to 2010 because that data is not currently available. As discussed in Section 11.6, Tintina attempted to obtain copies of the historical data from Teck, who purchased Cominco a number of years ago.

In the absence of assay certificates and QA/QC results for the historical data, Tintina drilled six diamond core holes in 2010 twinning historical drilled holes. This twin hole program focused on verifying historical data that was collected from the Johnny Lee UZ. Table 12.1 compares four 2010 Tintina UZ intercepts with older UII intercepts. These comparisons show that half of the new holes have higher grades and thicknesses than the older intercepts and half have lower grades and thicknesses. Figure 12.1 shows the location of the first four 2010 Tintina confirmation drillholes.

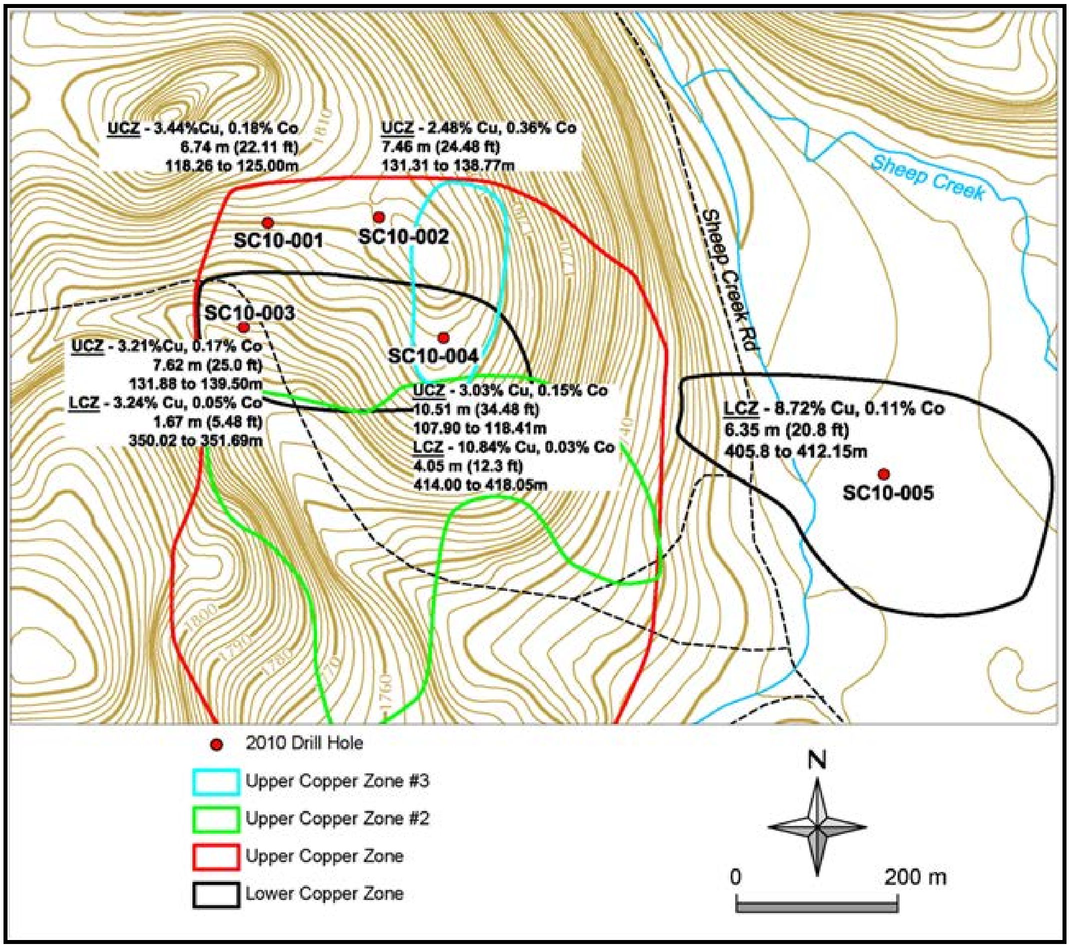
Table 12.1 Confirmation Hole Comparison

Twin Site Location	Company	Drillhole	Top Elevation (m)	Base Elevation (m)	Thickness (m)	Cu (%)	Co (%)
1	Tintina	SC10-001	118.26	125.00	6.74	3.44	0.18
	UII	SCC-19	115.21	123.75	8.54	3.05	0.18
	Difference (%)	N/A	3	1	-21	13	0
2	Tintina	SC10-002	131.31	138.77	7.46	2.48	0.36
	UII	SCC-23	132.59	140.51	7.92	3.57	0.54
	Difference (%)	N/A	-1	-1	-6	-31	-33
3	Tintina	SC10-003	131.88	139.50	7.62	3.21	0.17
	UII	SCC-17	130.76	137.46	6.70	2.76	0.19
	Difference (%)	N/A	1	1	14	16	-11

table continues…

Twin Site Location	Company	Drillhole	Top Elevation (m)	Base Elevation (m)	Thickness (m)	Cu (%)	Co (%)
4	Tintina	SC10-004	107.90	118.41	10.51	3.03	0.15
	UII	SCC-34	110.03	119.79	9.76	2.35	0.13
	Difference (%)	N/A	-2	-1	8	29	15

Figure 12.1 Tintina Confirmation Drilling Locations



Each new hole twinned an historic hole drilled through either or both the USZ and LSZ near Strawberry Butte in areas described as copper-cobalt resources by CAI. In each twinning, the historic collar was located as closely as possible either by physical location of the historic collar or by re-surveying the historic location. New holes were collared 3 m away from the historic location. SC10-001 was located 3 m south of the SCC-19 collar marker; SC10-002 was located 3 m south of the SCC-23 collar marker; SC10-003 was located 3 m south of the historic survey coordinates; SC10-004 was located 3 m south of the historic SCC-34 collar marker; and SC10-005 was located 3 m south of the historic survey coordinates.

Grades and grade x thicknesses for each of the confirmation twin holes are compared with the older holes in a series of down-hole copper grade histograms and cumulative grade-thickness plots in Figure 12.2 through Figure 12.5.

Figure 12.2 SC10-001 versus SCC-19

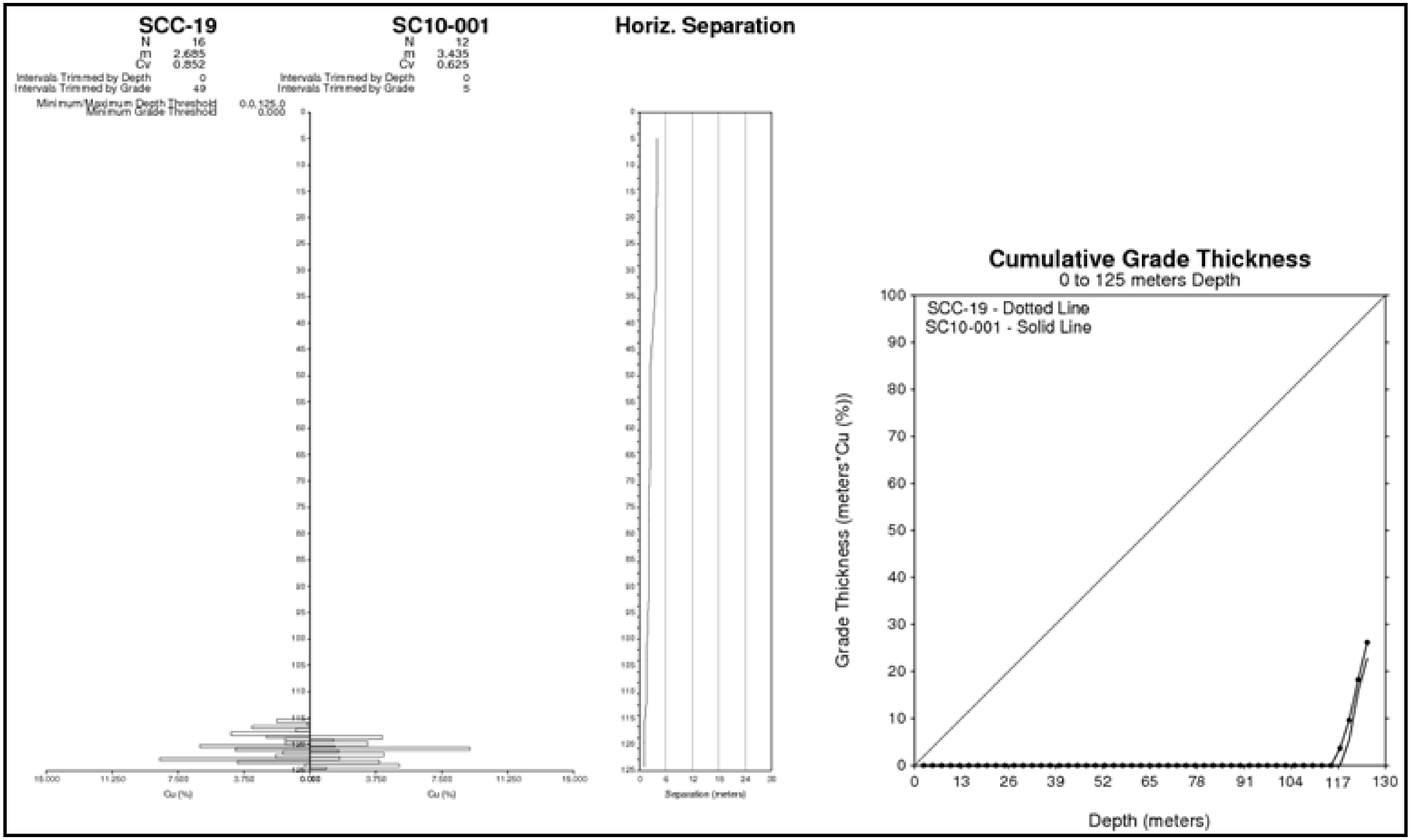


Figure 12.3 SC10-002 versus SCC-23

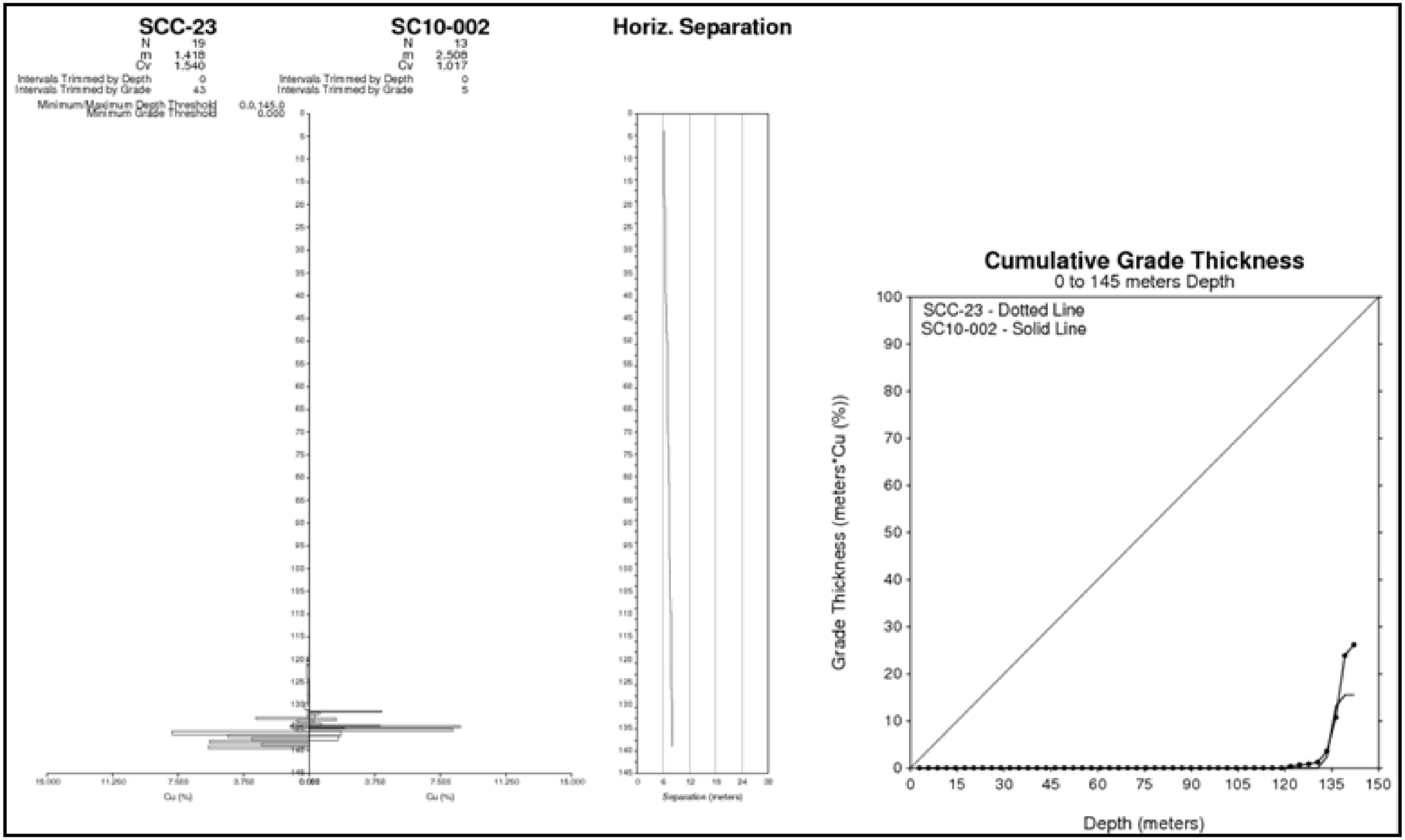


Figure 12.4 SC10-003 versus SCC-17

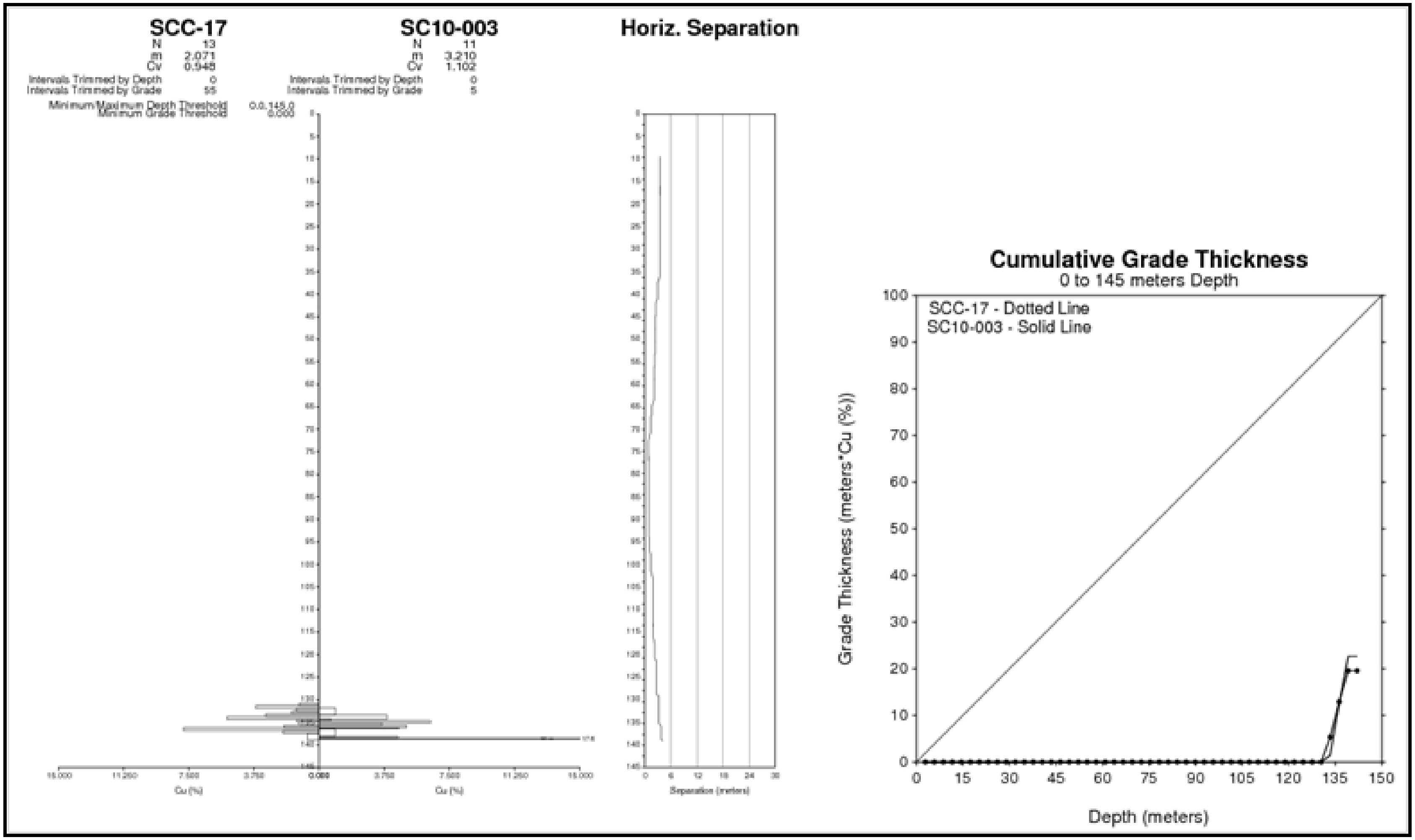
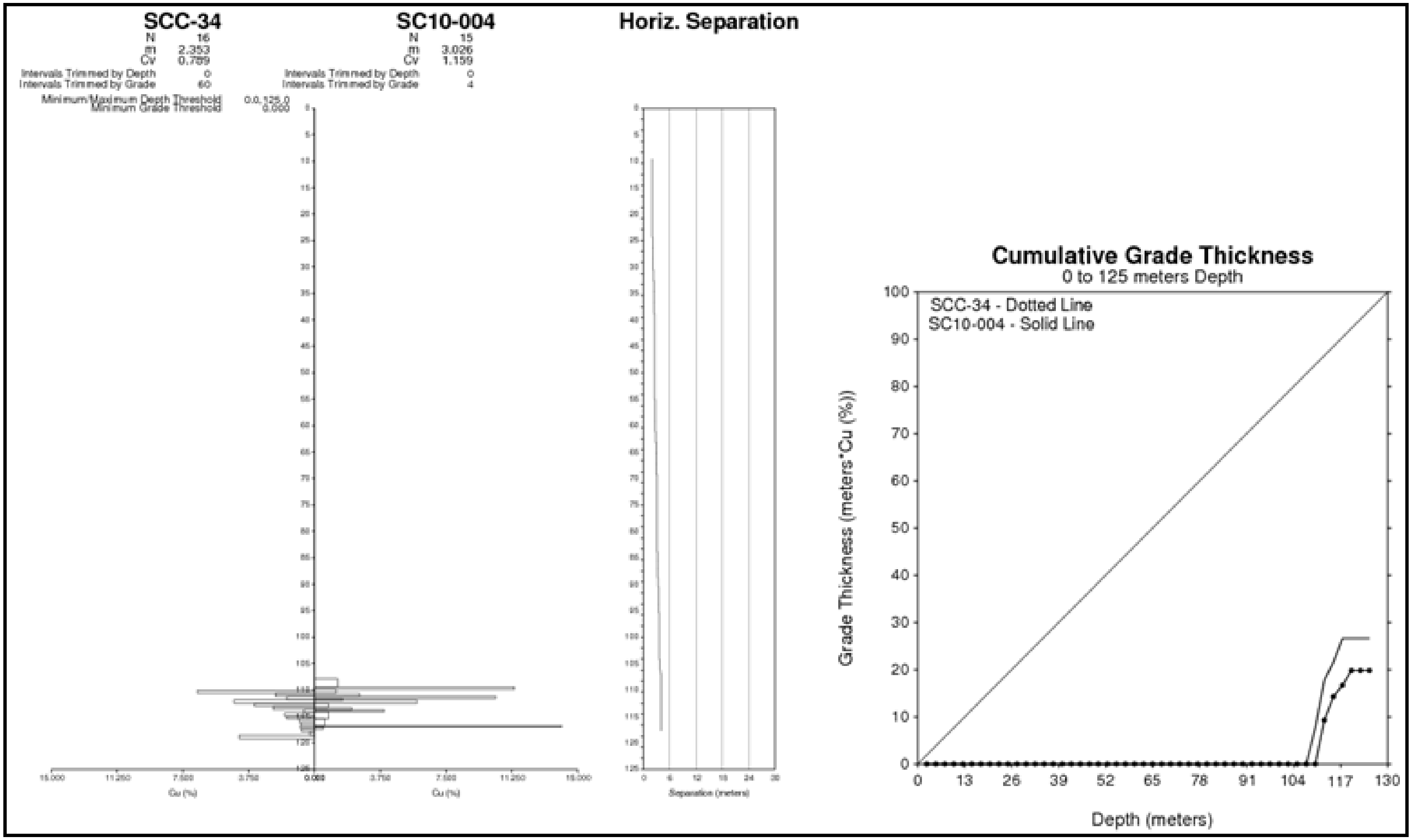


Figure 12.5 SC10-004 versus SCC-34



Tintina has drilled a significant number of holes in the Johnny Lee UZ, LZ, and MZ since their 2010 confirmation drilling program. Figure 10.1 shows the location of the Johnny Lee UZ and LZ drilling by company. Table 12.2 summarizes drillhole intersections from the Johnny Lee UZ by company.

Table 12.2 Johnny Lee UZ Drillhole Intersections

Company	No. of Drillholes	No. of 1 m Composites	Total (m)	Average Length (m)	% of Data	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)
CAI	5	28.0	27.5	5.51	2	2.35	0.13	0.01	13.5
UII	14	80.0	80.2	5.73	6	2.09	0.15	0.01	9.2
BHP	6	55.0	55.9	9.32	4	1.71	0.08	0.04	14.7
Tintina	112	1,237.0	1,234.5	11.02	88	2.02	0.09	0.01	16.5
CAI+UII+BHP	25	1,63.0	163.6	6.54	12	2.00	0.12	0.02	11.8
Total/Average	**137**	**1,400.0**	**1,398.1**	**10.21**	**100**	**2.02**	**0.09**	**0.01**	**15.9**

The data in Table 12.2 show that approximately 90% of the total UZ data have been collected by Tintina since their entry into the district in 2010. Tintina's drilling data shows an appreciably average thickness than the historic data but that thickness is highly influenced by the discovery of a thicker zone of mineralization at the south end of the zone by Tintina. The average copper grade for the UZ is nearly identical when Tintina's assay data is compared with the historical data.

Table 12.3 compares historical drilling within the Johnny Lee LZ with Tintina's drilling data. Figure 10.1 shows the location of the Johnny Lee UZ and LZ drilling by company.

Table 12.3 Johnny Lee LZ Drillhole Intersections

Company	No. of Drillholes	No. of 1 m Composites	Total (m)	Average Length (m)	% of Data	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)
CAI	9	30.0	29.5	3.28	15	5.81	0.02	0.30	6.7
UII	4	15.0	15.3	3.84	8	5.54	0.07	0.30	4.7
BHP	1	12.0	12.4	12.41	6	5.71	0.03	0.27	2.4
Tintina	19	136.0	134.9	7.10	70	5.74	0.04	0.34	4.3
CAI+UII+BHP	14	57.0	57.3	4.09	30	5.72	0.04	0.29	5.2
Total/Average	**33**	**193.0**	**192.2**	**5.82**	**100**	**5.74**	**0.04**	**0.32**	**4.6**

Tintina's drilling data in the LZ represents 70% of the total data and like the UZ, the average copper grade compares very well between the historic and Tintina drilling (i.e. 5.72% versus 5.74%, respectively). Like the UZ, Tintina's drilling data shows the LZ to be thicker than the historic data.

Table 12.4 compares historical drilling within the Lowry MZ with Tintina's drilling data. Figure 10.2 shows the location of the Lowry MZ drilling by company.

Table 12.4 Lowry MZ Drillhole Intersections

Company	No. of Drillholes	No. 2.5 m Composites	Total (m)	Average Length (m)	% of Data	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)
Tintina	25	273.0	686.7	27.47	79	1.88	0.08	0.005	12.88
CAI	4	71.0	178.3	44.58	21	1.40	0.07	0.031	10.80
Total/Average	**29**	**344.0**	**865.0**	**29.83**	**100**	**1.40**	**0.07**	**0.031**	**10.80**

Tintina's drilling data in the MZ represents about 80% of the total data. Unlike the Johnny Lee UZ and LZ comparisons, on average, the Tintina drilling shows the MZ to be thinner but higher grade than the historical data. The MZ is structurally more complex than the Johnny Lee UZ and LZ so drillhole location can highly influence thickness and grade comparisons.

RMI paired historical and Tintina drilling within the Johnny Lee UZ and LZ along with the Lowry MZ to compare how well copper grades compare. Table 12.5 compares the paired data using a maximum separation distance of 50 m.

Table 12.5 Copper Grade Comparison for Spatially Paired Data

Mineralized Zone	Historic Drilling				Tintina Drilling				% Grade Difference
	UCZ (m)	Cu (%)	Standard Deviation	CV	UCZ (m)	Cu (%)	Standard Deviation	CV	
Johnny Lee UZ	40.0	2.14	1.26	0.59	40.0	2.36	1.63	0.69	10
Johnny Lee LZ	6.4	8.18	3.56	0.44	6.0	8.17	4.81	0.59	0
Lowry MZ	45.0	2.91	1.95	0.67	43.1	2.67	2.21	0.83	-8

Note: CV = Coefficient of Variation.

The data in Table 12.5 show that the historical data compares reasonably well with the Tintina data (±10%) for data within 50 m of one another. The Tintina data shows a higher standard deviation and coefficient of variation than the historical data.

Quantile-Quantile (QQ) plots comparing historical and Tintina copper grades within the the Johnny Lee and Lowry mineralized zones using spatially paired data with a maximum separation distance of 100 m. That separation distance was selected to generate sufficient pairs for plotting purposes. Figure 12.6, Figure 12.7, and Figure 12.8 show the QQ plots for the Johnny Lee UZ, Johnny Lee LZ, and Lowry MZ, respectively.

Figure 12.6 QQ Plot – Historical versus Tintina Copper Grades – Johnny Lee UZ

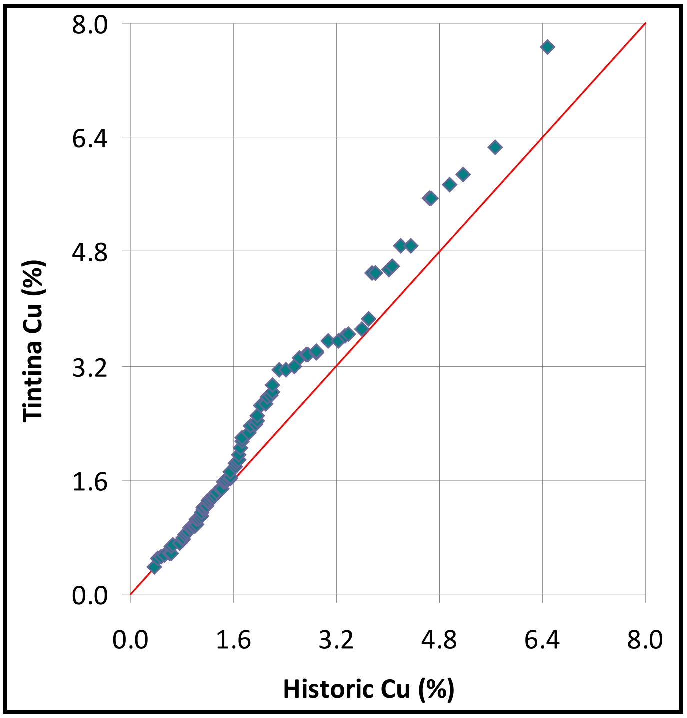


Figure 12.7 QQ Plot – Historical versus Tintina Copper Grades – Johnny Lee UZ

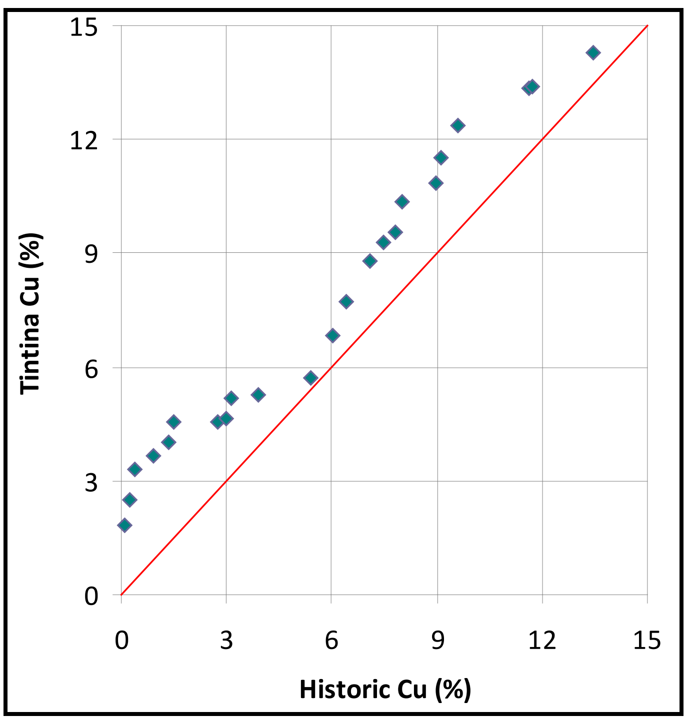
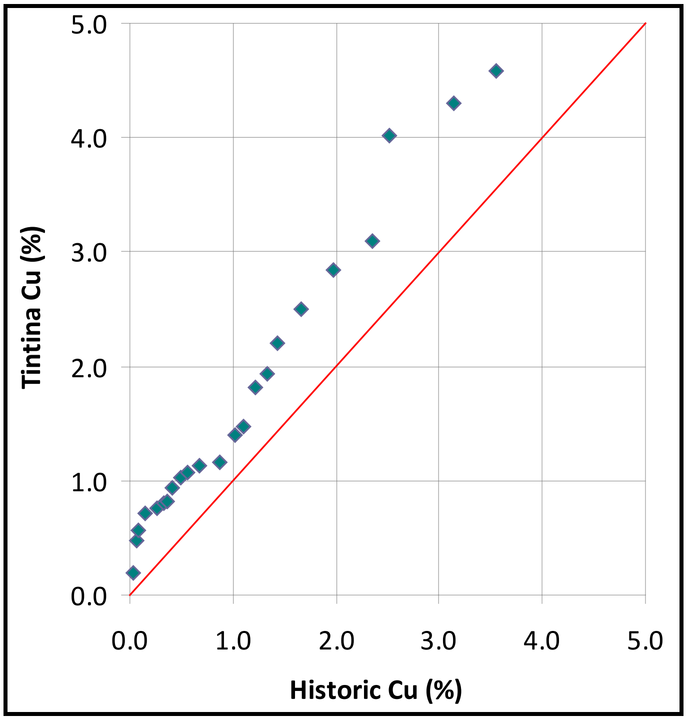


Figure 12.8 QQ Plot – Historical versus Tintina Copper Grades – Lowry MZ



The data shown in Figure 12.6, Figure 12.7, and Figure 12.8 reflect a similar trend of higher copper grades associated with the Tintina drilling data. It is RMI's opinion that this apparent bias is associated with differing analytical methods associated with the historical and modern data. When the historical data were analyzed, aqua regia was commonly used to digest the sample pulps. The Tintina data are based on a four acid digestion protocol used by ALS Chemex. That acid digestion method has been used by most commercial laboratories for a number of years now and is more apt to getting more of the copper into solution.

Based on various comparisons between historic and Tintina drilling data, it is RMI's opinion that the historical data are suitable to be used to estimate mineral resources. In general, the historic data tend to be slightly lower grade than the Tintina data and in the case of the Johnny Lee zone, tend to show the zones to be thinner than the Tintina data. The average thickness of the Johnny Lee zones is highly dependent upon hole location.

12.3 TINTINA ASSAY VERIFICATION

Since 2010, RMI has estimated and updated mineral resources for the Project several times. For the initial two estimates (Lechner 2010; 2011), RMI was able to verify 100% of the 2010 and 2011 Tintina electronic drillhole assays that were used for estimating Johnny Lee UZ and LZ mineral resources by comparing the records stored in their electronic database against signed ALS Chemex assay certificates. No errors were discovered.

For the updated mineral resource estimate of the Johnny Lee UZ (Lechner 2012), RMI randomly selected six 2011 drillholes that were used to define the Johnny Lee UZ (units 31 and 32). Signed assay certificates for drillholes SC11-017, SC11-029, SC11-035, SC11-063, SC11-064, and SC11-072 were compared against Tintina's electronic assay database. Only intervals from those holes that were used by the author for estimating mineral resources were checked. A total of 80 intervals from zones 31 and 32 were checked for copper, cobalt, gold, and silver (320 entries). No errors were discovered.

RMI has verified a significant portion of the 2012 Tintina drillhole assays that were used to estimate mineral resources for the Lowry MZ comprising 615 copper, cobalt, and silver assay records from 5 Tintina drillholes. No errors were discovered.

12.4 DISCUSSION

The original assay certificates and associated QA/QC data were unavailable for the historic drilling. However, the Tintina confirmation drilling results demonstrated that the older drilling is adequate for resource estimation purposes.

While on site, RMI made some random checks of down-hole survey records from the drillers against the electronic database and discovered no discrepancies. RMI has completed a representative review of Tintina's 2010, and 2011, and 2012 drilling campaigns and has been able to verify that their assay database is accurate. The QA/QC results demonstrate that the assays are representative and reproducible.

In RMI's opinion, the drillhole data that were used to estimate mineral resources for the Black Butte UZ, LZ, and MZ are adequate and reasonable.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Tintina contracted Arthur H. Winckers, P.Eng., to direct the various preliminary metallurgical test work programs to determine the flotation response of composite samples from the Johnny Lee UZ and LZ. The following sections pertain to work directed by Mr. Winckers with regards to the metallurgy of the Johnny Lee UZ and LZ.

13.1 MINERAL PROCESSING INVESTIGATIONS

The objective of the preliminary metallurgy program was to develop effective flotation conditions for the recovery of copper and other payable metals, and to identify potential amenability problems. The first phase of this program focused on testing samples from the UZ, to develop metallurgical response data and process design parameters. In the second phase, samples from the LZ were tested applying the process conditions developed from the UZ composite.

The process development studies were conducted on a UZ Master Composite followed by variability flotation response tests on UZ sub-composites. The effects of primary grind and regrind levels as well as pH levels and collector types on the metallurgy were investigated. Bond rod and ball mill work index determinations were performed on the Master Composite. The process conditions developed for the UZ composite were subsequently applied to the LZ composite.

The test work was conducted at the metallurgical division of Inspectorate, and the analytical work was conducted by Inspectorate's analytical division which has ISO 9001 accreditation and uses standard QA/QC procedures.

The results of these investigations indicated that the Johnny Lee UZ copper-cobalt mineralization is very fine grained and complex requiring a primary grind level of 80% passing 38 µm and a rougher concentrate regrind of 80% passing 8 µm for effective liberation and recovery of copper minerals to a marketable concentrate. The Johnny Lee LZ composite was found to be coarse grained and very responsive to the UZ conditions that were applied, as the two mineralized material types will be comingled for processing in the second year of production.

Initial work which included Quantitative Evaluation of Materials by Scanning Electron Microscopy (QEMSCAN or QS) mineralogy studies focused on the determination of optimum rougher flotation feed size and flotation conditions followed by developing cleaner conditions in batch flotation tests.

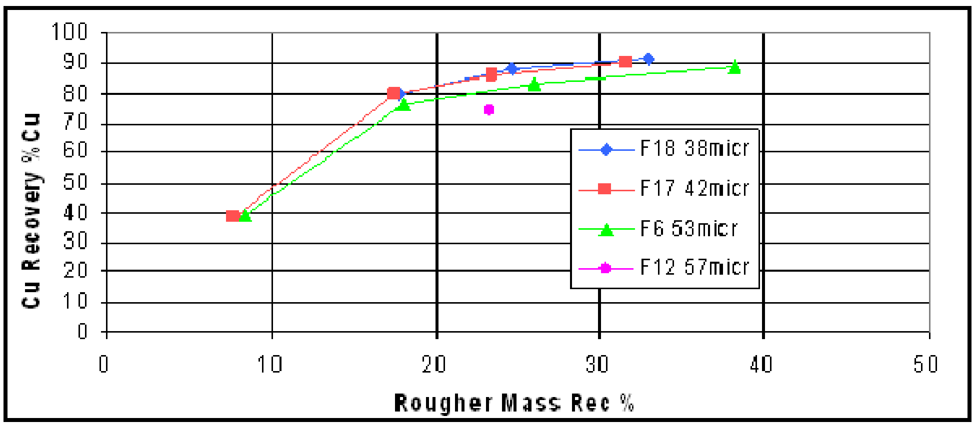
13.1.1 Investigations on Upper Zone Samples

The following is a brief overview of the investigations on the UZ composite.

Primary grind size levels between 108 and 42 µm P_{80} were tested initially, flotation was conducted at a near neutral pH level using sodium isopropyl xanthate (SIPX) as the primary collector while exploring a number of alternate secondary collectors. A primary P_{80} grind level of 42 µm and a combination of SIPX and A3894 were concluded to give the best results. A3894 is utilized as a CYTEC collector (a chemical manufacturer) composed of Alkyl-alkyl thionocarbamate, which is stated to be a good copper mineral collector selective against pyrite.

The results of the grind sensitivity tests, conducted at pH 8.0 to 8.5 are shown in Figure 13.1.

Figure 13.1 Rougher Mass Recovery versus Grind



A primary grind size level of 38 to 42 µm was indicated and a very high mass recovery of about 30% to the rougher concentrate was required to achieve close to a 90% copper rougher recovery; the ratio of concentration to achieve this recovery target is very low, at about three. The silver and cobalt recoveries to the rougher concentrate were very low at about 50%. These test results are in line with the information produced by the QEMSCAN mineralogy studies.

Difficulties experienced in upgrading the rougher concentrate in subsequent cleaner stages prompted a mineralogy study to determine the copper mineral liberation and associations; at 16 µm, about 60% of the copper sulphides in the concentrate are liberated and the balance was mostly associated with pyrite. The concentrate contained 63% cobaltiferous pyrite (containing over 40% of the cobalt in the concentrate), which was almost 80% liberated.

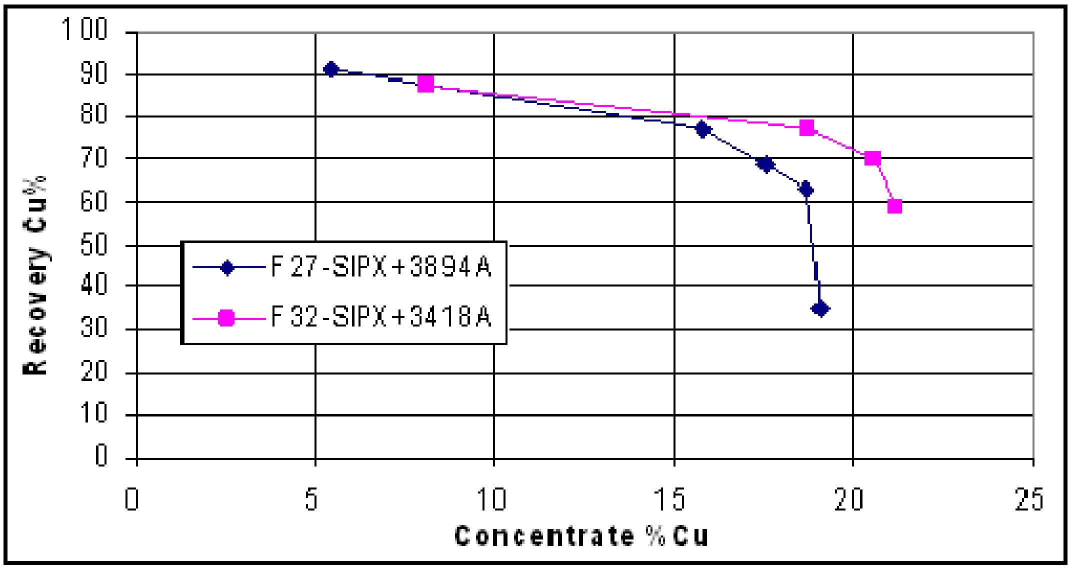
The high collector additions in the roughers needed to achieve less than 90% copper recoveries resulted in very high recoveries of nearly barren pyrite and high collector concentrations in the cleaners which adversely affected the selectivity.

More selective rougher conditions and finer re-grinding were considered to be necessary to improve selectivity in the cleaners. The results of tests F27 and F32 illustrate the effect of rougher conditions. The cleaner metallurgy of these tests is shown in Table 13.1 and Figure 13.2. The regrind level and cleaner conditions in both tests were the same.

Table 13.1 Effect of Rougher Conditions on Cleaner Metallurgy

	Rougher Conditions & Copper Recovery				Cleaner Concentrate	
Test	pH	Collector	Mass (%)	Cu (%)	Cu (%)	Rec (%)
F27	8.5	A3894	37.1	91.3	15.8	77
F32	9.5	3418A	23.0	87.4	18.8	77

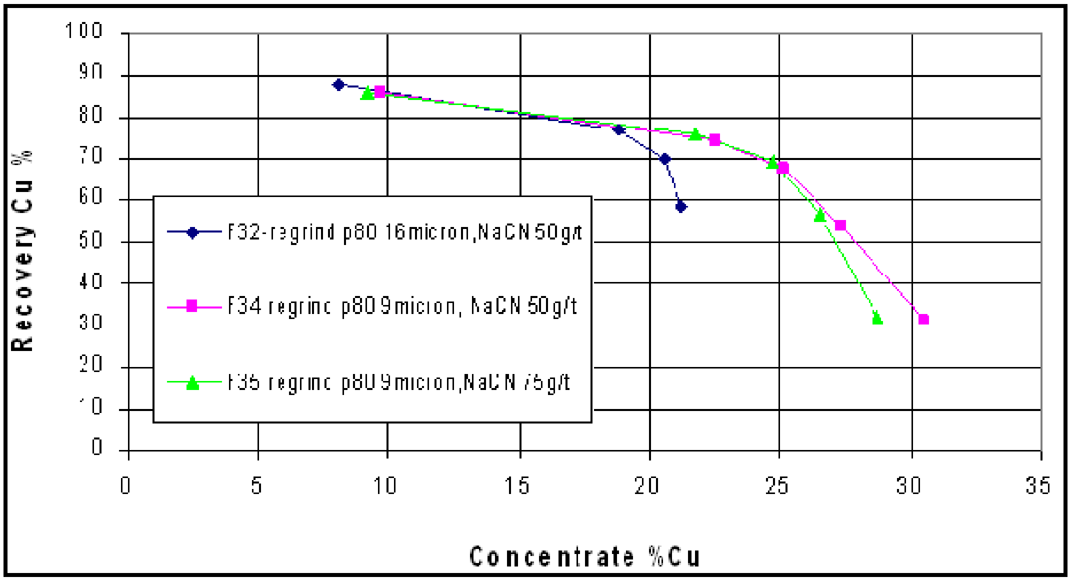
Figure 13.2 Effect of Collector on Cleaning



The more selective rougher conditions applied in F32 clearly produced superior overall copper cleaner metallurgy, even though the rougher recovery F27 was 4% higher. The cobalt and silver rougher recoveries in F32 at about 30% were much lower than in F27 because much of the cobaltiferous pyrite was rejected.

The effect of a finer regrind and sodium cyanide addition levels were explored in the next series of tests. The first cleaner conditions and results are shown in Figure 13.3.

Figure 13.3 Effect of Regrind and Sodium Cyanide on Cleaning



Test F34 with a 50 g/t sodium cyanide addition to the regrind mill produced the best results.

In preparation for the locked cycle test, two 3-stage batch cleaner tests were done generally following F35 conditions. In both tests, the P_{80} primary grind size was reduced to 37 µm and, in one of the tests (F37), the ratio between 3418A and SIPX was increased as summarized in Table 13.2.

Table 13.2 F34, F36, F37 Rougher Concentrates

F34, F36, F37 Rougher Concentrates										
					Assay			Distribution		
Test	Grind P_{80} (µm)	Collector (g/t) SIPX; 3418A	Product	Weight (%)	Ag (g/t)	Cu (%)	Co (%)	Ag (%)	Cu (%)	Co (%)
F34	42	60; 30	Rougher Concentrates	20.2	26.9	9.7	0.21	37.2	85.7	29.1
F36	37	60; 30	Rougher Concentrates	22.4	26.4	8.3	0.22	38.6	87.8	30.1
F37	37	30; 45	Rougher Concentrates	21.6	25.1	8.4	0.22	37.5	85.9	29.7

The finer primary grind slightly improved the copper, cobalt and silver rougher recoveries.

The metallurgy balance of F36 is listed in Table 13.3 and shows that the copper recovery decreases sharply at concentrate grades above 25% copper. The silver and cobalt recoveries to the cleaner concentrate are very low.

Table 13.3 F36 Metallurgy Balance

Combined Products	Weight %	Assay Ag (g/t)	Assay Cu (%)	Assay Co (%)	Distribution Ag (%)	Distribution Cu (%)	Distribution Co (%)
Third Cleaner Concentrate	2.7	35.4	30.6	0.07	6.2	38.8	1.1
Second Cleaner Concentrate	5.0	36.3	26.6	0.12	11.9	63.4	3.6
First Cleaner Concentrate	9.8	33.9	17.3	0.21	21.8	80.6	12.2
Rougher Concentrate	20.9	26.9	8.8	0.22	36.7	87.3	28.1
Calculated Feed	100.0	15.3	2.1	0.17	100.0	100.0	100.0

The program on the Master Composite was concluded with a locked cycle test under the optimized conditions used in F36.

The metallurgy projected from the results of the last three cycles is shown in Table 13.4.

Table 13.4 F38 Locked Cycle Test Cycles 4 to 6 Projected Metallurgy for Third and Second Stages of Cleaning

	Weight (%)	Assay Ag (g/t)	Assay Cu (%)	Assay Co (%)	Distribution Ag (%)	Distribution Cu (%)	Distribution Co (%)
Third Cleaner Concentrate	6.9	16.8	25.80	0.14	7.7	79.2	6.4
Second Cleaner Concentrate	8.5	15.8	21.70	0.15	9.0	82.2	8.2
First Cleaner Scavenger Tails	10.4	23.2	1.10	0.24	16.2	5.0	16.7
Rougher Concentrate	18.9	19.9	10.30	0.20	25.2	87.2	24.8
Total Final Tails	81.1	13.8	0.35	0.14	74.8	12.8	75.2
Calculated Head	100.0	14.9	2.24	0.15	100.0	100.0	100.0

A minor element analysis was performed on the concentrates of the last three cycles of the test. The concentrate was found to contain very low levels of deleterious elements; the only element that may incur a penalty is arsenic, which has a slightly elevated concentration of 0.28%. Minor element concentrates are summarized in Table 13.5.

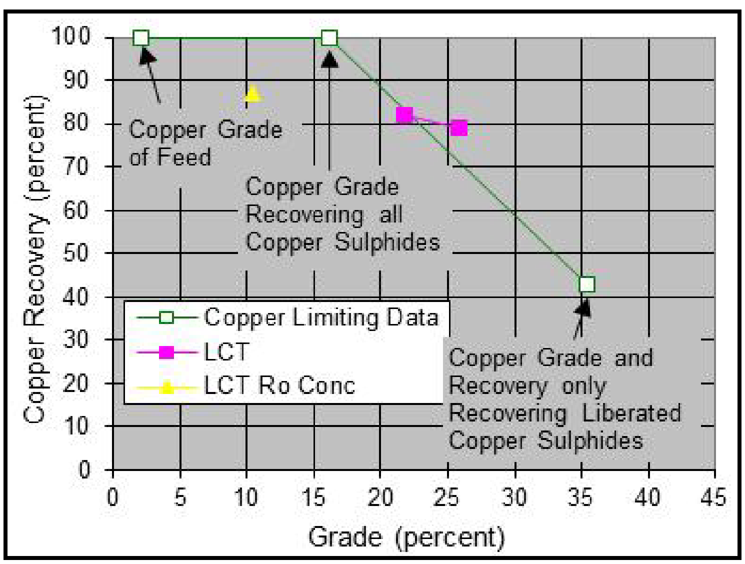
Table 13.5 Minor Element Concentrations

Locked Cycle Test Concentrate Minor Element Analysis			
Elements	Units	Third Cleaner Concentrate	Analytical Method
Ag	ppm	17	30-4A-TR
As	%	0.28	As-1A-OR
Bi	ppm	<2	30-4A-TR
Cd	ppm	<0.5	30-4A-TR
Co	ppm	1,419	30-4A-TR
F-	µg/g	59	ISE
Mo	ppm	8	30-4A-TR
Ni	ppm	586	30-4A-TR
Pb	ppm	887	30-4A-TR
Sb	ppm	92	Sb-4A-LL-ICP
Se	ppm	39	Se-4A-LL-ICP
Te	ppm	6.6	50-4A-UT
Zn	ppm	136	30-4A-TR
Hg	ppm	1.3	Hg-AR-TR-CVAA

The contained silver is not payable as the concentration is less than 1 oz/ton. The cobalt recovery is very low as most of the cobalt is present ion cobaltiferous pyrite.

The k80 feed size of the locked cycle test was 31 µm. The mineralogically limiting grade recovery curve derived from the QEMSCAN data shown in Figure 13.4 indicates that the locked cycle test results were consistent with the projections from the QS study.

Figure 13.4 Mineralogically Limiting Grade Recovery Chart – UZ Composite



A mineralogy study (QEMSCAN Bulk Modal Analysis) and a number of preliminary batch cleaner flotation tests were conducted on the UZ sub-composites following the conditions of the locked cycle test. The composite head grade and copper mineral liberation are shown in Table 13.6.

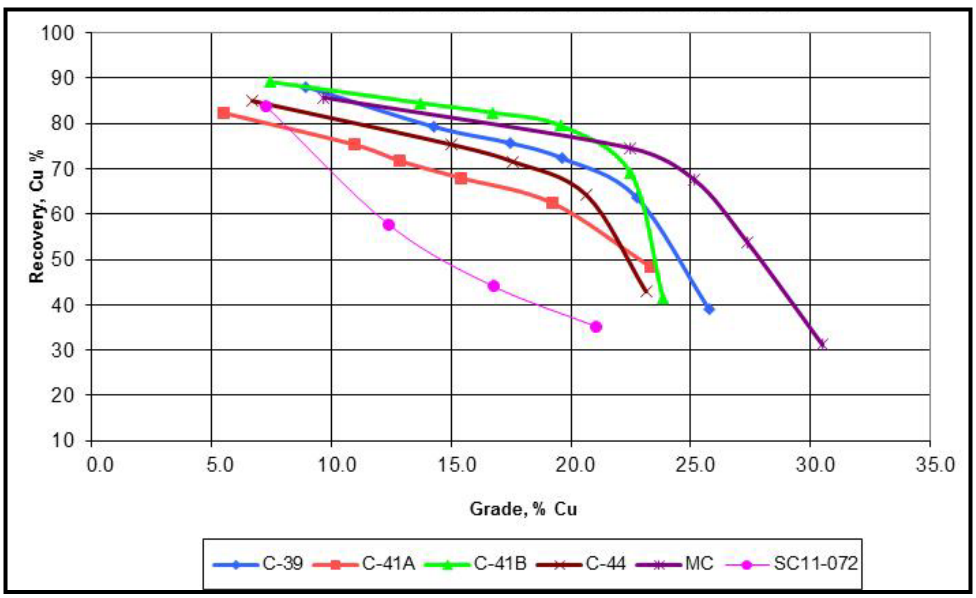
At a particle size of about 50 µm less than 30% of the chalcopyrite is liberated; insufficient material of the higher grade composite 36 was available for test work but the received sample was included in the Master Composite.

Table 13.6 Johnny Lee UZ Sub-composite Copper Mineralogy Mineral Distribution by Class of Associations

Sample	Composite 36	Composite 39	Composite 41A	Composite 41B	Composite 44
Grade % Cu	3.3	2.2	1.7	2.1	1.8
Size k80 µm	46	54	48	49	53
Minerals					
Liberated Cs	58	26	23	33	20
Binary – Cob	3	1	2	2	1
Binary – Py	14	26	37	35	36
Binary – Gn	10	13	6	6	4
Multiphase	15	34	33	24	38
Total	**100**	**100**	**100**	**100**	**100**

The grade-recovery curves derived from the batch cleaner tests, shown in Figure 13.5, indicate that there is a large variability in metallurgical response between drillhole samples of the UZ.

Figure 13.5 Copper Grade Recovery UZ Sub-composites

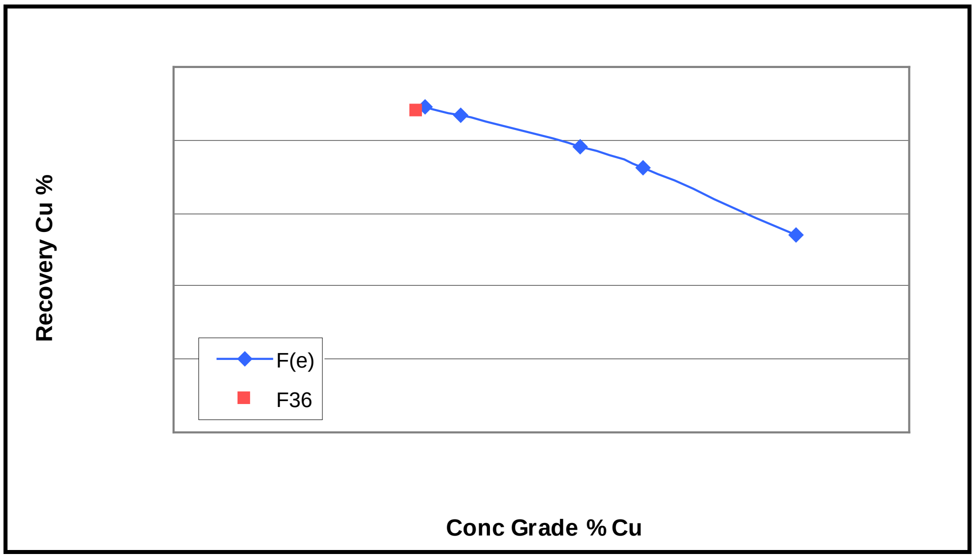


The higher grade sample of hole SC11-072 gave the poorest response. The samples exhibit a sharp decrease in recovery at concentrate grades above 20% copper.

A series of optimization tests on the Johnny Lee UZ Composite were conducted to determine if the conditions developed in previous test work could be further optimized; the basis for these tests was test F36. F36 conditions were applied to the locked cycle test F38 and formed the basis for the design criteria.

The best results of the four optimization tests were achieved in test F(e). In this test, lime was added to the grind and collector additions were increased substantially compared to test F36, as a result the copper flotation kinetics increased significantly. The rougher flotation time was reduced from 19 to 13 minutes which is a reduction of about 30%. The overall rougher metallurgy did not change as the same copper and rougher mass recovery were achieved as in test F36 as shown in Figure 13.6.

Figure 13.6 Rougher Test F(e) versus F36



Based on the results of test F(e), it was recommended to reduce the rougher flotation time in the design criteria by 30%; other parameters such as copper and mass recovery will remain unchanged. The reagent coditions of the two tests are compared in Table 13.7.

Table 13.7 F36 and F(e) Rougher Reagent Conditions

Test	F36	F(e)
Lime to Grind (g/t)	1,000	2,000
Lime to Roughers (g/t)	620	0
Rougher pH	9.5	10
SIPX (g/t)	60	110
A3418 (g/t)	30	40

13.1.2 INVESTIGATIONS ON LOWER ZONE SAMPLES

Intervals from nine diamond drillholes selected across the LZ mineralization were selected for preliminary test work. From these samples, five sub-composites were prepared with grades ranging from 0.6 to 11% copper with an overall weighted average composite grade of about 4% copper. Batch rougher and cleaner tests with UZ conditions were conducted on this sample in preparation for a locked cycle test. The test work indicated that good results can be obtained at a coarser primary grind without regrinding of the rougher concentrate. The locked cycle test however was conducted at UZ grind levels because material from both the UZ and LZ will be comingled in the mill feed. The results of the locked cycle test are shown in Table 13.8.

Table 13.8 Locked Cycle Test Results on LZ Composite

Parameter	Weightt (%)	Assays Ag (g/t)	Assays Cu (%)	Assays S (%)	Distribution Ag (%)	Distribution Cu (%)	Distribution S (%)
Cleaner Concentrate	14.3	8.7	27.0	35.1	23	96.6	20.2
First Cleaner Scavenger Tails	7.6	-	0.63	30.8	-	1.2	9.4
Rougher Concentrate	21.9	-	17.8	33.6	-	97.8	29.6
Rougher Tails	78.1	-	0.11	22.4	-	2.2	70.4
Calculated Head	100.0	5.5	3.98	24.9	-	100.0	100.0

Preliminary rougher and batch cleaner tests were done on the four sub-composites of this zone. The results of the kinetic rougher tests are shown in Table 13.9.

Table 13.9 Johnny Lee LZ Sub-composite Rougher Tests

Composite	Heads % Cu	Heads % Stot	Product	Weight (%)	Assay % Cu	Assay % Stot	Distribution Cu %	Distribution Stot %
LZ1	0.62	22.1	Rougher Concentrate	14.4	4.0	25.8	93.0	16.8
LZ2	1.53	16.0	Rougher Concentrate	25.5	5.9	27.8	97.9	44.5
LZ3	3.12	15.3	Rougher Concentrate	23.1	13.2	33.6	98.2	34.0
LZ4	6.82	34.6	Rougher Concentrate	38.4	17.8	36.7	98.2	40.8

The excellent results of the locked cycle test and of the rougher tests on the sub-composites indicate that the LZ copper mineralogy is much coarser grained and less complex than that of the UZ.

A QEMSCAN mineralogy study of the LZ composite indicated that 88% of the chalcopyrite is liberated at a particle size of 85% passing 53 µm (liberated is defined as greater than 80% exposed).

13.1.3 Cobalt and Silver Mineralogy and Metallurgy

Composites of the Johnny Lee Upper and Lower Zones were analyzed and tested. The head assays of the composites shown in Table 13.10 indicate that there is a significant difference in cobalt and silver content between the composites.

Table 13.10 Johnny Lee UZ and LZ Composite Analysis

Element	Unit	UZ	LZ
Copper	%	2.07	4.05
Gold	ppm	0.006	0.52
Silver (tot)	ppm	15	5.5
Silver (CN Sol)	ppm	2.4	N/A
Cobalt	ppm	1,644	523

The results of mineralogy studies and flotation tests show a large difference in mineral textural complexity and flotation response between the zones; the UZ material being very fine grained responded poorly compared to that of the coarse grained LZ. The cobalt and silver recoveries to concentrate were low for both zones.

Summary and Conclusions

- Locked cycle tests were conducted with the primary objective to optimize copper recovery and concentrate grade not to optimize cobalt and silver recoveries.
- Mineralogy studies and flotation test results indicate that the recoveries of cobalt and silver to the copper flotation concentrate will be low due to the complex fine grained nature of the minerals containing these elements.
- Optimizing reagent conditions in further flotation tests may improve the cobalt and silver flotation recoveries to the copper concentrate but not significantly.
- Approximately 17% of the cobalt and silver reported to the cleaner scavenger tailings; it may be possible to scalp of liberated cobalt minerals from this stream by lowering the pH to 8-9 with sulphuric acid and the addition of some copper sulphate. Recovery of these elements by hydrometallurgy processes could also be further explored.
- Scoping tests to produce a pyrite rougher flotation at pH 6 from the locked cycle test rougher tailings were marginally successful recovering 25 to 45% of the sulphur to the pyrite concentrates assaying about 42% sulphur, further test work is required to improve concentrate grade and recovery. A target pyrite concentrate grade and purity for roaster feed material have to be obtained from a potential buyer. Payment for contained cobalt and silver in the pyrite-sulphur concentrate is very unlikely due to the low concentration levels. The market for and the economics of this option need to be assessed.
- The rougher flotation tailings contained 70 to 75% of the cobalt and silver in the test feed; it is unlikely that these elements can be recovered economically from

the tailings due the low concentration levels; tests as proposed for the cleaner scavenger tailings are recommended.

- Producing a pyrite-sulphur concentrate from the rougher tailings combined with the cleaner scavenger tailings for use as backfill will reduce the long term tailings disposal and management costs.

DETAILS

Cobalt Mineralogy

QEMSCAN mineralogy studies were performed on the UZ Master Composite and on some flotation tests products of this composite.

The mineral composition and elemental deportment of the locked cycle test feed sample ground to a p80 of 31 µm, examined by G&T in report KM3218, are summarized in Table 13.11 and Table 13.12.

In the G&T report the following conclusions with regard to cobalt were made:

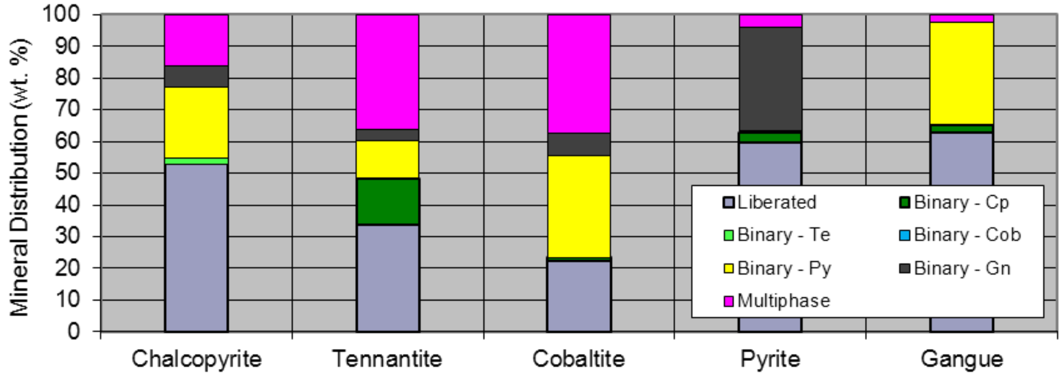
> *"Cobaltite was the principal cobalt bearing mineral, and contained about 60 percent of the total feed cobalt. Carrolite contained about 5% of cobalt. It is of significance to note that about 36percent of the feed cobalt was contained in pyrite.Consequently, the cobalt recovery from the flotation feed will be limited due to the need to reject pyrite from the copper flotation circuit in order to make a saleable copper concentrate grade." "The concentration of cobalt in pyrite averaged 1321ppm but ranged from 0 to 7350 ppm".*

Table 13.11 Mineral Composition of UZ Master Composite

Mineral Composition				Copper Department		Cobalt Department	
Sulphide Minerals	Mass (%)	Gangue Minerals	Mass (%)	Copper Sulphides	Mass (%)	Cobalt Minerals	Mass (%)
Chalcopyrite	5.6	Quartz/Feldspars	23.8	Chalcopyrite	93.3	Cobaltite	
Bornite	<0.01	Barite	16.7	Bornite	0.1	Carrollite	59.3
Covellite	0.01	Iron Oxides	1.3	Covellite	0.2	Bravoite	4.7
Tennantite	0.3	Muscovite	0.7	Tennantite	6.1	Pyrite	35.8
Cobaltite	0.4	Carbonates	0.3	Carrollite	0.2	-	-
Carrollite	0.03	Other Gangue	4.8				
Bravoite	<0.01	-	-	-	-	-	-
Pyrite	46.1	-	-	-	-	-	-
Arsenopyrite	0.01	-	-	-	-	-	-
Total	**52.4**	**Total**	**47.6**	**Total**	**100**	**Total**	**100**

The mineral distribution by class of association for cobaltite shows that only about 20% of the cobalt minerals are liberated at a p80 grind size of 31 µm, the balance occurs as inclusions in pyrite and multiphase particles.

Table 13.12 UZ Master Composite Mineral Distribution by Class of Association



As the LZ composite contained less than 0.1% cobalt co-mineralogy data were not generated for this sample.

Cobalt and Silver Flotation Response

The flotation response of cobalt and silver of the Upper and Lower Zone Master Composites are shown in Table 13.13 and Table 13.14.

Table 13.13 Johnny Lee UZ – Locked Cycle Test Metallurgy Projections

Products	Weight (%)	Assays Ag (g/t)	Assays Cu (%)	Assays Co (%)	Distribution Ag (%)	Distribution Cu (%)	Distribution Co (%)
Second Cleaner Concentrate	8.5	15.8	21.7	0.15	9.0	82.2	8.2
First Cleaner Scavenger Tails	10.4	23.2	1.1	0.24	16.2	5.0	16.7
Rougher Concentrate	18.9	19.9	10.3	0.20	25.2	87.2	24.8
Rougher Tails	81.1	13.8	0.35	0.14	74.8	12.8	75.2
Calculated Head	100.0	14.9	2.24	0.15	100.0	100.0	100.0

In the locked cycle test about 25% of the cobalt and silver reported to the rougher concentrate, this is consistent with the cobalt mineralogy data. Recoveries to the final concentrate for cobalt and silver were 8 to 9% respectively, as cobaltiferous pyrite and complex cobalt and silver particles were rejected in the cleaning stages. The cobalt and silver contained in the concentrate is below the payable level.

Silver mineralogy data were not produced due to the low silver content; the cyanide soluble silver content was only 16% of the total silver content suggesting that most of the silver is occluded in other minerals.

The locked cycle tests cleaner scavenger tailings contained about 16% of the cobalt and silver present in the flotation feed; studies to scalp of cobalt and silver from this process stream are recommended. This product which contained about 10% of the sulphur in the feed and graded 30% sulphur could be upgraded in sulphur content, after cobalt removal, if a market exists for such a product.

Preliminary pyrite flotation from locked cycle test rougher tailings recovered about 25% of the sulphur, cobalt and silver from the tailings to a concentrate containing 43% sulphur and 0.2% cobalt and 22 g/t silver. The cobalt and silver grades in this concentrate are too low to be of economic interest.

The Johnny Lee LZ metallurgy shows a similar response with regards to cobalt and silver as the UZ, low recoveries and concentrates without payable cobalt and silver content.

Table 13.14 Johnny Lee LZ – Locked Cycle Test Metallurgy Projections

Products	Weight (%)	Assays			Distribution		
		Ag (g/t)	Cu (%)	Co (%)	Ag (%)	Cu (%)	Co (%)
Cleaner Concentrate	14.3	8.7	27.0	0.08	23.0	96.6	14.5
First Cleaner Scavenger Tails	7.6	-	0.63	0.18	-	1.2	17.8
Rougher Tails	78.1	-	0.11	0.07	-	2.2	67.8
Head	100.0	5.5	3.98	0.08	-	100	100

Preliminary pyrite flotation from locked cycle test rougher tailings recovered approximately 45% of the sulphur in the tailings to a concentrate containing 42% sulphur and 0.11% cobalt. The cobalt and silver grades in this concentrate are too low to be of economic interest.

13.2 BASIS FOR PREDICTING THE COPPER RECOVERY OF THE UPPER ZONE MINERALIZATION

The recovery of copper to concentrate from the UZ mineralization was estimated based on the locked cycle test results on the Master Composite which graded 2.24% copper; the annual copper recovery was then calculated to reflect the higher annual mine production plan head grades which with has a LOM average 2.6% copper. The average annual copper recovery for the this head grade works out to 83.6% compared to the locked cycle test copper recovery of 82.2%.

The following steps were taken to calculate the copper recovery mine plan production head grades:

1. Calculate the recovery for the higher head grade assuming a constant tailings assay as produced in the locked cycle test.
2. Add 50% percent of the recovery increase calculated from step 1 to the locked cycle test recovery.

The locked cycle test and the LOM UZ metallurgy are compared in Table 13.15.

Table 13.15 Johnny Lee UZ Copper Metallurgy

	Head	Concentrate	
	% Cu	% Cu	Rec % Cu
Locked Cycle Test	2.24	21.7	82.2
LOM Average Grade	2.6	21.7	83.6

The recovery estimate for higher grade LOM mineralization has a low level of confidence as the available data on the response of higher grade mineralization or on the variability in mineralogy across the UZ is very limited.

13.3 Basis for Projecting the Copper Recovery for the Lower Zone Mineralization

The LOM average head grade of the LZ is 4.9% copper compared to a test composite grade of 4.0%. The rougher recovery in the locked cycle test, as shown in Table 13.8, was 97.8%; the higher grade sub-composites yielded 98% copper rougher recovery; this rougher recovery can be applied to the average LZ head grade of 4.9% copper. Based on a 99% cleaner recovery obtained in the locked cycle tests, an average copper recovery of 97% can be projected for the LZ mineralization.

13.4 Origin and Representativeness of Metallurgical Samples

Mineralized intervals of four diamond drillholes in the UZ were selected for the study; the samples include hanging wall dilution amounting to 15% of the weight of the 2 m interval above mineralization. A small number of samples were selected to represent the typical massive sulphide copper mineralization in the UZ but are not considered to be representative of the entire UZ mineralization. The samples selected are summarized in Table 13.16.

Table 13.16 Summary of Black Butte UZ Metallurgy Sub-composites

Composites	Hole ID	From (m)	To (m)	Length (m)
Composite 36	SC11-036	118.5	122.07	3.5
Composite 39	SC11-039	122.8	133.70	10.9
Composite 41a	SC11-041	75.6	86.37	10.1
Composite 41b	SC11-041	91.4	99.71	6.8
Composite 44	SC11-044	112.8	120.00	6.3
Total	-	-	-	**37.6**

A Master Composite was prepared from the sub-composites based on equal weight contributions.

The head grades of the Master Composite and sub-composites are shown in Table 13.17.

Table 13.17 Black Butte UZ Composite Analyses

Element	Unit	Black Butte UZ Composite Analyses					
		Master	36	39	41A	41B	44
Silver	ppm	10.8	4.5	16.1	8.1	9.3	18.4
Copper	%	2.07	3.55	2.42	1.94	2.45	1.98
Cobalt	ppm	1,644	1,202	2,573	999	572	1,141
Sulphur total	%	30.3	23.7	29.8	31.8	28.5	32.3

The cyanide soluble silver content was determined to provide an indication of the exposed surface area of the silver containing minerals; a 25% cyanide soluble silver content suggests that most of the silver is occluded in other minerals and not readily available for flotation.

High-grade intervals from drillhole SC11-072 (108.7 to 126.4 m) were shipped to Inspectorate to explore the response of higher grade mineralization. The target head grade of this sample was 3%, however, the prepared sample head grade was only 2.6% copper.

13.5 Lower Zone Metallurgy Samples

Samples of Johnny Lee LZ mineralization were collected from Tintina drill core and submitted to Inspectorate for initial metallurgical test work. Core samples from 11 diamond drillholes spaced across the LZ mineralization were selected to cover the spatial distribution as well as the grade range within the LZ. The sample selection and head grades are summarized in Table 13.18 and Table 13.19. Material submitted from hole SC11-29 was insufficient for test work.

Table 13.18 Summary of Black Butte LZ Metallurgy Composites

Composite ID	Drillhole	From (m)	To (m)	Length (m)
LCZ-1	SC11-010	450.1	458.8	8.7
	SC11-031	426.1	428.2	2.1
	SC11-032	374.5	375.7	1.2
	Total	N/A	N/A	**12.0**
LCZ-2	SC11-007	409.7	411.2	1.5
	SC11-008	353.4	357.4	4.0
	SC11-009	415.4	416.7	1.3
	Total	N/A	N/A	**6.8**
LCZ-3	SC11-011	409.7	422.7	13.0
	SC11-012	384.7	387.6	2.9
	SC11-015	449.3	456.6	7.3
	Total	N/A	N/A	**23.2**

table continues…

Composite ID	Drillhole	From (m)	To (m)	Length (m)
LCZ-4	SC11-048	359.9	367.6	7.7
LCZ-5	SC11-029	437.0	441.5	4.5
Total	N/A	N/A	N/A	**54.2**

Table 13.19 Black Butte LZ Composite Analyses

Element	Unit	Master Composite	Sub-composites			
			LZ-1	LZ-2	LZ-3	LZ-4
Silver (total)	ppm	5.5	3.5	4.2	4.4	7.9
Copper	%	4.1	0.6	1.5	3.2	6.7
Cobalt	ppm	523	282	196	446	944
Sulphur (total)	%	24	22.1	16.1	23.6	33.2

13.6 Grindability Tests

Standard Bond work index terminations were performed on the Upper and Lower Zone composites. Table 13.20 tabulates the results.

Table 13.20 Johnny Upper and Lower Zone Grindability Test Results

	Work Index (kWh/t)		Closing Screen (µm)		
Composite	Rod Mill	Ball Mill	Rod Mill	Ball Mill	Abrasion Index
Upper Zone	17.1	13.6	977	74	0.688
Upper Zone	-	14.8	-	53	-
UZ SC11-072	15.0	15.1	1,190	74	0.688
Lower Zone	12.6	11.8	1,190	74	-

The UZ samples are relatively hard and abrasive due to high silica content.

13.7 Mineral Processing Risk Factors

The samples selected for the test work are believed to be typical but not necessarily representative of the massive sulphide mineralization of the UZ and LZ of the Johnny Lee deposit. Results of tests performed on UZ drillhole composites showed significant variability in metallurgical response between composites. The test work completed to-date is appropriate for a PEA level of study, but more test work on a much larger suite of samples taken from the across the mineralization in the zone is required for a feasibility-level study.

The UZ composite which was tested is lower in grade at 2.24% copper than the LOM UZ production plan copper grade of 2.6%; a positive adjustment to the locked cycle test recovery was made to project the recovery for the higher grade ore in the mine production plan; there is a risk associated with this projection as acual testwork data on mine grade production samples is not available.

The process flowsheet and flotation conditions used in the tests on which the metallurgy projections are based are typical for the processing of complex massive sulphide mineralized material, and are used extensively in the industry; accordingly, the process risk is considered to be low.

The concentrate produced in the locked cycle tests contained very low levels of potentially deleterious elements; this provides a preliminary indication that the risk with regard to the effect of deleterious elements on the Project economics is relatively low.

The sulphide mineralization of the LZ is very different than that of the UZ, the latter being very fine grained and complex while the former is coarse-grained with easily liberated copper minerals. Material from both zones will be combined as mill feed in the second year of production. Developing optimal processing conditions to co-process these very different mineralization types will be essential to achieving good metallurgical performance.

14.0 MINERAL RESOURCE ESTIMATES

Mr. Michael J. Lechner, President of RMI was contracted to prepare an estimate of Mineral Resources for the Johnny Lee UZ, Johnny Lee LZ, and the Lowry MZ within the Project area. Mr. Lechner is a recognized QP by virtue of his education (B.A. Geology, University of Montana), experience (over 30 years of continuous employment in the fields of mineral exploration, mine operations, resource estimation and geologic consulting), and professional registration (P.Geo. in BC, Registered Geologist in Arizona, Certified Professional Geologist from the AIPG, and a Registered Member of the SME).
Mr. Lechner has no interest in Tintina or owns any Tintina securities and has operated for them as an independent consultant.

Mr. Lechner estimated resources for the Johnny Lee UZ in late 2010 and prepared a Technical Report, which discussed that work (Lechner 2010). Mr. Lechner estimated resources for the Johnny Lee LZ in late 2011 and prepared a technical report which discussed that work (Lechner 2012). Mr. Lechner updated the estimate of mineral resources for the Johnny Lee UZ in early 2012 (Lechner 2012). Resources were estimated for the Lowry MZ in early 2012 and a technical report summarized those results (Lechner, 2012).

Mineral resources were updated in late 2012 for the Johnny Lee UZ and LZ using infill drilling results from the 2012 drilling campaign. This section discusses the status of mineral resources for the three Black Butte mineralized zones (i.e. Johnny Lee UZ, Johnny Lee LZ, and Lowry MZ). Sections 14.1 through 14.13 discuss the Johnny Lee UZ resource estimate. Sections 14.14 through 14.22 discuss the estimate of resources for the Johnny Lee LZ. Sections 14.23 through 14.34 discuss the estimate of resources for the Lowry Zone.

14.1 DRILLHOLE DATA

RMI was provided with various electronic drillhole data for the Johnny Lee UZ, Johnny Lee LZ and Lowry MZ by Tintina personnel. These data (drillhole collars, downhole surveys, assays, geology, density, etc.) were provided as either MS Excel® spreadsheets, ASCII CSV files, or as MineSight® drillhole files. MineSight® is a commercial mine planning software package.

All the Project drillhole data are stored in a MS Access® database that was constructed and is managed by database consultant Jack Cote. The database resides on the Tintina corporate FTP site as well as at the Project site in Montana. All assay results are loaded directly from electronic certificates that are issued by ALS Chemex. QA/QC reports are prepared for each certificate load. Once the new assay data are loaded a copy of the updated database is uploaded to the Project office and ftp site. All data collected from core logging is hand entered by project personnel into the DDH3 Site Tool data entry program. This includes RQD, SG measurements, geologic coding of intervals, sample interval data, and collar and downhole survey data.

14.2 JOHNNY LEE UZ DRILLING DATA

As described in Section 10.0, Tintina has collected most of the drilling data used by RMI to estimate the Mineral Resources for the UZ. RMI has also used historic drilling data collected by CAI, UII, and BHP. The aerial distribution of the holes shown in Table 10.1 can be reviewed in Figure 10.1.

14.3 JOHNNY LEE UZ EXPLORATORY DATA ANALYSIS

The Johnny Lee UZ consists of several lenses of massive sulphide mineralization. Tintina's geologic staff generated 3D wireframes which represent those two copper sulphide lenses. The main massive sulphide zone is referred to as “upper copper zone 31”. A smaller, stratigraphically higher massive sulphide lense is referred to as “upper copper zone 32”.

Basic assay statistics for copper, silver and gold are tabulated in Table 14.1 through Table 14.4, respectively for the two Johnny Lee UZ massive sulphide units (i.e. UCZ 31 and UCZ 32). The term "Inc %" in columns 4 and 7 of Table 14.1 to Table 14.5 refers to incremental percentage of material between cut-off grades. For example, the first incremental percentage value of 33 in column 4 of Table 14.1 means that 33% of the assayed meterage is between 0.00 and a 1.00% copper cut-off grade. The term "Grd-Thk" in column 6 of Table 14.1 to Table 14.4 refers to grade times thickness. The "Inc %" column immediately to the right of "Grd-Thk" refers to the incremental grade times thickness product at various cut-offs.

Table 14.1 Johnny Lee UZ Copper Assay Statistics

UZ	Cu Cut-off (%)	Total (m)	Inc %	Mean Cu (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
All Data	0.00	1,398	33	2.02	2,822	10.6	1.93	0.96
	1.00	931	36	2.71	2,522	25.0	2.04	0.75
	2.00	434	11	4.18	1,817	13.7	2.18	0.52
	3.00	275	20	5.20	1,431	50.7	2.16	0.42
31	0.00	1,230	34	1.99	2,451	11.0	1.91	0.96
	1.00	813	36	2.69	2,182	25.4	2.03	0.75
	2.00	375	11	4.17	1,560	13.6	2.18	0.52
	3.00	236	19	5.19	1,226	50.0	2.16	0.42
32	0.00	168	30	2.20	371	8.3	2.07	0.94
	1.00	118	35	2.88	340	22.5	2.13	0.74
	2.00	60	12	4.30	257	13.9	2.21	0.51
	3.00	39	23	5.26	205	55.3	2.18	0.41

Note: Inc % = incremental percentage, Grd-Thk = grade times thickness, CV = Coefficient of Variation.

As shown in Table 14.1, approximately 67% of the drillhole assays in the main sulphide bed (UZ 31) are in excess of 1% copper and about 19% are above 3% copper. The CV for the UZ copper assays is 0.96, which suggests that high-grade outliers are not a significant issue.

Table 14.2 UZ Cobalt Assay Statistics

UZ	Co Cut-off (%)	Total (m)	Inc %	Mean Co (ppm)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
All Data	0.00	1,398	1	0.09	128	0.1	0.09	1.00
	0.01	1,379	27	0.09	128	10.2	0.09	0.98
	0.05	999	43	0.12	115	33.4	0.10	0.86
	0.10	399	29	0.18	72	56.2	0.13	0.72
31	0.00	1,230	1	0.09	112	0.1	0.09	1.02
	0.01	1,219	27	0.09	112	10.2	0.09	1.01
	0.05	892	45	0.11	101	35.2	0.10	0.89
	0.10	338	27	0.18	61	54.5	0.14	0.76
32	0.00	168	5	0.10	16	0.3	0.08	0.84
	0.01	160	32	0.10	16	10.4	0.08	0.79
	0.05	107	27	0.14	14	21.1	0.08	0.57
	0.10	62	37	0.18	11	68.2	0.07	0.41

Approximately 27% of the Johnny Lee UZ cobalt assays are above 0.10%. Cobalt shows a slightly higher CV than copper but is not alarmingly high.

Table 14.3 UZ Silver Assay Statistics

UZ	Ag Cut-off (%)	Total (m)	Inc %	Mean Ag (g/t)	Grd-Thk (g/t-m)	Inc %	Standard Deviation	CV
All Data	0	1,398	6	16	22,240	1.0	11	0.70
	5	1,320	20	17	22,023	9.3	11	0.66
	10	1,039	66	19	19,946	67.1	11	0.58
	30	115	8	44	5,032	22.6	16	0.37
31	0	1,230	5	16	19,708	0.9	11	0.68
	5	1,165	19	17	19,526	8.7	11	0.64
	10	934	68	19	17,818	69.0	11	0.56
	30	97	8	44	4,228	21.5	15	0.35
32	0	168	8	15	2,532	1.4	13	0.88
	5	155	30	16	2,497	14.6	13	0.82
	10	105	52	20	2,128	52.3	14	0.71
	30	18	11	44	804	31.8	20	0.45

Approximately 74% of the Johnny Lee UZ silver assays are above a 10 g/t cut-off grade with 8% above a 30 g/t cut-off. Silver shows the lowest CV of the four metals that were estimated by the author.

Table 14.4 UZ Gold Assay Statistics

UZ	Au Cut-off (%)	Total (m)	Inc %	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc %	Standard Deviation	CV
All Data	0.00	1,398	100	0.01	13	74.6	0.05	5.10
	0.25	5	0	0.71	3	3.5	0.40	0.57
	0.50	3	0	0.95	3	7.4	0.30	0.32
	1.00	2	0	1.25	2	14.5	0.00	0.00
31	0.00	1230	100	0.01	12	75.3	0.05	5.29
	0.25	3	0	0.95	3	0.0	0.30	0.32
	0.50	3	0	0.95	3	8.4	0.30	0.32
	1.00	2	0	1.25	2	16.3	0.00	0.00
32	0.00	168	99	0.01	1	68.9	0.03	3.08
	0.25	2	1	0.27	0	31.1	0.00	0.00
	0.50	0	0	0.00	0	0.0	0.00	0.00
	1.00	0	0	0.00	0	0.0	0.00	0.00

Gold assays tend to be generally quite low in the UZ with only two samples above a 0.5 g/t cut-off grade. Those two "high-grade" samples skew the CV at 5.1.

14.4 Johnny Lee UZ High-grade Outliers

RMI generated a series of cumulative probability plots after transforming the original copper, cobalt, silver, and gold assays using the cumulative normal distribution method. Figure 14.1 and Figure 14.2 show copper and cobalt probability plots for UZ unit 31, respectively. The black circle shown in these figures are capping limits selected by RMI to minimize the potential for over estimating contained metal. Table 14.5 summarizes the high-grade outlier capping limits that were established for copper, cobalt, silver, and gold. These limits were applied to the raw assays prior to creating drillhole composites.

Table 14.5 Johnny Lee UZ Grade Capping Limits

	Copper (%)		Cobalt (%)		Silver (g/t)		Gold (g/t)	
UZ Unit	Cap Limit	No. Capped	Cap Limit	No. Capped	Cap Limit	No. Capped	Cap Limit	No. Capped
31	15.0	5	1.0	4	60	13	1.0	1
32	8.0	6	0.5	0	50	2	0.5	0

Figure 14.1 Johnny Lee UZ Copper Cumulative Probability Plot

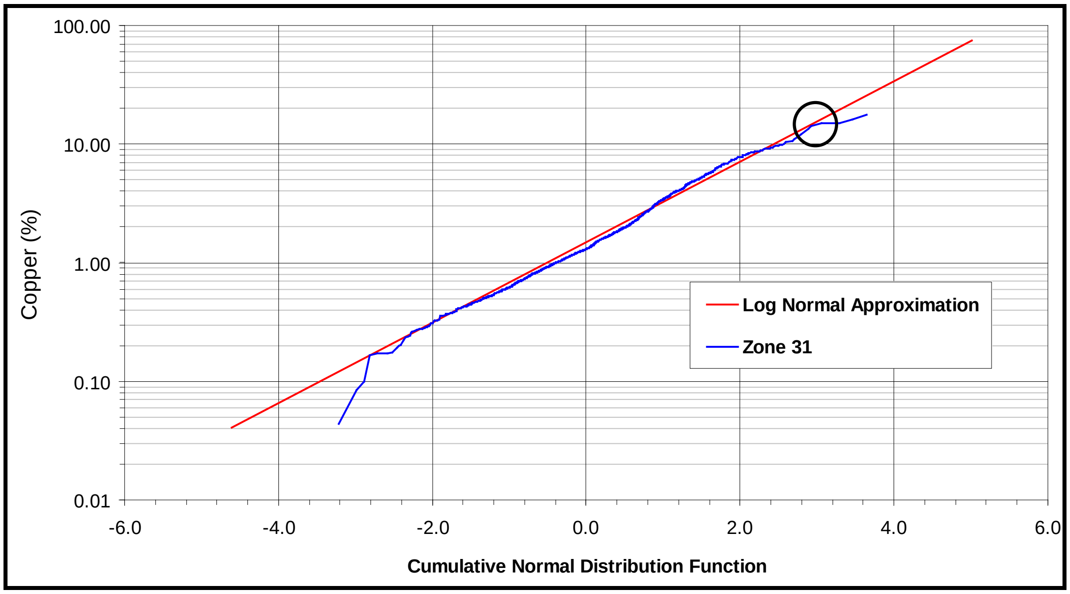
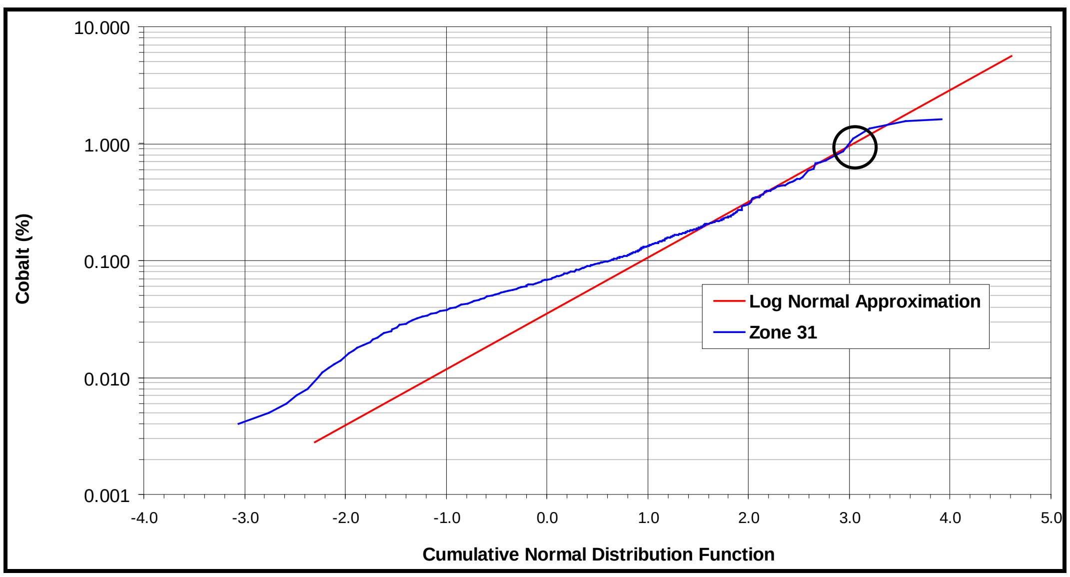


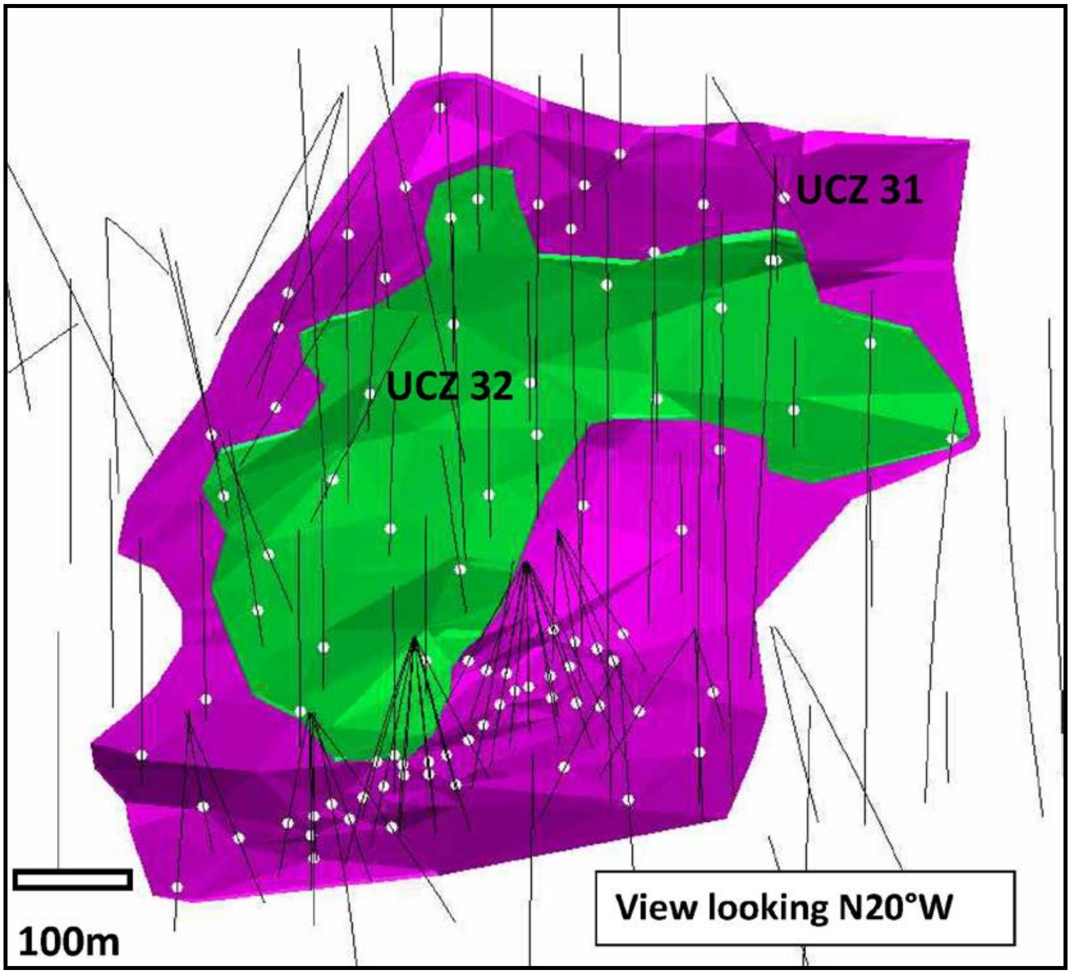
Figure 14.2 Johnny Lee UZ Cobalt Cumulative Probability Plot



14.5 Johnny Lee UZ Domains

Mr. Vincent Scartozzi, Senior Project Geologist with Tintina, constructed 3D wireframes to represent the two Johnny Lee UZ stratabound copper sulphide horizons. Those horizons consist of a lower more extensive unit referred to as UZ unit 31, and an upper less extensive horizon referred to as UZ unit 32. RMI reviewed the wireframes and requested that Tintina make minor changes to exclude and/or include several drillhole intervals. XYZ hanging wall and footwall drillhole pierce points were used to create the initial wireframe solids. Criteria such as minimum thickness (approximately 3 m) and copper grade (roughly a 1% cut-off grade) were used in conjunction with logged lithologic/mineralization observations to construct the wireframe. Figure 14.3 is a perspective view looking N20W downward at the main and secondary UZ wireframes. Block grades were only estimated for UZ units 31 and 32. The percentage of each model block inside of the two UZ's wireframes was stored in the block model for more accurate tonnage tabulations.

Figure 14.3 Johnny Lee UZ Wireframe Perspective



14.6 JOHNNY LEE UZ COMPOSITING

One-metre-long drillhole composites were created starting and ending inside of the Johnny Lee UZ wireframes (zones 31 and 32). There were a total of 1,232 UZ unit 31 composites with 92% of them exactly 1 m in length; approximately 3% were between 0.50 to 1.0 m in length, and about 5% greater than 1.0 m in length. If the last sample interval in a bore hole was less than 0.5 m in length, it was added to the previous 1 m composite to ensure that no composite was less than 0.5 m in length. The maximum UZ

31 composite length was 1.49 m in length. There were a total of 168 UZ unit 32 composites with 85% of them exactly 1 m long, 4% between 0.50 and 1.0 m in length, and 11% greater than 1.0 m in length. The grade estimates were weighted by composite length.

14.7 JOHNNY LEE UZ VARIOGRAPHY

RMI generated a number of variograms for the Johnny Lee UZ using several software packages (MineSight® and Sage 2001). Copper grade, grade-thickness, and grade indicator variograms and correlograms were generated and modelled. Figure 14.4 shows a copper grade (high-grade outliers capped) correlogram that was generated from drillhole composites that were generated from UZ units 31 and 32. A single spherical model was used to fit the data points. Vectors were drawn at 80% and 95% of the total variance (red bisectors) to show the spread of range. The UZ 31/32 copper composites show a high nugget effect reflecting local variability.

Figure 14.4 Johnny Lee UZ Copper Correlogram

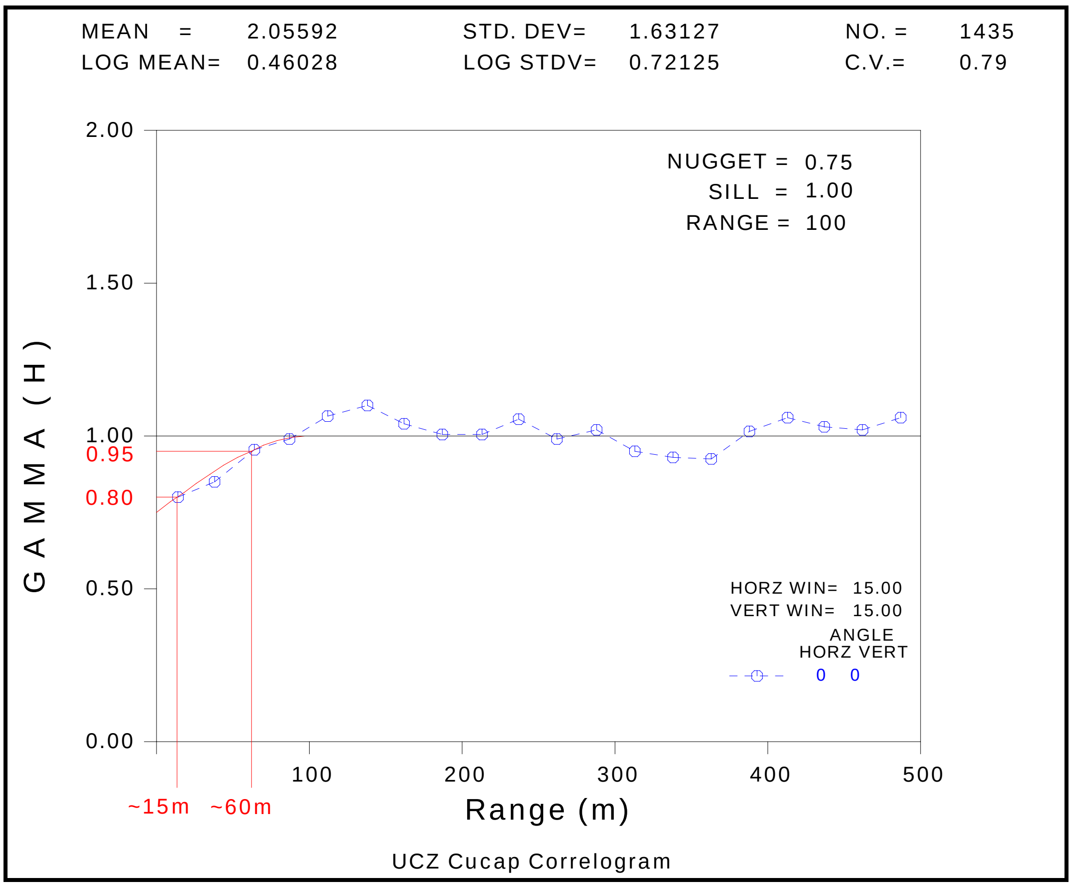


Figure 14.5 shows a 1% copper indicator correlogram that was generated from 1 m in long UZ 31/32 drillhole composites. Figure 14.6 shows a copper grade times thickness variogram (covariance function) that was generated from Johnny Lee UZ 31 composites that were generated from hanging wall to footwall contacts (100 drillholes).

Figure 14.5 Johnny Lee UZ 1% Copper Indicator Correlogram

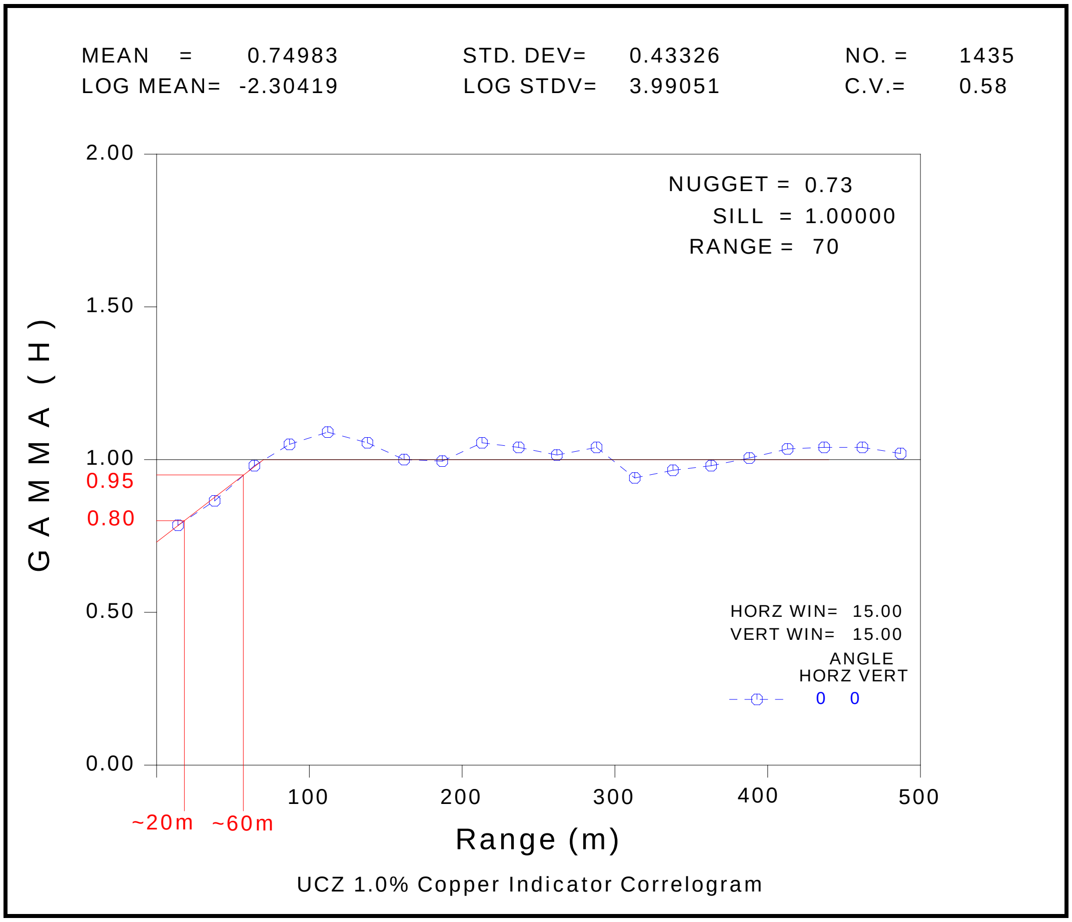
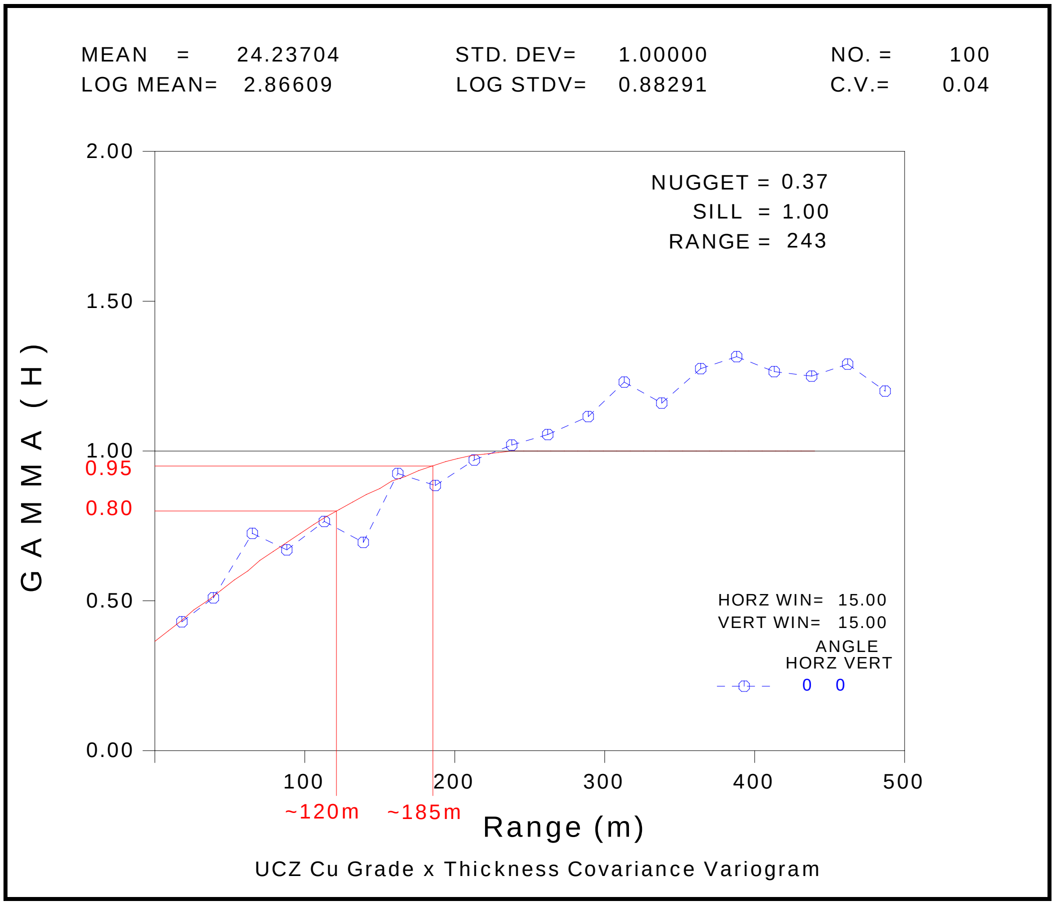


Figure 14.6 Johnny Lee UZ Copper Grade by Thickness Correlogram



14.8 JOHNNY LEE UZ GRADE ESTIMATION

RMI constructed a 3D block model using MineSight® software. Table 14.6 summarizes the limits of the model and size of the blocks.

Table 14.6 Block Model Limits

Parameter	Minimum	Maximum	Extent (m)	Size (m)	Number
Easting (columns)	506,200	506,925	725	5	145
Northing (rows)	5,180,000	5,181,250	1,250	5	250
Elevation (levels)	1,550	1,750	200	1	200

Because copper mineralization within the Johnny Lee UZ occurs as distinct stratabound layers within thick bedded sulphide accumulations RMI elected to use an estimation method which would provide constraints that would result in a distribution of block grades that closely follow bedding.

This method is based on selecting eligible composites to estimate each block based on the relative distance between each model block and the copper zone hanging wall and footwall contacts. The Cartesian distance between each block centroid and the upper and lower copper zone contacts was calculated and stored in the blocks. The relative distance between the hanging wall and footwall surfaces for each block was calculated using the following expression:

*relative distance (RELZ) = distance to footwall / (distance to footwall + distance to hanging wall) * 100*

For example, a RELZ value of 100 means that the block is located at the hanging wall contact while a RELZ distance of 0 means the block is located near the footwall contact. This method allows for a more uniform position of the block relative to irregular hanging wall and footwall contacts. The 1 m long drillhole composites were then backtagged with the block RELZ value. This ensured that the position of both the blocks and the drillholes relative to the zone contacts was established and could be used to select composites within similar stratigraphic positions as the blocks.

After examining various estimation methods, the author decided to use inverse distance (ID) methods to minimize grade smoothing, which is characteristic of ordinary kriging. Mineralization within the Johnny Lee UZ is quite stratabound so a method that minimizes smearing/smoothing was selected to provide a realistic distribution of grades. A two pass ID estimation plan was used for estimating copper, cobalt, silver, gold, lead, zinc, iron, sulphur, and barium. The first pass insured that blocks within the UZ wireframes (units 31 and 32) were filled with estimated block grades. The first pass used a maximum of three composites with no more than one composite per drillhole. The second pass required that at least two drillholes were required to estimate each block. The second estimation pass locally overwrote many of the block grades that were estimated by the first pass. RMI experimented with numerous combinations of ID power weights and minimum/maximum number of allowable samples. Comparisons were made between the ID block and nearest neighbour (NN) model grades at a zero cut-off grade. In order to minimize smearing of higher grade samples, RMI found that a limited number of samples should be used with an ID weighting power of 3.

The relative elevation option available in MineSight® was used to further select eligible drillhole composites. The actual Z or elevation coordinate for both the blocks and drillholes was substituted with the RELZ value that was previously described. A parameter (PAR20) in the estimation routine allowed RMI to open or restrict which composites could be used. For example, a block with a RELZ value of 50 (half way between the hanging wall and footwall contacts) could be estimated by composites with RELZ values of 50 ± the PAR20 value of 10. This means that the block with a RELZ value of 50 could be estimated by composites having RELZ values between 40 and 60, or 40 to 60% of the distance from the footwall. The resultant distribution of block grades appears to be very stratigraphic and is thought to be a good representation of the in situ distribution of copper grades.

As previously mentioned, the model blocks were coded with the two UZ wireframes so that an integer code (either 31 or 32) and the percentage of each block contained within the wireframes were stored. Along the margins of the wireframes some blocks contain a partial percentage of the wireframe or a mineralized component and an unmineralized component. Block grades were estimated for mineralized and unmineralized portions of each block using appropriate drillhole intervals (i.e. the mineralized portion of the block was estimated by drillhole composites located inside of the wireframe and the unmineralized portion of the block was estimated by only drillhole composites located outside of the wireframes). Grades were also estimated for blocks with no proportion of UZ horizon wireframes. This strategy will allow mine planners to use local information for dilution estimates.

The block estimation parameters are summarized in Table 14.7. The number of composites and drillholes used to estimate each block were captured along with the distance to the closest composite. These data were used to classify the blocks into Inferred Resources.

Table 14.7 Johnny Lee UZ Inverse Distance Estimation Parameters

Estimation Pass	ID Power	Composite Selection			Ellipse Dimensions (m)[1]			PAR20[2]
		Minimum	Maximum	Maximum/ Hole	Major Axis	Minor Axis	Vertical Axis	
1	3	1	3	1	200	200	200	10
2	3	2	3	1	200	200	200	10

Note: [1] The vertical axis range is replaced by the RELZ value.
[2] PAR20 refers to a ±RELZ tolerance for composite selection.

The apparent spherical ellipse dimension of 200 m is misleading because the relative elevation method and PAR20 parameter limits the actual search to a narrow band that parallels the hanging wall and footwall contacts of the wireframe. The number of composites used to estimate the block grades was based on comparisons made with a NN model. The limited number of composites used in the estimate define high, medium, and low-grade zones within the wireframe with the goal of minimizing grade smearing.

14.9 JOHNNY LEE UZ GRADE MODEL VERIFICATION

The estimated block grades were verified by visual and statistical methods. The block grades were compared with the drillhole composite grades in section and plan. It is RMI's opinion that the block grades look reasonable when compared with the sample data. Figure 14.8 is a plan map showing the outline of the two UZ massive sulphid ehorizons (UZ 31 and UZ 32), drillholes, and three lines of section. Figure 14.8 through Figure 14.12 are vertical cross sections through the block model showing composite and block copper grades.

Figure 14.7 UZ Plan Map

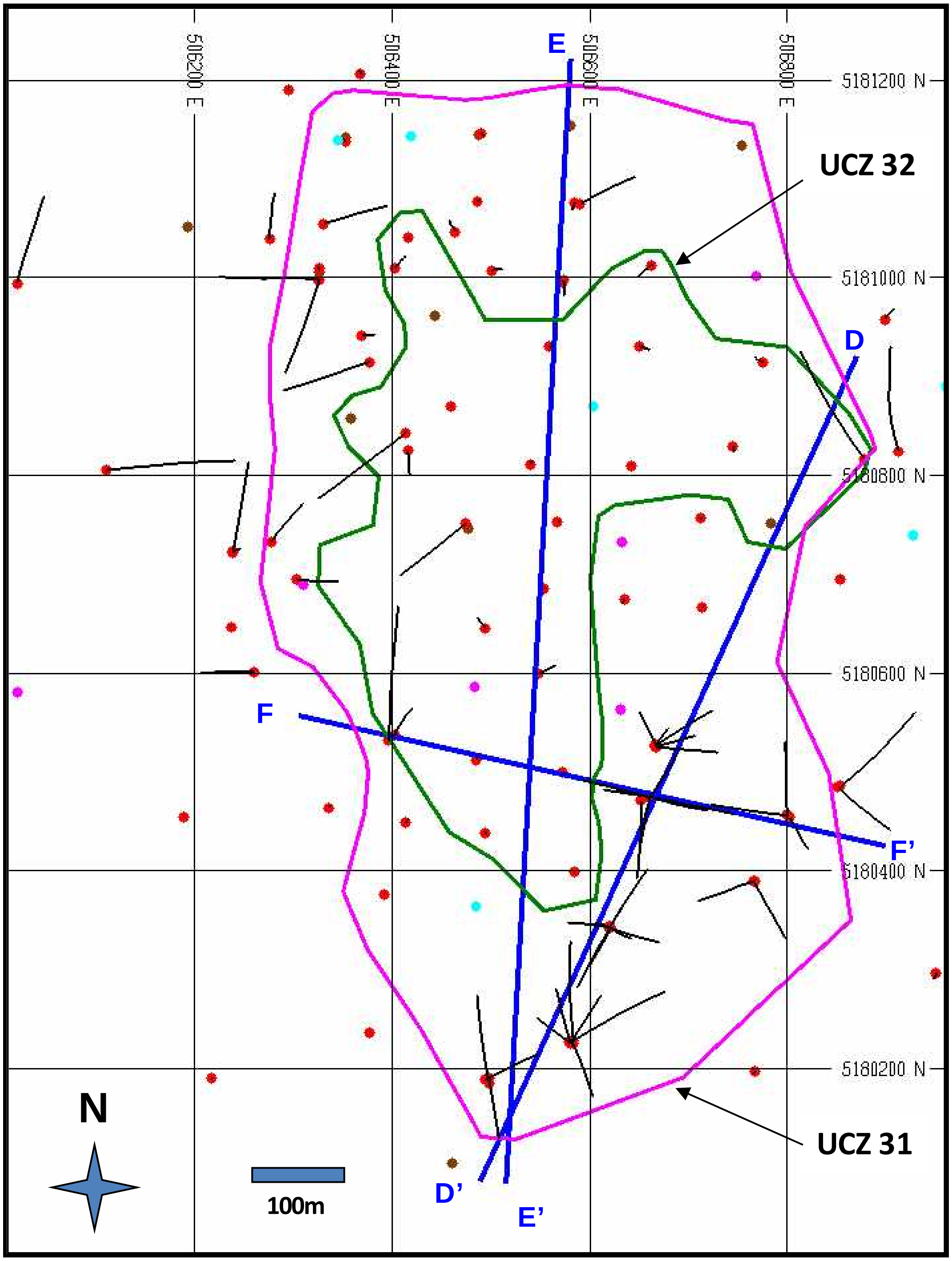


Figure 14.8 UZ Block Model Cross Section D-D'

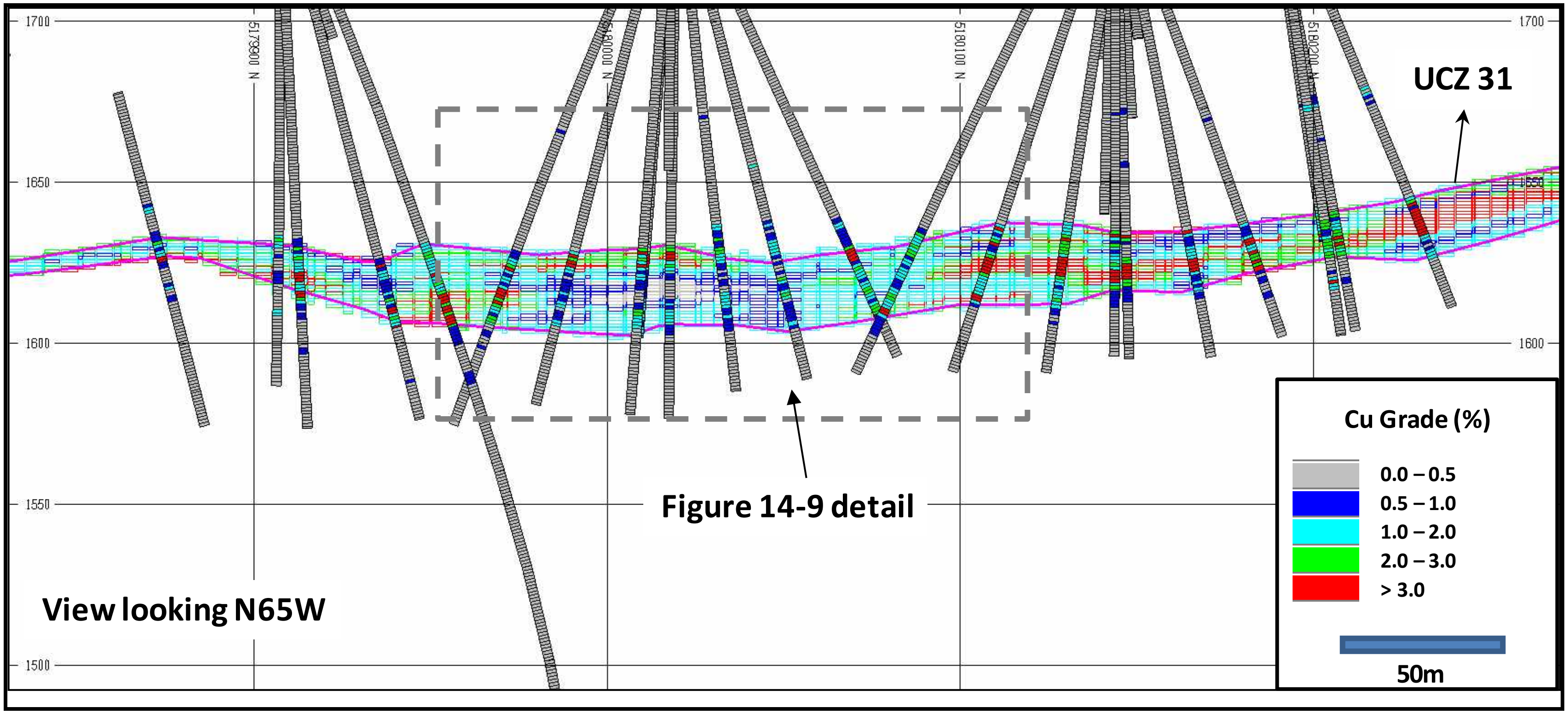


Figure 14.9 UZ Block Model Cross Section D-D' Detail

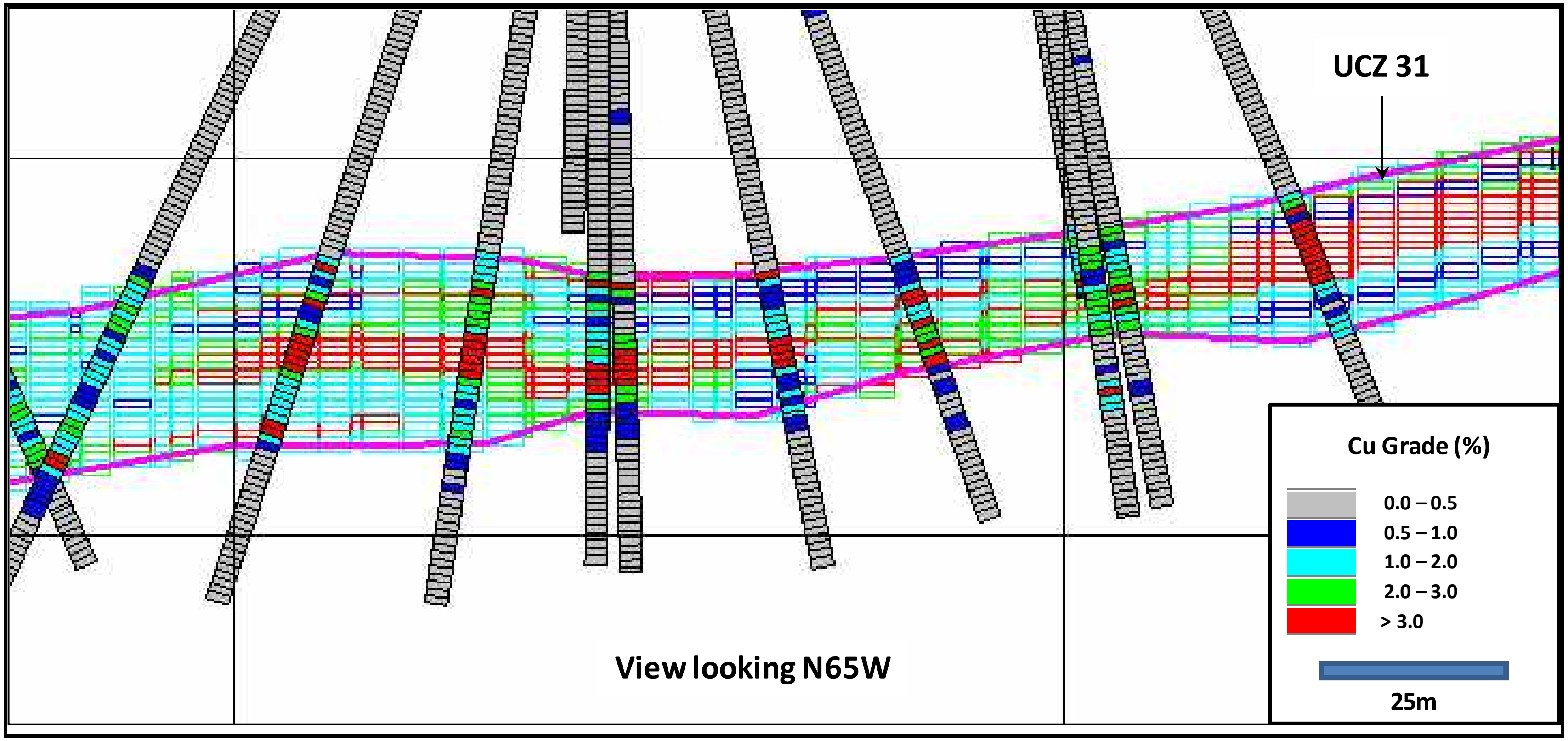


Figure 14.10 UZ Block Model Cross Section E-E'

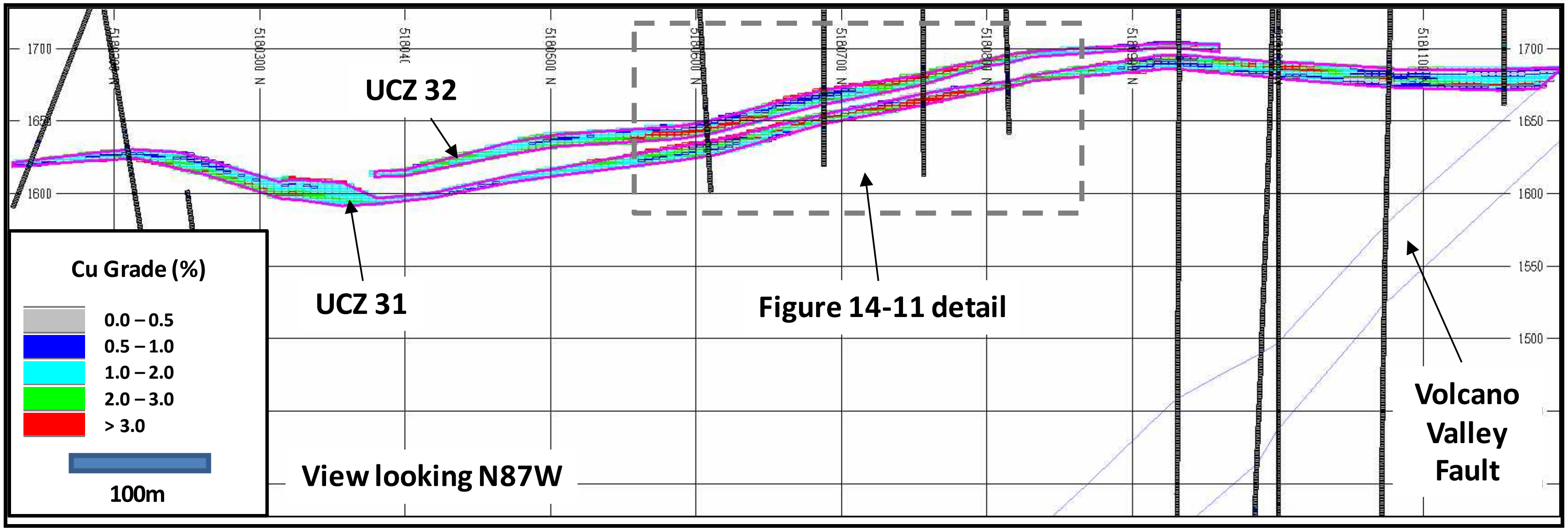


Figure 14.11 UZ Block Model Cross Section E-E' Detail

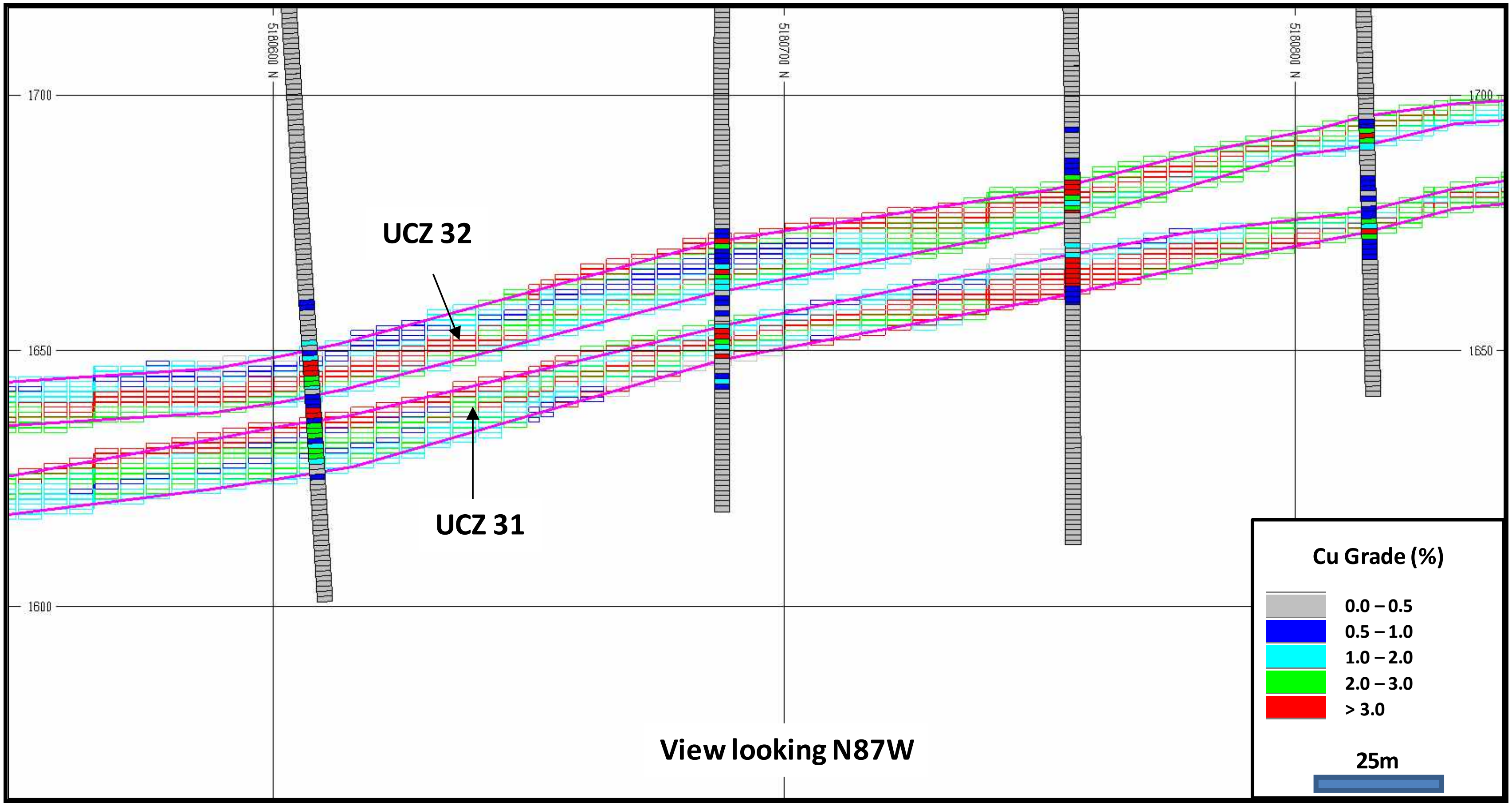
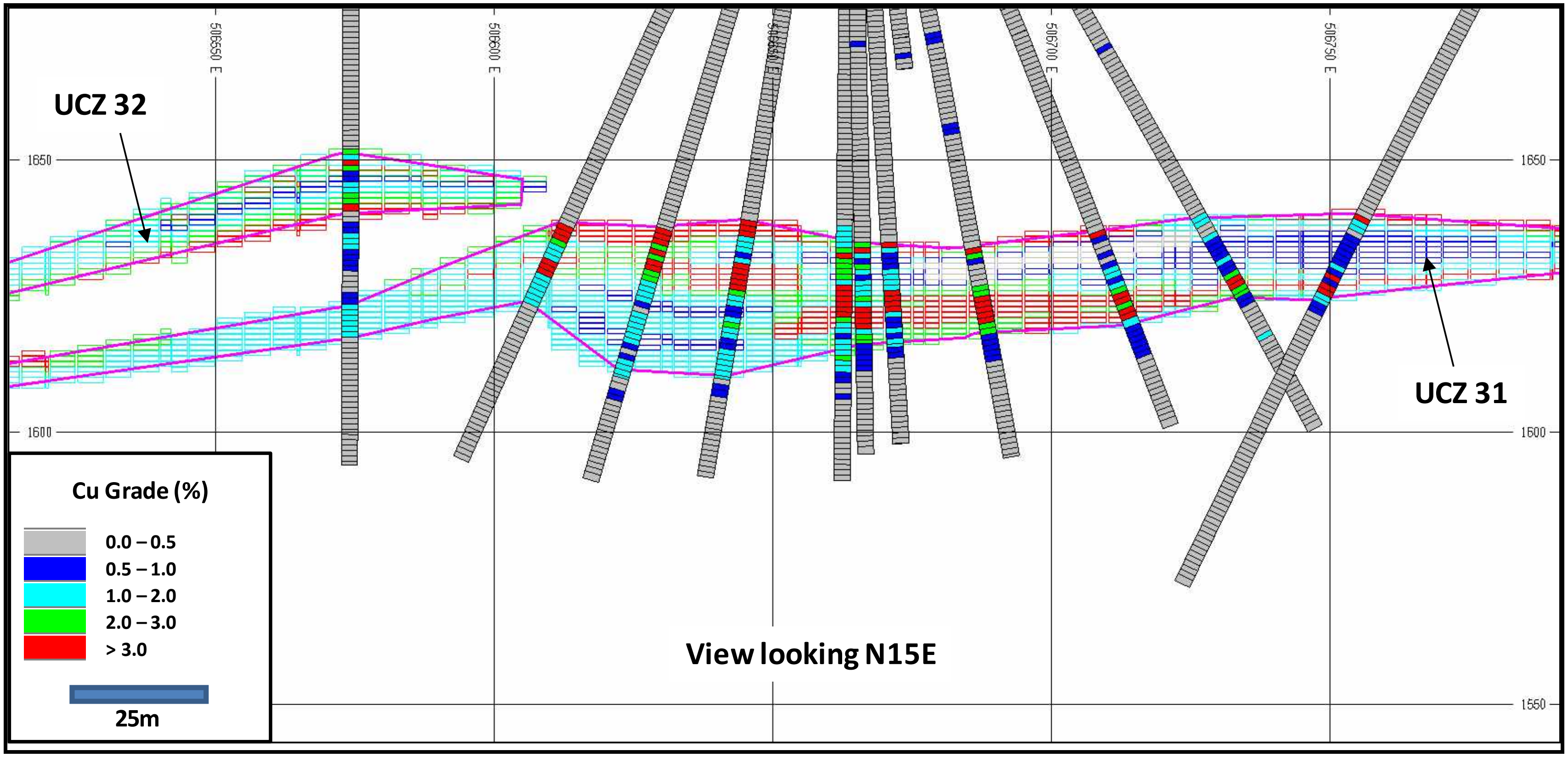


Figure 14.12 Block Model Cross Section F-F'



NN models were constructed for the primary metals of interest (i.e. copper, cobalt, silver, and gold). To check for possible global biases in the block model, the inverse distance weighted (IDW) grades were compared with the NN models for Measured and Indicated blocks at a zero cut-off grade. Table 14.8 shows the comparisons.

Table 14.8 Johnny Lee UZ Global Bias Check

Metal	ID Resource Grade	NN Grade	Difference (%)
Copper (%)	2.0641	2.0692	-0.25
Cobalt (ppm)	0.1008	0.1003	0.50
Gold (g/t)	0.0101	0.0102	-0.98
Silver (g/t)	15.49	15.60	-0.68

The data in Table 14.8 show a close comparison between the IDW and NN grades and show that the model is globally unbiased. Based on industry accepted practice models that show less than a 5% variance from a NN grade model are unbiased.

RMI also checked for local biases by creating a series of slices or "swaths" through the model columns (eastings), rows (northings), and levels (elevations) comparing the IDW and NN grades. Figure 14.13 through Figure 14.15 show the local variation between the IDW and NN copper models at a zero cut-off grade. The ID grade (cuidw) is shown in red, the NN grade (cunn) is shown in blue and the number of blocks per "swath" are shown by the black line which is read from the right side Y-axis.

Figure 14.13 Johnny Lee UZ Copper Swath Plot – Easting

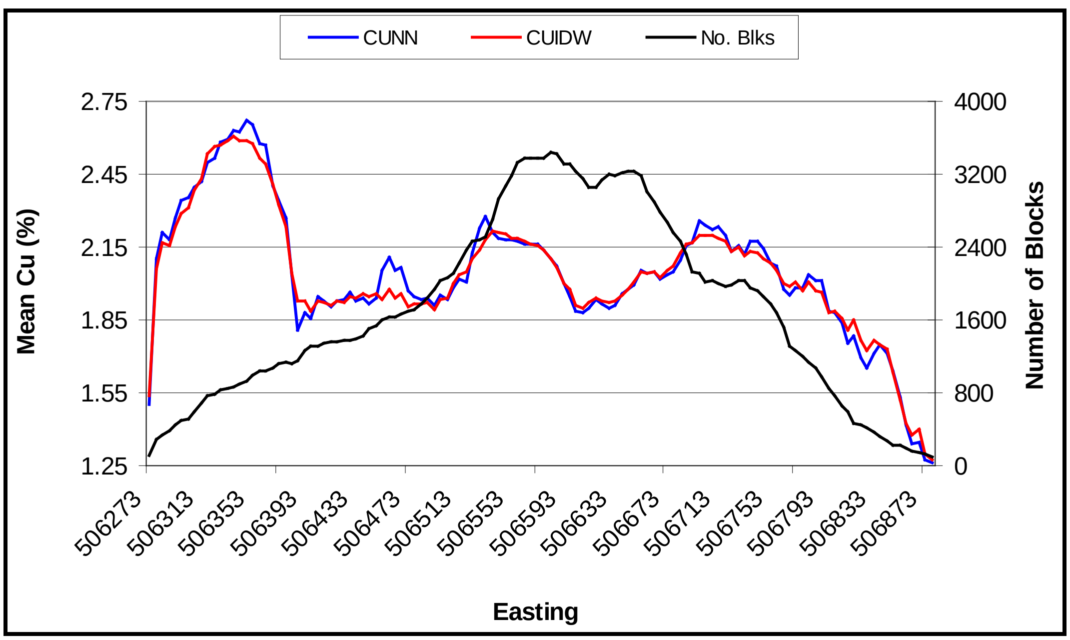


Figure 14.14 Johnny Lee UZ Copper Swath Plot – Northing

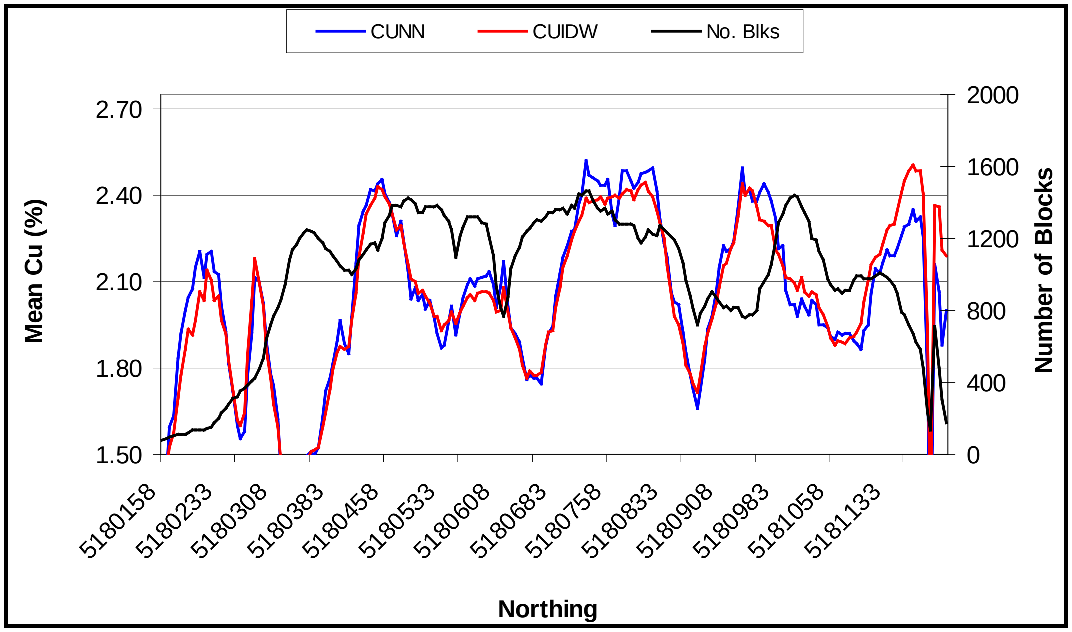
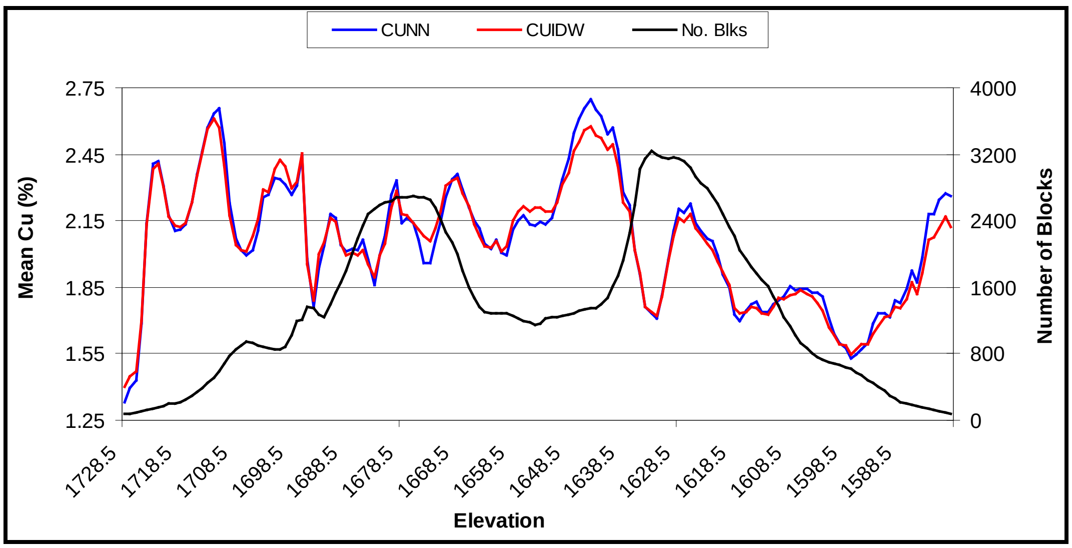


Figure 14.15 Johnny Lee UZ Copper Swath Plot – Elevation



The swath plots shown in Figure 14.14 through Figure 14.15 show some local variation with the NN model grades showing more variation and the IDW grades showing some smoothing. These graphs also provide an indication as to were higher grades exist.

Based on a visual examination and comparisons with a NN model, it is RMI's opinion that the model is globally unbiased and represents a reasonable estimate of undiluted (no external dilution) in-situ resources.

14.10 Johnny Lee UZ Resource Classification

Estimated blocks inside of the UZ wireframes (units 31 and 32) were classified as Measured Resources if they were estimated by two or more holes with one hole within 20 m. Blocks were classified as Indicated Resources if they were estimated by two more holes with one hole within 50 m and not classified as Measured. All other blocks inside of the two wireframes were classified as Inferred Resources.

14.11 Johnny Lee UZ Density Data

Tintina personnel have obtained bulk density determinations from 577 pieces of drill core taken from their 2010, 2011 and 2012 drilling programs in the Johnny Lee UZ. The average bulk density from these 577 samples was 3.60 g/cm^3. These determinations are representative of the various lithologic units including UZ massive sulphide horizons. The core was not rigorously dried but was not thought to contain much moisture. The core was weighed in-air and then weighed while submerged in water. A relative bulk density calculation was then made (bulk density = weight in-air/(weight in-air – weight in water).

Based on 181 UZ massive sulphide determinations, a bulk density value of 3.99 g/cm^3 was selected by RMI for Johnny Lee UZ horizons 31 and 32. Based on 357 determinations an average bulk density of 3.60 g/cm^3 was selected for non-copper sulphide zones within the bedded sulphide package. A bulk density value of 3.07 g/cm^3 was assigned to all other model blocks.

14.12 Black Butte Topographic Data

Surface topographic data were obtained from the US Geological Survey (USGS) website (seamless.usgs.gov) by selecting an area of interest (AOI) around the Project area and downloading the data as a standard digital elevation model (DEM) file. This data has a resolution of about 1/9 arcsecond or approximately 3 m and was in North American Datum (NAD)83 units. The data was translated into WGS84 datum using Manifold GIS software (version 8). The resultant XYZ topographic points were then triangulated into a surface using MineSight®. RMI compared the elevation of the surveyed drillhole collar locations against the topographic surface and found a close correspondence.

14.13 Johnny Lee UZ Resource Summary

A cut-off grade of 1.6% copper was used to define a Measured and Indicated Mineral Resource of 179,000 t with an average grade of 2.83% copper, 0.12% cobalt, 15.7 g/t silver, and 0.008 g/t gold. In addition to Measured and Indicated Resources there is an Inferred Resource of 1,255,000 t with an average grade of 2.52% copper, 0.10% cobalt, 15.2 g/t silver, and 0.008 g/t gold using a 1.6% copper cut-off grade. No external dilution factors were applied to these resources, although drillhole intersections within the wireframes used to define the resource contained dilutant material.

The cut-off grade was established by using:

- a copper price of US$2.75/lb
- a copper recovery of 81%
- mining costs of US$59/t
- processing costs of US$16.00/t
- G&A costs of US$5.00/t.

The price, costs, and recovery shown above were used to calculate a cut-off grade of 1.6% copper for the Johnny Lee UZ. In the author's opinion, this cut-off grade demonstrates reasonable prospects for economic extraction.

Table 14.9 to Table 14.11 summarizes Johnny Lee UZ Measured, Indicated and Inferred Resources at a number of cut-off grades, respectively. The disclosed Johnny Lee UZ Measured Resource is highlighted in grey. No credit was given to cobalt or silver in determining the cut-off grade.

Table 14.9 Johnny Lee UZ Undiluted Measured Mineral Resource

Cu Cut-off (%)	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
1.0	4,128	2.38	0.104	0.009	16.3	217	9.5	1.2	2,163
1.1	3,836	2.48	0.106	0.008	16.3	210	9.0	1.0	2,010
1.2	3,566	2.58	0.109	0.008	16.3	203	8.6	0.9	1,869
1.3	3,280	2.70	0.111	0.008	16.2	195	8.0	0.8	1,708
1.4	3,047	2.80	0.113	0.008	16.1	188	7.6	0.8	1,577
1.5	2,862	2.89	0.115	0.008	16.2	182	7.3	0.7	1,491
1.6	**2,659**	**2.99**	**0.118**	**0.007**	**16.3**	**175**	**6.9**	**0.6**	**1,393**
1.7	2,426	3.12	0.121	0.007	16.3	167	6.5	0.5	1,271
1.8	2,238	3.24	0.124	0.007	16.3	160	6.1	0.5	1,173
1.9	2,092	3.34	0.127	0.007	16.1	154	5.9	0.5	1,083
2.0	1,931	3.45	0.131	0.007	16.1	147	5.6	0.4	1,000
2.1	1,807	3.55	0.133	0.007	15.9	141	5.3	0.4	924
2.2	1,696	3.64	0.136	0.007	16.0	136	5.1	0.4	872
2.3	1,584	3.74	0.139	0.007	16.1	131	4.9	0.4	820
2.4	1,511	3.80	0.139	0.007	16.1	127	4.6	0.3	782
2.5	1,434	3.88	0.141	0.007	16.0	123	4.5	0.3	738

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.10 Johnny Lee UZ Undiluted Indicated Mineral Resource

Cu Cut-off (%)	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
1.0	9,839	2.27	0.110	0.010	15.7	492	23.9	3.2	4,966
1.1	9,281	2.35	0.112	0.010	15.6	481	22.9	3.0	4,655
1.2	8,715	2.42	0.115	0.010	15.6	465	22.1	2.8	4,371
1.3	8,116	2.51	0.117	0.010	15.5	449	20.9	2.6	4,044
1.4	7,519	2.60	0.120	0.010	15.5	431	19.9	2.4	3,747
1.5	7,022	2.68	0.123	0.009	15.5	415	19.0	2.0	3,499
1.6	**6,520**	**2.77**	**0.125**	**0.009**	**15.5**	**398**	**18.0**	**1.9**	**3,249**
1.7	5,996	2.87	0.129	0.009	15.5	379	17.0	1.7	2,988
1.8	5,516	2.97	0.132	0.009	15.4	361	16.0	1.6	2,731
1.9	5,075	3.06	0.135	0.009	15.3	342	15.1	1.5	2,496
2.0	4,668	3.16	0.138	0.009	15.2	325	14.2	1.4	2,281
2.1	4,320	3.25	0.141	0.009	15.1	309	13.4	1.3	2,097
2.2	3,999	3.34	0.144	0.009	15.0	294	12.7	1.2	1,929
2.3	3,706	3.42	0.146	0.008	15.0	279	11.9	1.0	1,787
2.4	3,441	3.51	0.148	0.008	14.9	266	11.2	0.9	1,648
2.5	3,173	3.60	0.151	0.008	14.8	252	10.6	0.8	1,510

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.11 Johnny Lee UZ Undiluted Inferred Mineral Resource

Cu Cut-off (%)	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
1.0	2,482	1.90	0.084	0.008	16.0	104	4.6	0.6	1,277
1.1	2,267	1.99	0.087	0.008	15.8	99	4.3	0.6	1,152
1.2	2,034	2.08	0.089	0.008	15.8	93	4.0	0.5	1,033
1.3	1,798	2.19	0.092	0.008	15.7	87	3.6	0.5	908
1.4	1,580	2.31	0.096	0.008	15.7	80	3.3	0.4	798
1.5	1,404	2.42	0.099	0.008	15.4	75	3.1	0.4	695
1.6	**1,255**	**2.52**	**0.102**	**0.008**	**15.2**	**70**	**2.8**	**0.3**	**613**
1.7	1,122	2.62	0.105	0.008	15.0	65	2.6	0.3	541
1.8	1,010	2.72	0.108	0.009	14.9	61	2.4	0.3	484
1.9	909	2.81	0.110	0.009	14.8	56	2.2	0.3	433
2.0	819	2.91	0.114	0.009	14.7	53	2.1	0.2	387
2.1	737	3.01	0.117	0.009	14.7	49	1.9	0.2	348
2.2	658	3.11	0.120	0.009	14.7	45	1.7	0.2	311
2.3	592	3.20	0.123	0.009	14.7	42	1.6	0.2	280
2.4	535	3.29	0.126	0.009	14.7	39	1.5	0.2	253
2.5	483	3.39	0.129	0.009	14.8	36	1.4	0.1	230

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

14.14 General Discussion – Johnny Lee UZ Resource

RMI is not aware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other factors that could materially affect the Johnny Lee UZ Inferred Mineral Resources discussed in this report.

14.15 Johnny Lee LZ Drilling Data

Like the updated Johnny Lee UZ resource, the Johnny Lee LZ resource is based on a majority of drilling data that has been recently collected by Tintina. Data collected by previous operators (i.e. CAI, UII, and BHP) were used in conjunction with the newly acquired Tintina drilling data to estimate mineral resources for the Johnny Lee LZ. Table 14.12 summarizes the drillhole data that were used by RMI to estimate mineral resources for the Johnny Lee LZ. The information in Table 14.12 includes the company that drilled the hole, beginning and ending depth of the horizon, the LZ intersection length, average copper, cobalt, silver, and gold grades for the LZ intersections, and which LZ zone (i.e. 11=high-grade zone and 12=low-grade zone).

Table 14.12 Johnny Lee LZ Drillhole Data

Drillhole	Company	From Depth (m)	To Depth (m)	Length (m)	LZ Intersections				LZ Zone
					Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)	
SC10-003	Tintina	347.70	351.69	3.99	1.79	0.03	3.6	0.26	11
SC10-004	Tintina	414.00	418.05	4.05	10.84	0.08	8.3	0.21	11
SC10-005	Tintina	405.80	412.15	6.35	8.72	0.11	5.0	0.05	11
SC11-007	Tintina	409.66	411.24	1.58	1.38	0.01	3.2	0.07	11
SC11-008	Tintina	355.22	357.40	2.18	2.45	0.04	6.6	0.70	11
SC11-009	Tintina	395.88	399.00	3.12	0.04	0.01	4.9	0.56	12
SC11-010	Tintina	456.60	458.75	2.15	0.73	0.03	20.0	0.05	12
SC11-011	Tintina	409.65	422.70	13.05	3.18	0.02	2.5	0.35	11
SC11-012	Tintina	363.61	369.61	6.00	1.58	0.02	3.3	0.08	11
SC11-015	Tintina	449.29	456.59	7.30	3.14	0.04	6.1	0.46	11
SC11-023	Tintina	421.35	424.59	3.24	0.19	0.28	5.1	0.89	12
SC11-029	Tintina	437.00	440.63	3.63	13.97	0.02	7.5	0.23	11
SC11-031	Tintina	426.08	428.24	2.16	0.64	0.03	2.0	0.01	12
SC11-032	Tintina	381.25	383.13	1.88	0.05	0.03	6.4	1.12	12
SC11-036	Tintina	364.11	366.63	2.52	0.04	0.02	5.1	2.25	12
SC11-039	Tintina	340.70	343.75	3.05	0.05	0.01	5.1	0.22	12
SC11-048	Tintina	356.87	367.60	10.73	5.27	0.06	4.8	0.50	11
SC12-100	Tintina	412.00	424.10	12.10	8.55	0.03	2.6	0.55	11
SC12-101	Tintina	382.95	397.75	14.80	5.60	0.04	2.5	0.27	11
SC12-102	Tintina	429.70	441.35	11.65	3.18	0.10	3.1	0.25	11
SC12-103	Tintina	444.50	447.15	2.65	14.09	0.01	2.7	0.69	11

table continues...

Drillhole	Company	From Depth (m)	To Depth (m)	Length (m)	LZ Intersections Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)	LZ Zone
SC12-104	Tintina	460.10	477.43	17.33	8.32	0.04	7.9	0.28	11
SC12-105	Tintina	387.60	390.90	3.30	0.08	0.03	6.2	0.07	12
SC12-106	Tintina	363.88	366.41	2.53	0.03	0.01	1.5	0.27	12
SC12-107	Tintina	437.08	440.62	3.54	0.06	0.04	11.7	0.55	12
SC12-109	Tintina	391.41	396.34	4.93	0.07	0.03	10.1	0.37	12
SC12-110	Tintina	405.34	408.04	2.70	2.33	0.02	3.8	0.04	11
SC12-123	Tintina	361.75	363.93	2.18	10.76	0.01	6.2	0.75	11
SC12-124	Tintina	360.32	363.90	3.58	4.40	0.02	3.4	0.37	11
SC12-129	Tintina	325.22	327.40	2.18	0.01	0.00	2.3	0.27	12
SC12-130	Tintina	399.00	401.00	2.00	0.16	0.03	6.5	0.67	12
SC12-142	Tintina	340.50	349.54	9.04	3.24	0.02	2.5	0.40	11
SC-50	CAI	367.89	370.33	2.44	7.75	0.01	3.2	0.40	11
SC-51	CAI	397.61	404.77	7.16	5.80	0.01	1.7	0.19	11
SC-52	CAI	330.71	331.93	1.22	1.50	0.02	6.8	0.11	11
SC-55	CAI	463.60	470.31	6.71	10.12	0.02	12.5	0.43	11
SC-57	CAI	482.50	486.16	3.66	6.47	0.02	6.3	0.31	11
SC-58	CAI	478.54	480.06	1.52	0.02	0.02	9.2	0.00	11
SC-63	CAI	457.20	462.08	4.88	0.60	0.07	5.8	0.43	11
SC-90	CAI	383.26	384.54	1.28	11.64	0.02	10.9	0.10	11
SC-91	CAI	310.29	310.96	0.67	0.02	0.01	9.9	0.05	11
SCC-17	CAI	355.70	358.14	2.44	6.82	0.05	3.0	0.34	11
SCC-20	CAI	343.05	344.97	1.92	1.21	0.02	1.8	0.13	11
SCC-21	CAI	394.56	400.66	6.10	4.78	0.04	4.0	0.24	11
SCC-34	CAI	413.61	418.49	4.88	7.56	0.15	7.6	0.41	11
SCC-36	BHP	365.15	367.89	2.74	0.07	0.04	4.3	1.26	12
SCC-46	BHP	400.35	412.76	12.41	5.71	0.03	2.4	0.27	11
Total/Average	**n/a**	**n/a**	**n/a**	**231.52**	**4.78**	**0.04**	**4.95**	**0.37**	**n/a**

14.16 JOHNNY LEE LZ EXPLORATORY DATA ANALYSIS

The Johnny Lee LZ consists of several lenses of massive sulphide mineralization. Only the thickest and most continuous horizon was modelled. Basic copper assay statistics were tabulated at four different cut-off grades and are summarized in Table 14.13. The data summarized in Table 14.13 include the number of metres at each cut-off grade, mean grades, standard deviations, and CVs for the main high-grade portion of the LZ (i.e. LCZ 11) and a low-grade portion of the LZ (i.e. LCZ 12). Incremental data (i.e. statistics for material between cut-off grades) are also tabulated. For example, 64% of the LZ intersections are above a 1% copper cut-off grade, with 36% less than that cut-off. Forty-five percent of the total hanging wall to footwall LCZ assays are greater than 3% copper. The majority of the 47 drillholes that intersected the LCZ horizon contain high-grade copper (LCZ unit 11) but about 17% of the total LCZ meterage is quite low-grade

(LCZ unit 12) for copper. The LCZ unit 12 intersections however, contain higher precious metal grades than the high-grade copper zone.

Table 14.13 Johnny Lee LZ Copper Assay Statistics

LZ	Cu Cut-off (%)	Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Standard Deviation	CV
All Data	0.00	232	36	4.78	1,108	1.6	5.60	1.17
	1.00	147	13	7.40	1,090	3.7	5.51	0.74
	2.00	116	6	9.03	1,050	3.0	5.10	0.56
	3.00	103	45	9.86	1,016	91.7	4.81	0.49
11	0.00	192	24	5.74	1,102	1.2	5.70	0.99
	1.00	146	15	7.47	1,089	3.5	5.51	0.74
	2.00	116	7	9.03	1,050	3.0	5.10	0.56
	3.00	103	54	9.86	1,016	92.2	4.81	0.49
12	0.00	39	96	0.14	5	71.0	0.23	1.64
	1.00	1	4	1.06	2	29.0	0.10	0.09
	2.00	0	0	0.00	0	0.0	0.00	0.00
	3.00	0	0	0.00	0	0.0	0.00	0.00

Similar assay statistics were tabulated for cobalt, silver, and gold and are summarized in Table 14.14 to Table 14.16, respectively.

Table 14.14 Johnny Lee LZ Cobalt Assay Statistics

LZ	Co Cut-off (%)	Total Metres	Inc. Percent	Mean Co (%)	Grd-Thk (%-m)	Inc. Percent	Standard Deviation	CV
All Data	0.00	232	17%	0.04	10	2.6%	0.09	2.05
	0.01	191	65%	0.05	9	37.2%	0.09	1.88
	0.05	42	11%	0.14	6	18.2%	0.17	1.21
	0.10	16	7%	0.26	4	42.0%	0.23	0.88
11	0.00	192	18%	0.04	8	2.9%	0.08	1.96
	0.01	157	62%	0.05	8	35.6%	0.09	1.79
	0.05	37	12%	0.13	5	20.0%	0.15	1.16
	0.10	14	7%	0.23	3	41.5%	0.21	0.91
12	0.00	39	13%	0.05	2	1.6%	0.11	2.36
	0.01	34	76%	0.05	2	43.9%	0.12	2.21
	0.05	4	7%	0.24	1	10.4%	0.27	1.11
	0.10	1	3%	0.65	1	44.1%	0.00	0.00

Table 14.15 Johnny Lee LZ Silver Assay Statistics

LZ	Ag Cut-off (g/t)	Total Metres	Inc. Percent	Mean Ag (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Standard Deviation	CV
All Data	0.00	232	60	4.96	1,148	27.8	4.40	0.89
	5.00	92	26	9.06	829	36.5	4.36	0.48
	10.00	31	13	13.40	410	34.4	4.95	0.37
	30.00	0	0	33.30	15	1.3	0.00	0.00
11	0.00	192	64	4.59	883	31.8	4.02	0.88
	5.00	69	24	8.77	602	36.3	3.97	0.45
	10.00	22	11	12.92	282	30.2	4.44	0.34
	30.00	0	0	33.30	15	1.7	0.00	0.00
12	0.00	39	42	6.73	265	14.4	5.57	0.83
	5.00	23	36	9.93	227	37.4	5.27	0.53
	10.00	9	22	14.58	128	48.3	5.86	0.40
	30.00	0	0	0.00	0	0.0	0.00	0.00

Table 14.16 Johnny Lee LZ Gold Assay Statistics

LZ	Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Standard Deviation	CV
All Data	0.00	232	50	0.37	85	14.2	0.41	1.10
	0.25	115	27	0.64	73	26.6	0.43	0.67
	0.50	53	14	0.95	50	24.2	0.46	0.48
	1.00	22	9	1.37	30	35.0	0.44	0.32
11	0.00	192	53	0.32	62	17.0	0.32	0.98
	0.25	91	27	0.57	51	30.8	0.31	0.54
	0.50	39	14	0.83	32	29.1	0.30	0.36
	1.00	11	6	1.26	14	23.1	0.16	0.13
12	0.00	39	38	0.59	23	6.8	0.64	1.09
	0.25	24	25	0.89	22	15.5	0.66	0.74
	0.50	14	10	1.25	18	11.1	0.63	0.51
	1.00	10	26	1.49	15	66.6	0.59	0.40

14.17 JOHNNY LEE LZ HIGH-GRADE OUTLIERS

RMI generated a series of cumulative probability plots after transforming the original copper, cobalt, silver, and gold assays using the cumulative normal distribution method. Figure 14.16 through Figure 14.19 show copper, cobalt, silver, and gold probability plots for the Johnny Lee LZ, respectively. The black circle shown in these figures are capping limits selected by RMI to minimize the potential for over estimating contained metal. Table 14.17 summarizes high-grade outlier capping limits for copper and cobalt for the Johnny Lee LZ. These limits were applied to the raw assays prior to creating drillhole composites.

Table 14.17 Johnny Lee LZ Grade Capping Limits

	Cap Limit	No. Capped
Copper	20.0%	3
Cobalt	0.20%	6
Silver	20.0 g/t	3
Gold	1.25 g/t	10

Figure 14.16 Johnny Lee LZ Cumulative Probability Plot – Copper

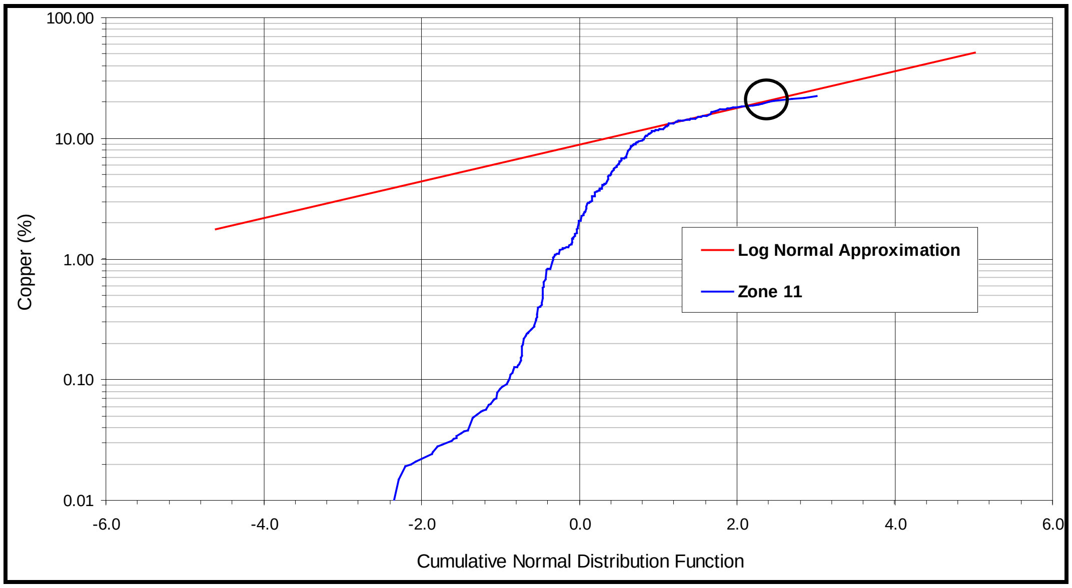


Figure 14.17 Johnny Lee LZ Cumulative Probability Plot – Cobalt

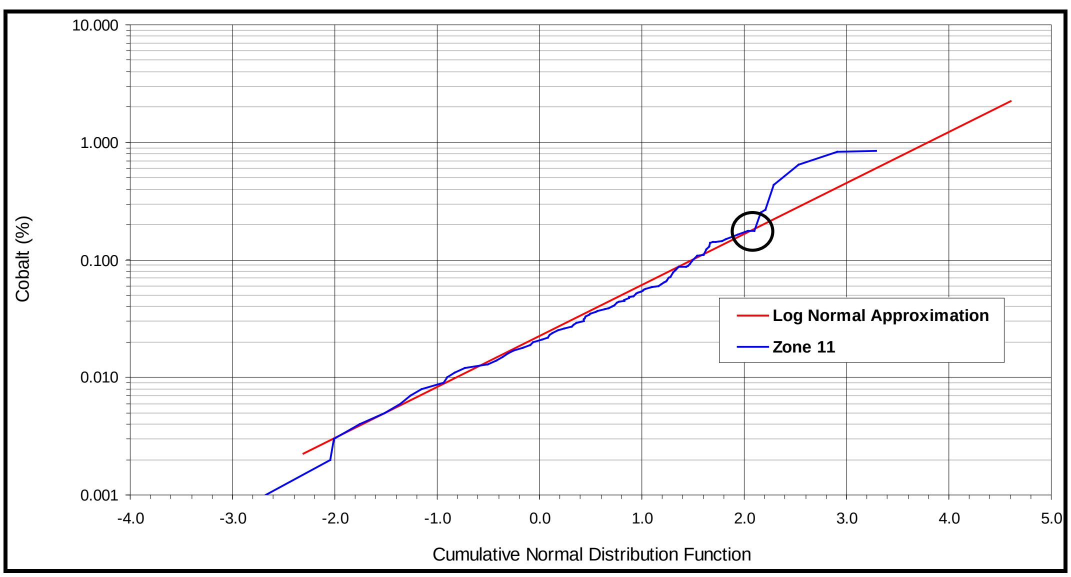


Figure 14.18 Johnny Lee LZ Cumulative Probability Plot – Silver

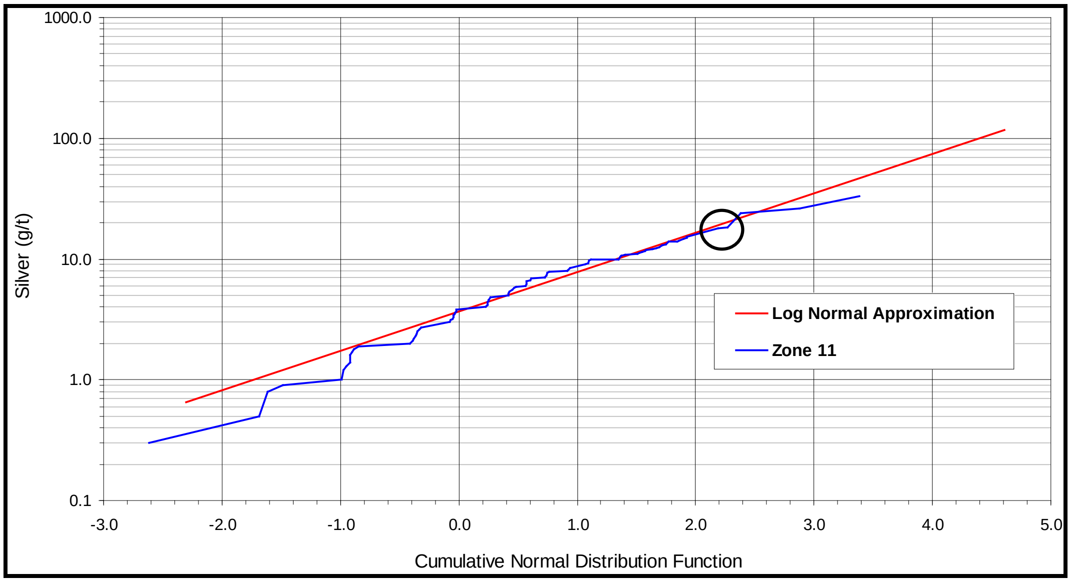
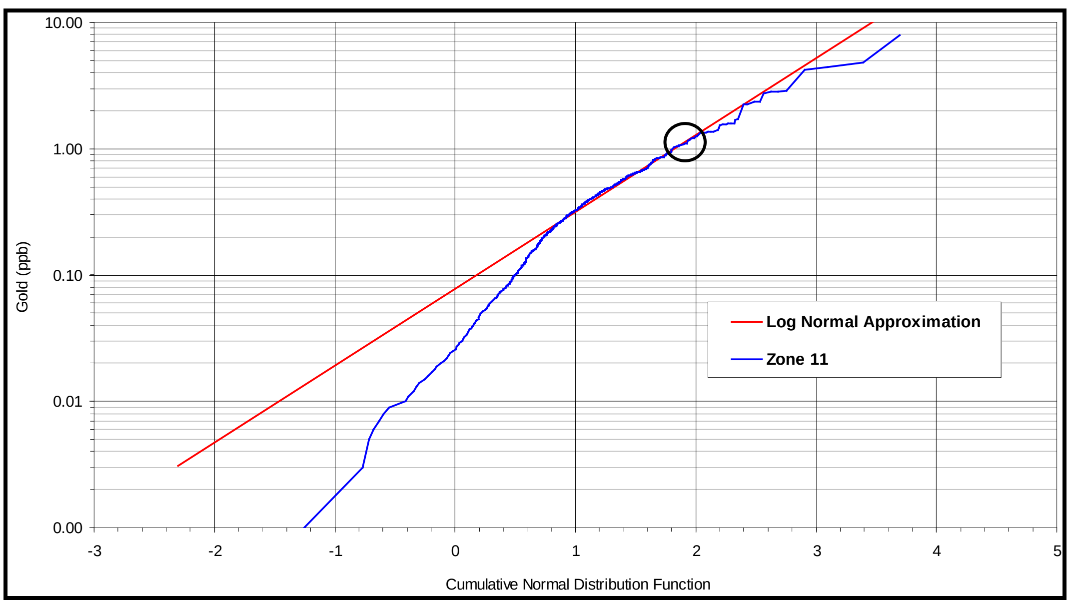


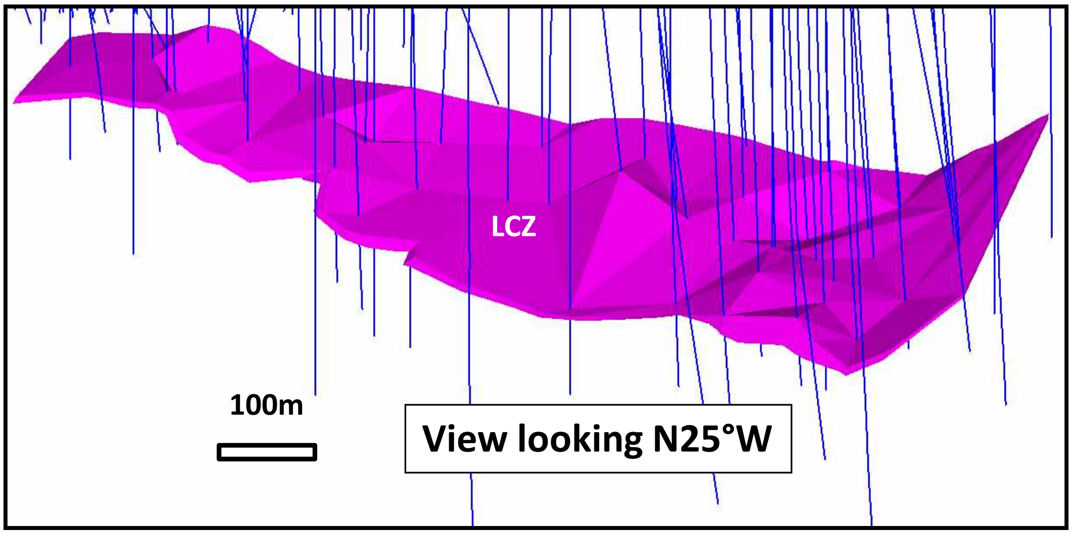
Figure 14.19 Johnny Lee LZ Cumulative Probability Plot – Gold



14.18 JOHNNY LEE LZ DOMAIN

Mr. Vincent Scartozzi, Senior Geologist with Tintina, constructed a 3D wireframe to represent the Johnny Lee LZ stratabound copper sulphide horizon. RMI reviewed the wireframes and requested that Tintina make minor changes to exclude and/or include several drillhole intervals. XYZ hanging wall and footwall drillhole pierce points were used to create the initial wireframe solid. Criteria such as minimum thickness (approximately 2 m) and copper grade (roughly a 2% cut-off grade) were used in conjunction with logged lithologic/mineralization observations to construct the wireframe. The wireframe was extended approximately 30 to 40 m outward from the perimeter drillholes that intersected the horizon. The Johnny Lee LZ wireframe was then intersected with two district fault structures (the VVF and the Buttress Fault). Figure 14.20 is a perspective view looking N62E at the Johnny Lee LZ horizon (red) which is shown to be truncated by the VVF and Buttress Fault shown in blue hues. Block grades were only estimated for Johnny Lee LZ. The percentage of each model block inside of the Johnny Lee LZ wireframe was stored in the block model for more accurate tonnage calculations.

Figure 14.20 Johnny Lee LZ Wireframe Perspective



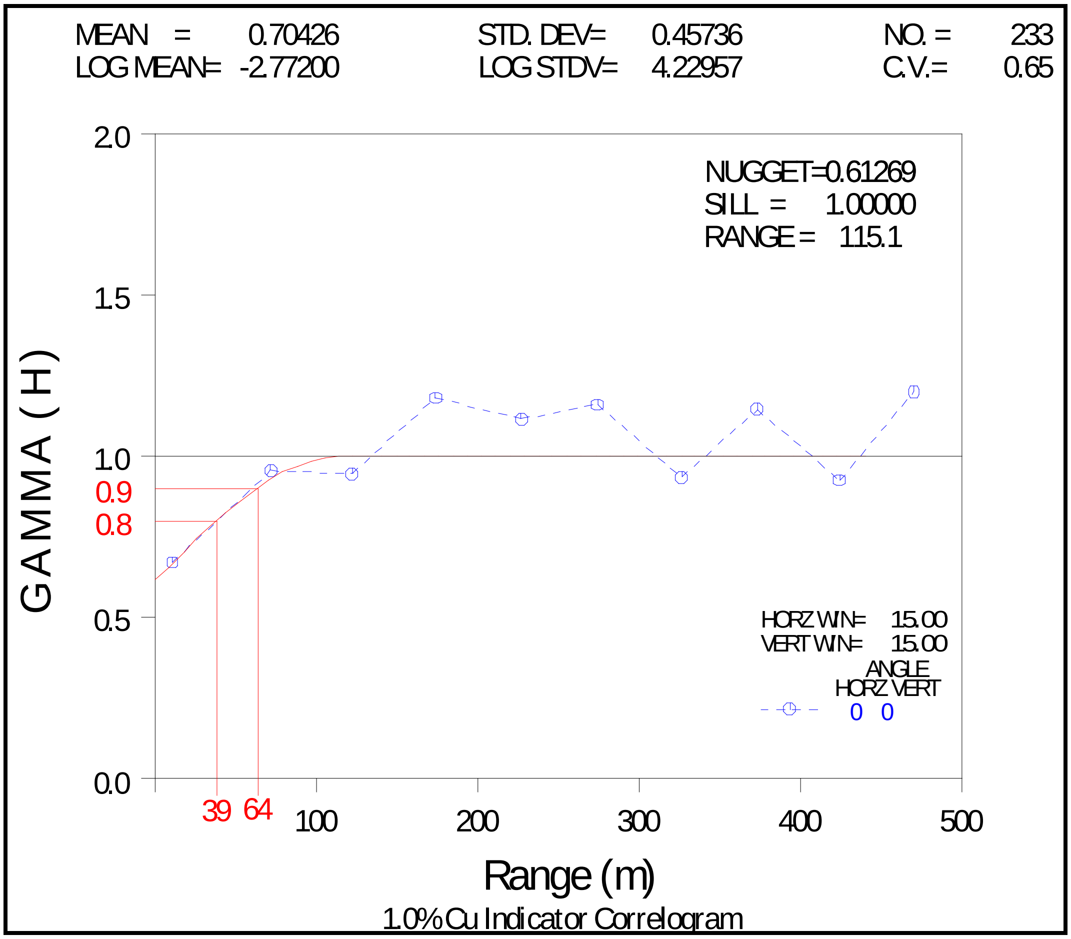
14.19 JOHNNY LEE LZ COMPOSITING

One-metre-long drillhole composites were created starting and ending inside of the Johnny Lee LZ wireframe (LZ zones 11 and 12). There were a total of 193 LZ unit 11 composites with 83% of them exactly 1 m in length; approximately 8% were between 0.50 to 1.0 m in length, and about 8% were greater than 1.0 m in length. If the last sample interval in a bore hole was less than 0.5 m in length, it was added to the previous 1 m composite to ensure that no composite was less than 0.5 m in length. The maximum LZ 11 composite length was 1.44 m in length. Out of a total of 40 LZ unit 12 composites, 68% were exactly 1 m long, 15% were between 0.50 and 1.0 m in length, and 17% were greater than 1.0 m in length. The grade estimates were weighted by composite length.

14.20 JOHNNY LEE LZ VARIOGRAPHY

Copper grade and copper indicator correlograms were generated from 1 m composites located inside of the Johnny Lee LZ wireframe. Figure 14.21 is a 1% copper indicator correlogram that was modelled with a single spherical model resulting in a maximum range of 115 m. Ranges of 39 m and 64 m are shown in red on Figure 14.21 at 80% and 90% of the total variance, respectively.

Figure 14.21 Johnny Lee LZ 1% Copper Indicator Correlogram



14.21 JOHNNY LEE LZ GRADE ESTIMATION

RMI constructed a 3D block model for the Johnny Lee LZ using MineSight® software. Table 14.18 summarizes the limits of the model and size of the blocks.

Table 14.18 Johnny Lee LZ Block Model Limits

Parameter	Minimum	Maximum	Extent (m)	Size (m)	Number
Easting (columns)	506,100	507,450	1,350	5	270
Northing (rows)	5,180,550	5,181,175	625	5	125
Elevation (levels)	1,185	1,500	315	1	315

RMI constructed inverse distance grade models for the LZ using the same relative elevation method as discussed in Section 14.8. Distances from the LZ hanging wall and footwall surfaces were calculated for each block inside of the LZ wireframe. A relative elevation was then calculated for each block and that relative location was used to select eligible composites.

An inverse distance weighing estimator (third power) was used in a two pass estimation strategy. Copper, cobalt, silver, gold, lead, zinc, iron, sulphur, barium, and arsenic were estimated using this method.

Table 14.19 summarizes the ID parameters that were used to estimate base and precious metals.

Table 14.19 Johnny Lee LZ Base Metal Estimation Parameters

Estimation Pass	ID Power	Composite Selection			Ellipse Dimensions (m) [1]			PAR20 [2]
		Min	Max	Max/Hole	Major Axis	Minor Axis	Vertical Axis	
1	3	1	3	1	250	250	250	10
2	3	2	3	1	250	250	250	10

Notes: [1] The vertical axis range is replaced by the RELZ value.
[2] PAR20 refers to a ±RELZ tolerance for composit selection.

Like the Johnny Lee UZ, the apparent spherical ellipse dimension of 250 m is misleading because the relative elevation method and PAR20 parameter limit the actual search to a narrow band that parallels the hanging wall and footwall contacts of the wireframe. The number of composites used to estimate the block grades was based on comparisons made with a nearest neighbour model. The limited number of composites used in the estimate define high-, medium-, and low-grade zones within the wireframe with the goal of minimizing grade smearing.

14.22 JOHNNY LEE LZ GRADE MODEL VERIFICATION

The estimated block grades were verified by visual and statistical methods. The block grades were compared with the drillhole composite grades in section and plan. It is RMI's opinion that the block grades look reasonable when compared with the sample data.

Figure 14.22 is a plan map that shows the outline of the LZ wireframe, two cross section reference lines (G-G' and H-H'). The plan map also shows a dotted red line that sub-divides the LZ into a high-grade core zone and an outer low-grade zone. Figure 14.23 is a section drawn longitudinally through the high-grade core zone of the LZ. Figure 14.24 is a north-south trending cross sections drawn through the LZ. Both cross sections show colour-coded estimated copper block grades and colour-coded 1 m drillhole composite grades.

Figure 14.22 Johnny Lee LZ Plan Map Showing Lines of Section

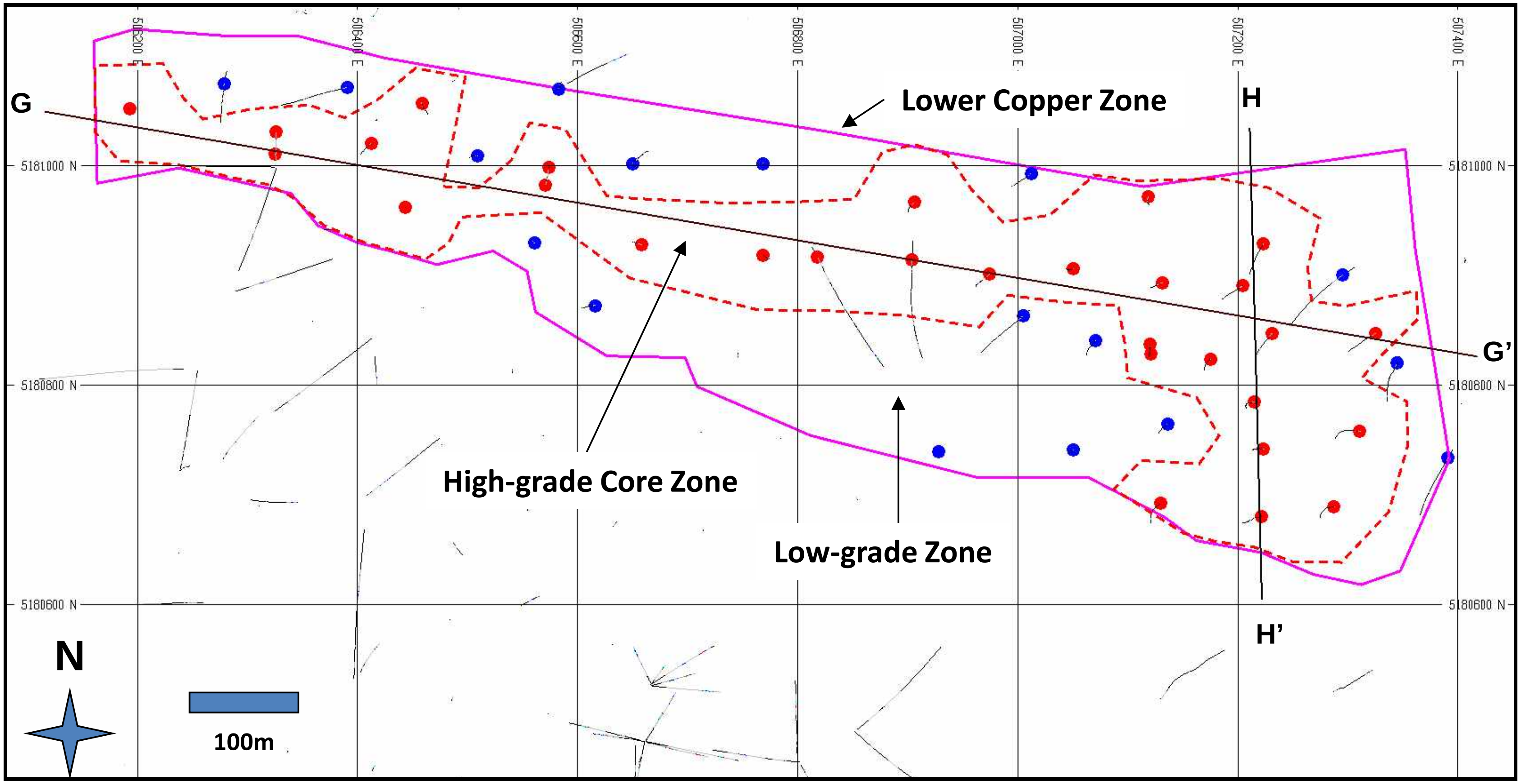


Figure 14.23 Johnny Lee LZ Block Model Cross Section G-G'

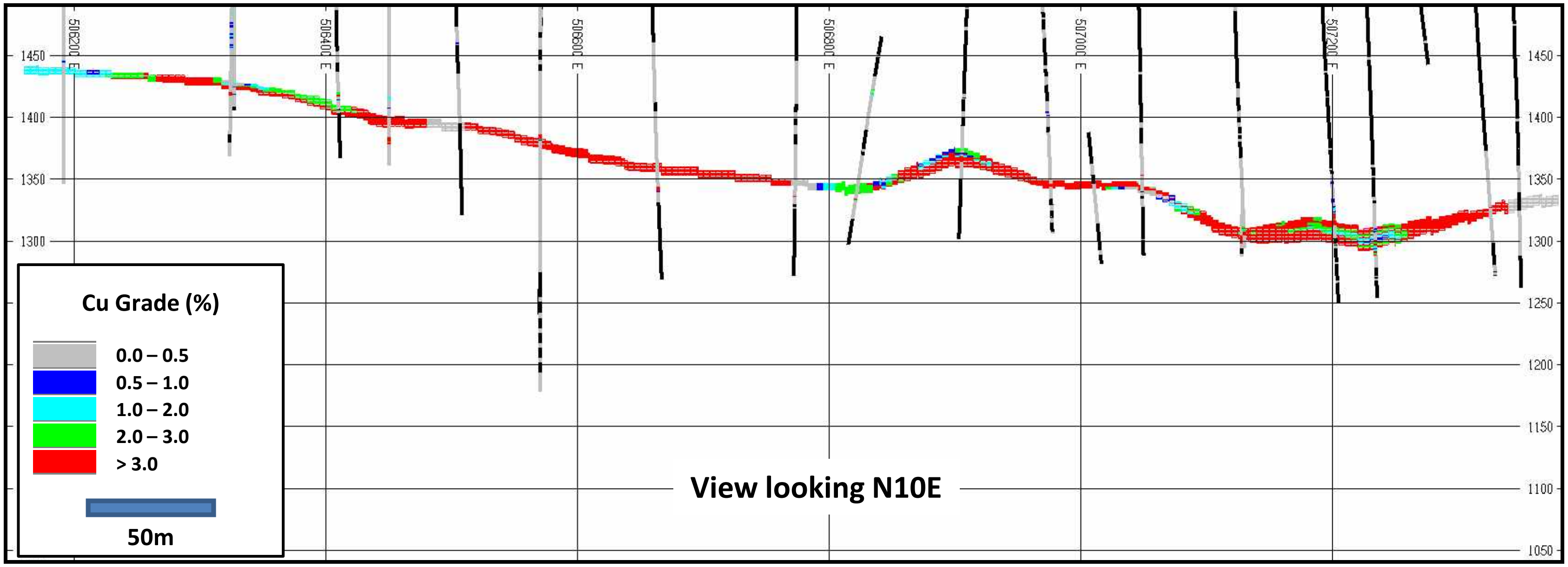
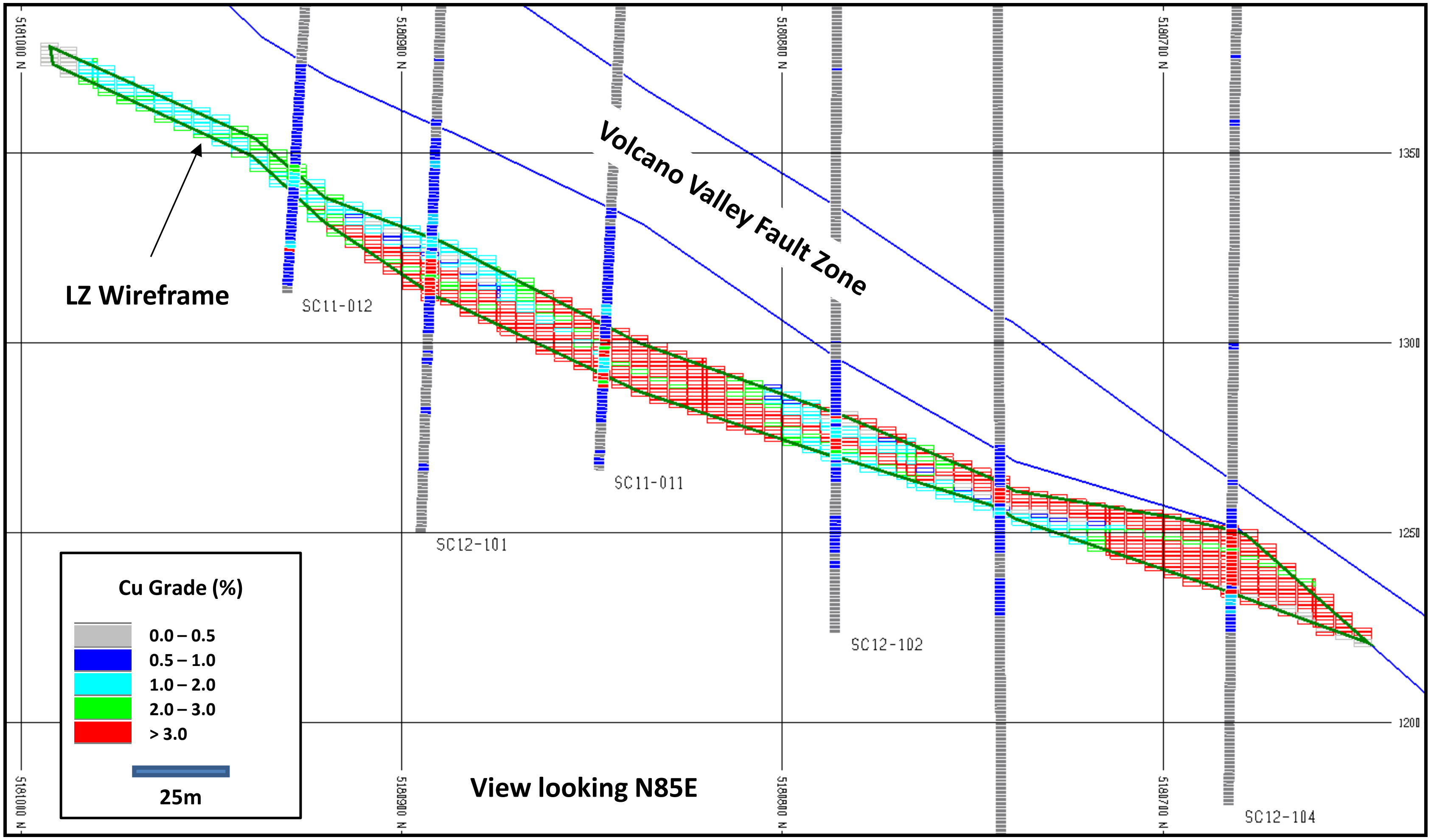


Figure 14.24 Johnny Lee LZ Block Model Cross Section H-H'



NN models were constructed for the primary metals of interest (i.e. copper, cobalt, gold, and silver). To check for possible global biases in the block model, the ID and NN grades were compared for Inferred blocks at a zero cut-off grade. Table 14.20 compares copper and cobalt grades estimated by ID and NN methods for Indicated and Inferred blocks.

Table 14.20 Johnny Lee LZ Global Bias Check

Metal	Resource Grade	Nearest Neighbour Grade	Percent Difference
Indicated Blocks			
Copper (%)	5.7813	5.7497	0.55%
Cobalt (%)	0.0318	0.0302	5.30%
Silver (g/t)	4.4211	4.2960	2.91%
Gold (g/t)	0.3033	0.3036	-0.10%
Inferred Blocks			
Copper (%)	0.7500	0.7104	5.57%
Cobalt (%)	0.0313	0.0302	3.64%
Gold (g/t)	0.4346	0.4191	3.70%
Silver (g/t)	5.5465	4.9012	13.17%

The data in Table 14.20 show a close comparison between the ID and NN grades for Indicated blocks, while the comparison for Inferred blocks is not as close. The Inferred resource blocks show more variance when compared to the Indicated blocks, reflecting the uncertainty in those estimates. Based on industry-accepted practices, models that show less than a 5% variance from a NN grade model are thought to be unbiased.

RMI also checked for local biases by creating a series of slices or "swaths" through the model columns (eastings), rows (northings), and levels (elevations) comparing the ID and NN grades. Figure 14.25 through Figure 14.27 show the local variation between the ID and NN copper models at a zero cut-off grade for Indicated blocks. The ID grade (CUIDW) is shown in red, the NN grade (CUNN) is shown in blue and the number of blocks per "swath" are shown by the black line which is read from the right side Y-axis.

Figure 14.25 Johnny Lee LZ Copper Swath Plot – Easting

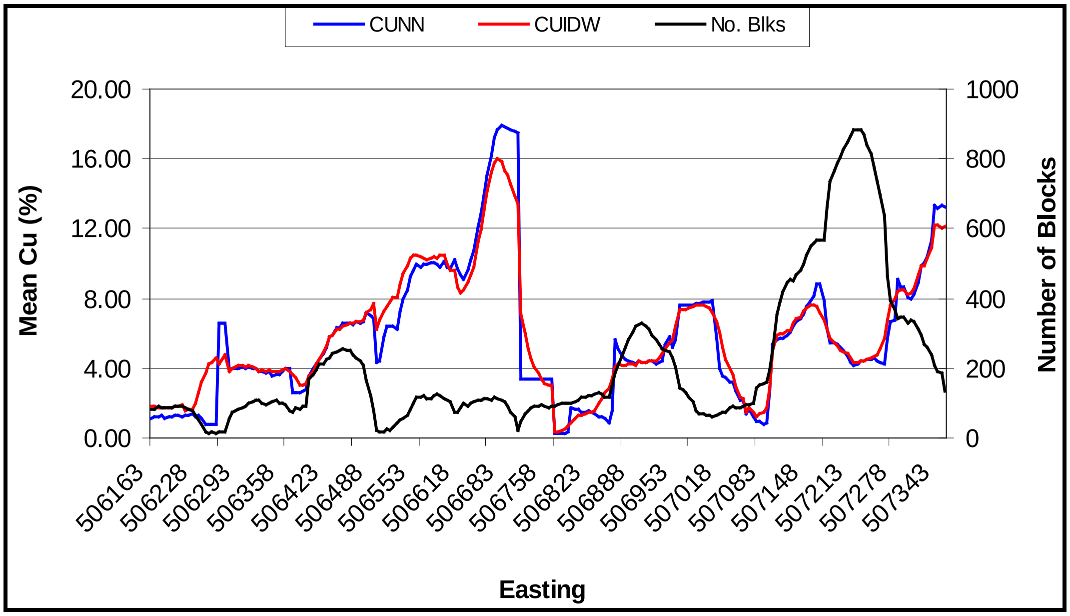


Figure 14.26 Johnny Lee LZ Copper Swath Plot – Northing

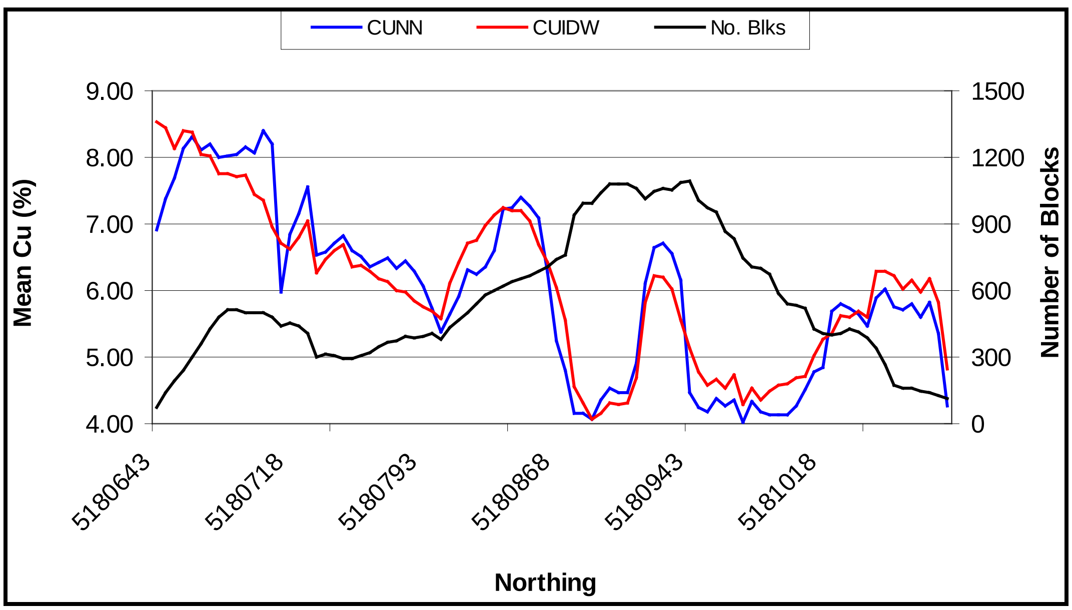
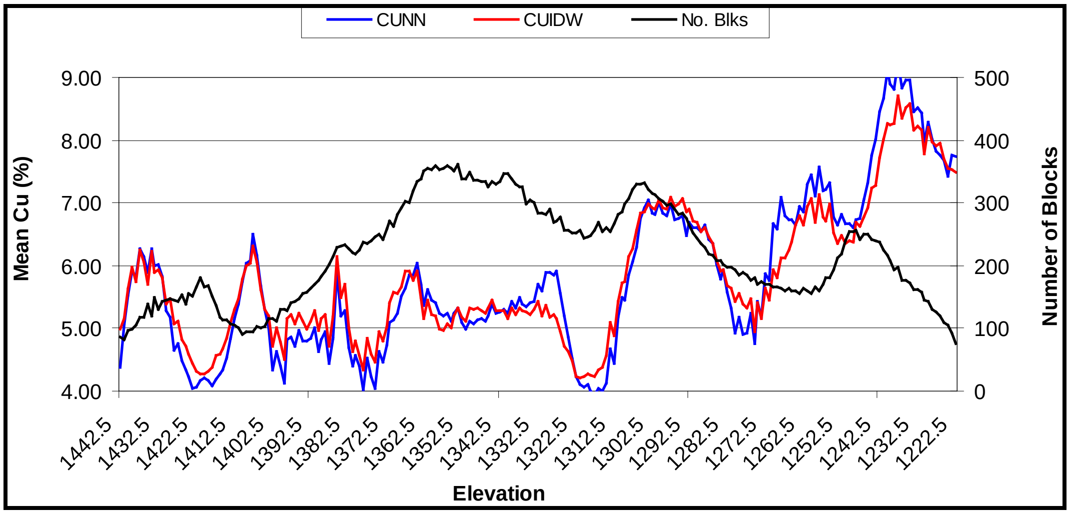


Figure 14.27 Johnny Lee LZ Copper Swath Plot – Elevation



The swath plots shown in Figure 14.25 through Figure 14.27 show some local variation with the NN model grades showing more variation while the ID grades show some smoothing.

Based on a visual examination and comparisons with a NN model, it is RMI's opinion that the Johnny Lee LZ model is globally unbiased and represents a reasonable estimate of undiluted in-situ resources.

14.23 Johnny Lee LZ Resource Classification

Johnny Lee LZ blocks were classified into Indicated and Inferred categories. RMI constructed a 3D wireframe based on mineralized continuity. Blocks located inside of this wireframe were flagged as Indicated Resources provided the blocks were within 50 m of a drill hole. Blocks inside of the Indicated wireframe that were located beyond 50 m of a drill hole were re-coded as Inferred blocks. All other blocks inside of the LZ copper wireframe were coded as Inferred.

14.24 Johnny Lee LZ Density Data

Tintina personnel obtained bulk density determinations from 256 pieces of drill core taken from their 2010 to 2012 drilling programs. The core was not rigorously dried but was not thought to contain much moisture. The core was weighed in-air and then weighed while submerged in water. A relative bulk density calculation was then made (bulk density = weight in-air / (weight in-air - weight in water)). Table 14.21 summarizes bulk density statistics from the drill core samples that were analyzed by Tintina's personnel. The data in Table 14.21 are subdivided into samples located inside of the LZ wireframe (primarily massive and semi-massive sulphide material), and samples collected from outside of the LZ wireframe (a variety of lithologies with varying amounts of pyrite).

Table 14.21 Johnny Lee LZ Bulk Density Determinations

Sample Location	Count	Mean	Minimum	Maximum	Standard Deviation
Inside Wireframe	53	3.49	2.61	4.22	0.40
Outside Wireframe	103	3.07	2.36	4.21	0.50

Tintina sent 58 core sample pieces from their 2012 LZ drilling program to ALS Chemex in Reno, Nevada, for bulk density determination after they had made their own bulk density measurements. The average bulk density as determined by Chemex was about 4% higher than Tintina's measurements (3.23 versus 3.10). Of the 58 samples sent to Chemex, 27 were obtained from massive sulphide material from inside of the LZ wireframe. For this sample set, the Chemex results were about 5% higher than Tintina's results (3.64 versus 3.47).

While the Tintina results may be somewhat low compared to the Chemex results, RMI chose to use an average dry bulk density value of 3.49 for tabulating resources for the Johnny Lee LZ. A density value of 3.07 was assigned to blocks located outside of the LZ wireframe.

14.25 Johnny Lee LZ Resource Summary

A cut-off grade of 1.5% copper was used to define an undiluted Indicated Mineral Resource for the LZ of 2,387,000 t with an average grade of 6.40% copper, 0.03% cobalt, 0.30 g/t gold, and 4.5 g/t silver. Using the same cut-off grade, there is an Inferred Resource of 205,000 t with an average grade of 5.33% copper, 0.3% cobalt, 0.21 g/t gold, and 4.1 g/t silver. The cut-off grade was established by using:

- a copper price of US$2.75/lb
- a copper recovery of 84%
- mining costs of US$50/t
- processing costs of US$16.00/t
- G&A costs of US$5.00/t
- refining costs of US$5.53/t.

The cut-off grade for the Johnny Lee LZ differs from that of the Johnny Lee UZ due to slightly differing recoveries and mining cost estimates. The same cut-off grades have been used for each deposit for the last several resource estimates for comparison purposes.

No credit was given to cobalt, gold, or silver in determining the cut-off grade since little metallurgical work has been completed at this stage of the Project. Table 14.22 and Table 14.23 summarize the Indicated and Inferred Resources at several cut-off grades, respectively. Johnny Lee LZ resources of record are highlighted in gray in Table 14.22 and Table 14.23.

Table 14.22 Johnny Lee LZ Undiluted Indicated Mineral Resource

Cu Cut-off (%)	Tonnes (000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (M lb)	Co (M lb)	Au ('000 oz)	Ag ('000 oz)
1.0	2,518	6.13	0.032	0.300	4.5	340	1.8	24	364
1.1	2,504	6.16	0.033	0.300	4.5	340	1.8	24	362
1.2	2,491	6.19	0.033	0.300	4.5	340	1.8	24	360
1.3	2,456	6.26	0.033	0.301	4.5	339	1.8	24	355
1.4	2,423	6.32	0.033	0.302	4.5	338	1.8	24	351
1.5	2,387	6.40	0.033	0.304	4.5	337	1.7	23	345
1.6	2,344	6.49	0.033	0.306	4.5	335	1.7	23	339
1.7	2,304	6.57	0.033	0.308	4.6	334	1.7	23	341
1.8	2,264	6.65	0.033	0.311	4.6	332	1.6	23	335
1.9	2,233	6.72	0.033	0.312	4.6	331	1.6	22	330
2.0	2,206	6.78	0.033	0.314	4.6	330	1.6	22	326
2.1	2,179	6.84	0.033	0.315	4.6	328	1.6	22	322
2.2	2,155	6.89	0.034	0.316	4.7	327	1.6	22	326
2.3	2,128	6.95	0.034	0.317	4.7	326	1.6	22	322
2.4	2,102	7.01	0.034	0.319	4.7	325	1.6	22	318
2.5	2,073	7.07	0.034	0.321	4.7	323	1.6	21	313

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.23 Johnny Lee LZ Undiluted Inferred Mineral Resource

Cu Cut-off (%)	Tonnes (000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (M lb)	Co (M lb)	Au ('000 oz)	Ag ('000 oz)
1.0	222	5.02	0.024	0.199	4.1	25	0.1	1.4	29
1.1	220	5.06	0.024	0.200	4.1	25	0.1	1.4	29
1.2	219	5.07	0.024	0.201	4.1	24	0.1	1.4	29
1.3	217	5.11	0.024	0.202	4.1	24	0.1	1.4	29
1.4	212	5.20	0.024	0.204	4.1	24	0.1	1.4	28
1.5	205	5.33	0.025	0.207	4.1	24	0.1	1.4	27
1.6	203	5.37	0.025	0.208	4.1	24	0.1	1.4	27
1.7	199	5.45	0.025	0.210	4.1	24	0.1	1.3	26
1.8	193	5.55	0.025	0.212	4.1	24	0.1	1.3	25
1.9	191	5.60	0.025	0.214	4.1	24	0.1	1.3	25
2.0	185	5.72	0.025	0.217	4.1	23	0.1	1.3	24
2.1	178	5.86	0.025	0.222	4.2	23	0.1	1.3	24
2.2	174	5.95	0.025	0.225	4.2	23	0.1	1.3	23
2.3	169	6.06	0.026	0.229	4.2	23	0.1	1.2	23
2.4	165	6.13	0.026	0.232	4.3	22	0.1	1.2	23
2.5	162	6.22	0.026	0.235	4.3	22	0.1	1.2	22

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

14.26 GENERAL DISCUSSION – JOHNNY LEE LZ RESOURCE

RMI is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other factors that could materially affect the Johnny Lee LZ resources discussed in this report.

14.27 LOWRY DRILLING DATA

As described in Section 10.0, Tintina has collected approximately 85% of the drilling data used by RMI to estimate the Mineral Resources for the Lowry MZ. RMI has also used historic drilling data collected by CAI. The aerial distribution of the holes is shown in Table 10.3 and can be reviewed in Figure 10.2.

14.28 LOWRY EXPLORATORY DATA ANALYSIS

The Lowry MZ was modelled as two separate wireframes consisting of a large single lens of massive sulphide mineralization and a small isolated lens. The two wireframes were created using logged geologic information and drillhole copper grades. A code of “21” was assigned to intervals located inside of the MZ wireframes. All other drill hole intervals in the vicinity of the MZ wireframe were assigned a code of “99”.Basic assay statistics were tabulated at four different cut-off grades for uncapped copper, cobalt, silver, and gold, and are presented in Table 14.24 to Table 14.27. The data summarized in Table 14.24 to Table 14.27 include the number of metres at each cut-off grade, mean grades, standard deviations, and CV. Incremental data (i.e. statistics for material between cut-off grades) are also tabulated; for example, Table 14.24 shows that 52% of the MZ intersections are above a 1% copper cut-off grade and 48% are less than that cut-off.

Table 14.24 Lowry MZ Copper Assay Statistics

	Uncapped Cu Statistics Above Cut-off							
MZ	Cu Cut-off (%)	Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Standard Deviation	CV
All Data	0.00	11,180	94	0.26	2,912	32.5	0.88	3.37
	1.00	712	3	2.76	1,964	17.3	2.23	0.81
	2.00	355	1	4.12	1,460	14.0	2.49	0.61
	3.00	190	2	5.53	1,052	36.1	2.69	0.49
21	0.00	864	48	1.78	1,539	11.3	2.22	1.25
	1.00	451	23	3.03	1,365	18.7	2.47	0.82
	2.00	248	13	4.35	1,077	17.6	2.68	0.62
	3.00	139	16	5.80	806	52.4	2.82	0.49
99	0.00	10,316	97	0.13	1,373	56.4	0.46	3.48
	1.00	261	1	2.30	599	15.7	1.63	0.71
	2.00	107	1	3.59	383	10.0	1.89	0.53
	3.00	51	0	4.80	246	18.0	2.12	0.44

Table 14.25 Lowry MZ Cobalt Assay Statistics

MZ	Uncapped Co Statistics Above Cut-off							
	Co Cut-off (ppm)	Total Metres	Inc. Percent	Mean Co (ppm)	Grd-Thk (%-ppm)	Inc. Percent	Standard Deviation	CV
All Data	0	11,180	68	161	1,801,037	10.5	366	2.27
	100	3,606	24	447	1,611,684	33.4	543	1.21
	500	963	6	1050	1,010,992	23.9	758	0.72
	1,000	331	3	1756	581,368	32.3	935	0.53
21	0	864	15	785	678,255	1.0	862	1.10
	100	731	32	919	671,622	12.0	873	0.95
	500	456	27	1293	590,077	24.2	916	0.71
	1,000	227	26	1874	425,681	62.8	996	0.53
99	0	10,316	72	109	1,122,782	16.3	219	2.01
	100	2,875	23	327	940,062	46.2	324	0.99
	500	507	4	831	420,915	23.6	485	0.58
	1,000	104	1	1500	155,687	13.9	721	0.48

Table 14.26 Lowry MZ Silver Assay Statistics

MZ	Uncapped Ag Statistics Above Cut-off							
	Ag Cut-off (g/t)	Total Metres	Inc. Percent	Mean Ag (g/t)	Grd-Thk (%-m)	Inc. Percent	Standard Deviation	CV
All Data	0	11,180	55	7.2	80,654	9.4	9.3	1.28
	5	5,005	28	14.6	73,050	35.4	9.6	0.66
	15	1,866	13	23.9	44,515	37.7	9.8	0.41
	30	357	3	39.5	14,106	17.5	11.5	0.29
21	0	864	17	12.5	10,768	3.0	8.7	0.70
	5	715	50	14.6	10,441	37.5	8.1	0.55
	15	282	28	22.7	6,398	46.0	6.8	0.30
	30	40	5	35.9	1,440	13.4	5.6	0.16
99	0	10,316	58	6.8	69,885	10.4	9.2	1.35
	5	4,290	26	14.6	62,610	35.0	9.8	0.67
	15	1,584	12	24.1	38,117	36.4	10.2	0.42
	30	317	3	40.0	12,666	18.1	11.9	0.30

Table 14.27 Lowry MZ Gold Assay Statistics

MZ	Uncapped Au Statistics Above Cut-off							
	Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (%-m)	Inc. Percent	Standard Deviation	CV
All Data	0.00	11,180	100	0.01	65	76.6	0.06	10.03
	0.05	52	0	0.29	15	2.9	0.80	2.72
	0.10	22	0	0.60	13	1.6	1.15	1.92
	0.20	15	0	0.83	12	18.9	1.35	1.63
21	0.00	864	99	0.01	9	47.8	0.12	10.92
	0.05	8	0	0.58	5	1.4	1.04	1.80
	0.10	6	0	0.77	5	3.5	1.16	1.51
	0.20	4	0	1.25	4	47.3	1.35	1.08
99	0.00	10,316	100	0.01	55	81.4	0.05	9.31
	0.05	43	0	0.24	10	3.2	0.73	3.06
	0.10	16	0	0.53	9	1.2	1.14	2.14
	0.20	11	0	0.70	8	14.2	1.33	1.90

The CV for copper, cobalt, and silver based on raw assays for the Lowry MZ are seen to be relatively low. A single 4 g/t gold assay highly skewed the CV for that metal. Grade capping (Section 14.29) slightly reduced the CV for the key metals (copper and cobalt).

14.29 LOWRY HIGH-GRADE OUTLIERS

RMI generated a series of cumulative probability plots after transforming the original copper, cobalt, silver, and gold assays using the cumulative normal distribution method. Figure 14.28 to Figure 14.31 show copper, cobalt, silver, and gold probability plots for the Lowry MZ (code 21 intervals). The black circle in Figure 14.28 to Figure 14.31 indicates the capping limits selected by RMI to minimize the potential for over estimating contained metal.

Table 14.28 summarizes high-grade outlier capping limits that were selected for copper, cobalt, silver, and gold for the Lowry MZ based on a review of probability plots. These limits were applied to the raw assays prior to creating drillhole composites.

Figure 14.28 Lowry Zone Cumulative Probability Plot – Copper

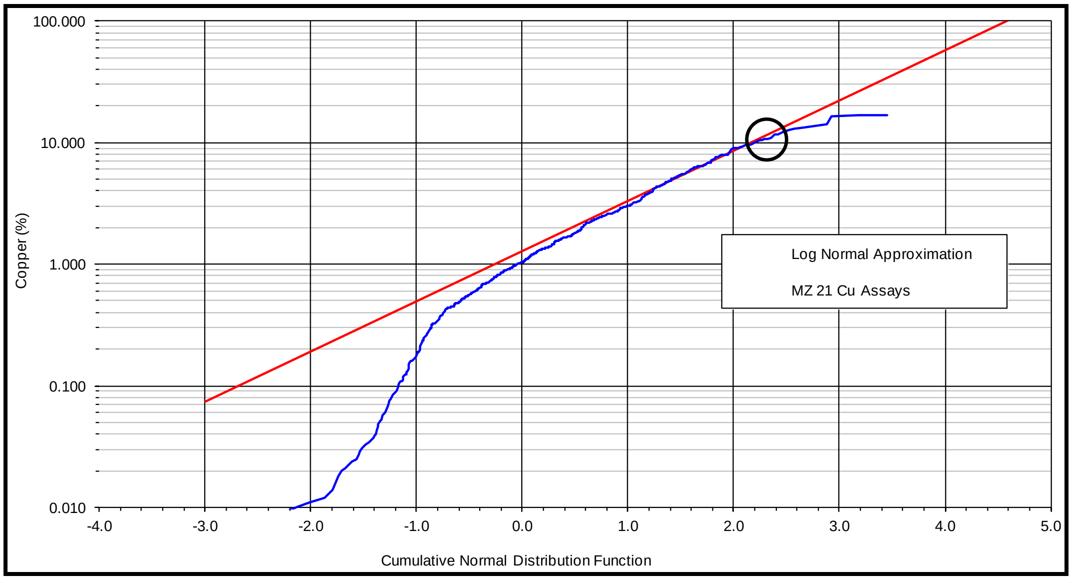


Figure 14.29 Lowry Zone Cumulative Probability Plot – Cobalt

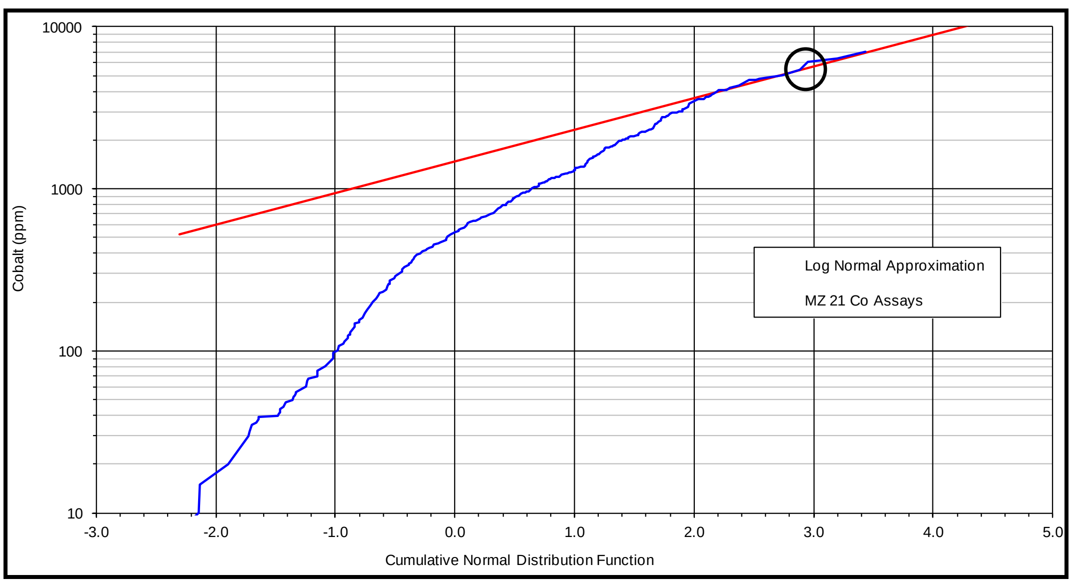


Figure 14.30 Lowry Zone Cumulative Probability Plot – Silver

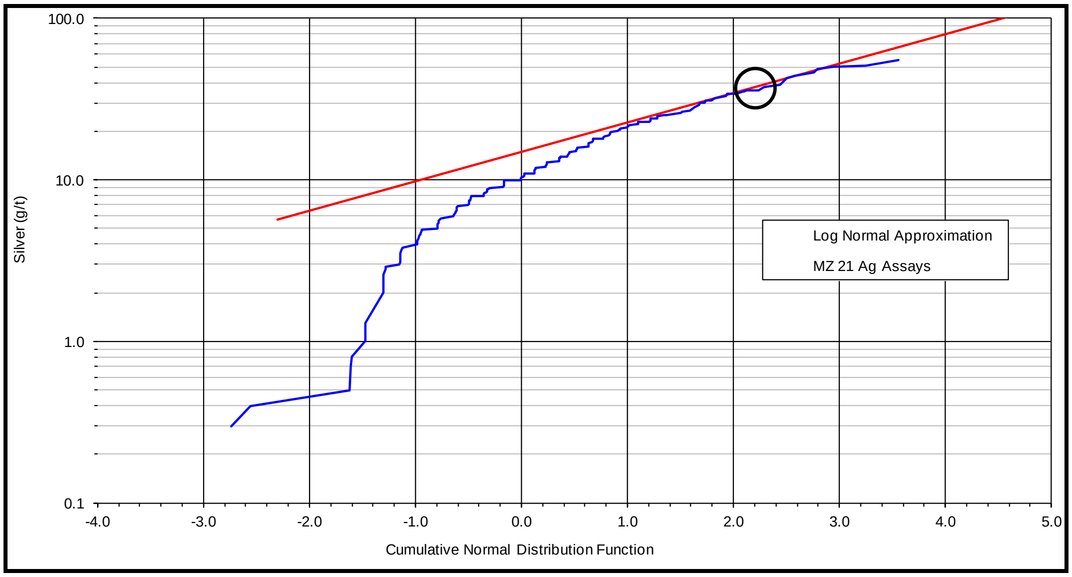


Figure 14.31 Lowry Zone Cumulative Probability Plot – Gold

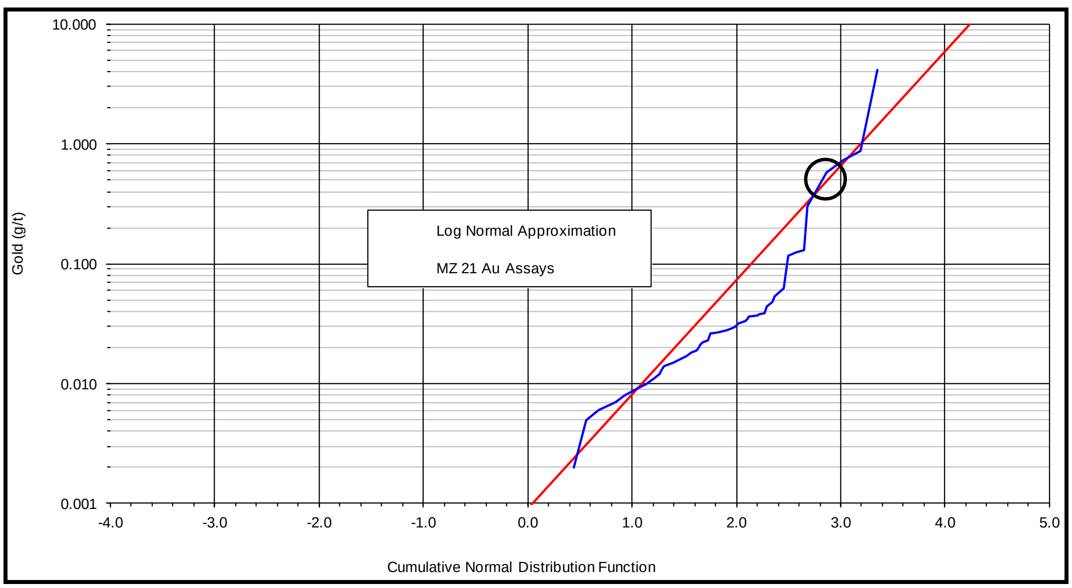


Table 14.28 Lowry Grade Capping Limits

Metal	Cap Limit	No. Capped
Copper	12.5%	8
Cobalt	5,000 ppm	4
Silver	30 g/t	21
Gold	0.50 g/t	4

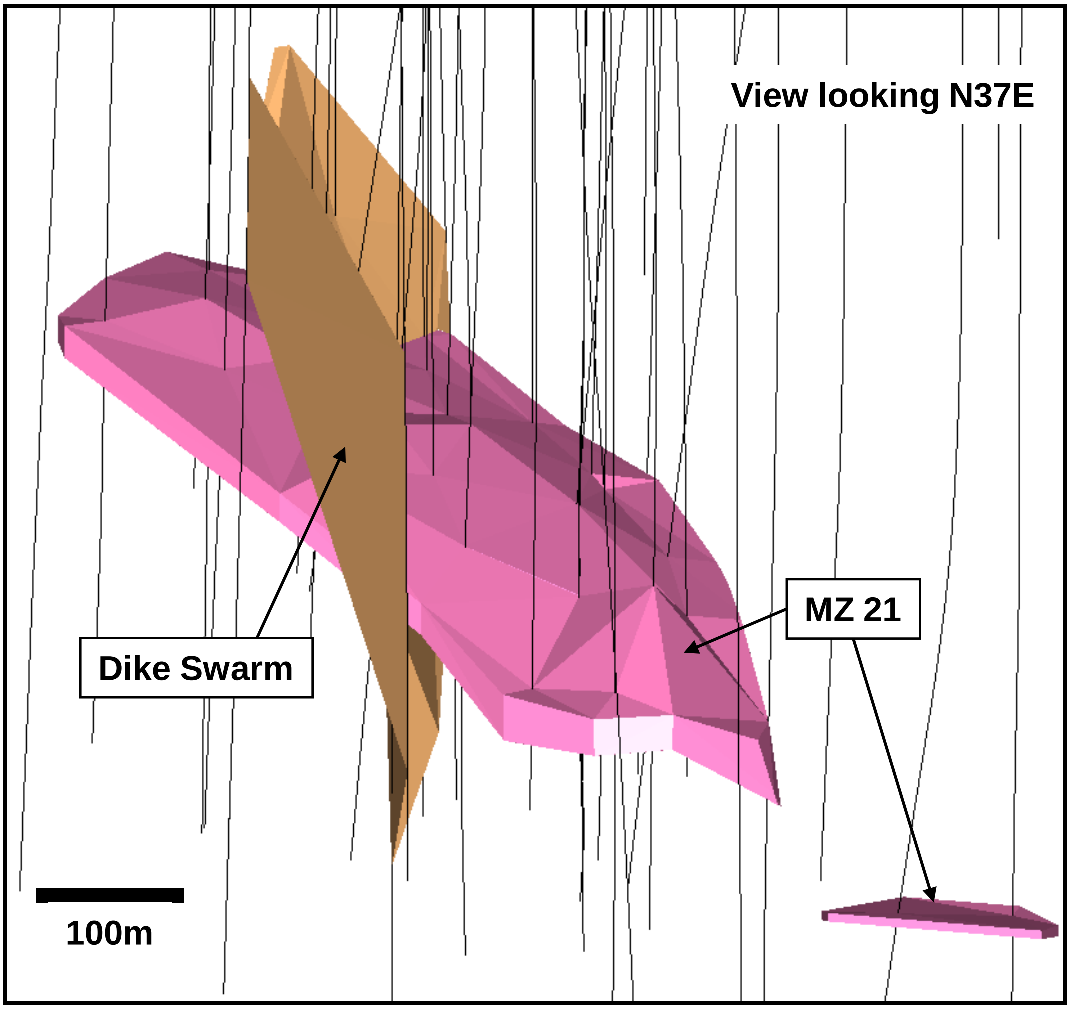
14.30 LOWRY DOMAINS

Mr. Vincent Scartozzi, Senior Geologist with Tintina, constructed 3D wireframe to represent the Lowry MZ stratabound copper sulphide horizon. RMI reviewed the wireframes and believe that they fairly represent the mineralized zone based on the current drillhole spacing. The mineralized horizon was modelled as two separate lenses that are offset by a steep northeast trending fault zone referred to as the "hanging wall fault".

XYZ hanging wall and footwall drillhole pierce points through the mineralized horizon were used to create the initial wireframe solid. Criteria such as minimum thickness (approximately 2 m) and copper grade (roughly a 1% cut-off grade) were used in conjunction with logged lithologic/mineralization observations to construct the wireframe. The wireframe was extended approximately 30 to 40 m outward from the perimeter drillholes that intersected the horizon. Block grades were only estimated for blocks within the Lowry wireframe. The percentage of each model block inside of the Lowry wireframe was stored in the block model for more accurate tonnage calculations.

Figure 14.32 is a perspective view looking to the northeast showing a pink-coloured wireframe that represents the Lowry mineralized copper zone. This wireframe was used to constrain the estimate of block grades. A northeast-striking dike swarm is also shown in tan.

Figure 14.32 Lowry MZ Wireframe Perspective



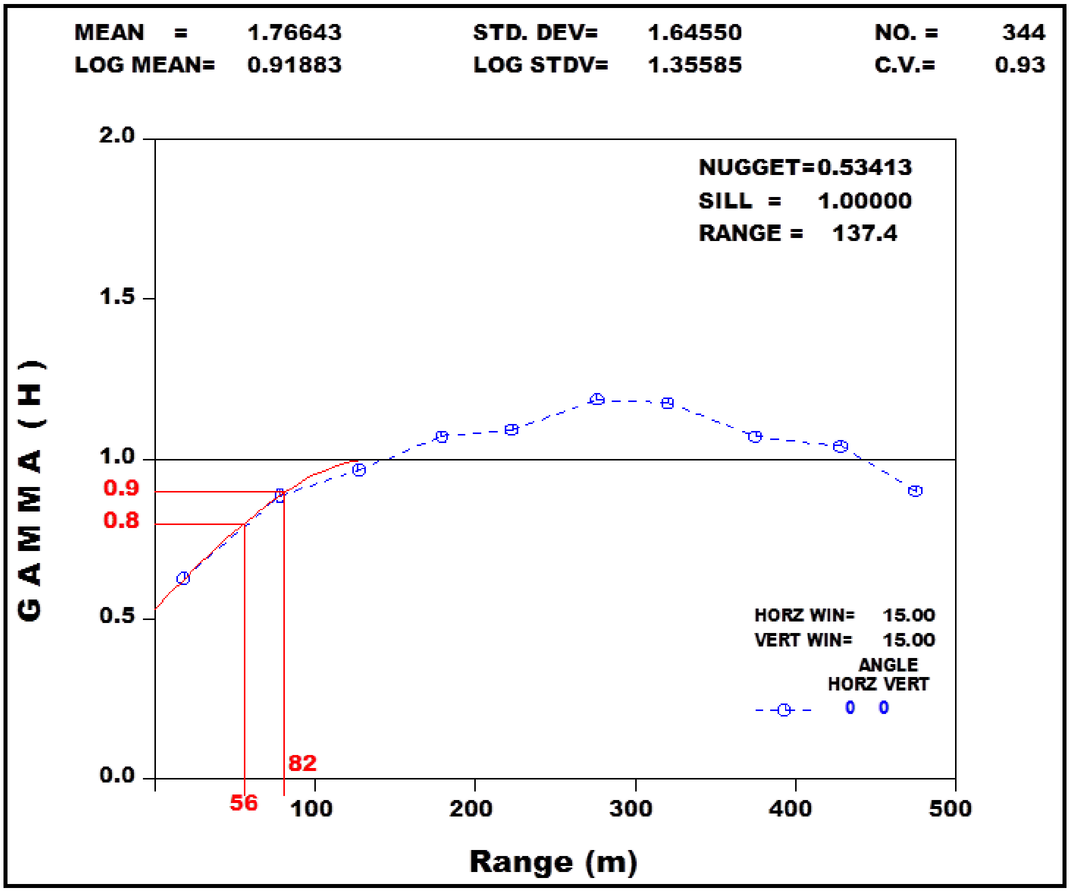
14.31 LOWRY ASSAY COMPOSITING

Two and a half (2.5) metre-long drillhole composites were created starting and ending inside of the Lowry MZ wireframe (MZ zone 21). About 85% of the original assay intervals inside of the MZ wireframe were less than 2 m in length. There were a total of 344 MZ unit 21 composites, with 92% of them exactly 2.5 m in length; approximately 2% were between less than 2.5 m and approximately 7% greater than 2.5 m in length. If the last sample interval in a bore hole was less than 1.25 m in length, it was added to the previous 2.5 m composite to ensure that no composite was less than 1.25 m in length. The maximum MZ 21 composite length was 3.70 m in length. The grade estimates were weighted by composite length.

14.32 LOWRY VARIOGRAPHY

Copper grade and copper indicator correlograms were generated from the Lowry MZ composites. Figure 14.33 shows a representative copper correlogram that was generated from the 2.5 m MZ 21 composites.

Figure 14.33 Lowry MZ Copper Grade Correlogram



14.33 LOWRY GRADE ESTIMATION

RMI constructed a 3D block model for the Lowry MZ using MineSight® software. Table 14.29 summarizes the limits of the model and size of the blocks.

Table 14.29 Lowry MZ Block Model Limits

Parameter	Minimum	Maximum	Extent (m)	Size (m)	Number
Easting (columns)	508,900	509,350	450	5.0	90
Northing (rows)	5,179,300	5,180,000	700	5.0	140
Elevation (levels)	1,000	1,500	500	2.5	200

A significant amount of the copper mineralization within the Lowry Zone occurs as distinct stratabound layers within thick bedded sulphide accumulations. A portion of the Lowry copper mineralization appears to be related to remobilization and replacement of pre-existing minerals. RMI elected to use the same method of selecting samples that was used for the Johnny Lee UZ and LZ (Section 14.8). This relative elevation method results in a distribution of block grades that closely follow bedding.

A two-pass ID estimation method was selected by RMI for estimating base and precious metals for the Lowry Zone. An ID power of three was selected. The number of composites and drillholes used to estimate each block were captured along with the distance to the closest composite. These data were used to classify the blocks into Inferred Resources.

Table 14.30 summarizes the ID parameters that were used to estimate base and precious metals.

Table 14.30 Lowry MZ Grade Estimation Parameters

Estimation Pass	ID Power	Composite Selection			Ellipse Dimensions (m)[1]			PAR20 [2]
		Minimum	Maximum	Max/Hole	Major Axis	Minor Axis	Vertical Axis	
1	3	1	3	1	200	200	200	10
2	3	2	3	1	200	200	200	10

Note: [1] The vertical axis range is replaced by the RELZ value.
[2] PAR20 refers to a ±RELZ tolerance for composite selection.

Like the Johnny Lee UZ and LZ models, the percentage of MZ wireframe occupying each 5 m by 5 m by 2.5 m block was stored. Grades for the mineralized portion of each block inside of the MZ wireframe were estimated using the relative elevation method described previously using only composites from within the MZ wireframe (i.e. MZ code 21 composites). Grades were also estimated for unmineralized blocks including the fractional portion of blocks that were not 100% inside of the MZ wireframe. The same relative elevation method was used to estimate grades for the unmineralized blocks using only composites located outside of the MZ wireframe.

14.34 Lowry Grade Model Verification

The estimated Lowry MZ block grades were verified by visual and statistical methods. The block grades were compared with the drillhole composite grades in section and plan. It is RMI's opinion that the block grades look reasonable when compared with the composited sample data. Figure 14.34 is a plan map showing the outline of the MZ copper sulphide zone, drillholes, and two cross section reference lines (I-I' and J-J'). Figure 14.35 and Figure 14.36 are vertical cross sections through the block model showing composite and block copper grades. A northeast trending zone of narrow discontinous, primarily alkalic dikes cuts across the MZ and is shown in Figure 14.35 and Figure 14.36. These dikes are thought to be associated with local/regional Ecocene intrusive activity. It was not possible to model individual dikes with the current drill hole spacing so a band or zone of possible dike bodies was modelled. RMI recommends that additional drilling efforts be undertaken to determine how these dikes may affect insitu resources.

Figure 14.34 Lowry MZ Plan Map Showing Section Reference Lines

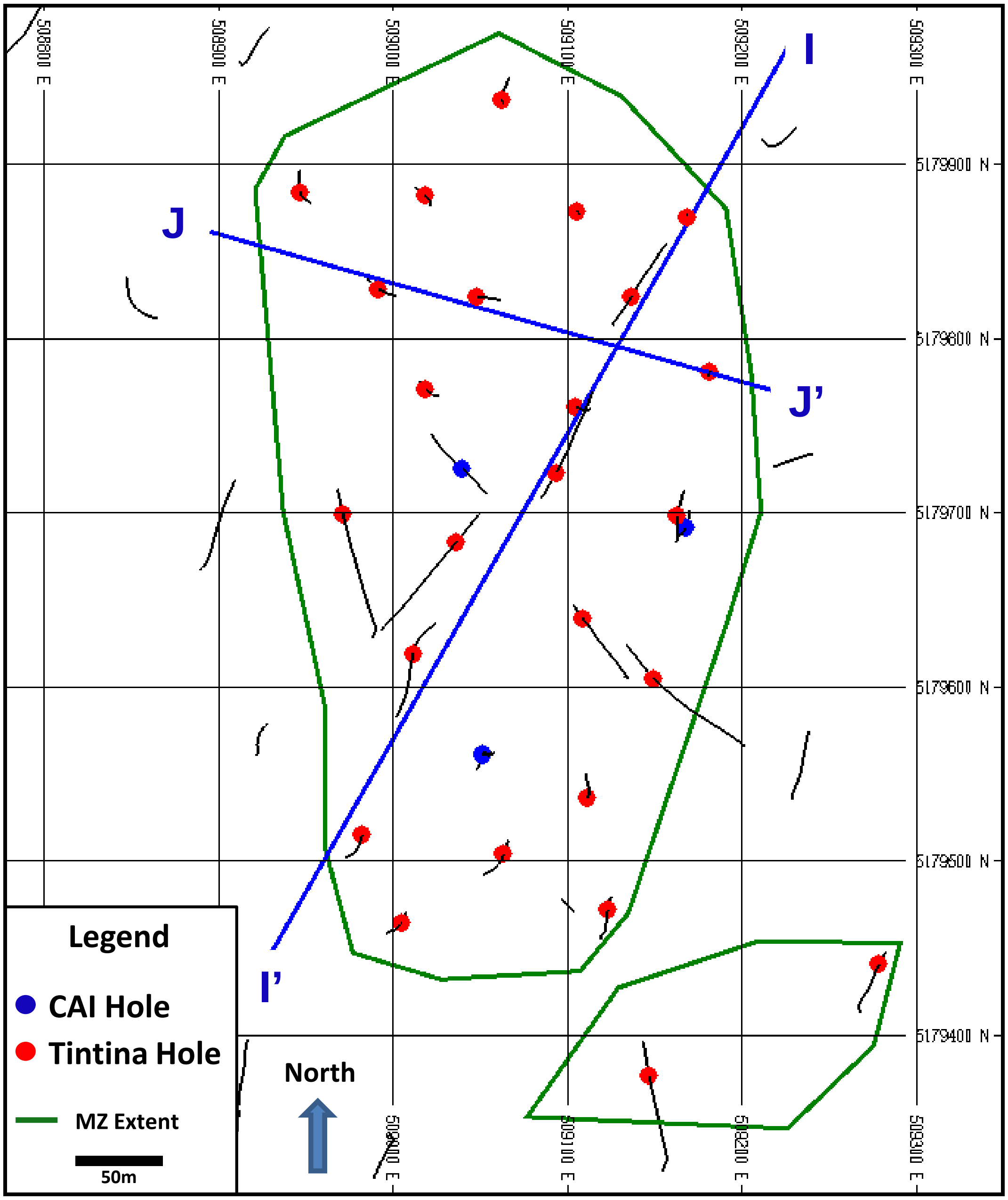


Figure 14.35 Lowry Cross Section I-I'

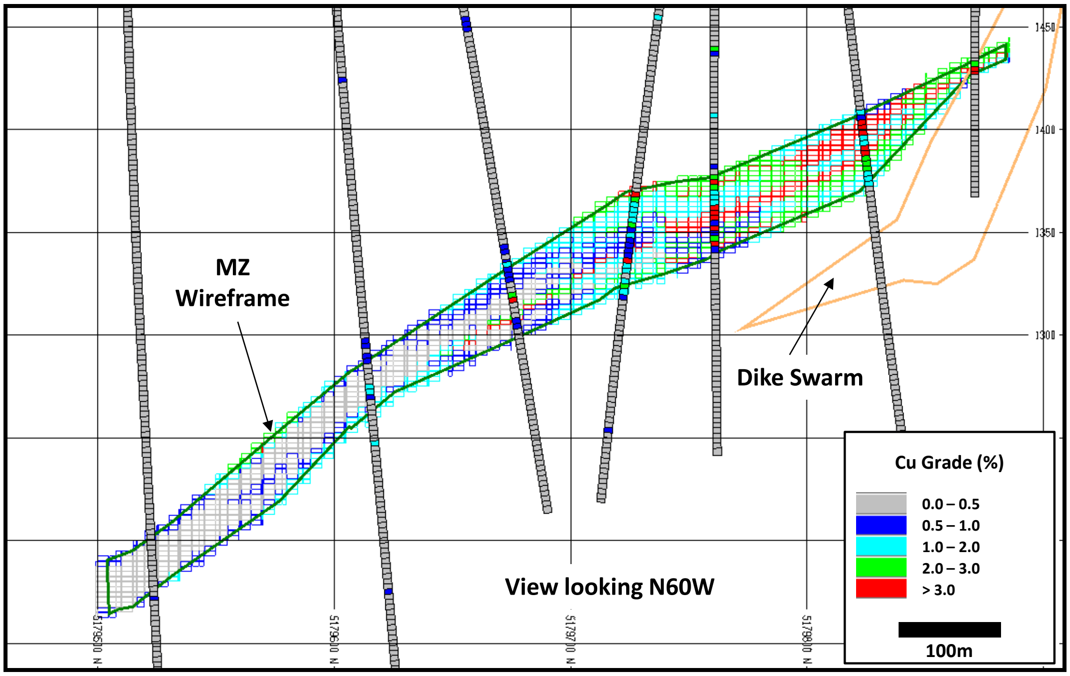
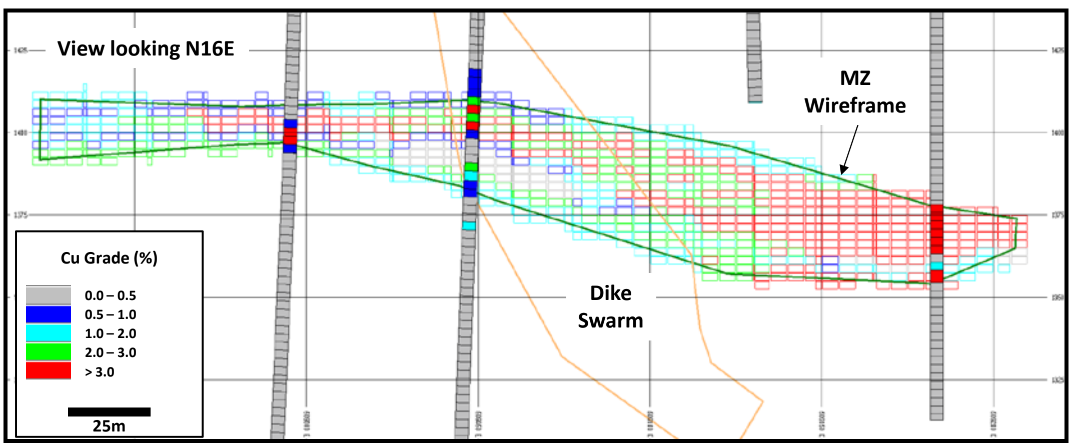


Figure 14.36 Lowry Cross Section J-J'



NN models were constructed for the primary metals of interest (i.e. copper, cobalt, gold, and silver). To check for possible global biases in the block model, the ID and NN grades were compared for estimated blocks at a zero cut-off grade. Table 14.31 compares copper, cobalt, silver, and gold grades estimated by ID and NN methods.

Table 14.31 Lowry MZ Global Bias Check

Metal	ID Resource Grade	NN Grade	Percent Difference
Copper (%)	1.7337	1.7307	0.17%
Cobalt (%)	0.0768	0.0770	-0.26%
Silver	0.0062	0.0058	6.90%
Gold	12.15	12.16	-0.07%

The data in Table 14.31 show a close comparison between the ID and NN grades for copper, cobalt, and silver showing that the model is globally unbiased. Based on industry accepted practice models that show less than a 5% variance from a NN grade model are thought to be unbiased. From a percentage basis gold grades appear to be biased but the grades are very low and not thought to be material.

RMI also checked for local biases by creating a series of slices or "swaths" through the model columns (eastings), rows (northings), and levels (elevations) comparing the ID and NN grades. Figure 14.37 through Figure 14.39 show the local variation between the ID and NN copper models at a zero cut-off grade. The ID grade (CUIDW) is shown in red, the NN grade (CUNN) is shown in blue and the number of blocks per "swath" are shown by the black line which is read from the right side Y-axis.

Figure 14.37 Lowry MZ Copper Swath Plot – Easting

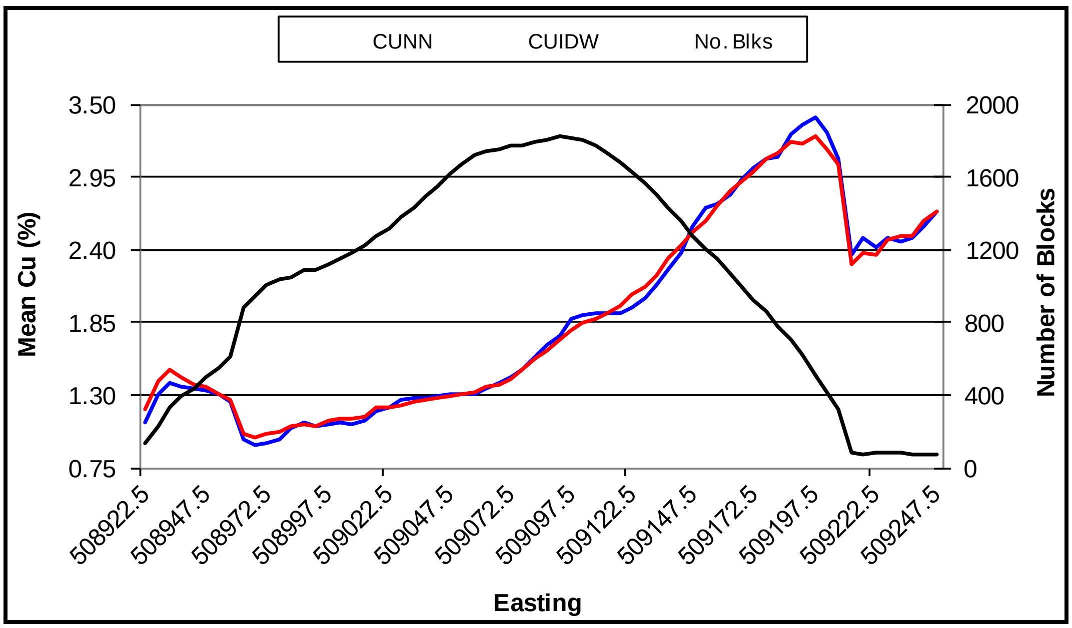


Figure 14.38 Lowry MZ Copper Swath Plot – Northing

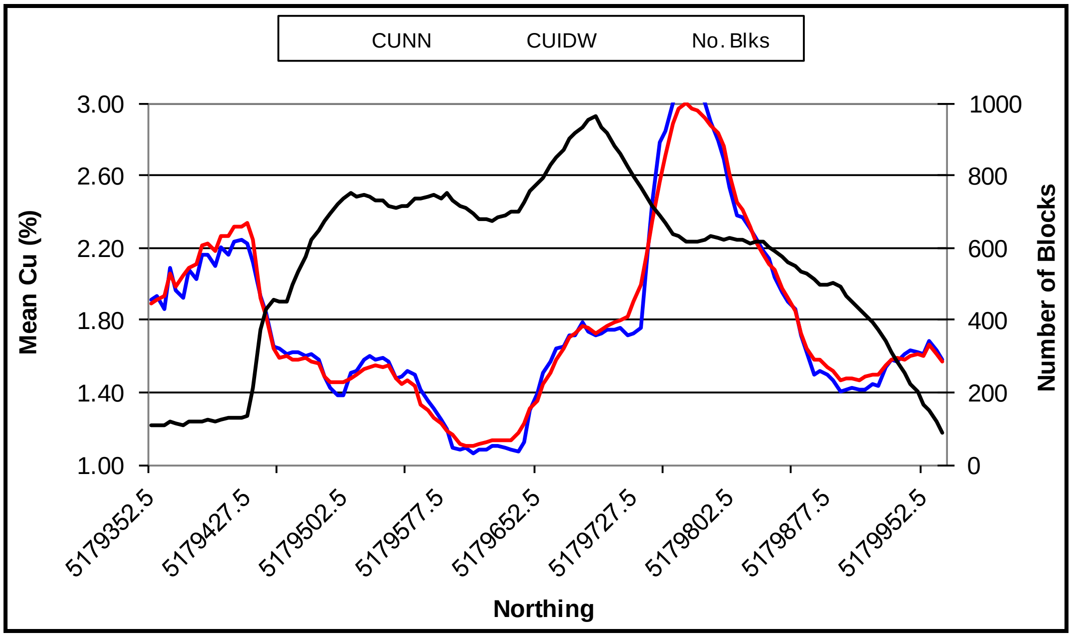
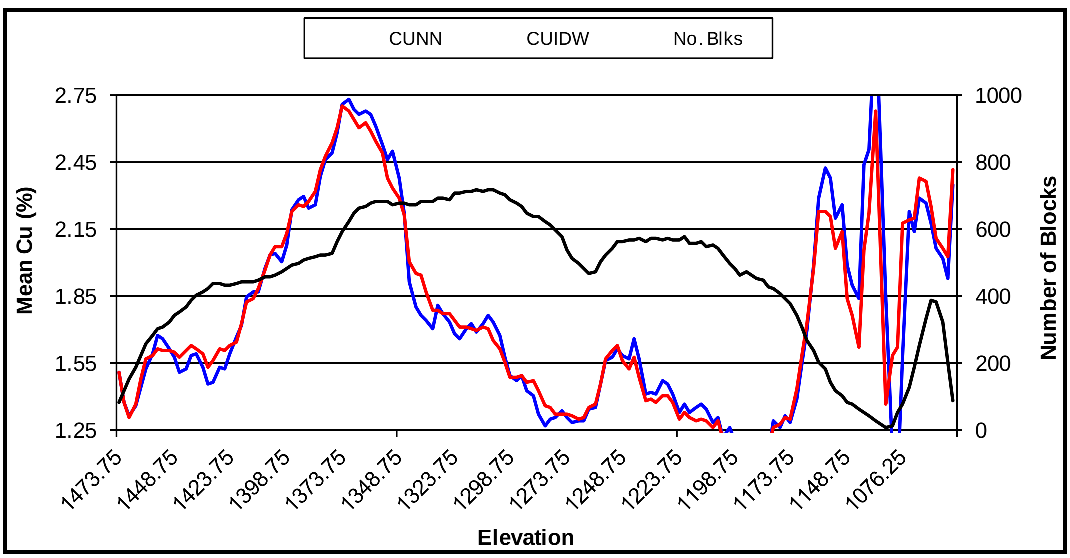


Figure 14.39 Lowry MZ Copper Swath Plot – Elevation



The swath plots shown in Figure 14.37 through Figure 14.39 show some local variation with the NN model grades showing more variation and the ID grades showing some smoothing. These graphs also provide an indication as to where higher grades exist. For example, copper grades tend to increase going from south to north (i.e. Figure 14.38).

Based on a visual examination and comparisons with a nearest neighbor model, it is RMI's opinion that the Lowry MZ model is globally unbiased and represents a reasonable estimate of undiluted in-situ resources.

14.35 LOWRY RESOURCE CLASSIFICATION

Blocks located inside of the Lowry MZ wireframe that were estimated by two or more drillholes were initially classified as Indicated Resources. Then a 3D solid was used to reset a subset of the initially classified Indicated blocks to Inferred. The 3D solid that was used to reset the blocks defines areas of weak mineralization or poor continuity. All blocks located inside of the dike swarm were also set to Inferred. RMI notes that all of the remaining Indicated Resource blocks were estimated by three drillholes with the average distance to the closest drillhole being 31 m.

14.36 LOWRY DENSITY DATA

A bulk density value of 3.18 g/cm^3 was used to tabulate resource tonnage. This value is based on the arithmetic average of 117 bulk density determinations from a variety of lithologies obtained from recent Tintina diamond core hole samples. Massive and semi-massive sulphide lithologies yielded greater density values than the average value but, given the variety of lithologies within the MZ wireframe, the average bulk density value was determined to be more appropriate. RMI recommends that additional work be undertaken to model the MZ so that appropriate bulk density values can be assigned to different lithologic units.

14.37 LOWRY RESOURCE SUMMARY

A cut-off grade of 1.6% copper was used to define an undiluted Inferred Mineral Resource for the Lowry Zone of 5,139,000 t with an average grade of 2.60% copper , 0.12% cobalt, 0.009 g/t gold, and 14.6 g/t silver. The cut-off grade was established by using:

- a copper price of US$2.75/lb
- a copper recovery of 81%
- mining costs of US$57/t
- processing costs of US$16.00/t
- G&A costs of US$5.00/t.

Table 14.22 summarizes resources at several cut-off grades. No credit was given to cobalt, gold, or silver in determining the cut-off grade. Undiluted Lowry Inferred Resources are summarized in Table 14.32.

Table 14.32 Lowry MZ Undiluted Indicated Mineral Resource

Cu Cut-off (%)	Tonnes ('000)	Metal Grades				Contained Metal			
		Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
0.0	9,560	1.74	0.08	0.006	12.3	367	17	1.8	3,781
0.2	8,901	1.86	0.08	0.006	12.7	365	16	1.7	3,634
0.4	8,383	1.96	0.08	0.007	13.0	362	15	1.9	3,504
0.6	7,815	2.07	0.09	0.006	13.3	357	16	1.5	3,342
0.8	7,160	2.19	0.09	0.006	13.6	346	14	1.4	3,131
1.0	6,413	2.34	0.09	0.006	13.9	331	13	1.2	2,866
1.2	5,539	2.54	0.10	0.007	14.1	310	12	1.2	2,511
1.4	4,790	2.73	0.10	0.007	14.6	288	11	1.1	2,248
1.6	**4,099**	**2.94**	**0.10**	**0.006**	**15.1**	**266**	**9**	**0.8**	**1,990**
1.8	3,568	3.12	0.11	0.006	15.6	245	9	0.7	1,790
2.0	3,075	3.32	0.11	0.007	16.0	225	7	0.7	1,582
2.2	2,639	3.52	0.11	0.007	16.3	205	6	0.6	1,383
2.4	2,307	3.70	0.12	0.007	16.7	188	6	0.5	1,239
2.6	1,944	3.92	0.12	0.007	17.1	168	5	0.4	1,069
2.8	1,663	4.13	0.13	0.007	17.5	151	5	0.4	936
3.0	1,464	4.30	0.13	0.007	17.9	139	4	0.3	843

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.33 Lowry MZ Undiluted Inferred Mineral Resource

Cu Cut-off (%)	Tonnes ('000)	Metal Grades				Contained Metal			
		Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
0.0	1,778	1.68	0.08	0.006	11.1	66	3	0.3	635
0.2	1,724	1.73	0.08	0.006	11.4	66	3	0.3	632
0.4	1,641	1.80	0.08	0.006	11.8	65	3	0.3	623
0.6	1,545	1.88	0.09	0.006	12.1	64	3	0.3	601
0.8	1,459	1.95	0.09	0.006	12.4	63	3	0.3	582
1.0	1,303	2.08	0.09	0.007	12.8	60	3	0.3	536
1.2	1,116	2.24	0.10	0.007	13.2	55	2	0.3	474
1.4	937	2.42	0.10	0.007	13.7	50	2	0.2	413
1.6	801	2.58	0.10	0.008	14.1	46	2	0.2	363
1.8	713	2.69	0.10	0.008	14.4	42	2	0.2	330
2.0	622	2.80	0.10	0.008	14.7	38	1	0.2	294
2.2	508	2.96	0.11	0.008	15.3	33	1	0.1	250
2.4	403	3.13	0.11	0.008	15.5	28	1	0.1	201
2.6	261	3.48	0.12	0.008	15.0	20	1	0.1	126
2.8	197	3.73	0.12	0.007	15.5	16	1	0	98
3.0	152	3.99	0.13	0.006	16.1	13	0	0	79

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

14.38 GENERAL DISCUSSION – LOWRY RESOURCE

RMI is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other factors that could materially affect the Lowry Inferred Mineral Resources that are the subject of this report.

Additional infill drilling, geotechnical studies, metallurgical test work, and environmental permitting will be required to determine the economics of this project and whether any portion of the resources will be affected by mining, processing, or permitting. The northeast trending dike swarm will need to be modelled in more detail, which will require additional drilling.

The reader should be aware that no resources from the Lowry deposit have been included in the mine plan for the purpose of assessing project economics for this updated PEA-level study.

15.0 MINERAL RESERVE ESTIMATES

There are no mineral reserves on the Property.

16.0 MINING METHODS

The preliminary mine plan presented in this section is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized.

In this section, the term "mineralized zone" refers to those estimated mineral resource blocks that are incorporated in the PEA mine design plan. In this section, the term "mining shapes" refers to conceptual plans containing mineralized zones that are above a nominal 1.9% copper cut-off grade after including planned dilution.

The Johnny Lee deposit consists of the UZ and LZ. The UZ is the larger of the two and ranges in depth from 20 to 190 m from the surface with a total vertical extent of 150 m. The LZ is steeper dipping and ranges in depth from 300 to 500 m from surface with a total vertical extent of 240 m. Although the deposit contains other metals, only copper was considered in the mine plan for this study. The process recovery of the other metals is under investigation.

The UZ mineralized zone ranges in thickness from 3 to 26 m and varies in dip from 0° to 20°. In some areas, the mineralized zone is a single lens while, in other areas, there are two sub-parallel lenses separated by 2 to 16 m of material below the cut-off grade. The hanging wall and footwall of the mineralized zone are not visual geologic contacts and will be defined by drillhole and sample assays during mining. The UZ is much larger than the LZ in size, and makes up 78% of the tonnage within the overall Johnny Lee mining shape.

The LZ mineralized zone ranges in thickness from 2 to 16 m and the dip varies from 20° to 37°. The hanging wall and footwall of the mineralized zone are not visual geologic contacts and will be defined by drillhole and sample assays during mining. The LZ constitutes 22% of the tonnage within the overall Johnny Lee mining shape.

Portions of the UZ are amenable to surface mining methods; however, these methods are not compatible with current and planned future surface land uses. This study focuses on the evaluation of potential underground mining methods only.

A comprehensive geotechnical/geohydrological assessment of the UZ and LZ has not yet been completed; however, drill core recoveries indicate that the rock mass may be of fair to good quality. Both the UZ and the LZ are at shallow depths and not likely to be subjected to high stress. Excavations have been designed and sequenced to account for poor ground conditions expected in the vicinity of the VVF.

The chosen mining method is selective enough to address issues related to changing ground conditions in the presence of faults. A number of active faces will be available to ensure that the designed production rate can be achieved despite poor ground conditions expected in some areas of the mine.

At the direction of Tintina, the planning cut-off grade was increased to 1.9% copper to ensure all tonnes mined generated a positive operating margin to optimize the Project economics and provide a faster payback for the capital costs. No marginal cut-off grade was used in the evaluation and there are no low-grade stockpiles in the plan.

Tintina supplied two block models that were used to estimate the mineralized material contained in the mining shapes: one block model for the UZ and one block model for the LZ. The two block models were used to estimate copper grades and rock densities. The specific gravity of the rock within the mineralized zones has been estimated as 3.99 for the UZ and 3.49 for the LZ. The specific gravity for all non-mineralized rock in both models has been estimated as 3.07.

The mining shapes include all planned dilution. Unplanned over-break dilution was estimated at 10% for both the UZ and LZ. For the UZ, dilution was assigned a grade of 1.3% copper, which was estimated using the UZ block model. For the LZ, dilution was assigned a grade of 1% copper, which was the dilution grade used in the previous PEA report (Tetra Tech 2012). In all cases, a mining recovery factor of 98% was used to estimate the amount of material that can be recovered from the planned mining shapes.

Table 16.1 summarizes the subset of Mineral Resources contained in the mine plan by mining area, and shows the dilution and recovery assumptions. A nominal 1.9% copper cut-off grade was used for planning purposes.

Table 16.1 Subset of Mineral Resources in Mine Plan

Area/ Class	In Stope ('000 t)	In Stope Cu Grade (%)	In Stope Cu ('000 lb)	Mining Recovery (%)	Dilution (%)	Dilution Cu Grade (%)	Diluted Mineral Resources Within Stopes*		
							'000 t	Cu Grade (%)	Cu ('000 lb)
UZ Northeast									
Measured	380	2.96	24,790	98.0	10	1.30	414	2.79	25,480
Indicated	1,499	2.79	92,221	98.0	10	1.30	1,632	2.64	95,050
Inferred	384	2.50	21,146	98.0	10	1.30	418	2.38	21,921
UZ Northwest									
Measured	666	2.88	42,220	98.0	10	1.30	725	2.72	43,452
Indicated	2,015	2.67	118,612	98.0	10	1.30	2,194	2.53	122,521
Inferred	117	2.62	6,793	98.0	10	1.30	128	2.49	7,023
UZ Southeast									
Measured	1,015	2.91	65,009	98.0	10	1.30	1,105	2.75	66,872
Indicated	1,321	2.63	76,456	98.0	1	1.30	1,438	2.49	79,044
Inferred	214	2.39	11,288	98.0	10%	1.30	233	2.28	11,729

table continues…

Area/ Class	In Stope ('000 t)	In Stope Cu Grade (%)	In Stope Cu ('000 lb)	Mining Recovery (%)	Dilution (%)	Dilution Cu Grade (%)	Diluted Mineral Resources Within Stopes*		
							'000 t	Cu Grade (%)	Cu ('000 lb)
UZ Southwest									
Measured	191	2.94	12,398	98.0	10	1.30	208	2.78	12,746
Indicated	573	2.80	35,299	98.0	10	1.30	624	2.65	36,379
Inferred	108	2.70	6,408	98.0	10	1.30	117	2.56	6,615
Lower Zone									
Measured	-	-	-	-	-	-	-	-	-
Indicated	2,215	5.43	265,006	98.0	10	1.00	2,411	4.99	265,017
Inferred	180	4.37	17,390	98.0	10	1.00	197	4.03	17,475
Total Johnny Lee									
Measured	2,252	2.91	144,418	98.0	10	1.30	2,452	2.75	148,551
Indicated	7,622	3.50	587,593	98.0	10	1.21	8,299	3.27	598.011
Inferred	1,004	2.85	63,025	98.0	10	1.25	1,093	2.69	64,764

Note: * after mining recovery was applied.

16.1 MINE ACCESS AND DEVELOPMENT

The Johnny Lee UZ and LZ will be accessed from a single portal, three main ramps, and one decline. There will be six raises that reach the surface, which will provide secondary egress and ventilation circuits. All personnel and materials will be transported through the portal and down the decline to the working areas. All mineralized material and waste will be trucked up the decline to stockpiles located on surface within 100 m of the portal. Paste backfill will be pumped from the paste plant through a pipe that will extend from the plant to the portal and down the decline to the working areas.

Tintina provided AMEC with a conceptual design for an exploration decline that will be driven to gather geological information. The exploration decline will be 1,500 m long from the portal to the end. The development of the exploration decline and portal construction are not included in the schedule or cost estimates, and are assumed to be completed before the commencement of mine development. The maximum gradient of the exploration decline is -15%.

Mine development designs were planned in order to satisfy the following objectives:

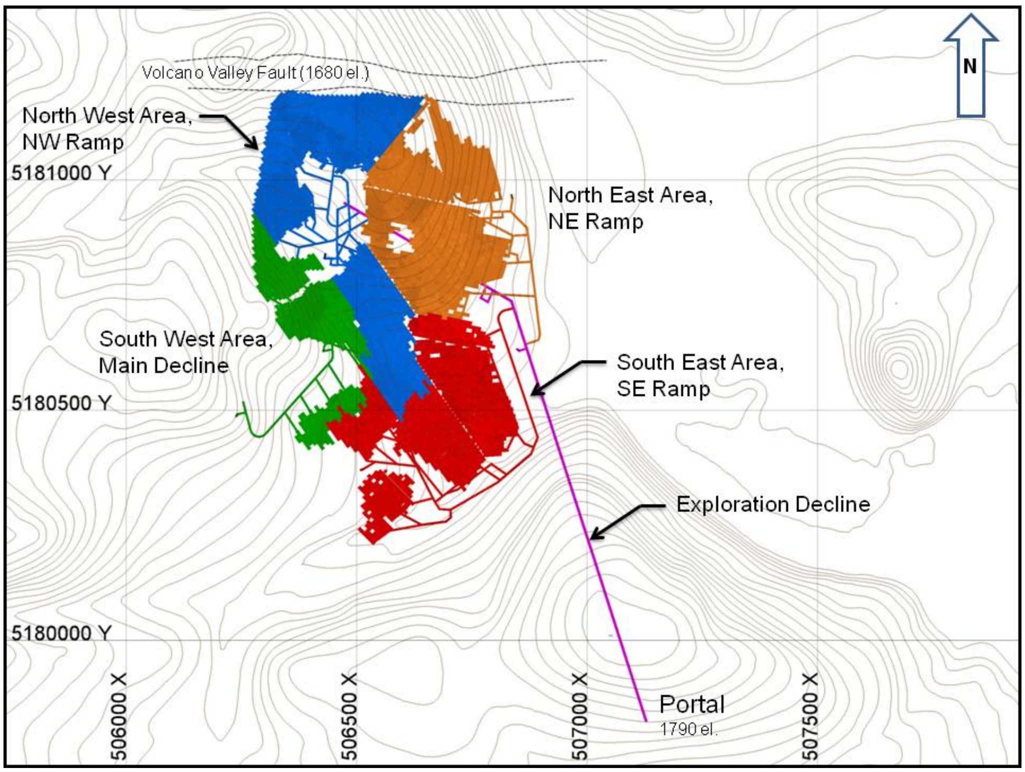
- Provide four independent access-ways to the five main production areas to reduce congestion and support a 3,300 t/d operation. All four main access-ways will originate from the exploration decline, about 1,000 m from the portal.
- Provide multiple intakes and exhaust airways to surface that will support a high volume of ventilation.
- Isolate exhaust air from the main mining areas.
- Avoid development near the VVF.

All development designs were planned with a maximum gradient of ±15%. All waste development was planned as a 5 m by 5 m drift cross section, with the exception of the decline to the LZ, which was planned as a 5.5 m by 5.5 m drift cross section for ventilation purposes.

The UZ will be accessed from three ramps and the main decline, which will originate from the exploration decline. Figure 16.1 illustrates the following four general mining areas that will be accessed from this development:

- UZ – Northeast Area
- UZ – Northwest Area
- UZ – Southeast Area
- UZ – Southwest Area.

Figure 16.1 Upper Zone Development

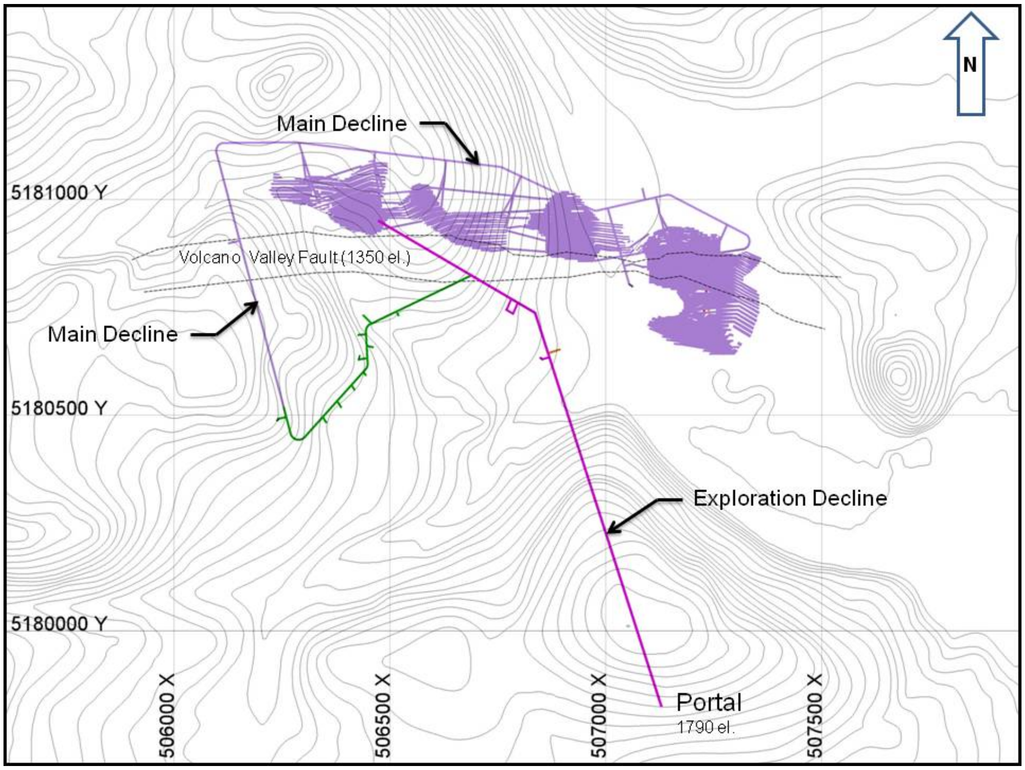


Source: AMEC 2013.

All main haulage-ways were planned at a minimum offset of 25 m from the production mining areas. Primary development in the footwall was designed to be a minimum of 20 m below the production areas. Secondary development in the footwall was planned to be between 10 m and 20 m below the production areas, depending on the thickness of the mineralized zone in that area.

The LZ will be accessed from the main decline, which will originate from the exploration decline. Nine levels have been planned at 25 m vertical spacing to access the drift-and-fill mining areas. Figure 16.2 shows a plan view of the LZ mining areas.

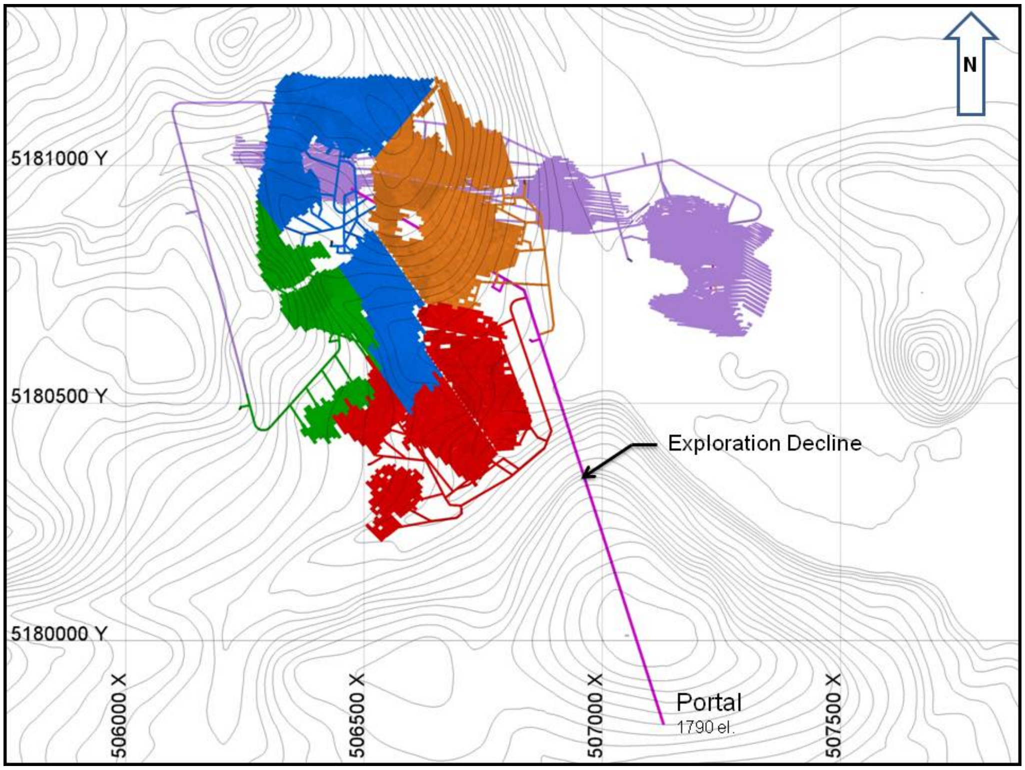
Figure 16.2 Lower Zone Development



Source: AMEC 2013.

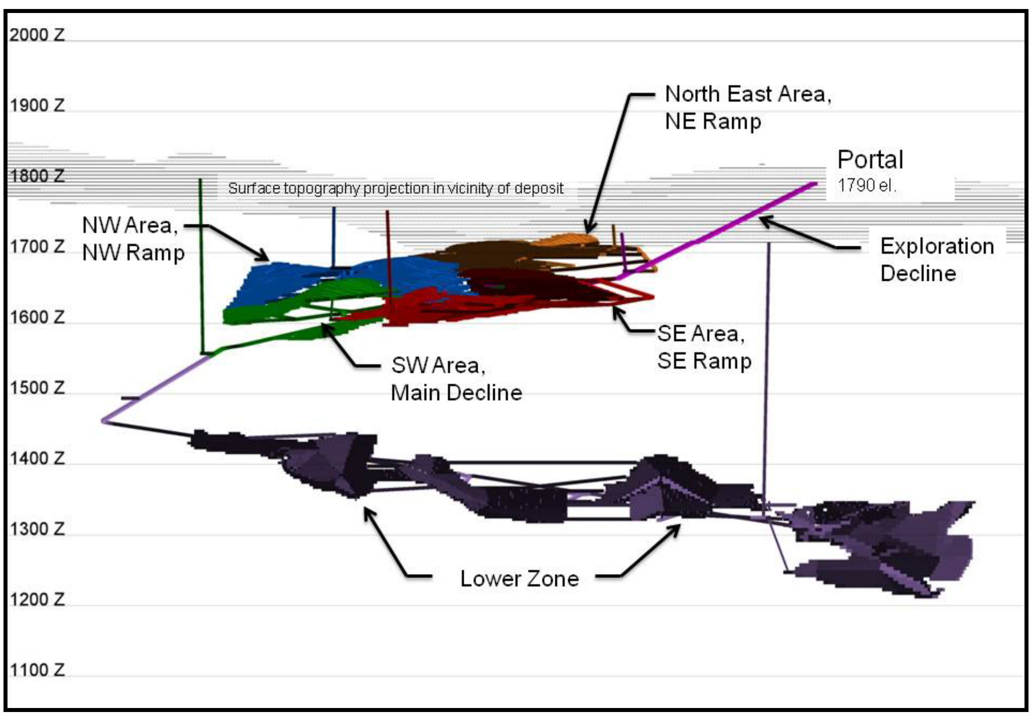
Figure 16.3 is a plan view of the UZ and LZ mining areas and primary development. Figure 16.4 is a longitudinal section view of the mine plan looking north. Figure 16.5 is a longitudinal section view of the deposit looking east. A 3D view of the planned mining operation is shown in Figure 16.6.

Figure 16.3 Plan View of Johnny Lee Mining Areas and Primary Development



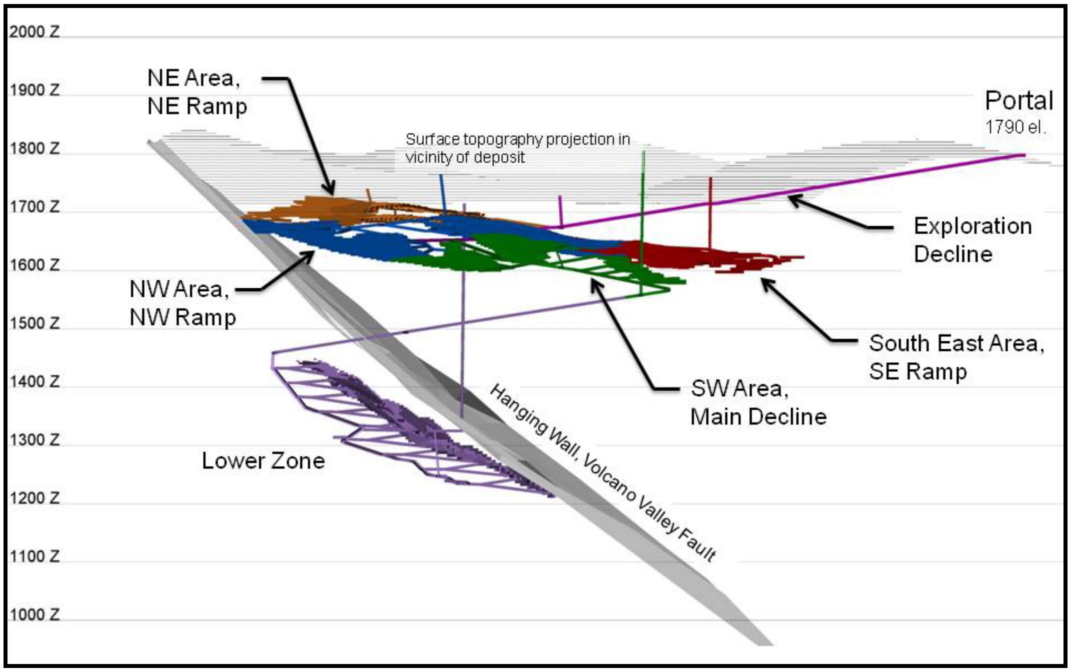
Source: AMEC 2013.

Figure 16.4 Longitudinal Section Looking North



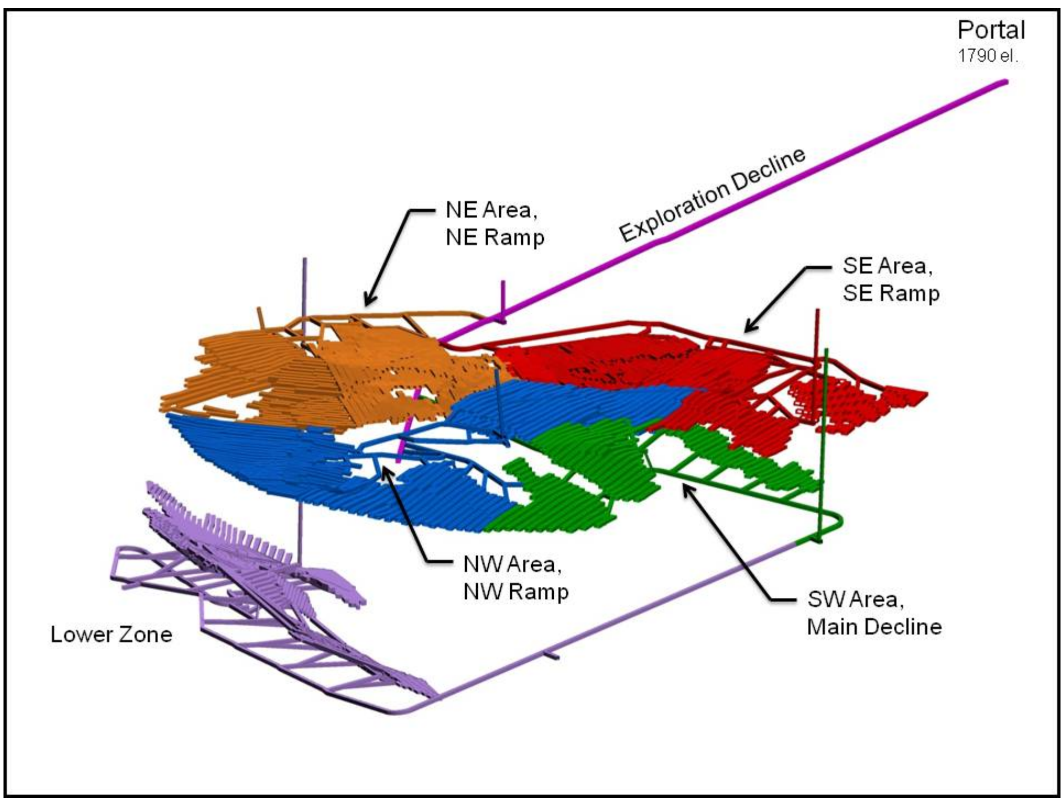
Source: AMEC 2013.

Figure 16.5 Longitudinal Section Looking East



Source: AMEC 2013.

Figure 16.6 3D View of the Mine Plan



Note: No scale. Viewing direction of S65E, plunge of 17°.
Source: AMEC 2013.

16.2 MINING AND BACKFILL METHODS

The drift-and-fill mining method was selected for the UZ due to the overall thinness and shallow dip of the mineralized zones, and for flexibility in adapting to local variations. Although the ground conditions may allow for larger stoping methods, the geometry of the deposit was the controlling factor in selecting this method. In most areas, there are either one or two mining horizons in vertical extent (4 to 10 m). In a few areas, the thickness allows multiple-stacked mining horizons. The following design constraints and general assumptions were made regarding this mining method:

- All drift-and-fill mining shapes were planned at 8 m wide and 4 to 5 m high. The average height of the mining shapes is 4.5 m.
- The drift-and-fill panels were laid out 'off-strike' to target a maximum gradient of ±12% for the majority of the drifts. Localized gradients are ±15%. A few areas of a few panels have gradients of ±16%.
- It was estimated that 80% of all drift-and-fill mining shapes will be driven 5 m wide and then slashed out 3 m for a total width of 8 m. The entire excavation will then be backfilled.
- Twenty percent of the mineralized zones are thick enough to allow for a bulk benching method. In this method, an 8 m-wide drift will be driven first and left open. The access will then be ramped down to mine an 8 m wide by 4.5 m bench below the first drift. The entire excavation will then be backfilled.
- Based on the limited geotechnical information available for the study, ground support in all areas was assumed to be 1.8 m resin bolts on a 1.2 m by 1.2 m pattern with welded wire mesh installed within 2.5 m of the sill. It was assumed that a mechanized rock bolter will be used to install the bolts and mesh, and shotcrete will not be required for ground support.
- Drilling of the 5 m wide and 8 m wide drifts was planned using a two boom face jumbo (electric/hydraulic). All slashing and benching would be mined using the same drill.
- The explosive assumed for this mining method is ammonium nitrate/fuel oil (ANFO), which would be loaded using an explosives truck.
- Mucking of the production faces, slashes, and benches was assumed to be done using a 5.4 m^3 load-haul-dump (LHD) machine. Mineralized material will be removed from the mining areas and loaded directly into 40 t trucks in the level access drifts or, alternatively, into muck bays. The trucks are not scheduled to enter the drift-and-fill areas.
- Paste backfill was assumed for this mining method, and 100% of the paste fill was planned to be cemented. A paste fill plant will be located near the process concentrator on surface, and a pipe distribution system will deliver the paste fill to the underground mining areas

- All mining in the UZ is planned in a "bottom-up" sequence. The lowest mining horizons in a particular area will be mined first, backfilled, and then mining will progress upward. This sequence minimizes the cement requirements for the paste fill.

The drift-and-fill mining method was also selected for the LZ due to the overall thinness and variable dip of the mineralized zones. The following design constraints and general assumptions were made regarding the mining method for this zone:

- All drift-and-fill mining shapes above the 1320 elevation (which constitute 48% of the tonnage of the LZ) were planned as 3.5 m by 3.5 m drifts. These smaller drifts were planned above this elevation to reduce dilution in this thinner portion of the mineralized zones.
- All drift-and-fill mining shapes below the 1320 elevation were planned as 5 m by 5 m drifts (52% of the total).
- Neither slashing nor benching were planned in the LZ.
- All drift-and-fill mining shapes were planned at a 0% gradient on sublevels.
- Unlike the sequencing in the UZ, mining in the LZ is not all planned as "bottom-up" mining. Four sills have been planned so that the production rate can be maximized in this higher-grade zone. The four initial production sublevels will be mined in a bottom-up sequence. However, in the second half of the mine life, mining will be required underneath the backfill from the four initial production sublevels. Backfilling of the initial mining horizons will require high-strength paste fill with a higher than normal cement content.
- For the 5 m by 5 m mining shapes, ground support, drilling, blasting, mucking, and backfilling were planned in the same manner as the UZ mining.
- For the 3.5 m by 3.5 m mining shapes, ground support will be installed using handheld jackleg drills. A smaller 1.8 m^3 LHD will be used for mucking together with a smaller single boom face drill jumbo (electric/hydraulic).

16.3 PRODUCTION AND DEVELOPMENT RATES

Expected production rates were estimated for each of the five production areas:

- UZ – Northeast Area
- UZ – Northwest Area
- UZ – Southeast Area
- UZ – Southwest Area
- LZ.

For scheduling purposes, these five main areas were further subdivided into drift-and-fill mining panels. The maximum number of active faces that each panel could support was estimated in order to determine the expected production rate for each panel. The following key assumptions were used to estimate the sustainable production rate out of each mining panel:

- drift-and-fill drive average advance rate of 1.8 m/d per heading to ensure multi-heading production efficiencies
- minimum pillar of 16 m between active work areas
- slash average advance rate of 6 m/d per heading
- paste backfill set-up time of 4 days
- paste backfill cure time of 30 days
- paste backfill plant capacity of 2,030 t/d.

Based on these assumptions, it was estimated that the UZ can yield 912,500 t/a (2,500 t/d) and the LZ can yield 292,000 t/a (800 t/d) from the mineralized zones. To achieve the 1.2 Mt/a production rate, it is estimated that 18 active stope areas will be required in the UZ and 14 will be required in the LZ. An active stope area is defined as a drift-and-fill stope in various stages of the mining cycle including driving the heading, slashing, backfill set-up, paste fill pumping, and the 30-day backfill curing time.

To achieve a production rate of 3,300 t/d, the following number of heading blasts will be required:

- UZ – 4.1 heading blasts out of an 8 m by 4.5 m drift-and-fill stope
- LZ – 2.5 heading blasts out of a 5 m by 5 m drift-and-fill stope.

To sustain a production rate of 3,300 t/d, the average long-range drifting rate was estimated at 30 m/d, which includes waste development and stope heading advancement. This rate does not include the UZ slashing or benching.

16.4 Proposed Life of Mine Schedules

16.4.1 Proposed Production and Development Schedule

Table 16.2 shows the projected LOM production schedule. The mine life is estimated to be 12 years, including 1.5 years of pre-production development and 10.5 years of production.

It is assumed that underground mine development will start at the beginning of Year 1. All initial development will start underground from the pre-existing exploration decline, approximately 1 km from the portal. Initial development in mineralized material will start in Q2 of Year 2 and the full production rate of 3,300 t/d will be achieved in Q3 of Year 3.

Concentrator start-up is planned for Q3 of Year 2. In Year 2, the UZ will provide 575,000 t of concentrator feed.

The drifting rate (mineralized material plus waste) peaks at 33 m/d in Year 3 before settling to about 30 m/d long term.

Table 16.2 LOM Proposed Production and Development Schedule

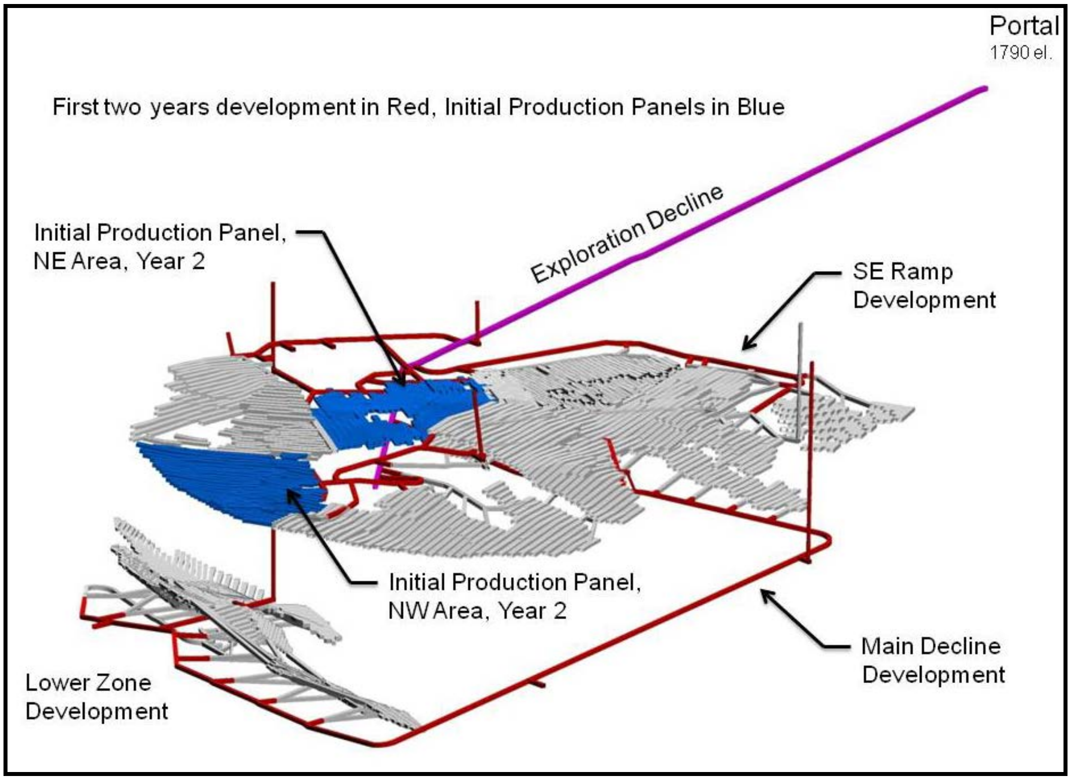
	LOM Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
UZ – Northeast Area													
Measured & Indicated ('000 t)	2,046	-	296	208	242	211	197	253	243	217	100	77	-
Measured & Indicated Cu Grade (%)	2.67	-	2.81	2.79	2.65	2.52	2.60	2.81	2.74	2.60	2.39	2.39	-
Measured & Indicated Cu ('000 lb)	120,530	-	18,354	12,778	14,126	11,740	11,308	15,706	14,717	12,461	5,273	4,068	-
Inferred ('000 t)	418	-	13	3	16	16	30	79	49	95	67	52	-
Inferred Cu Grade (%)	2.38	-	2.43	2.25	2.45	2.29	2.48	2.62	2.65	2.28	2.15	2.15	-
Inferred Cu ('000 lb)	21,921	-	687	162	867	786	1,635	4,548	2,869	4,771	3,160	2,438	-
UZ – Northwest Area													
Measured & Indicated ('000 t)	2,919	-	256	341	196	222	237	214	242	363	413	336	100
Measured & Indicated Cu Grade (%)	2.58	-	2.89	2.89	2.66	2.52	2.48	2.47	2.47	2.45	2.47	2.50	2.64
Measured & Indicated Cu ('000 lb)	165,974	-	16,260	21,701	11,498	12,308	12,957	11,637	13,182	19,639	22,446	18,502	5,844
Inferred ('000 t)	128	-	10	14	7	7	9	8	9	18	22	17	7
Inferred Cu Grade (%)	2.49	-	2.82	2.82	2.45	2.15	2.22	2.24	2.24	2.47	2.51	2.54	2.57
Inferred Cu ('000 lb)	7,023	-	644	860	375	338	416	389	440	972	1,238	956	394
UZ – Southeast Area													
Measured & Indicated ('000 t)	2,543	-	-	136	240	421	407	332	316	136	136	254	166
Measured & Indicated Cu Grade (%)	2.60	-	-	2.82	2.76	2.67	2.68	2.65	2.55	2.54	2.54	2.36	2.32
Measured & Indicated Cu ('000 lb)	145,917	-	-	8,471	14,564	24,781	24,002	19,400	17,755	7,585	7,630	13,240	8,489
Inferred ('000 t)	233	-	-	0	0	36	33	27	53	18	18	30	17
Inferred Cu Grade (%)	2.28	-	-	2.10	2.10	2.31	2.31	2.31	2.31	2.30	2.30	2.20	2.17
Inferred Cu ('000 lb)	11,729	-	-	19	20	1,820	1,683	1,396	2,701	909	917	1,440	825
UZ – Southwest Area													
Measured & Indicated ('000 t)	832	-	-	176	177	-	-	-	-	60	142	133	143
Measured & Indicated Cu Grade (%)	2.68	-	-	3.31	3.31	-	-	-	-	2.21	2.21	2.22	2.22
Measured & Indicated Cu ('000 lb)	49,125	-	-	12,812	12,907	-	-	-	-	2,949	6,946	6,507	7,004

table continues…

	LOM Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Inferred ('000 t)	117	-	-	34	35	-	-	-	-	6	14	14	15
Inferred Cu Grade (%)	2.56	-	-	2.89	2.89	-	-	-	-	2.06	2.06	2.12	2.12
Inferred Cu ('000 lb)	6,615	-	-	2,181	2,198	-	-	-	-	261	614	656	706
LZ													
Measured & Indicated ('000 t)	2,411	-	-	208	264	264	263	272	278	277	238	186	161
Measured & Indicated Cu Grade (%)	4.99	-	-	4.90	5.29	5.29	5.22	4.91	4.88	4.86	4.64	4.74	5.01
Measured & Indicated Cu ('000 lb)	265,017	-	-	22,471	30,817	30,817	30,240	29,461	29,904	29,739	24,388	19,427	17,754
Inferred ('000 t)	197	-	-	27	28	28	29	20	14	15	16	12	8
Inferred Cu Grade (%)	4.03	-	-	3.69	4.19	4.19	4.10	4.43	4.21	3.84	3.60	3.87	4.02
Inferred Cu ('000 lb)	17,475	-	-	2,200	2,563	2,563	2,667	1,929	1,321	1,233	1,250	1,062	687
Total Johnny Lee													
Measured & Indicated ('000 t)	10,751	-	552	1,069	1,119	1,118	1,104	1,071	1,079	1,054	1,030	986	570
Measured & Indicated Cu Grade (%)	3.15	-	2.85	3.32	3.40	3.23	3.23	3.23	3.18	3.12	2.94	2.84	3.11
Measured & Indicated Cu ('000 lb)	746,562	-	34,614	78,233	83,912	79,647	78,507	76,204	75,557	72,372	66,682	61,744	39,090
Inferred ('000 t)	1,093	-	23	79	86	86	101	134	125	151	136	125	47
Inferred Cu Grade (%)	2.69	-	2.61	3.12	3.19	2.90	2.88	2.80	2.65	2.45	2.39	2.38	2.52
Inferred Cu ('000 lb)	64,764	-	1,331	5,422	6,023	5,508	6,401	8,261	7,331	8,146	7,179	6,551	2,612
Capital Development													
Lateral (m)	9,370	3,432	3,257	1,215	435	189	184	184	151	139	184	-	-
Vertical (m)	1,084	327	123	616	6	-	12	-	-	-	-	-	-
Expensed Development													
Lateral (m)	10,313	-	421	988	1,066	1,066	1,066	1,066	1,066	1,066	1,014	938	558
Waste ('000 t)	1,692	310	326	220	121	101	101	101	98	97	97	76	45
Paste Fill ('000 t)	7,272	-	353	705	740	740	740	740	740	740	716	682	379

Figure 16.7 shows the development planned during the first two years of the Project and the two initial production areas planned in the UZ. The estimated mineralization to waste ratio is 9:1 for the UZ, 4:1 for the LZ, and 7:1 for the entire deposit.

Figure 16.7 First Two Years of Development



Note: No scale. Viewing direction of S65E, plunge of 17°.
Source: AMEC 2013.

An underground development contractor will be mobilized at the start of Year 1 and kept on site for 2.5 years to drive the majority of the capital development for the mine. The Owner's mining crews will be hired at the start of Year 2. A steep ramp up of Owner's crews was is assumed for the first half of Year 2 before production of mineralized material starts mid-year. The Owner's crews will be responsible for all production mining. The Owner's crews will drive the southeast ramp in the first half of Year 2 to provide a working area for training and ramp-up of Owner's personnel during this six-month period.

Once mobilized, the contractor will simultaneously drive three capital development headings. The Northeast and Northwest Ramps will each be driven at a rate of 2.5 m/d to open up the initial production mining areas in the second half of Year 2. The main decline is the priority development heading and will be driven at a rate of 5 m/d during Years 1 and 2. The main decline development will provide access to high grade mineralized material in the LZ and in the Southwest Area of the UZ. The total contractor development rate will be 10 m/d in Year 1, after initial set-up. Blasting on demand and dedicated equipment are planned for the main decline development during the first two years in order to achieve the development rate of 5 m/d.

A short ventilation raise will be established at the start of Year 1 that extends from the pre-existing exploration decline to surface. This ventilation raise will provide flow-through ventilation close to the starting point of the three initial development drives. A second ventilation raise will be established from the bottom of the main decline development to surface during Q3 of Year 1. The second raise will provide flow-through ventilation that will allow for continued prioritized development of the main decline during Year 2. The main LZ ventilation raise and escape hoisting system will be installed at the end of Year 2. Production from the upper portions of the LZ will start at the beginning of Year 3. The contractor will develop the bottom section of the LZ in the first half of Year 3, after which, the contractor will demobilize and Owner's crews will begin production in the lower sections of the LZ.

16.4.2 Mine Mobile Equipment/Fleet Plan

The mobile equipment fleet for the Johnny Lee deposit was calculated based on first principles using the mine production and development schedule. During the pre-production phase of the Project, the contractor will use equipment provided by the Owner. All mobile equipment is planned to be secured using capital leases. New capital leased equipment will replace equipment after a particular lease expires during the mine life.

During steady-state production, the available run time for major equipment is estimated to be 16 h/d based on 2 shifts per day, 10-hour shift schedules, and accounting for downtime due to safety meetings, lunch, travel and pre-operation checks. The LHD and scissor lift equipment will have an estimated utilization/availability level of 70% while the rock bolters and the drill jumbos will have an estimated utilization/availability level of 80%.

Forty-tonne trucks will be used to haul rock from the work areas to surface stockpiles within 100 m of the portal. Rehandling of rock from the surface stockpiles to the waste dump or concentrator is discussed in Section 18.0.

Other than the 3.5 m by 3.5 m headings, all work areas will use the same sized equipment, which includes a two boom drill jumbo, a mechanized rock bolter, and a 5.4 m^3 LHD. The smaller 3.5 m by 3.5 m areas of the LZ will use a single boom drill jumbo, 1.8 m^3 LHD, and hand-held jacklegs for rock bolting.

The underground paste backfill crew will be assigned a scissor lift. The projected mobile equipment requirements are provided in Table 16.3.

Table 16.3 Mobile Equipment Schedule Plan

Equipment Summary	Year											
	1	2	3	4	5	6	7	8	9	10	11	12
Jumbo												
Two Boom	2	3	5	4	4	4	4	4	4	4	4	2
Single Boom	-	-	1	1	1	1	1	1	1	1	1	1
Rock Bolter												
Mechanized Bolter	2	2	4	4	4	4	4	4	4	4	3	2
Handheld Stoper	-	-	3	3	3	3	3	2	2	3	2	1
LHD												
5.4 m^3	2	3	4	4	4	4	4	4	4	4	4	3
1.8 m^3	-	-	1	1	1	1	1	1	1	1	1	1
Scissor Lift	2	2	4	4	4	4	4	4	4	4	3	1
Haul Truck – 40 t	1	4	4	4	4	4	4	4	4	4	4	3
Auxiliary Equipment												
Grader	1	1	1	1	1	1	1	1	1	1	1	1
Forklift	1	1	1	1	1	1	1	1	1	1	1	1
Boom Truck	1	1	1	1	1	1	1	1	1	1	1	1
ANFO Loader	1	2	3	3	3	3	3	3	3	3	3	3
Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1
Underground Light Truck	2	2	9	9	9	9	9	9	9	9	9	9

16.4.3 PROJECTED MINE WORKFORCE

The personnel required to support the underground mine includes hourly employees and salaried staff for the Owner and for the contractor. The total workforce required for the Project is listed by year in Table 16.4.

Table 16.4 Projected Mine Workforce

Total Workforce	Year											
	1	2	3	4	5	6	7	8	9	10	11	12
Chief Engineer	-	1	1	1	1	1	1	1	1	1	1	1
Mine Engineer	-	1	2	2	2	2	2	2	2	2	2	2
Surveyor	-	1	2	2	2	2	2	2	2	2	2	2
Technician	-	1	2	2	2	2	2	2	2	2	2	2
Chief Geologist	-	1	1	1	1	1	1	1	1	1	1	1
Geologist	-	1	1	1	1	1	1	1	1	1	1	1
Underground Geologist	-	2	4	4	4	4	4	4	4	4	4	4
Mine Superintendent	-	1	1	1	1	1	1	1	1	1	1	1
Safety	-	1	1	1	1	1	1	1	1	1	1	1
Trainer	-	1	2	2	2	2	2	2	2	2	2	2

table continues...

Total Workforce	Year 1	2	3	4	5	6	7	8	9	10	11	12
Mine Captain	-	1	1	1	1	1	1	1	1	1	1	1
Underground Supervisor	-	2	4	4	4	4	4	4	4	4	4	4
Mine Clerk	-	1	1	1	1	1	1	1	1	1	1	1
Underground Miners	-	38	98	120	120	120	120	116	116	116	96	64
Underground Mine Paste Crew	-	-	12	12	12	12	12	12	12	12	12	-
Underground Maintenance		9	18	14	14	14	14	14	14	14	14	10
Subtotal	-	59	151	169	169	169	169	165	165	165	145	97
Contractors												
Underground Management												
Project Engineer	1	1	1	-	-	-	-	-	-	-	-	-
Safety Supervisor	1	1	1	-	-	-	-	-	-	-	-	-
Clerk	1	1	1	-	-	-	-	-	-	-	-	-
Development Shift Bosses	3	3	2	-	-	-	-	-	-	-	-	-
Surveyors	2	1		-	-	-	-	-	-	-	-	-
Underground Miners	21	24	27	-	-	-	-	-	-	-	-	-
Maintenance Personnel	7	7	3.5	-	-	-	-	-	-	-	-	-
Support Personnel	1	1	1	-	-	-	-	-	-	-	-	-
Total Personnel	37	98	185	169	169	169	169	165	165	165	145	97

16.4.4 MINERALIZED MATERIAL AND WASTE HANDLING PLAN

At full production, mineralized material (for concentrator feed) and waste rock from the Johnny Lee deposit will be handled using rubber-tired equipment including four 5.4 m^3 LHDs and one 1.8 m^3 LHD. Mineralized material and waste will be hauled from the mining areas to the surface using a fleet of 40 t diesel trucks. The trucks were sized at 40 t after consideration of capital, operating, and ventilation requirements for various truck size alternatives.

The main decline will be driven 5 m wide by 5 m high with an arched back, which provides enough clearance for a 40 t haulage truck to operate safely without interference from ventilation ducting and other mine services. Truck loading arrangements will be developed at the intersection of the decline with each main level access. The back of the intersection will be increased to 6.5 m to facilitate side-loading of the trucks by the LHDs. Based on an average tramming distance of 300 m along the level from a remuck bay (in the production area or along the decline) to the truck loading area, three 5.4 m^3 LHDs will be required to be in operation at full production for the UZ. In the LZ, one 5.4 m^3 LHD and one 1.8 m^3 LHD will be required. These LHDs will be primarily dedicated to the haul truck fleet; however, while the haul trucks are en route to the surface, these LHDs can also be used for production, clean-up, or in other miscellaneous areas.

The proposed backfill method will be paste backfill delivered to the production areas via a piping distribution system from the main decline to the production area. Therefore, no back haulage of material will be required for the truck fleet.

Waste and low-grade material produced primarily from development headings in the mine will be hauled to surface stockpiles within 100 m of the portal. The requirements, design, and cost of all surface stockpile areas, surface equipment, and surface operators were determined by Tetra Tech and are discussed in Section 18.0.

Maintenance and fuelling of all trucks and LHDs will be conducted on the surface. The requirements, design, and cost of all surface shops, their associated equipment, and operators were determined by Tetra Tech and are discussed in Section 18.0.

16.4.5 PLANNED MINE VENTILATION

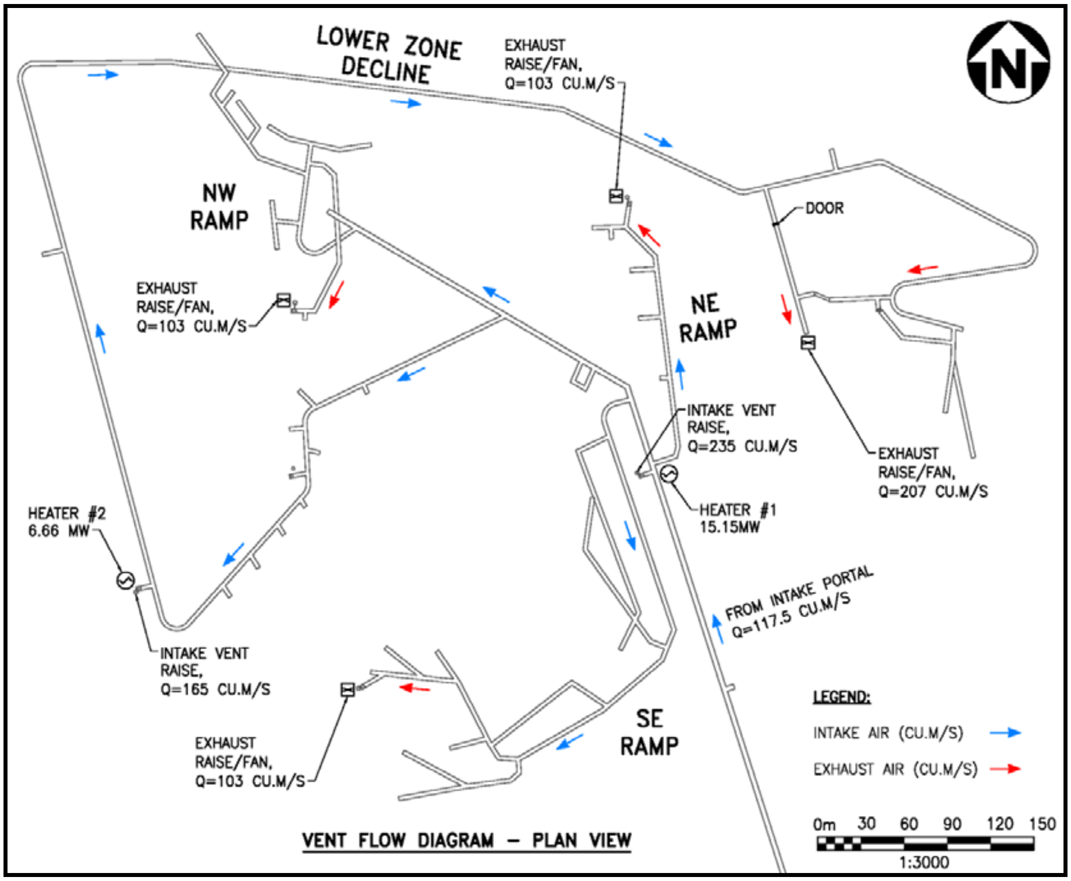
The proposed mine will draw a total airflow of 517 m^3/s (1.1 Mcfm). The required airflow rates are based on the maximum production rate of 3,300 t/d for a mechanized drift-and-fill operation.

Intake air will be drawn through the portal and two ventilation raises. The airflow will be distributed to the working areas through the Northeast, Northwest, and Southeast Ramps and down the main decline to the LZ. Air will be exhausted to surface through four ventilation raises in the northeast area, northwest area, southeast area, and the LZ. The dimensions of the planned ventilation raises and the mine airways were determined using air velocity guidelines.

To determine the airflow distribution across the mine and size the ventilation system mine air heaters, a preliminary ventilation network was built in VnetPC (ventilation software) to represent the mine layout at its maximum production rate.

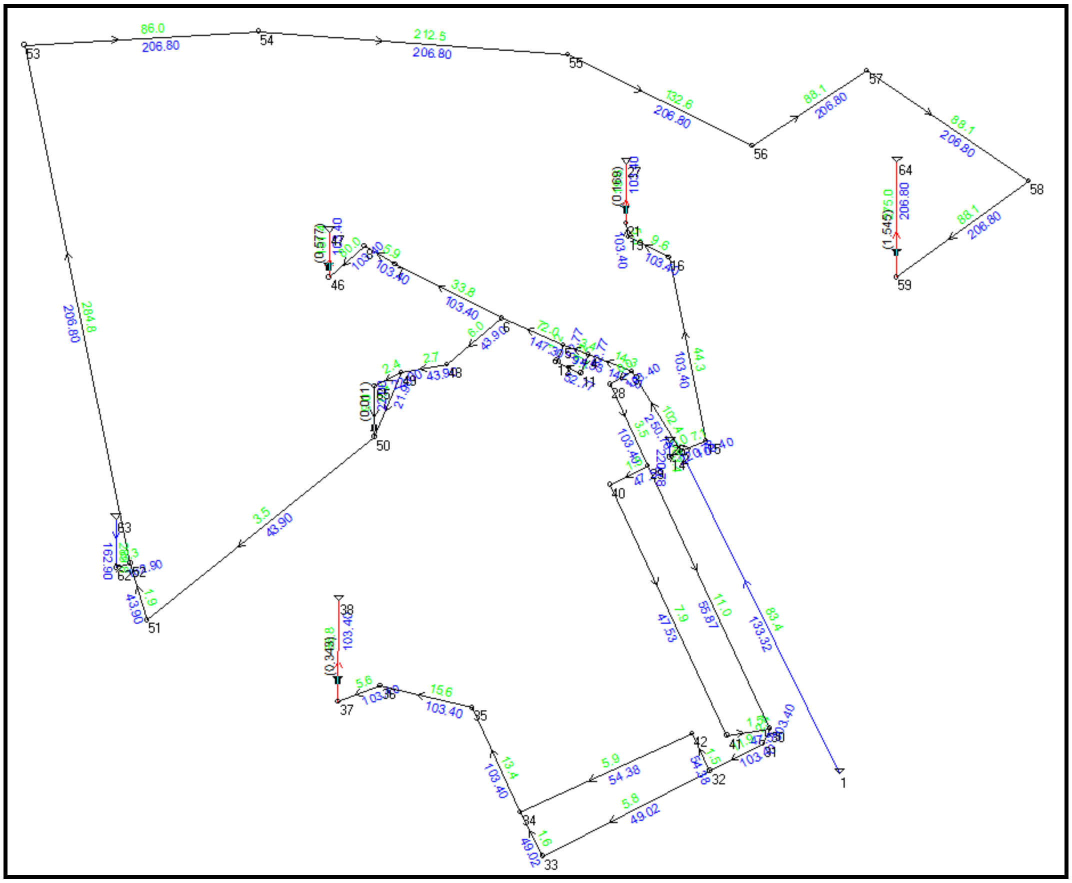
Figure 16.8 shows the primary ventilation circuits, exhaust raises and fans, intake raises, and mine air heater locations. Figure 16.9 shows the VnetPC network analysis results.

Figure 16.8 Ventilation Schematic



Source: AMEC, 03 May 2013.

Figure 16.9 VnetPC Analysis Network Diagram



Notes: no scale; Source: AMEC 2013.

Main Underground Fans

Two fans will be installed underground at the bottom of each of the four exhaust raises. The UZ exhaust raises in the Northeast, Southeast and Northwest Areas will exhaust at an average airflow rate of 103 m^3/s (220,000 cfm) each for most of the mine life. Each of the UZ raises will reach a peak exhaust airflow rate of 188 m^3/s (400,000 cfm), and each raise will have two 75 kW (100 hp) underground fans, each capable of pulling air at a rate of 103 m^3/s (220,000 cfm). For each of these fan installations, one fan will be running for normal operating points and the second fan will run during increased production activity in that area.

The LZ exhaust raise will exhaust at an airflow rate of 207 m^3/s (440,000 cfm) and will have two 225 kW (300 hp) underground fans running continuously.

Secondary Underground Fans

The auxiliary ventilation system includes auxiliary fans for the development and operation of the mine. For the first three years, three auxiliary fans with a motor size of 186 kW (250 hp), will be required for primary development.

During operations, auxiliary fans will be required to push air through a 1.1 m (42 in) diameter duct over a length of 250 m (820 ft). These fans are 56 kW (75 hp) in size and will provide airflow of 24 m^3/s (50,000 cfm) to each production area, which will be sufficient to ventilate the required underground mobile equipment and personnel.

Mine Air Heaters

The heating system will consist of two propane air heating arrangements. One heater will be located at the top of each of the two ventilation intake raises.

Heater No. 1 will be located at the top of the intake raise located on the exploration decline. Heater No. 2 will be located at the top of the intake raise located on the LZ main decline. Heater No. 1 will heat the intake air to 22.2°C and in turn heat the mine air to 2.8°C. The burner size of this heating system will be 15.2 MW. Two propane tanks will be required for this system.

Heater No. 2 will heat the intake air from the winter design temperature of -30°C to 2.8°C. The burner size of this heating system will be 6.7 MW. This system will incorporate an additional propane tank.

Together, the two heaters will utilize 1.08 ML of propane fuel per year. The heating system propane consumption costs are calculated at $0.56/t processed.

16.4.6 Emergency Egress and Refuge

During initial development and as the mine progresses, the primary escape-way for mine personnel is up the exploration decline and out the portal.

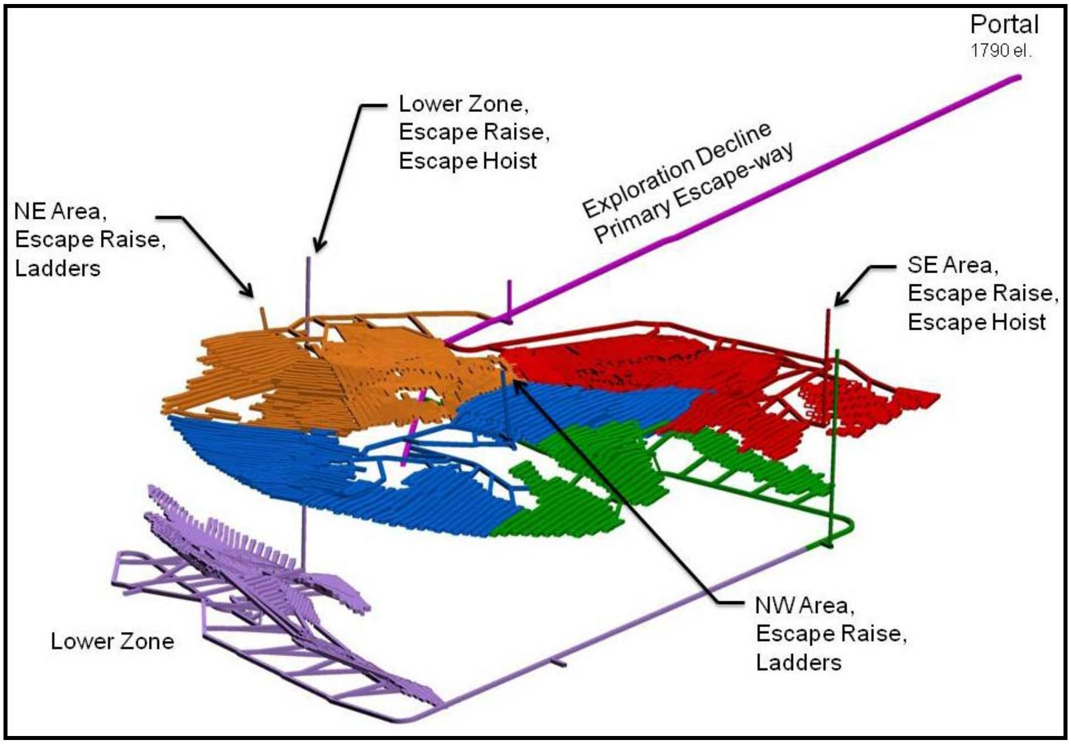
By the start of Year 4, the mine will have four secondary escape raises to surface for mine personnel. Two of these raises are less than 91.4 m (300 ft) in height and will have ladders installed to the surface. Two of the escape raises will be serviced by an escape hoist located at the top of the raises. Figure 16.10 shows the primary escape-way as well as the locations of the four escape raises.

Production from the five main areas will not begin until the secondary escape-way is completed for each area.

After the contractor is mobilized at the beginning of Year 1, a 20-person portable refuge chamber will be installed near the bottom of the exploration decline where the development activity starts. This unit will be moved down the decline as development progresses. A second 20-person portable refuge chamber will be installed permanently in the UZ in Year 3. The first unit will be installed permanently in the LZ in Year 3 after development in that area is completed. The refuge units will be strategically located in high-traffic areas of the mine.

The two escape raises that have hoists installed at the top are also planned as ventilation air exhaust raises. Since the exhaust fans will be arranged underground, the top of the egress raises will have a concrete collar foundation and a tripod style headframe over the raise that will support the secondary egress conveyance. Each secondary egress raise will incorporate a 60 hp hoist mounted in a sea container and will provide hoisting speeds of up to 0.9 m/s. The bullet-shaped conveyance will be capable of evacuating two men per trip, and rope guided to eliminate rotation or drifting into the raise walls.

Figure 16.10 Proposed Mine Egress



Notes: No scale. Viewing direction of S65E, plunge of 17°.
Source: AMEC 2013.

16.4.7 BACKFILL

Paste backfill will support the mined out drift-and-fill areas underground. A paste plant (described in Section 18.3) will be constructed on site that will feed a 150 mm (6″) schedule 80 paste line. The paste line will extend from the surface plant to the underground production areas. The paste line will be installed in the existing exploration decline in Year 1. All new primary development starting in Year 1 will have the paste line installed as part of the development cycle. The paste fill delivery network will expand as mine development progresses.

16.4.8 PROJECTED POWER SUPPLY REQUIREMENTS

The underground mine power will be carried from the portal at 4,160 V.

The 4,160 V power will be delivered down the exploration decline via 5 kV armoured cable to the main 5 kV substation. The main substation will be located in an underground excavation off the exploration decline, approximately 1 km from the portal. At the main substation, 4,160 V sub-feeders will carry power to unit substations/load centres for conversion to 480 V for low voltage motor control and other loads.

Unit substations will consist of three 500 kVA, 4.16 kV-480/277 V skid-mounted units to serve the packaged dewatering pump skids and other nearby loads. Nine 1,000 kVA, 4.16 kV-480/277 V substations will serve the 480 V loads in the production and development areas, such as auxiliary fans, drills, and sump pumps. Four additional 4.16 kV-480/277 V substations will serve the underground main fan loads.

The two secondary egress hoists will be powered by two 500 KVA, 4.16 kV-480/277 V surface unit substations. Allowances have been made for the temporary loads required by the raise-bore machine during the development phase. For hoist back-up power, a 100 kVA, 480/277 V diesel generator will be located adjacent to each hoist unit substation. In the event of a power outage, the generator will automatically start and pick up the load (hoist only) via an automatic transfer switch.

The maximum estimated operating electrical load for the Johnny Lee deposit is approximately 4 MVA.

16.4.9 PROJECTED MINE WATER SUPPLY REQUIREMENTS

It is estimated that a water supply of approximately 1,200 m^3/d will be required for the mine, which accounts for:

- mobile equipment fleet water usage – 1,075 m^3/d
- underground delineation drilling – 10 m^3/d
- dust suppression system – 200 m^3/d.

To supply the required quantities of water to the working areas, a 3 in water supply line has been included in the primary development.

16.4.10 MINE DISCHARGE REQUIREMENTS

A detailed water balance for the mine could not be completed at this stage due to the absence of detailed hydrological information.

An allowance has been made in the mine plan to discharge 136 m^3/h (500 gpm) at the portal utilizing an underground pumping system. This estimate is for preliminary cost estimating purposes only and will require hydrogeological studies in the next phase of the Project.

The planned dewatering system consists of three 100 hp skid-mounted pump stations. Each pump skid includes a 30,000 L receiving tank and two centrifugal pumps connected in series. Each pump skid also contains two back-up end suction pumps. Each pump skid system delivers the design head of 216 m and a flow rate of 1,893 L/min (500 gpm).

The first of the three pump-skids will be installed permanently in Q1 of Year 1 in an excavation near the bottom of the exploration decline at the 1650 elevation. The second pump skid will be installed in Year 2 at the 1490 elevation in an excavation off the main decline about mid-way between the UZ and LZ. The third pump skid will be installed in Year 3 at the 1310 elevation in an excavation on the main decline in the LZ.

The discharge water will be staged out of the mine. Pump skid no. 3 will feed pump skid no. 2, which in turn will feed pump skid no. 1. Pump skid no. 1 will then pump the water to the portal (1790 elevation). The total distance from the lowest pump skid to the portal is 3,500 m.

16.4.11 Underground Fire Protection

Fire protection for the underground mine will conform to the more stringent guidelines of either the Mine Safety and Health Administration (MSHA) regulations, company site regulations, or local fire codes where applicable.

16.4.12 Sewage Disposal

Three portable ablution facilities will be installed underground: two located within the UZ, and one within the LZ. Sewage regularly transported from underground would be treated in the surface plant. Other than to provide workforce size as a key input parameter, the design and cost of surface sewage facilities is by others.

16.4.13 Mine Communications

The underground mine will be serviced by a leaky feeder system. The leaky feeder cable will be hung in all the main underground areas such as the decline ramps, footwall drifts, and in the vicinity of permanent infrastructure. The system will be serviced by a head-end unit located on surface. As part of this system, portable and stationary radios will be used for personnel communication. Additional communication systems that could potentially benefit the Project should be investigated in further study; for example, fiber optic systems, personal emergency device (PED) systems, closed-circuit televisions (CCTV), etc.

17.0 RECOVERY METHODS

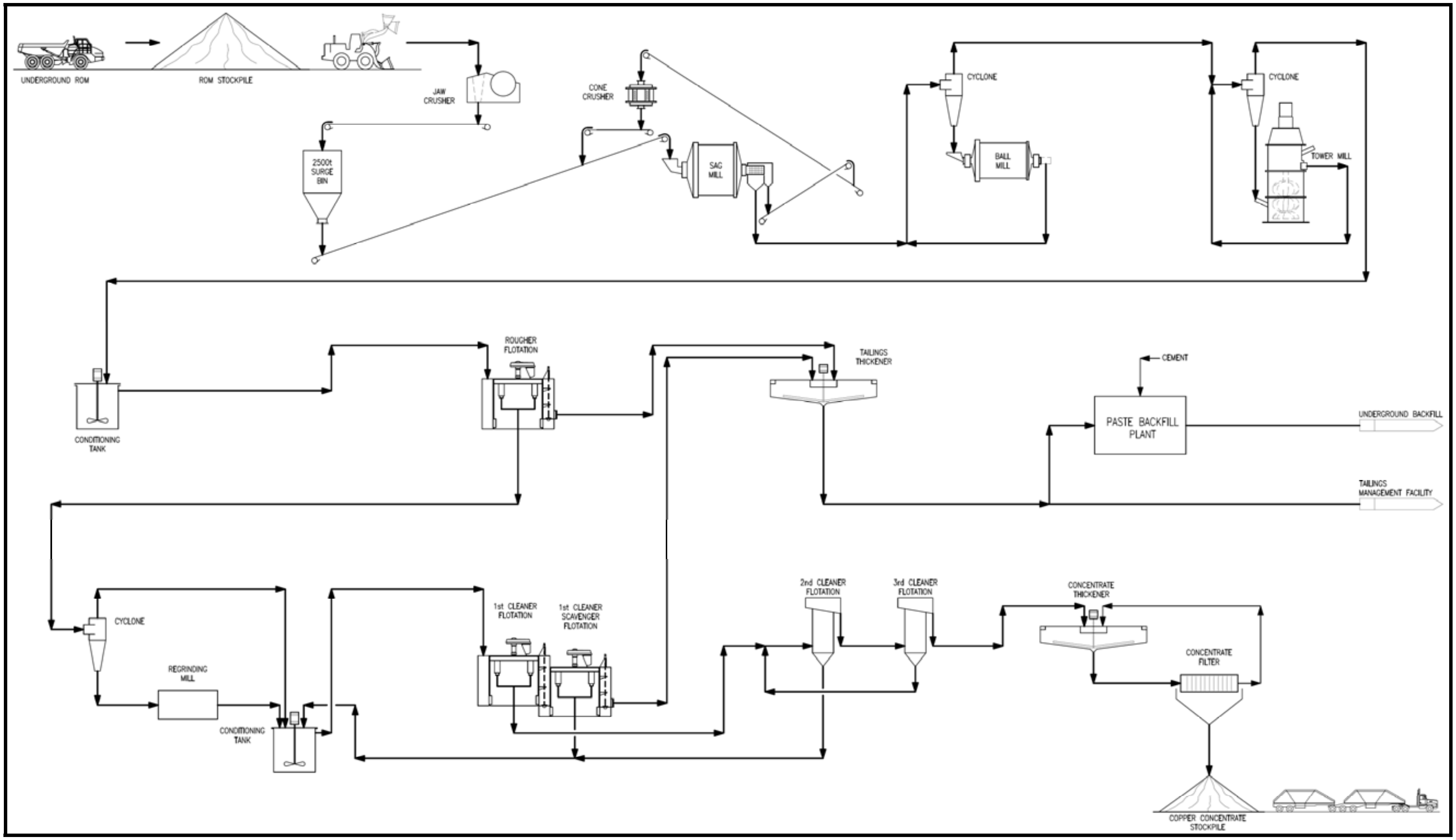
17.1 GENERAL DESCRIPTION

Tetra Tech designed a 3,300 t/d process plant for the Project to process massive sulphide mineralization containing copper and associated cobalt and silver. The process plant will operate in two 12-hour shifts per day, 365 d/a; the plant will process mineralized material at an annual rate of 1,204,500 t. The proposed availability is 70% for the crushing plant and 92% for the grinding and flotation plant.

The mill feed will be crushed by a jaw crusher to 80% passing 125 mm, and then ground to 80% passing 38 µm in a SAG/ball mill/tower mill primary grinding circuit. The ground material will be processed using copper rougher flotation followed by copper rougher concentrate regrinding; the reground copper rougher flotation concentrate will then be upgraded by three stages of cleaner flotation. Copper rougher flotation tailings, together with the copper cleaner scavenger flotation tailings, will be dewatered by thickening prior to being delivered to the backfilling plant or to the TMF. The third cleaner flotation concentrate, which will on average contain approximately 23.5% copper, will be thickened and then pressure-filtered before it is shipped to smelters. The LOM average copper recovery is estimated to be approximately 88.3%.

A simplified process flowsheet is provided in Figure 17.1.

Figure 17.1 Proposed Process Flowsheet



17.2 PROCESS DESIGN CRITERIA

Process design criteria were developed for the Project is based on a mill throughput of 3,300 t/d (1,204,500 t/a). Table 17.1 outlines the main process design criteria.

Table 17.1 Process Design Criteria

Description		Unit	Value
General			
Type Of Deposit		-	Massive Sulphide Mineralization
Mill Feed Characteristics			
Specific Gravity (Upper Zone)		-	3.81
Specific Gravity (Lower Zone)		-	3.52
Specific Gravity (Average)		-	3.75
Moisture Content		%	4.0
Bond Ball Mill Work index		kWh/t	13.6
Bond Rod Mill Work index		kWh/t	17.1
Abrasion Index (Average)		-	0.6885
Operating Schedule			
Shift/Day		-	2
Hours/Shift		h	12
Hours/Day		h	24
Days/Year		D	365
Plant Availability/Utilization			
Overall Plant Feed		t/a	1,204,500
Overall Plant Feed		t/d	3,300
Crusher Plant Availability		%	70.0
Grinding and Flotation Plant Availability		%	92.0
Crushing Rate		t/h	196
Grinding Rate		t/h	149
Flotation Rate		t/h	149
Design Factor	Grinding/Rougher Flotation	-	1.15
	Regrinding/Cleaner Flotation	-	1.25
Head Grades (Average)		Cu %	3.11
Copper Recovery (Average)		%	88.3
Copper Concentrate Grade (Average)		Cu %	23.5
Copper Concentrate Mass Recovery (Average)		%	11.7
Copper Concentrate Production (Average)		t/a	140.800

17.3 PROCESS DESCRIPTION

17.3.1 CRUSHING OPERATIONS

Haul trucks will transport the run-of-mine (ROM) material from the underground mine to the primary crushing area. The ROM material will feed a stationary grizzly with 550 mm by 450 mm openings at an average feed rate of 196 t/h. The grizzly will screen out of the oversize which will be reduced by a rock breaker. The grizzly undersize will discharge into a dump hopper and feed to a vibrating grizzly feeder with 100 mm openings. The grizzly feeder oversize will be directed to a 760 by 1,000 mm primary jaw crusher driven by a 110 kW motor. The jaw crusher will crush the oversize material to approximately 80% passing 125 mm. The grizzly screen undersize will join with the jaw crusher discharge and be transported to a surge bin with a live capacity of 2,500 t via a 700 mm wide feed belt conveyor.

A fogging system will be installed to minimize fugitive dust emissions.

17.3.2 GRINDING CIRCUIT OPERATION

A SAG/ball mill/tower mill primary grinding circuit consisting of one SAG mill, one ball mill, one tower mill, and one pebble crusher is designed to grind the crushed mill feed to a particle size of 80% passing 38 µm, which is required for effective liberation of the copper minerals from other minerals in the mineralization, especially for the upper zone mineralization. Tetra Tech has proposed three stages of grinding to achieve the grind size.

PRIMARY GRINDING – SAG MILL GRINDING CIRCUIT

The crushed mineralization will be reclaimed from the surge bin at a controlled rate of 149 t/h, using two belt feeders, and then fed via a conveyor to a 6,100 mm by 2,740 mm effective grinding length (EGL) (20 ft by 9 ft) SAG mill, powered by a 1,450 kW drive motor. Process water will be added to make the slurry solid density in the SAG mill to 75% solids. The SAG discharge will feed onto the trommel screen attached to the SAG mill. The screen oversize will then return to the SAG mill feed conveyor after being crushed by a cone crusher with an installed power of 90 kW. The screen undersize will report to the ball mill grinding circuit.

The proposed transfer particle size between the SAG mill grinding and the downstream ball mill grinding will be 80% passing 850 µm. Lime will be added to the SAG mill to maintain a pulp pH at approximately 10.0 to suppress pyrite.

SECONDARY GRINDING – BALL MILL GRINDING CIRCUIT

The SAG mill screen undersize, together with the primary ball mill discharge, will discharge into the hydrocyclone feed pump box in the primary ball mill grinding circuit. The combined slurry will be pumped to the hydrocyclone cluster consisting of three 600 mm diameter hydrocyclones. The hydrocyclone underflow, with a solid density of approximately 70 to 75%, will feed to a 4,115 mm diameter by 6,700 mm long (13.5 ft by 20 ft) ball mill powered by a 1,450 kW drive motor. The circulation load for this circuit will be approximately 300%. The hydrocyclone overflow will be directed to the hydrocyclone feed pump box in the tower mill grinding circuit.

Tertiary Grinding – Tower Mill Grinding Circuit

The classification in the secondary ball grinding circuit will consist of 11, 380-mm hydrocyclones. The hydrocyclone underflow with a solid density of approximately 65 to 70% will feed to a mill with an installed power of 1,100 kW. The proposed circulation load is approximately 200%. The hydrocyclone overflow with a particle size of 80% passing 38 μm will be directed to copper rougher flotation circuit.

The grinding mills have been sized according to a Bond ball mill work index (BWI) of 13.60 kWh/t at an operating availability of 92%.

Steel balls will be used as the grinding media in the grinding circuit. The steel balls will be added as necessary to maintain sufficient steel load for optimum grinding efficiency. The solid density in the grinding system will be controlled by maintaining process water addition.

17.3.3 Copper Rougher Flotation Circuit

The hydrocyclone overflow (at 28% solids) from the tower mill grinding circuit will flow by gravity into the rougher flotation circuit, consisting of seven 50 m^3 flotation tank cells. The slurry pH at the rougher flotation will be maintained at 10.0 to 10.5 with addition of lime. The rougher concentrate produced from rougher flotation will be advanced to the rougher concentrate regrind circuit. The rougher flotation tailings, together with the copper cleaner scavenger flotation tailings, will be pumped to the tailings thickener feed well. The thickener overflow will be recycled as process water while the thickener underflow will be sent to the paste backfill plant or the TMF.

Reagents used in the circuit will include lime as a pH conditioner to suppress pyrite, SIPX, and 3418A as collectors, and methyl isobutyl carbinol (MIBC) as a frother.

17.3.4 Copper Rougher Concentrate Regrind and Copper Cleaner Flotation Circuit

The copper rougher concentrate will be pumped into a hydrocyclone cluster with 21, 100-mm hydrocyclones for classification. The hydrocyclone underflow will then be reground in two stirred mills with an installed power of 1,100 kW in an open circuit. The proposed regrind size is 80% passing 10 μm.

The regrinding mill discharge, together with the hydrocyclone overflow, will be conditioned with lime to depress pyrite. Lime will be added to adjust the pulp's pH up to 11.8. The conditioned pulp will then be fed to a bank of four 30 m^3 copper cleaner flotation cells. 3418A will be added to collect copper minerals, while MIBC will be used as a frother. The first copper cleaner flotation concentrate will be further cleaned in the second cleaner flotation with a 3,600 mm diameter by 8,000 mm height flotation column. The first copper cleaner tailings will be scavenged in one 30 m^3 flotation cell. The copper cleaner scavenger flotation concentrate with the second copper cleaner flotation tailings will return to the first copper cleaner flotation conditioning tank. The cleaner scavenger flotation tailings together with the copper rougher flotation tailings will be pumped to the tailings dewatering thickener.

The second copper cleaner flotation concentrate will be further upgraded in the third copper cleaner circuit by a 3,600 mm diameter by 8,000 mm height column. The third cleaner flotation concentrate will be the final copper concentrate, which will report to the copper concentrate thickener. The second copper cleaner flotation tailings will return to the first cleaner flotation conditioning tank. The third copper cleaner tailings will return to the preceding cleaner flotation column feed pump box.

The collector and the frother added in the two cleaner flotation circuits are 3418A, and MIBC. Lime will be added to suppress pyrite. The cleaner flotation will be carried out at pH approximately 11 to 11.8.

17.3.5 Copper Concentrate Dewatering

The third copper cleaner flotation concentrate will be thickened and further dewatered to a moisture content of 10% by a pressure filter. The dewatered copper concentrate will be stored in the concentrate storage building prior to being shipped to overseas smelters.

The final copper flotation concentrate will be pumped from the third copper cleaner flotation concentrate receiving standpipe to the thickener feed well where the copper concentrate slurry will be mixed with flocculant solution. The thickener proposed is a 10,000 mm diameter high-rate thickener. The thickener underflow with a solid density of approximately 60% will be pumped to a 6,000 mm diameter by 6,500 mm high concentrate stock tank with a 10-hour storage capacity, prior to the pressure filtration. The thickener overflow will be sent to the process water tank for reuse in the grinding/flotation circuits.

The thickened copper concentrate slurry will further be dewatered by a 160 m^2 pressure filter to a moisture content of 10%. The filtration cake will be conveyed to the copper concentrate storage and load-out shed. A belt scale and a sampling system will be installed to acquire data for overall metallurgical accounting. Filtrate will return to the copper concentrate thickener.

17.3.6 Tailings Disposal

The copper rougher flotation tailings, together with the copper cleaner scavenger flotation tailings, will be directed to the feed well of a high-rate thickener with a diameter of 27,000 mm. The flow rate reporting to the thickener will be approximately 132 t/h. Flocculant will be added to improve settling of the tailings. Thickener underflow slurry with a solid density of 60% will be pumped to the paste backfill plant or to the TMF.

17.3.7 WATER SUPPLY

There will be two separate water supply systems: a fresh water supply system and a process water supply system.

FRESH WATER SUPPLY SYSTEM

Fresh water will be supplied to the Property from wells drilled on the mine site to supply the fresh and potable water for the Project. An 11,000 mm by 11,000 mm fresh water and fire water storage tank will hold operating fresh water prior to distribution within the plant. Fresh water will mainly service the following areas:

- fire water – fresh water will be distributed for emergency purposes and electrical, diesel, and jockey pumps will be connected to the fresh tank
- gland and seal water – fresh water will be pumped to various slurry pumps via the fresh water distribution piping system
- mill lubrication cooling water – lubrication cooling water will be supplied from the fresh water storage tank
- potable water – fresh water from the line to the fresh water tank will be chlorinated and filtered as potable water. The treated water will be stored in a separate 3,000 mm diameter by 4,000 mm high holding tank prior to distribution.

PROCESS WATER

Process water comprise fresh water, the water reclaimed from the TMF, the overflows of the flotation tailings and the copper concentrate thickeners, and water from the underground mine. The reclaimed water and fresh water will be directed to a 12,500 m diameter by 14,000 mm high process water tank, from which the water will be distributed to the process plant and other service locations.

17.3.8 AIR SERVICE

Two separate air supply systems will service the process plant. Low-pressure air for the flotation cells will be supplied by air blowers. High-pressure air for the overall process plant will be supplied by plant air compressors.

Instrumentation service air will be provided from plant air compressors. Compressed air will be dried and stored in air receivers for distribution to various instruments. Filtration air will also be provided from plant air compressors.

17.3.9 QUALITY CONTROL

The final concentrate and intermediate streams will be monitored by an on-line x-ray diffraction analyzer, which will include pH control and reagent addition control systems. The assay data will be fed back to central control room and used to optimize process conditions. Routine samples of intermediate products and final products will be collected and analyzed in an assay laboratory where standard assays will be performed. The data obtained will be used for product quality control and routine process optimization. Feed and tailings samples will also be collected and subjected to routine assay.

The assay laboratory will consist of a full set of assay instruments for base metal analysis, including an atomic absorption spectrophotometer (AAS), an inductively coupled plasma (ICP), experimental balances, and other determination instruments such as pH and redox potential metres.

Metallurgical performance and flowsheet optimization tests will be conducted in an on-site metallurgical laboratory. The laboratory will be equipped with laboratory crushers, ball mills, particle size analysis devices, laboratory flotation cells, balances, and pH meters.

17.4 Process Control Philosophy

The plant control system will consist of a distributed control system (DCS) with PC-based operator interface stations (OIS) located at the central control room. The DCS, in conjunction with the OIS, will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generation. The plant central control room will be staffed by trained personnel 24 h/d.

The central room will also control and monitor the primary crushing facility and the coarse material reclaim from the coarse material surge bin.

The process control will be enhanced with the installation of an automatic sampling system. The system will collect samples from various streams for on-line analysis and the daily metallurgical balance. Vendors' instrumentation packages will be integrated with the central control system.

A closed-circuit television (CCTV) system will monitor various facilities and conveyor discharge points. The cameras will be monitored from the central control room.

17.5 Metallurgical Performance Projection

According to the metallurgical performance projections developed from the metallurgical test results and the proposed mine plan, annual concentrate production is projected and shown in Table 17.2.

Table 17.2 Annual Concentrate Production Projection

Year	Mill Feed		Copper Concentrate		
	Tonnage (t/a)	Grade (Cu %)	Recovery (Cu %)	Grade (Cu %)	Production (t/a)
1	575,000	2.84	84.3	21.7	63,400
2	1,147,500	3.31	88.2	23.2	144,400
3	1,204,500	3.39	89.0	23.6	154,000
4	1,204,500	3.21	88.8	23.7	144,800
5	1,204,500	3.20	88.8	23.7	144,500
6	1,204,500	3.18	88.6	23.6	144,000
7	1,204,500	3.12	88.6	23.6	141,000
8	1,204,500	3.03	88.4	23.7	136,500
9	1,166,500	2.87	87.7	23.5	125,300
10	1,110,800	2.79	87.0	23.2	116,100
11	616,900	3.07	89.0	24.0	70,300
Total	11,843,700	3.11	88.3	23.5	1,384,300

18.0 INFRASTRUCTURE

18.1 INTRODUCTION

The Property is located in Meagher County, Montana, US, about 17 miles north of the town of White Sulphur Springs. The Property is accessed by 1.5 miles of well-maintained county graveled road which branches off from US Highway 89, an all-weather state-maintained highway (Figure 18.1). Figure 18.2 illustrates the overall Project site layout.

18.2 ROADS

The Project will be accessed by an existing county road leading from US Highway 89. This access road will require minimal upgrading. A connecting network of roads between the laydown area, the TMFs, the portal, and the mining operations staging points will be constructed.

Single lane site roads are required to access the various ancillary facilities including the process plant site, auxiliary buildings and primary crushing building and the TMF.

18.3 BUILDINGS

Figure 18.3 illustrates the location of the site buildings.

Figure 18.1 Road Access to the Property

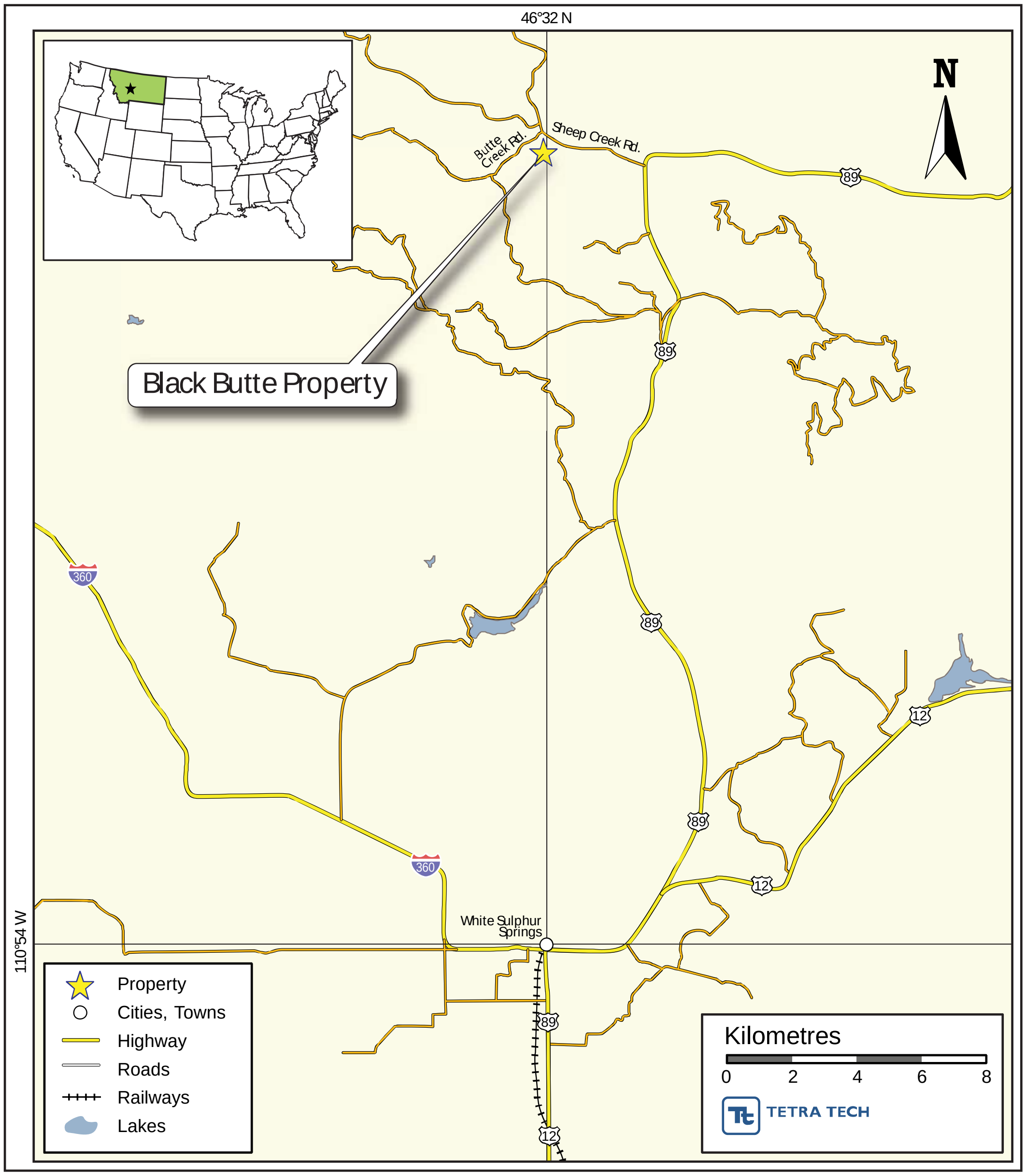


Figure 18.2 General Arrangement

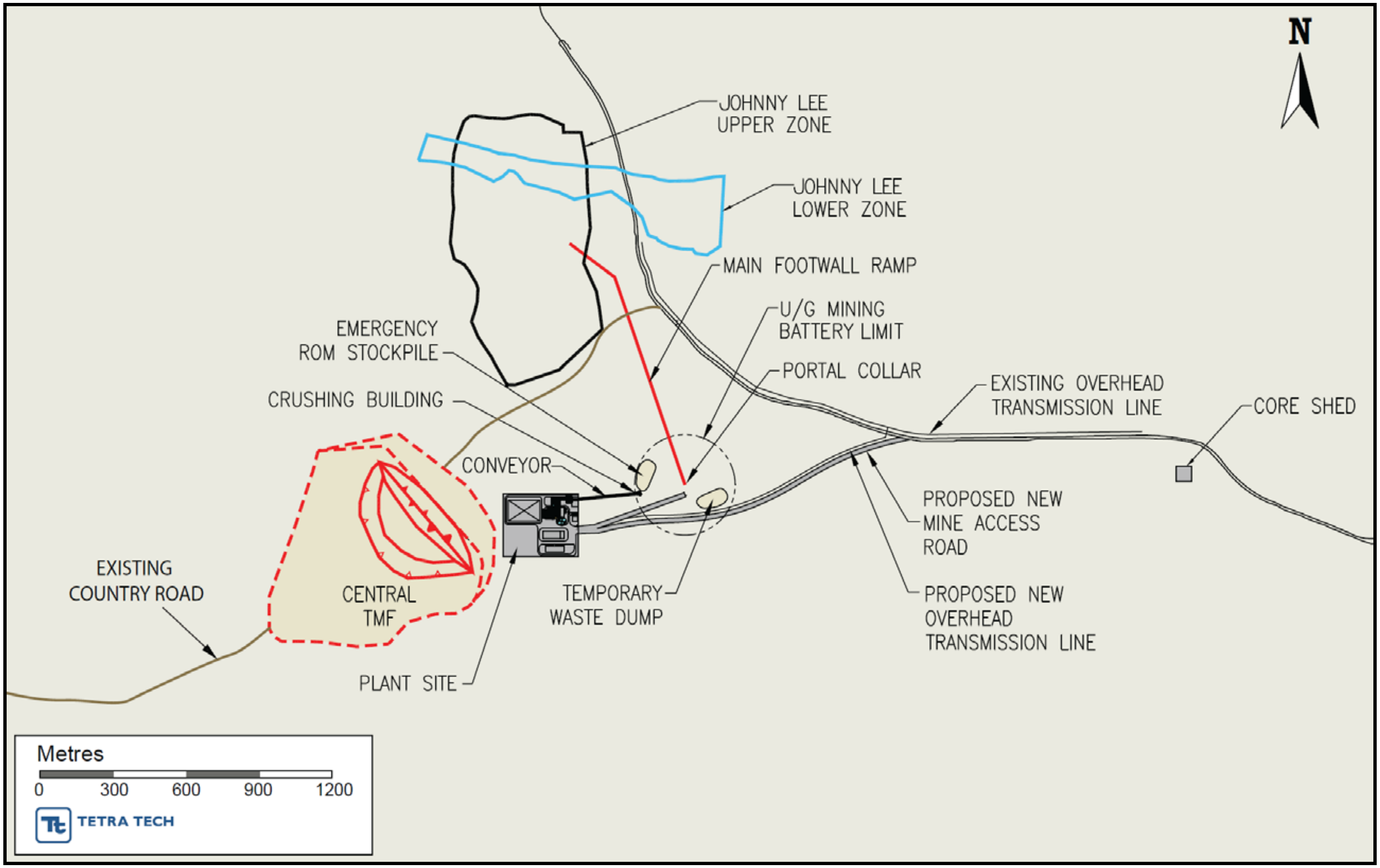
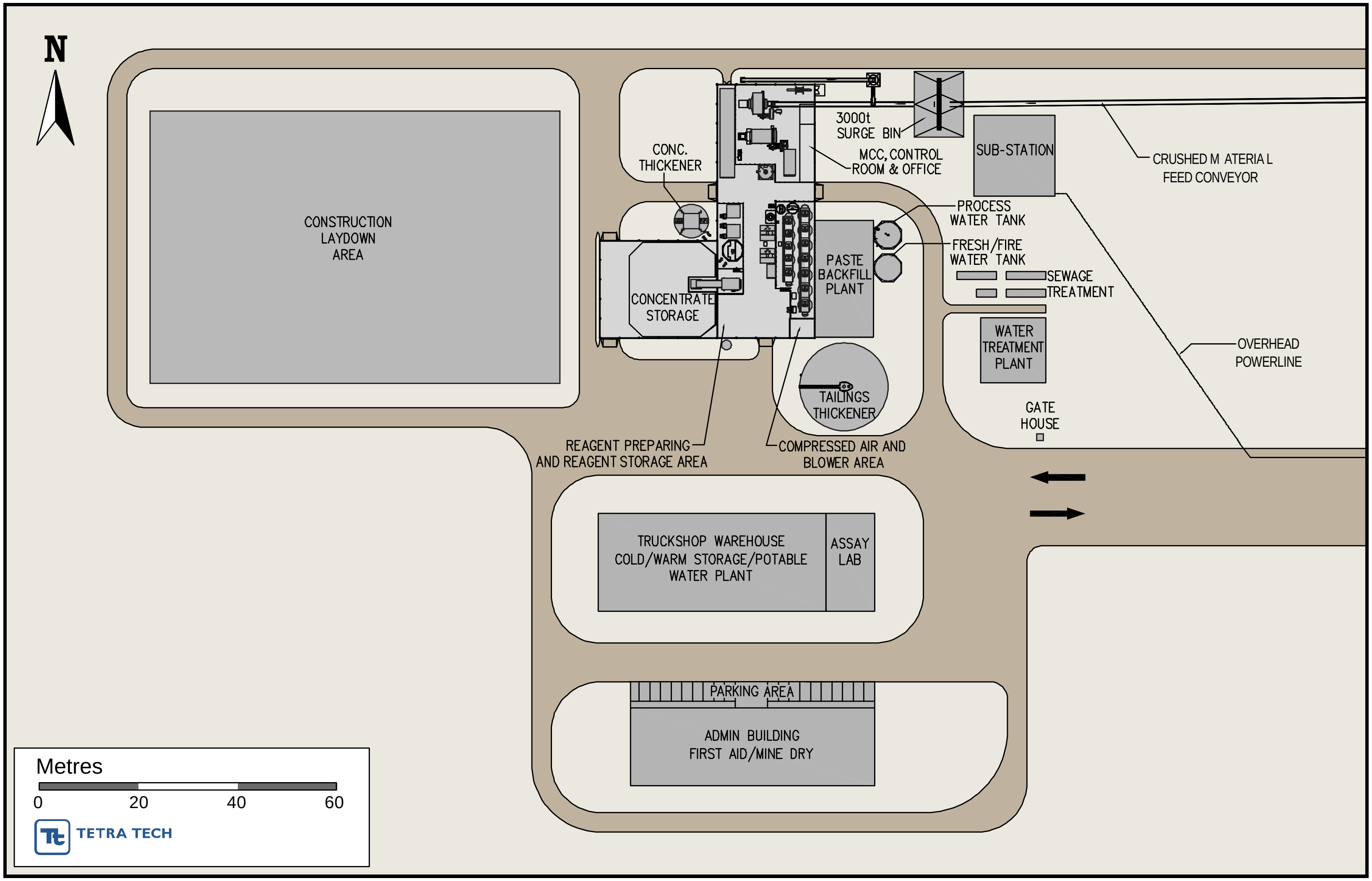


Figure 18.3 Plant and Ancillary Layout



18.3.1 Mill Building

The mill building will be a pre-engineered steel structure with insulated steel roof and walls. The building will have elevated steel platforms throughout for ongoing operations and maintenance. The building will house an overhead crane coverage clear-span. The building foundation will consist of concrete spread footings, grade walls along the building perimeters and a slab-on-grade floor. The floor surfaces will have localized areas that are sloped toward sumps for cleanup operations. Operations and maintenance activities will be staged in the designated laydown area.

The building will house the SAG mill, primary and secondary ball mills, rougher flotation and cleaner floation columns, regrind area, reagents area, concentrate surge tank, concentrate filter press and laydown areas. There is a mezzanine level above for the control room, offices and electrical room.

Adjacent to the building, alongside the north wall, are areas for the tailings thickener, concentrate thickener and water services.

Adjacent to the building, alongside the west wall, is the concentrate stockpile and loadout structure.

An optical fibre backbone is included throughout the plant in order to provide a path for the data requirements for voice, data, and control system communications. A fibre backbone for a site ethernet-type system is included, which will provide data and voice bandwidth.

18.3.2 Concentrate Building

The concentrate building is a stockpile and loadout facility with a full clear-span interior and will be a “sprung” structure constructed on top of concrete spead footings, grade walls along the building perimeters and a slab-on-grade floor. The building will be designed with insulation and “almost zero” air leakage envelope, to contain or limit all dispersement of concentrate dust. Modular, steel interior retaining walls provided for fleet load out vehicle to operate and manage the concentrate and loadout facility. Load out occurs at sliding cargo door end.

18.3.3 Administration Building

The administration building is a single-storey steel structure with insulated steel roof and walls located in close proximity to the process area. The building will be supported on concrete spread footings with concrete grade walls along its perimeter. This facility will house mine dry, lockers, shower facilities, first aid, with emergency vehicle parking and office areas for the administrative, engineering, and geology staff.

18.3.4 MAINTENANCE/TRUCK SHOP AND WAREHOUSE COLD/WARM STORAGE

The facility is a pre-engineered steel structure with insulated roof and walls, and limited interior support steel structures. The building will be supported on concrete spead footings and concrete grade walls along its perimeters. Sumps and trenches will be constructed to collect wastewater in the maintenance bays. Floor hardener will be applied to concrete surfaces in high-traffic areas.

The facility will house a wash bay complete with repair bays, parts storage area, welding area, machine shop, electrical room, mechanical room, compressor room, lube storage room. The facility will also house the cold/warm storage warehouse and areas to support warehouse and maintenance personnel.

The facility is designed to service and maintain both the mining haul fleet and the process plant fleet.

18.3.5 FUEL STORAGE

Diesel fuel requirements for the mining equipment, and process and ancillary facilities will be supplied from above-ground diesel fuel storage tanks located near the truck shop. The diesel fuel storage tank will have a capacity sufficient for approximately three days of operation. Diesel storage will consist of above-ground tanks and a containment pad, complete with loading and dispensing equipment conforming to regulations. A fuel dedicated service truck will transport diesel to the mining equipment and the process plant fleet.

18.3.6 ASSAY LABORATORY

The assay laboratory is a single-storey modular building. The building foundation will consist of concrete spread footings. The facility will house the assay and metallurgical laboratory required for all necessary laboratory equipment for metallurgical grade testing and control, and will be equipped with all appropriate HVAC and chemical disposal equipment as needed. The facility floor will be reinforced as needed to accommodate specialized equipment.

18.3.7 CONVEYING

Conveyors are to be vendor supplied including all structural support frames, trusses, bents, and take-up structures.

Overland conveyors are supported on concrete pre-cast panels spaced at regular intervals. Elevated conveyors are supported with vendor supplied steel trusses and bents on concrete foundations.

18.3.8 PRIMARY CRUSHING (JAW) BUILDING

The primary crushing building will be of concrete construction, with multiple levels housing the ROM mineralized material feed hopper and feed hopper grizzly, the vibrating feed grizzly, jaw crusher, the primary apron feeder and the crushed material surge bin feed belt conveyor.

The structure will be earth retaining on three sides, stick-built and enclosed up to the dump pocket. ROM mineralized material will be discharged into the dump pocket at the top. The structure will be supported on concrete spead footings and concrete grade walls along its perimeters. Interior steel platforms will be provided to support equipment for ongoing opertions and maintenance. There will be no control room adjacent to the dump pocket. There will be no rockbreaker adjacent to the dump pocket. The area will be equipped with a dust control system to control fugitive dust, and a crushing area overhead crane.

The primary crushing building is not within the boundaries of the process plant site, instead it is located within the battery limits of the portal approximately 100 m from the east boundary of the process plant site.

18.3.9 CRUSHED MATERIAL SURGE BIN

The crushed material surge bin is a production surge facility which will allow for a steady feed of fine mineralized material to the sag mill feed circuit. The facility will be an engineered post-and-beam steel structure connected to two adjacent, side-by-side bins, supported over a heavy concrete mat foundation. The surge bins will have a combined live capacity of 2,500 t. It will be fed crushed mineralized material by the crushed material surge bin feed belt conveyor and onto the surge bin feed belt feeder split to either bin. Two reclaim belt feeders will feed onto the belt feed sag mill feeder. The area will be equipped with feed weight scale.

18.3.10 SECONDARY CRUSHING (PEBBLE) BUILDING

The secondary crushing building will be an engineered post-and-beam steel structure with an insulated steel roof and walls, and, multiple interior platform levels housing the pebble feed surge bin, pebble crusher belt feeder, pebble crusher, and, pebble discharge belt. The building will be supported on concrete spead footings and concrete grade walls along its perimeters. The area will be equipped with conveyor belt self-magnets, metal detector, and lube unit.

18.3.11 WATER TREATMENT PLANT BUILDING

The water treatment plant is a modular building. The building foundation will consist of concrete spread footings and concrete grade walls along its perimeter. The facility will be equipped with all appropriate equipment as needed. The facility floor will be reinforced as needed to accommodate specialized equipment.

18.3.12 HVAC and Fire Protection

The cost for HVAC systems in ancillary buildings (based on costs per square metre) has been calculated from in-house data based on building function and site-specific climatic conditions.

Building heating and cooling loads were estimated based upon experience of similar projects in similar climates. Quantities for HVAC equipment (fans, heaters, air conditioning units, air handling units, etc.) were selected based upon the estimated heating and cooling loads for each building.

Fire protection is included based on information from recent similar projects.

All process areas will be heated to a minimum temperature of 5°C on a design winter day. This will be achieved by providing multiple propane-fired heating units along the perimeter walls and above all doorways. All process areas will be ventilated year-round to prevent a build-up of contaminants and humidity.

All occupied areas, such as offices, first aid, washrooms and change rooms, will be heated to a minimum temperature of 20°C on a design winter day. This will be achieved by supplying filtered and tempered outdoor air mixed with return air. The air will be distributed through ductwork into the individual rooms.

Air conditioning will be limited to control rooms, laboratories, and those electrical rooms where heat gains from electrical equipment are excessive. Electrical rooms where heat gains are not significant will be cooled using filtered outdoor air.

Small rooms, electrical rooms and remote buildings will be heated using electric heat.

Washrooms, change rooms and janitorial rooms will be mechanically exhausted to atmosphere. Make-up air will either be transferred from adjacent areas or supplied as filtered, tempered outdoor air.

18.3.13 Plumbing

All plumbing fixtures will be hard-piped by gravity to a sanitary drainage system.

All sinks and showers will be hard-piped with both potable hot and potable cold water.

Water will be heated in hot water storage tanks near the end users. Heating will be by propane or electricity.

All fixtures connected to the sanitary system will be vented.

All cold-water piping will be insulated to prevent condensation, and all hot water piping will be insulated for heat conservation.

Oil separators will be provided in truck shops and truck washes.

18.3.14 Fire Protection

A fire water tank will be built capable of sustaining two hours of firefighting at the design water flow rate. Firewater will be distributed around the site in valved loops, enabling water to flow in either direction.

Branches from the firewater distribution into each building will be provided with isolating valves.

The fire water system will be pressurized by a firewater pump packaget that consisting of a jockey pump, a main electric pump and a standby diesel-fired pump.

Yard hydrants will be positioned around the site such that all the buildings outside walls and all fuel tanks can be reached by a 30 m hose and a 15 m hose stream.

Sprinkler systems will be provided in lube rooms, air compressor rooms, blower rooms, truck shops, warehousers, laboratories, elevated mill offices, the mining equipment storage building and the administration building. Sprinklers will also be used to protect conveyors located in enclosed areas.

Fire hose stations will be provided in any building taller than 14 m, and will be located such that all areas of each building are within reach of a 30 m hose and a 15 m hose stream.

18.3.15 Dust Control

Dust control systems will be provided at the primary crushing apron feeder.

The dust collection equipment will consist of dry baghouse and the collected fines will be returned to the process stream.

The dust will be pneumatically conveyed from the exhaust hood to the dust collector through steel ducting.

The dust ducting will include test ports, dampers and clean-outs.

18.4 Tailings Management Facility

The TMF will be a lined impoundment designed to store 5.92 Mt of tailings (50% of total tailings production) and up to 1.63 Mt of PAG waste over the LOM. The remainder of the tailings will be used as mine backfill. The embankment borrow material will be excavated from within the impoundment area; therefore the excavation will provide for increased tailings storage capacity as well as construction material to build the embankment. The impoundment will be lined with a 100 mil HDPE liner overlying a prepared low-permeability subgrade. The TMF will be constructed in two stages, to limit capital costs and provide the flexibility for variations in capacity requirements over the life of the mine. The first stage is designed to store tailings and PAG waste rock from the first four years of the mine life, with the second staged sized to store the remaining tailings and PAG waste rock over the remaining life of the mine.

The impoundment will have interior slopes of 3H:1V, to facilitate liner installation. The downstream slope of the final embankment will be constructed at a 2H:1V slope. The second stage excavation will be developed upslope of, and tied into, the starter impoundment. The materials excavated from the basin will be used to construct the tailings embankment raise.

18.5 SEEPAGE MANAGEMENT

Seepage collection and control measures will be required for all tailings storage and management facilities. Seepage collection measures are necessary to satisfy permit requirements and to ensure that the seepage water is collected and treated for re-use or disposal.

An under-drain system will be placed above an HDPE basin liner in order to promote consolidation of the tailings and reduce seepage gradients. Collected water will be recycled for mill process water, with excess water being treated for disposal. The under-drain system will be designed to preclude air entry (to prevent oxidation of the tailings) and will be decommissioned after closure.

18.6 INSTRUMENTATION INSTALLATION AND MONITORING

The performance of the TMF will be monitored through the use of several methods, including vibrating wire piezometers, groundwater monitoring wells and drain monitoring sumps.

18.7 TAILINGS DELIVERY SYSTEM

The preliminary design tailings delivery system is based on the estimated plant site elevation, solids content of the tailings, and grade of the proposed pipeline route.

Tailings will be delivered to the impoundment using a 4” diameter Schedule 40 steel pipeline. The pipeline will follow a secondary access road from the plant and will be positioned to deposit tailings along the embankment and partially around the perimeter of the impoundment. Tailings pipelines will not be placed near the reclaim barge so that tailings deposition does not interfere with reclaim water collection. The second stage of construction will include a tailings pipeline extension.

18.8 Reclaim Water System

Reclaim water for use in the mill processes will be pumped from a floating barge to a reclaim head tank at the crest of the hill located southwest of the plant site. This head tank will store a 24-hour supply of mill process water, which will be gravity fed to the plant site. The water will be pumped to the head tank using a 6" diameter DR 17 HDPE pipe. The barge will be positioned at the south end of the pond to minimize the pumping distance to the head tank.

18.9 Waste Rock Storage Area

Waste rock will be produced mainly during the start of mining operations, when the portal to the deposit is excavated. It is estimated that approximately 1.63 Mt of waste rock will be produced during excavation of the mine adit and during mining. The amount of waste rock that will be PAG is currently unknown.

If the waste rock is deemed to be non-PAG, it will be placed in a waste rock storage area adjacent to the mine adit entrance and/or used for construction of the tailings embankment. The waste rock will be placed in a manner to conceal it from view from the public access roads as much as possible. The proximity to the entrance of the mine access tunnel will reduce waste hauling costs during production. The waste dump slopes will be constructed to a maximum slope of 2H:1V, to facilitate reclamation.

PAG rock will be deposited underground as mine backfill and/or co-disposed with the tailings in the TMF. The TMF will be designed to store 100% of the waste rock delivered to surface, based on the assumption that all the waste rock will be PAG.

18.10 Additional Water Management Facilities

A number of additional facilities have been identified for water management. The conceptual level design of these facilities has not yet been completed at this stage of development. However, an allowance for these items (including an allowance for cost) are included as they will need to be evaluated and incorporated into subsequent design studies.

18.10.1 Water Supply

Regional weather stations show that the predicted lake evaporation of the area exceeds the annual precipitation, which may create a surface water deficit during mining operations and post-closure. Additional water from surface sources or dewatering of the underground mine may be required to offset water loss from the TMF due to evaporation. A fresh water supply system may be required to provide potable water and any additional make-up water that may be required. This will need to be defined in subsequent design studies.

18.10.2 WATER TREATMENT PLANT AND DISPOSAL SYSTEM

It is possible that on-going dewatering of the mine may result in a water surplus, partuclarly during the latter stages of the mine life.

Therefore, it has been assumed that a water treatment plant may be required to treat excess mine water prior to disposal. The treatment plant is assumed to be required in the latter years of the mine life, when the mine water inflows would be expected to be greatest. Disposal of the treated water would be completed using spray evaporation within the tailings impoundment or by means of a land application and disposal system.

18.11 POWER DISTRIBUTION, ENERGY EFFICIENCY, AND UTILIZATION

18.11.1 POWER/ELECTRICAL

The Black Butte Mine is estimated to have a load between 7 and 9 MW. There are two nearby transmission lines (data provided by Fergus Electric and Heberley & Assoc); a 69 kV line at White Sulphur Springs substation, approximately 23 miles from site, and a 100 kV line to the east, approximately 17 miles from site.

POWER SOURCE OPTIONS

Option 1 – Extend 69 kV Line from a Substation within White Sulphur Springs

For the estimated load, 69 kV is a good fit. The nearest service point for 69 kV is the substation at White Sulphur Springs, 23 miles to the south. Further investigation performed by the electric utility revealed that there is not adequate power available in this substation.

Option 2 – Obtain Power from 100 kV Transmission Line to the East

For the estimated load, 100 kV will provide very robust service. The preferred tap location is Kings Hill. A tap substation is required at Kings Hill to install transmission line protection devices. The tap substation is estimated to cost US$950,000. The 100 kV transmission line would then extend from the tap substation, east along US Highway 89, to the Black Butte mine location approximately 17 miles away. The cost of the transmission line is estimated at US$3.1 million. The total cost including tap substation, is US$4.05 million.

The 100 kV line routing may run through areas controlled by the National Forest. This could make obtaining permits and right of way more difficult. Routing the line alongside the highway should simplify this permitting process.

ON-SITE ELECTRICAL DISTRIBUTION

The on-site electrical substation will be located as close as possible to the grinding/mill loads as these are the largest loads. Utility voltage will be stepped down to 13.8 kV for site wide power distribution.

18.12 Underground Mine Related Infrastructure Projections

The planned infrastructure for the underground mine includes the main access-ways and ventilation raises, as well as the following:

- eight main fans installed in four different underground locations
- two mine air heaters installed in two surface locations
- two emergency hoist systems in two surface locations
- a mine discharge system that includes three pump skids
- a main underground substation and power distribution system
- a leaky-feeder communications system.

18.13 Proposed Paste Backfill Plant

A scoping-level plan for the paste plant has been included as part of the processing facilities that will provide backfill for the underground mine. Currently, there is no test work available on tailings filtering, thickening, or paste backfill strength; therefore, AMEC has based the scoping study on typical parameters developed for other paste backfill projects that have proceeded to higher levels of study, detailed design, or construction. The paste plant has been designed with a backfill capacity of 2,030 t/d.

18.13.1 Process Description

The paste backfill plant will be constructed as part of the mill concentrator building. The mill concentrator will supply services such as compressed air, instrument air, gland water, process water, building heating, fire protection, etc. Electrical supply will be common to the other equipment in the mill concentrator; therefore, separate transformers, motor control centre (MCC) rooms and control systems will not be needed.

The paste backfill plant will be fed from the tailings thickener and underflow pumps in the proposed mill concentrator. Flocculent will be added to the thickener feed well. It is assumed that this thickener underflow feed at 59% solids will be ready for filtration to make paste backfill.

Two rotary disc filters will be used to dewater the thickener underflow slurry. The disc filters will dewater the thickener underflow to a solids concentration of approximately 85%. The cake from the disc filters will fall onto a conveyor belt that feeds the paste mixer. The disc filters will generate a filtrate that will flow to the waste water return pumpbox.

The disc filters will make a filter cake that will be delivered via a belt conveyor to the front of a twin screw paste mixer. A weightometer below the conveyor belt will provide a real-time measure of the mass flow to the paste mixer.

Cement and fly ash will be used as binders in the cemented paste backfill (CPB) product. A silo for cement and a silo for fly ash will be located immediately outside the paste plant. It is assumed that the binder will consist of 50% fly ash and 50% Portland cement, and that adequate paste strength can be obtained using 7% binder. Individual rotary valves and screw conveyors will be used to transport the cement and fly ash to the twin screw paste mixer.

The paste mixer will mix the combination of tailings filter cake, cement, and fly ash with process water to form a paste slurry measuring 75% solids content. The paste leaves the end of the paste mixer and drops into the suction side of a positive displacement paste pump.The paste pump will pump the mixed paste of tailings, fly ash, and cement to the collar of the vertical borehole and into the underground distribution system.

In the event of a power outage, the paste plant will be equipped with an emergency head tank of process water that will discharge by gravity to the borehole to provide a full volume flush of the paste borehole. Compressed air contained within the concentrator piping system will be used to assist the water flush of the paste borehole during a power outage.

The battery limit for the paste backfill plant is at the concentrator building wall. The underground distribution system is not included in the paste backfill plant estimate.

19.0 MARKET STUDIES AND CONTRACTS

There are no market studies or contracts material to the Project.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The Property is located on 4,700 acres (1,900 ha) of leased mineral rights on private ranch land, located in Meagher County, about 17 miles (24 km) north of the town of White Sulphur Springs, Montana. The Property abuts fragmented pieces of land west of the Property, whose surface is controlled by the USDA Forest Service, on the Lewis and Clark National Forest and other private owners. The Property can be accessed via gravel county and ranch roads located west of US Highway 89 (Figure 18.1).

The Project involves initial permitting of an exploration decline, from which underground development drilling will be conducted, followed by an application for a mine-operating permit from the State of Montana.

This section of the updated PEA describes:

- the mine permitting and environmental assessment (EA) process
- the environmental setting
- the current status of baseline studies.

20.2 PROJECT SETTING

The Project area lies to the east of the topographic feature called Black Butte in the headwaters of the Sheep Creek drainage. Sheep Creek is a tributary to the Smith River, which is in turn a tributary of the Missouri River.

The Property is accessible via 1.5 miles of well-maintained county graveled road, which branches west from US Highway 89, an all-weather state-maintained highway. US Highway 89 connects the Property with White Sulphur Springs, Montana. White Sulphur Springs is county seat for Meagher County and the largest town in the area with a population of approximately 984.

Sheep Creek is a minor tributary to the Smith River and drains a basin of approximately 194 square miles (NRIS 2011) within the Missouri river watershed. The Project area is located in the approximate upper third of the drainage. There are no gauging stations on Sheep Creek or its tributaries. The nearest gauging station is located on the Smith River just below its confluence with Sheep Creek. Base flows at this location range from approximately 90 ft^3/s to peak flows of approximately 1,500 ft^3/s (USGS Station

No. 06077200). The actual percentage of flow from Sheep Creek at this station on the Smith River is not known, but Sheep Creek accounts for approximately 19% of the surface drainage basin area above this location.

Butte Creek is another tributary to Sheep Creek located west of Black Butte. Sheep Creek and Butte Creek are high-quality streams that are used for irrigation, stock water, and fishing (RMI 2010).

The Project site ranges in elevation from approximately 5,600 ft along Sheep Creek to 6,800 ft atop Black Butte. Sub-irrigated low-lying hay meadows, and shrub-dominated wetlands and riparian areas occur along Sheep Creek and Little Sheep Creek. The topography of the remainder of the Project area consists of buttes, ridges, and valleys that form gently rolling hills. Timber cover consists primarily of Douglas fir on north facing slopes, hill tops, and grass and mountain sagebrush-covered valley floors and draws, which comprise approximately 40% of the resource area. Land uses are predominantly agricultural with hay and livestock production the primary activities. Outfitters also use the Sheep Creek drainage for big game hunting and fishing.

Sheep Creek supports livestock and irrigation as well as fisheries, and mine development on the Property must protect in-stream flow and water quality. The Project area has hosted only very minor historical prospecting and there are no significant environmental liabilities on the Property.

Precipitation data indicate an average annual liquid precipitation of about 13″ to 16″ with the annual snowfall between 37″ and 83″ depending on the station location and period of record (WRCC 2011). Annual evaporation rates for the Project area are believed to be between 35″ and 40″ per year.

20.3 MINE PERMITTING

The Project area lies entirely on privately owned ranch property. Tintina has a mineral lease agreement with the underlying Property owners who own the surface, mineral, and water rights. Because the Project is on private property and located in Montana, the Montana Department of Environmental Quality (DEQ) will be the sole agency responsible for permitting mining operations for the Project.

An operating permit is required for mining operations within the State of Montana, under the administrative supervision of the Montana DEQ's Permitting and Compliance Division's Hard Rock Mining Program. The required permit fee is US$500 and additional fees are typically required to cover DEQ's review of the application or the cost of a third party environmental review, depending upon the magnitude and complexity of the proposed action.

The State of Montana has various rules, regulations, and procedures that must be followed by a proponent attempting to acquire an operating permit for a mining project. The operating permit application consists of:

- an operating plan
- environmental baseline studies
- a reclamation plan.

20.3.1 Operating Plan

The operating plan is submitted by the proponent for review by the DEQ and specifies all major aspects of the mining process including: mine access, the type of mining and milling operations proposed, reagents and equipment to be used, planned mining and milling rates, types of liners to be used and installation procedures for mined wastes and tailings repositories, and the location of all support facilities and proposed surface disturbances related to mining.

20.3.2 Environmental Baseline Studies

Environmental baseline studies require performing research and/or collection of physical and chemical baseline data, including hydrology and hydrogeology, water quality, air quality, geology, soils, vegetation, wildlife (including aquatics and fisheries), and cultural features, and possibly others. The purpose is to characterize environmental baseline conditions at the site prior to construction or mining activities. Some of the baseline disciplines (i.e. hydrology and wildlife) may require up to two full years of seasonal baseline data to be collected before the state will deem the application complete and initiate EAs.

Tintina has met with DEQ's Hard Rock Mining Program personnel to discuss site-specific informational needs prior to initiating their baseline studies, and has retained the services of an interdisciplinary group of consultants with expertise in permitting projects in Montana. This will ensure that Tintina's plan meets DEQ requirements thereby reducing potential for unanticipated regulatory challenges or delays.

20.3.3 Reclamation Plan

A mine closure and reclamation plan is required to obtain a mine-operating permit under the Montana *Metal Mines Reclamation Act* (MMRA). The reclamation plan states reclamation goals and objectives, and describes how they would be implemented. The reclamation plan must consider site-specific conditions and circumstances, including the post-mining land use of the mine site. Disturbed lands must be reclaimed in a manner consistent with the requirements and standards set forth in MMRA.

The mine plan and reclamation program should be sufficiently detailed to assure DEQ reviewers that the proponent has the necessary understanding, resources, technical capability, and intent to develop the mine in a safe and environmentally sound manner, and to demonstrate that there are no major issues or concerns that have not been addressed or cannot be adequately mitigated. One of the more important issues for the Project will be the potential for acid rock drainage (ARD) and metal leaching to surface or groundwater, which requires considerable information to be gathered prior to the application stage.

The plan must provide details of reclamation activities, particularly those relating to control of erosion, and provide for construction of a graded, vegetative cover with landscaping and contouring that minimize the amount of precipitation infiltrating into disturbed areas. The re-established vegetative cover must also meet standards for noxious weed control. The plan must provide measures to prevent objectionable or non-compliant post mining groundwater discharges. It must also provide sufficient measures to ensure public safety and to prevent the pollution of air or water and the degradation of adjacent lands. Sufficient detail must be provided for DEQ to calculate a reclamation bond to adequately fund the entire closure plan.

20.3.4 MINE OPERATING PERMIT PROCESS

The DEQ's role during the permitting process is to issue timely and complete permit decisions for mining and reclamation of hard rock minerals, which ensure that mineral development occurs with adequate protection of environmental resources. DEQ also ensures that appropriate public involvement complies with the *Montana Environmental Policy Act* (MEPA) and other public notice and participation statutes.

The DEQ has 90 days to review the initial operating permit application submittal and determine its completeness for evaluation and its compliance with Montana environmental statutes. Following the determination that the operating plan is complete and compliant, preparation of an EA or EIS can commence under the MEPA. Environmental review under the MEPA is a public process that identifies the possible environmental impacts of the proposed project and requires agencies to describe those impacts to the decision maker, the project applicant, and the public. The MEPA review helps the state determine whether it can accommodate the statutory rights to development in a way that does not conflict with the public's constitutional and environmental rights.

Three types of written environmental reviews are possible under MEPA. These include:

- a checklist EA
- an EA
- an EIS.

An EIS will almost certainly be required for the Project based on the information reviewed to date. An EIS requires the agency to explain why it made particular decisions, what voluntary or enforceable mitigation efforts have been included in the decision, and what unavoidable environmental impacts may occur as a result of the decision. The types of resources or topics that may require investigation during the MEPA process include air, water, soils, geology, environmental geochemistry, vegetation and wildlife (including threatened and endangered species), cultural resources, Native American interests, noise, visual quality, land use, transportation, and socioeconomics.

Tintina has prepared and submitted an amendment to its exploration license to construct an exploration decline, for which an EA is currently (05/09/2013) being prepared by the Montana DEQ. Tintina has also initiated work on an operating permit application for submission to the Montana DEQ. To this end, they have initiated environmental baseline studies as discussed in Section 20.4.

20.3.5 OTHER LIKELY PERMIT REQUIREMENTS

MAJOR FACILITY SITING

The *Major Facility Siting Act* (MFSA) has been implemented by the State of Montana to provide a mechanism for the review of the construction of energy-related facilities such as power plants, power lines, pipelines, and geothermal facilities. The MFSA is deemed necessary to ensure that location, construction, and operation of facilities is in compliance with state law, and that a facility is not constructed or operated within Montana without a certificate of compliance. The MFSA is also designed to:

- ensure the protection of the state's environmental resources
- ensure the consideration of socioeconomic impacts
- provide citizens with an opportunity to participate in facility siting decisions
- establish a coordinated and efficient method for the processing of all authorizations required for regulated facilities.

If an MFSA permit is triggered, the proponent for a certificate under the Montana MFSA must file an application with the DEQ. Information concerning the need for the transmission line or pipeline, the proposed location, baseline data, and reasonable alternative locations must be included in the application. For transmission lines for a particular commercial facility, the application is usually applied for by the applicant in conjunction with the local power company.

SURFACE WATER DISCHARGE PERMIT

The Montana Pollutant Discharge Elimination System (MPDES) permit is issued by the Montana DEQ's Permitting and Compliance Division, and is required for all point-source discharges to State surface waters, regardless of any permits that are issued by other programs or agencies. Substantial application and maintenance fees are required for an MPDES permit. For those proposed discharges that are directly related to a hard rock or placer mining, or an exploration project, Hard Rock Mining Program hydrologists will assist the applicant in obtaining an MPDES permit from the DEQ. Requirements of the permit usually include pre-operational, operational, and post-operational water quality monitoring for specific parameters, depending on the specific site and proposed activity. These monitoring requirements can significantly extend the length of time to acquire an MPDES permit. The Project will almost certainly require an MPDES permit

GROUNDWATER DISCHARGE PERMIT

This Montana Groundwater Pollution Control System permit (MGWPCS permit) is issued by the Montana DEQ's Permitting and Compliance Division for discharges directly to groundwater, such as through a percolation pond or land application discharge (LAD) system. It is also required when the possibility exists of a discharge to groundwater from a “sealed” impoundment, such as a tailings pond or a heap leach pad/pond system. Substantial application and maintenance fees are required for a MGWPCS permit. An MGWPCS, or groundwater discharge permit, is required only if a hard rock or placer operator is proposing a discharge to groundwater and is operating entirely under a Small Miner Exclusion Statement (SMES). This separate permit is *not* required if the operator holds an operating permit or an exploration license. An operating permit or exploration license supersedes the requirement for a groundwater discharge permit because groundwater discharges permitted under an operating permit or exploration license would be subject to the same level of review and monitoring as those permitted under a separate groundwater permit.

MONTANA STREAMBED PRESERVATION ACT – 310 PERMIT

A 310 Permit is issued by the County Conservation Districts, in cooperation with the Montana Fish, Wildlife & Parks (FWP). It is only required for certain perennial streams, and is necessary when an applicant intends to ford a stream, install a culvert, or install a bridge. It is also required for stream alteration or diversion.

DREDGE/FILL – FEDERAL CLEAN WATER ACT – SECTION 404 PERMIT

A federal Section 404 Permit is issued by the US Army Corps of Engineers. This permit is required whenever an operator proposes to remove material from (dredge), or place material in (fill), in waters of the United States including wetlands.

AIR QUALITY PERMIT

An air quality permit is issued by the Montana DEQ's Permitting and Compliance Division under the authority of *Montana Air Quality Act*. It is required when emissions from a project are expected to exceed certain threshold values for various parameters. Generally, an Air Quality Permit is required if emissions of any pollutant, including fugitive dust, exceed 25 t/a. An annual fee is required, based upon a facility’s total emissions. In most cases, an air quality permit is only needed for larger developments (e.g. large open pit mines, or mines with a sizeable tailings impoundment, on-site large-scale power generation, or large-scale milling facilities, etc.).

WATER RIGHTS

Operators need to secure the necessary water rights/permits when using water in their processing or operation. One-time-only users, such as drillers who may need a limited amount of water in a water truck or pipe diversion, can generally take the water as long as consideration is given to downstream water users, and stream banks are not altered, or a sedimentation problem created. It is strongly recommended that an operator contact a local land owner and inquire about water sources.

Hard Rock Impact Act

Under the MMRA, prior to issuing an operating permit, the DEQ must first certify that an applicant is in compliance with the various requirements of the *Montana Hard Rock Impact Act* (HRIA). The HRIA only applies to large-scale hard rock and placer mineral developers that would employ over 75 employees.

If an operating permit applicant is proposing an operation that would employ over 75 people, the applicant must enter into negotiations with a local committee (near the proposed mine area) made up of local officials and individuals. The negotiations centre on the HRIA's requirements for the pre-payment of taxes by the applicant to mitigate socioeconomic impacts to the local area caused by an influx of people to work at the mine. Socioeconomic concerns usually include local school capacity, water and sewage infrastructure, road maintenance, and other related issues.

State Historic Preservation Act

The Montana SHPO works to preserve significant historic, archaeological, and cultural places as a resource to the people of Montana. In general, historic preservation determinations are made during a review of cultural resources of potentially disturbed ground within mining permit areas and are conducted by architectural historians, historic architects, and archaeologists. Features to be evaluated include those that are at least 50 years old.

20.4 Environmental Baseline Review

Baseline studies describe and evaluate baseline (existing) conditions at the Project site, prior to construction or operation of the proposed facility. The purpose of the studies is to collect information and physical data associated with resources that may be affected by construction and operation of the facility. This facilitates the evaluation of possible impacts and provides a benchmark against which future changes can be measured. The physical data are typically evaluated through comparison with state standards or guidelines.

An initial consultation with the DEQ is recommended in order to understand the DEQ's internal process for mine permitting as well as define the types of baseline information and data that the DEQ requires in order to evaluate the mine's potential to impact the area. The DEQ may collect and evaluate different types of media as part of a baseline study as summarized in Table 20.1.

Table 20.1 Potential Resources for Baseline Environmental Assessment/Study

Study	Resources
Surface Water	Wetlands
Groundwater	Vegetation
Rock/Sediment	Climate
Soil	Historical/Cultural
Fish and Wildlife	Geology and Topography

Site-specific environmental baseline studies were initiated by Tintina in 2010 following initial consultation with the Montana DEQ. These studies are designed to collect environmental baseline data for aquatic, terrestrial, and human resources. Some of this work was initiated to acquire baseline data for permitting of the exploration decline, so that the original study areas were limited in scope to the immediate area that would be influenced by the decline. However, the majority of studies (particularly those requiring a longer period of record for environmental permitting of the entire proposed mine facility, i.e. surface and groundwater studies and waste rock characterization) were designed and implemented to cover the full area of influence for the future mine. The following sections describe Tintina's current understanding of the Project area environment. An increased understanding of the existing environment will be obtained through ongoing baseline investigations and monitoring. A report describing these studies in detail is currently being prepared for release in mid-2013.

20.4.1 AIR RESOURCES AND WEATHER DATA

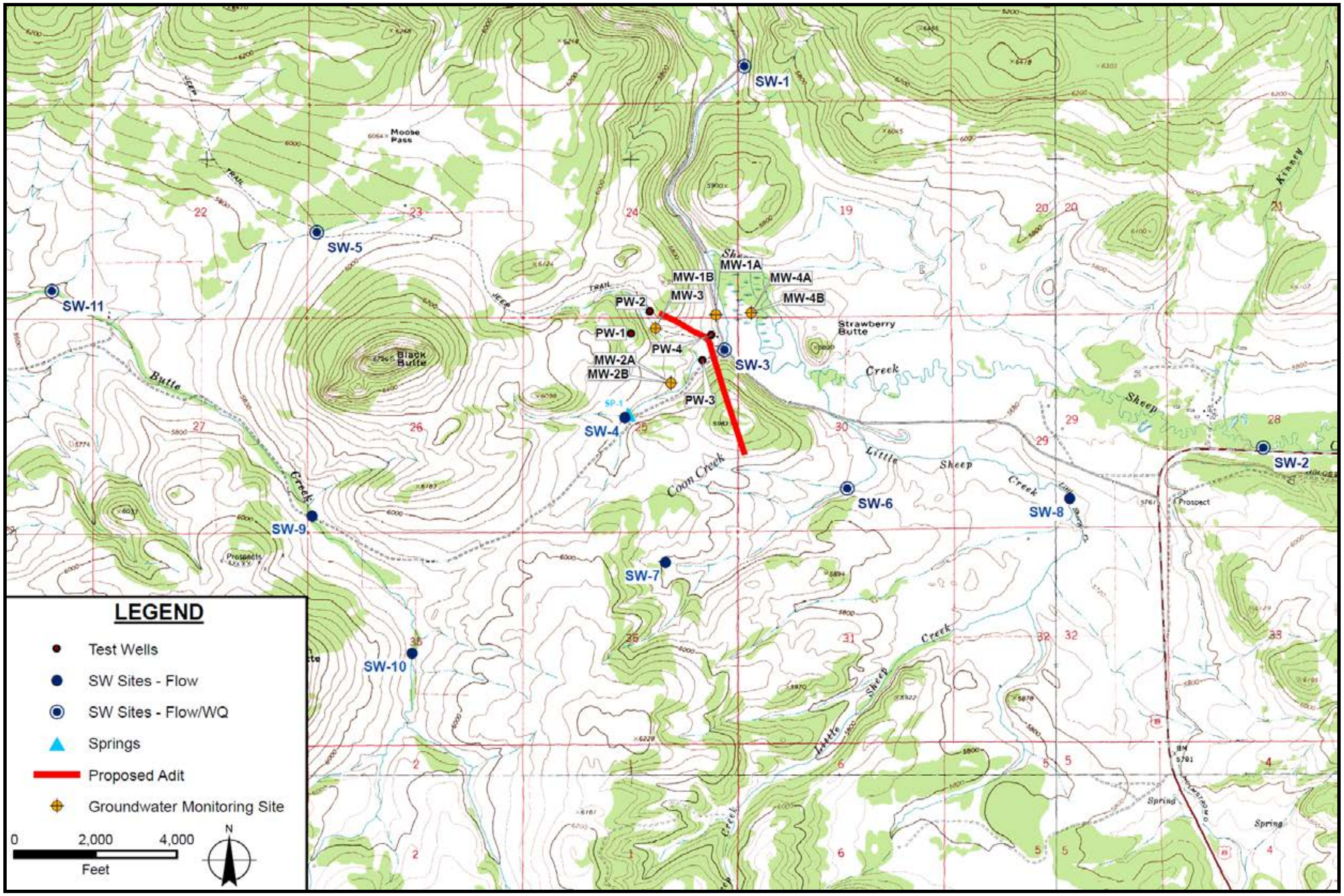
An air quality permit will be required for the construction and operations of the Black Butte Copper Mine, with an application submitted to the Montana DEQ at least 75 to 90 days prior to construction. In order for an air quality permit application to be submitted or to determine the need for a permit, an inventory of all equipment (stationary, portable, and mobile) that will be required for the Project is needed. The inventory includes the manufacturer and model of the equipment to insure all vendor emission factors and rates are included in the emission inventory calculation. Where those factors do not exist, the US Environmental Protection Agency (EPA) emission factors based on standard industrial classification codes (SIC) can be used. If the inventory projects less than 25 t/a, Tintina will request a finding from the DEQ stating that a permit is not needed. Additional background data required for the Black Butte Copper Mine's operating permit application would likely require an on-site meteorological station and a minimum of a one year period of record for data. Tintina installed and began collecting data from a meteorological station in March 2012.

20.4.2 WATER RESOURCES

GENERAL HYDROLOGIC SETTING

The location of surface and groundwater quality monitoring sites (Figure 20.1), frequency of required sampling, and field and analytical parameter lists were discussed with the Montana DEQ prior to initiating water resource baseline studies in June 2011. Quarterly sampling of surface and groundwater was agreed upon for all surface and groundwater monitoring sites.

Figure 20.1 Surface and Groundwater Monitoring Sample Locations



SURFACE WATER QUALITY AND QUANTITY

Sheep Creek and Butte Creek are high quality streams that are used for stock water and fishing (RMI 2010). Numerous surface water quality monitoring stations were identified both up and down gradient of proposed mine facilities. Three quarterly sampling events were completed in 2011, four were completed in 2012, and two events have been completed in 2013. Surface water sites exhibit neutral to slightly alkaline pH and low to moderate specific conductance, with calcium and bicarbonate-dominated major ion chemistry. Metals data suggests infrequent excursions above DEQ-7 water quality standards for aluminum, lead, iron, and manganese. Sheep Creek is listed under Section 303d of the *Federal Clean Water Act* for the State of Montana, from its headwaters to its junction with the Smith River, due to elevated fecal coliform bacteria and trace detections of mercury. Tintina has requested 303d designation de-listing of Sheep Creek from the DEQ based on more recent and extensive water quality sampling results.

GROUNDWATER MONITORING SITES AND WATER LEVEL DATA

Groundwater baseline studies were also initiated in 2011 and, to date, 11 groundwater monitoring installations have been completed (Figure 20.1). Groundwater quality and water level data has been collected from monitoring wells (MW-1a/1b, and MW-2a/2b, MW-3, and MW-4a/4b), and bedrock hydrologic pump test wells (PW-1, PW-2, PW-3, and PW-4), all of which are located in the vicinity of and down-gradient of proposed mine activity and facility locations. These wells were completed in alluvium (1a/2a/4a) and bedrock (1b/2b/4b, PW-1 through 4), respectively. In addition, one groundwater monitoring well (MW-3) was completed in the mineralized interval of the upper copper zone in order to measure in situ baseline water quality in the undisturbed mineralized interval. Four additional pumping wells were installed in the vicinity of the proposed adit decline in 2012 for hydraulic conductivity testing; however, these wells are also monitored for quarterly water quality. One new pair of wells MW-5a/5b is scheduled for installation in the summer of 2013 near the centre of section 30 of T12N, R7E (Figure 20.1) to monitor water quality down gradient of the proposed exploration decline portal.

Groundwater sampling was conducted at one colluvial/bedrock well pair (MW-1a/1b) in 2011, 2012, and again in January and May of 2013. Results show some differences in pH, conductivity, and the ratio of alkalinity to sulfate between colluvial water and deep groundwater. DEQ-7 groundwater (human health) standards were exceeded in some water samples from the colluvial well for dissolved thallium, and in the bedrock well for dissolved arsenic and thallium. Data from all of the wells will be used to evaluate the need for additional groundwater monitoring in support of the larger mine permit application.

As a part of the initial water resource evaluation, nine seeps and 13 springs in the Project area have been identified, mapped, and some sampled for water quality and flow (Hydrometrics 2011; 2012). Observed flow rates at the springs ranged from 1 g/min to as much as 50 g/min. The springs generally exhibit neutral to slightly alkaline pHs. Background nitrate concentrations were low and metal concentrations were within regulatory limits.

Aquifer Testing

Two aquifer testing programs have been completed for the Project which used open PQ and HQ exploration core holes along with newly constructed pumping wells for testing in the vicinity of two proposed decline alignments. The tests provide initial estimates of water volumes expected during development of the mineralized material deposit, for planning purposes. Results of this work showed that the shallow bedrock in the vicinity of the exploration adit, as it passes beneath the surface projection of a creek some 2,600 ft north of the portal, is moderately fractured and has the potential to produce between 175 and 615 gpm of inflow to the exploration adit. The high end of this range is a very conservative estimate, but may be representative of initial inflows, prior to any adit fracture grouting, which should significantly reduce this potential inflow. The extent of drawdown predicted near the adit portal will likely be offset by effects of re-infiltration of water in the adjacent land application disposal area, which was not evaluated in this simulation. The quality of the shallow groundwater is very good, exceeding only the secondary standard for iron. As the adit penetrates deeper, the bedrock becomes significantly tighter and predicted inflows to the adit from this shallow near surface zone through the mineralized material zone are minimal (on the order of 15 gpm).

Wetlands Delineation

A baseline wetland inventory and mapping program was conducted to clearly delineate any wetland areas within the Project area of influence. A large wetland complex, charged by surface and groundwater, is present on the floodplain of Sheep Creek and Little Sheep Creek. Other linear wetlands, originating from springs and occurring along stream bottoms, dissect upland habitats and flow down-gradient into Sheep and Little Sheep Creeks.

Although wetlands, seeps, and springs are present in various places throughout the Project area, a preliminary layout of mine portal areas and support facility sites for the overall mine-operating permit has avoided disturbance of all wetland areas. However, small wetlands do occur in areas initially proposed for possible tailings impoundment sites under the future larger scale mine operating permit proposal. If in fact these sites are selected for development, Tintina will need to obtain State and Federal permits and adhere to regulations for replacing wetland ecosystem resources.

Water Rights

Tintina has negotiated the use of the water rights of the lessor (the land, mineral rights and water rights owners) as a part of its mining lease agreement. Additional water right acquisitions are being evaluated.

20.4.3 Soil Resources

An Order II soil survey, including new mapping of soils, was completed in the Project area to supplement existing mapping by the NRCS (NRCS 2011). Table 20.2 lists the soils mapped in the Project area. The depth and volume of salvageable topsoil and sub-soils were determined. Physical data collected on soils included depth, percent slope of the land surface, saturation percentages, texture, organic matter content, and coarse fragment content. Chemical data have included soil pH, nutrient content (N, K, P) and electrical conductivity. Most soils in the area are rated as being either poor or fair for use as a topsoil source. Poor ratings were generally due to shallow depths to bedrock, or a high percentage of rock fragments within the soil.

Calculations documenting the availability of soil volumes needed for reclamation purposes will be required for the mine operating permit application. Operationally, once the suitable depths are determined, topsoil, and subsoil will be stripped from all proposed disturbance areas (i.e. waste rock and tailings storage areas, roads, soil stockpile areas) prior to construction. Salvaged topsoil and subsoil will be stockpiled separately and will be seeded with an approved seed mix to prevent weed invasion and minimize erosion.

Table 20.2 Soil Types near the Proposed Black Butte Copper Exploration Adit

Map Unit Number	Name	Description	Topsoil Source Rating
38E	Woodhall-Woodhurst complex	Loamy-skeletal, mixed, superactive Ustic rgicryolls	Poor
340D	Burnette-Lymanson-Adel loams	Fine, smectic, Pachic Argicryolls	Fair
1176D	Stubbs-Copenhaver complex	Fine loamy, mixed, superactive Pachic Argicryolls	Fair

During the soil survey, constant and falling head tests using large percolation test pits, and other tests using a double-ring infiltrometer were used to measure porosity and permeability of various colluvial and bedrock units in areas likely to be used for LADs of excess water. In addition, soil profiles were measured in trenches to characterize the nature of soil and colluvial materials in the A, B, and C soil horizons. Several promising areas for land application of water were located to the south of the mine portal and will be recommend for use in combination with various source control techniques (grouting and groundwater pumping) during dewatering of the exploration decline.

20.4.4 Mine Waste Geochemical Characterization

Baseline environmental geochemistry of mine wastes is needed to secure regulatory approval of Tintina's plans for development of an exploration decline for the Project. Although there are no formalized guidelines for waste characterization, the DEQ generally follows best management practices defined by the US EPA, the US Forest Services (USFS), and the industry, as summarized in the Global Acid Rock Drainage (GARD) guide. Tests of representative samples are needed to describe the acid generation and metal release potential in order to identify selective handling criteria and monitoring/mitigation requirements, rule out the presence of asbestiform minerals, and support adequate waste and water management strategies for the decline project. A secondary goal of this work is the development of a comprehensive sampling and analysis plan for future characterization of the overall environmental geochemistry for the proposed Black Butte Copper Mine. The objectives and methods of the environmental geochemistry program have been reviewed with and approved by the DEQ, who will be involved in reviewing geochemistry results for the Project as it proceeds.

Testing and Test Results -Acid Generation Potential and Metal Mobility

The zone of exploration interest targeted by the 2012 Johnny Lee Decline is the USZ, which hosts copper-cobalt mineralization in the calcareous shale of the lower Newland Formation (RMI 2012). The sulphur content of the 135,000 tons of rock to be mined from the evaluation adit is variable, ranging from below detect to more than 40% by weight. Statistical evaluation of 248 whole rock analysis (ICP analysis of a four-acid digestion) were used to validate the selection of 6 to 28 samples representing each of six lithologic material types for static and kinetic testing of acid generation potential and metal mobility (Table 20.3).

Table 20.3 Sample Lithology and Type of Testing

Lithologic Unit	Lithology	Percent of Material Mined	Number and Type of Testing			
			ABA	NAG pH	SPLP	Humidity Cell
Igneous Intrusive (IG)	Igneous intrusive	1	9	9	1	0
Sulphide Zone (0/1 SZ)	Sulphide zone	5	0	0	0	0
Lower Newland Fm hanging wall (Ynl)	Calcareous shale and dolostone	41	28	28	1	1
Lower Newland “Nose interbeds” (Ynl 0)	Dolostone	6	6	6	1	1
Sulphide Zone (SZ Sub 0)	Massive sulphide	Unknown	8	8	1	0
Upper Sulphide Zone (USZ)	Massive sulphide with cobalt and copper	11	11	11	1	1
Copper Ore	Copper ore	10	0	0	0	0
Lower Newland Fm footwall (Ynl B)	Shale and conglomerate	26	7	7	1	1

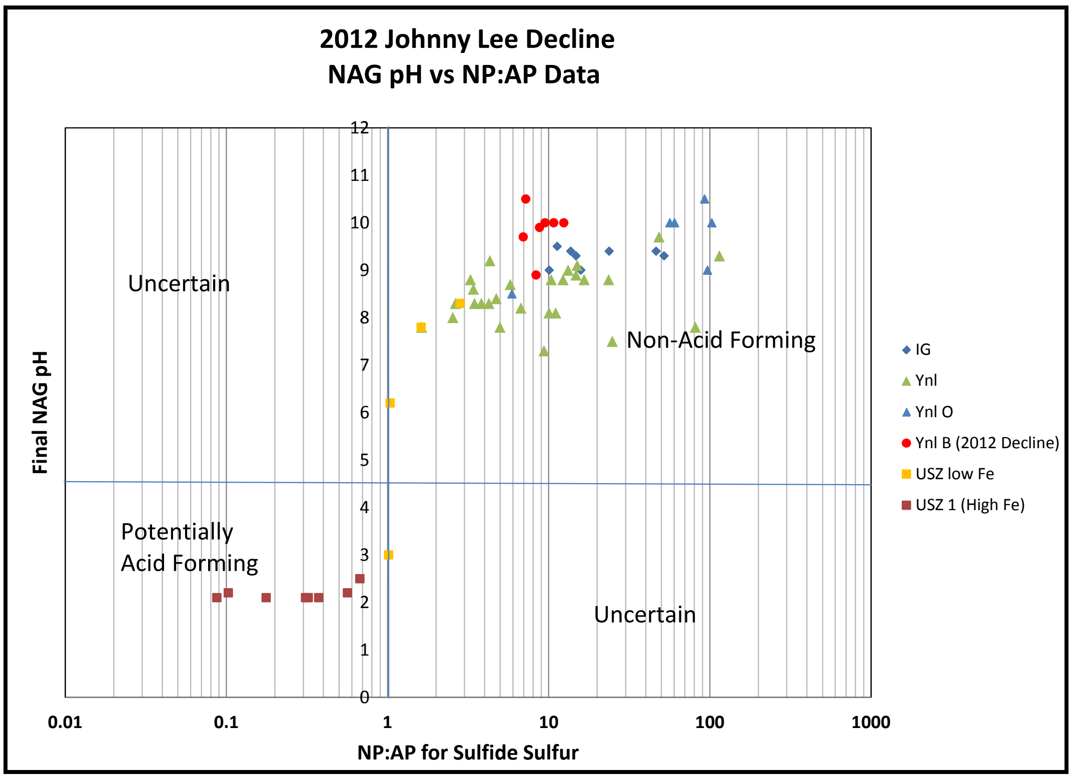
Results of this study (Figure 20.2) indicate that the igneous intrusive (*IG*), lower Newland dolomite “nose” (*Ynl 0*), lower Newland footwall shale and conglomerate (*Ynl B*), and much of the undifferentiated lower Newland Formation (*Ynl*), are strongly net neutralizing and are unlikely to generate acid, and can therefore be handled as non-potentially acid generating (NAG) rock. With the exception of the *IG*, these lithotypes also have low potential to release metals in concentrations that are likely to exceed groundwater standards, indicating that they can safely be stockpiled. Based on SPLP tests, potential does exist for leachate concentrations to exceed MDEQ surface water standards for aluminum, iron, chromium, and selenium, particularly from the *IG*, suggesting that care should be taken to prevent discharge from the rock pile facilities to surface water.

The *USZ and 0/1 SZ* should be handled as potentially acid generating (PAG) rock. Due to the occurrence of sulphide interbeds, which increase in number with proximity to the *USZ*, the *Ynl* requires further evaluation prior to and during construction of the adit.

Results of metal mobility testing using the SPLP method indicate low levels of potential metal release, but are limited by the elevated pH (in some cases, above 9) associated with the high carbonate content of the samples and resulting disequilibrium in the bottle roll tests. For this reason, and to confirm static test predictions of acid generation potential, kinetic tests of composite samples of *USZ, Ynl, Ynl 0*, and *Ynl B* lithotypes were initiated early in 2013, to provide further information for rock handling during construction of the evaluation adit.

There are no identified asbestiform minerals in any of the lithotypes to be mined from the 2012 Johnny Lee Decline at the Black Butte Copper Project.

Figure 20.2 Comparison of NAG pH with NP:AP by Lithology



ROCK MANAGEMENT

The results of this study provide clear guidance for management of rock that would be produced from the 2012 Johnny Lee Decline. The lithotypes *USZ* and *0/1 SZ* should be placed in the lined PAG facility based on lithotype. The non-acidic but potentially metalliferous *IG* unit should also be placed in the PAG facility, along with any identified acid subsections of the *Ynl*. All remaining lithologies can be placed in the NAG facility.

MONITORING AND MITIGATION

Geological mapping with onsite operational NAG pH testing should be used during development of the exploration decline where needed to screen the *Ynl* for selective handling, with offsite confirmation testing of a subset of samples to allow results to be added to the existing data. Rock with visual sulfide, or a NAG pH of less than 4.5, should be managed as PAG. Additional SPLP metal mobility tests of *Ynl* NAG and PAG composites are also recommended, to evaluate the effectiveness of management strategies.

Classification of the lithotypes *Ynl O* and *Ynl B* as NAG rock should be verified with limited operational sampling to validate the results of this baseline study and to support efforts to characterize these units for the overall full scale mining operation.

Monitoring of water quality and weathering products within the decline, and in the NAG and PAG waste rock facilities, could provide in situ data of use in confirming laboratory test results and interpreting future kinetic test work.

20.4.5 BIOLOGICAL RESOURCES

Reconnaissance-level baseline studies have been conducted to characterize wildlife habitat and assess the potential for plants and animals of conservation concern to be present within the proposed Project area. Databases maintained by the Montana Natural Heritage Program and the Montana FWP were also queried to obtain natural resources information relevant to the Project area.

VEGETATION RESOURCES

Reconnaissance level baseline vegetation studies were conducted in the area during the summer of 2011 (Elliot 2011). Wetland, riparian, shrub, conifer forest, and sagebrush/grassland habitat based communities were identified and described. No plant Species of Concern (SOC) are listed in the vicinity of the Project area; however, nine SOCs are known to exist in other areas of Meagher County (MNHP 2011). It is possible that these species are also present, but have not yet been identified in the Project area.

Noxious weeds observed in the Project area include Canada thistle, musk thistle, and hound's tongue. Tintina shall make reasonable and conscientious efforts to identify and control and suppress the introduction of all weeds that its operations introduce, or are likely to have introduced. Noxious weeds will be controlled using appropriate mechanical, biological, and chemical treatments that meet the requirements of Montana and federal laws and a weed control plan will be developed between the land owners, county weed control officials, and Tintina.

WILDLIFE RESOURCES

Reconnaissance-level baseline wildlife studies have been conducted in 2011 to characterize wildlife habitat and assess the potential for animals of conservation concern to be present within the proposed Project area (Elliot 2011). Databases maintained by the Montana Natural Heritage Program and the Montana FWP were also queried to obtain natural resources information relevant to the Project area.

Wildlife species or their sign (tracks, scats, skeletal remains, nests, beds, or calls) observed during field studies include white-tailed deer, mule deer, elk, coyote, beaver, Richardson's ground squirrel, pocket gopher, red-tailed hawk, Swainson's hawk, northern harrier, kestrel, Canada goose, Clark's nutcracker, eastern kingbird, barn swallow, tree swallow, savannah sparrow, lark sparrow, gold finch, rock dove, northern flicker, yellow-rumped warbler, mourning dove, raven, American robin, ruffed grouse, magpie, and red-winged blackbird.

Wildlife SOCs are not known to have been surveyed or identified specifically within the Project area, but SOCs have been identified in Township 12 N Range 6 E of Meagher County including 20 birds, 5 mammals, 1 amphibian, and 5 fish species (Montana Natural Heritage Program 2011).

FISHERIES AND AQUATIC LIFE

Sheep Creek and Little Sheep Creek are perennial streams that meander through a broad floodplain of sub-irrigated meadows and shrub-dominated wetlands. Sheep Creek has riffles and pools with cobble and gravel substrates. There is evidence of abandoned beaver dams, and oxbows are a prominent feature of the broad floodplain. It is likely that brook trout, rainbow trout, westslope cutthroat trout, and hybrids of rainbow and westslope cutthroat trout are present in waters of the Project area. No critical habitat locations have been identified at this time; however, some may exist in the area.

Benthic invertebrate communities in the Project area will ultimately require quantitative sampling of baseline conditions in order to provide a basis for the quantitative evaluation of project related effects. No taxonomic information was available for review.

20.4.6 CULTURAL RESOURCES

The Montana DEQ encouraged Tintina to conduct a cultural resource inventory (Tetra Tech 2011) prior to filing the Exploration License Amendment to construct the exploration decline. Tintina contracted an intensive pedestrian inventory of 970 acres of private land (Sections 24, 25 and part of 30 of T12N, R6E) within the Project area which covers the central portion of the lease block, including a two-square mile area surrounding the mineral deposit area. This area also includes most of the proposed facilities identified during conceptual planning of the larger scale mine operating permit area, including the mine portal, plant sites, temporary waste rock storage facilities, portal pad facilities, and access roads.

20.4.7 SOCIOECONOMIC RESOURCES

The 2010 population of Meagher County was 1,891. Meagher County is sparsely populated by Montana and US standards. The land area is 2,391.8 square miles and the population density is 0.8 people per square mile, while the average for Montana in 2010 was 6.8 people per square mile. The population in Meagher County has decreased slightly since 2000, but it is higher than the 1990 population of 1,824. The US Census Bureau reports that migration out of the county is greater than migration into the county (loss is 2.1%), and the number of births has also decreased, which are the primary causes of the decline in population in the county. Meagher County has a significantly higher proportion of its population over the age of 65 (21.2%) compared to Montana (14.6%) and the US (12.9%). In addition, the percent of the population under the age of 5 is 5.6% in Meagher County, 6.4% in Montana, and 6.9% in the US.

Meagher County is rural and the main industries are farming and ranching, which employ 173 people or 16.9% of the population. Interestingly, the percentage of people employed by farming and ranching has decreased by 23.8% since 2001. Other major industries that employ people include: retail trade (9.5%); arts, entertainment and recreation (5%); accommodation and food services (6.7%); other services (6.7%); and government (14.1%). Growth industries for jobs include: retail trade (+34%); real estate (+142.3%); education (+12%); arts, entertainment and recreation (+4.8%); and other services (+5.9%). Industries showing a loss of jobs include: farming/ranching (-23.8%); accommodation and food services (-7.5%); and government (-16.1%).

The unemployment rate is an indication of the potential number of available employees for Tintina's Project. Considering the nationwide economic conditions, both Meagher County and Montana reported unemployment rates for August 2011 below that of the national average (i.e. 7.8% and 7.1%, respectively).

Income is reported by the US Census as per capita and household. The per capita data takes the total income for the county or state and divides it by the total population in each for an indication of the income per person. The Meagher County and Montana per capita incomes are US$18,866 and US$22,881 respectively. The median household income for Meagher County and the State of Montana are US$32,409 and US$42,222, respectively. The percentages of the populations in Meagher County and the State of Montana that are considered below the poverty level are 19% and 15%, respectively.

Operationally, Tintina expects to employ about 175 people with about 80% of the work force (140 people) working a seven-day-on/seven-day-off schedule. The remaining work force (about 35 people) would work a regular five-day work week. Average incomes in the mining industry include US$86,738 for general managers and operations managers/superintendents, US$65,557 for first line supervisors, and US$52,884 for non-supervisory miners (http://www.bls.gov/oco/cg/cgs004.htm#earnings; Bureau of Labor and Statistics).

20.4.8 LAND USE

Land uses in the Project area are predominantly agricultural, with hay and livestock production the primary activities. In addition, outfitters use the Sheep Creek drainage for big game hunting and fishing.

The proposed mine facilities fall entirely within land leased and controlled by Tintina. The land consists of two tracts of private property owned by the Bar Z Ranch, three members of the Hanson family, and/or Rose Holmstrom, who together control 100% of the surface and/or mineral rights. Lease payment agreements between Tintina and the surface, mineral, and water rights owners vary but the leases are each for 30 years and are renewable for subsequent periods of 10 years (RMI 2010). The leases stipulate that only underground mining will be practiced. Post mining land uses are expected to revert to farming, ranching, outfitting/guide services, and recreational access.

20.4.9 Water Management

Water management at the Property will be a critical issue because of the sulphide mineralogy of the deposit and the need to protect surface and groundwater resources from contamination. Oxidation of sulphide mineralized material in contact with water can mobilize trace metal contaminants. Tintina is in the process of preparing a water management plan incorporating a number of critical components to provide source and migration control of these potential contaminants.

Tintina plans two methods of source control for water generated from underground workings. The first is the construction and pumping of perimeter abstraction wells that will attempt to dewater the block of ground prior to mining. Water generated would be disposed of in LAD systems or by direct injection back to the groundwater system distal to the mining area. A second method of source control will be the implementation of an aggressive underground grouting program in advance of driving development and production headings.

In addition to possible groundwater injection, disposal of any mine water discharge would be to surface LAD areas via a surface drip emitter discharge system or traditional impact-type irrigation systems. A major component of this method of water disposal is through evaporation, so often the impact-type systems work best, particularly during the spring-summer-early fall seasons when vegetation growth and evaporation are high. Use of these surface LAD systems could be most effective during initial dewatering of the block of ground to be mined when large volumes of water need to be disposed of, as opposed to smaller sustained mine-inflow later in the mining cycle. However, because water needs to be disposed of on a year around basis, large area underground drain field systems would be constructed to dispose of water below the frost level, returning water into the near surface colluvial and/or shallow fractured bedrock system. Tintina has conducted shallow and deep percolation testing to identify areas suitable for these types of disposal.

Two waste rock storage facilities (a PAG and a NAG facility) will need to be constructed for placement of initial mine waste generated prior to the construction of a tailings impoundment, where the waste would be ultimately stored (probably, underwater) on a long-term basis. These waste rock storage facilities would be constructed using a composite compacted clay/HDPE geotextile bottom liner, with an internal waste rock seepage collection system that reports to HDPE-lined seepage collection ponds, which could be pumped to a water treatment facility for treatment as necessary prior to disposal in LAD systems. Diversion structures would channel surface water away from the waste rock facility. The use of a temporary cover may be considered to minimize the infiltration of precipitation into the waste rock facility, especially during periods of predicted rain or snowfall, although there is merit to using this contained facility as a field scale kinetic test of environmental geochemistry for waste rock. Ultimately, PAG waste rock would be placed back underground if the Project is abandoned early, during, or prior to construction of the exploration decline. Once the larger scale mine is in operation, waste would either be stored in underground mine workings voids or placed in the tailings impoundment. Tailings facilities have not been designed at this stage of the Project; however, conceptual designs propose to use sub-aqueous deposition of tailings operationally and at closure to minimize sulphide oxidation.

Water treatment facilities are planned for construction and operation at the site to treat whatever volume of water remains following minimization through source control and LAD disposal of water meeting groundwater quality standards. Treatment facilities being considered include lime treatment, reverse osmosis with thermal evaporation of brines, sulphide precipitation, and zero discharge strategies. Other methods, such as absorptive media treatment of RO brines and anaerobic biological treatment systems for nitrates may be considered as the planning of the Project progresses.

Best Management Practices (BMPs) and a storm-water management plan will be prepared and implemented at the site to prevent co-mingling of unaffected surface and groundwater from waters that come in contact with the mining or milling process and to control run-off from the site and adjacent areas. A spill prevention and containment plan will be developed for fuels and lubricant storage and use areas

20.4.10 POTENTIAL POSITIVE EFFECTS OF PROJECT ON LOCAL COMMUNITIES

Potential positive effects of the proposed Project development include:

- reduction of unemployment in the region
- increased tax base for local, state and federal government
- economic stimulus for existing local businesses
- long-term, meaningful employment for residents in mining operations and related positions (e.g. environmental monitors, service industry sector)
- economic development and contract opportunities for existing and new businesses
- community infrastructure improvements.

21.0 CAPITAL AND OPERATING COSTS

Tetra Tech developed a capital cost estimate (CAPEX) and operating cost estimate (OPEX) for the Project, based on the findings of this study. A summary of both the CAPEX and OPEX is provided in Table 21.1, and discussed in greater detail in the subsections that follow. The CAPEX and OPEX provide the basis for the economic analysis in Section 22.0.

Table 21.1 Summary of Capital and Operating Costs

Cost Type	Total ($)	Unit Cost ($/t milled)	Estimate Accuracy Range
Total Capital Costs	217,753,000	-	±40%
Total Operating Costs	-	66.48	±40%

The estimate base date is Q1 2013; no allowance for escalation was included. Quotations provided by vendors are budgetary and non-binding. All costs are expressed in US dollars unless otherwise stated. A foreign currency exchange rate of US$1.00/Cdn$1.00 was utilized for the estimate.

21.1 CAPITAL COST ESTIMATE

21.1.1 INTRODUCTION

This section describes the methodology of the development of the Project capital cost estimate.

The capital cost estimate is a Class 5 estimate prepared in accordance with the AACE International Estimate Classification System with an expected accuracy range of ±40%.

21.1.2 CAPITAL COST SUMMARY

Table 21.2 outlines the CAPEX subtotals by area, and the total CAPEX for the Project.

Table 21.2 Capital Cost Summary

Item	Total Cost ($)
Direct Costs	
Overall Site	2,790,724
Mine Capital	54,406,432
Mine Surface Facilities	12,017,674
Processing	52,218,559
Water Management (Knight Piésold)	11,069,469
Utilities	5,314,571
Buildings	8,242,691
Off-site Infrastructure	4,066,207
Plant Mobile Equipment	2,063,212
Subtotal	**152,199,539**
Indirect Costs	26,567,854
Owner's Costs	5,642,746
Contingency	33,342,538
Total Capital Costs	**217,752,677**

21.1.3 Contributors to the Estimate

The estimate was developed by Tetra Tech, in conjunction with.

- AMEC: underground mine and paste plant
- Knight Piésold: tailings and reclaim, and water management
- Tintina: Owner's costs.

21.1.4 Components of the Estimate

The estimate consists of four main parts:

- direct costs
- indirect costs
- contingency
- Owner's costs.

21.1.5 Estimate Base Date, Exchange Rate, and Validity Period

Tetra Tech prepared this estimate with a base date of Q1 2013. No escalation beyond Q1 2013 was applied to the estimate.

The budget quotes used in this estimate were obtained in Q1 2013 and have a 90-day period of validity.

21.1.6 ESTIMATE APPROACH

The capital cost estimate is based on the following:

- assembly and structure per the Project work breakdown structure (Table 21.3)
- equipment costs, based on in-house data or quotations from similar projects
- vendor quotations (budgetary, non-binding)
- prices and quantities as supplied by the other consultants
- preliminary material take-offs by discipline, as required
- electrical, plate work, instrumentation, and piping expressed as percentage.

Equipment and material costs are included as FCA (free carrier) or FOB (free board marine) manufacturer plant and exclusive of spare parts, taxes, duties, freight and packaging. These costs are included in the indirect section of the estimate.

The estimated installation hours were based on in-house experience and published references.

The allowance for freight costs and spares costs are based on a percentage of the value of materials and equipment. With the exception of the mining equipment, the costs are inclusive of freight.

There are repair facilities located close to the Project site; therefore, spares costs have been included as a lump sum.

The estimate assumes the construction man-hours/workweek to be a 10 h/d with a 3-week-on and 1-week-off rotation. Due to proximity of municipalities for labour supply, there will be no need for a temporary construction or permanent operations camp.

Owner costs were included as a percent of the direct costs.

Table 21.3 Work Breakdown Structure

Major	Major Description	Area	Area Description	Sub-area	Sub-area Description
11	Overall Site	111	General Development	11110	Bulk Earthworks/Site Preparation
				11115	Environmental Works
				11140	Site Roads at Mine
21	Mine Underground (AMEC)	211	Mine Development	21100	Mine Development
22	Mine Surface Facilities (AMEC)	223	Backfill Plant	22300	Backfill Plant
31	Process	311	Crushing	31110	Primary Crushing
				31120	Primary Crushing Conveyance
		312	Ore Stockpile and Conveying	31210	Crushed Ore Stockpile and Reclaim
				31220	Crushed Ore Storage Conveyance
		313	Process Plant	31320	Grinding and Classification
				31330	Pebble Crushing
				31340	Flotation and Regrind
				31350	Concentrate Handling and Loadout
				31360	Reagents
41	Water Management (Knight Piésold)	411	Tailings (Knight Piésold)	41110	Tailing Disposal and Reclaim
				41120	Tailing Management Facilities
		412	Seepage (Knight Piésold)	41210	Seepage Collection and Sediment Control
		413	Water Management	41310	Water Management Systems
				41320	Fresh Water Supply
				41330	Gland Water

table continues...

Major	Major Description	Area	Area Description	Sub-area	Sub-area Description
51	Utilities	511	Main Substations	51160	Main Substation
		512	Utilities – Fuel Supply, Storage and Distribution	51220	Diesel
		513	Utilities – Water Systems	51310	Water Distribution System
				51320	Potable Water
				51330	Process Water
		514	Utilities – Waste Disposal	51410	Solid Waste Disposal
				51420	Sewage – STP
		515	Utilities – Air	51510	Plant and Instrument Air
61	Ancillary Buildings	611	Ancillary Buildings	61110	Administration and Mine Dry
				61115	Assay and Metallurgical Laboratory
				61125	Maintenance
				61130	Core Storage
				61140	Gatehouse and Fencing
				61150	Weighbridge
65	Off-site Infrastructure	651	Off-site Infrastructure	65110	Allowance for Power Line Upgrade & Substation
72	Plant Mobile Equipment	721	Mobile Equipment	72110	Surface Mobile Equipment
91	Indirects	911	Indirects – Mine Area	91110	Construction Indirects
				91120	Initial Fills
				91130	Spares
				91140	Freight and Logistics
				91150	Commissioning and Start-up
				91160	Engineering Procurement and Construction Management (EPCM)
				91161	EPCM (AMEC)
				91170	Vendor Commissioning and Assistance
98	Owner's Costs	981	Owner's Costs	98100	Owner's Costs
99	Contingency	991	Contingency	99110	Contingency

21.1.7 Elements of Costs

Direct Costs

Labour Rates, Productivity and Travel Allowances

A blended labour rate of $64/h was used throughout the estimate.

The labour rates include:

- vacation and statutory holiday pay
- fringe benefits and payroll burdens
- overtime and shift premiums
- small tools
- consumables
- personal protection equipment
- contractor's overhead and profit
- living out allowance.

A productivity factor of 1.15 was applied to the labour portion of the estimate. This implies an efficiency of approximately 87%, and allows for inefficiencies such as extended work hours, potential climatic conditions, and to the 3-week in 1-week out rotation.

Duties and Taxes

Duties and taxes have not been included in the CAPEX.

Cost Basis by Discipline, Bulk Earthworks Including Site Preparation, Access and Haul Roads

Excavation of top soil and an allowance for rock excavation was assumed. Structural fill pricing are based on aggregates being produced at site utilizing a portable crushing and screening plant; the mobilization and set-up costs of the aggregate plant are included in the unit rates. The actual cost of aggregate production is included in the unit rates. Earthwork quantities do not include any allowance for bulking or compaction of materials; these allowances are included in the unit prices.

For the purposes of developing the estimate, Tetra Tech assumed:

- The topsoil thickness will average 150 mm in thickness, and will be stripped and stockpiled on-site.
- Five percent of excavated material will be unsuitable for re-use.

- An average of 50% of the excavated material will be in-rock excavation and 50% of that will be rippable rock; the balance will require drilling and blasting.
- Surplus excavated material will stockpiled within 5 km of the Project site.
- Allowable ground bearing pressure is assumed to be minimum 400 kPa at the plant site location; equipment foundations may require greater ground bearing capacity (to be confirmed by selected vendors and a geotechnical engineer in the next phase of the Project).
- The primary crushers will be located on rock.

An allowance of $100,000 has been included for environmental works.

Mining

Pre-stripping and mining equipment are included in the estimate provided by AMEC. An allowance for mining procurement, as well as a contingency amount, were included with the construction indirects. Mine engineering and construction management are included in Owner's costs.

Concrete

Concrete quantities are based on estimated quantities with an allowance included for overpour and wastage.

Typically, all concrete is based on a 28 d compressive strength of 30 MPa. Concrete unit rates include for formwork, reinforcing steel, placement, and finishing of concrete.

Structural Steel

Structural steel quantities are based on estimated quantities with no allowance made for growth and wastage. Allowances are included for cut-offs, bolts, and connections.

Craneage is included for all tonnages, at a rate of $250/t.

Platework and Liners

Preliminary quantities for platework and metal liners for tanks, launders, pump-boxes, and chutes are estimated using recent similar projects and in-house data.

Mechanical

The equipment estimate has been prepared based on the Project equipment list and process flow diagrams. The mechanical pricing is based on budgetary quotes obtained for the recent projects.

Heating, Ventilation, Air Conditioning, and Fire Protection

HVAC and fire protection is included as a percentage of the process equipment cost and is based on experience with recent similar recent projects.

Piping and Valves

Piping and valves allowances were included as a percentage of process equipment, based on experience with recent similar projects.

Electrical

Electrical allowances were included as a percentage of process equipment, based on experience with recent similar projects.

The power supply and substation are based on a quote from the local utility.

Instrumentation

Instrumentation is included as a percentage of the equipment list allowance assigned to each area and based on experience with recent similar projects.

Mobile Process Equipment

The estimated requirements for mobile equipment are included.

21.1.8 PERMANENT ACCOMMODATION, CONSTRUCTION CAMPS, AND CATERING

No allowance for permanent construction camps and catering during construction have been included.

21.1.9 TAXES AND DUTIES

Taxes and duties on materials were excluded from the estimate.

21.1.10 CONSTRUCTION INDIRECTS

Construction Indirects are based on a percentage of the direct costs.

This also includes a percentage for the water treatment including EPCM and contingency.

21.1.11 INITIAL FILLS

Initial fills are included in the indirect portion of the estimate.

21.1.12 SPARES

A nominal allowance of has been included for spares due to proximity of distribution facilities.

21.1.13 EPCM

An allowance of 10% of the direct costs has been included for EPCM activities for the process facilities. The EPCM allowance for tailings is 6% of the direct costs.

21.1.14 Logistics and Freight

A provision of 4% has been made for freight, calculated on the overall cost of materials process equipment.

21.1.15 Owner's Costs

An allowance has been included for the Owner's costs (including insurance, site orientation, and mine electrical costs), based on a percentage of the direct costs.

Exclusions

The following are not included in the capital cost estimate:

- force majeure
- schedule delays such as those caused by:
  - major scope changes
  - unidentified ground conditions
  - labour disputes
  - environmental permitting activities
  - abnormally adverse weather conditions
- receipt of information beyond the control of the EPCM contractors
- cost of financing (including interests incurred during construction)
- taxes
- schedule acceleration costs
- contractors camps including catering and housekeeping.

21.1.16 Costs Incurred Prior to Release of Detail Engineering and Construction Assumptions

The following assumptions have been made in the preparation of this estimate:

- All material and installation subcontracts were competitively tendered on an open shop, lump sum basis.
- Site work is continuous and is not constrained by the Owner or others.
- Skilled tradespersons, supervisors, and contractors are readily available.

- The geotechnical nature of the site is assumed to be sound, uniform, and able to support the intended structures and activities. Adverse or unusual geotechnical conditions requiring piles or soil densification have not been allowed for in this estimate.

21.1.17 Contingency

A contingency allowance of 20% for the process equipment and 25% for mining is included in the estimate.

It is expected that this allowance will adequately cover minor changes to the current scope to be expected during the next phase of the Project. The overall contingency for the Project is calculated to be 21.8% of the direct costs.

21.2 Mining Costs – Basis of Estimate

The capital and operating cost estimates for the underground mine has been prepared as an AMEC Type 5 estimate with 0 to 2% of full project definition. The estimate for AMEC's scope is considered to be a t a scopying level with an expected accuracy range of ±40%. This estimate is part of a larger study and is limited to underground development, associated underground infrastructure, and underground mining portion of the Project. The following items are included in the overall Project CAPEX but are not included specifically in the mine CAPEX:

- contingency
- EPCM costs
- Owner's management and supervision costs for the first 18 months of the Project.

An underground development contractor will be mobilized at the start of Year 1 and kept on site for 2.5 years to drive the majority of the capital development for the mine. The Owner's mining crews will be hired at the start of Year 2. A steep ramp up of the Owner's crews was assumed for the first half of Year 2 before production of mineralized material starts mid-year. The Owner's crews will be responsible for all production mining. The Owner's crews will drive the southeast ramp in the first half of Year 2 to provide a working area for training and ramp-up of Owner's personnel during this six-month period.

The mine capital and operating cost estimates do not include power consumption costs as these are included in the overall project cost estimate.

The scope includes the underground work to develop and mine the Johnny Lee UZ and LZ.

21.2.1 Project Data

The underground mine capital cost estimate is based on the following project data:

- design criteria
- block model provided by Tintina
- geotechnical data provided by Tintina
- AMEC mine plan
- preliminary general arrangement drawings
- single line electrical drawings.

Tintina will provide all mobile and fixed equipment to the contractor.

The contractor's work schedule will be 14-days-on, 7-days-off, and two 10 hour shifts per day.

The Owner's operating work schedule will be 7-days-on, 7-days-off, and two 10 hour shifts per day.

All quantities are displayed as metric with the exception of pipe sizes.

21.2.2 DIRECT COSTS

LABOUR RATES

Contractor Labour Rates

Contractor labour rates were established based on AMEC's recent experience with labour cost on underground mine development projects. Wage rates include base rates, scheduled overtime, payroll taxes and insurances, small tools and consumables, per diem and housing allowance. At Tintina's request, payroll burdens were calculated as 35% of the base rate plus overtime to be consistent with the overall study. The wage rates are based on a work schedule of 14-days-on, 7-days-off, and two shifts of 10 hours per day (see Table 21.4).

Table 21.4 Summary of Contractor Labour Rates

Description	Rate ($/h)
Development Lead Miner	89.96
Development Miner	76.84
Construction Leader	89.96
Construction Miner	86.55
Nipper	62.01
Truck Driver	76.84
Mechanic	76.84
Electrician	76.84

Owner Operating Labour Rates

Tintina provided base labour rates and bonus percentages. At the Owner's request, payroll burdens were calculated as 35% of the base rate plus overtime to be consistent with the overall study. The rates are based on a 7-days-on and 7-days-off at 10 hours per day (see Table 21.5).

Table 21.5 Summary of Owner Labour Rates

Description	Rate ($/h)
Miner Level 5	68.05
Miner Level 4	63.35
Miner Level 3	51.70
Miner Level 2	45.30
Miner Level 1	35.55
Underground Mechanic	62.45
Mechanic Helper	35.55
Electrician	52.55

Mobile Equipment

The major mobile equipment costs were based on 2013 pricing guidance from a major underground mining equipment supplier. The costs for minor equipment were developed from AMEC's in-house data. A diesel fuel price of $1.04/liter was used to develop operating cost.

In addition to the base cost of the mobile equipment, the following allowances were added to the base cost to calculate the equipment costs used in the estimate:

- development allowance: 2.5%
- spares allowance: 3 to 8% based on number of units
- freight allowance: 2%.

Fixed Equipment

Fixed equipment prices were developed from budgetary quotes or AMEC in-house data.

In addition to the base cost of the fixed equipment, the following allowances were added to the base cost to calculate the equipment cost used in the estimate:

- development allowance: 5%
- spares allowance: 3.3 to 8% based on number of units
- freight allowance: 5%.

UNDERGROUND DEVELOPMENT AND PRODUCTION DRIFTING

Underground mine development cost and advance rates were developed from bottom-up estimates. Cycle times were developed; then the equipment and crew required to accomplish the cycle were identified.

Mining cost and production rates for the mineralized material were developed from bottom-up estimates. Cycle times were developed; then the equipment and crew required to accomplish the cycle were identified.

The direct unit advancement costs for the various sizes of development and production headings are shown in Table 21.6.

Table 21.6 Summary of Direct Heading Advance Costs

Size/Description	Owner/Contractor	Material	$/m
5.5 m high by 5.5 m wide	Contractor	Waste	4,053
5.0 m high by 5.0 m wide	Contractor	Waste	2,746
5.0 m high by 5.0 m wide	Owner	Waste	2,598
5.0 m high by 5.0 m wide	Owner	Mineralized	2,645
4.5 m high by 5.0 m wide UZ	Owner	Mineralized	2,399
4.5 m high by 3.0 m wide UZ Slash	Owner	Mineralized	751
4.5 m high by 8.0 m wide UZ	Owner	Mineralized	3,065
4.5 m high by 8.0 m wide UZ Bench	Owner	Mineralized	2,570
3.5 m high by 3.5 m wide LZ	Owner	Mineralized	1,912
5.0 m high by 5.0 m wide LZ	Owner	Mineralized	2,781

UNDERGROUND STATION EXCAVATIONS

The mine development schedule and associated capital costs accounts for a 15% allowance for muck-bays and miscellaneous underground excavations in the three primary access ramps and the main decline. In addition, there are three main pump station excavations and one main substation excavation included in the capital development costs. The mine development capital costs also include muck-bays, sumps, and electrical load center stations for each mining area.

RAISES

There are 12 raises that will be excavated over the first six years of development that will serve as ventilation airways. Six of the raises reach the surface and six are internal to the mine. Ten of the 12 raises will also serve as personnel escape-ways and are planned to be supported. One of 10 supported raises will be supported using shotcrete and the other 9 are planned to be supported using rock bolts and welded wire mesh. The 12 raises will be excavated by various methods including Alimak, drop raising, and raise-boring.

A summary of the LOM planned raises is shown in Table 21.7. The capital costs shown in the table include all excavation and support costs. In addition to the $5.6 million of capital raise costs shown in Table 21.7, there are $0.86 million of raise contractor mobilization and de-mobilization costs included in the CAPEX.

Table 21.7 LOM Raise Development and Support Costs

Line Item	Raise Description	Year	Size (m)	Depth (m)	Cost ($/m)	Total Cost ($ million)
1	Alimak	1	4.3 by 4.3	52	5,488	0.29
2	Alimak	2	3.8 by 3.8	85	4,878	0.41
3	Drop Raise	4	3.1 by 3.1	6	1,668	0.01
4	Drop Raise	6	3.1 by 3.1	12	1,668	0.02
5	Raise Bore	1	4.3 diameter	245	4,307	1.06
6	Drop Raise	2	3.1 by 3.1	27	1,668	0.05
7	Alimak	2	4.3 by 4.3	145	5,488	0.80
8	Alimak	1	3.8 by 3.8	30	4,878	0.15
9	Drop Raise	2	3.1 by 3.1	11	1,668	0.02
10	Raise Bore	2	4.9 diameter	388	6,157	2.39
11	Alimak	3	4.3 by 4.3	78	5,488	0.43
12	Drop Raise	3	4.3 by 4.3	5	1,753	0.01
			Total	**1,084**	-	**5.62**

AMEC's in-house estimating models were used to estimate the cost of the Alimak and drop raises. Budgetary quotes were used to price the raise-bored raises.

The emergency hoist installed on the vent raise was based on a budget estimate. An allowance was applied for installing and commissioning the hoist. The cost of bringing power to the top of the raise was not included in the underground mine capital cost estimate.

ELECTRICAL

The electrical equipment costs for the underground project were developed from the requirements to power the electrical fixed and mobile equipment. This included drills, pumps, fans and miscellaneous electrical and lighting. The estimate considered the underground mine development and mining plan for locations and distance to the equipment. Equipment was priced based on budget quotes and AMEC's in-house database.

The underground mine electrical power loads were estimated by AMEC, but are not included in the mine capital and operating cost estimates. Power consumption costs are included in the overall project cost estimates.

Dewatering

Utilizing an underground pumping system, an allowance has been made in the mine plan to discharge 136 m^3/h (500 gpm) at the portal. This estimate is for preliminary cost estimating purposes only and will require hydro geological studies in the next phase of the Project. Based on the pumping allowance, AMEC selected the appropriate pumps and pipe sizes for the dewatering lines. The main pump skid equipment costs were based on a budget quote. Smaller pumps were priced using AMEC's in-house data.

Ventilation

AMEC evaluated the ventilation requirements of the mine and developed a ventilation plan. The costs of the required fans, ventilation doors, mine air heaters, and air flow regulators are included in the estimate. The cost of the mine air heaters was factored from a recent quote. The cost for the vent doors, flow regulators, and auxiliary fans was based on AMEC's in-house data. The ventilation raise costs are discussed above.

21.2.3 Indirect Costs

Engineering, Procurement and Construction Management

EPCM costs are excluded from the mine capital cost estimate. The underground mine EPCM costs are included in the overall project capital cost estimate.

Vendor Representatives

No vendor representatives are included in the estimate.

Commissioning and Start-Up

Commissioning of the underground ventilation fans, pump skids, and the emergency hoist is included in the direct cost of installation.

Freight

Freight costs were included as a percentage of the equipment and material cost.

Taxes and Duty

Taxes and duties on materials were excluded from the estimate.

Contingency

Contingency is not included specifically for the underground mine; it is included in the overall project capital cost estimate.

21.2.4 Owner's Costs

Owner's project management and supervision costs for the first 18 months of the Project are not included in the mine CAPEX.

21.2.5 Spare Parts

Freight costs were included as a percentage of the equipment and material costs.

21.2.6 Capital Cost Assumptions and Exclusions

Assumptions

The following assumptions have been made in the preparation of the mining portion of the CAPEX:

- The Owner will provide all mobile and fixed equipment to the contractor.
- The Owner will utilize capital leases for all of the mobile equipment. All leased equipment will be new and replaced if needed at the end of the lease period. Where the lease period ends, near the end of the mine life, no new equipment is leased, and it is assumed that the Owner will maintain and operate the existing equipment.
- An estimate of salvage value has not been included in the estimate.
- The underground dewatering system is based on 500 gpm capacity.
- Electrical service will be provided by the Owner to the contractor's surface facilities, the portal, and to the emergency hoists at no cost to the contractor.
- All mobile equipment provided by the Owner will be assembled, tested and ready for use by the underground contractor.
- The Owner will provide the following services to the underground contractor:
    - mine rescue service and equipment
    - use of mine-yard loader as required
    - loading and unloading freight trucks as required
    - cap lamps and maintenance
    - telephone and internet service
    - fire protection
    - sewage disposal
    - garbage disposal
    - water supply (service and potable)
    - snow removal
    - dry and dry operation
    - hazardous material disposal.
- Trucking of underground waste rock and mineralized material is costed to surface stockpiles within 100 m of the portal. Trucking costs beyond the near-portal stockpiles is by others and not included in the estimate.
- Dewatering cost for the underground includes cost of pumping to the portal. No costs for handling or treating the water on the surface are included.

- Ground support for the underground development and mining was based on limited geotechnical information provided by the Owner.
- No adverse weather conditions will impede construction.
- Security service during construction will be supplied by the Owner.

EXCLUSIONS

The following are not included or accounted for in this estimate:

- escalation during construction
- all surface work except installation of the emergency hoist
- schedule delays and force majeure events, such as those caused by:
  - scope changes
  - labour disputes
  - extreme weather events, seismic disturbances
  - unidentified ground conditions
  - political events, changes in laws and regulations
  - late receipt of information
  - shortages of material.
- hazardous or contaminated materials handling and/or disposal
- working capital
- geotechnical studies
- reclamation and closure costs
- environmental permitting
- sunk costs.

21.3 UNDERGROUND MINE CAPITAL COST ESTIMATE

21.3.1 SUMMARY

The underground mine CAPEX includes both initial and sustaining capital cost for the life of the mine. Initial capital ends at the end of Year 2. Sustaining capital starts at Year 3 and continues through the end of the mine life. Table 21.8 presents the LOM capital costs for the underground mine. The costs do not include contingency, EPCM, and capitalized power.

Table 21.8 LOM Underground Mine Capital Costs

Description	Total Capital ($ million)
Initial Capital Cost	54.9
Sustaining Capital Cost	69.3
Total Capital	**124.3**

21.3.2 INITIAL CAPITAL COST

The initial capital cost by year is shown in Table 21.9.

Table 21.9 Initial Capital Costs ($ million)

Description	Year 1	Year 2	Total
Mine Development			
Lateral	14.78	14.96	**29.74**
Vertical	1.90	4.07	**5.97**
Mobile Equipment Leasing	3.75	2.48	**6.23**
Fixed Equipment			
Ventilation	0.21	3.62	**3.83**
Electrical	1.66	1.00	**2.66**
Dewatering	0.32	0.26	**0.58**
Hoists	-	1.27	**1.27**
System Installation			
Ventilation	0.03	1.43	**1.45**
Electrical	1.00	1.14	**2.14**
Dewatering	0.02	0.02	**0.04**
Hoists	-	0.05	**0.05**
Other	0.99	-	**0.99**
Total	**24.65**	**30.29**	**54.94**

21.3.3 SUSTAINING CAPITAL COST

The capital cost by year is shown in Table 21.10.

Table 21.10 Sustaining Capital Cost ($ million)

Description	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10 to 12	Total
Mine Development									
Lateral	5.53	1.13	0.49	0.48	0.48	0.39	0.36	0.48	9.34
Vertical	0.48	0.01	-	0.02	-	-	-	-	0.51
Mobile Equipment Leasing	6.64	5.31	5.48	6.81	5.50	6.40	5.07	10.98	52.19

table continues...

Description	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10 to 12	Total
Fixed Equipment									
Ventilation	1.85	0.37	0.03	0.03	0.03	0.03	-	-	2.35
Electrical	0.26	0.54	-	-	0.50	-	-	-	1.30
Dewatering	0.26	-	0.02	-	-	-	-	-	0.28
Hoists	-	-	-	-	-	-	-	-	-
System Installation									
Ventilation	0.60	0.60	-	-	-	-	-	-	1.20
Electrical	0.62	0.38	0.06	0.06	0.64	0.06	0.06	0.11	1.98
Dewatering	0.02	-	-	-	-	-	-	-	0.02
Hoists	-	-	-	-	-	-	-	-	-
Other	0.15	-	-	-	-	-	-	-	0.15
Total	**16.40**	**8.34**	**6.09**	**7.40**	**7.15**	**6.88**	**5.49**	**11.56**	**69.31**

21.4 PASTE BACKFILL PLANT – CAPITAL COSTS

In keeping with the Order of Magnitude (±40%) cost estimate, the paste backfill plant has been estimated on a factored basis.

The mechanical equipment for the paste backfill plant has been individually listed and capital costs for equipment and installation have been assigned based on other AMEC paste backfill projects that have been recently estimated. Based on the installed costs of mechanical equipment, factors have been applied to obtain concrete, structural, architectural, electrical, piping and instrumentation costs.

Indirect costs have similarly been factored from previous paste backfill projects. It should be noted that Owner's costs, taxes and camp costs are not included. A contingency of 20% has been included in the CAPEX to reflect the uncertainties associated with a PEA-level study. The paste backfill plant capital cost estimate does not include EPCM costs or contingency. The capital costs are summarized in Table 21.11.

Table 21.11 Paste Backfill Plant Capital Costs

Item	Item Description	Estimated Costs ($ million)
1	Underground Mining	-
2	Civil	0.28
3	Concrete	0.75
4	Structural/Architectural	1.68
5	Mechanical	3.73
6	Piping (Services)	0.26
7	Piping (Paste Process)	0.86

table continues...

Item	Item Description	Estimated Costs ($ million)
8	Electrical	1.31
9	Instrumentation and Controls	0.49
10	Owner's Costs	-
11	**Indirects**	
12	Temporary Construction	0.56
13	Construction Equipment	0.15
14	Freight	0.37
15	Taxes	-
16	Start-up and Commissioning	0.19
17	First Fills and Spares	0.19
18	Camp	-
19	**Total**	**10.80**

21.5 OPERATING COST ESTIMATE

21.5.1 SUMMARY

The total LOM operating cost for the proposed mine is estimated at $66.46/t milled on average. The estimate includes mining, processing, tailing management, G&A and surface service costs.

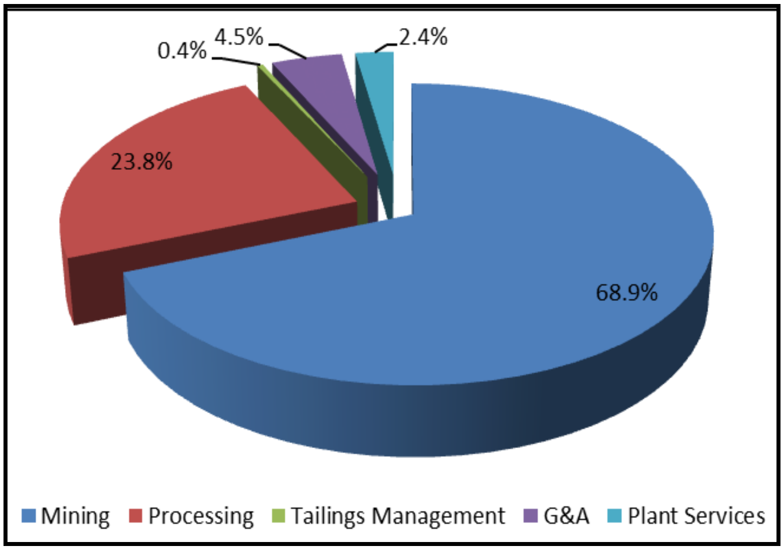
A total of 11,844,000 t mineralization from the underground mine will be processed during LOM based on the proposed mining schedule. The nominal annual process rate is approximately 1,204,500 t/a (LOM average annual process rate is approximately 1,077,000 t/a) or 3,300 t/d (LOM daily average rate is 2,950 t/d) at 365 d/a. The unit cost is estimated based on the LOM average mill feed rate. The accuracy for the estimate is expected to be within a range of ±40%. The breakdown operating cost estimates are presented in Table 21.12. Figure 21.1 shows the cost breakdown at the LOM average throughput.

Table 21.12 Overall Operating Cost

Area	LOM Average Unit Operating Cost ($/t milled)
Mining*	45.83
Processing	15.83
Tailing Management	0.25
G&A	2.97
Plant Services	1.60
Total	**66.48**

Notes: *Including backfill cost and mining power cost

Figure 21.1 LOM Average Operating Cost Distribution



21.5.2 MINE OPERATING COSTS

The mine operating costs include mining of mineralized material, expensed waste development, paste plant operating costs and backfill material, loading and haulage, maintenance costs, dewatering, heating, and mine overhead labor. All production mining was planned to be done by the Owner's mining crews.

The average LOM mine unit operating cost per tonne processed is summarized in Table 21.13. The $45.83/t mine operating cost excludes maintenance supervision and maintenance shop costs.

Table 21.13 LOM Underground Mine Operating Costs

Description	Cost ($)
Production Mining	27.14
Haulage	4.29
Paste Backfill	9.50
Underground Maintenance Labour	1.37
Propane	0.56
Mine Overhead	2.96
Total (per tonne processed)	**45.83**

The underground workforce includes LHD operators, truck operators, jumbo drillers, rock bolter operators, underground construction workers, paste backfill crews, service crews and nippers. Table 21.14 shows the annual cost associated with each pay grade including bonus and burden. Miner Level 5 personnel include the jumbo drillers, rock bolter operators, utilities leaders, and jackleg miners. Miner Level 4 personnel consist of service leaders and construction workers. Miner Level 3 personnel are the LHD operators. Miner Level 2 operators are primarily haul truck drivers. Miner Level 1 operators consist of nippers and helpers. The pay grade spread for miners in Year 4 is shown in Table 21.14 excluding the paste fill crew.

Table 21.14 Miner Pay Grades

Underground Mine Workforce (Year 4)	Rotation	Total Workers	Annual Rate ($)
Miner Level 5	7-days-on, 7-days-off	56	123,851
Miner Level 4	7-days-on, 7-days-off	16	115,297
Miner Level 3	7-days-on, 7-days-off	20	94,094
Miner Level 2	7-days-on, 7-days-off	16	82,446
Miner Level 1	7-days-on, 7-days-off	8	64,701
	Total	120	-

The direct unit advance cost of driving the various sized mineralized drift-and-fill headings as well as the expensed waste development was developed by first principals and is presented in Table 21.6. Cycle times were developed; then the equipment and crew required to accomplish the cycle were identified. The direct advance unit costs do not include maintenance labour, but do include field parts, diesel fuel, lubricants, and tire costs.

The direct unit advancement costs for the various sizes of development and production headings are shown in Table 21.15.

Table 21.15 Summary of Direct Heading Advance Costs

Size/Description	Owner/Contractor	Material	$/m
5.0 m high by 5.0 m wide	Owner	Waste	2,598
5.0 m highby 5.0 m wide	Owner	Mineralized	2,645
4.5 m high by 5.0 m wideUZ	Owner	Mineralized	2,399
4.5 m high by 3.0 m wide UZ Slash	Owner	Mineralized	751
4.5 m high by 8.0 m wide UZ	Owner	Mineralized	3,065
4.5 m high by 8.0 m wide UZ Bench	Owner	Mineralized	2,570
3.5 m high by 3.5 m wide LZ	Owner	Mineralized	1,912
5.0 m high by 5.0 m wide LZ	Owner	Mineralized	2,781

Table 21.16 shows the equipment and operating costs planned in all production areas with headings greater than or equal to 4 m in height.

Table 21.16 Equipment Costs for Headings Greater than or Equal to 4 m Height

Equipment Type	$/h
Two Boom Jumbo Drills	59.79
Rock Bolter	58.54
LHD – 5.4 m^3	96.22
ANFO Loader	58.59
Scissor Lift	53.50

One operator was assigned to each piece of equipment during each task for operating cost estimation. In addition, for cost estimation there were three personnel assigned to loading the face and installing services as the face advances. These personnel are the services leader, construction worker, and a helper.

In the LZ, approximately half of the production is planned in the upper portion where the drift-and-fill heading sizes are planned at 3.5 m by 3.5 m. Mining of these smaller headings was planned using a smaller drill jumbo and LHD along with handheld pneumatic rock bolting. Table 21.17 shows the equipment and operating costs planned in the LZ small production areas.

Table 21.17 Equipment Costs for LZ 3.5 m by 3.5 m Headings

Equipment Type	$/h
Single Boom Jumbo Drill	35.21
Jackleg for Bolting	0.43
LHD – 1.8 m^3	60.86
ANFO Loader	58.59
Scissor Lift	53.50

For the 3.5 m by 3.5 m headings, the same personnel assumptions were made regarding loading explosives and installing services. One operator was assigned to the drill jumbo, LHD, and the handheld jackleg.

Operating times for each piece of equipment was estimated from first principles. The jumbo operating time includes drilling and moving in and out of the heading. The rock bolter operating time includes drilling, installation of bolts, and moving in and out of the headings. The LHD operating time includes mucking the heading out as well as moving in and out of the heading. Operating consumables waste was estimated at 10%.

The ANFO explosives loader operating time includes time to load and move in and out of the heading to be blasted. The scissor lift operating time includes moving in and out of the heading as well as time to hang the services. Hanging of utilities was assumed every five rounds for cost estimating. Hanging of utilities was estimated to take three hours per heading to complete.

The planned underground paste fill crew consists of 3 people per shift (12 on the payroll). This crew would be responsible for preparing headings for backfill, installing and stripping utilities, and installing and stripping paste fill bulkheads as necessary. An underground forklift and scissor lift have been assigned to the paste crew for cost estimating. Table 21.18 shows the personnel planned at each pay grade for the underground paste crew. The annual rate shown includes bonus and 35% burden. For each shift a lead miner, a LHD operator, and a construction helper is planned.

Table 21.18 Underground Paste Fill Personnel Pay Grade

Pay Grade	Rotation	Total Workers	Annual Rate ($)
Miner Level 4	7-days-on, 7-days-off	4	115,297
Miner Level 3	7-days-on, 7-days-off	4	94,094
Miner Level 2	7-days-on, 7-days-off	4	82,446
	Total	12	-

A construction crew is incorporated in the operating cost to account for miscellaneous mine construction projects. This crew will be a day shift crew of two people at an hourly rate of $63.35, which equates to an annual operating cost of $922,000 per year. There would be four construction personnel on the payroll working a one-week-on/one-week-off schedule.

The secondary loading and hauling costs include operation of a 5.4 m^3 LHD, a 1.8 m^3 LHD (LZ only), and 40 t trucks for hauling to surface stockpiles within 100 m of the portal. The average one-way truck haul distance for the UZ was 1,467 m with a maximum LHD tramming distance of 300 m. The average one-way truck haul distance for the LZ was 3,435 m with a maximum LHD tramming distance of 300 m. For the operating cost estimate, one worker was assigned to each piece of equipment. The secondary loading and haulage costs do not include maintenance labor, but do include field parts, diesel fuel, lubricants, and tire costs. The secondary loading and hauling costs are summarized in Table 21.19.

Table 21.19 Secondary Loading and Hauling Costs

Location	Equipment	Equipment Cost ($/h)	Operator Cost ($/h)	Total Cost ($/h)
UZ	LHD – 5.4 m^3	96.22	58.14	154.36
	40-t Truck	87.15	45.30	132.45
LZ	LHD – 1.8 m^3	60.86	58.14	119.01
	40-t Truck	87.15	45.30	132.45

Auxiliary equipment costs are not included in the direct heading advance costs or the haulage costs. The field parts, diesel fuel, lubricants, and tire costs for auxiliary equipment was estimated at $0.90/t. This includes costs for personnel trucks for the maintenance crew, a grader, boom truck, forklift, ANFO loader, and a fuel/lube truck. The auxiliary equipment was planned to be intermittently run throughout each production shift.

Dewatering pump maintenance costs were estimated at $0.05/t and are included in the estimate. Propane costs for the mine air heaters were estimated at $630,000 per calendar year. The propane usage is seasonal, but on an annualized basis, the propane cost is $0.56/t based on a $2.20/gal price.

The average LOM underground maintenance labour cost is estimated at $1.37/t. For Year 4 onward, 14 underground maintenance personnel are included in the operating costs. Table 21.20 shows the maintenance personnel included in the mine operating cost and their associated annual cost. The annual rates shown include bonus and 35% burden. There will be three mechanics on each shift. The electrician and mechanic helper will work a 40 hour week.

Table 21.20 Underground Maintenance Labour Rates

Maintenance Labour (Year 4)	Rotation	Total Workers	Annual Rate ($)
Mechanic	7 days on, 7 days off	12	113,971
Electrician	40 hours/week	1	95,902
Mechanic Helper	40 hours/week	1	64,879
	Total Personnel	14	-

The underground mine overhead cost includes all the full time professional staff and supervision as well as the light vehicles that would be required to support them. Except for the front line supervisors, it was planned that all staff would be on a 40 hour work week with some overlap to account for shortages of senior level personnel onsite at a given time. Seven light trucks are planned to support the overhead staff. The overhead staff planned for the Project is shown in Table 21.21 along with the annual costs. The annual rates shown in Table 21.10 include bonus and 35% burden.

Table 21.21 Mine Overhead Personnel

Underground Mine Overhead Staff	Rotation	Total Workers	Annual Rate ($)
Chief Engineer	40 hours/week	1	186,000
Mine Engineer	40 hours/week	2	131,750
Surveyor	40 hours/week	2	108,500
Technician	40 hours/week	2	108,500
Chief Geologist	40 hours/week	1	155,000
Geologist	40 hours/week	1	124,000

table continues...

Underground Mine Overhead Staff	Rotation	Total Workers	Annual Rate ($)
Underground Geologist	40 hours/week	4	116,250
Mine Superintendent	40 hours/week	1	217,000
Safety	40 hours/week	1	170,500
Trainer	40 hours/week	2	170,500
Mine Captain	40 hours/week	1	201,500
Underground Supervisor	7 days on, 7 days off	4	170,500
Mine Clerk	40 hours/week	1	69,750

21.5.3 Paste Backfill Plant – Operating Costs

Plant electrical costs have been estimated from mechanical equipment using 76% plant availability and 80% load factor. Electrical cost of $0.07/ kWh was provided by Tintina.

Costs for cement and fly ash have been estimated based on an operating paste backfill plant in Nevada. It is assumed that the binder will consist of 50% fly ash and 50% Portland cement and that adequate paste strength can be obtained using 7% binder.

A full-time plant operator and part-time maintenance person have been included. Instrument repair and supervision are assumed to be supplied by mill concentrator personnel.

An allowance has been made for filter bags, consumables and mechanical spares. A share of overhead costs (5%) has been included to cover mill concentrator overheads such as heating, ventilation, compressed air, water supply, etc. A contingency of 10% has been included in the operating cost estimate to reflect the uncertainties associated with a PEA. The operating costs are summarized in Table 21.22.

Table 21.22 Paste Backfill Plant Operating Costs

Item	Description	Backfill Tonnage 740,000 t/a		Mining Tonnage 1,204,500 t/a	
		$ '000/a	$/t	$ '000/a	$/t
1	Paste Mix Plant Electrical Cost	336	0.45	336	0.28
2	Binder Cost	3,755	5.08	3,755	3.12
3	Surface Plant Operation/Maintenance Crews	571	0.77	571	0.47
4	Filter Bags, Consumables	189	0.26	189	0.16
5	Spare Parts (10% of Mechanical Equipment)	350	0.47	350	0.29
6	**Subtotals**	**5,201**	**7.03**	**5,201**	**4.32**
7	Overheads (5%)	260	0.35	260	0.22
8	Contingency (10%)	520	0.70	520	0.43
9	**Totals**	**5,982**	**8.08**	**5,982**	**4.97**

21.5.4 PROCESSING OPERATING COSTS

The estimated process operating cost is \$15.83/t milled or \$17.05 million per year. The estimate is based on a total mill feed of 11,844,000 t (LOM) or an average annual process rate of 1,077,000 t or 2,950 t/d and 365 d/a.

A summary of the plant operation costs is shown in Table 21.23.

Table 21.23 Summary of Process Operating Cost

Description	Labour Force	Annual Cost (\$)	Unit Cost (\$/t milled)
Labour Force			
Operating Staff	15	1,628,000	1.51
Operating Labour	22	1,556,000	1.45
Maintenance	15	991,000	0.92
Subtotal Labour Force	**52**	**4,175,000**	**3.88**
Major Consumables			
Metal Consumables	-	5,771,000	5.36
Reagent Consumables	-	1,672,000	1.55
Supplies			
Maintenance Supplies	-	1,396,000	1.29
Operating Supplies	-	246,000	0.23
Subtotal Consumables and Supplies	**-**	**9,085,000**	**8.43**
Power Supply	-	3,791,000	3.52
Subtotal Power	**-**	**3,791,000**	**3.52**
Total (Process)	-	**17,051,000**	**15.83**

The estimated labour force cost is \$3.88/t milled. A total of 52 people are estimated for the process operation, including 15 staff for management and professional services, 22 operators for operating, and 15 personnel for maintenance and assaying. The estimate is based on 12 hours per shift, 24 h/d, and 365 d/a.

The operating cost for the major metal consumables is estimated to be \$5.36/t milled. The metal consumables include mill and crusher liners and mill grinding media.

The estimated reagent cost is \$1.55/t milled. Reagent consumptions are estimated from the laboratory test results and comparable operations. The reagent costs are from the current budget prices from potential suppliers or Tetra Tech's database.

The maintenance supplies are estimated at \$1.30/t milled. The power cost is estimated based on the average power requirement of 54.2 MWh and a unit electric energy price of \$0.07/kWh for transmission line on site.

All operating cost estimates exclude taxes, permitting costs, or other government imposed costs, unless otherwise noted. The estimate includes:

- labour force, including supervision, operation, and maintenance; salary/wage levels are based on labour rates provided from the client; benefit burden of 35% including holiday and vacation payment, pension plan, various other benefits, and tool allowance
- power supply from the electrical transmission line
- crusher/mill liner and mill grinding media consumptions estimated from the BWi and the Tetra Tech's in-house database
- maintenance supply costs, including building maintenance costs, based on approximately 7% of major equipment capital costs
- laboratory supplies, service vehicles consumables and other costs based on Tetra Tech's in-house database and industry experience
- reagent costs based on the consumption rates from the test results and quoted budget prices or Tetra Tech's in-house database

21.5.5 General and Administrative Costs and Surface Services Costs

G&A costs are estimated to average $2.97/t over the course of the LOM. Tetra Tech and Tintina developed the costs.

The G&A costs include:

- labour cost for administrative personnel
- services expenses related to general administration, travelling, human resources, safety and security
- allowances for insurance, regional taxes and licenses
- sustainability, including environment, community liaison and engineering consulting.

A summary of the G&A costs are provided in Table 21.24.

Table 21.24 G&A Operating Costs

	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
G&A Labour Force			
G&A	13	1,499,000	1.39
G&A Hourly Personnel	4	258,000	0.24
Subtotal G&A Labour Force	**17**	**1,757,000**	**1.63**
G&A Expense			
General Office Expense	-	40,000	0.03
Computer Supplies including Software	-	50,000	0.04
Communications	-	40,000	0.04
Travel	-	30,000	0.03

table continues...

	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
Audit	-	100,000	0.09
Consulting/External Assays	-	50,000	0.05
Head Office Allowance: Marketing	-	100,000	0.09
Environmental	-	200,000	0.19
Insurance	-	433,000	0.40
Regional Taxes and Licenses Allowance	-	100,000	0.09
Legal Services	-	30,000	0.03
Warehouse	-	20,000	0.02
Recruiting	-	20,000	0.02
Medicals and First Aid	-	20,000	0.02
Relocation Expense	-	20,000	0.02
Training/Safety	-	50,000	0.05
Liaison Committee/Sustainability	-	40,000	0.04
Others	-	100,000	0.09
Subtotal G&A Expense	**-**	**1,443,000**	**1.34**
Total	**17**	**3, 200,000**	**2.97**

The surface service costs were estimated at $1.60/t milled and are detailed in Table 21.25. The costs include:

- labour costs for surface service personnel and maintenance workshop
- surface mobile equipment and light vehicle operations
- portable water and waste management
- general maintenance including yards, roads, fences, and building maintenance
- electrical power for site services, including lighting
- building heating.

Table 21.25 Surface Services Operating Costs

Surface Service	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
Surface Service Personnel	12	713,000	0.66
Small Vehicles/Equipment	-	30,000	0.03
Potable Water and Waste Management	-	100,000	0.09
Building Maintenance	-	50,000	0.05
Building Heating	-	315,000	0.29
Electrical Power	-	307,000	0.29
Road Maintenance	-	210,000	0.19
Total	**12**	**1,725,000**	**1.60**

21.5.6 TAILINGS MANAGEMENT COST

The estimated operating costs of the TMF is $0.25/t milled.

22.0 ECONOMIC ANALYSIS

This updated PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Tetra Tech has prepared an economic evaluation of the Project based on a pre-tax financial model.

As of April 26, 2013, Tetra Tech's long-term consensus copper price used in the base case was US$3.05/lb.

The pre-tax financial results are:

- 30.5% IRR
- 3.6-year payback on the US$218 million initial capital costs
- US$218 million NPV at an 8% discount rate.

Tintina commissioned PwC in Vancouver, BC, to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes (Section 22.4).

The following post-tax financial results were calculated:

- 20.2% IRR
- 4.7-year payback on the US$218 million initial capital costs
- US$110 million NPV at an 8% discount rate.

Sensitivity analyses were conducted to analyze the sensitivity of the Project merit measures (NPV and IRR) to the main inputs.

22.1 Pre-tax Model

22.1.1 Mine/Metal Production in Financial Model

The life-of-project average material tonnages, grades and metal production are shown in Table 22.1.

Table 22.1 Metal Production from the Black Butte Mine

Description	Value
Total Tonnes to Mill ('000)	11,844
Average Annual Tonnes to Mill ('000)	1,077
LOM (years)	11
Average Grade	
Copper (%)	3.11
Total Production	
Copper ('000 lb)	716,014
Average Annual Production	
Copper ('000 lb)	65,092

22.1.2 BASIS OF FINANCIAL EVALUATIONS

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage processed, head grades, and recoveries.

Copper payable values were calculated based on base case metal price. Net invoice value was calculated each year by subtracting the applicable refining and smelting charges from the payable metal value. At-mine revenues are then estimated by subtracting transportation and insurance costs. Unit operating costs for mining, processing, power, fuel, and G&A were applied to annual mined/milled tonnages to determine the overall operating cost which was deducted from the revenues to derive the annual operating cash flows.

Initial and sustaining capital costs as well as working capital have been incorporated on a year-by-year basis over the LOM. Mine reclamation costs are also applied to the capital expenditure. Capital expenditures are then deducted from the operating cash flow to determine the net cash flow before taxes.

Initial capital expenditures include costs accumulated prior to first production of concentrate; sustaining capital includes expenditures for mining and processing additions, replacement of equipment, and tailings embankment construction.

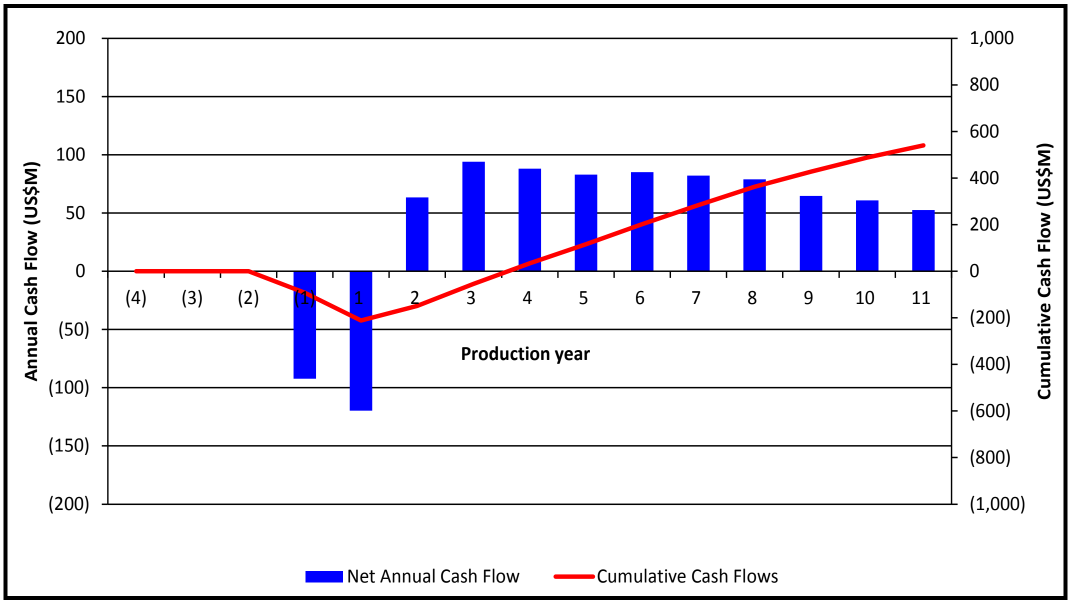
The pre-production period is assumed to be three years.

Working capital is assumed to be three months of the annual operating cost and fluctuates from year to year based on the annual cost. The working capital is recovered at the end of the LOM.

Sustaining capital costs were estimated at US$115 million, and mine closure and reclamation costs were estimated at US$14 million.

The undiscounted annual net cash flow (NCF) and cumulative net cash flow (CNCF) are illustrated in Figure 22.1.

Figure 22.1 Pre-tax Undiscounted Annual and Cumulative Net Cash Flow



22.2 SUMMARY OF FINANCIAL RESULTS

Tetra Tech evaluated the base case using a consensus copper price of US$3.05/lb.

The pre-tax financial model was established on a 100% equity basis, excluding debt financing, and loan interest charges. The financial results for the base case and for alternative cases are presented in Table 22.2.

Table 22.2 Summary of Financial Results

Copper Price ($/lb)	Pre-Tax			Post-Tax		
	IRR (%)	NPV at 8% (millions $)	Payback (Years)	IRR (%)	NPV at 8% (millions $)	Payback (Years)
2.5	11.3	28	6.2	5.5	-21	8.3
3.05*	30.5	218	3.6	20.2	110	4.7
3.5	44.7	373	2.8	30.4	210	3.6

Note: *Base case copper price.

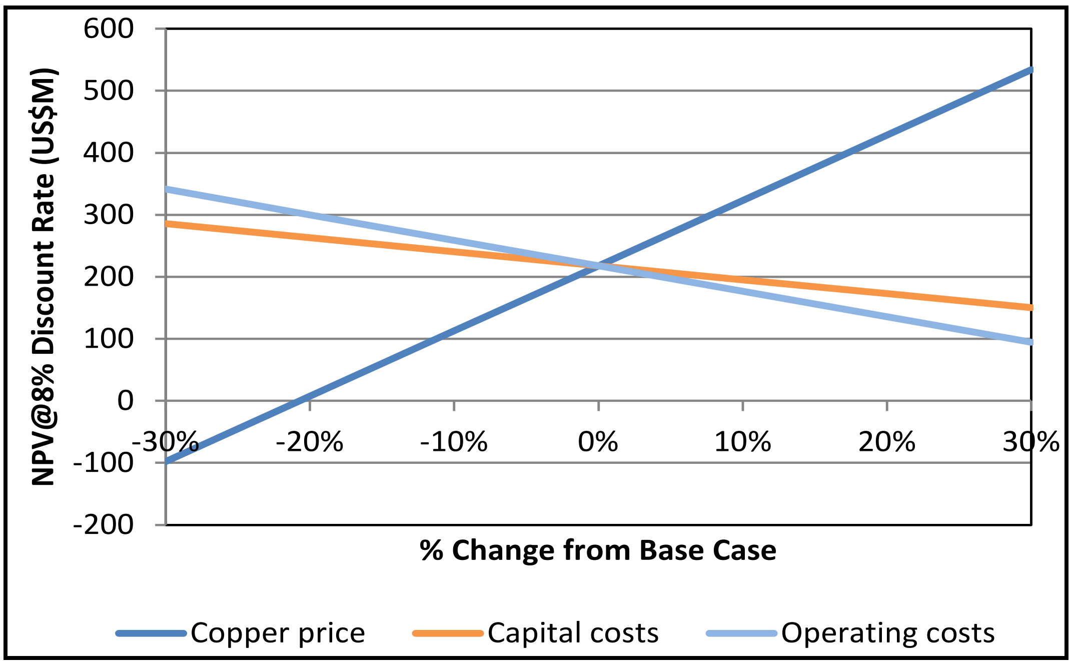
22.3 SENSITIVITY ANALYSIS

Tetra Tech investigated the sensitivity of NPV and IRR to the key Project variables. Using the base case as a reference, each of key variables was changed between -30% and +30% at a 10% interval while holding the other variables constant. The following key variables were investigated:

- copper price
- capital costs
- operating costs.

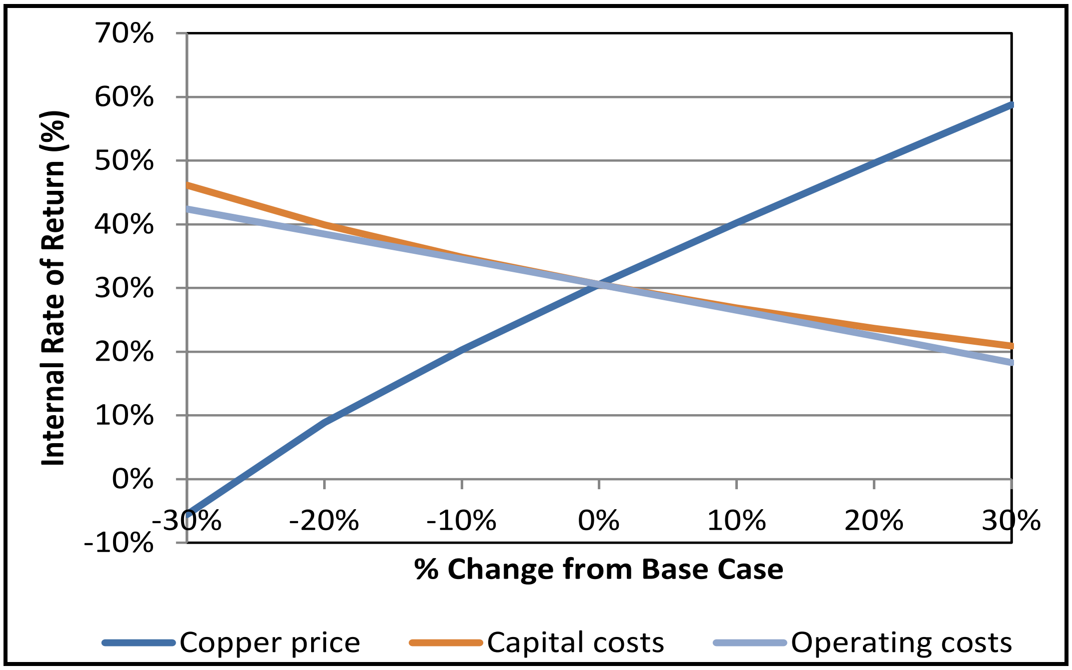
The Project's pre-tax NPV, calculated at 8% discount rate, is most sensitive to copper price and, in decreasing order, operating costs and capital costs (Figure 22.2).

Figure 22.2 Pre-tax NPV Sensitivity Analysis



As shown in Figure 22.3, the Project's pre-tax IRR is most sensitive to the copper price followed by capital costs and operating costs.

Figure 22.3 Pre-tax IRR Sensitivity Analysis



22.4 POST-TAX FINANCIAL ANALYSIS

Tintina commissioned PwC to prepare a tax model for use in the post-tax preliminary economic assessment of the Project.

The following general tax regime was recognized as applicable at the time of report writing.

22.4.1 US FEDERAL INCOME TAX REGIME

For US federal income tax purposes, in accordance with the Internal Revenue Code of 1986 as amended (IRC), a taxpayer is required to calculate taxes under both the regular corporate tax system and the Alternative Minimum Tax (AMT) system and pay whichever method results in the higher amount of taxes.

The statutory US federal corporate income tax rate is 35% and the tax rate under AMT is 20%. The Montana state income tax rate is 7%. As state taxes are deductible for federal tax purposes, the applicable combined regular statutory income tax rate for the Project will be 39.55% of regular taxable income. However, the Project will be subject to either regular tax or AMT over the life of the mine as calculated in each year.

Net operating losses generated in a given year may be carried forward for 20 years and applied to taxable income when it arises, or carried back 2 years and applied against taxable income from the Project's prior years. The IRC also provides certain deductions to provide an incentive for investments by mining companies, including depletion and development expenditures.

Table 22.3 US Federal Tax Rate

Taxable Income	Tax
$0 to $50,000	15%
$50,000 to $75,000	25%
$75,000 to $100,000	34%
$100,000 to $335,000	39%
$335,000 to $10,000,000	34%
$10,000,000 to $15,000,000	35%
$15,000,000 to $18,333,333	38%
$18,333,333 to ∞	35%

22.4.2 DEPLETION

Generally speaking, depletion, like depreciation, is a form of cost recovery. Just as the owner of a business asset is allowed to recover the cost of an asset over its useful life, a miner is allowed to recover the cost of the mineral property. Depletion is taken over the period that minerals are being extracted.

For federal income tax purposes, two forms of depletion are allowed: cost depletion and percentage depletion. The taxpayer is required to use the method that will result in the greatest deduction.

COST DEPLETION

Cost depletion is calculated based on the adjusted basis of the depletable property, multiplied by the units of ore produced over the proven and probable reserves. For purposes of this report, it was assumed that the adjusted basis of the depletable property was zero. Accordingly, the cost depletion deduction would also be zero. Therefore, the Project would only be entitled to deductions under percentage depletion.

PERCENTAGE DEPLETION

Under the percentage depletion method, a flat percentage of 15% of adjusted gross income from copper mining is used to calculate the depletion allowance. However, the deduction for depletion cannot exceed 50% of the adjusted taxable income from the activity. This limitation is computed without regard to the depletion allowance. The amount of the deduction allowable under percentage depletion is not limited by the basis of the property, except for AMT purposes. Thus, even though the basis of the property is reduced by the amount of depletion taken, if the basis becomes zero, the depletion based on the percentage of adjusted gross income may continue to be claimed for tax purposes.

22.4.3 MONTANA CORPORATE LICENSE TAX REGIME

The corporation license tax is a franchise tax levied on corporations for the privilege of doing business in Montana. Corporations making a "water's edge" election are required to pay tax at a rate of 7% of net income earned in Montana.

In computing net income, gross income is the same as for federal corporate tax purposes. Allowable deductions include all ordinary and necessary business expenses, certain losses and depreciation of assets, resource depletion allowance, interest paid on business debts, taxes paid (except all taxes measured by net income or profits), certain charitable contributions, certain energy-related investments, and net operating losses.

Corporations conducting business that is taxable both within and without the state (multi-state corporations) are required to allocate income to Montana based on an equally-weighted, three-factor apportionment formula where sales, property, and payroll are the three factors.

Net operating losses generated in a given year may be carried forward for seven years and applied to taxable income when it arises, or carried back three years and applied against taxable income from the Project in those years.

22.4.4 MONTANA METAL MINES LICENSE TAX

Montana mining operations in which metal or gems are extracted are subject to a license tax, which is based on the gross value of the product.

The gross value to which the tax rate is applied is the monetary payment the mining company receives from the metal trader, smelter, roaster, or refinery. This is determined by multiplying the quantity of metal received by the metal trader, smelter, roaster, or refinery by the quoted price for the metal, and then subtracting basic treatment and refinery charges, quantity deductions, price deductions, interest and penalty, metal impurity, and moisture deductions as specified by contract between the mining company and the receiving metal trader, smelter, roaster, or refinery. Deductions also are allowed for the cost of transportation from the mine or mill to the smelter, roaster, or refinery.

Concentrate shipped to a smelter, mill, or reduction work is taxed at 1.81% of gross value over $250,000. Gross value under $250,000 is exempt from metal mines license taxation. They instead pay the Resource Indemnity and Ground Water Assessment Tax (RIGWAT) at a rate of one-half (1/2) of one (1%) percent.

There is no provision in the legislation to carry losses forward to offset future profits in the mining licence tax calculation.

Montana Metal Mines Gross Proceeds Tax

A yearly ad-valorem tax is imposed on the gross proceeds of metal mines. Gross proceeds means the monetary payment or refined metal received by the mining company from the metal trader, smelter, roaster, or refinery, determined by multiplying the quantity of metal received by the quoted price for the metal and then subtracting basic treatment and refinery charges, quantity deductions, price deductions, interest and penalty, metal impurity, and moisture deductions as specified by contract.

The taxable value of metal mines is equal to 3% of annual gross proceeds. This amount is subject to local mill levies in the jurisdiction in which the taxable value of the mining operation is allocated.

Mines that produce less than 20,000 tons of ore in a year are exempt from property taxation on one-half of the merchantable value.

At the long-term copper price of US$3.05/lb used for this study, total estimated taxes payable on Black Butte profits are US$208 million over the 11-year mine life. The components of the various taxes that will be payable are shown in Table 22.4.

Table 22.4 Components of the Various Taxes

Tax Component	LOM Amount (US$ million)
Montana Metalliferous Mines License Tax	30
Metal Mines Gross Proceeds Tax Rate	50
Montana State Income Tax	23
Federal Income Tax	105
Total Taxes	**208**

The base case post-tax financial results are summarized in Table 22.5.

Table 22.5 Summary of Post-tax Financial Results

Description	Value
Copper Price (US$/lb)	3.05
Net Cash Flow (US$ million)	333
Discounted Cash Flow NPV (US$ million) at 8%	110
Payback (years from start of mill operations)	4.7
IRR (%)	20.2

22.5 ROYALTIES

A 2% NSR was applied in the financial analysis.

22.6 SMELTER TERMS

Typical smelter terms have been applied for the delivery of copper concentrate to an East Asian smelter.

Copper concentrate contracts will generally include payment terms as follows:

- copper – pay 100% of content less 1.0 unit at the London Metal Exchange (LME) price for Grade A copper less a refining charge of US$0.075 per accountable pound. The refining charge is not subject to price participation.
- treatment charge – US$75/dmt of concentrate delivered.
- penalty charge – US$3/dmt of concentrate for each 0.1% arsenic over 0.2%.

22.7 TRANSPORTATION LOGISTICS

Transportation costs for the copper concentrate are as follows:

- trucking – US$21.51/wmt
- rail – US$55.00/wmt
- port storage and handling – US$21.00/wmt
- ocean transport to Asian port – US$42.00/wmt
- moisture content – 9%.

The trucking cost of US$21.51/wmt is based on:

- $4.00/gal fuel cost
- current dollars
- 146,000 tons per year
- 24 h/d, 365 d/a operation
- load and unload times not to exceed 20 min each.

22.7.1 INSURANCE

An insurance rate of 0.15% was applied to the provisional invoice value of the copper concentrate.

23.0 ADJACENT PROPERTIES

Within Section 34, Township 12 North, Range 6 East, are a number of patented claims and unpatented load mining claims controlled by Holcim, a large Swiss cement company. Holcim operates a small open-cut iron oxide mine on the patented claims and produce only a few thousand tonnes of iron oxide mineralized material per year from the mine. The mine only operates during fair weather months. The iron oxide is trucked to Holcim's cement plant near Three Forks, Montana. Past drilling by CAI has shown that the iron oxide concentrations are gossans formed from weathering of the USZ. RMI is not aware of any resources located on adjacent properties.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information to add to this technical report.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 GEOLOGIC INTERPRETATION

The copper-cobalt mineralization at Black Butte has been recognized since the early to mid 1980s. A significant amount of work was completed by major mining companies in developing a geologic model and testing that model by a number of core drilling campaigns.

The Black Butte bedded sulphide accumulations best fit a shale-hosted massive sulphide deposit type model. The host rocks contain no volcanic component and in terms of setting and geometry, the sulphide occurrences are quite similar to typical Proterozoic and Phanerozoic shale-hosted zinc and lead rich deposits. However, the high concentrations of copper, cobalt, and barium are unusual in shale hosted sulphide occurrences. Mt. Isa (Perkins 1984) and Walford Creek (Rohrlach et. al. 1998) in Australia make reasonable analogies (Zieg 1992). Most geologists interpret the genesis of the Black Butte sulphides as having formed at sysnsedimentary hydrothermal vents sites during deposition of the host shale. Sulphides are involved in soft sediment folding, and sulphide accumulations include evidence of vent biota grown over subaqueous hydrothermal hot springs. These are intricate growths of tubes interpreted as having formed around algal or bacterial filaments and are most abundant with greater sulphide accumulations (McGoldrick and Zieg 2004).

The Black Butte exploration model is a middle Proterozoic synsedimentary subaqeous hydrothermal vent field developed at structural intersections during prolonged synsedimentary extensional faulting along the northern margin of the Helena embayment.

25.2 MINERAL RESOURCE ESTIMATES

Historic and 2010-2012Tintina drilling data were used to update mineral resource estimates for the Johnny Lee UZ, Johnny Lee LZ, and Lowry MZ. The historic drilling data collected by CAI, UII, and BHP have been validated by twin hole drilling and various spatially paired comparisons between the older and newer data. In general, spatial pairing of older data with newer data that is supported by QA/QC results shows that the older data may be biased low when compared to the new data. This apparent low bias may be associated with differing analytical methods that were used. Most of the older samples were digested using aqua regia while the Tintina samples used a four acid digestion, which may have put more copper into solution. Currently, Tintina's drilling data used to model the Johnny Lee UZ, Johnny Lee LZ, and Lowry MZ comprise 88%, 74%, and 85% of the total.

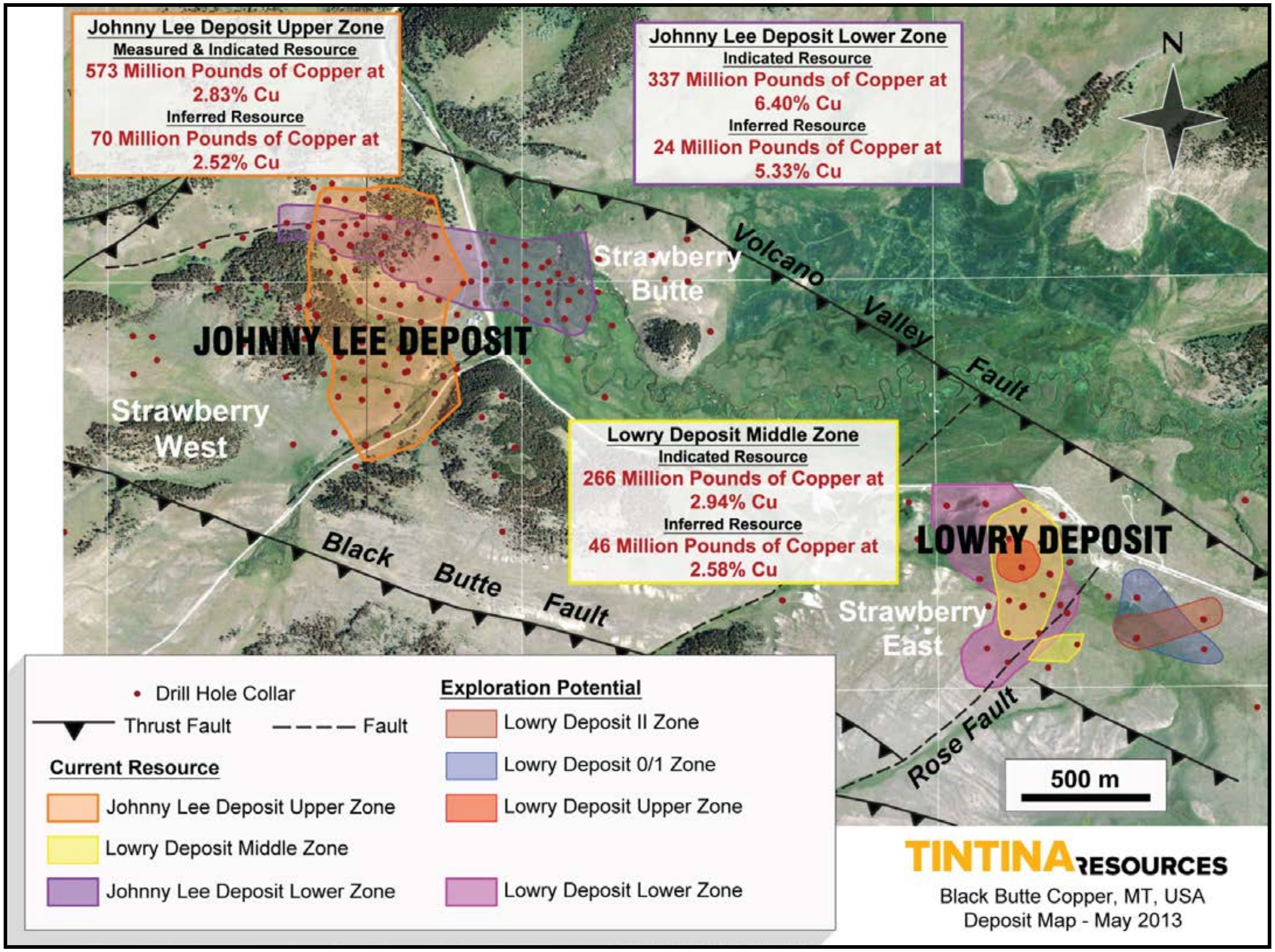
Inverse distance block grade estimation methods were used in conjunction with a "relative elevation" option which allows for drill hole samples to be selected based on relative distances from hanging wall and footwall surfaces. This method allows for much more stratigraphic control in the estimation process. Table 25.1 summarizes the current Black Butte Mineral Resource inventory Note that Lowry resources were not used for economic contributions for this updated PEA. Only Johnny Lee UZ and LZ resources were used. Figure 25.1 is a plan map showing the distribution of the three mineral resource areas.

Table 25.1 Undiluted Black Butte Mineral Resources

Zone	Tonnes ('000)	Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
Measured and Indicated									
Johnny Lee UZ[1]	9,179	2.83	0.12	0.008	15.7	573	24.9	2.5	4,642
Johnny Lee LZ	2,387	6.40	0.03	0.304	4.5	337	1.7	23.3	345
Lowry MZ	4,099	2.94	0.10	0.006	15.1	266	9.0	0.8	1,990
Total Measured and Indicated	**15,665**	**3.40**	**0.10**	**0.053**	**13.9**	**1176**	**35.6**	**26.6**	**6,977**
Inferred									
Johnny Lee UZ[1]	1,255	2.52	0.10	0.008	15.2	70	2.8	0.3	613
Johnny Lee LZ[2]	205	5.33	0.03	0.207	4.1	24	0.1	1.4	27
Lowry MZ[1]	801	2.58	0.10	0.008	14.1	46	2.0	0.2	363
Total Black Butte Inferred	**2,261**	**2.80**	**0.09**	**0.026**	**13.8**	**140**	**4.9**	**1.9**	**1,003**

Notes: [1] A copper cut-off grade of 1.6% was used.
[2] A copper cut-off grade of 1.5% was used.
Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Figure 25.1 Black Butte Mineral Resource Areas



25.3 METALLURGICAL STUDIES

Preliminary test work and mineralogy studies were performed on Master Composites prepared from samples of typical massive sulphide UZ and LZ mineralization.

The mineralogical studies on the UZ Master Composite showed that chalcopyrite is the predominant copper mineral, with a small amount of tennantite present. Approximately 92% of the copper is contained in chalcopyrite whereas tennantite which may contribute arsenic to the copper concentrate contains 7.5% of the copper. The cobalt minerals include cobaltite, carrolite, and bravoite. Cobaltite contains 42% of the cobalt. Pyrite represents 45% of the sample mass and about 45% of the cobalt was contained in pyrite at an average concentration of 1,320 ppm.

The mineral distribution by class of association shows a very complex sulphide mineralization. At a grind size of 62 µm, chalcopyrite is poorly liberated and strongly associated with pyrite and complex multiphase associations.

Cobalt minerals are extremely poorly liberated and strongly associated with pyrite. The association between chalcopyrite and cobalt minerals is very weak indicating that the recovery of cobalt to the copper concentrate will likely be poor. The same is the case for silver.

The results of these investigations indicated that the UZ copper-cobalt mineralization is very fine grained and complex requiring a primary grind level of 80% passing 38 µm and a rougher concentrate regrind of 80% passing 8 µm for effective liberation and recovery of copper minerals to a marketable concentrate using otherwise conventional flotation conditions including rougher flotation at pH 9.5 with SIPX and 3418A as collectors followed by regrinding and three stage cleaning at pH 11.0 with low cyanide additions.

The mineralogy study of the LZ Master Composite indicated the chalcopyrite liberation at a grind size of 85% passing -53 µm is very high at 88%; this is shown in the excellent results of the flotation tests which achieved a 96.6% copper recovery in the locked cycle test. The test results confirm that the sulphide mineralization is much coarser grained and less complex than the UZ material. The copper grade of the LZ composite at 4% is much higher than the UZ composite of 2.2% but the cobalt and silver grades are much lower.

The results of the locked cycle tests and the projected metallurgical recoveries for the composites are shown in Table 25.2.

Table 25.2 Projected Metallurgy for Johnny Lee UZ and LZ Composites

Zone – Parameter	Weight (%)	Assays		Distribution	
		Ag (g/t)	Cu (%)	Ag (%)	Cu (%)
UZ					
Cleaner Concentrate	7.8	16.0	23.3	9.0	81.0
Head Grade	-	15.0	2.2	-	-
LZ					
Cleaner Concentrate	14.3	8.7	27.0	22.6	96.6
Head Grade	-	5.5	4.0	-	-

The results of the locked cycle tests on the UZ composite are consistent with the conclusions of the mineralogy studies. The projected metallurgy based on the results of a single locked cycle tests is only an indication of the plant metallurgy for material of this composition.

The copper recovery estimates for the mine production plan, which has higher grades than the metallurgy composites of the respective zones, were adjusted upwards from the locked cycle test results to reflect the higher copper grades. Test work to support the metallurgy projections for the higher grade material was not performed due non-availability of proper samples.

Minor element analysis of the concentrate indicated that it contained very low levels of potentially deleterious elements. The concentrations of arsenic at 0.4% and at 0.2% cobalt and nickel are slightly elevated and may incur minor penalties. The concentrate does not contain payable silver and cobalt values.

From the results of the mineralogy and metallurgy studies, it can be concluded that the production of payable levels of cobalt and silver to the concentrates is very unlikely.

25.4 RISKS AND UNCERTAINTIES

Copper grades within the drilled mineralized horizons tend to be quite variable even within short distances, but in general, copper is typically in excess to 1 to 2% copper in relatively persistent lenses, locally spiking above 10%. Close spaced drilling from underground drill stations will be required to predict local grades.

As discussed in Section 13.0, the current metallurgical work suggests that there may be several different mineralized material types within the Johnny Lee sulphide system. Tintina's geologic staff will need to coordinate with their metallurgical consultants to determine if an mineralized material type model can be constructed which will allow for more confidence in potential recoverable copper metal.

RMI and Arthur H. Winckers are not aware of any other significant risks associated with the current Indicated and Inferred Resources. At this juncture there does not appear to be anything that would preclude the permitting of this project. Ongoing testing and cost estimates associated will PEA will provide additional insight into the potential risks and uncertainties associated with the Project.

25.5 RECOVERY METHODS

A 3,300 t/d process plant has been designed for the Project to process massive sulphide mineralization. The proposed process plant will consist of one stage of crushing, a SAG mill/ball mill/tower mill/pebble crusher (SABC) primary grinding circuit, copper rougher flotation, followed by rougher flotation concentrate regrinding and cleaner flotation processes. A copper concentrate will be produced from the plant.

The equipment that has been incorporated into the design is widely operated in the industry.

25.6 MINING RISKS

AMEC notes the following mining risks:

- A comprehensive hydrogeological assessment of the UZ and LZ has not been completed. Inflows higher than the planned 500 gpm could affect:
  - the LZ development rates, LZ production rates, and LZ mine operating costs
  - the overall mine operating cost estimate.
- A high mine development rate of 5 m/d is planned in order to accelerate access of the higher grade material in the LZ. A number of factors could reduce this development rate.
- The selection and design of the mining methods for the deposit are based on limited geotechnical information. The excavation and ground support design are preliminary estimates based on rock core photographs. Rock strength testing and rock mass classifications have not yet been completed and may have an impact on operating costs and schedules.
- Material testing has not been done as a basis for the paste fill plant design and operating cost estimates.
- The LZ full production rate of 800 t/d has been planned for the mine life. A lower production rate may be necessary during the second half of the mine life and detailed mine planning is recommended in the next phase of the Project.

25.7 ENVIRONMENTAL

The environmental baseline study process for the Project is well underway, and the mine permitting requirements are well defined. Mine planning in response to data collection is ongoing. The principal challenges for the Project are waste and water management.

25.8 ECONOMIC ANALYSIS

Tetra Tech performed a base case, 100% equity, pre-tax economic analysis of the Project, based on the following:

- price of copper – US$3.05/lb
- total LOM production of 11,844,000 t of mineralized material
- average grade of 3.11% copper and average process recovery of 88.3%
- total of 716,014,000 lb of copper recovered over the 11-year LOM and 65,092,000 lb of copper recovered per year
- LOM payable copper value of US$2,081,979,000 with an on-site operating cost estimate of US$787,370,000 and a total LOM capital cost estimate of US$346,007,000.

The resulting pre-tax discounted cash flow NPV at 8% is $217,926,000, the IRR is 30.5%, and the payback period is 3.6 years.

The resulting post-tax discounted cash flow NPV is 8% at $109,967,000, the IRR is 20.2%, and the payback period is 4.7 years.

In addition to the possible impact on overall economics that could result from variations in process recovery or mineralized material grades, sensitivity analyses show that the Project economics are particularly sensitive to changes in copper price with lesser influence from operating and capital costs. It is apparent that the copper price would have a very significant impact on profitability of the Project.

This updated PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

26.0 RECOMMENDATIONS

26.1 GENERAL

Tetra Tech recommends that Tintina continue investigating the Property and proceed to the next phase of study.

26.2 GEOLOGY

RMI highly recommends that Tintina continue to obtain assay certificates and QA/QC data for the CAI, UII, and BHP drilling campaigns. It is RMI's opinion that data collected by these companies is valid based on comparisons with Tintina drilling data.

Tintina should collect representative bulk density samples from unmineralized hanging wall material, massive sulphide zones, and unmineralized footwall material from underground exposures. These determinations would be used to corroborate bulk densities that have been estimated from diamond core samples. The cost for these tasks is nominal provided that Tintina geologists or geologic technicians perform the task. The costs associated with obtaining these confirmatory bulk density determinations should not amount to more than US$2,500.

Eventually Tintina will need a more detailed topographic map for the Project area. The current surface used by RMI fits the surveyed drillhole collars reasonably well and this is not a material issue regarding the Inferred Resources that are the subject of this report.

In RMI's opinion, a detailed aerial survey and subsequent digital terrain model will cost between US$15,000 and US$30,000.

26.2.1 JOHNNY LEE UZ

If Tintina obtains a permit for the exploration data, RMI recommends that Tintina consider drilling a series of infill core holes from underground drill stations. The goal of this infill program is to test the current grade model and, more importantly, to provide additional information for mine planning. This drilling program would consist of drilling between 5,000 and 10,000 m for selected areas of the mineralized zone. The program could be carried out over a five-year period with an estimated cost ranging from US$1.5 million to US$3.0 million. The program would be contingent upon recommendations from future feasibility level studies.

RMI recommends that Tintina complete detailed geologic mapping and sampling within all exposed mineralized zones encountered while driving the exploration decline and mineralized material drift. A significant portion of the cost for this activity should be considered as a sunk cost (salary of Tintina's geologic staff). The cost for face/back samples is estimated to range between US$10,000 and US$20,000.

RMI recommends that Tintina conduct grade/tonnage reconciliation studies for material that will be mined from the mineralized material drift. The actual mined material will be compared with the current exploration model. The cost for this activity is nominal.

26.2.2 Johnny Lee LZ

Similar to the Johnny Lee UZ, RMI recommends that Tintina drill several infill holes from the end of the exploration decline down to the Johnny Lee LZ. These holes would test the current resource model and provide additional information for mine planning purposes. This program would be contingent upon recommendations from future feasibility level studies. The program could be staged over a period of years during mine development. The recommended meterage to complete this initial infill program is 1,500 to 3,000 m with an estimated cost ranging between US$450,000 and US$900,000.

26.3 Metallurgy and Mineral Processing

26.3.1 Metallurgy

For the next level of the study, the following recommendations have been made:

- The optimum primary grind size needs to be re-evaluated because the mineralogy study of the test feed sample indicated that a finer primary grind may be beneficial.
- The mineralogy data suggest that a coarser regrind level may be acceptable, compared to the regrind size of 8 µm used tested in the previous locked cycle tests. Further tests are required to optimize the regrind size.
- Alternate reagent schemes and processes to optimize silver and cobalt recoveries should be explored.
- Flowsheet optimization tests for processing the blended mill feed from the upper zone and the lower zone are recommended, including flash flotation in the primary grinding circuit and the introduction of a first cleaning stage prior to regrinding.
- Further variability flotation test work and mineralogy studies are recommended to investigate the effect of the mineralization types on metallurgical performance.
- Determinations of the design related parameters for next phase study, including comminution circuit design parameters, and filtration rates and thickening rates for concentrate and tailings dewatering. Pilot plant tests may be required to generate the samples for the tests.
- Comminution tests work to determine grinding circuit design parameters and tests to determine the variability in mineralized material hardness across the

deposit is also recommended. This will include SAG mill design tests on HQ core a Verti Mill design test as the final primary grinding stage and regrinding mill tests.

The cost estimate for the above scope of work is summarized in Table 26.1.

Table 26.1 Estimated Costs for Feasibility Metallurgical Test Work

Activity	Estimated Cost (US$)
Bench Scale Tests	600,000
Pilot Plant Run	75,000
Comminution Studies	100,000
Equipment Vendor Tests	25,000
Mineralogy Studies	50,000
Total	**850,000**

26.3.2 Mineral Processing Design

Further optimizations on plant designs including primary comminution circuits, regrinding circuit and layout are recommended. The costs associated with the optimizations will be part of the costs for the next phase of study.

26.4 Tailings Management Facility

It is recommended that a geotechnical site investigation be completed. This investigation will characterize the foundation conditions for the impoundment, with a focus on determining the depth and geotechnical characterization of the overburden in the area. Soil characterization would be determined through in-situ density testing as well as laboratory index testing of collected samples. Rock core would be logged with a focus on rock mass quality, hydraulic conditions would be determined through the use of in-situ hydraulic testing during drilling, and rock core samples would be collected for laboratory strength testing.

The results of the geotechnical investigation will be used to establish the volumes of available material for the infill borrow construction method. The TMF design will be altered based on site conditions to accommodate several factors including: terrain shaping, embankment fill sources, and seepage control measures.

The data collected during the geotechnical site investigation would form the basis for the feasibility design of the waste and water management facilities.

The cost to complete the above work is estimated to be US$540,000.

26.5 MINING

AMEC recommends the following during the next level of the study:

- A comprehensive hydrogeological and geotechnical assessment of the Johnny Lee deposit is required. The hydrogeological and geotechnical assessment, including drilling additional holes could cost $400,000 to $600,000.

  A detailed geotechnical assessment, including the determination of rock strengths and behavior, is required to confirm the selected mining methods and the assumptions used in the design and support of mine excavations. This information will be needed to support more accurate mining cost estimates in future studies.

  The detailed hydrogeological assessment is required to:

  - Predict ground water flows in the vicinity of the VVF and how it could affect LZ development, the LZ production rate, and LZ mine operating costs.
  - Complete a detailed mine water balance estimate to be used as a basis to design the mine dewatering system.
  - Estimate the affect of potential mine inflows on mine development rates and costs.
- An assessment of the tailings from the process plant will be required as a basis for the paste plant design, detailed mine planning, and cost estimates. The paste backfill material assessment including tailings testing, binder, and a mix design option evaluation, could cost from $80,000 to $400,000 depending on many variables.
- A detailed mine plan is needed as a basis for confirming estimated production rates and operating costs. Detailed mine planning could cost from $100,000 to $300,000 depending on geologic model changes due to additional drilling and other variables.

26.6 CAPITAL COST

The source and availability of labour for construction should be verified in the next phase of the study.

26.7 ENVIRONMENTAL

Tintina will continue to collect data for environmental baseline studies and succeed with permitting efforts by completing the thorough, well-defined program they have initiated, in consultation with key stakeholders and regulatory agencies. The estimated cost for this recommended work is $650,000 to $800,000 for environmental baseline studies excluding the EIS or mine operating permit.

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28.0 CERTIFICATES OF QUALIFIED PERSON

28.1 ARTHUR H. WINCKERS, P.ENG.

I, Arthur H. Winckers, P.Eng., of North Vancouver, British Columbia, do hereby certify:

- I am the President of Arthur H. Winckers & Associates Inc. with a business address of 4345 Raeburn Street, North Vancouver, British Columbia V7G 1K1.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of the Technical University of Delft in the Netherlands with a M.Sc. degree in Mining Engineering with a specialty in Mineral Processing (1965). I am a Registered Professional Engineer in the Province of British Columbia (#8693), and a Registered Member of the CIM. I have more than 40 years of professional experience in the mineral processing industry, as Mill Superintendent at a number of base-metal concentrators operated by Cominco Ltd., as Senior Metallurgist for Teck Cominco Limited and since 2002 as an independent mineral processing consultant in due diligence evaluation of base-metal and gold projects, design and management of metallurgical studies, process/mill design, pre-feasibility studies and project management. My experience includes the management of a number of metallurgy studies to develop the flowsheet and design criteria for the processing of volcanogenic massive sulphide deposits such as the San Nicolas deposit in Mexico, the Hackett River deposits in Nunavut and the Kutcho Creek deposit in northern BC. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.5, 13.0, 25.3, 25.4, 26.3.1, 27.0 (geology and metallurgy only), and 28.1 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report. I co-authored the technical report entitled "Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana" and dated August 30, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at North Vancouver, British Columbia.

Original document signed and sealed by
Arthur H. Winckers, P.Eng.

Arthur H. Winckers, P.Eng.
President
Arthur H. Winckers & Associates Inc.

28.2 MICHAEL J. LECHNER, P.GEO.

I, Michael J. Lechner, P.Geo., of Stites, Idaho, do hereby certify:

- I am a consulting geologist and President of Resource Modeling Incorporated with a business address of 124 Lazy J Drive, PO Box 295, Stites, Idaho 83552.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of the University of Montana with a B.A. degree in Geology (1979). I am a Registered Professional Geologist in the State of Arizona (#37753), a Certified Professional Geologist with the American Institute of Professional Geologists (#10690), a Professional Geologist with British Columbia (#155344), and a Registered Member of SME (#4124987RM). From 1979 to the present I have been actively employed in various capacities of the mining industry. I have worked as an exploration geologist exploring for precious and base metals throughout western North America (eight years), a mine geologist working at precious metal mines in California and Nevada (10 years), and a geologic consultant during which time I have estimated Mineral Resources for numerous precious and base metal deposits located throughout the world (16 years). I have worked on a number of exhative-type deposits as an explorationist, mine geologist, and resource estimator. Examples of some previous projects include the Royal Mountain King Mine (1987-1993), Greens Creek (1999-2000), Rubstovsk (2004), and El Roble (2013). I have been working on the Black Butte deposits for Tintina Resources Inc. since 2010. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on September 20, 2011.
- I am responsible for Sections 1.2 to 1.4, 1.6 to 1.9, 4.0 to 12.0, 14.0, 23.0, 25.1, 25.2, 26.2, 27.0 (geology and metallurgy only), and 28.2 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report. I co-authored the technical report entitled "Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana" and dated August 30, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Stites, Idaho.

Original document signed and sealed by
Michael J. Lechner, P.Geo.

Michael J. Lechner, P.Geo.
President
Resource Modeling Inc.

28.3 ANDREA CADE, P.GEO.

I, Andrea Cade, P.Geo., of Vancouver, British Columbia, do hereby certify:

- I am a Project Manager with Tetra Tech WEI Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia V6B 1M1.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of the University of Western Ontario, (M.Sc. Geology, 2003) and Simon Fraser University (B.Sc. Earth Science, 2001). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#37690). My relevant experience includes 11 years in mineral exploration and mining including copper-gold exploration on the nearby Mount Polley property and copper exploration in the Highland Valley area of British Columbia. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.1, 1.16, 1.17, 2.0, 3.0, 19.0, 24.0, 26.1, and 28.3 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Vancouver, British Columbia.

Original document signed and sealed by
Andrea Cade, P.Geo.

Andrea Cade, P.Geo.
Project Manager
Tetra Tech WEI Inc.

28.4 HARVEY WAYNE STOYKO, P.ENG.

I, Harvey Wayne Stoyko, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Manager of Estimating with Tetra Tech WEI Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia V6B 1M1.
- This certificate applies to the technical report entitled “Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana”, dated July 12, 2013 (the “Technical Report”).
- I am a graduate of the University of Saskatchewan, (B.Sc. Mechanical Engineering, 1985). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#17092). My relevant experience with respect to mine development and costing includes over 25 years of combined mining experience. This includes capital cost engineering/cost control for both greenfield and brownfield studies, and acquisitions/mergers or development of properites (construction) with Placer Dome. I have also been involved as an Owners representative with the planning, costing/cost control and execution of mine/concentrate handling facilities including plant, road, rail and port with the Port of Vancouver (Kinder Morgan) for Comino’s Red Dog Project. At Tetra Tech, I have directed the preparation of capital cost estimates for technical studies and I am responsible for project controls for ongoing studies and EPCM projects. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.14.1, 21.0 (except 21.2 to 21.5), 26.6 and 28.4 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report. I co-authored the technical report entitled “Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana” and dated August 30, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Vancouver, British Columbia.

Original document signed and sealed by
Harvey Wayne Stoyko, P.Eng.

Harvey Wayne Stoyko, P.Eng.
Manager of Estimating
Tetra Tech WEI Inc.

28.5 JIANHUI (JOHN) HUANG, PH.D., P.ENG.

I, Jianhui (John) Huang, Ph.D., P.Eng., of Burnaby, British Columbia, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech WEI Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia V6B 1M1.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of North-East University (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals (M.Eng., 1988), and Birmingham University (Ph.D., 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30898). My relevant experience with respect to mineral engineering includes more than 30 years of involvement in mineral process for base metal ores, gold, silver and rare metal. I have relevant experience in copper recovery from various ores including massive sulphide mineralization. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.11, 1.12 (except 1.12.1), 1.14.2, 17.0, 18.1 to 18.3, 18.11, 21.5.1, 21.5.4, 21.5.5, 25.5, 26.3.2, and 28.5 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report. I co-authored the technical report entitled "Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana" and dated August 30, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Vancouver, British Columbia.

Original document signed and sealed by
Jianhui (John) Huang, Ph.D., P.Eng.

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech WEI Inc.

28.6 SABRY ABDEL HAFEZ, PH.D., P.ENG.

I, Sabry Abdel Hafez, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of Assiut University, (B.Sc Mining Engineering, 1991; M.Sc. in Mining Engineering, 1996; Ph.D. in Mineral Economics, 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#34975). My relevant experience is in mine evaluation. I have more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from the conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. I have been involved in the technical studies of several base metals, gold, coal, and aggregate mining projects in Canada and abroad. I have recently been involved in the technical reports for the Copper Fox's Schaft Creek project feasibility study, Pretium Resources' Brucejack project feasibility study, AQM's Zafranal PEA, Castle Resources' Granduc project PEA study and Seabridge's KSM project prefeasibility study. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.15, 22.0, 25.8, and 28.6 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report. I co-authored the technical report entitled "Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana" and dated August 30, 2012.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Sabry Abdel Hafez, Ph.D., P.Eng.

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.

28.7 KEN BROUWER, P.ENG.

I, Ken Brouwer, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Managing Director with Knight Piésold Ltd. with a business address at Suite 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia, (BApSc., 1982). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#15117). My relevant experience includes 28 years of consulting for a wide variety of tailings, waste rock and water management facilities at various project locations around the world. I have also been providing permitting and design support for several mining projects in Montana since 1985, including extensive ongoing support at the Montana Tunnels Mine, Stillwater Mine, East Boulder Mine and Montana Resources Mine. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was February 1, 2011.
- I am responsible for Sections 1.12.1, 18.4 to 18.10, 21.5.6, 26.4, and 28.7 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report. I co-authored the technical report entitled "Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana" and dated August 30, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Vancouver, British Columbia.

Original document signed and sealed by
Ken Brouwer, P.Eng.

Ken Brouwer, P.Eng.
Managing Director
Knight Piésold Ltd.

28.8 LISA KIRK, P.G.

I, Lisa Bithell Kirk, P.G., of Bozeman, Montana, do hereby certify:

- I am a Principal Geochemist with Enviromin, Inc. with a business address at PO Box 1685, Bozeman, MT 59771.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of the University of Pennsylvania (B.A. in Geology/Environmental Science, 1983) and the University of Colorado (M.S. in Geochemistry, 1990). I am presently a Doctoral Candidate in Microbial Geochemistry at Montana State University ~~with anticipated completion in August 2013~~. I am a member in good standing of the Association of Professional Engineers and Geoscientists of Wyoming (PG-2959) and a Registered Member of the Society of Mining, Metallurgy, and Exploration (#4053453). My relevant experience is 29 years of experience in characterization, assessment, and management of mined materials. My project experience in waste rock characterization includes the design of environmental geochemical sampling and analytical programs for more than 15 copper, silver, gold, phosphate, talc, and coal mine sites, and third party review or evaluation of comparable programs at 14 additional mine sites throughout North and South America. These testing programs have involved development of statistically relevant sampling programs; analytical testing protocols, including custom testing methods; development of field sampling methods; *in situ* sampling and monitoring; and laboratory and field research. I have taught short courses in mine waste characterization and acid rock drainage prediction, and am an active member of the Acid Rock Drainage Technical Initiative and the International Network for Acid Prevention. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was October 27, 2011 for one day.
- I am responsible for Sections 1.13, 20.0, 25.7, 26.7, 27.0 (environmental only), and 28.8 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report. I co-authored the technical report entitled "Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana" and dated August 30, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Bozeman, Montana

Original document signed and sealed by
Lisa Bithell Kirk, P.G.

Lisa Bithell Kirk, P.G.
Principal Geochemist
Enviromin, Inc.

28.9 SRIKANT ANNAVARAPU, RM SME

I, Srikant Annavarapu, RM SME, of Mesa, Arizona, do hereby certify:

- I am a Principal Mining Engineer with AMEC E&C Services Inc. with a business address at 1640 S. Stapley Drive, Suite 241, Mesa, Arizona, 85204.
- This certificate applies to the technical report entitled "Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana", dated July 12, 2013 (the "Technical Report").
- I am a graduate of the Indian Institute of Technology, Kharagpur, India (B.Tech. degree in mining engineering, 1980) and the University of Arizona (M.S. degree in mining and geological engineering, 1998). I am a Professional Engineer in Arizona (#36554) and a registered member of the Society of Mining, Metallurgical and Exploration, Inc. I have practiced my profession for 28 years during which time I have been involved in the design of underground mining projects, including mine design, ground stabilization instrumentation, monitoring, and analysis, for various mines. My previous experience includes PFS design of the Barrick Cortez underground gold mine (drift-and-fill) in Nevada, USA; PEA design for Kamoa underground copper project (room-and-pillar with backfill) in the Democratic Republic of Congo; and PEA design for Bokan Mountain underground rare-earths mine (transverse stoping with backfill) in Alaska, USA. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.10, 15.0, 16.0, 18.12, 18.13, 21.2, 21.3, 21.4, 21.5.2, 21.5.3, 25.6, 26.5, and 28.9 of the Technical Report.
- I am independent of Tintina Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of July, 2013 at Mesa, Arizona

Original document signed and sealed by
Srikant Annavarapu, RM SME

Srikant Annavarapu, RM SME
Principal Mining Engineer
AMEC E&C Services Inc.