Report to:

TINTINA RESOURCES

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

Document No. 1291880100-REP-R0001-01

TETRA TECH
WARDROP

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

EFFECTIVE DATE: AUGUST 30, 2012

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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
tonnes seconds per hour metre cubed	ts/hm^3
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt
year (annum)	a

Abbreviations and Acronyms

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
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
Hard Rock Impact Act	HRIA
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
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
tailings management facility	TMF
Tetra Tech Wardrop	Tetra Tech
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 Wardrop (Tetra Tech) to prepare a 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.

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-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 of 8.483 Mt of Indicated mineral resources and 3.719 Mt of Inferred mineral resources support a 14-year life-of-mine (LOM).

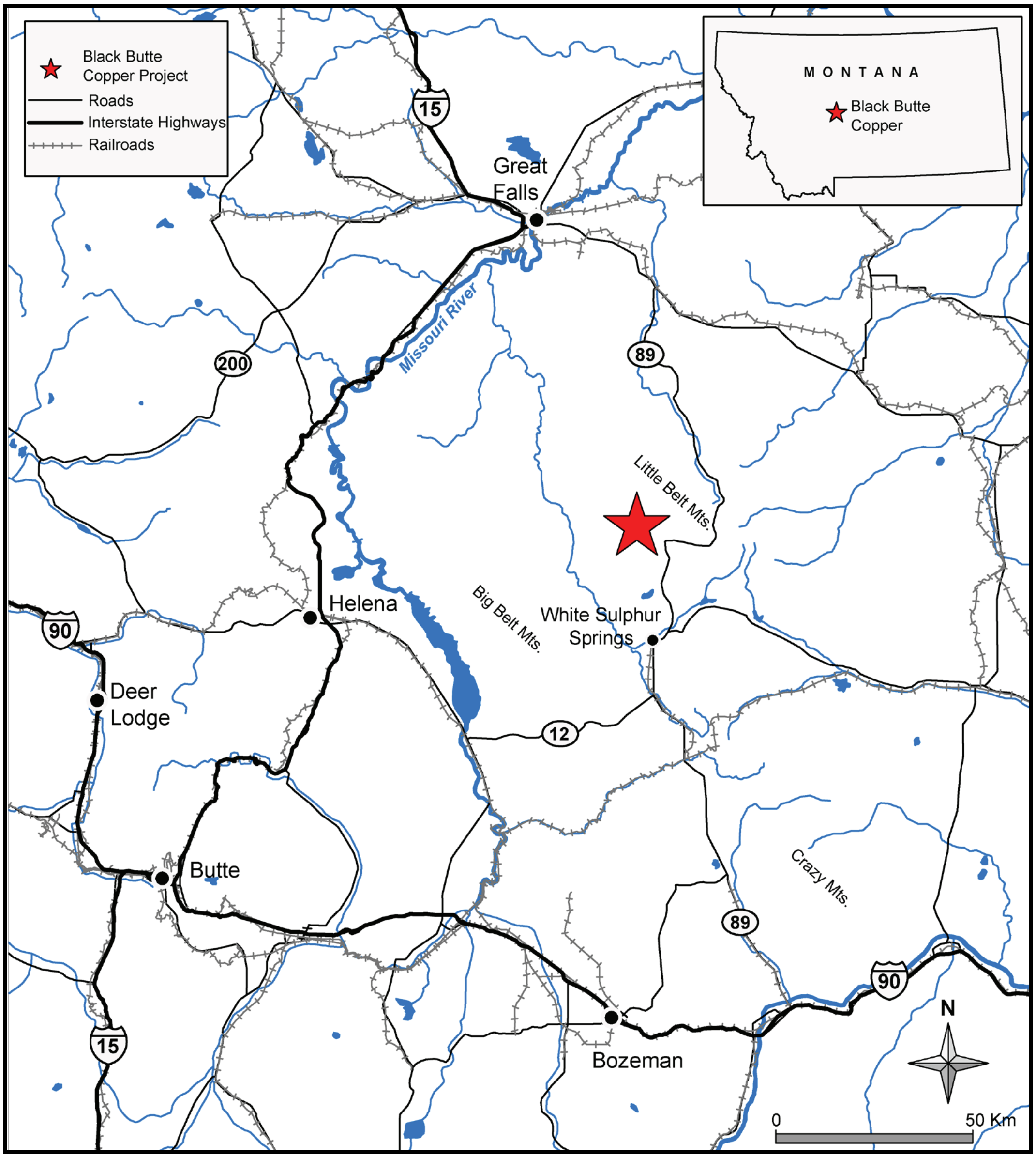
Table 1.1 outlines general information of the Project.

Table 1.1 General Project Information

Description	Unit	Amount
Estimated Mineral Resources (Indicated)	Mt	8.48
Estimated Mineral Resources (Inferred)	Mt	3.72
Copper Price	US$/lb	2.97
LOM	years	14
Milling Rate (Nominal)	t/d	3,300
Total Project Capital Cost	US$ million	210.6
Average Overall Operating Cost	US$/t milled	68.93
Net Present Value (NPV) at 8% Discount Rate	US$ million	145.8
Internal Rate of Return (IRR)	%	20.4
Payback Period	Years	5.5

All dollar figures presented in this PEA are stated in US dollars, unless otherwise specified. The London Metal Exchange (LME) three-year trailing metal prices, effective date of July 6, 2012, with an exchange rate of Cdn$1.00 to US$1.00 has been used, unless otherwise specified.

Figure 1.1 General Location Map



This PEA has been prepared by Tetra Tech for Tintina incorporating work by the following independant 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:** Mineral Processing and Metallurgical Testing
- **Stantec Consulting Ltd.** (Stantec): 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, formerly called the "Sheep Creek 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 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 and grass and mountain sagebrush covered valley floors and draws and open to partly timbered ridge tops. Timber covers approximately 10% of the resource area.

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 Sheep Creek 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 Sheep Creek are concentrated along the northern margin of the Helena embayment along the Volcano Valley Fault (VVF) zone.

The Newland Shale hosts the Sheep Creek 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*

The focus of this report is on copper-cobalt mineralization hosted in calcareous shale of the lower Newland Formation within bedded sulphide horizons referred to as the Upper Sulphide Zone (USZ). In the Sheep Creek area north of the Black Butte Fault (BBF), four separate lenses of massive sulphide occur along this stratigraphic horizon and are separated by conglomerate lenses or cut into separate structural blocks by northeast trending, down to the southeast normal faults. Only one lens, the Johnny Lee USZ, contains enough drillhole information to allow some detailed definition of its geometry and compositional character. With the exception of its higher copper grades, the mineralogical and textural attributes of the Johnny Lee USZ are typical of the USZ throughout the Sheep Creek area. The Johnny Lee USZ 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. (in press). 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.

Sulphides are concentrated in the Johnny Lee deposit 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 LZ reaches over 17 m thick and consists of bedded and replacement pyrite with high concentrations of replacement chalcopyrite in silicified shale and conglomerate.

1.5 METALLURGY

Tintina contracted Arthur H. Winckers, P.Eng. to conduct various metallurgical tests to determine the flotation response of representative composite samples from the Johnny Lee UZ and LZ. The objective of the preliminary metallurgical program was to develop effective flotation conditions for the recovery of copper and other payable metals 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.

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-cobalt 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 composite is based on correlations from locked cycle test and is currently estimated to be 81%. The test, based on samples with a head grade of 2.24% copper, yielded a concentrate grade of 23.3% copper. 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 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.

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 3D 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 3.93 based on 60 determinations was used to tabulate tonnage. A cut-off grade of 1.6% copper was used to estimate an Indicated Mineral Resource of 8,483,000 tonnes with an average grade of 2.96% copper, 0.12% cobalt, and 16.9 g/t silver. In addition to the Indicated Resource, there is an estimated Inferred Resource containing 1,257,000 t at an average grade of 2.64% copper, 0.10% cobalt, and 16.4 g/t silver using a copper cut-off grade of 1.6%. The cutoff 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 Indicated mineral resources and Inferred mineral resources are tabulated in Table 1.2 and Table 1.3, respectively using a copper cut-off grade of 1.6%.

Table 1.2 Undiluted Johnny Lee UZ Indicated Resources

Tonnes ('000)	Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
8,483	2.96	0.12	16.9	553	22.0	4,609

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

Tonnes ('000)	Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
1,257	2.64	0.10	16.4	73	3	663

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 3.80 based on 17 determinations was used to tabulate tonnage. A cut-off grade of 1.5% copper was used to define an Inferred mineral resource of 2,462,000 t with an average grade of 4.71% copper, 0.06% cobalt, and 5.1 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 and refining costs of US$5.53/t. The estimate of undiluted LZ Inferred mineral resources is tabulated in Table 1.4.

Table 1.4 Undiluted Johnny Lee LZ Inferred Resources

Tonnes ('000)	Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
2,462	4.71	0.06	5.1	256	2.9	404

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 not included in the PEA analysis.

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 3D 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 3.35 g/cm^3 based on 24 determinations was used to tabulate tonnage. A cut-off grade of 1.6% copper was used to define estimated Inferred mineral resources of 5,139,000 t with an average grade of 2.60% copper, 0.12% cobalt, 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$59/t, processing costs of US$16.00/t, and G&A costs of US$5.00/t. The estimate of undiluted Lowry MZ Inferred mineral resources is tabulated in Table 1.5.

Table 1.5 Undiluted Lowry MZ Inferred Resources

Tonnes ('000)	Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
5,139	2.60	0.12	14.6	294	14	2,412

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 1.6 and Table 1.7 shows the total undiluted Black Butte Indicated and Inferred mineral resource inventories. Note that at this time there is no recognized Indicated resource for the Johnny Lee LZ and Lowry MZ.

Table 1.6 Total Undiluted Black Butte Indicated Mineral Resources

Zone	Tonnes ('000)	Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
Johnny Lee UZ	8,483	2.96	0.12	16.9	553	22	4,609
Johnny Lee LZ	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Lowry MZ	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Total	**8,483**	**2.96**	**0.12**	**16.9**	**553**	**22**	**4,609**

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 Total Undiluted Black Butte Inferred Mineral Resources

Zone	Tonnes ('000)	Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
Johnny Lee UZ	1,257	2.64	0.10	16.4	73	3.0	663
Johnny Lee LZ	2,462	4.71	0.06	5.1	256	2.9	404
Lowry MZ	5,139	2.60	0.12	14.6	294	14.0	2,412
Total	**8,858**	**3.19**	**0.10**	**12.21**	**623**	**19.9**	**3,479**

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.

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

1.10 MINING

The mining concept provides for ramp access to the UZ and LZ and employs mechanized drift-and-fill mining method. The designs and estimates are based on Indicated and Inferred mineral resources (from the December 2011 block model). The designs and estimates are not considered optimized; production schedules have not been smoothed, and required project resources have not been leveled.

Based on a nominal cut-off grade of 1.60% copper, the resource is estimated at 12.2 Mt grading 2.99% copper, 965 ppm cobalt, and 13 ppm silver, after allowances for mining recovery and dilution.

The “base case” in the assessment is founded on a nominal production rate of 3,300 t/d at steady state. This is considered to be a median estimate of mining capacity for the mineralized material body. While not optimized, there is good confidence in the ability to sustain this rate. Overall mine life is projected to be nominally 14 years. Costs and schedules in this report exclude collaring the mine portal and driving the first 900 m of decline, which are assumed to be completed with an “underground exploration program”. The full scope of the underground exploration program has not been planned in detail. At a “scoping study” level, the demarcations between the underground exploration program, pre-production capital development phase, and start of the operating phase are not precisely known. More detailed scope and schedule for these would be the subject of subsequent studies.

Pre-production capital to develop the underground mine, lease equipment during the capital period and purchase select equipment is estimated at US$53.7 million (after the mine portal has been collared and initial 900 m of decline development has been completed). This includes pre-production capital development and certain mobile and fixed mine plant equipment during period Year 1 and the first quarter of Year 2.

A further US$37.9 million is included as “sustaining” capital cost from the second quarter of Year 2 onwards to complete lateral and vertical development, purchase/install equipment as the mine ramps up to peak production rate, and replace/rebuild equipment during the mine life.

Over the LOM, all combined mine operating costs (direct and indirect) amount to approximately US$49/t of mineralized material mined, excluding mobile equipment maintenance and electrical power.

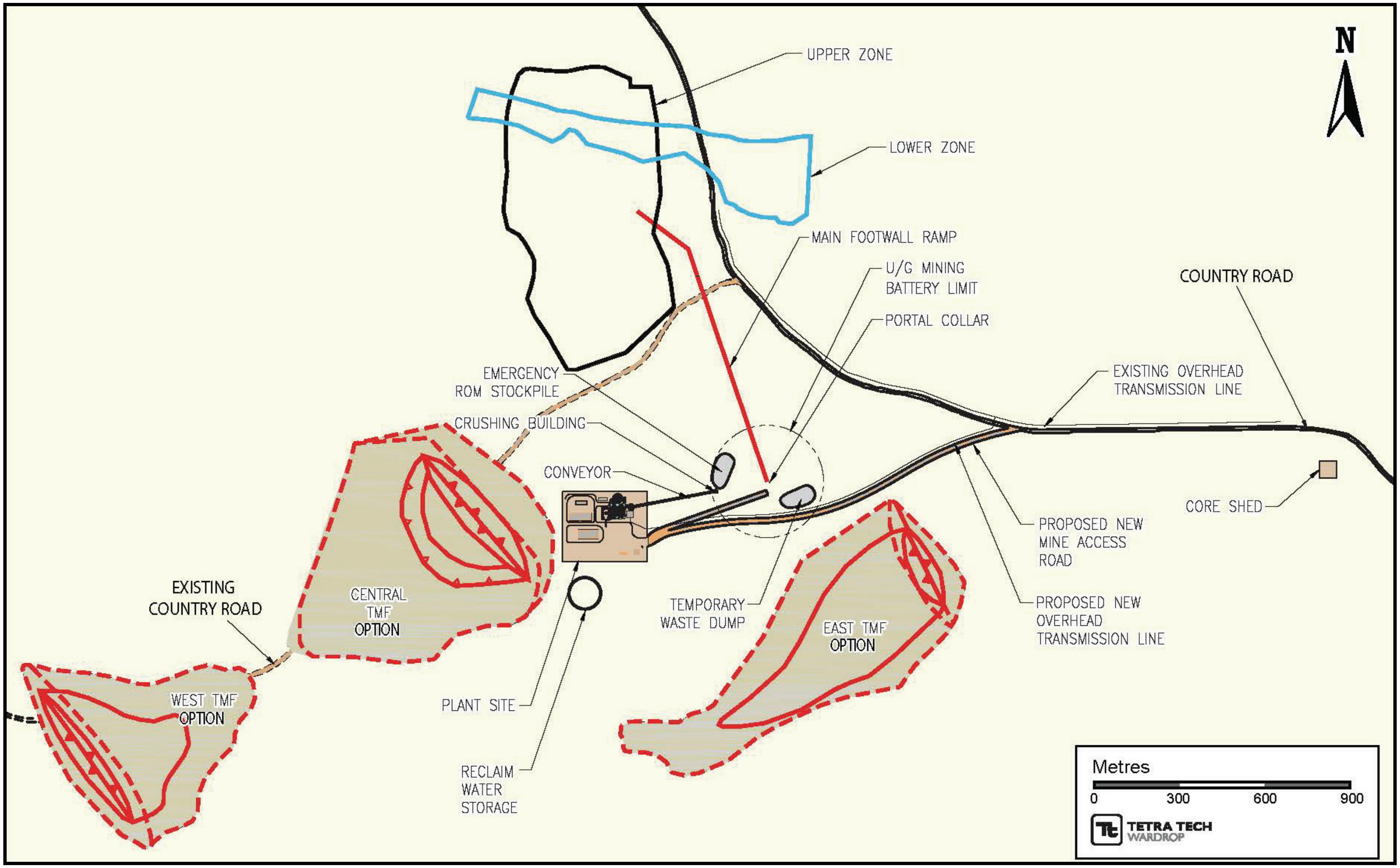
A detailed risk assessment has not been undertaken as an integral part of this mining concept study; however no technical “fatal flaws” to success of the Project are evident.

The opportunity for a higher production rate may be reconsidered once greater confidence in the geometry and geology is established – especially for the LZ. Economic trade-off of higher capital expenditures versus (temporarily) higher production rate and optimization of production rate is recommended for future work.

1.11 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, primary and secondary ball mills, rougher floations 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 electrival 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 in included providing 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 in addition to 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, machines shop, electrical room, mechncial 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 anciallary 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 with complete with all the required laboroatory equipment for metallurgical grade testing and control and will be equipped with all the required heating, ventilation, and air conditioning (HVAC) systems and chemical disposal equipment.

1.11.1 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 mm high-density polyethylene (HDPE) liner to minimize seepage from the facility. The TMF will store 6.1 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 for the first four years of operations. 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, partuclarly 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.

1.12 PROCESS

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, 350 d/a; the plant will process mineralized material at an annual rate of 1,155,000 t. The crushing plant availability will be 65% and grinding and flotation plant availability will be 94%.

The mill feed will be crushed by a jaw crusher to 80% passing 120 mm, and then ground to 80% passing 38 µm in a SAG/ball mill/pebble crushing circuit (SABC). 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% copper, will be thickened and then pressure-filtered before it is shipped to smelters.

1.13 ENVIRONMENTAL

Tintina proposes to conducted advanced exploration of the Johnny Lee deposit through development of an 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 like require permitting of power lines under the *Major Facility Siting Act*, a Surface Water Discharge Permit, a Groundwater Discharge Permit, water use permits, 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), 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 one year of data collection completed. Key environmental issues for the project will be acid rock drainage risk to surface and groundwater resources from waste rock and tailings, due to elevated sulphide content of some portions of the mineralized material 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$210.6 million is estimated for the Project (Table 1.8). 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 PEA. The accuracy range of the capital cost estimate is ±30%.

This estimate is prepared with a base date of Q2 2012 and does not include any escalation past this date. The quotations used in this estimate were obtained in Q2 2012 and are budgetary and non-binding.

Budget quotations were obtained for all major equipment. The vendors provided equipment prices, delivery lead times, freight costs to a designated marshalling yard, and spares allowances. For non-major equipment (i.e. equipment less than $100,000), 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, taxes, duties, freight, and packaging will be covered separately in the Indirects section of the estimate.

Table 1.8 Capital Cost Summary

Item	Total Cost (US$)
Direct Costs	
Overall Site	2,611,844
Mine Capital (Stantec)	53,642,988
Processing	58,618,969
Water Management (Knight Piésold)	9,478,403
Utilities	5,194,203
Buildings	8,242,691
Off-site Infrastructure	4,066,207
Plant Mobile Equipment	2,063,212
Subtotal	**143,918,516**
Indirect Costs	30,346,106
Owner's Costs	6,294,657
Contingency	30,076,498
Total Capital Costs	**210,635,778**

1.14.2 Operating Cost

On site operating costs are estimated to be US$68.93/t milled including mining, processing, G&A, and plant services. A total of 12,156,000 t mineralization from the underground mine will be processed during the LOM based on the proposed mining schedule. On average, the annual process rate is approximately 960,000 t/a or 2,740 t/d at 350 d/a. The unit cost is estimated based on the LOM average mill feed rate. Table 1.9 summarizes the operating costs.

Table 1.9 Operating Cost Summary

Area	Unit Operating Cost (US$/t milled) at LOM
Mining*	48.96
Processing	14.63
Tailings Management	0.29
G&A	3.39
Plant Services	1.66
Total	**68.93**

Note: *Including backfill cost

1.15 ECONOMIC ANALYSIS

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

- price of copper – US$2.97/lb
- total LOM production of 12,156,000 t of mineralized material
- average grade of 2.99% copper and average process recovery of 86.4%
- total of 692,751,000 lb of copper recovered over the 14-year LOM and 49,482,000 lb of copper recovered per year
- LOM recovered copper value of US$2,057,469,000 with an operating cost estimate of US$837,856,000 and a capital cost estimate of US$286,140,000.

The resulting discounted cash flow NPV at 8% is $145,811,000, the IRR is 20.4%, and the payback period is 5.5 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 increases in copper price would have a very significant positive impact on profitability of the Project.

This 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 PEA and the resultant economic evaluation, Tetra Tech recommends that this study should be further investigated in order to further assess the economic viability of the Project.

Detailed opportunities and recommendations complete with costs are provided in Section 26.0 of this technical report.

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 the 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:

- George Darling, P.Eng. from June 8 to 9, 2011
- Ken Brouwer, P.Eng. on February 1, 2011
- Lisa Kirk, P.G., Ph.D. on October 27, 2011
- Michael Lechner, P.Geo. 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	Harvey Wayne Stoyko, P.Eng.
3.0	Reliance on Other Experts	Tetra Tech	Harvey Wayne Stoyko, P.Eng.
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.

table continues…

Report Section		Company	QP
11.0	Sample Preparation, Analyses, and Security	RMI	Michael J. Lechner, P.Geo.
12.0	Data Verification	RMI	Michael J. Lechner, P.Geo.
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	Stantec	George Darling, P.Eng.
16.0	Mining Methods	Stantec	George Darling, P.Eng.
17.0	Recovery Methods	Tetra Tech	John Huang, P.Eng.
18.0	Infrastructure	-	-
	18.1 Introduction	Tetra Tech	Harvey Wayne Stoyko, P.Eng.
	18.2 Roads	Tetra Tech	Harvey Wayne Stoyko, P.Eng.
	18.3 Buildings	Tetra Tech	Harvey Wayne Stoyko, 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 Dumps	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	Harvey Wayne Stoyko, P.Eng.
19.0	Market Studies and Contracts	Tetra Tech	Harvey Wayne Stoyko, P.Eng.
20.0	Environmental Studies, Permitting, and Social or Community Impact	Enviromin	Lisa Kirk, P.G., Ph.D.
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	Stantec	George Darling, P.Eng.
	21.3 Mining Capital Cost Estimate	Stantec	George Darling, P.Eng.
	21.4 Operating Cost Estimate	-	-
	21.4.1 Summary	Tetra Tech	John Huang, P.Eng.
	21.4.2 Mining Operating Estimate	Stantec	George Darling, P.Eng.
	21.4.3 Process Operating Cost	Tetra Tech	John Huang, P.Eng.
	21.4.4 General and Administrative Costs and Surface Services Costs	Tetra Tech	John Huang, P.Eng.
	21.4.5 Tailings Management Costs	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.
24.0	Other Relevant Data and Information	Tetra Tech	Harvey Wayne Stoyko, P.Eng.
25.0	Interpretation and Conclusions	All	Sign-off by Section

table continues…

Report Section		Company	QP
26.0	Recommendations	All	Sign-off by Section
27.0	References	All	Sign-off by Section
28.0	Certificates of Qualified Person	All	Sign-off by Section

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.

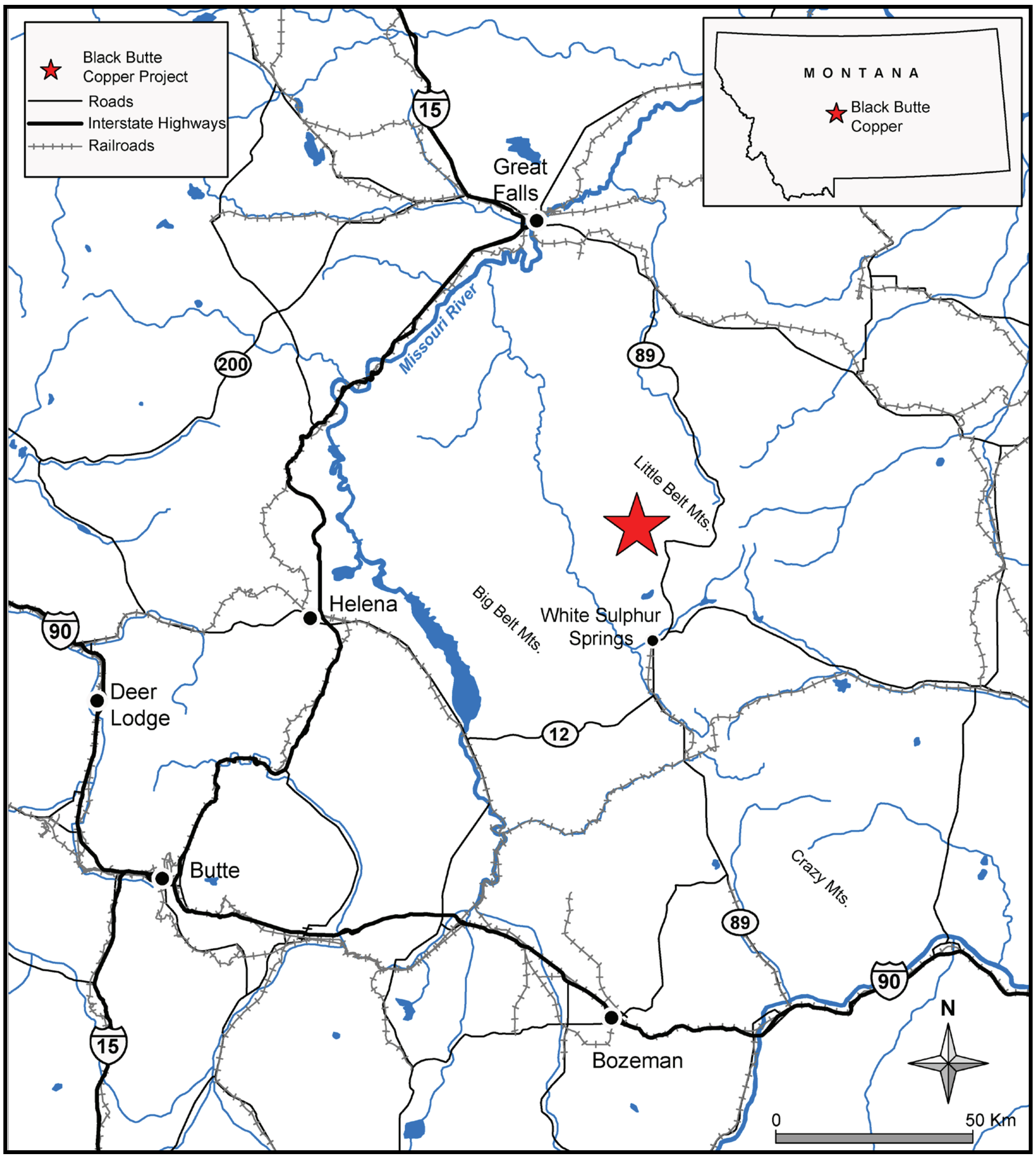
The QPs who prepared this report relied upon information provided by the following experts who are not QPs:

- 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.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located in Meagher County, Montana, about 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 U.S. 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 TintinaGold 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 data and each anniversary data 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; and $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 mine workings on the Property. 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.

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

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Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
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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

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Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
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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
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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

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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

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Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number	Claim Name	Section, Township, Range	Recorded Document Number	BLM Serial Number
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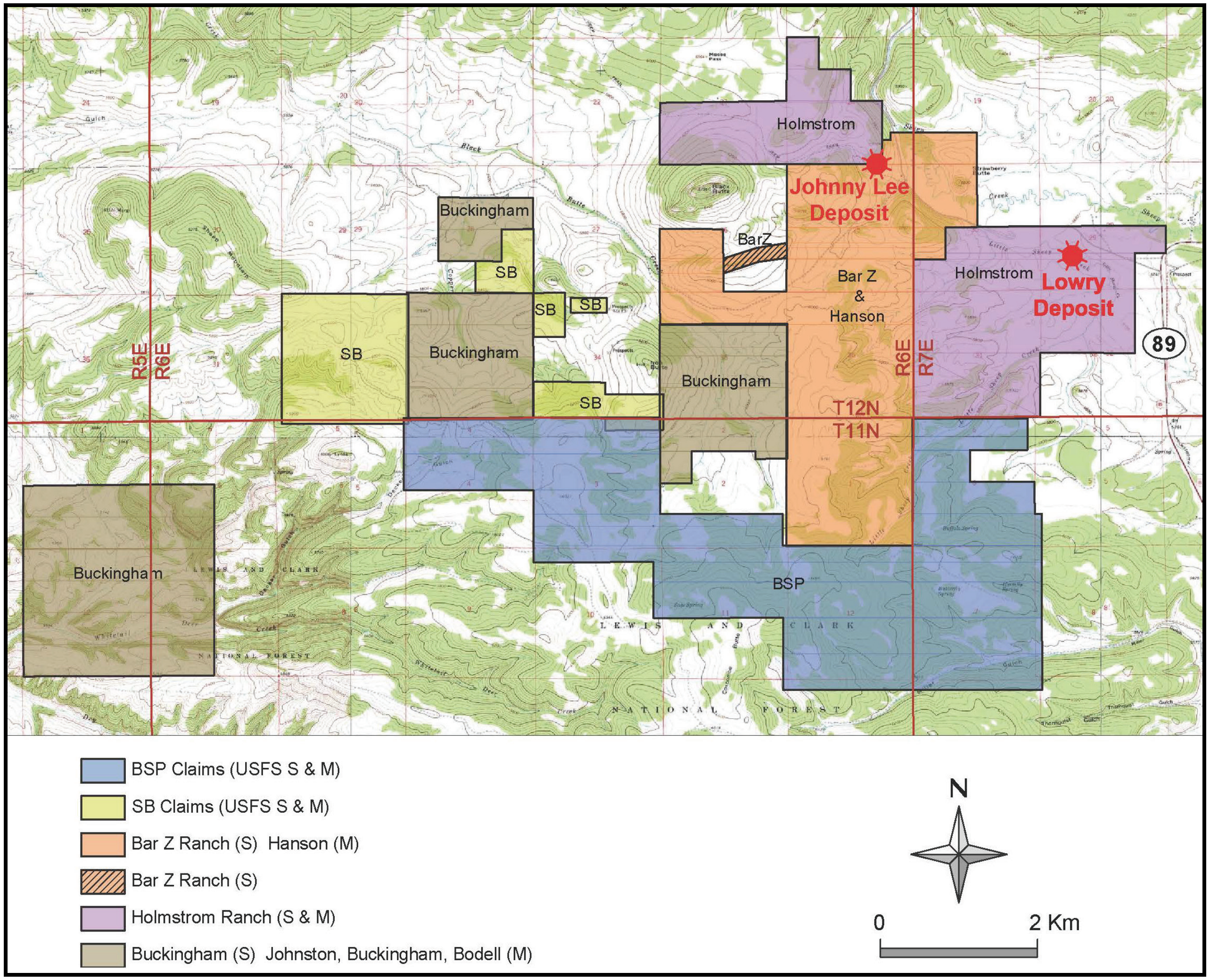
BSP-135	S 7, T11N, R7E	138388	MMC-223714				
BSP-136	S 7&18, T11N, R7E	138389	MMC-223715				
BSP-137	S 7, T11N, R7E	138390	MMC-223716				
BSP-138	S 7&18, T11N, R7E	138391	MMC-223717				
BSP-139	S 7, T11N, R7E	138392	MMC-223718				
BSP-140	S 7&18, T11N, R7E	138393	MMC-223719				
BSP-141	S 7, T11N, R7E	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



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.

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, and grass and mountain sagebrush-covered 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 which 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 approximately 984. This is the county seat of Meagher County which 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, Montana, population 633, which lies 80 km distant and has the nearest railhead; and Great Falls, Montana, with approximately 56,690 population, 132 km distant. Great Falls has an international airport.

Agriculture drives the local economy, and most agricultural operations specialize in cattle ranching but some produce grain. The region has high quality hunting and fishing and some locals have outfitting businesses for both big game and for fishing, including some primarily utilizing the Sheep Creek drainage. The few small logging operations in the area must haul logs to mills outside the valley, often for as much as 325 km. 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

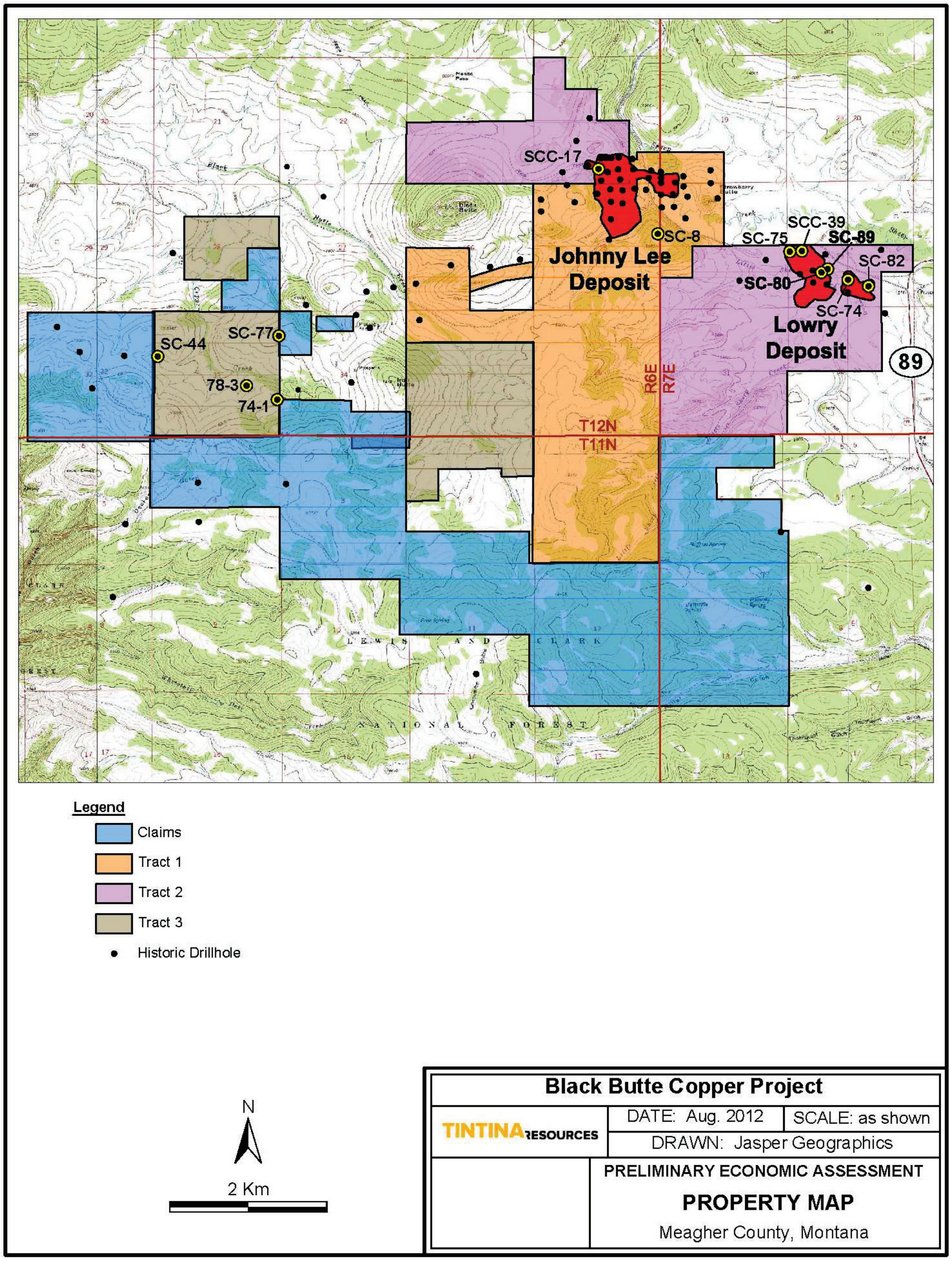
Tract 1 (Hanson/Bar Z lease), Tract 2 (Holmstrom lease), and Tract 3 (Buckingham lease; see Section 4.0) are ranch properties which were initially homestead properties and railroad lands, consolidated over time into the fee simple tracts now under lease to Tintina, complete with mineral rights. The same 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, but 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 and in 1910 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, personal comm.). Based on material on the dump, Mr. Lee only encountered unoxidized copper mineralized work. His cabin and outbuildings lay above the copper resource upon where Tintina's work is now focused and Tintina has recognized the efforts of Mr. Lee by naming the resource the "Johnny Lee Deposit".

During the first half of the twentieth 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 Hanson/Bar Z Ranch lease. Work focused on the iron potential, and while prospectors dug a few prospect pits and drove a few small adits, none penetrated the redox boundary into sulphide-bearing rock.

CAI carried out the first modern exploration work on the Property. CAI leased Tract 1 and Tract 3 (see 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 UII in 1985, and then UII was subsequently taken over by 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).

Figure 6.1 Claims Map



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 the USZ are the UZ #1 and the UZ #2. In the north end of the USZ resource, the UZ #1 is coincident with the USZ. Further south, the UZ #1 lay at or near the base of the much thicker USZ, and UZ #2 lay separated from and above the UZ #1 but still within the USZ. UZ #2 has a more limited areal extent than UZ #1. An UZ #3 was also encountered and had more limited areal extent than UZ #2.

CAI calculated 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 conceived and therefore 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 Project (i.e. Section 1.2). 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 completed both the USZ and LSZ resource estimates prior to the enactment of NI 43-101 and therefore the estimates are not compliant with NI 43-101 standards. 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 Project (i.e. Section 1.2). 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 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 base line environmental work but they did complete preliminary resource calculations (mentioned above) and completed initial metallurgical testing. These reports are proprietary and are not available to Tintina. There has been 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 which 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) 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 Black Butte are concentrated along the northern margin of the Helena embayment along 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 thick and an upper carbonate-dominated part which measures approximately 350 m thick. The shale is even 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 throughout, more typically sulphides are concentrated in several discrete stratigraphic horizons of greater lateral extent. The carbonate-rich upper Newland Formation is further divided into seven units 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; and Unit VII, a silty limestone and silty calcareous shale (Zieg 1981; 1986). The Greyson Shale, a shallow water silty shale measuring approximately 700 m thick, overlies the Newland, and is in turn overlain by the Spokane Shale, a red argillite measuring at least 300 m thick. 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 comprise the Lower belt portion of the Belt Supergroup.

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 may reflect a local representative of this event. No subsequent deformation or magmatism affected the area until Late Cretaceous regional compression, and at Black Butte, no intrusions accompanied this event. 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). Oligocene basaltic magmatism produced some basalt flows in the Sheep Creek 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

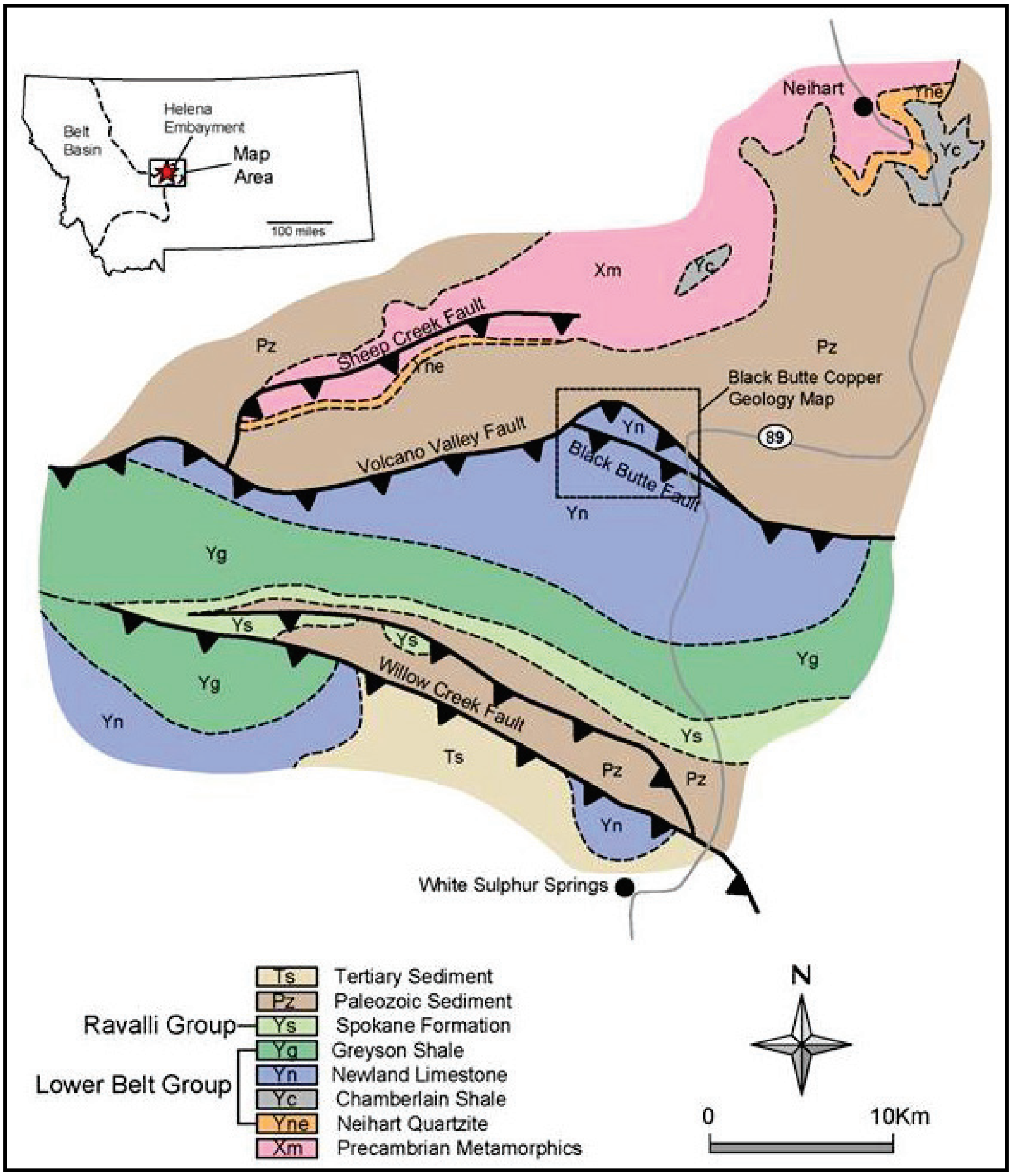
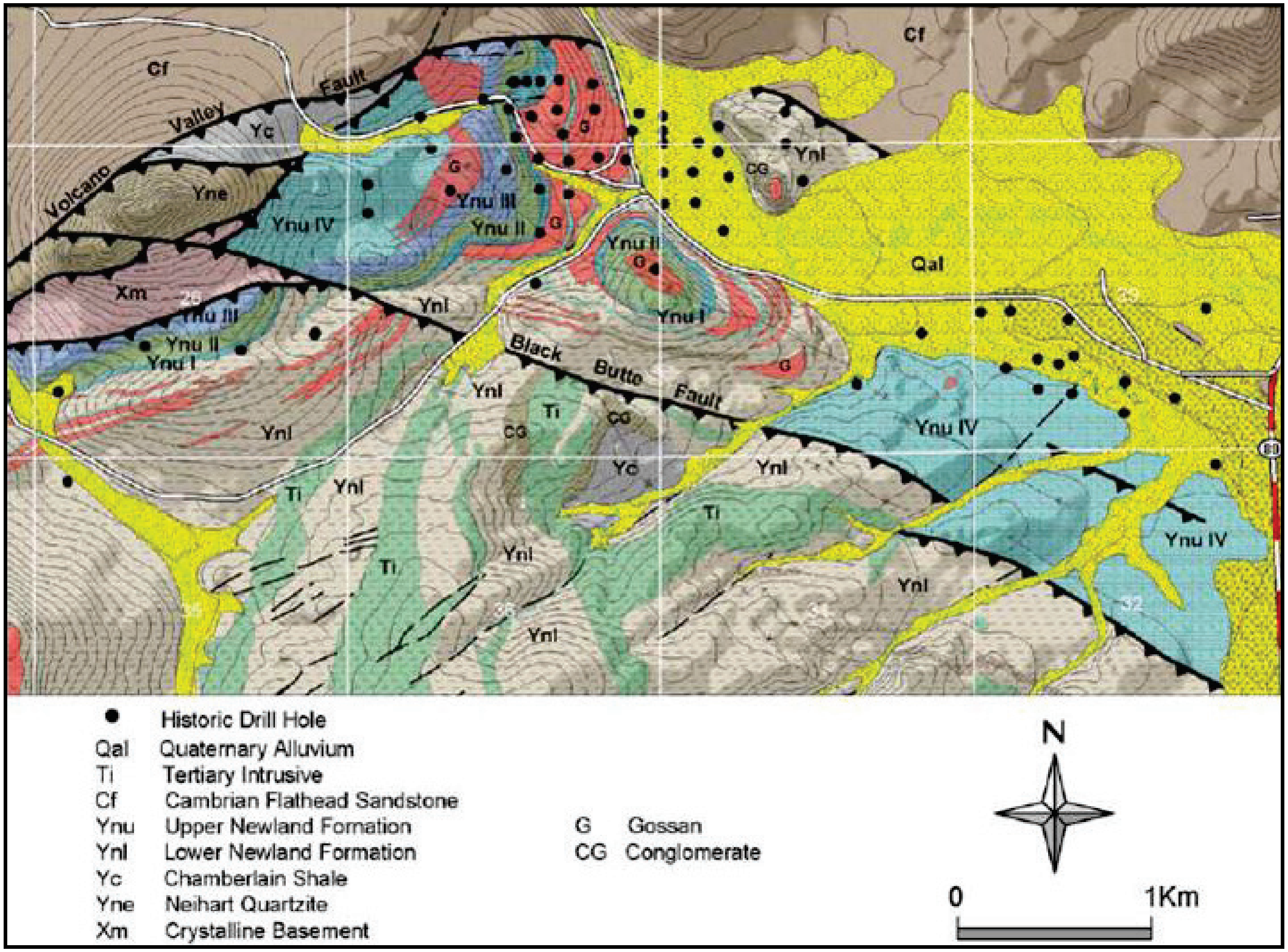


Figure 7.2 District Scale Geologic Map



The VVF forms the most mappable structure through the Project area, as it carries Proterozoic sediments and crystalline basement complex on its south side against Paleozoic sediments on its north side. Because regionally it cuts through the entire Paleozoic stratigraphic section and is intruded by Eocene intrusive, its age is best interpreted as Laramide (late Cretaceous). Drilling at Black Butte shows that the VVF cuts, conceals, and apparently utilizes portions of earlier, Pre-cambrian normal faults that form a northern margin to most Newland Formation exposures. The northernmost of these structures is known as the “Buttress” Fault. Allochthonous debris of a source-proximal derivation is preserved in lower Newland shale along the east-west fault trend 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 across succession of ”stair stepped” down thrown fault blocks of Newland and Chamberlain shale as shown in Figure 7.3. West of Butte Creek, the Copper Creek segment of the Volcano Valley fault shows an orientation of roughly N80E. At Butte Creek, a N50E trending structure offsets the VVF in sinistral fashion to a point 1 km northeast of its previous location. From this point, the Black Butte segment of the VVF continues east north of Black Butte for approximately 2 km and gradually arcs toward the southeast for 7 km at a bearing of S45E toward Newlan Creek. From its entrance into the Newlan Creek valley, the Newlan Creek segment of the VVF continues with an easterly bearing for at least 16 km. The flexures in the VVF at Butte Creek and at Newlan Creek are joined by a S65E trending northeast directed reverse fault called the Black Butte Fault (BBF) which carries Chamberlain shale over Newland Shale. The area enclosed 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).

In and around the Property area, bedded pyrite in the Newland shale persists along at least 25 km of strike length, and in some places occurs intermittently across 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 shales with the underlying Chamberlain Formation shale. 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 25 km of the east-west strike length of the district shows it consists of highly 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 sulfphide 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 sysnsedimentary 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 interpreted as having formed around algal or bacterial filaments (McGoldrick and Zieg 2004). Clearly, secondary pyrite, silicification, dolomitization, barite, and chalcopyrite all replaced the earlier pyritic"muds" deposited near and adjacent to vent sites. Lead isotope ratios from USZ galena samples completed by CAI 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, because the lower contact of an overlying debris flow cut through the west margin of the sulphide lens, a submarine slope failure must have carried a portion of the deposit away. Overlying Unit 0 carbonate carries through this area uninterrupted, so the event must have preceded it's deposition.

Figure 7.4 is plan map of a portion of the Johnny Lee deposit showing the surface projection traces of the UZ and LZs and lines of section for geologic cross sections shown in Figure 7.5 and Figure 7.6. These geologic cross sections have not been updated with Tintina's 2011 drillholes which are shown on the plan map (Figure 7.4).

Figure 7.3 Structural Setting

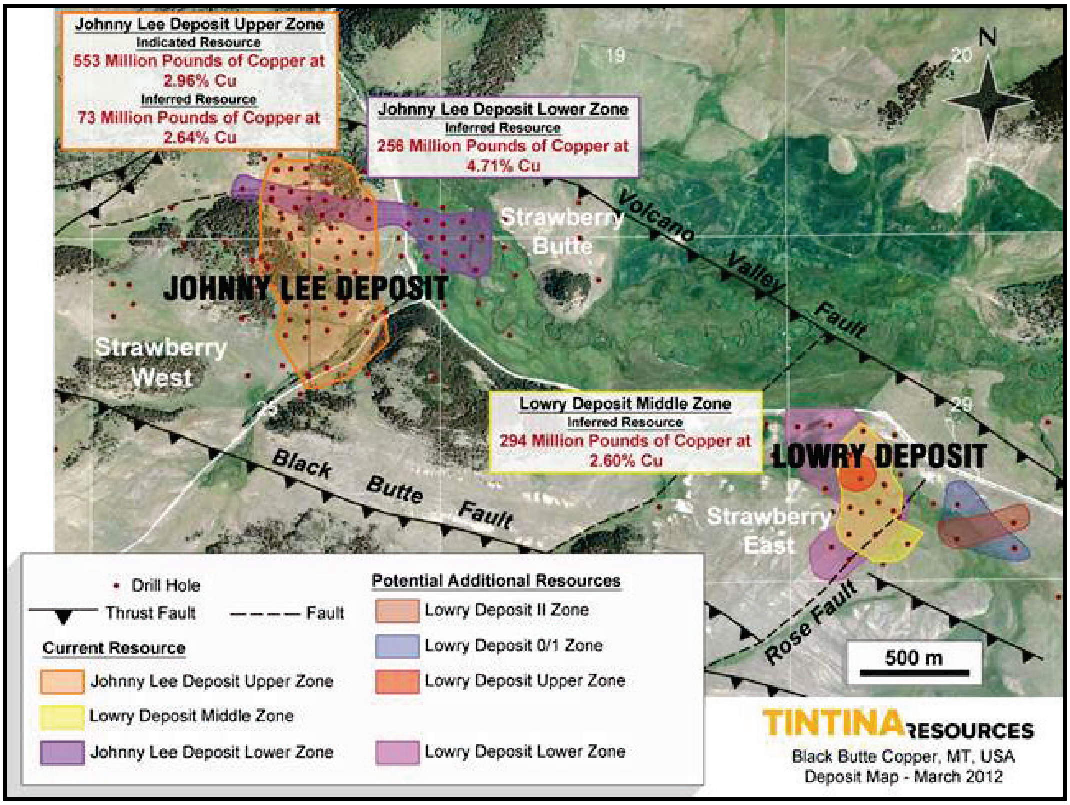


Figure 7.4 Plan Map Showing Lines of Section

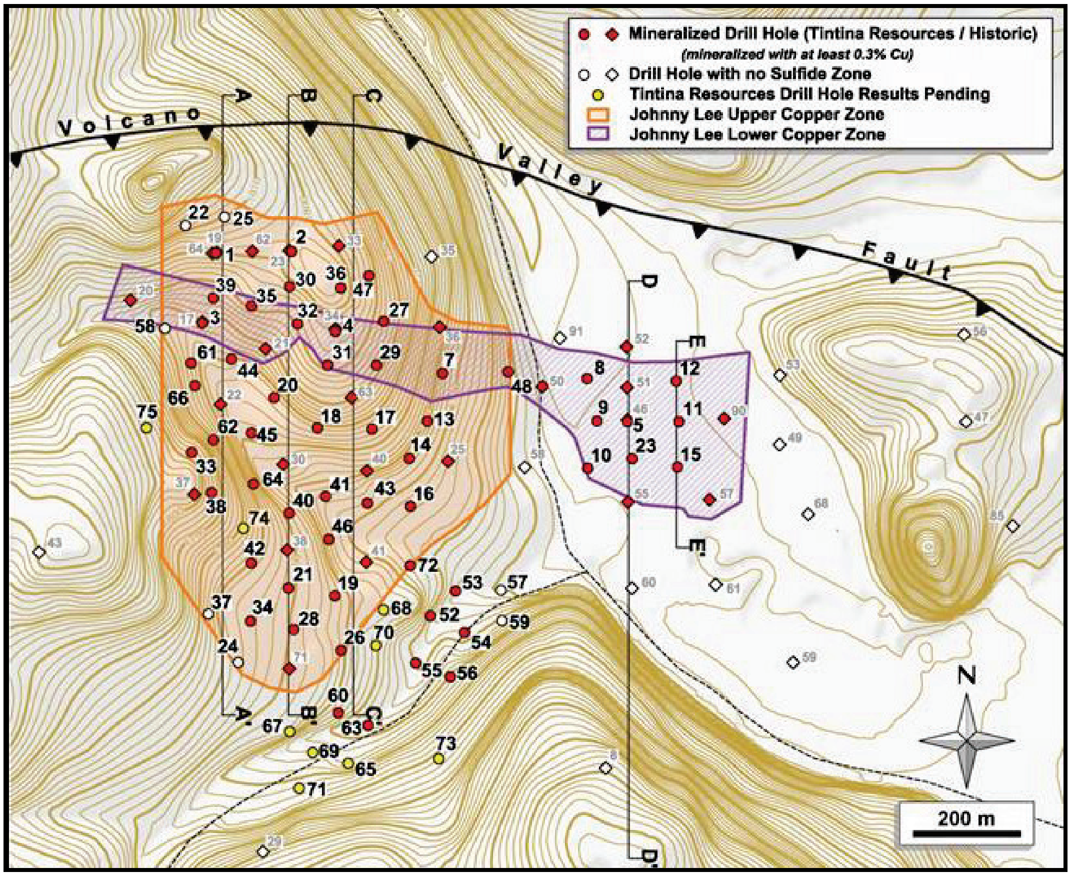


Figure 7.5 Sections A-A' and B-B'

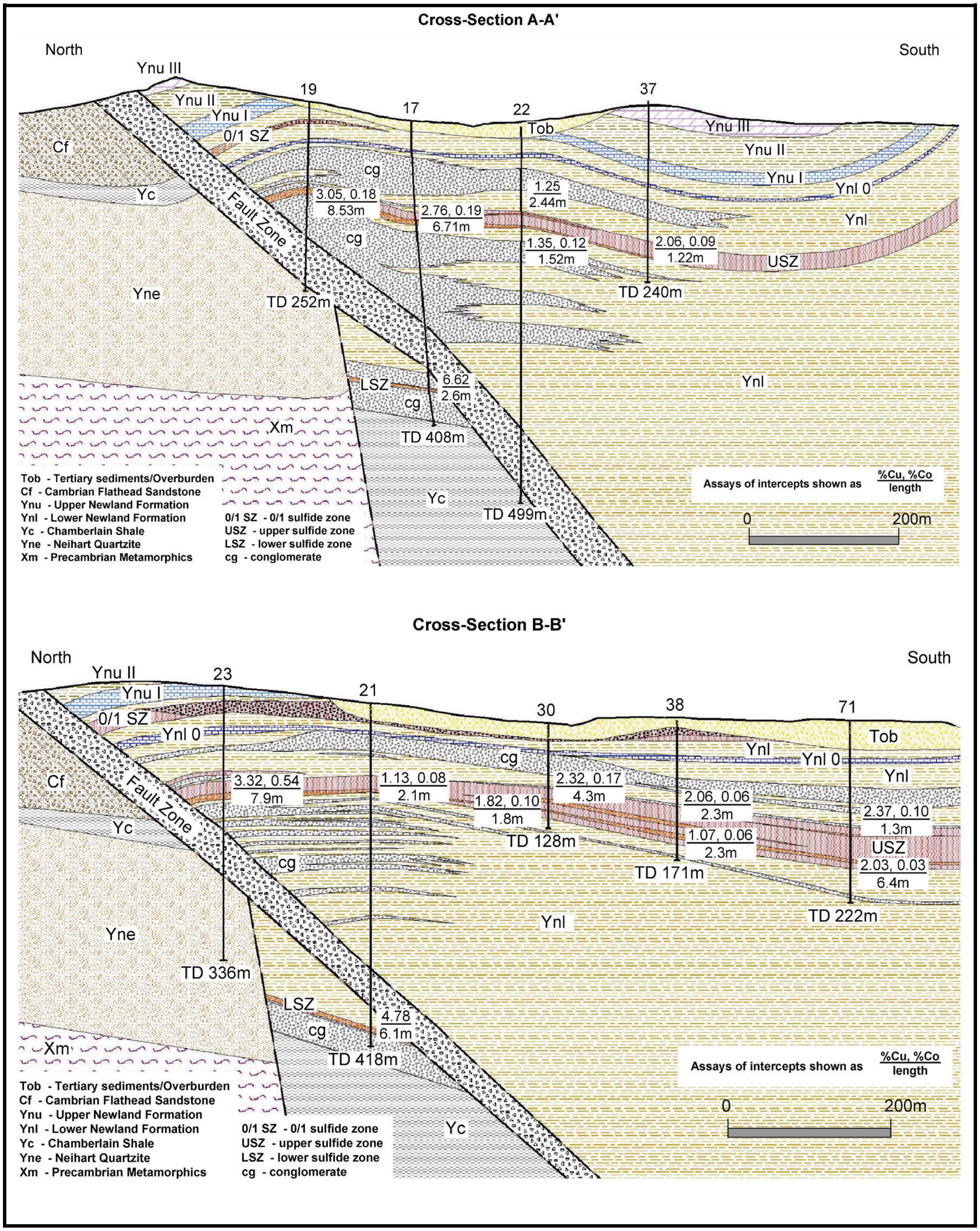
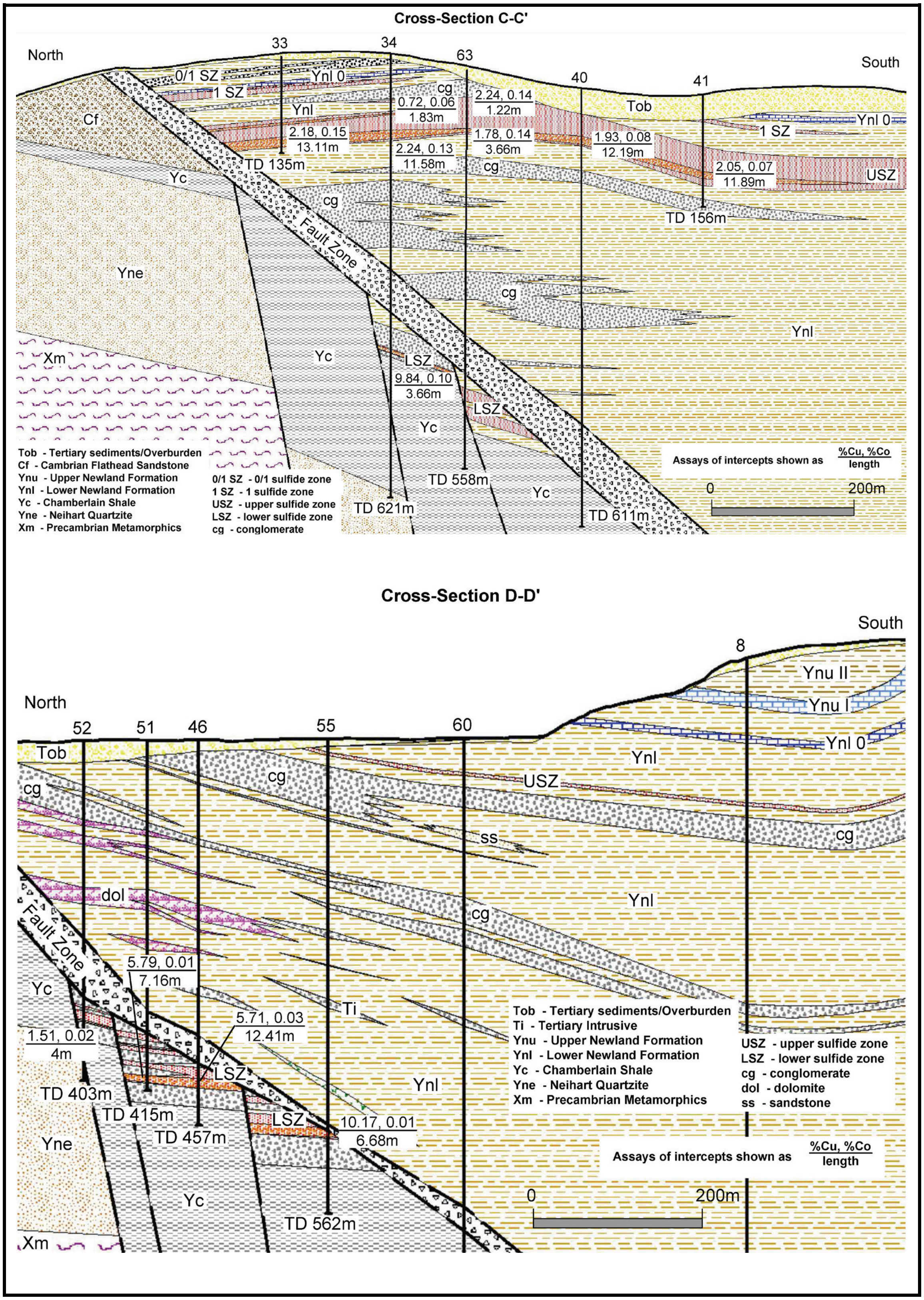


Figure 7.6 Sections C-C' and D-D'



7.2 MINERALIZATION

7.2.1 UPPER SULPHIDE ZONE

The USZ (Zieg et. al. 1991) stratigraphy lies 60 to 90 m below the contact between upper Newland and lower Newland and is hosted in lower Newland calcareous 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 cut into separate structural blocks by northeast trending, down to the southeast normal faults. Only the lens of USZ included in the Johnny Lee deposit called the Johnny Lee UZ, contains enough drillhole information to allow some 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 to the sulphide sheet, besides the VVF on the north, include an erosional trace on the east, a conglomeratic lens on the west, and an erosional surface cutting the sub-mineralized material grade portion of the 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. (in press). Pyrite occurs as laminations and beds of very fine grained pyrite as microcrystals and spheroidal aggregates form 1 to 25 µm or sometimes larger in diameter. Rhodes (2011 in-house report) interprets crinkly thin laminations within pyrite beds as microbial mat texture. 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 varieties. Pyrite and sometimes marcasite aggregates contain rims, patches, and sometimes cores of chalcopyrite and tennantite, and in many cases amorphous copper-cobalt-nickel-arsenic-rich material. Pyrite grain rims contain enrichments of 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 second 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 occurs both intergrown with and crosscutting pyrite. Gangue mineralogy in the Johnny Lee UZ is usually barite but can be dolomite or fine grained quartz. 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 some ongoing research by Dr. Gammons.

Copper-enriched horizons, informally called (in ascending order) UZs #1, #2, and #3, appear stratiform. These copper-rich horizons are now 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-rich zones disappear.

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 maintain levels 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 grades are associated with barite-chalcopyrite veins and masses of intergrown pyrite, barite, and chalcopyrite that replace and crosscut the bedded sulphide.

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 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. To the south east, at 2.5 km, the Lowry UZ also contains copper-cobalt but in lower concentrations. A typical example of the Johnny Lee UZ copper mineralization is shown in Figure 7.7, which shows fine-grained chalcopyrite mineralization with a copper grade of 1 to 2%.

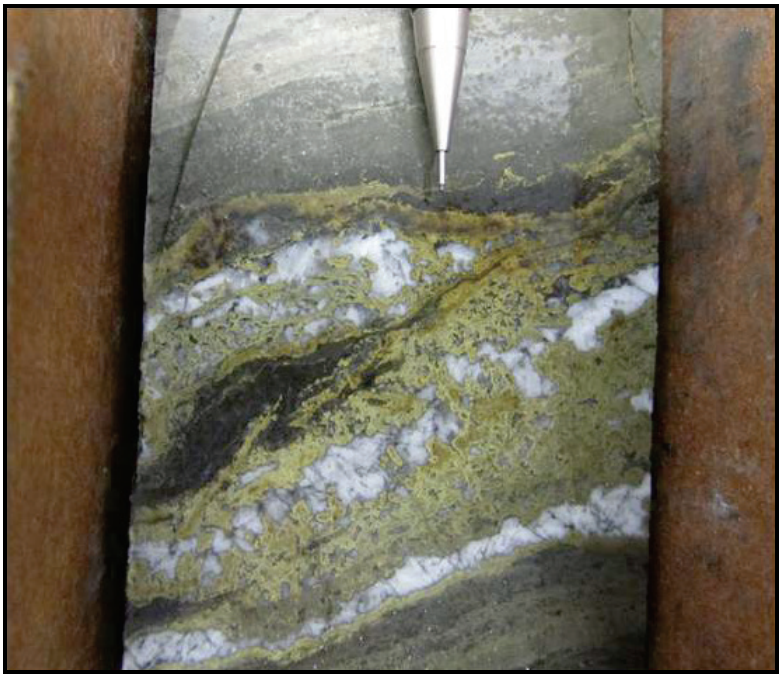
Figure 7.7 Fine-grained Chalcopyrite Mineralization – SC10-002

Higher grade Johnny Lee UZ chalcopyrite mineralization is shown in Figure 7.8, taken from a portion of Tintina drillhole SC10-004. The interval starting at 111.18 m ran about 10.4% copper.

Figure 7.8 Bedded Massive Sulphide Mineralization – SC10-004

Figure 7.9 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 7.9 Chalcopyrite-Barite Mineralization – SC11-055



7.2.2 MIDDLE SULPHIDE ZONE

East of the Johnny Lee deposit, the MSZ is well developd in the Lowry deposit area and where sufficiently copper mineralized is called the Lowry MZ. The MSZ begins about 30 m below the USZ and it's upper part consists of dolomitized carbonate breccia which shows textures typical of solution breccias, including occurrences of banded dolomite filling large vugs, and replacement pyrite occurrences. Below this, the breccia is overprinted by silicifiication and replacement chalcopyrite and some pyrite. Breccia fragments include some bedded pyrite, including vent fauna, but the sulphide content of the MSZ is considerably lower than that of the USZ. The MSZ shows textural, mineralogical, and metallurgical similarities more similar to the LSZ. The MSZ shows extreme variability in thickness, sulphide content, copper and cobalt content, and degree of alteration over lateral distances of 200 m or less, and appears best developed along northeast trending high angle faults near the VVF. 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 Middle Zone, and the VVF cuts away the base of the thickest MSZ occurrences. The Lowry Middle Zone remains incompletely drilled and is open to the south.

Five kilometres west at Butte Creek, the stratigraphy below the USZ includes shows 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.

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 a dolomite, dolomitic shale, black shale, and shale clast conglomerate protolith. The LSZ contains higher copper contents and lower cobalt contents than the USZ. Chalcopyrite is the only copper mineral identified within the Johnny Lee LZ. Coarse grained chaotic fragmental or crosscutting sulphide textures dominate the Johnny Lee LZ, through bedded pyrite does occur in and above it. Up to four additional sulphide zones that lie in the hanging wall of the LSZ and show more bedded pyrite, some replacive dolomite and pyrite intergrowths, occasional barite, and generally less chalcopyrite. Some silicification can occur in hanging wall sulphide zones.

Alteration associated with Johnny Lee Lower Zone sulfide 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 in crosscutting replacement veins and ribbons, porphyrotopes, and trains of porphyrotopes, occurs throughout the hanging wall of the Johnny Lee LZ for metres and overprints hanging wall zones. Dolomite alteration also occurs footwall to and distal to the LSZ but this distal alteration shows finer grain sizes and weak development. Associated mineralization commonly contains disseminated chalcopyrite, and in places sphalerite or galena. In more distal mineralization, barite can occur in a similar fashion to the carbonate.

The LSZ is also represented in the Lowry deposit LZ, and there, the footwall consists of shattered silicified sediment rather than conglomerate as is observed elsewhere. The footwall contains frequent 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. Also in the Lowry LZ hangingwall, zones of silicification, replacive carbonate-sullphide masses, and scattered pyrite beds can contain mineralized material-grade concentrations of chalcopyrite. Distal LSZ mineralization from the Lowry LZ appears similar to distal LSZ mineralization and hanging wall sulphide zones near the Johnny Lee LZ where it often 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. Capping the LSZ stratigraphy is a 60 m stylolitic dolomite unit called the Hangingwall Dolomite.

The greatest known concentrations of LZ copper mineralization are in the Johnny Lee LZ, and to a lesser extent in the Lowry 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.

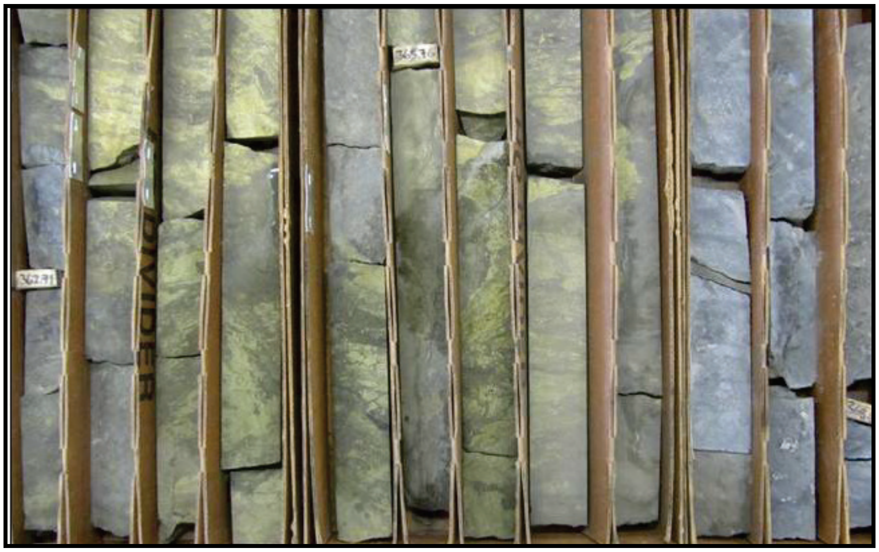
The Johnny Lee LZ has been modelled as the 3D wireframe using a nominal copper cut-off of 2%. The dimensions of the wireframe are approximately 1,175 m in the east west direction and approximately 125 m in the north-south direction. The unit is wider on the far east end reaching about 300 m in the north-south 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 7.10. This interval contains about 15% copper.

Figure 7.10 Finely Bedded LSZ Mineralization – SC10-004

Another example of very high-grade copper mineralization from the Johnny Lee LZ is shown in Figure 7.11 from a 2011 Tintina core hole (SC11-048). The interval between the two wooden run blocks ran 8.5% copper.

Figure 7.11 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 make reasonable analogies (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 centers at the seabottom. In more distal areas, zinc, lead, and silver become more concentrated. The process resulted in multiple stratabound zones of copper-cobalt mineralization in at least five stratigraphic levels within the host shales.

In all bedded sulphide zones, a biogenic component to mineralization is evidenced by the abundant preservation of two early fabrics, microbial mats within the massive pyrite beds and abundant masses of pyritized microbial tubiform structures. 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 concentration of these early thermophiles around seafloor hydrothermal vent centres.

According to Zieg and Lietch (1994) a synsedimentary origin for the Sheep Creek 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. Sheep Creek 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 δ^{34}S values, suggesting a sulphur-deficient source fluid. One chalcopyrite-pyrite, sulphur isotopic, pair yielded a temperature of 276°C. Analyses of USZ barite show δ^{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 (δ^{34}S -14 to 18 $^{0}/_{00}$) and barite (δ^{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 Sheep Creek base metal mineralization, and also with the results of Strauss and Schieber (1990).

The role of microbial life forms appears to have been critical to both reduction of sulphate and as a substrate for much sulphide precipitation in the initial stages of development of the sulphide zones. The lack of higher 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.

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 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 is addressing these and will field-check areas of concern.

Tintina also compiled previous soil geochemical data and conducted their own soil sampling program in 2011. Tintina's staff compiled available historical soil geochemistry from the area and 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.

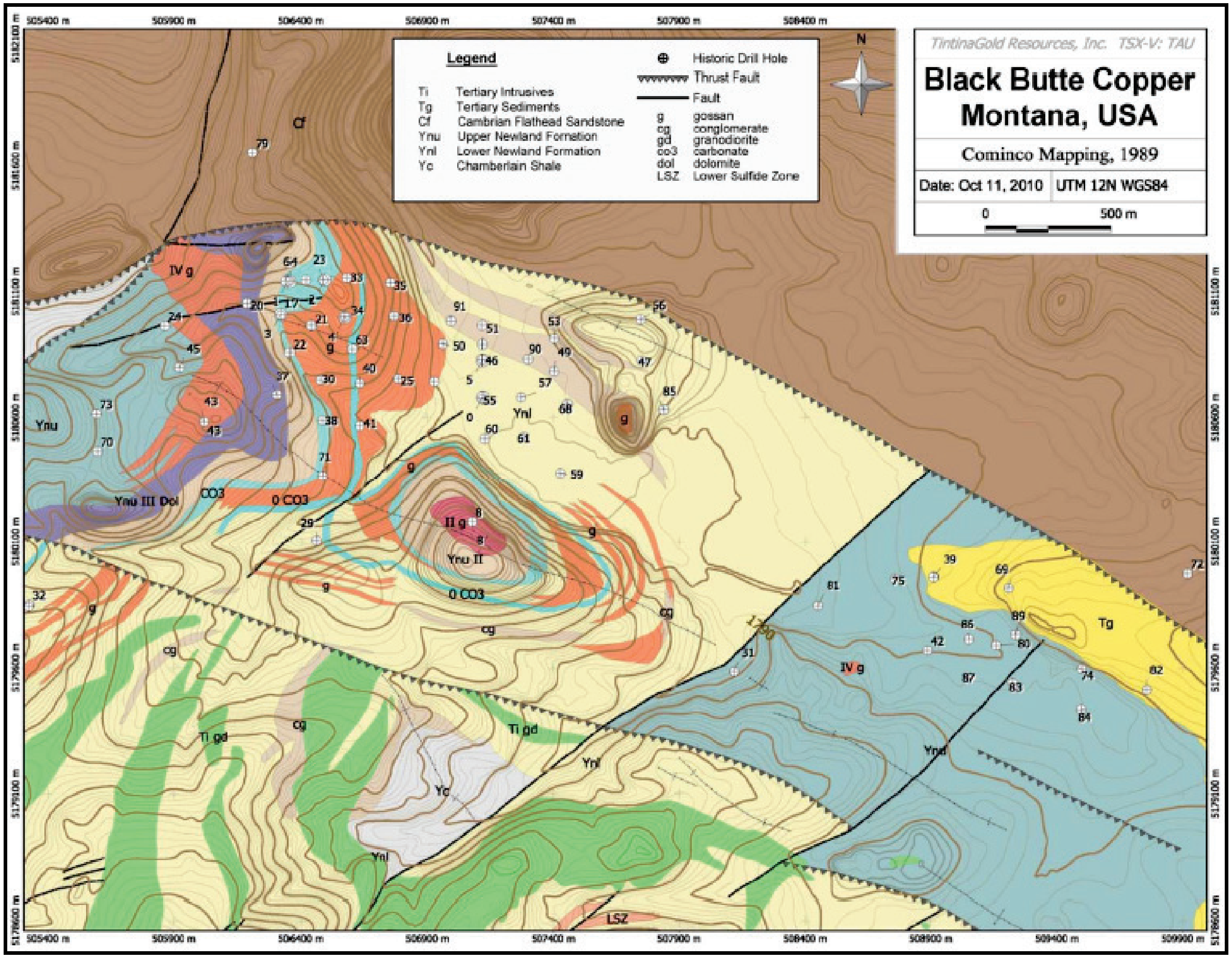
During 2011, Tintina crews collected 744 soil samples over areas geologically permissive for mineralization and not covered by earlier surveys. Field crews collected samples with a hand auger, and attempted to collect from the ”B” soil horizon. If the B horizon was not present, samples were collected from the ”C” horizon. The survey consisted of collecting soil samples from 19 north-south oriented lines. These sample lines were spaced 300 m apart in most areas with samples collected at 60 m intervals along the sample line. In areas which were thought to be 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 will require field review.

In March 2012, Tintina contracted Aeroquest to complete an airborne magnetics and resistivity survey over the district. The data has been collected, and final processing is pending. Initial results show the highly conductive nature of the Johnny Lee UZ. Many other conductors are apparent on the Property and many are consistent with known trends of sulphide mineralization. Magnetic data shows the transition from a highly magnetic area to the north with very thin or absent sequences of Belt rocks, to the less magnetic area on the south with thick sequences of Belt rocks. Eocene intrusives follow northeast and northwest trends and form strong linears on the map. Analysis of the results and comparisons with district geology continue.

Figure 9.1 is an historic geologic map that was completed by CAI. This figure also shows the location of drillholes prior to Tintina's entry into the district.

Figure 9.1 Historic Drillhole Locations



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 and LZ has been collected by Tintina as a result of their 2010 and 2011 drilling campaigns. Approximately 65% of the UZ drilling was completed by Tintina. Drilling data collected by CAI, UII, and BHP were also used by RMI for estimating mineral resources for the Johnny Lee zones. Drillhole data that were used for the Johnny Lee UZ are summarized Table 10.1.

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	54	10,846.34
Total	**73**	**16,595.76**

Table 10.2 tabulates the drillholes that were used to estimate mineral resources for the Johnny Lee LZ. Note that 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	5	2,321.66
UII	4	1,889.46
BHP	1	456.59
Tintina	15	6,979.28
Total	**25**	**11,646.99**

Table 10.3 tabulates the drillholes that were used to estimate mineral resources for the Lowry MZ. Note that one of the CAI holes (SC-87W) was wedged off another diamond hole (SC-87).

Table 10.3 Summary of Lowry MZ Drilling by Company

Company	No. of Holes	Drilled (m)
CAI	4	2,558.79
Tintina	10	6,715.20
Total	**14**	**9,273.99**

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). This table 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.

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 color coded by the company that drilled them. There are more drillholes shown in Figure 10.1 within the UZ than were used for the 2010 resource estimate. 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)	Zone	Company
SC10-001	506,353.76	5,181,138.65	1,801.55	0.00	-90.00	142.07	UZ	Tintina
SC10-002	506,490.51	5,181,146.29	1,802.41	0.00	-90.00	144.78	UZ	Tintina
SC10-003	506,325.98	5,181,009.87	1,775.67	0.00	-90.00	365.88	UZ & LZ	Tintina
SC10-004	506,573.20	5,180,996.13	1,799.27	0.00	-90.00	429.91	UZ & LZ	Tintina
SC10-005	507,119.35	5,180,825.79	1,709.93	0.00	-90.00	422.43	LZ	Tintina
SC10-006	509,162.70	5,179,683.30	1,726.20	0.00	-90.00	579.12	MZ	Tintina
SC11-007	506,775.28	5,180,915.26	1,747.01	0.00	-90.00	475.49	UZ & LZ	Tintina
SC11-008	507,044.04	5,180,906.06	1,708.72	0.00	-90.00	420.00	LZ	Tintina
SC11-009	507,062.33	5,180,827.03	1,709.32	240.00	-87.00	450.95	LZ	Tintina
SC11-010	507,045.22	5,180,738.69	1,710.28	0.00	-90.00	545.59	LZ	Tintina
SC11-011	507,215.58	5,180,825.55	1,709.87	0.00	-90.00	457.20	LZ	Tintina
SC11-012	507,210.76	5,180,901.49	1,709.26	0.00	-90.00	420.01	LZ	Tintina
SC11-013	506,745.36	5,180,829.58	1,750.35	0.00	-90.00	94.49	UZ	Tintina
SC11-014	506,713.05	5,180,756.83	1,745.64	0.00	-90.00	102.11	UZ	Tintina
SC11-015	507,212.95	5,180,740.61	1,710.97	0.00	-90.00	518.38	LZ	Tintina
SC11-016	506,713.82	5,180,667.04	1,739.19	0.00	-90.00	120.09	UZ	Tintina
SC11-017	506,642.23	5,180,810.25	1,762.65	0.00	-90.00	114.91	UZ	Tintina
SC11-018	506,540.74	5,180,811.75	1,765.96	0.00	-90.00	124.97	UZ	Tintina
SC11-019	506,572.95	5,180,500.03	1,751.43	0.00	-90.00	157.58	UZ	Tintina
SC11-020	506,459.30	5,180,869.96	1,767.55	0.00	-90.00	124.97	UZ	Tintina
SC11-021	506,485.50	5,180,512.81	1,758.41	0.00	-90.00	188.37	UZ	Tintina

table continues…

BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	Zone	Company
SC11-023	507,128.19	5,180,756.62	1,710.04	0.00	-90.00	511.45	LZ	Tintina
SC11-024	506,391.91	5,180,376.13	1,766.53	0.00	-90.00	216.10	UZ	Tintina
SC11-026	506,584.17	5,180,398.57	1,750.91	0.00	-90.00	180.14	UZ	Tintina
SC11-027	506,662.83	5,181,012.74	1,785.06	0.00	-90.00	111.86	UZ	Tintina
SC11-028	506,494.30	5,180,438.79	1,758.26	0.00	-90.00	190.20	UZ	Tintina
SC11-029	506,650.40	5,180,930.50	1,780.90	0.00	-90.00	512.06	UZ & LZ	Tintina
SC11-030	506,486.44	5,181,077.50	1,797.06	0.00	-90.00	155.45	UZ	Tintina
SC11-031	506,559.60	5,180,930.36	1,789.00	0.00	-90.00	548.64	UZ & LZ	Tintina
SC11-032	506,501.18	5,181,007.12	1,790.42	0.00	-90.00	469.39	UZ & LZ	Tintina
SC11-033	506,277.78	5,180,732.83	1,803.12	32.40	-76.50	227.08	UZ	Tintina
SC11-034	506,413.27	5,180,448.44	1,763.70	33.40	-88.10	199.95	UZ	Tintina
SC11-035	506,416.26	5,181,040.87	1,785.68	0.00	-90.00	156.06	UZ	Tintina
SC11-036	506,585.20	5,181,076.58	1,800.77	0.00	-90.00	147.52	UZ	Tintina
SC11-038	506,303.16	5,180,695.42	1,801.78	101.50	-77.90	220.07	UZ	Tintina
SC11-039	506,330.69	5,181,055.43	1,783.56	0.00	-90.00	160.93	UZ	Tintina
SC11-040	506,495.21	5,180,645.54	1,758.16	331.70	-85.10	153.62	UZ	Tintina
SC11-041	506,554.40	5,180,686.40	1,748.20	0.00	-90.00	129.69	UZ	Tintina
SC11-042	506,402.77	5,180,538.25	1,767.57	17.30	-80.10	205.13	UZ	Tintina
SC11-043	506,635.88	5,180,675.66	1,743.88	219.00	-87.70	129.54	UZ	Tintina
SC11-044	506,369.22	5,180,941.93	1,770.40	0.00	-90.00	140.21	UZ	Tintina
SC11-045	506,415.87	5,180,826.12	1,761.80	177.40	-79.20	146.91	UZ	Tintina
SC11-046	506,548.61	5,180,599.52	1,749.66	63.80	-82.80	149.96	UZ	Tintina
SC11-047	506,590.46	5,181,075.11	1,800.29	61.80	-64.40	139.90	UZ	Tintina
SC11-048	506,913.20	5,180,824.70	1,720.19	342.50	-74.40	431.90	LZ	Tintina
SC11-049	509,063.00	5,179,934.90	1,736.30	16.20	-89.10	594.40	MZ	Tintina
SC11-052	506,801.90	5,180,456.20	1,726.50	276.40	-59.90	173.43	UZ	Tintina
SC11-053	506,799.90	5,180,457.30	1,726.50	357.50	-58.30	153.47	UZ	Tintina
SC11-054	506,802.70	5,180,453.80	1,726.50	149.20	-73.60	148.74	UZ	Tintina
SC11-055	506,767.20	5,180,390.00	1,731.80	253.10	-64.60	155.45	UZ	Tintina
SC11-056	506,767.00	5,180,389.10	1,731.90	153.50	-62.90	161.54	UZ	Tintina
SC11-060	506,583.30	5,180,228.20	1,742.00	354.90	-64.60	309.37	UZ	Tintina
SC11-061	506,326.10	5,180,997.90	1,774.60	198.30	-66.00	242.93	UZ	Tintina
SC11-062	506,413.10	5,180,842.90	1,762.90	232.00	-58.90	215.49	UZ	Tintina
SC11-063	506,579.80	5,180,226.50	1,742.00	59.60	-58.50	222.50	UZ	Tintina
SC11-064	506,474.30	5,180,751.60	1,756.80	231.80	-61.10	185.47	UZ	Tintina
SC11-065	506,583.10	5,180,225.30	1,742.10	157.00	-68.10	161.50	UZ	Tintina
SC11-066	506,377.40	5,180,915.00	1,768.10	252.10	-62.60	215.49	UZ	Tintina
SC11-067	506,498.50	5,180,185.60	1,746.80	347.60	-62.70	197.80	UZ	Tintina
SC11-068	506,661.80	5,180,475.10	1,744.60	0.00	-90.00	148.74	UZ	Tintina
SC11-069	506,494.50	5,180,189.20	1,746.80	60.70	-70.10	182.88	UZ	Tintina
SC11-070	506,652.20	5,180,471.60	1,745.40	180.40	-61.70	173.13	UZ	Tintina

table continues…

BHID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Depth (m)	Zone	Company
SC11-071	506,498.10	5,180,190.30	1,746.60	169.60	-68.60	167.64	UZ	Tintina
SC11-072	506,667.30	5,180,527.70	1,743.90	57.30	-62.70	148.70	UZ	Tintina
SC11-074	506,396.70	5,180,532.10	1,768.10	2.00	-59.80	206.04	UZ	Tintina
SC11-076	509,126.50	5,179,808.00	1,727.70	0.00	-90.00	581.56	MZ	Tintina
SC11-077	508,996.20	5,179,458.00	1,733.50	0.00	-90.00	779.68	MZ	Tintina
SC11-079	509,118.90	5,179,455.80	1,731.30	0.00	-90.00	723.60	MZ	Tintina
SC11-081	509,201.50	5,179,566.10	1,726.30	300.00	-80.00	639.47	MZ	Tintina
SC11-082	509,002.00	5,179,582.40	1,729.20	30.00	-82.00	682.14	MZ	Tintina
SC11-084	509,001.40	5,179,824.70	1,728.00	0.00	-90.00	594.06	MZ	Tintina
SC11-085	509,268.00	5,179,413.10	1,731.80	0.00	-90.00	767.59	MZ	Tintina
SC11-090	509,154.60	5,179,322.20	1,735.20	0.00	-90.00	773.58	MZ	Tintina
SC-50	506,961.67	5,180,889.97	1,713.72	0.00	-90.00	408.43	LZ	UII
SC-51	507,120.63	5,180,888.73	1,709.12	0.00	-90.00	415.44	LZ	UII
SC-55	507,121.15	5,180,674.14	1,711.25	0.00	-90.00	561.75	LZ	UII
SC-57	507,274.68	5,180,679.07	1,711.07	0.00	-90.00	496.52	LZ	UII
SC-62	506,418.55	5,181,143.93	1,799.49	0.00	-90.00	139.14	UZ	UII
SC-63	506,603.44	5,180,870.46	1,776.16	0.00	-90.00	557.78	UZ	UII
SC-64	506,344.48	5,181,140.23	1,801.33	0.00	-90.00	131.80	UZ	UII
SC-71	506,485.12	5,180,363.67	1,759.17	0.00	-90.00	221.59	UZ	UII
SC-80	509,162.80	5,179,686.50	1,726.20	0.00	-90.00	610.51	MZ	CAI
SC-86	509,053.27	5,179,710.62	1,725.53	0.00	-90.00	615.39	MZ	CAI
SC-87	509,058.62	5,179,561.49	1,729.50	0.00	-90.00	681.84	MZ	CAI
SC-87W	509,058.62	5,179,561.49	1,729.50	0.00	-90.00	651.05	MZ	CAI
SC-90	507,301.03	5,180,830.94	1,709.83	0.00	-90.00	439.52	LZ	UII
SCC-17	506,325.71	5,181,005.73	1,775.10	0.00	-88.50	407.82	UZ & LZ	UII
SCC-19	506,352.54	5,181,141.66	1,801.93	0.00	-90.00	252.07	UZ	UII
SCC-20	506,192.42	5,181,052.08	1,788.52	0.00	-90.00	442.57	LZ	UII
SCC-21	506,443.38	5,180,962.17	1,778.90	0.00	-90.00	417.58	UZ & LZ	UII
SCC-22	506,359.09	5,180,858.10	1,767.20	0.00	-90.00	499.26	UZ	UII
SCC-23	506,487.55	5,181,145.18	1,802.31	0.00	-90.00	336.19	UZ	UII
SCC-25	506,784.21	5,180,751.82	1,739.33	0.00	-90.00	121.62	UZ	UII
SCC-30	506,476.46	5,180,746.01	1,756.36	0.00	-90.00	128.02	UZ	UII
SCC-33	506,580.14	5,181,154.03	1,795.89	0.00	-90.00	134.72	UZ	UII
SCC-34	506,573.56	5,180,998.83	1,799.61	0.00	-90.00	621.49	UZ & LZ	UII
SCC-35	506,754.46	5,181,134.17	1,758.12	0.00	-90.00	176.17	UZ	UII
SCC-36	506,768.58	5,181,002.38	1,752.61	0.00	-90.00	425.20	UZ	BHP
SCC-37	506,310.12	5,180,689.80	1,801.40	0.00	-90.00	240.49	UZ	BHP
SCC-38	506,483.85	5,180,586.13	1,759.04	0.00	-90.00	170.69	UZ	BHP
SCC-40	506,632.62	5,180,733.63	1,750.82	0.00	-90.00	611.43	UZ	BHP
SCC-41	506,631.22	5,180,563.44	1,742.26	155.00	-89.30	156.36	UZ	BHP
SCC-46	507,120.97	5,180,828.54	1,709.90	0.00	-90.00	456.59	LZ	BHP

Figure 10.1 Johnny Lee Drillhole Collar Locations

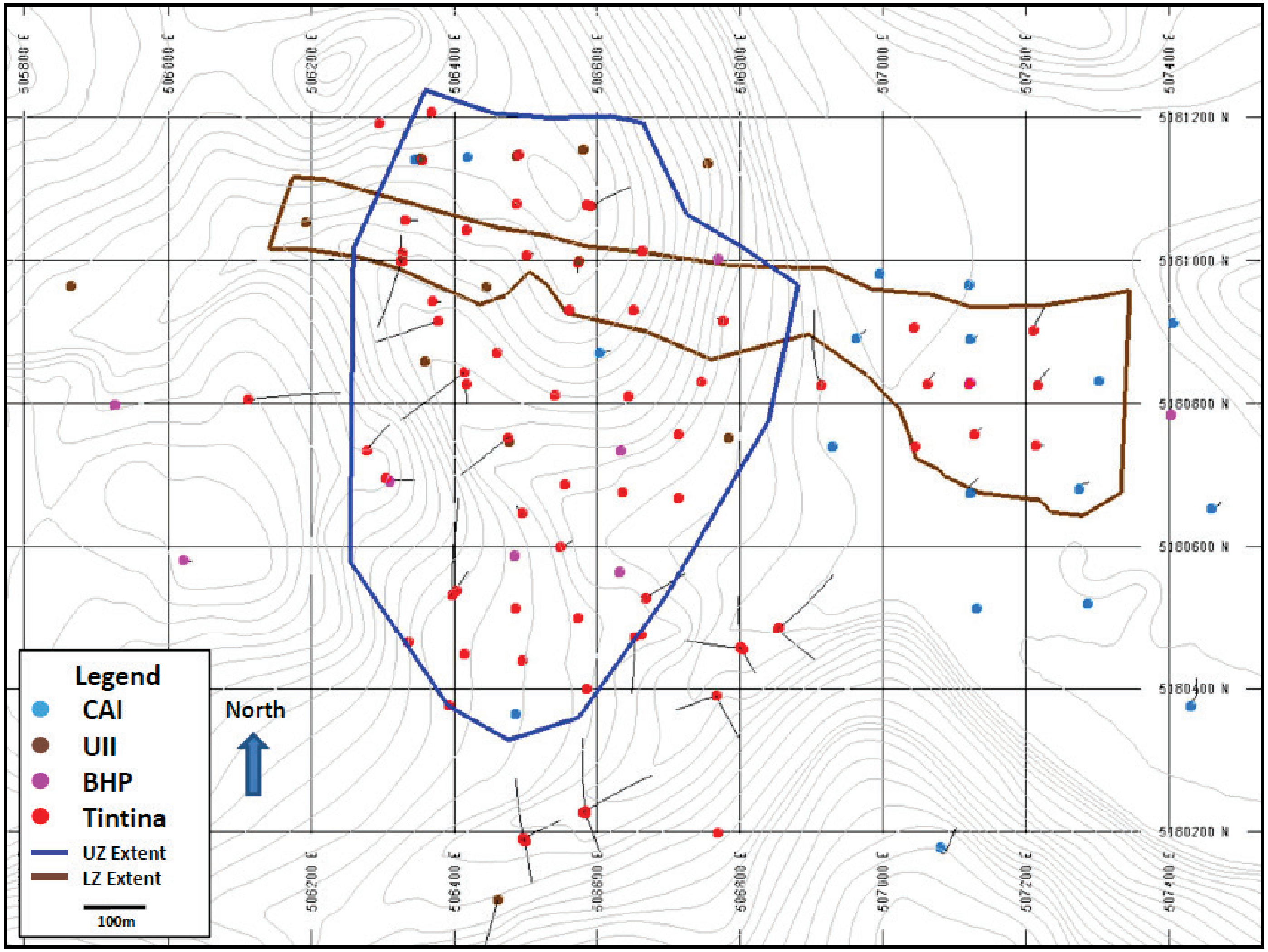
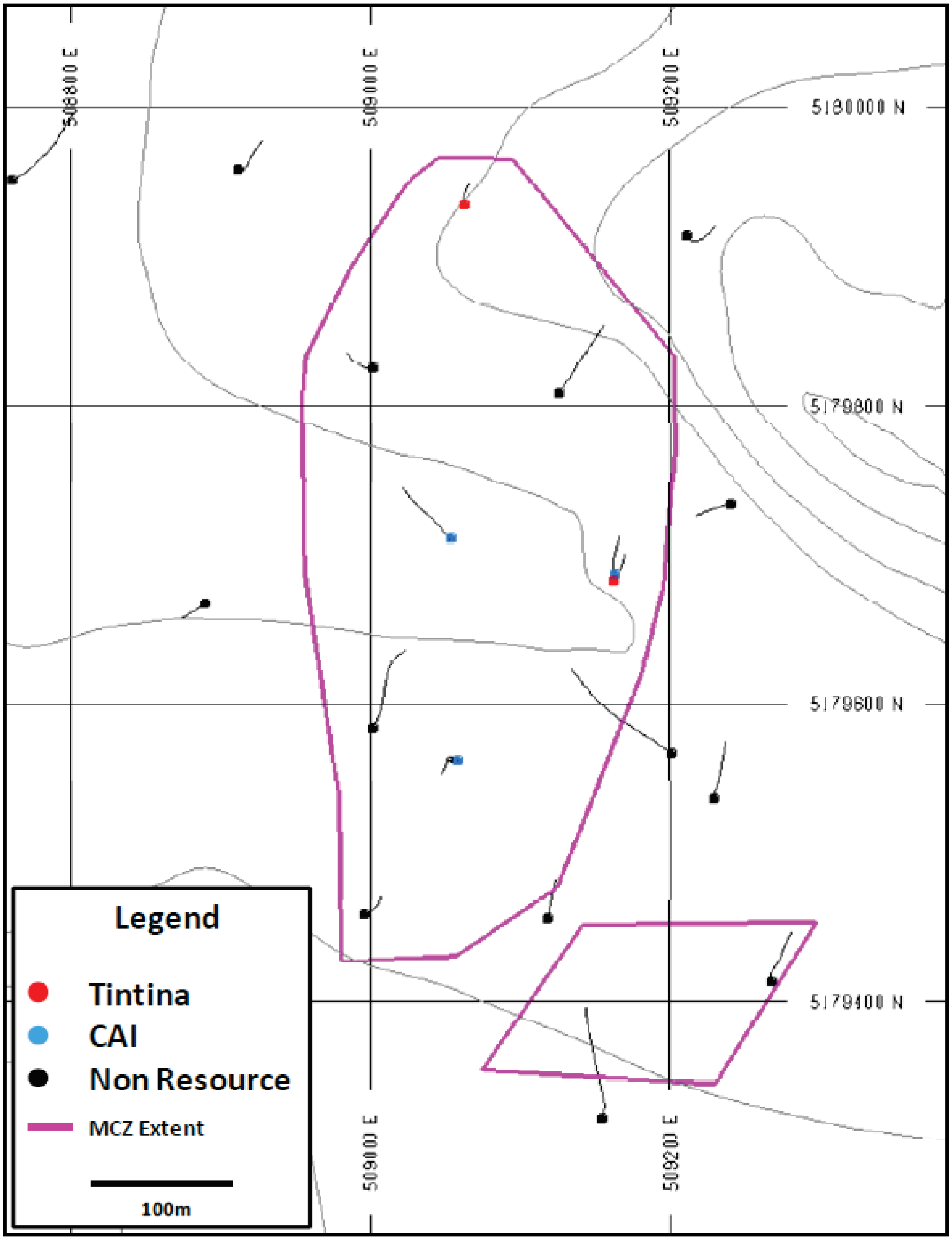


Figure 10.2 Lowry MZ Drillhole Locations



10.2 RELEVANT JOHNNY LEE UZ RESULTS

Samples were collected by Tintina, CAI, UII, and BHP from 73 diamond core holes from the Johnny Lee UZ covering an area measuring approximately 550 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 2 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)	UZ	Company
SC10-001	118.26	125.00	6.74	3.44	0.18	12.9	31	Tintina
SC10-002	131.31	138.77	7.46	2.48	0.36	5.8	31	Tintina
SC10-003	131.88	139.50	7.62	3.21	0.17	21.6	31	Tintina
SC10-004	107.90	118.41	10.51	3.03	0.15	10.0	31	Tintina
SC11-013	50.99	56.82	5.83	3.55	0.17	10.4	32	Tintina
SC11-014	57.85	73.15	15.30	2.21	0.10	16.3	31	Tintina
SC11-017	78.74	83.65	4.91	3.56	0.07	9.4	31	Tintina
SC11-018	90.18	94.18	4.00	2.98	0.11	13.1	31	Tintina
SC11-018	72.45	76.76	4.31	2.36	0.15	11.4	32	Tintina
SC11-019	99.50	110.84	11.34	2.09	0.06	11.8	32	Tintina
SC11-021	150.85	154.23	3.38	2.69	0.03	14.5	31	Tintina
SC11-027	76.20	83.65	7.45	2.52	0.16	10.3	32	Tintina
SC11-028	144.07	148.21	4.14	2.78	0.10	22.0	32	Tintina
SC11-029	85.75	88.76	3.01	2.99	0.13	8.6	31	Tintina
SC11-029	63.98	73.00	9.02	2.66	0.15	13.5	32	Tintina
SC11-030	115.50	127.00	11.50	2.05	0.14	8.5	31	Tintina
SC11-031	93.00	103.60	10.60	2.05	0.11	8.1	31	Tintina
SC11-033	202.07	205.07	3.00	2.00	0.12	32.4	31	Tintina
SC11-034	179.87	183.24	3.37	2.25	0.04	19.8	31	Tintina
SC11-035	100.88	103.88	3.00	2.30	0.06	14.7	32	Tintina
SC11-039	124.36	133.70	9.34	2.49	0.31	19.9	31	Tintina
SC11-041	92.91	99.71	6.80	2.45	0.06	11.3	31	Tintina
SC11-041	76.24	86.37	10.13	2.08	0.11	9.2	32	Tintina
SC11-044	113.70	120.00	6.30	3.15	0.16	21.8	31	Tintina
SC11-045	119.20	122.30	3.10	2.13	0.03	9.2	31	Tintina

table continues…

BHID	From (m)	To (m)	Length (m)	Cu (%)	Co (%)	Ag (g/t)	UZ	Company
SC11-046	98.47	108.10	9.63	2.65	0.11	9.3	32	Tintina
SC11-046	111.76	122.86	11.10	2.30	0.18	19.1	31	Tintina
SC11-054	95.06	98.06	3.00	2.21	0.08	47.8	31	Tintina
SC11-055	104.50	123.92	19.42	2.19	0.07	14.8	31	Tintina
SC11-060	121.40	149.18	27.78	2.64	0.06	19.9	31	Tintina
SC11-061	157.75	165.92	8.17	2.09	0.14	18.2	31	Tintina
SC11-062	161.49	168.63	7.14	2.92	0.14	43.9	31	Tintina
SC11-064	107.58	116.00	8.42	4.80	0.12	53.6	32	Tintina
SC11-066	156.09	161.10	5.01	3.26	0.17	17.2	31	Tintina
SC11-068	109.75	128.43	18.68	2.91	0.11	14.4	31	Tintina
SC11-069	122.45	129.14	6.69	2.32	0.07	13.9	31	Tintina
SC11-072	110.97	132.00	21.03	2.70	0.10	15.7	31	Tintina
SC-64	118.87	123.90	5.03	3.26	0.10	9.4	31	CAI
SC-71	168.25	171.91	3.66	2.09	0.03	17.9	31	CAI
SCC-17	130.76	137.46	6.70	2.76	0.19	18.5	31	UII
SCC-19	115.21	123.75	8.54	3.05	0.18	10.4	31	UII
SCC-23	132.59	140.51	7.92	3.62	0.54	5.4	31	UII
SCC-30	92.35	96.62	4.27	2.84	0.17	7.0	32	UII
SCC-33	109.73	124.36	14.63	2.04	0.15	6.4	31	UII
SCC-34	108.20	119.79	11.59	2.24	0.13	9.9	31	UII
SCC-36	31.39	42.37	10.98	2.30	0.10	6.1	31	BHP

10.3 RELEVANT JOHNNY LEE LZ RESULTS

Samples were collected by Tintina, CAI, UII, and BHP from 25 diamond core holes from the LZ covering an area measuring approximately 1,200 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 10.

Table 10.6 Relevant Johnny Lee LZ Intervals

BHID	From (m)	To (m)	Length (m)	Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)	Company
SC10-003	350.40	351.69	1.29	4.08	601	4.3	0.57	Tintina
SC10-004	414.00	418.05	4.05	10.84	765	8.3	0.21	Tintina
SC10-005	401.00	412.15	11.15	4.99	691	3.9	0.20	Tintina
SC11-007	409.66	411.24	1.58	1.38	91	3.2	0.07	Tintina
SC11-008	353.38	357.40	4.02	1.56	239	3.8	0.38	Tintina

table continues…

BHID	From (m)	To (m)	Length (m)	Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)	Company
SC11-009	415.42	416.67	1.25	1.35	10	1.0	0.06	Tintina
SC11-011	409.65	422.70	13.05	3.18	179	2.5	0.35	Tintina
SC11-012	384.65	387.55	2.90	2.37	1,240	6.8	1.74	Tintina
SC11-015	449.29	456.59	7.30	3.14	424	6.1	0.46	Tintina
SC11-029	437.00	441.50	4.50	11.39	1,941	8.0	0.30	Tintina
SC11-048	359.92	367.60	7.68	6.96	865	6.4	0.69	Tintina
SC-50	367.89	370.33	2.44	7.75	104	3.2	0.39	CAI
SC-51	397.61	404.77	7.16	5.80	102	1.7	0.19	CAI
SC-55	463.60	470.28	6.68	10.16	150	12.5	0.43	CAI
SC-57	482.50	484.94	2.44	9.39	217	8.7	0.38	CAI
SC-90	383.26	384.54	1.28	11.64	243	10.9	0.09	CAI
SCC-17	355.70	358.14	2.44	6.82	488	3.0	0.34	UII
SCC-20	343.05	344.97	1.92	1.21	192	1.8	0.13	UII
SCC-21	394.56	400.66	6.10	4.78	427	4.0	0.24	UII
SCC-34	413.61	417.27	3.66	9.84	1,010	7.7	0.40	UII
SCC-46	400.35	412.76	12.41	5.71	295	2.4	0.27	BHP

10.4 Relevant Lowry MZ Intervals

Samples were collected by Tintina and CAI 14 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)	Company
SC10-006	384.02	430.64	46.62	2.57	1,159	13.3	0.01	Tintina
SC11-076	319.36	356.87	37.51	2.38	1,257	13.1	0.00	Tintina
SC11-084	324.12	334.19	10.07	2.29	1,787	14.9	0.03	Tintina
SC11-085	659.98	668.13	8.15	3.31	1,069	12.6	0.01	Tintina
SC-80	393.50	444.40	50.90	2.80	1,118	11.3	0.00	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 truck mounted core-drilling rig capable of recovering HQ (2.5") diameter core from the depths required. In holes SC10-001 through -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 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.

10.8 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.

10.9 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 drilling programs, sawn core samples were collected throughout the UZ and LZ from HQ diameter drill intersections as a part of the 2010 verification drilling program and 2011 infill drill program. 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 above where core appeared to consistently contain more than approximately 0.5% chalcopyrite, and ended 30 ft 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. Labs used were Silver Valley Laboratories in Kellogg, Idaho; Bondar Clegg in Vancouver, BC, and BHP's in-house lab at Sunnydale, California. A program of QA/QC involving regular injection of blanks, standards, and duplicates was conducted by all companies. 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 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 prepped and the pulps sent to the ALS Chemex lab in Vancouver, BC. ALS Chemex is an internationally recognized certified lab (ISO 9001:2000). ALS Chemex crushes the core to 70% less than 2 mm, pulverizes a 250 g split to 85% less than less than 75 µm, completes a four-acid digestion on a split, and carries 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.

In 2011, Tintina purchased and began using three other certified standards that were purchased from WCM Minerals. Table 11.2 summarizes the expected values from all SRMs used by Tintina. Landscaping marble pieces purchased from a local hardware store were used for blank material in 2011.

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 drilling campaign (see 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 RMI's opinion for the resource estimate, 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

The majority of the Black Butte data were collected in the 1980s and 1990s by highly reputable major mining companies (i.e. CAI, UII, and BHP). Tintina has made a concerted effort to obtain assay certificates and QA/QC data from Teck. 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 and 2011 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 11.1 summarizes the number of QA/QC samples that were submitted by Tintina from their 2010 and 2011 drilling campaigns that are relative to this report.

Table 11.1 Summary of Submitted QA/QC Samples

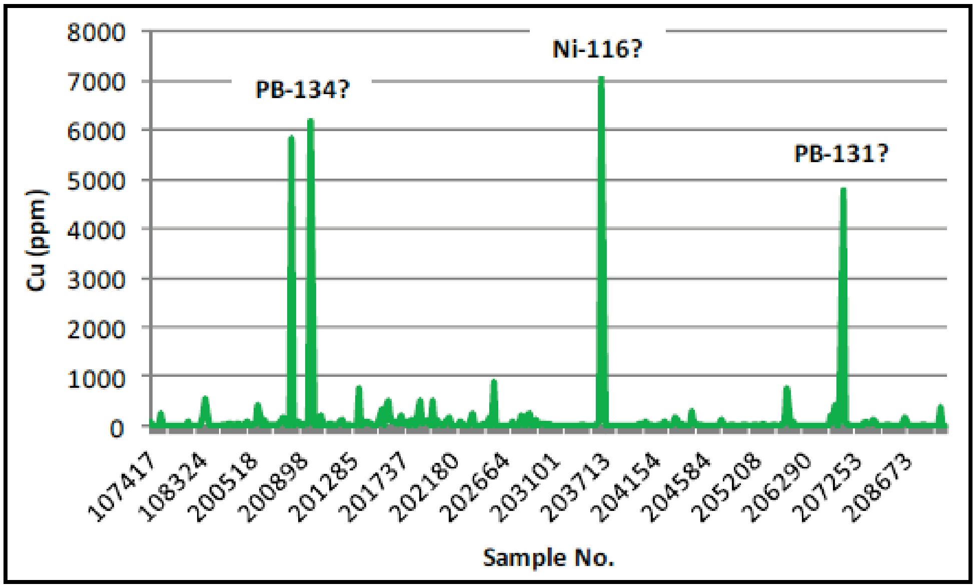
Sample Type	Description	No. of Samples
Blank	Barren outcrop – landscaping marble	348
Standard PB 131	WCM Minerals Certified Standard	173
Standard PB 134	WCM Minerals Certified Standard	111
Standard CU 145	WCM Minerals Certified Standard	22
Standard NI 116	WCM Minerals Certified Standard	20
HQ-NQ Duplicates	Prepared and assayed by ALS Chemex	321
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 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 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.

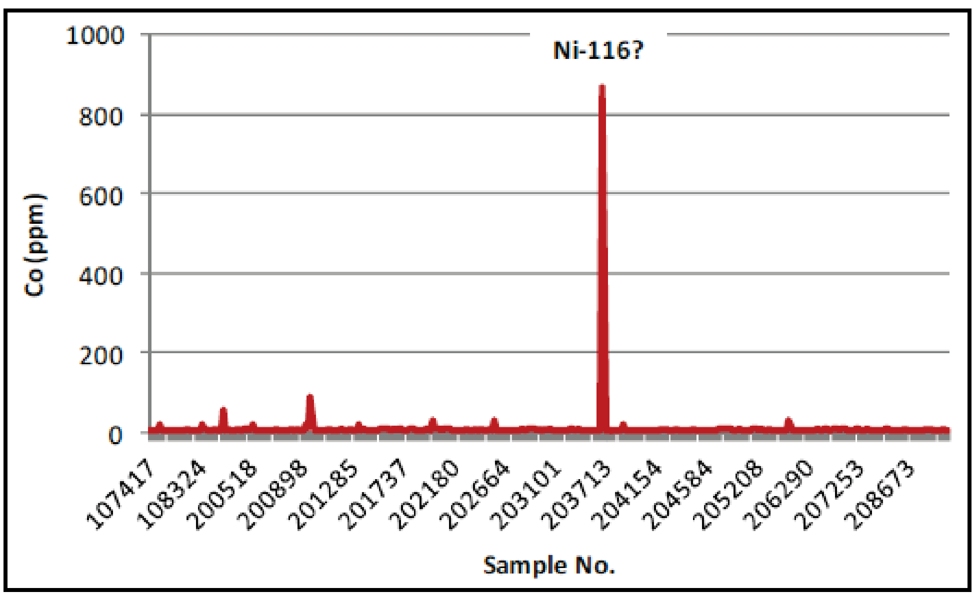
For the 2011 drill campaign, Tintina used commercially available landscape marble pieces which eliminated the anomalous copper values associated with the Newland Formation blanks used in 2010. Figure 11.2 through Figure 11.5 show values obtained from blank material submitted for the 2010 and 2011 drill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

Figure 11.2 Copper Blank Performance



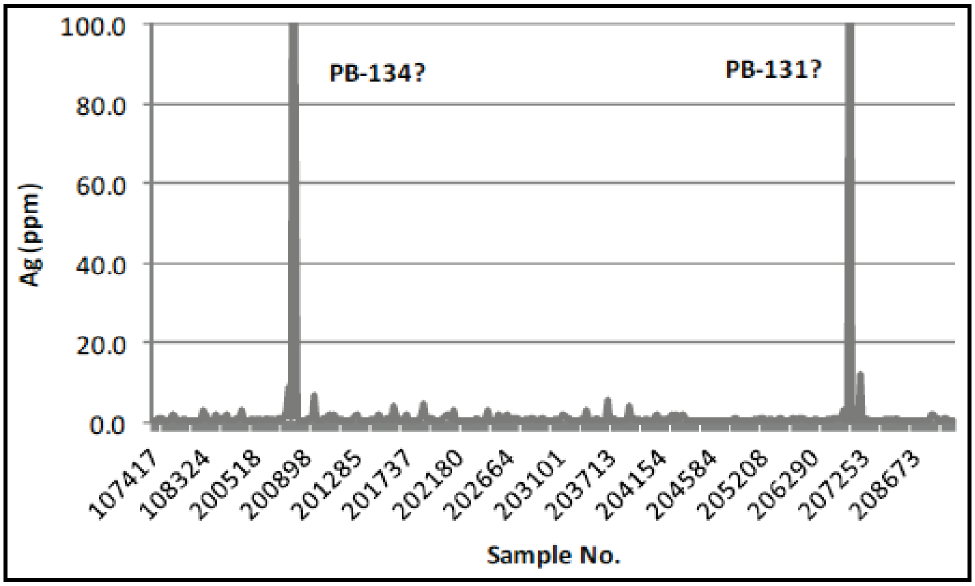
As illustrated in Figure 11.2 there were four apparent failures. The copper, lead, and zinc values that were obtained from those four barren samples are very close to three of the standards that were used by Tintina. Tintina's geologic staff was able to confirm that there was a clerical error in the database that switched the labels of blanks with standards. RMI notes that the 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



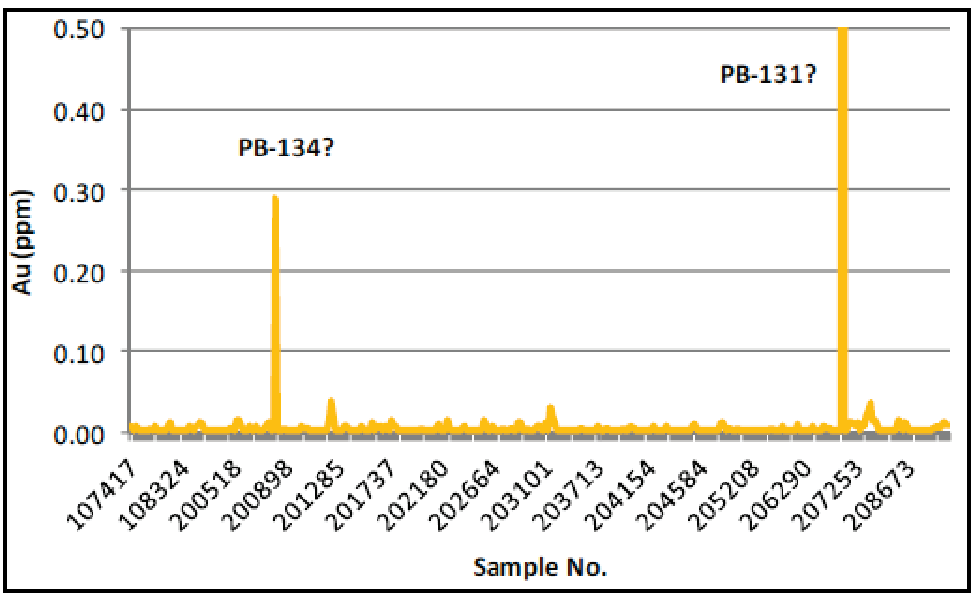
The apparent failure of sample number 203705 is no doubt due to the sample label error.

Figure 11.4 Silver Blank Performance



The apparent failure of two samples is thought to be associated with the sample label mix up that was previously discussed.

Figure 11.5 Gold Blank Performance



The apparent failure of two samples is thought to be associated with the sample label mix up that was previously discussed.

11.9 2010-2011 Tintina SRM Performance

For their 2010 and 2011 drilling programs, Tintina submitted 449 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 11.2 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.

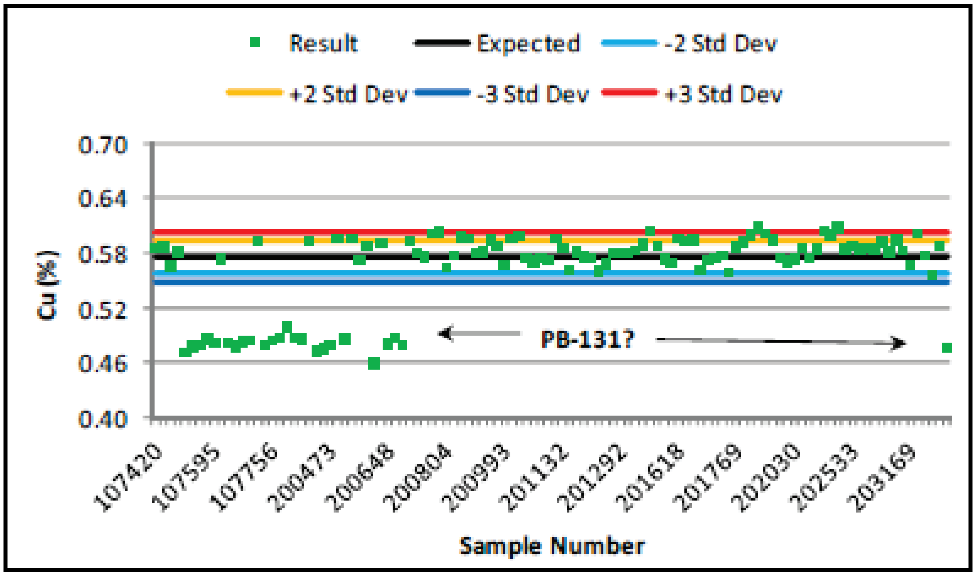
In 2010, the commercial standard PB 134, which was purchased from WCM Minerals, was inserted into the sample stream five times for the samples used by the author for estimating resources for the Johnny Lee UZ (Lechner 2010). This same standard along with three other WCM Minerals standards, (PB 131, CU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

Figure 11.6 through Figure 11.9 track the performance of SRM PB134 that was assayed by ALS Chemex for copper, lead, zinc, and silver, respectively.

Table 11.2 SRM Expected Values

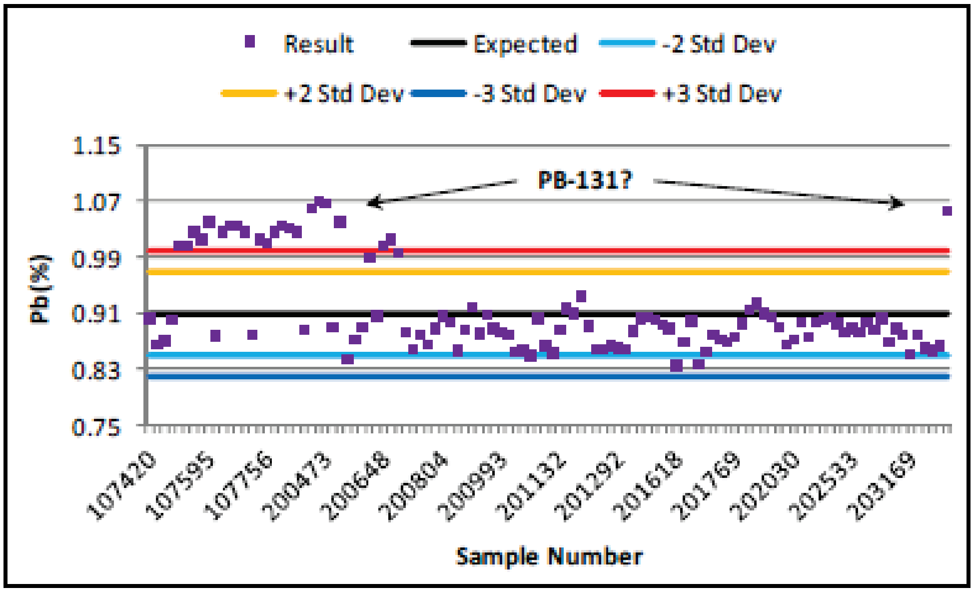
WCM Standard	Copper (%)		Ag (g/t)		Lead (%)		Zinc (%)		Co (%)	
	Expected Value	Standard Deviation	Expected Value	Standard Deviation	Expected Value	Standard Deviation	Expected Value	Standard Deviation	Expected Value	Standard Deviation
PB 131	0.470	0.012	262.0	10.8	10.400	0.035	1.890	0.059	n/a	n/a
PB 134	0.580	0.009	184.0	5.5	0.910	0.030	1.720	0.058	n/a	n/a
CU 145	3.100	0.090	93.0	3.4	n/a	n/a	n/a	n/a	n/a	n/a
NI 116	0.780	0.013	n/a	n/a	n/a	n/a	n/a	n/a	0.058	0.002

Figure 11.6 Copper Standard PB 134 Performance



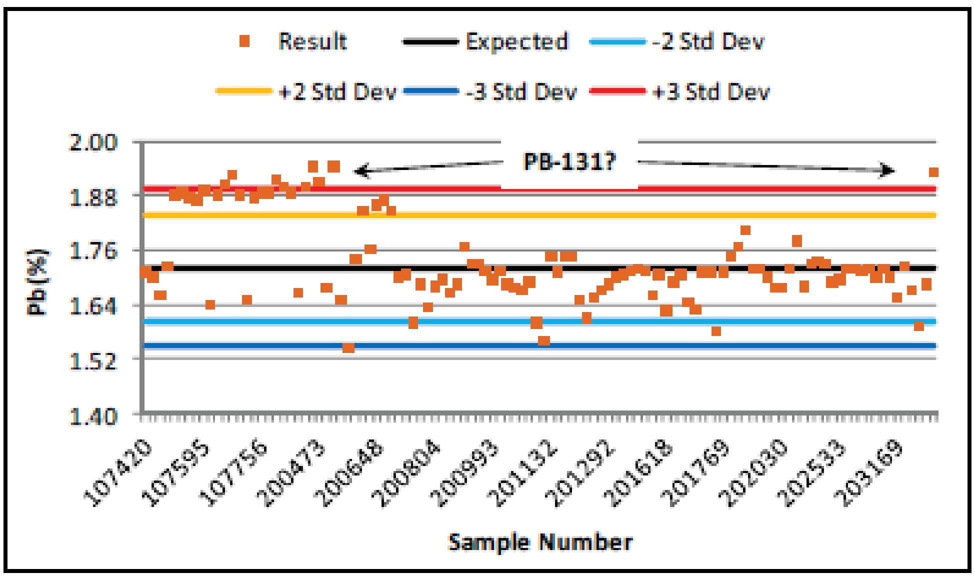
The apparent failures of the copper SRM PB 134 are associated with the sample label switch that was previously discussed. RMI has confirmed with Tintina's geologic staff that this was due to the wrong label attributed to the SRMs and blanks. ALS Chemex tended to return copper values that were higher than the expected value. A few samples were slightly above +3 standard deviations which should be used as a failure limit. RMI has recommended that Tintina closely monitor SRM performance for all future drilling campaigns. All samples outside of three standard deviations need to be compared against other QA/QC samples associated with that sample batch to see if there is a problem with the batch and if necessary have the lab re-assay the entire batch.

Figure 11.7 Lead Standard PB 134 Performance



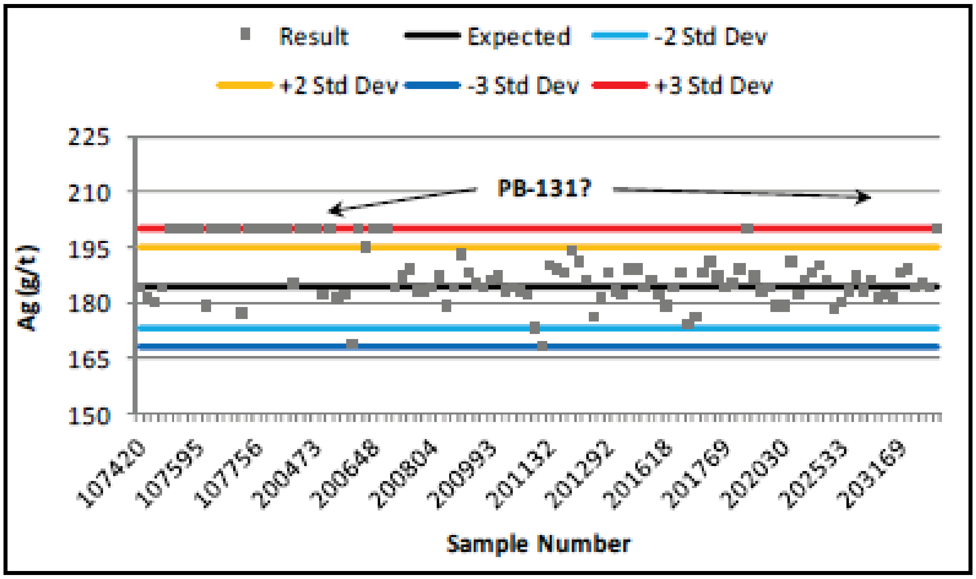
The apparent failures of the lead SRM PB 134 are associated with the sample label switch that was previously discussed.

Figure 11.8 Zinc Standard PB 134 Performance



The apparent failures of the zinc SRM PB 134 are associated with the sample label switch that was previously discussed.

Figure 11.9 Silver Standard PB 134 Performance



The apparent failures of the silver SRM PB 134 are associated with the sample label switch that was previously discussed. ALS Chemex did not perform well from late April through early June 2011 for copper and lead. A reasonable job was performed for silver.

Figure 11.10 through Figure 11.12 track the performance of WCM Minerals standard PB 131 as assayed by ALS Chemex for copper, lead, and zinc.

Figure 11.10 Copper Standard PB 131 Performance

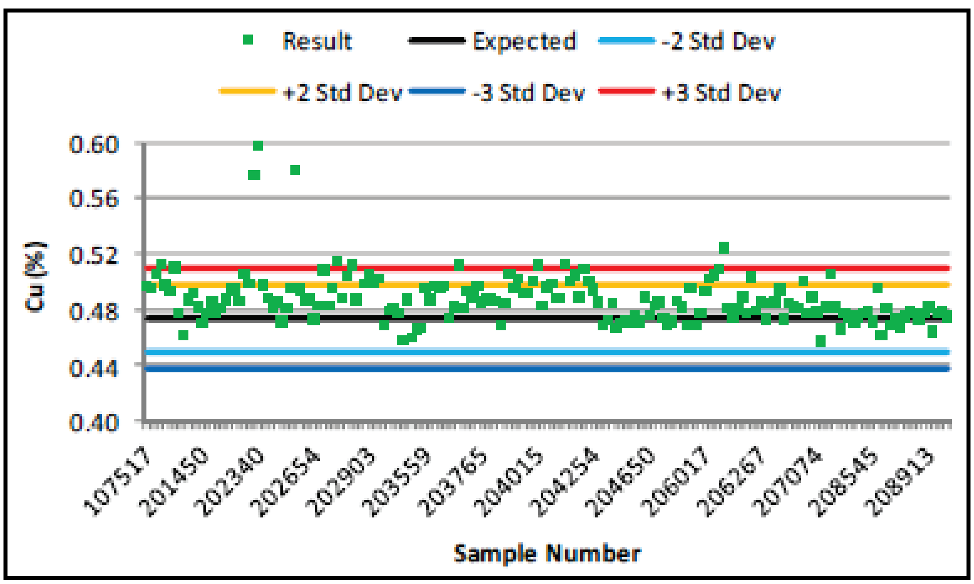


Figure 11.11 Lead Standard PB 131 Performance

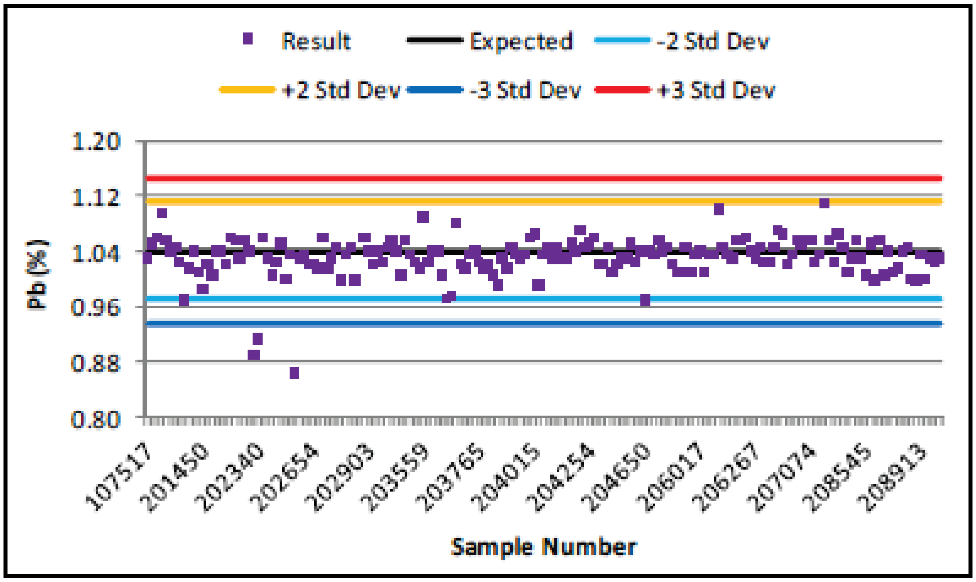
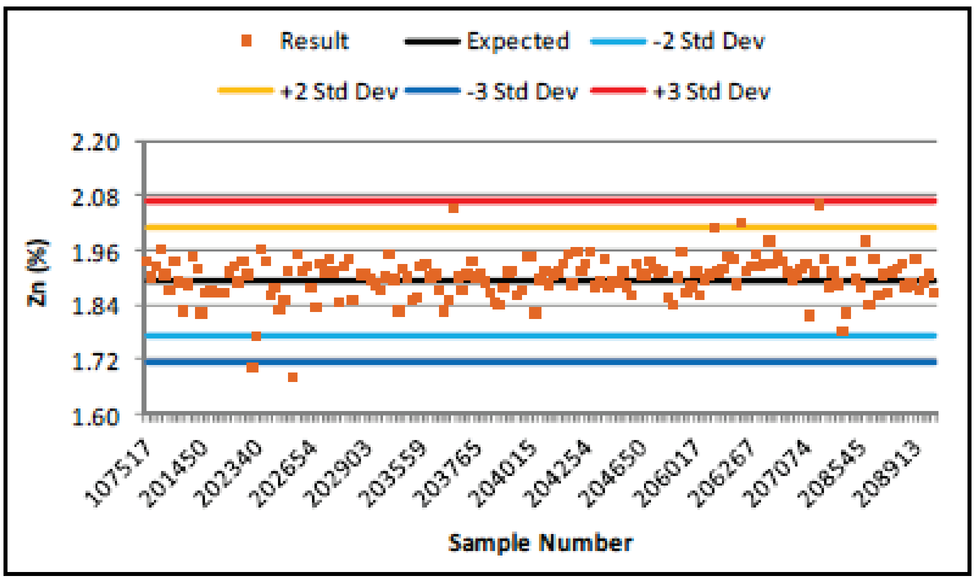


Figure 11.12 Zinc Standard PB 131 Performance



ALS Chemex did a better job of assaying standard PB 131 than PB 134 although there were a few samples in May 2011 that failed. RMI has recommended that Tintina review and modify their database entry program so that they will be immediately alerted to any QA/QC sample that is out of tolerance and to implement a remedial action plan.

11.10 2010-2011 Tintina Duplicate Samples

As a part of their QA/QC program Tintina notifies 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 and 2011 drilling campaigns, 321 duplicate pulp samples were prepared and assayed by ALS Chemex. Table 11.3 summarizes basic descriptive statistics for the original and duplicate pulp for those 321 sample pairs.

Table 11.3 Original-Duplicate Sample Comparison

	Cu (%)		Co (ppm)		Ag (g/t)		Au (g/t)	
Pair Count	Original 321	Duplicate 321	Original 321	Duplicate 321	Original 321	Duplicate 321	Original 321	Duplicate 321
Minimum	0.001	0.001	5	5	0.5	0.5	0.0025	0.0025
Maximum	20.200	20.500	8,540	8,540	115.0	185.0	1.6300	1.6250
Mean	0.781	0.799	254	254	10.1	10.6	0.0181	0.0187
Standard Deviation	3.341	3.389	916	918	16.6	19.3	0.1465	0.1434
Coefficient of Variation	4.28	4.24	3.60	3.61	1.65	1.81	8.08	7.69
Mean Grade Difference	-2%		0%		-5%		-3%	

The original pulp tended to assay slightly below the duplicate pulp assay (e.g. the mean original copper pulp grade was 2% lower than the duplicate 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 11.13 through Figure 11.16 shows QQ plots based on the original and duplicate pulp results for copper, cobalt, silver, and gold, respectively.

Figure 11.13 Copper QQ Plot – Original versus Duplicate Pulps

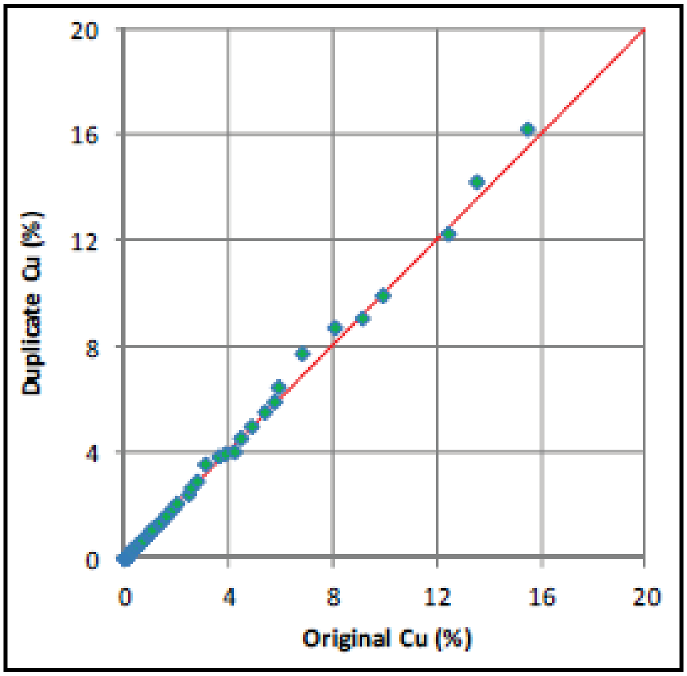


Figure 11.14 Cobalt QQ Plot – Original versus Duplicate Pulps

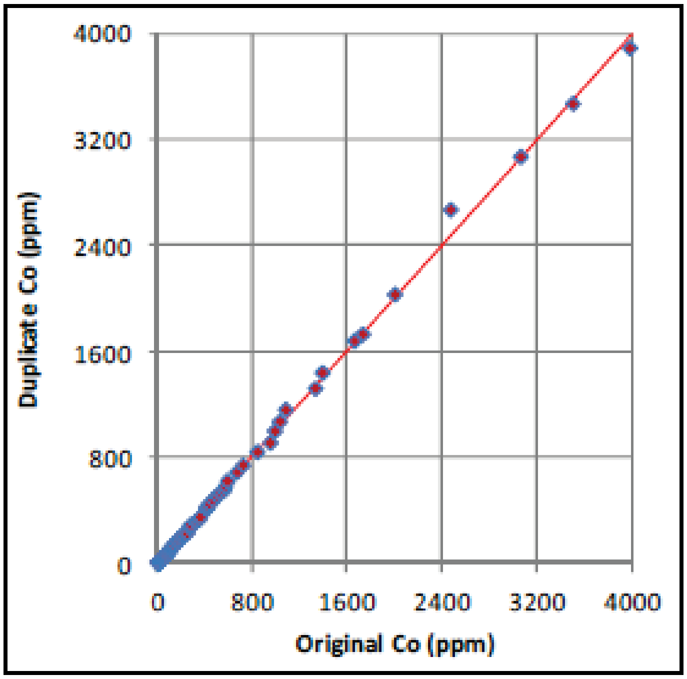


Figure 11.15 Silver QQ Plot – Original versus Duplicate Pulps

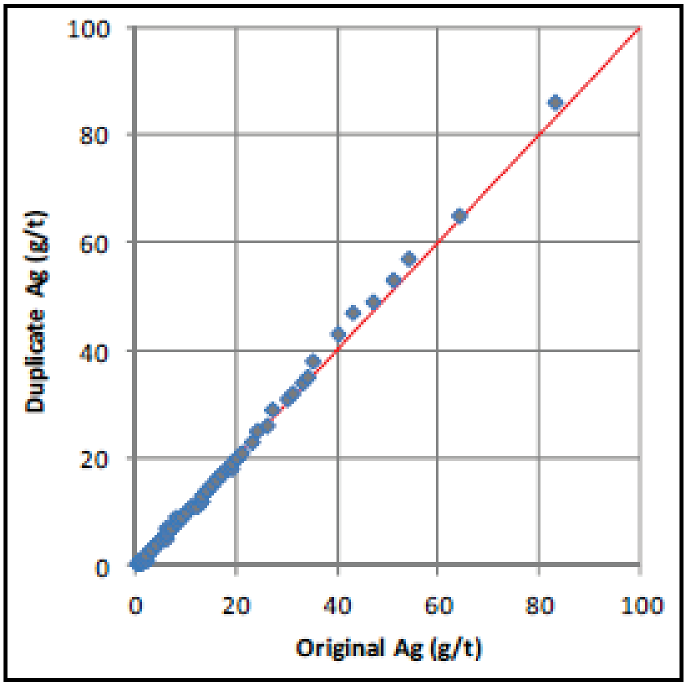
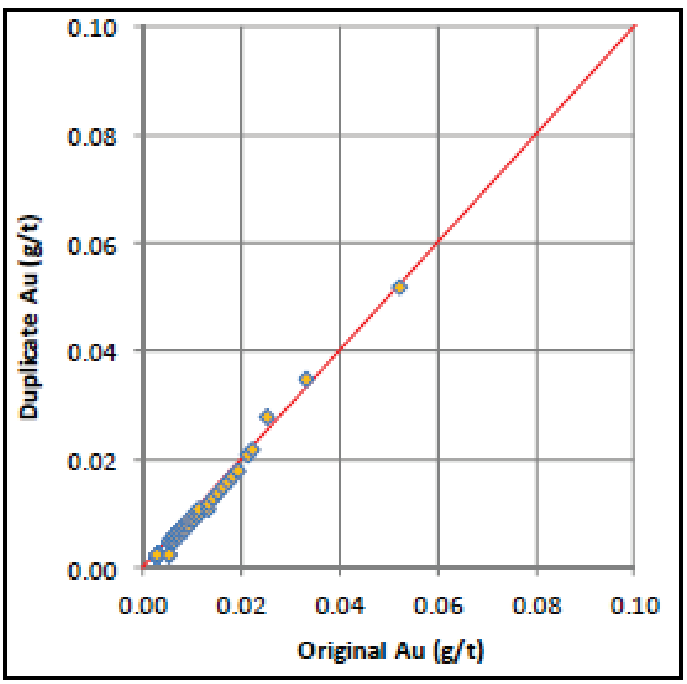


Figure 11.16 Gold QQ Plot – Original versus Duplicate Pulps



11.11 2010-2011 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 labs as illustrated by XY scatter graphs as illustrated in Figure 11.17 through Figure 11.19 for copper, cobalt, and silver, respectively.

Figure 11.17 and Figure 11.18 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 labs 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 lab. RMI also recommended that Tintina send coarse reject splits to a secondary lab so that they can prep and assay their own independent sample.

Figure 11.17 ALS Chemex versus Inspectorate – Cu

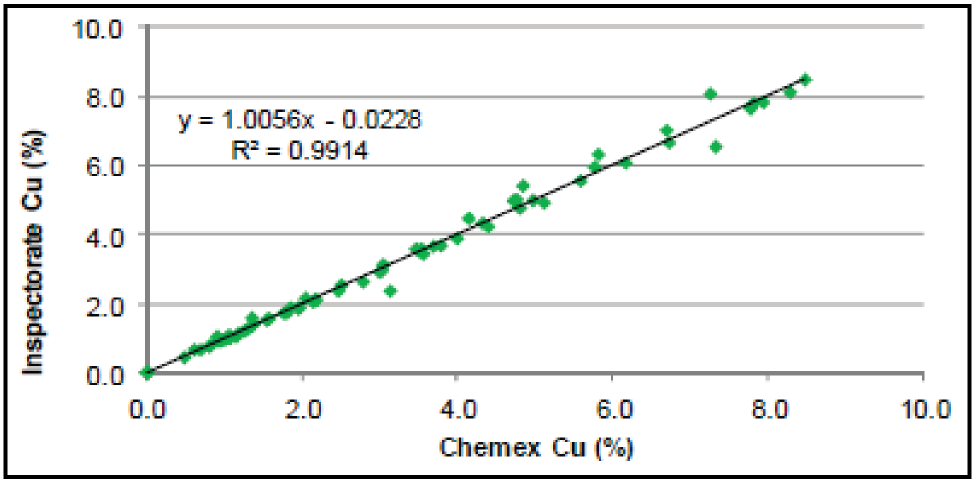


Figure 11.18 ALS Chemex versus Inspectorate – Co

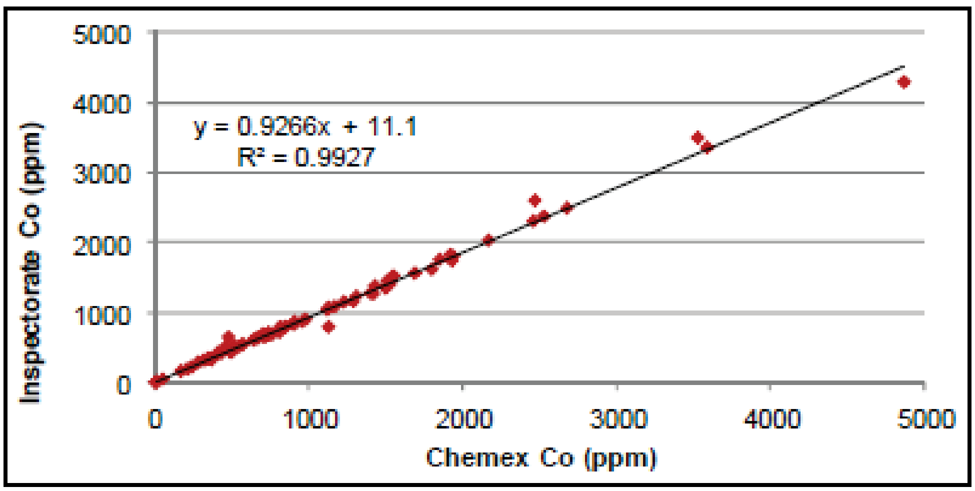
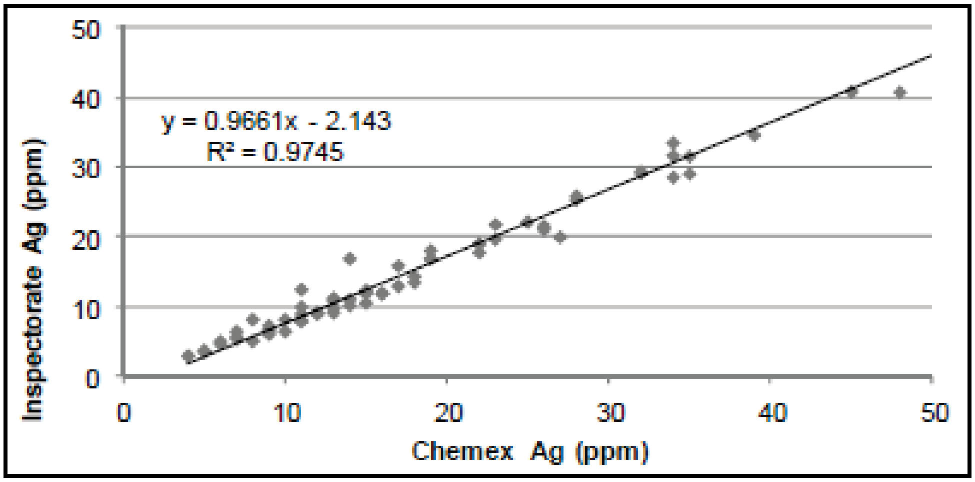


Figure 11.19 ALS Chemex versus Inspectorate – Ag



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.

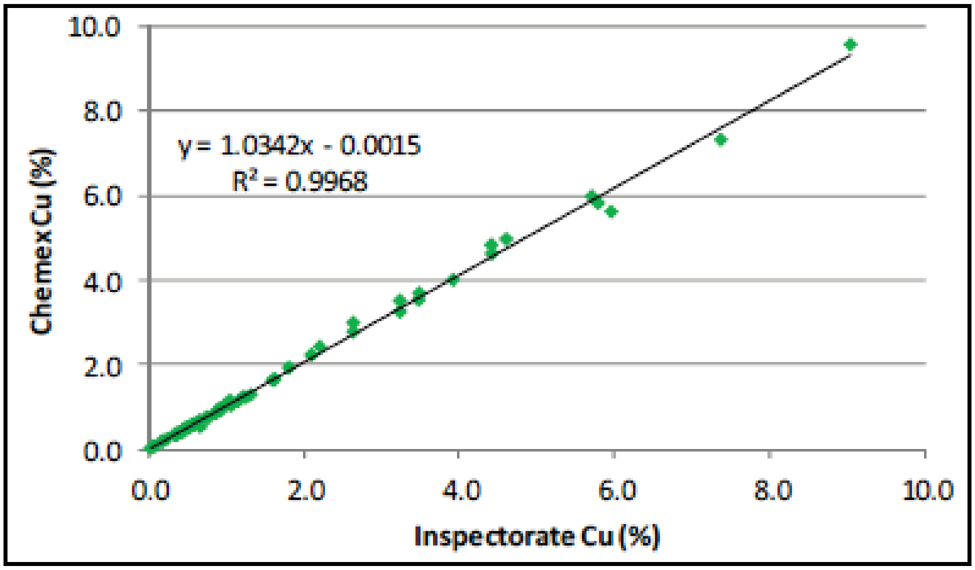
Table 11.5 Inspectorate versus Chemex Same Pulp Assay Comparison

	Cu (%)		Co (ppm)		Ag (ppm)	
Parameter	**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 labs 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 11.20 is a XY scatter graph that compares the Inspectorate copper grade with ALS Chemex.

Figure 11.20 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) and subsequent QA/QC results from Tintina's 2011 drilling campaign, 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.

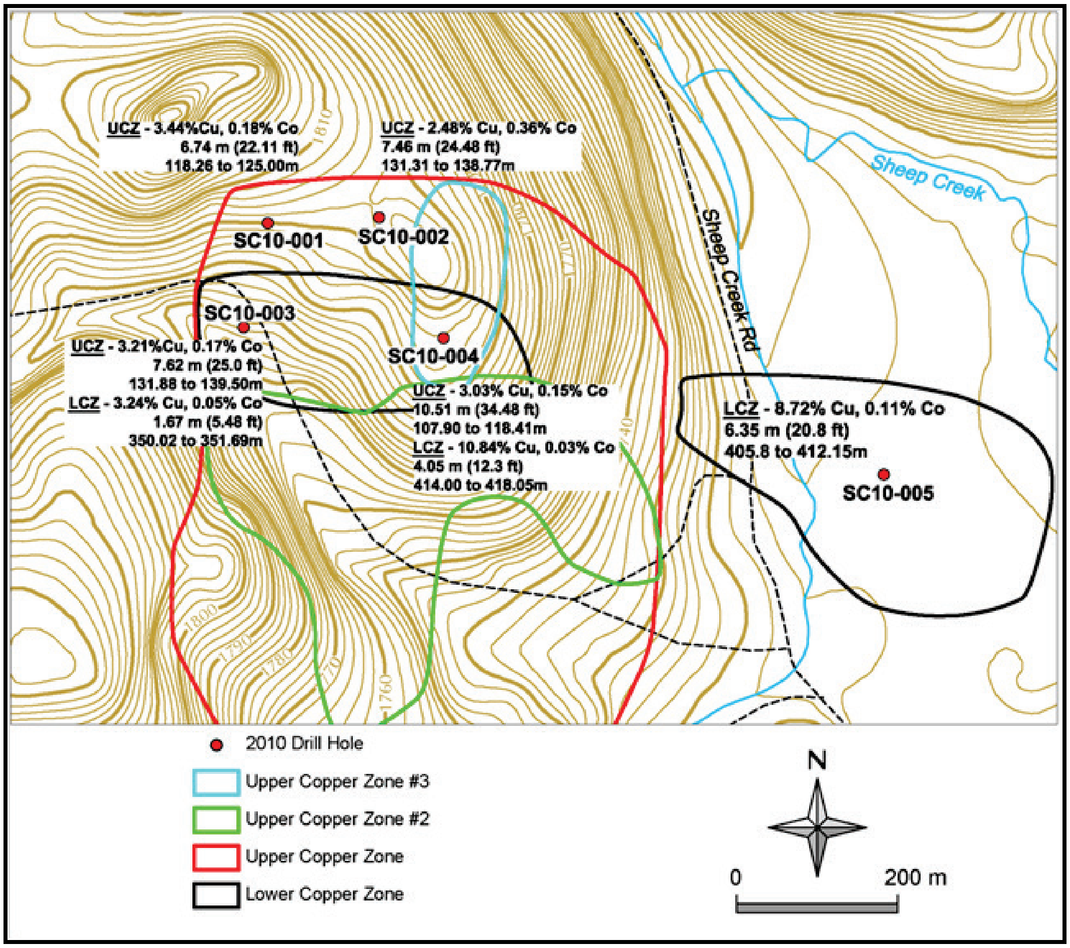
12.2 TINTINA CONFIRMATION DRILLING

For the resource estimates, RMI has stated that they had not been able to verify the drilling data collected prior to 2010 because that data is not currently available. In the absence of assay certificates and QA/QC results, Tintina drilled four twin diamond core holes in the northern portion of the USZ. A fifth hole (SC10-005) was drilled to target the LSZ and was not used in the RMI report. The Tintina holes were drilled as part of a Phase I drilling campaign designed to verify older drillhole results that were obtained from the USZ and LSZ. Table 12.1 compares the 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 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
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.

12.3 Tintina Assay Verification

RMI has estimated mineral resources for the Black Butte project four 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 verified a significant portion of the Tintina drillhole assays that were used to estimate mineral resources for the Lowry MZ. One hundred fifty copper, cobalt, and silver assay records from five 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 drilling campaigns and has been able to verify that their 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.

Figure 12.2 SC10-001 versus SCC-19

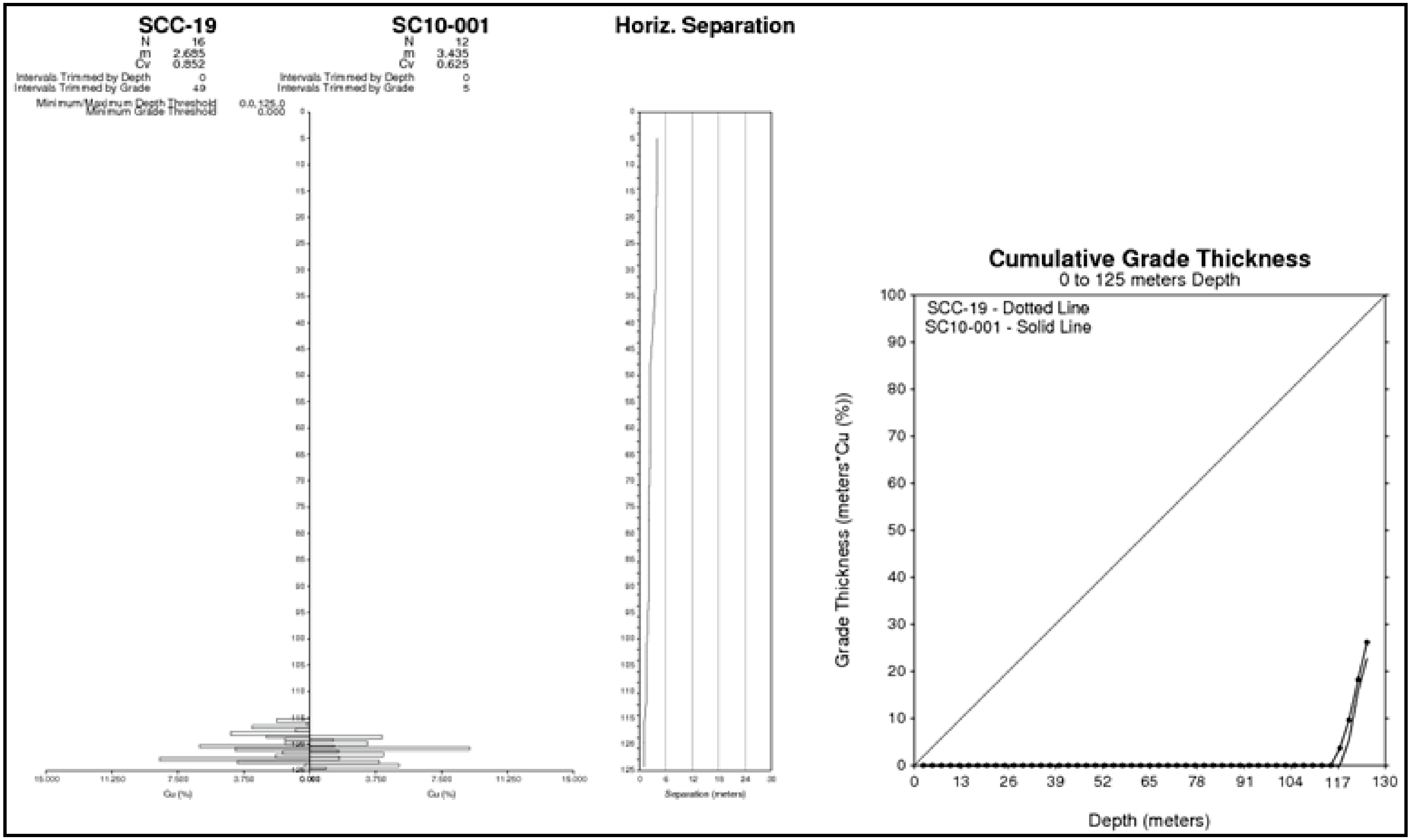


Figure 12.3 SC10-002 versus SCC-23

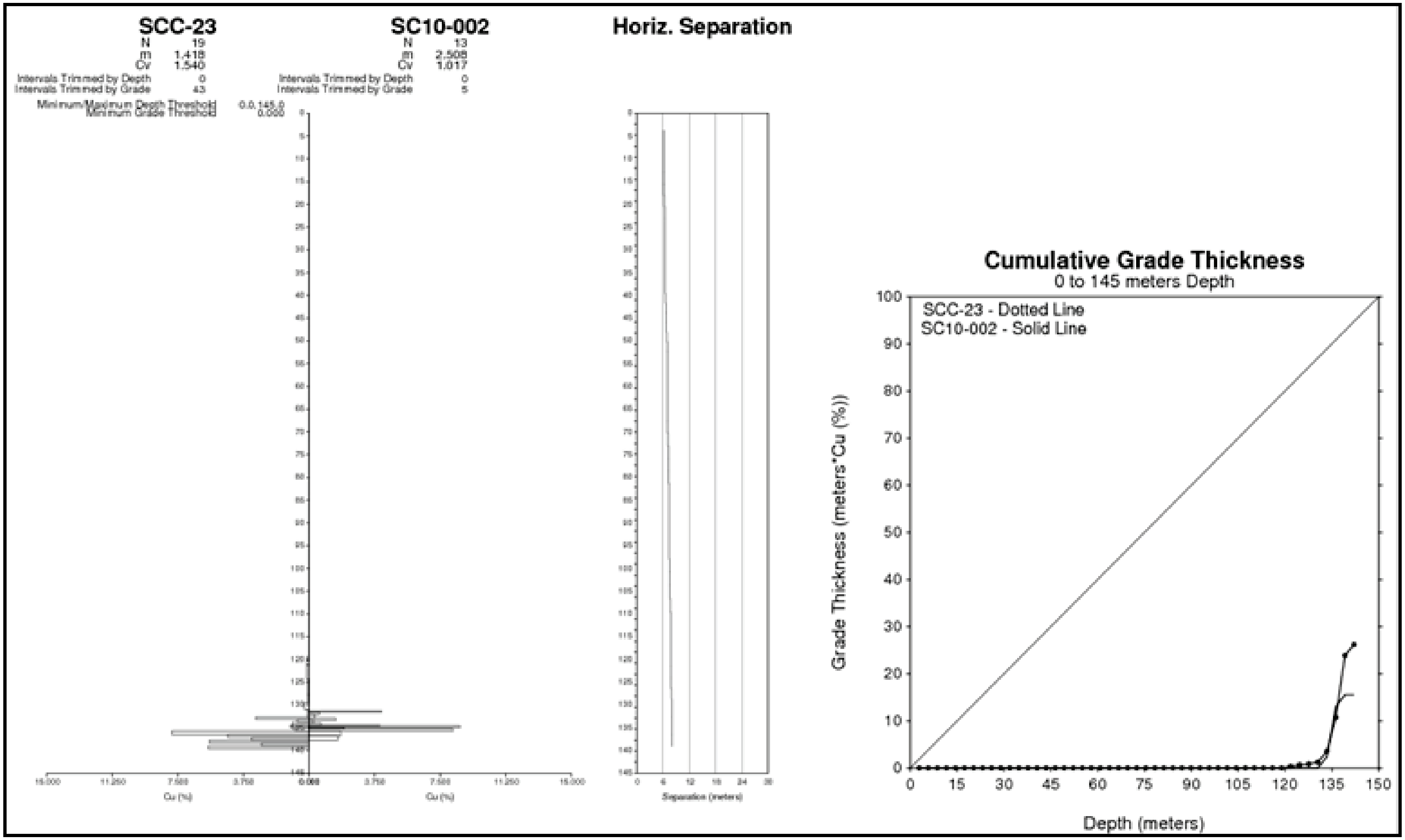


Figure 12.4 SC10-003 versus SCC-17

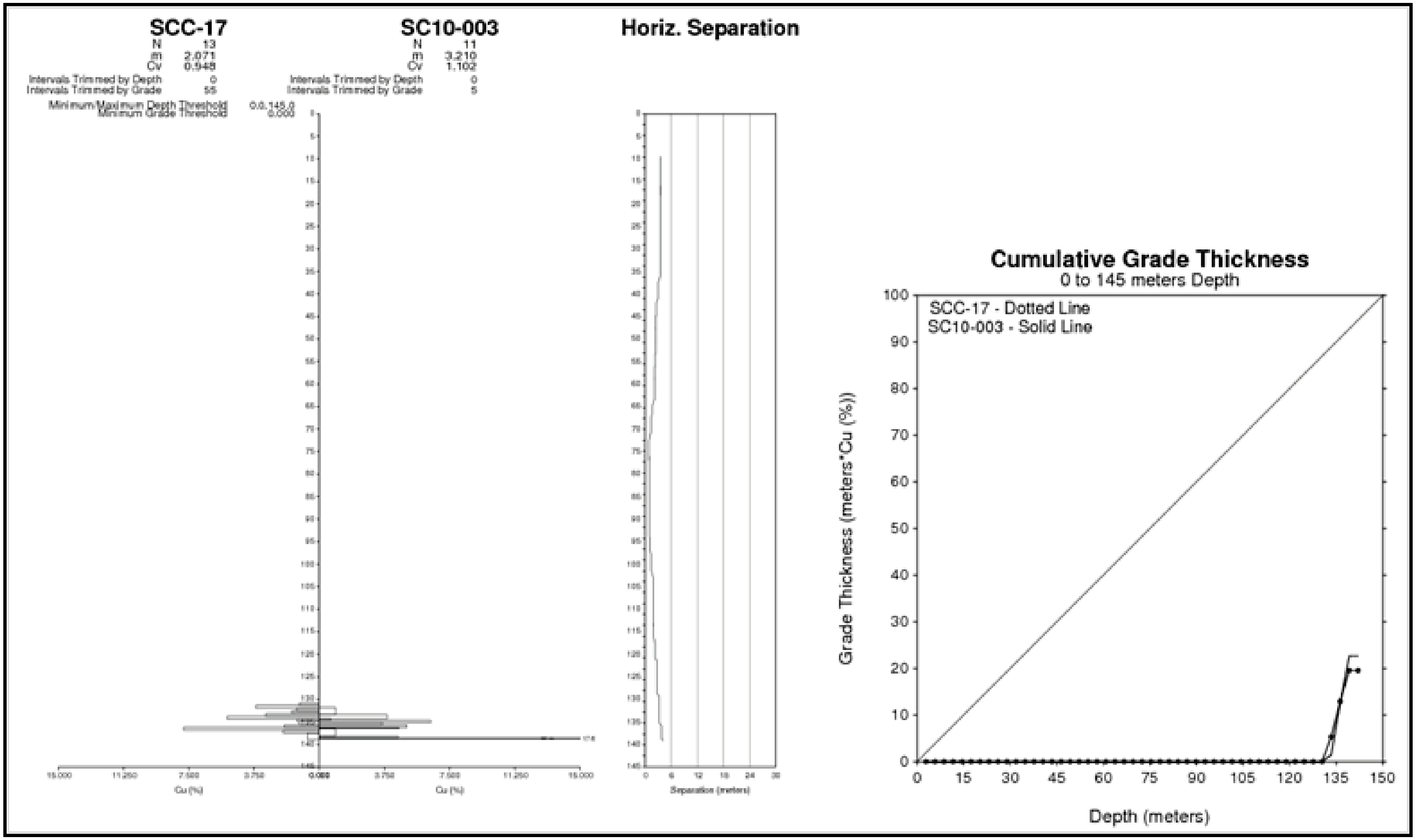
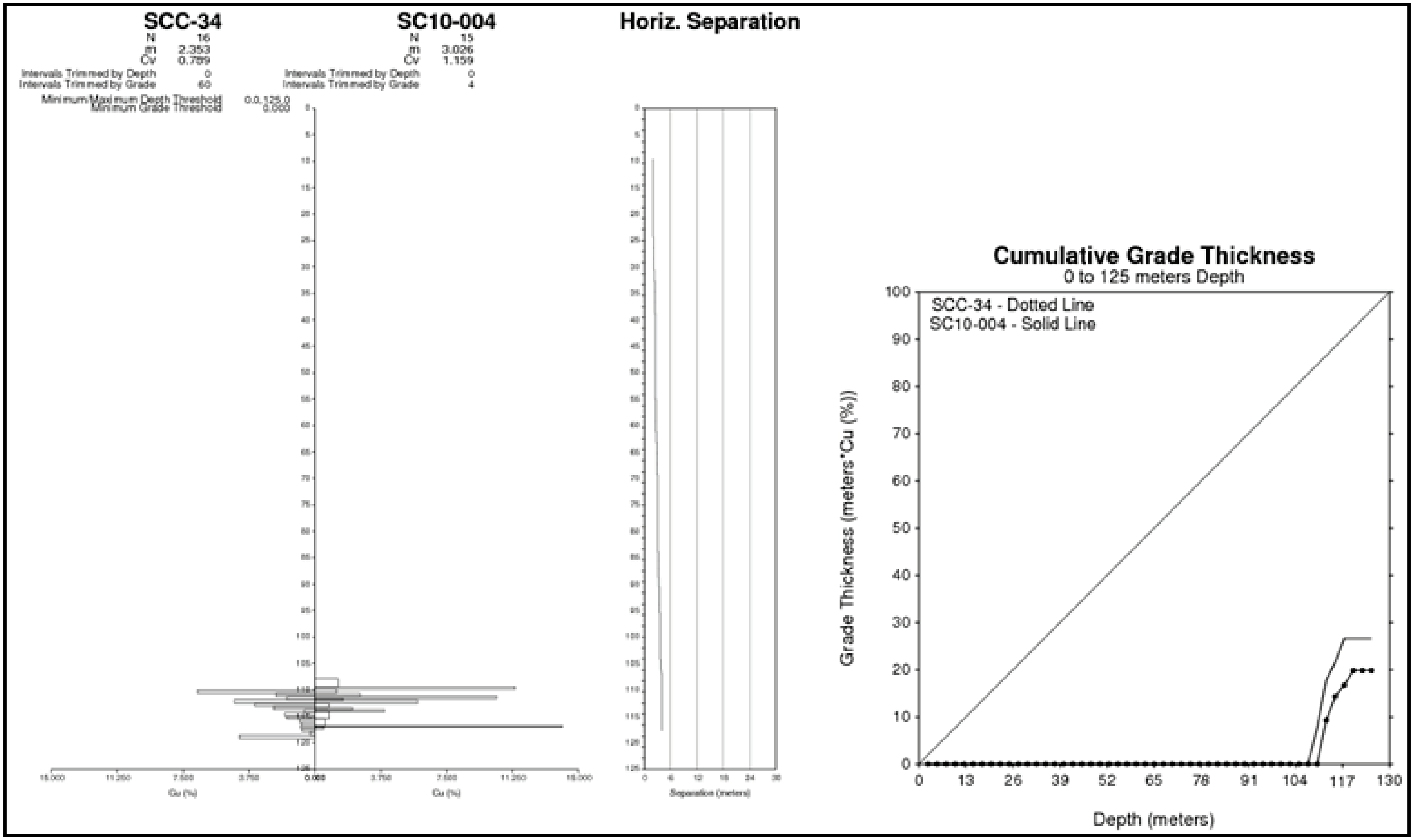


Figure 12.5 SC10-004 versus SCC-34



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Tintina contracted Arthur H. Winckers, P.Eng. to conduct various metallurgical tests to determine the flotation response of representative composite samples from the Johnny Lee UZ and LZ. The following sections pertain to work conducted by Mr. Winckers with regards to the metallurgy of the zones of interest within the deposit.

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 which were applied as the two mineralized material types will be comingled for processing in the fourth year of production.

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

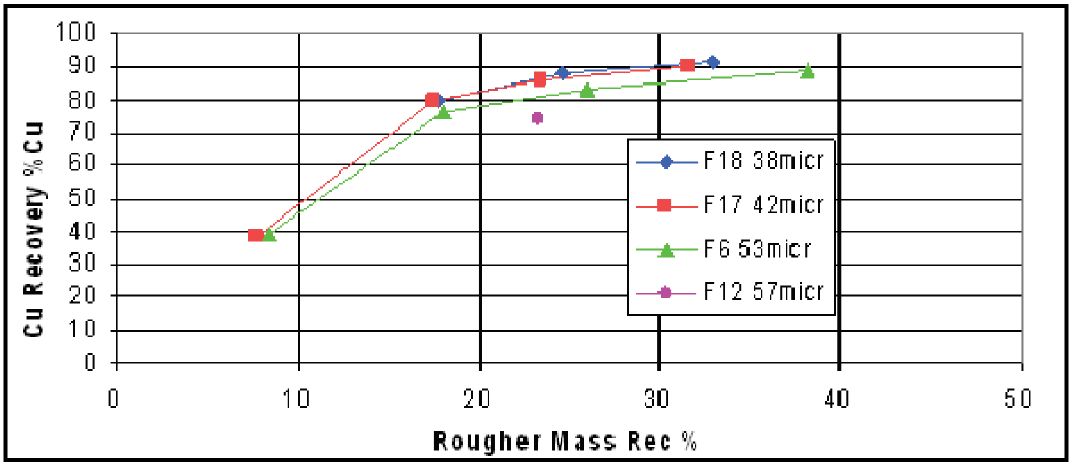
13.1.1 INVESTIGATIONS ON UZ SAMPLES

Below 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 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 mineralogy study data.

Difficulties experienced in upgrading the rougher concentrate in subsequent cleaner tests prompted a mineralogy study to determine the 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.

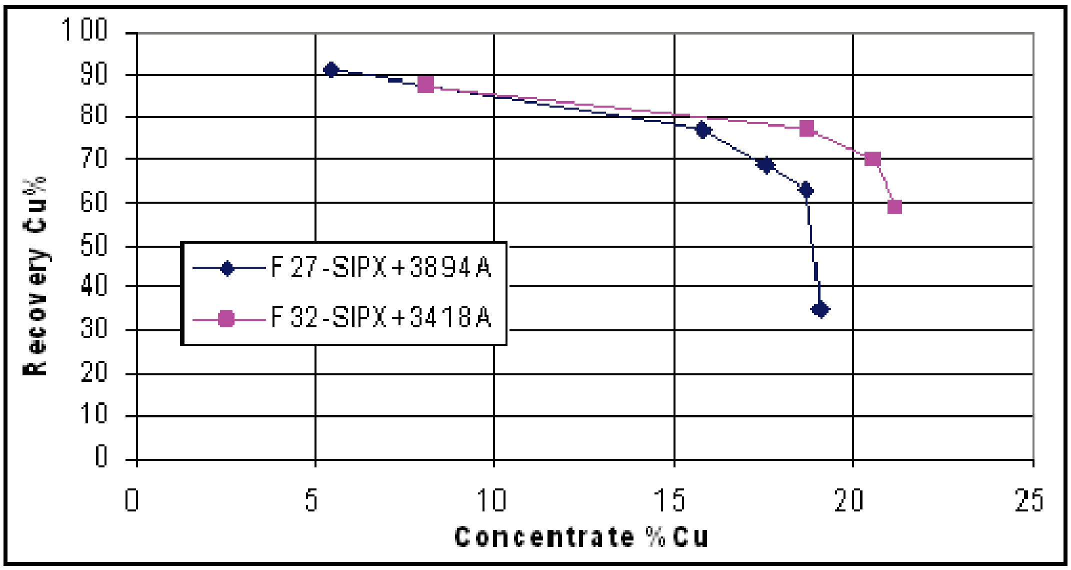
The high collector additions in the roughers needed to achieve less than 90% copper recoveries resulted in very high recoveries of nearly barren pyrite and high collector concentrations in the cleaners which adversely affected the selectivity.

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

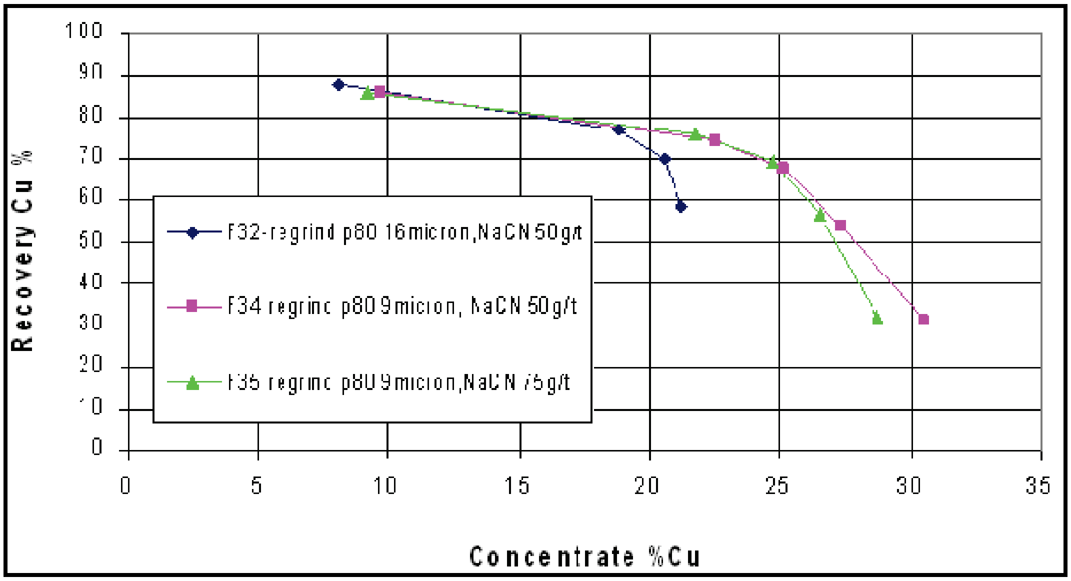
Figure 13.2 Effect of Collector on Cleaning



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

The effect of a finer regrind and sodium cyanide addition levels were explored in 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

Subsequent to the completion of the locked cycle test, the sub-composites and a sample from drillhole SC11-072, assaying 2.6% copper, were tested up to the first cleaner stage using conditions applied in the locked cycled test. There was insufficient material of Composite 36 for cleaner tests. The results are shown in Table 13.6.

Table 13.6 Results of First Cleaner Test on UZ Sub-composites

Composite	Product	Weight (%)	Cu (%)	Recovery (%)
Composite 39	Cleaner Concentrate 1-3	10.0	19.6	72.3
	Rougher Concentrate 1-5	26.9	8.9	88.1
	Calculated Feed	100.0	2.7	100.0
Composite 41A	Cleaner Concentrate 1-2	6.4	19.2	62.4
	Rougher Concentrate 1-5	29.2	5.5	82.3
	Calculated Feed	100.0	2.0	100.0
Composite 41B	Cleaner Concentrate 1-3	9.1	19.6	79.5
	Rougher Concentrate 1-5	26.9	7.5	89.2
	Calculated Feed	100.0	2.3	100.0
Composite 44	Cleaner Concentrate 1-3	6.1	20.7	64.2
	Rougher Concentrate 1-5	25.0	6.7	85.0
	Calculated Feed	100.0	2.0	100.0

table continues…

Composite	Product	Weight (%)	Cu (%)	Recovery (%)
SC11-072	Cleaner Concentrate 1-3	12.0	12.4	57.7
	Rougher Concentrate 1-5	29.9	7.2	83.9
	Calculated Feed	100.0	2.6	100.0
UZ Master Composite	Cleaner Concentrate 1-4	7.6	22.5	74.6
	Rougher Concentrate 1-5	18.8	10.2	84.2
	Calculated Feed	100.0	2.3	100.0

As shown in Figure 13.4 and Table 13.7, the flotation response of the sub-composites varies and appears to be more directly related to copper mineral liberation than to the head grade; e.g. C41B with 33% liberated chalcopyrite has the highest recovery.

Figure 13.4 Copper Grade Recovery of UZ Sub-composites

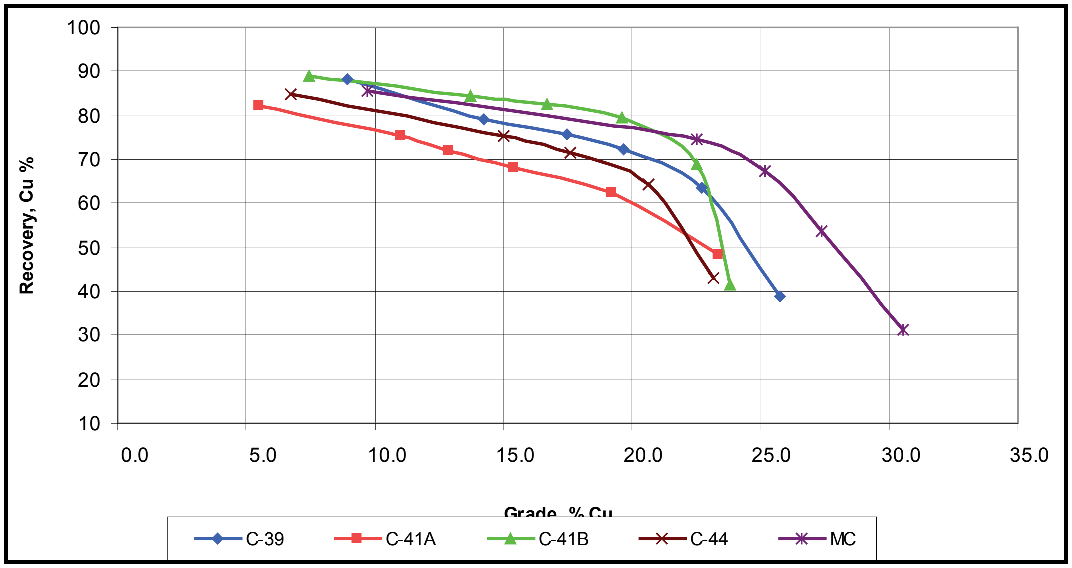


Table 13.7 UZ Sub-composites Mineral Distribution by Class of Associations

Copper Sulphides					
k80	**46**	**54**	**48**	**49**	**53**
Minerals	**Comp 36**	**Comp 39**	**Comp 41 A**	**Comp 41B**	**Comp 44**
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 results obtained on the SC11-072 sample were inferior to those obtained on the sub-composites. The mineralogy of this sample is believed to more complex and is further evidence of the variability in mineralogical texture across the UZ.

13.1.2 Investigations on LZ Samples

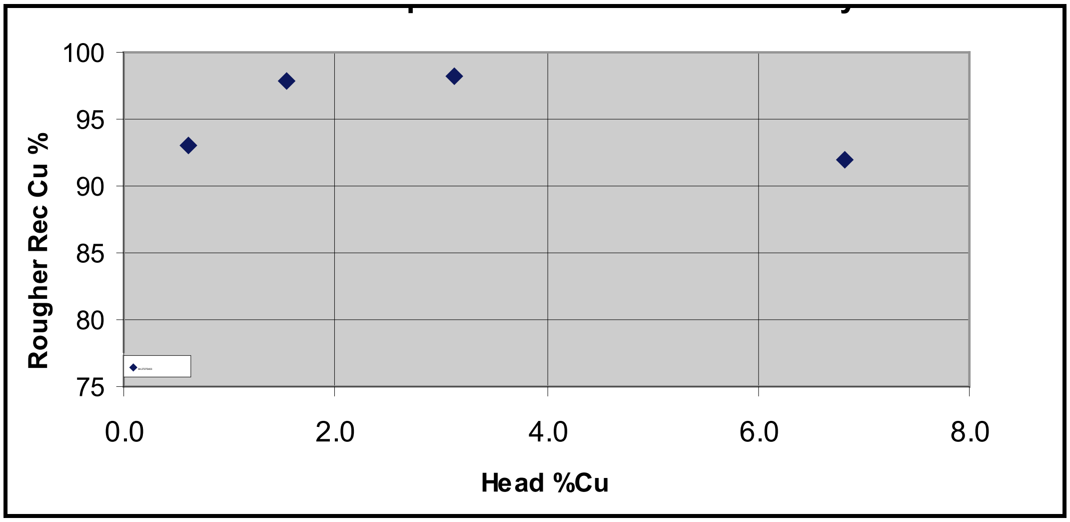
Intervals from nine diamond drillholes selected across the LZ mineralization were selected for testing. From these samples, five sub-composites were prepared with grades ranging from 0.6 to 11% copper with an overall weighted average composite grade of about 4% copper. Batch rougher and cleaner tests 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 with UZ grind levels because material from both the UZ and LZ with 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			Distribution		
		Ag (g/t)	Cu (%)	S (%)	Ag (%)	Cu (%)	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

Open circuit batch cleaner tests were conducted on four of the five LZ sub-composites from which the overall LZ composite was prepared. The tests were conducted at P_{80} primary grind levels of 53 and 38 µm, comparable results were obtained. The head grade rougher recovery relationship of the four samples is shown in the figure below which indicates that the highest grade sub-composite LZ-4 is somewhat less responsive than the other three (see Figure 13.5).

Figure 13.5 LZ Sub-composite Copper Heads versus Recovery



The results of the batch cleaner tests conducted at a primary grind of 38 µm without regrinding of the rougher concentrate are presented in Table 13.9.

Table 13.9 LZ Sub-composites – Batch Cleaner Test Results

Sub-composite	Head % Cu	Product	Weight (%)	Assay		Distribution	
				Cu (%)	Fe (%)	Cu (%)	Fe (%)
LZ-1	0.62	Second Cleaner Concentrate	3	16.6	25	89.5	4.6
LZ-2	1.53	Second Cleaner Concentrate	8	18.5	27	94.8	14.7
LZ-3	3.12	Second Cleaner Concentrate	12	24.2	27	95.0	18.1
LZ-4	6.96	Second Cleaner Concentrate	22	28.1	25	87.6	21.6

Higher copper recoveries were obtained but a third cleaner stage would be required at lower head grades to obtain the desired concentrate grades.

13.1.3 UZ-LZ Blended Composite

A composite comprising 3.25 parts UZ and 1.0 part LZ has been prepared and was tested with UZ conditions to determine the metallurgical response of a UZ-LZ blend. The ratio between the two mineralized types is based on the projected mill feed blend starting in year four of mine production.

A series of batch cleaner tests were done, all at a primary grind level of 38 µm, P_{80} to study the effect of regrind on the metallurgy. The 15 minute regrind gave the best results as shown in Table 13.10. A significant decrease in copper recovery was noted going from the second to the third cleaning stage; this response is typical for the UZ composite.

Table 13.10 UZ-LZ Blend Composite Batch Cleaner Tests – Effect of Regrind Time

Test No.	Regrind (min)	Copper Grade, % Cu			Copper Distribution, %	
		Calculated Head	Third Cleaner Concentrate	Second Cleaner Concentrate	Third Cleaner Concentrate	Second Cleaner Concentrate
F38	60	2.48	25.8	19.7	70.3	77.3
F41	30	2.52	27.8	24.0	73.3	78.2
F42	15	2.59	28.7	22.3	64.5	82.3

Finally, a locked cycle test was performed under test number F42. The steady state metallurgy based on cycles six to eight is shown in Table 13.11.

Table 13.11 Johnny Lee UZ-LZ Blend – Locked Cycle Test Results Based on Cycles 6 to 8

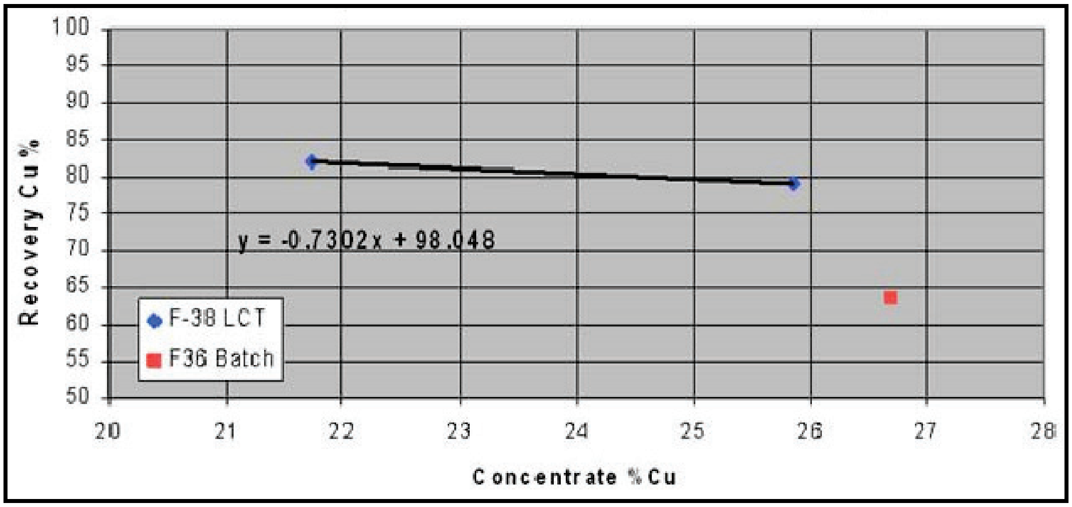
Products	Weight (%)	Assays (%)			
		Cu (%)	Fe (%)	Cu (%)	Fe (%)
Second Cleaner Concentrate	12.6	18.53	29.96	89.8	18.4
First Cleaner Scavenger Tails	12.9	0.36	24.38	1.8	15.4
Rougher Tailings	74.5	0.29	18.13	8.3	66.1
Calculated Head	100.0	2.59	20.42	100.0	100.0

The concentrate was below target; a third cleaning stage would have been required.

13.2 Basis for Predicting the Metallurgy of UZ and LZ Mineralization

In order to develop a concentrate grade recovery correlation for the LZ mineralization from the locked cycle test the second cleaner concentrate grade and recovery were calculated by making the assumption that the second cleaner concentrate would comprise the third cleaner concentrate plus 50% of the copper contained in the third cleaner tailing. The concentrate grade recovery equation developed from the two data sets is shown in Figure 13.6.

Figure 13.6 Locked Cycle Test Concentrate Grade versus Recovery



The projected metallurgy derived from the correlation equation based on a target 81% copper recovery is shown in Table 13.12.

Table 13.12 Projected Metallurgy of UZ Based on Locked Cycle Test Results

	Weight (%)	Cu (%)	Recovery (%)
Cleaner Concentrate	7.8	23.30	81.0
Head Grade	100.0	2.24	100.0

The projected metallurgy for the LZ mineralization will be based on the results of the locked cycle on the LZ composite. The silver and cobalt content of the zone is very low and payable levels of these elements in the concentrate were not achieved in the locked cycle test.

In the absence of metallurgy information on the effect of head grade, the metallurgy projections for the mine plan will be based on the recoveries as obtained in the locked cycle tests regardless of head grade. The LZ composite at 2.24% copper is lower in head grade than that estimated in the mine plan, therefore this will make the recovery projections more conservative.

13.3 Origin and Representativeness of Metallurgical Samples

13.3.1 UZ 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 13.13.

Table 13.13 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 13.14.

Table 13.14 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 2.5% copper.

13.4 LZ 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 13.15 and Table 13.16.

Table 13.15 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
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 13.16 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

The silver and cobalt content of the LZ mineralization is considerably lower than that of the UZ.

13.5 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. 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 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 after the fourth 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).

This report focuses on the estimate of resources for the Lowry zone based on new drillhole data that have been collected by Tintina. 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 Lowery Zone.

14.1 DRILLHOLE DATA

RMI was provided with various electronic drillhole data for the UZ and LZ 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. Basic assay statistics for copper, cobalt, and silver are tabulated in Table 14.1 through Table 14.3, respectively for the two Johnny Lee UZ massive sulphide units. The term "Inc %" in columns 4 and 7 of Table 14.1 to Table 14.4 refers to incremental percentage of material between cut-off grades. For example, the first incremental percentage value of 32 in column 4 of Table 14.1 means that 32% 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 14.3 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	738	32	2.10	1547	10	2.02	0.96
	1.00	500	35	2.79	1394	24	2.13	0.76
	2.00	242	11	4.26	1030	13	2.26	0.53
	3.00	160	22	5.20	831	54	2.24	0.43
31	0.00	596	32	2.09	1246	10	2.02	0.97
	1.00	403	35	2.78	1120	24	2.14	0.77
	2.00	192	11	4.28	821	13	2.28	0.53
	3.00	127	21	5.23	665	53	2.27	0.43
32	0.00	142	32	2.13	301	9	2.02	0.95
	1.00	96	33	2.85	274	22	2.08	0.73
	2.00	50	12	4.18	208	14	2.15	0.51
	3.00	33	23	5.09	166	55	2.15	0.42

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

As shown in Table 14.1, approximately 68% of the drillhole assays in the main sulphide bed (UZ 31) are in excess of 1% copper and about 22% 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	738	1	1,068	787,844	0	1,151	1.08
	100	733	26	1,074	787,551	8	1,151	1.07
	500	538	37	1,346	724,315	25	1,235	0.92
	1,000	267	36	1,970	525,401	67	1,510	0.77
31	0	596	1	1,096	653,616	0	1,220	1.11
	100	593	24	1,102	653,426	7	1,221	1.11
	500	449	39	1,351	605,752	26	1,309	0.97
	1,000	214	36	2,028	433,681	66	1,641	0.81
32	0	142	1	948	134,228	0	782	0.82
	100	140	36	956	134,126	12	781	0.82
	500	89	26	1,325	118,564	20	757	0.57
	1,000	53	37	1,736	91,720	68	740	0.43

Roughly one third of the Johnny Lee UZ cobalt assays are above 1,000 ppm (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	738	7	16	11,936	1.1	13	0.81
	5	688	53	17	11,807	32.5	13	0.76
	15	293	30	27	7,932	37.2	15	0.55
	30	75	10	46	3,490	29.2	17	0.37
31	0	596	7	16	9,710	1.2	13	0.79
	5	552	52	17	9,596	31.8	13	0.74
	15	241	31	27	6,504	38.6	14	0.53
	30	59	10	47	2,760	28.4	16	0.35
32	0	142	4	16	2,225	0.6	14	0.88
	5	136	59	16	2,211	35.2	14	0.85
	15	52	25	28	1,428	31.4	17	0.62
	30	16	11	46	730	32.8	21	0.45

Only about 10% of the Johnny Lee UZ silver assays are above 30 g/t (40% are above 15 g/t). Silver shows the lowest CV of the three metals.

14.4 JOHNNY LEE UZ HIGH-GRADE OUTLIERS

RMI generated a series of cumulative probability plots after transforming the original copper, cobalt, silver, gold, and zinc 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 14.4 summarizes the high-grade outlier capping limits that were established for copper, cobalt, and silver. These limits were applied to the raw assays prior to creating drillhole composites.

Table 14.4 Johnny Lee UZ Grade Capping Limits

	Copper (%)			Cobalt (%)			Silver (g/t)		
UZ Unit	Cap Limit	No. Capped	Metal Loss (%)	Cap Limit	No. Capped	Metal Loss (%)	Cap Limit	No. Capped	Metal Loss (%)
31	14.0	4	0.2	1.0	4	1.2	80	5	0.3
32	7.5	8	2.4	0.5	0	0.0	50	3	3.9

Figure 14.1 Johnny Lee UZ Copper Cumulative Probability Plot

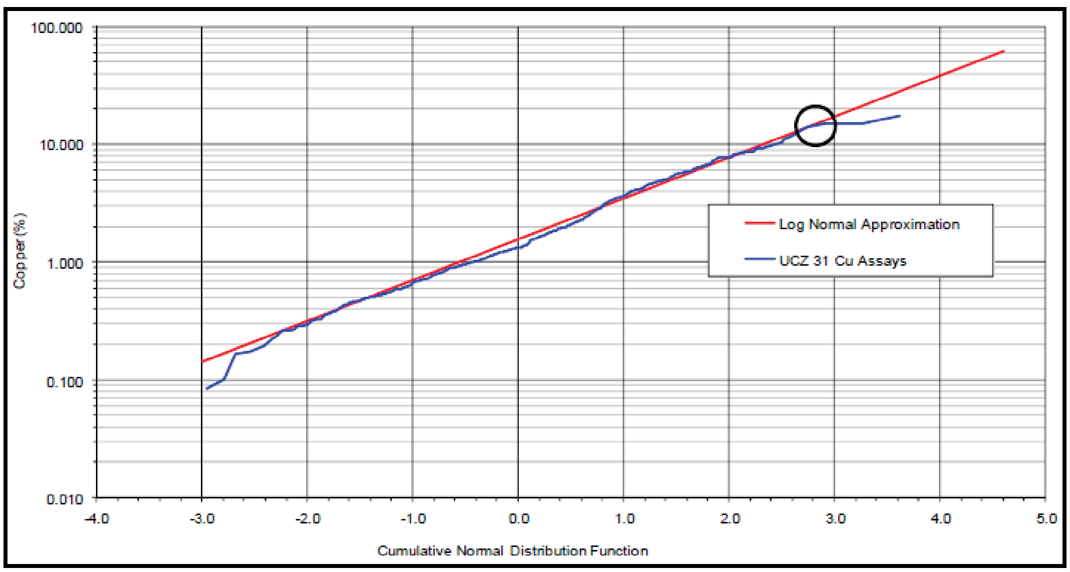
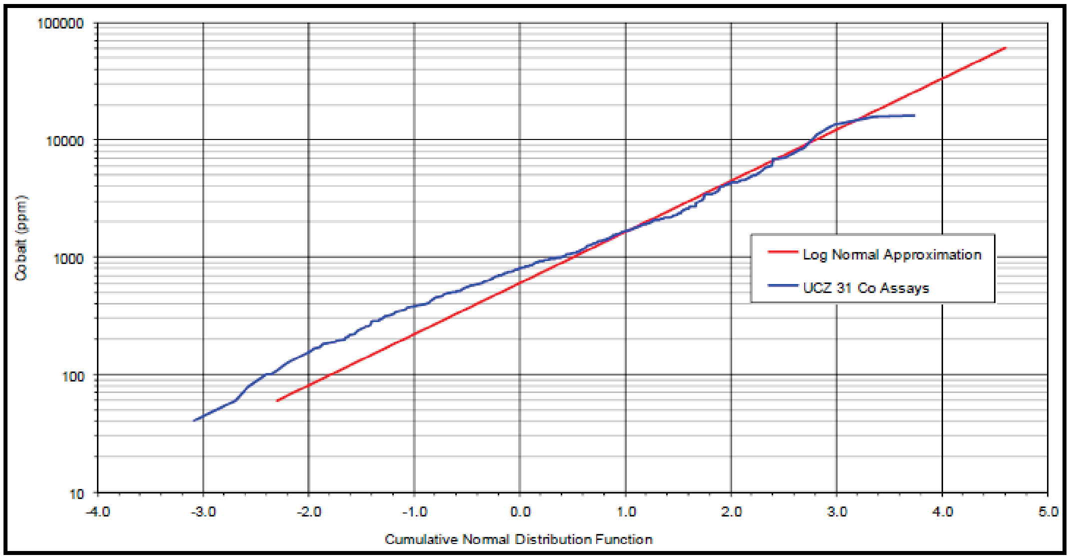


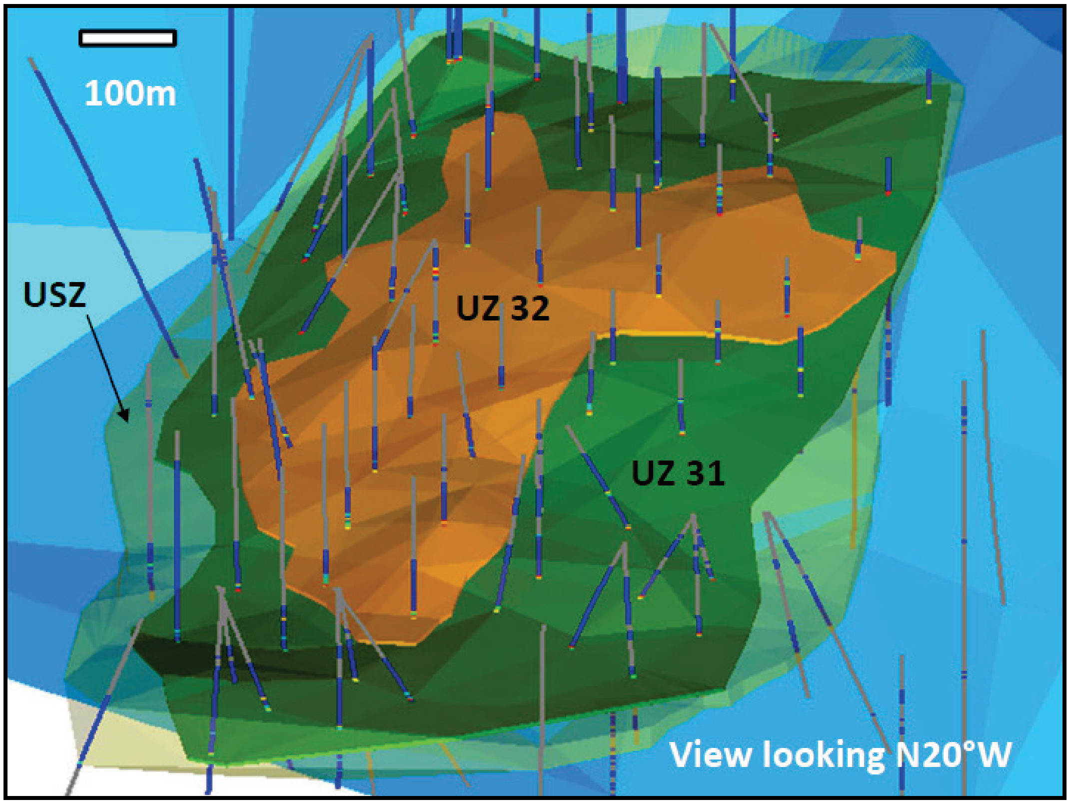
Figure 14.2 Johnny Lee UZ Cobalt Cumulative Probability Plot



14.5 JOHNNY LEE UZ DOMAINS

Mr. Vincent Scartozzi, a Senior Geologist with Tintina, constructed a 3D wireframe to represent two Johnny Lee UZ stratabound copper sulphide horizons. The two 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. The wireframe was extended approximately 30 to 40 m outward from the perimeter drillholes that intersected the horizon. Figure 14.3 is a perspective view looking N20W downward at the main and secondary UZ wireframes. The VVF shown in blue was modelled as a thick zone of shearing and was used to clip the up dip portion of unit 31. The USZ was modelled as a separate wireframe that locally is coincident with the UZ units but in general is more extensive than the actual copper zones and is shown as a light green semi-transparent wireframe. 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

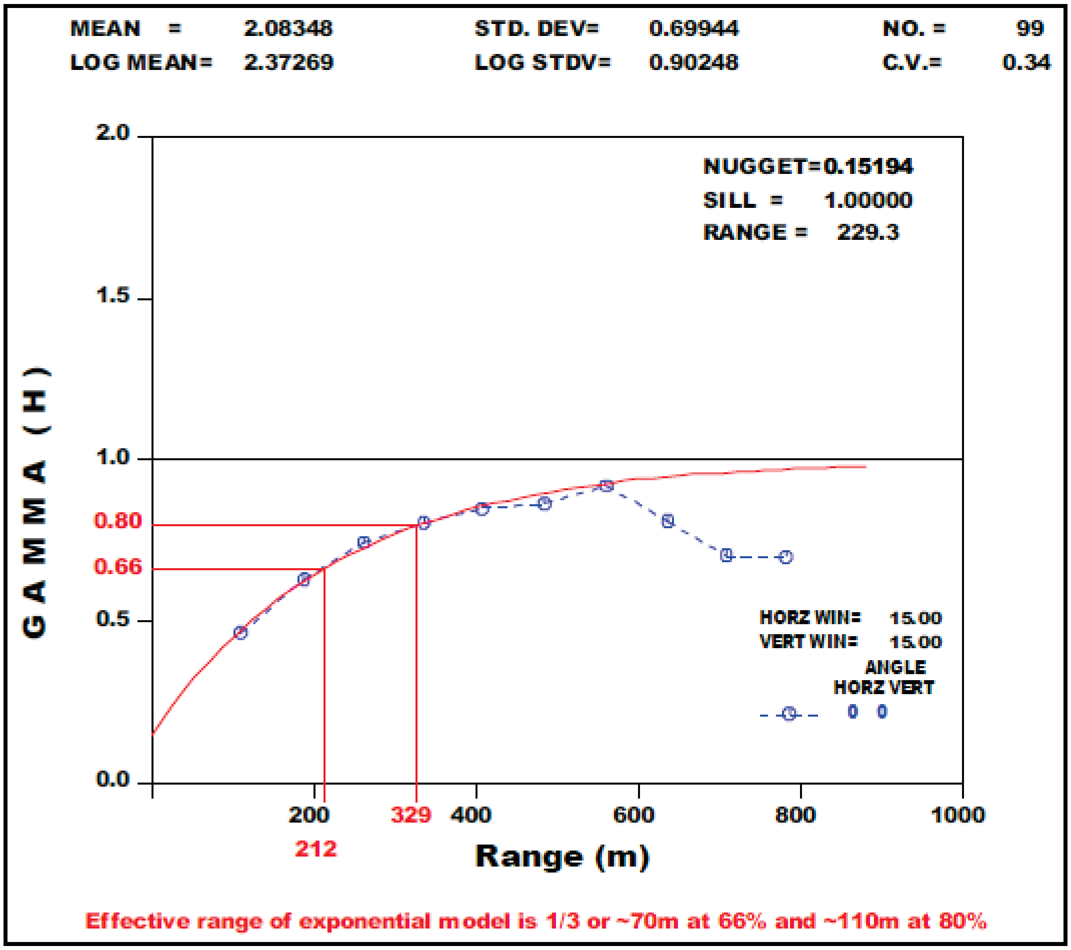
One-metre-long drillhole composites were created starting and ending inside of the Johnny Lee UZ wireframes (zones 31 and 32). There were a total of 630 UZ unit 31 composites with 95% of them exactly 1 m in length; approximately 4% were between 0.50 to 1.0 m in length, and about 1% less than 0.5 m in length. There were a total of 153 UZ unit 32 composites with 95% of them exactly 1 m long, 3% between 0.50 and 1.0 m in length, and 2% less than 0.5 m in length. The grade estimate was weighted by composite length.

14.7 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 relative variogram that was generated from 99 hanging wall to footwall composites that were generated from UZ units 31 and 32. An exponential model was used to fit the data points. Vectors were drawn at two-thirds and 80% of the total variance (red bisectors) to show the spread of range.

Note that the effective or practical range for exponential models is one-third of the range shown along the X-axis.

Figure 14.4 Johnny Lee UZ Copper Correlogram



A series of Johnny Lee UZ directional copper grade correlograms are presented in Figure 14.5 through Figure 14.8. The correlograms were constructed using 1 m composites, using Sage 2001, a commercially available software package.

Figure 14.5 Johnny Lee UZ Copper Correlograms #1

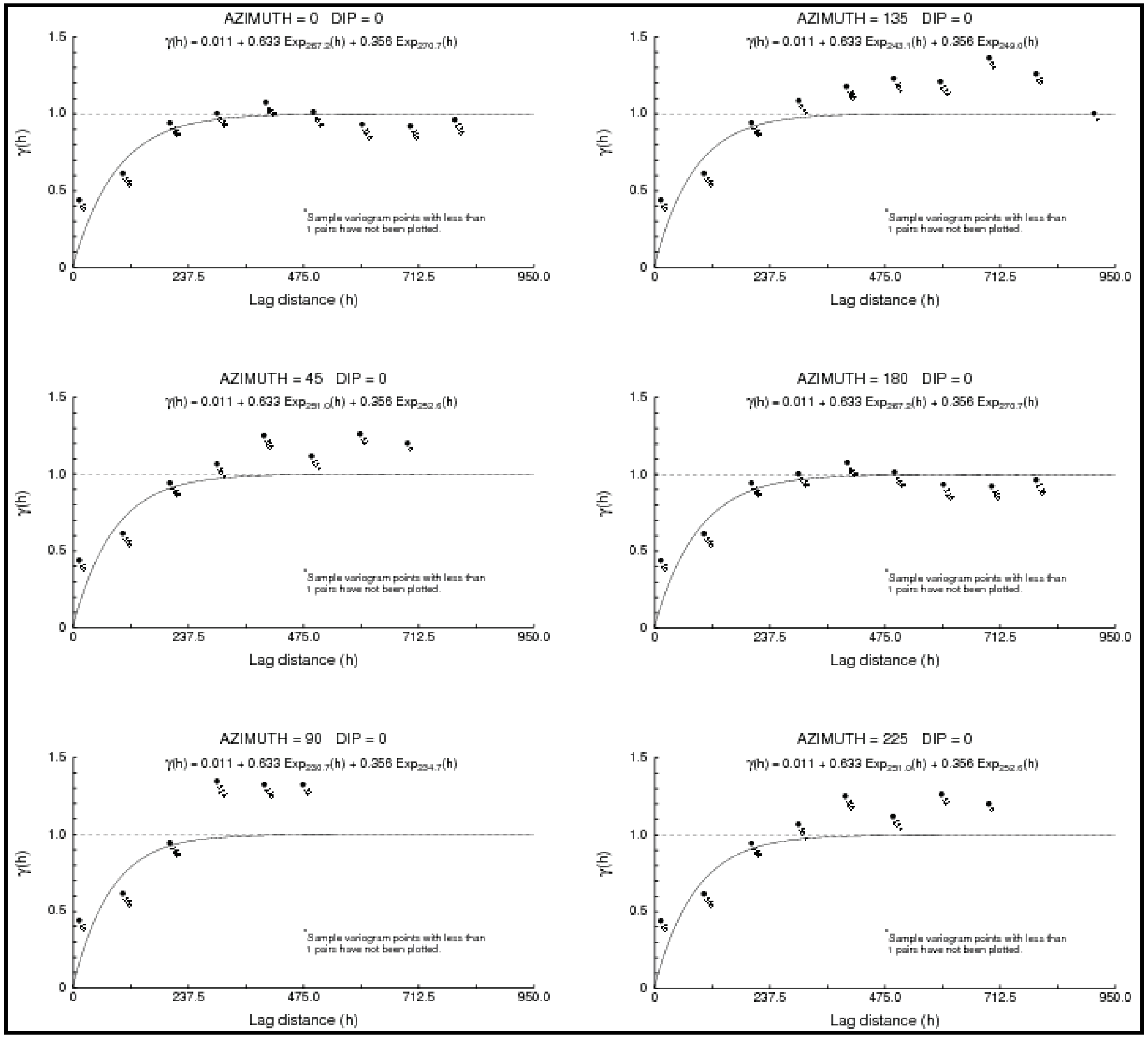


Figure 14.6 Johnny Lee UZ Copper Correlograms #2

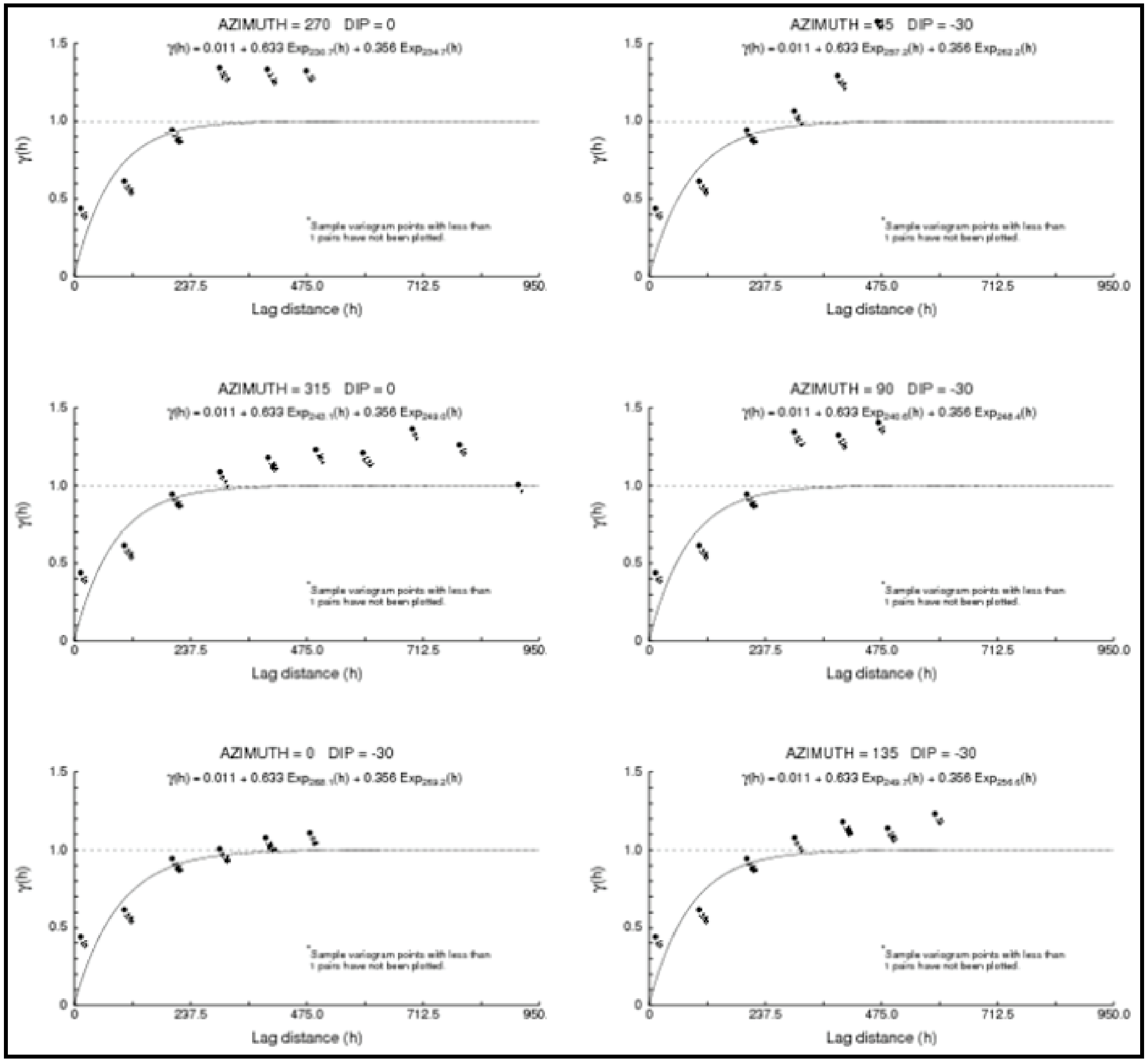


Figure 14.7 Johnny Lee UZ Copper Correlograms #3

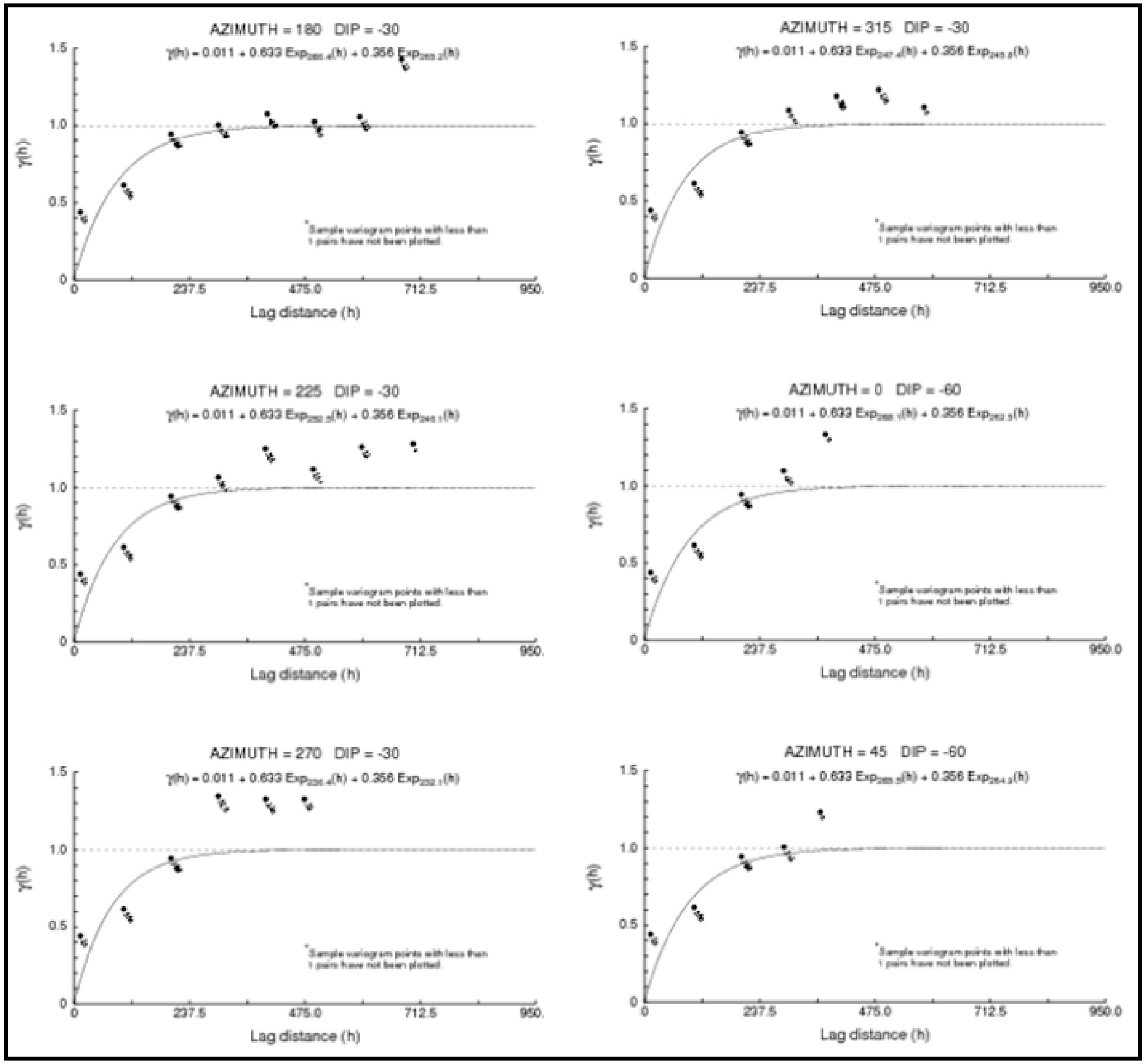
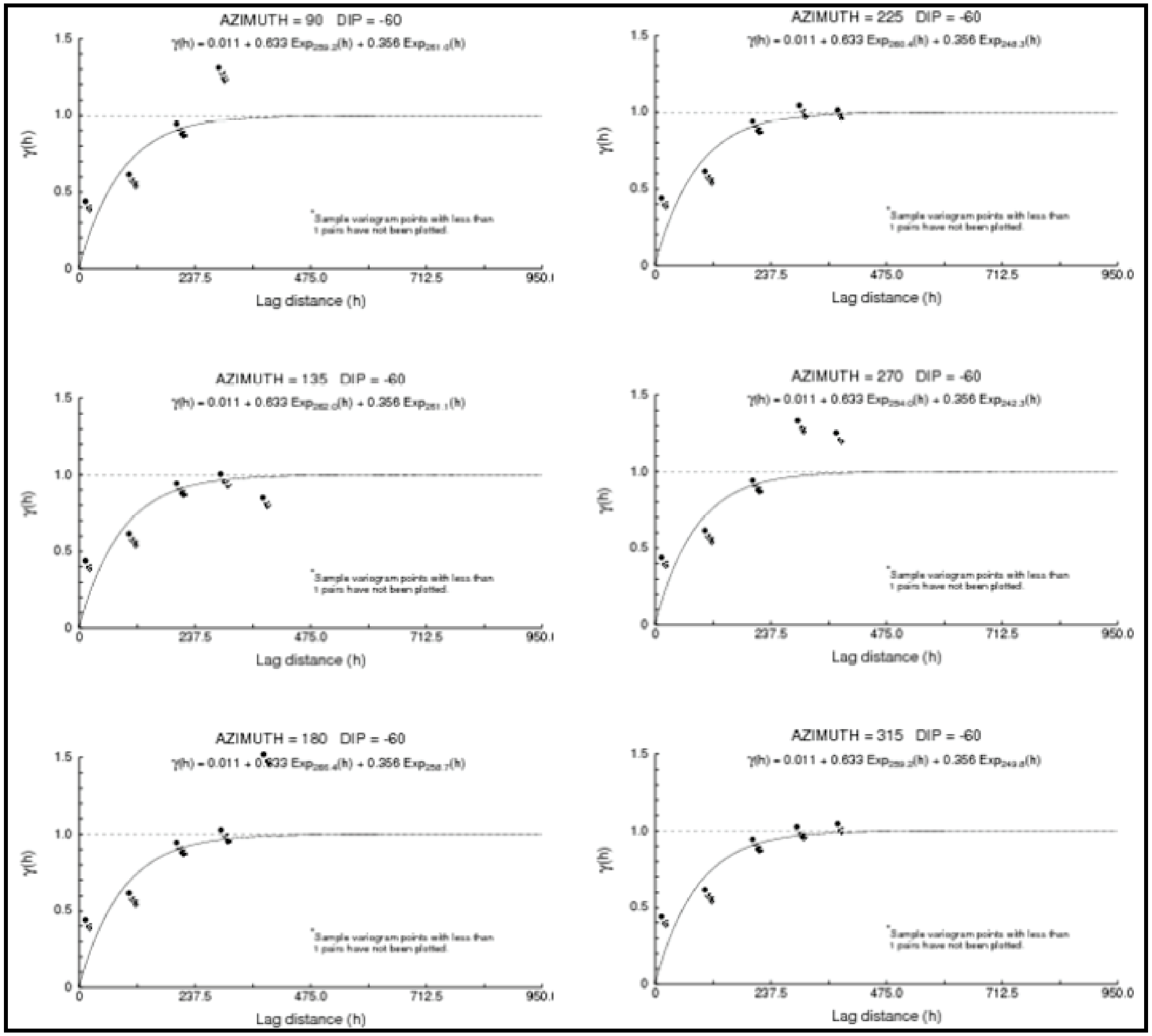


Figure 14.8 Johnny Lee UZ Copper Correlograms #4



14.8 Johnny Lee UZ Grade Estimation

RMI constructed a 3D block model using MineSight® software. Table 14.5 summarizes the limits of the model and size of the blocks.

Table 14.5 Block Model Limits

Parameter	Minimum	Maximum	Extent (m)	Size (m)	Number
Easting (columns)	506,200	506,900	700	5	140
Northing (rows)	5,180,100	5,181,225	1,125	5	225
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.

A two pass inverse distance (ID) estimation plan was used for estimating copper, cobalt, silver, gold, lead, zinc, iron, sulfur, 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 a high ID weighting power (5).

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 is thought to be a good representation of the actual 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 14.6. 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 14.6 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	5	1	3	1	200	200	200	10
2	5	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.

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 14.9 is a cross section through the block model showing composite and block copper grades for the 2010 UZ resource model. For reference, Figure 7.4 in Section 7.0 is a plan map showing the line of section for Figure 14.9 (i.e. section C-C'). Figure 7.6 in Section 7.0 is the corresponding geologic cross for section C-C', which does not have the 2011 infill drillholes shown in Figure 14.9. Figure 14.10 is a more detailed view of Section 14-9 showing how the stratabound-like distribution of block grades.

Figure 14.9 Johnny Lee UZ Block Model Cross Section C-C’

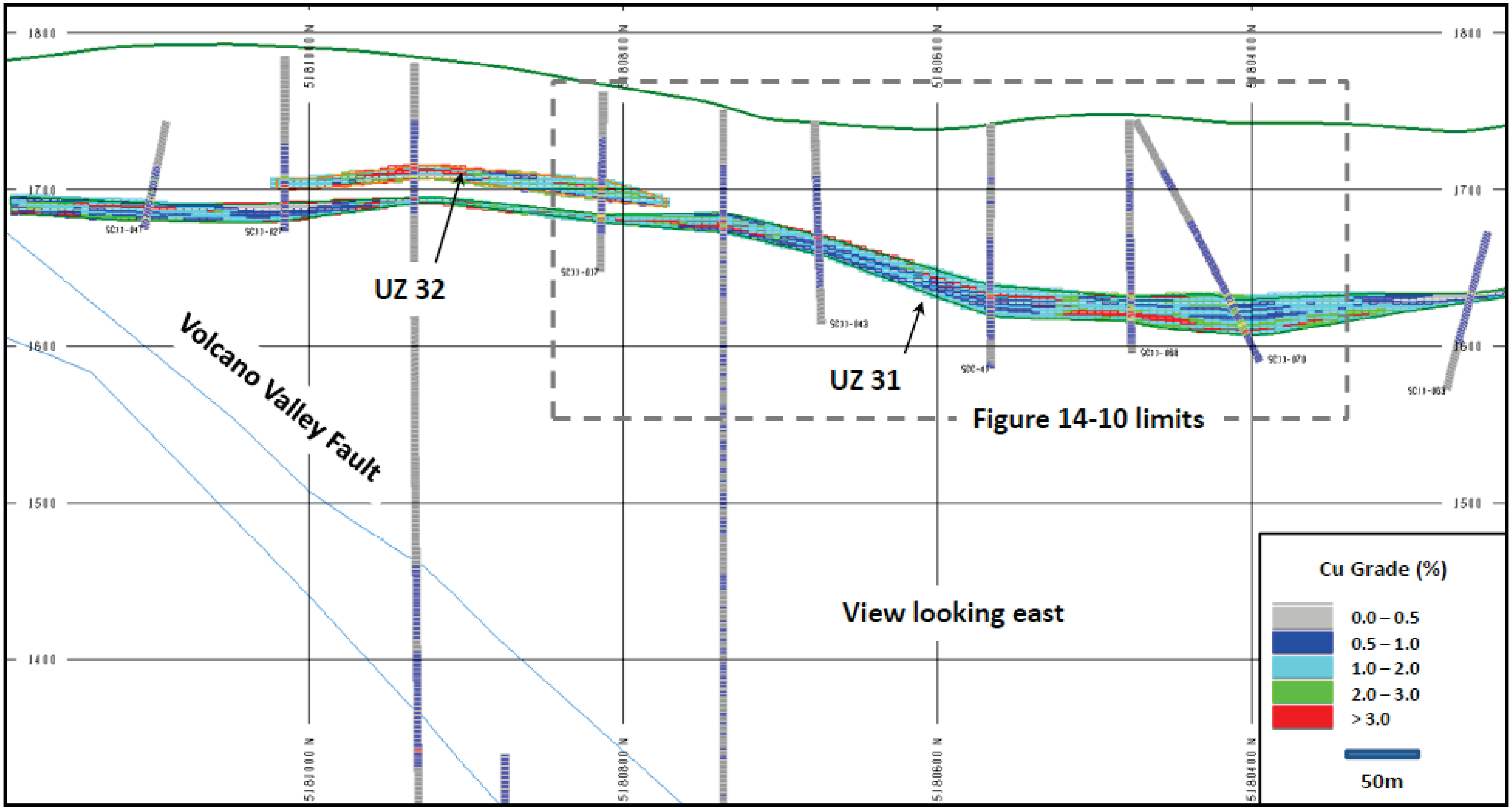
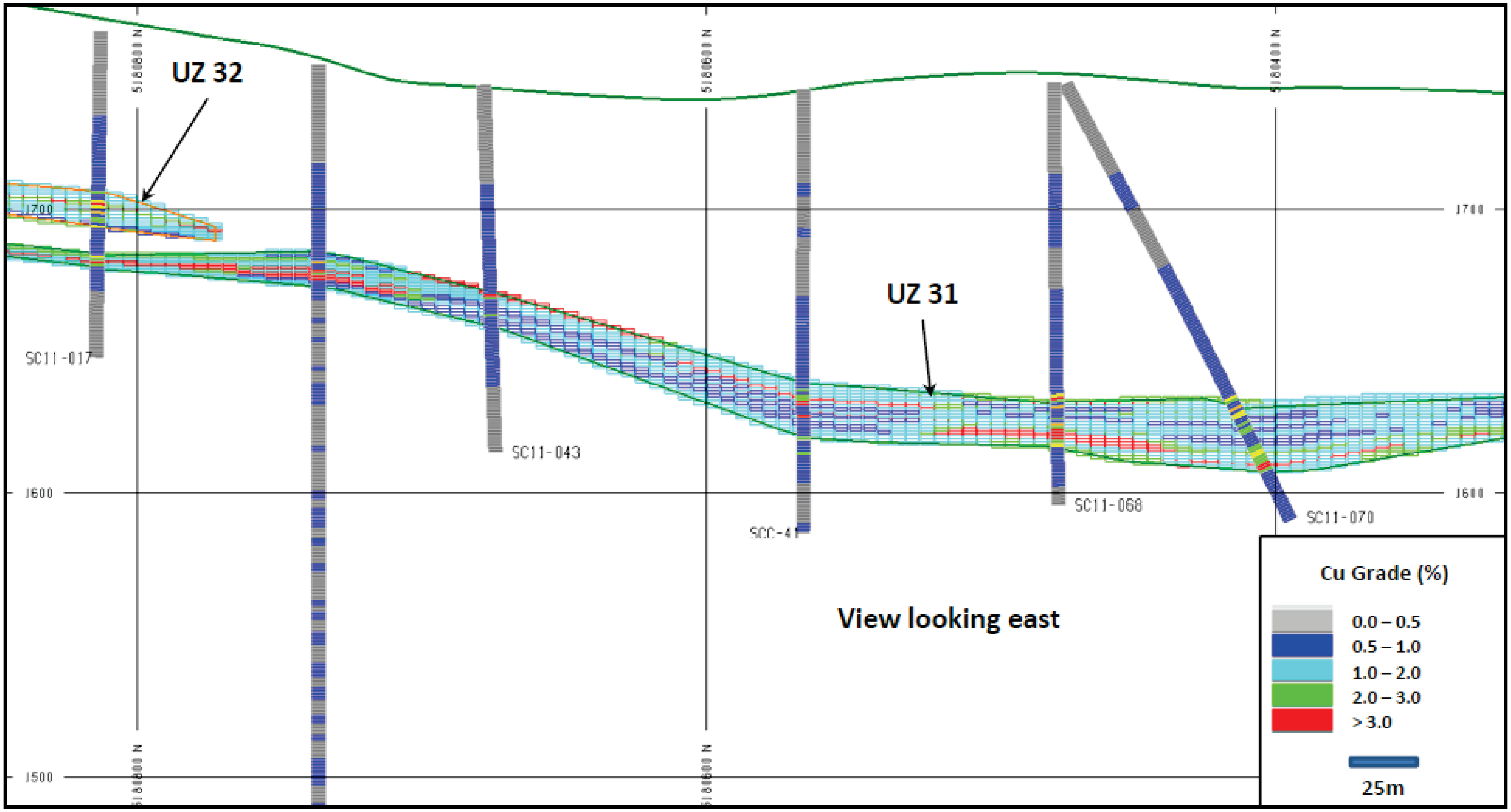


Figure 14.10 Johnny Lee UZ Block Model Cross Section C-C' Detail



NN models were constructed for the primary metals of interest (i.e. copper, cobalt, and silver). To check for possible global biases in the block model, the ID and NN grades were compared for all estimated blocks at a zero cut-off grade. Table 14.7 compares copper and cobalt grades estimated by ID and NN methods.

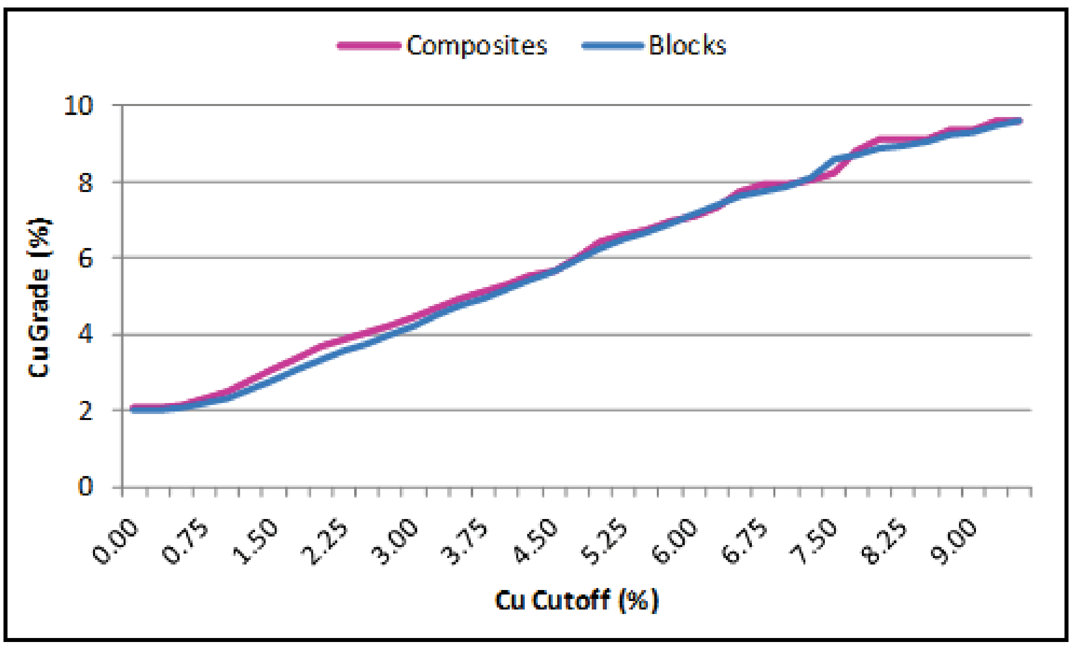
Table 14.7 Johnny Lee UZ Global Bias Check

Metal	ID Grade	NN Grade	Difference (%)
Copper (%)	2.0418	2.0373	0.22
Cobalt (ppm)	983.8	978.9	0.50
Gold (g/t)	0.0105	0.0105	0.00
Silver (g/t)	15.98	15.94	0.25

The data in Table 14.7 show a close comparison between the ID 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.

Johnny Lee UZ copper drillhole composite grades were compared with the estimated copper block grades at a variety of cut-off grades. Figure 14.11 shows that the model block grades closely follow the underlying drillhole composite grades.

Figure 14.11 Johnny Lee UZ Copper Grade Comparison



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 14.12 through Figure 14.13 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 14.12 Johnny Lee UZ Copper Swath Plot – Easting

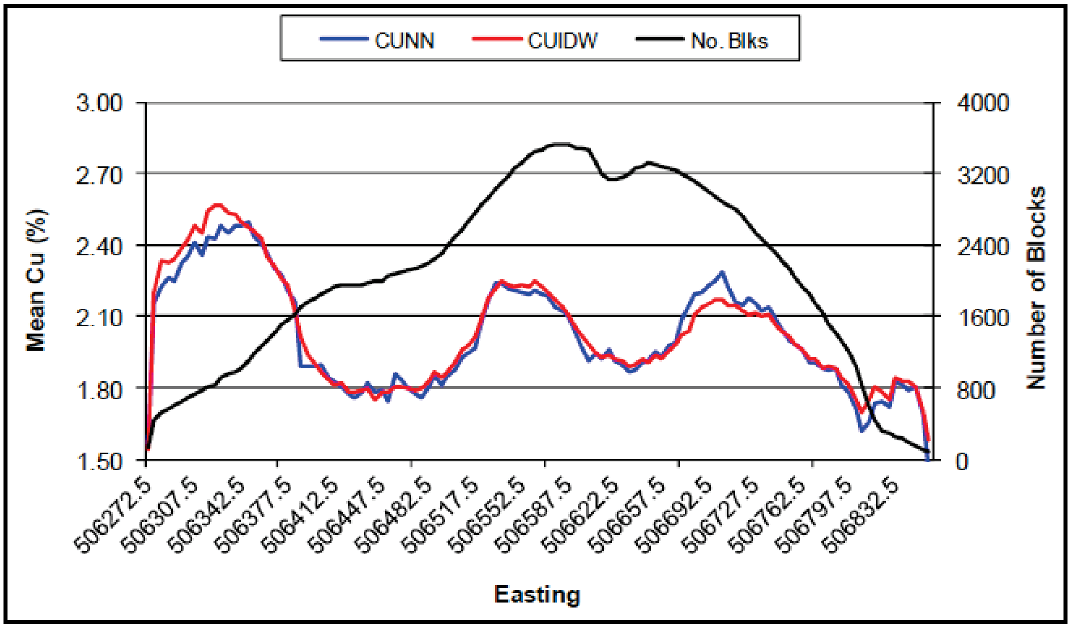


Figure 14.13 Johnny Lee UZ Copper Swath Plot – Northing

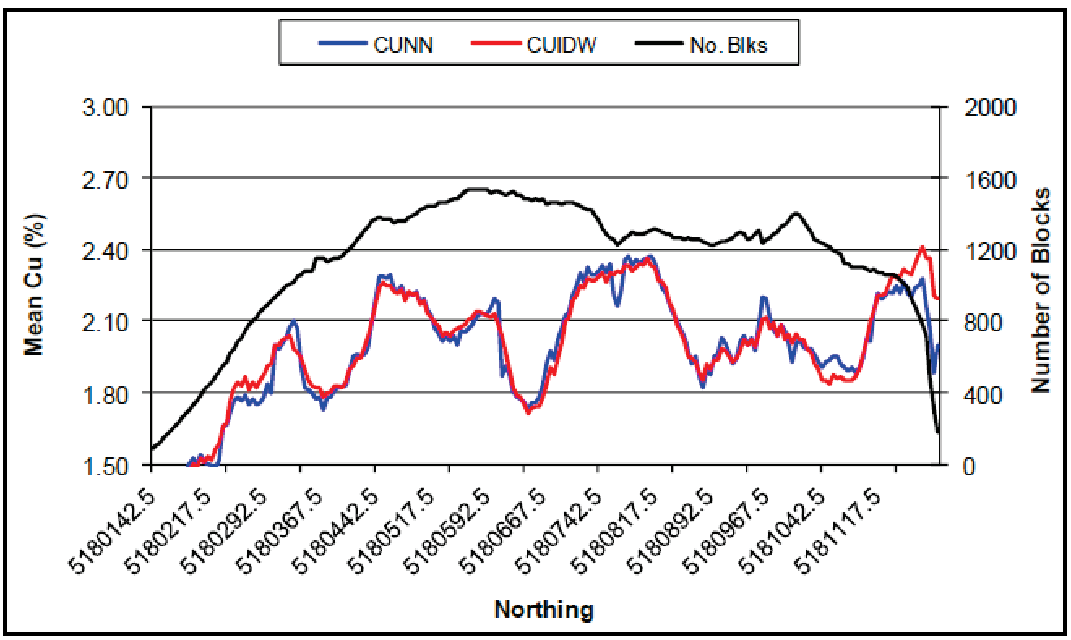
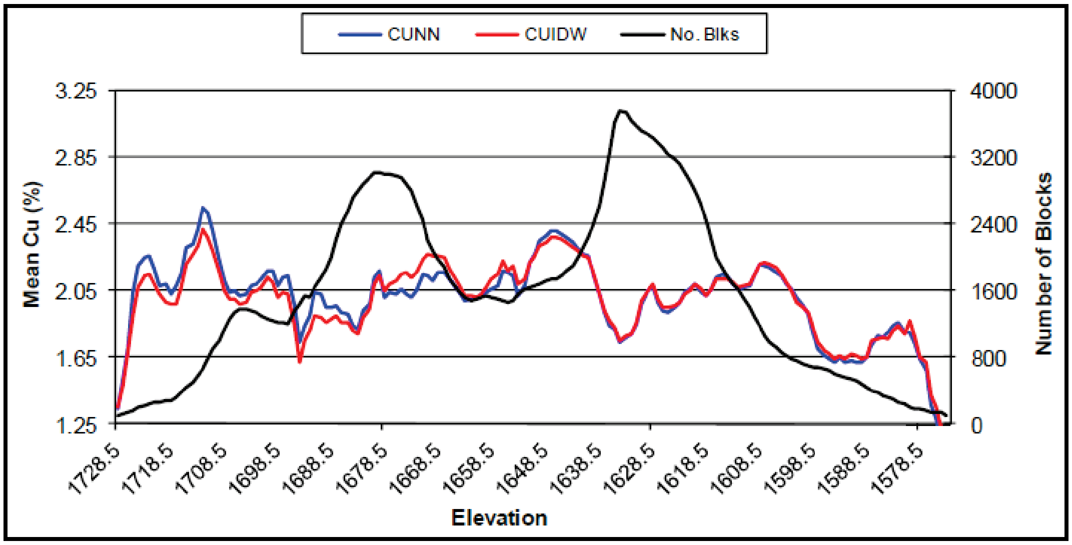


Figure 14.14 Johnny Lee UZ Copper Swath Plot – Elevation



The swath plots shown in Figure 14.12 through Figure 14.14 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 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 in-situ resources.

14.10 Johnny Lee UZ Resource Classification

Estimated blocks inside of the UZ wireframes (units 31 and 32) were classified as Indicated Resources if, 1) the block was estimated by two or more holes with one hole within 50 m or 2) the block was estimated by one hole within 25 m. 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 349 pieces of drill core taken from their 2010 and 2011 drilling programs in the Johnny Lee zone. The average bulk density from these 349 samples was 3.54 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 60 UZ massive sulphide determinations, a bulk density value of 3.93 g/cm^3 was selected by RMI for Johnny Lee UZ horizons 31 and 32. Based on 148 determinations an average bulk density of 3.61 g/cm^3 was selected for non-copper sulphide zones within the bedded sulphide package. A bulk density value of 2.80 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 an undiluted Indicated Mineral Resource of 8,483,000 t with an average grade of 2.96% copper, 0.12% cobalt, and 16.9 g/t silver. In addition to Indicated Resources there is an Inferred Resource of 1,257,000 t with an average grade of 2.64% copper, 0.10% cobalt, and 16.4 g/t silver using a 1.6% copper cut-off grade. 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. Table 14.8 summarizes Johnny Lee UZ Indicated Resources at a number of cut-off grades. The disclosed Johnny Lee UZ Indicated Resource is highlighted in grey. No credit was given to cobalt or silver in determining the cut-off grade.

Table 14.8 Johnny Lee UZ Undiluted Indicated Mineral Resource

Cu Cutoff (%)	Tonnes ('000)	Estimated Metal Grades			Contained Metal		
		Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
0.50	15,757	2.10	0.10	16.2	729	35	8,207
0.60	15,378	2.14	0.10	16.2	725	34	8,010
0.70	14,876	2.19	0.10	16.3	718	33	7,796
0.80	14,394	2.23	0.11	16.5	707	35	7,636
0.90	13,779	2.30	0.11	16.6	698	33	7,354
1.00	13,002	2.38	0.11	16.6	682	32	6,939
1.10	12,101	2.47	0.11	16.6	659	29	6,458

table continues…

Cu Cutoff (%)	Tonnes ('000)	Estimated Metal Grades			Contained Metal		
		Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
1.20	11,284	2.57	0.11	16.6	639	27	6,022
1.30	10,385	2.69	0.12	16.8	616	27	5,609
1.40	9,696	2.78	0.12	16.7	594	26	5,206
1.50	9,110	2.87	0.12	16.8	576	24	4,921
1.60	**8,483**	**2.96**	**0.12**	**16.9**	**553**	**22**	**4,609**
1.70	7,770	3.08	0.13	17.0	527	22	4,247
1.80	7,205	3.19	0.13	17.1	507	21	3,961
1.90	6,653	3.30	0.13	16.9	484	19	3,615
2.00	6,137	3.41	0.14	16.9	461	19	3,335
2.10	5,758	3.50	0.14	16.8	444	18	3,110
2.20	5,361	3.60	0.14	16.9	425	17	2,913
2.30	5,005	3.70	0.14	16.9	408	15	2,719
2.40	4,782	3.76	0.14	16.9	396	15	2,598
2.50	4,558	3.83	0.14	16.9	385	14	2,477
2.60	4,316	3.90	0.15	16.8	371	14	2,331
2.70	4,016	3.99	0.15	16.7	353	13	2,156
2.80	3,735	4.09	0.15	16.4	337	12	1,969
2.90	3,460	4.18	0.15	16.4	319	11	1,824
3.00	3,198	4.28	0.15	16.4	302	11	1,686

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.

Figure 14.15 shows grade-tonnage curves from the data tabulated in Table 14.8.

Figure 14.15 Johnny Lee UZ Indicated Grade-tonnage Curves

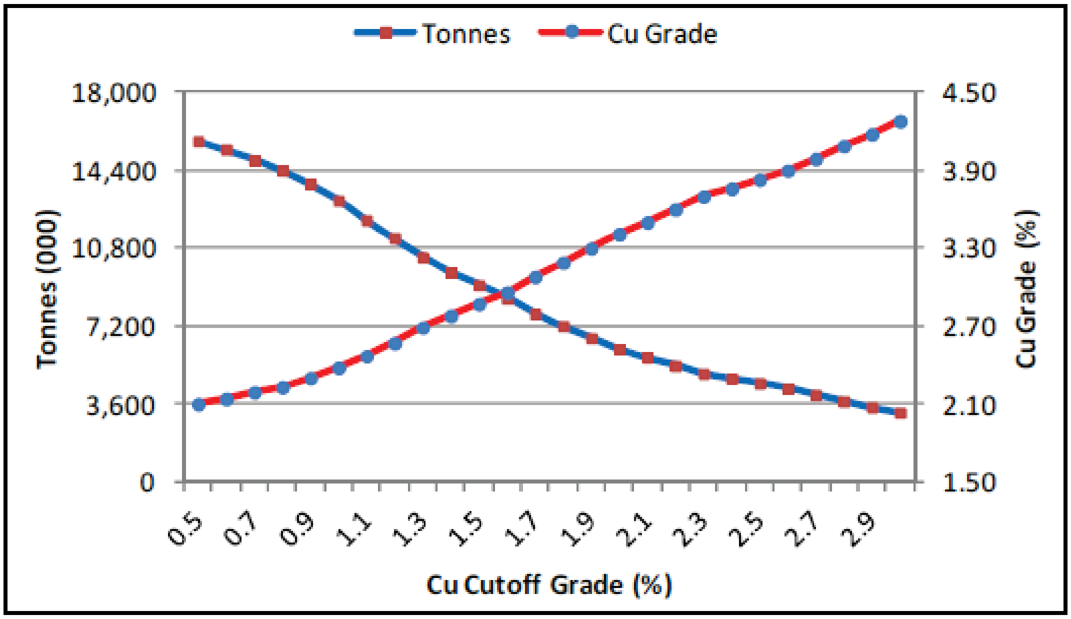


Table 14.9 summarizes Johnny Lee UZ Inferred Resources at a number of cut-off grades. The disclosed Johnny Lee UZ Inferred Resources is highlighted in grey. No credit was given to cobalt or silver in determining the cut-off grade.

Table 14.9 Johnny Lee UZ Undiluted Inferred Mineral Resource

Cu Cutoff (%)	Tonnes ('000)	Estimated Metal Grades			Contained Metal		
		Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
0.50	2,472	1.90	0.09	16.4	104	5	1,303
0.60	2,433	1.92	0.09	16.5	103	5	1,291
0.70	2,368	1.95	0.09	16.5	102	5	1,256
0.80	2,301	1.99	0.09	16.5	101	5	1,221
0.90	2,213	2.04	0.09	16.6	100	4	1,181
1.00	2,079	2.11	0.09	16.4	97	4	1,096
1.10	1,944	2.18	0.09	16.3	93	4	1,019
1.20	1,806	2.26	0.10	16.3	90	4	946
1.30	1,662	2.34	0.10	16.4	86	4	876
1.40	1,514	2.44	0.10	16.4	81	3	798
1.50	1,379	2.54	0.10	16.4	77	3	727
1.60	**1,257**	**2.64**	**0.10**	**16.4**	**73**	**3**	**663**
1.70	1,159	2.72	0.11	16.4	69	3	611
1.80	1,060	2.81	0.11	16.3	66	3	556
1.90	969	2.90	0.11	16.3	62	2	508
2.00	886	2.99	0.11	16.2	58	2	461

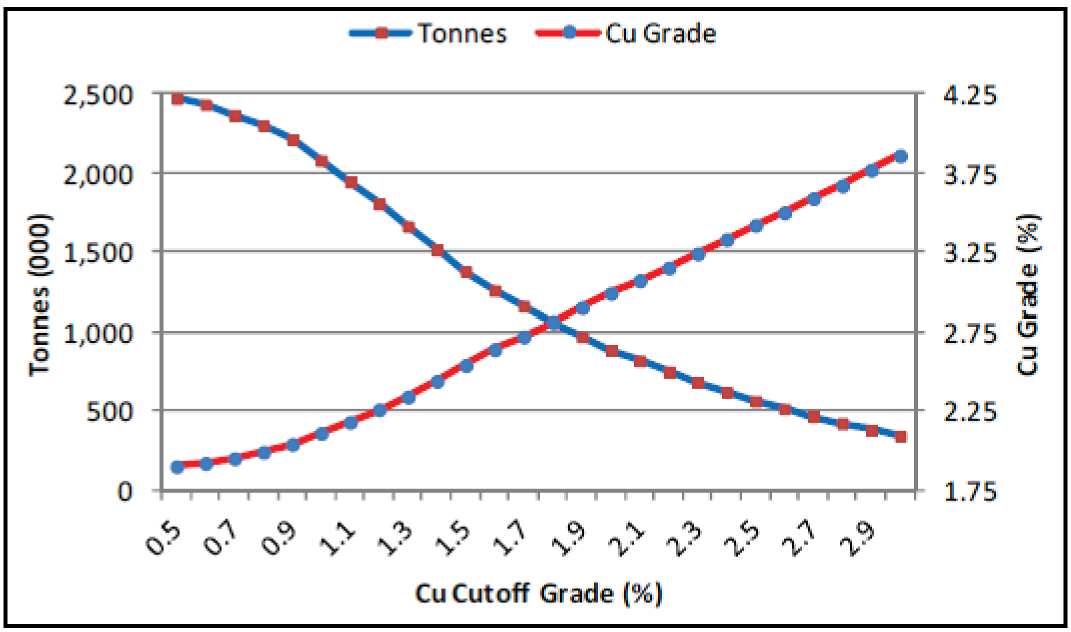
table continues…

Cu Cutoff (%)	Tonnes ('000)	Estimated Metal Grades			Contained Metal		
		Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
2.10	818	3.07	0.11	16.2	55	2	426
2.20	750	3.15	0.11	16.2	52	2	391
2.30	684	3.24	0.12	16.1	49	2	354
2.40	620	3.33	0.12	16.1	46	2	321
2.50	564	3.42	0.12	16.0	43	1	290
2.60	514	3.50	0.12	16.0	40	1	264
2.70	466	3.59	0.13	16.0	37	1	240
2.80	423	3.67	0.13	16.0	34	1	218
2.90	380	3.77	0.13	16.0	32	1	195
3.00	341	3.86	0.13	16.1	29	1	177

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.

Figure 14.16 shows grade-tonnage curves from the data tabulated in Table 14.9.

Figure 14.16 Johnny Lee UZ Inferred Grade-tonnage Curves



14.14 General Discussion – Johnny Lee UZ 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 UZ Inferred Mineral Resources discussed in this report.

A significant amount of additional infill drilling, geotechnical studies, metallurgical test work, and environmental permitting will be required to determine the economics of the Project and whether any portion of the resources will be affected by mining, processing, or permitting.

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 14.10 summarizes the drillhole data that were used by RMI to estimate mineral resources for the Johnny Lee LZ. The information in Table 14.10 includes the company that drilled the hole, beginning and ending depth of the horizon, the LZ intersection length and average copper, cobalt, silver, and gold grades for the LZ intersections. The grades shown in Table 14.10 are based on capped raw assays (see Section 14.16 for capping limits).

Table 14.10 Johnny Lee LZ Drillhole Data

Drillhole	Company	From Depth (m)	To Depth (m)	Length (m)	LZ Intersections Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)
SC10-003	Tintina	350.40	351.69	1.29	4.08	601	4.3	0.57
SC10-004	Tintina	414.00	418.05	4.05	7.88	765	8.3	0.21
SC10-005	Tintina	401.00	412.15	11.15	4.06	595	3.9	0.20
SC11-007	Tintina	409.66	411.24	1.58	1.38	91	3.2	0.07
SC11-008	Tintina	353.38	357.40	4.02	1.56	239	3.8	0.31
SC11-009	Tintina	415.42	416.67	1.25	1.35	10	1.0	0.06
SC11-010	Tintina	450.10	458.75	8.65	0.48	202	7.2	0.03
SC11-011	Tintina	409.65	422.70	13.05	3.12	179	2.5	0.30
SC11-012	Tintina	384.65	387.55	2.90	1.89	1,240	6.8	1.00
SC11-015	Tintina	449.29	456.59	7.30	2.68	424	6.1	0.46
SC11-023	Tintina	421.35	424.59	3.24	0.19	2,578	5.1	0.68
SC11-029	Tintina	437.00	441.50	4.50	7.30	1,299	8.0	0.30
SC11-031	Tintina	426.08	428.24	2.16	0.64	283	2.0	0.01
SC11-032	Tintina	374.51	375.72	1.21	0.47	130	4.0	0.01
SC11-048	Tintina	359.92	367.60	7.68	5.55	865	6.4	0.59

table continues…

Drillhole	Company	From Depth (m)	To Depth (m)	Length (m)	LZ Intersections			
					Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)
SC-50	CAI	367.89	370.33	2.44	6.07	104	3.2	0.39
SC-51	CAI	397.61	404.77	7.16	4.77	102	1.7	0.19
SC-55	CAI	463.60	470.28	6.68	5.97	150	12.5	0.42
SC-57	CAI	482.50	484.94	2.44	6.86	217	8.7	0.38
SC-90	CAI	383.26	384.54	1.28	9.38	243	10.9	0.09
SCC-17	UII	355.70	358.14	2.44	6.44	488	3.0	0.28
SCC-20	UII	343.05	344.97	1.92	1.21	192	1.8	0.13
SCC-21	UII	394.56	400.66	6.10	4.54	427	4.0	0.24
SCC-34	UII	413.61	417.27	3.66	8.40	1,010	7.7	0.40
SCC-46	BHP	400.35	412.76	12.41	5.47	295	2.4	0.27
Total	**N/A**	**N/A**	**N/A**	**120.56**	**4.17**	**484**	**5.0**	**0.31**

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 assay statistics were tabulated at four different cut-off grades for uncapped copper, cobalt, silver, and gold in Table 14.11 (left portion of table). The data summarized in Table 14.11 include the number of metres at each cut-off grade, mean grades, standard deviations, and CVs. Incremental data (i.e. statistics for material between cut-off grades) are also tabulated. For example, 66% of the LZ intersections are above a 1% copper cut-off grade, with 34% less than that cut-off. Basic statistics are also summarized for grades after high-grade outlier values were capped (refer to Section 14.16 regarding grade capping).

The CV, which is the ratio of standard deviation over the mean, for the Johnny Lee LZ copper assays was reduced from 1.10 to 0.96 by grade capping. Cobalt shows the highest CV suggesting that there are more outlier values in that population. Silver grades are not particularly significant with only 2% of the LZ assays above 15 g/t. Approximately 10% of the LZ gold assays are above 1 g/t.

Table 14.11 Johnny Lee LZ Assay Statistics

Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
Cu Cutoff (%)	Total (m)	Inc %	Mean Cu (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.00	120.56	34	5.13	619	2	5.66	1.10	4.17	502	2	4.01	0.96
1.00	79.32	15	7.64	606	4	5.49	0.72	6.18	490	5	3.55	0.58
2.00	61.46	3	9.48	583	2	4.89	0.52	7.59	466	2	2.73	0.36
3.00	57.83	48	9.91	573	93	4.72	0.48	7.90	457	91	2.50	0.32

Uncapped Co Statistics Above Cut-off								Capped Co Statistics Above Cut-off				
Co Cutoff (g/%)	Total (m)	Inc %	Mean Co (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV	Mean Co (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.00	120.56	22	0.05	6.29	3	0.12	2.25	0.05	5.84	3	0.09	1.91
0.01	93.83	57	0.07	6.13	25	0.13	1.99	0.06	5.67	27	0.10	1.68
0.05	24.86	9	0.18	4.53	12	0.21	1.16	0.16	4.08	13	0.16	0.95
0.10	14.06	12	0.27	3.77	60	0.25	0.93	0.24	3.32	57	0.18	0.74

Uncapped Ag Statistics Above Cut-off								Capped Ag Statistics Above Cut-off				
Ag Cutoff (g/t)	Total (m)	Inc %	Mean Ag (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV	Mean Ag (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.00	120.56	58	5.04	607	24	4.66	0.92	5.04	607	24	4.66	0.92
5.00	51.06	30	9.03	461	41	4.66	0.52	9.03	461	41	4.66	0.52
10.00	15.20	10	14.02	213	24	5.77	0.41	14.02	213	24	5.77	0.41
15.00	2.62	2	25.49	67	11	4.79	0.19	25.49	67	11	4.79	0.19

table continues…

Uncapped Au Statistics Above Cut-off							
Au Cutoff (g/t)	Total (m)	Inc %	Mean Au (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.00	120.56	57	0.35	42	15	0.41	1.18
0.25	51.65	21	0.69	35	22	0.43	0.62
0.50	26.47	12	0.99	26	23	0.41	0.42
1.00	11.90	10	1.40	17	40	0.24	0.17

Capped Au Statistics Above Cut-off				
Mean Au (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.31	37	17	0.31	1.00
0.60	31	25	0.26	0.44
0.81	21	26	0.19	0.23
1.00	12	32	0.00	0.00

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 14.17, Figure 14.18, and Figure 14.19 show copper, cobalt, 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 14.12 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 14.12 Johnny Lee LZ Grade Capping Limits

Metal	Capping Limit	No. Capped	Apparent Metal Reduction (%)
Copper	10.0%	56	18.8
Cobalt	0.6%	3	7.2
Silver	N/A	N/A	N/A
Gold	1 g/t	14	11.0

Figure 14.17 Johnny Lee LZ Cumulative Probability Plot – Copper

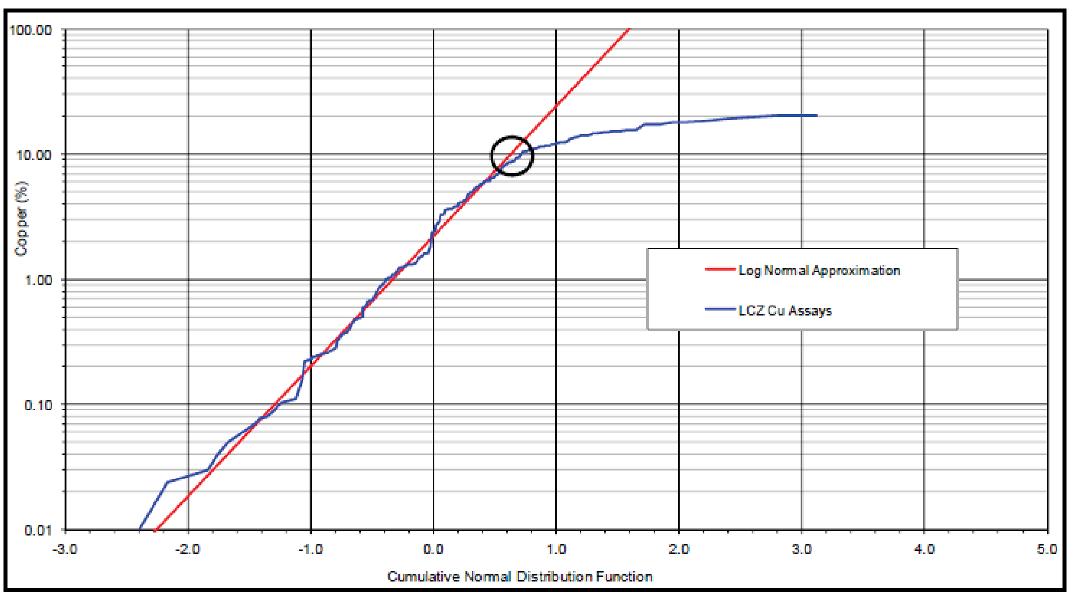


Figure 14.18 Johnny Lee LZ Cumulative Probability Plot – Cobalt

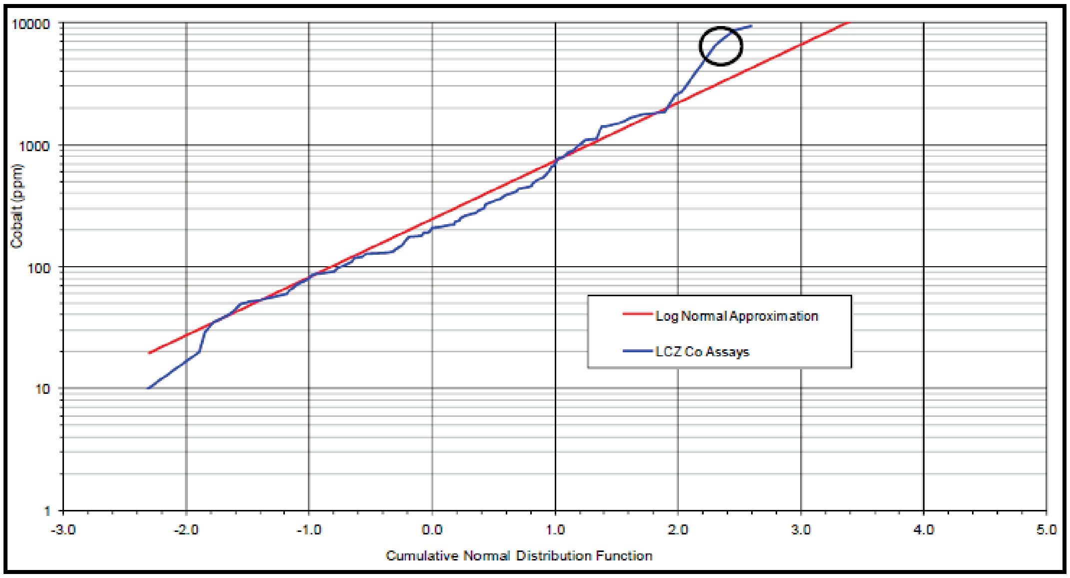
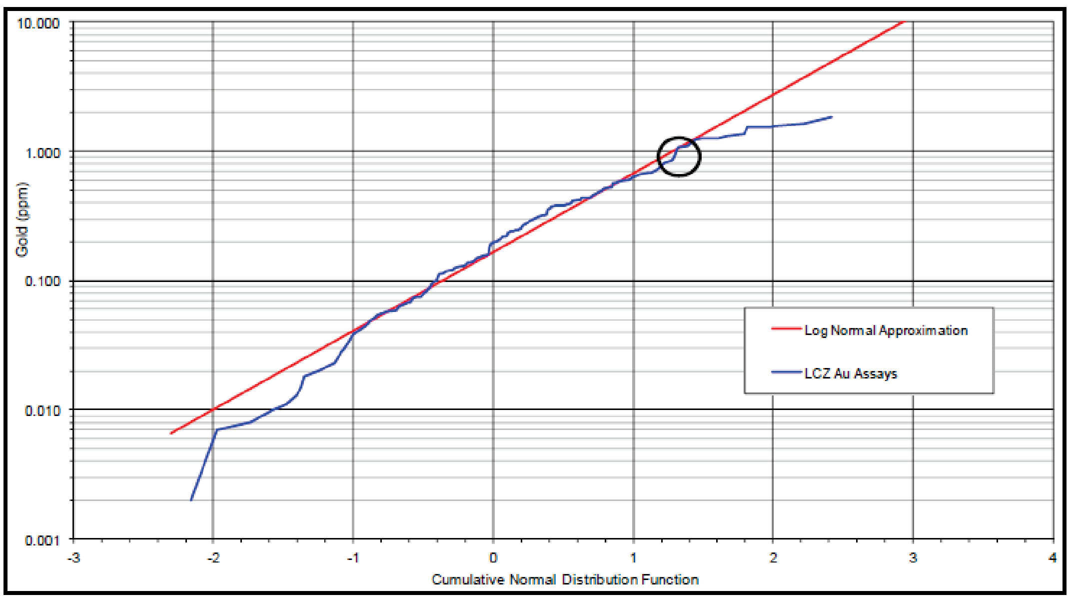


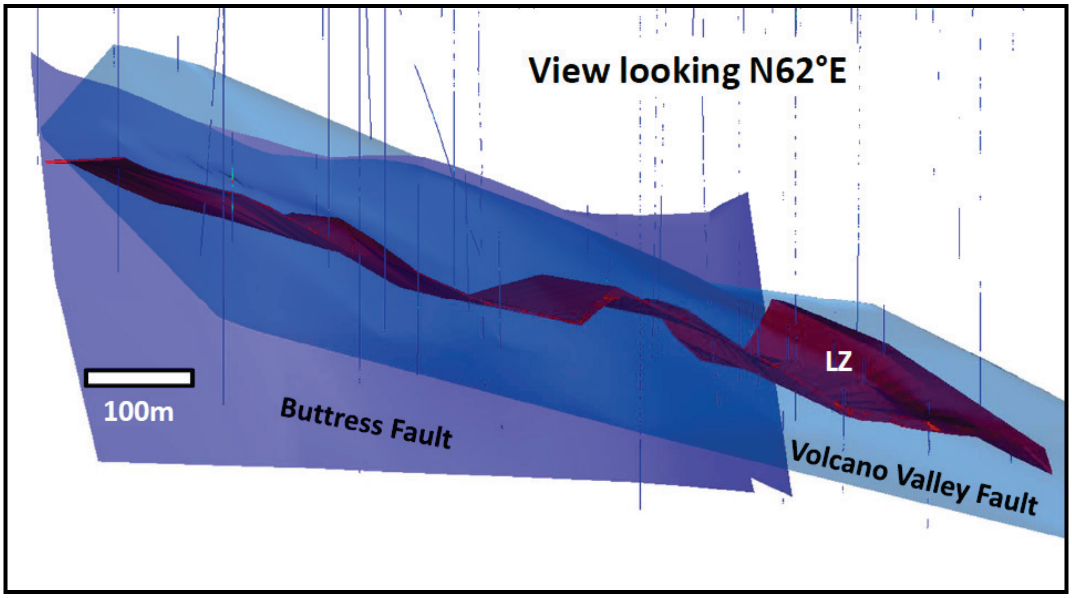
Figure 14.19 Johnny Lee LZ Cumulative Probability Plot – Gold



14.18 JOHNNY LEE LZ DOMAIN

Mr. Vincent Scartozzi, a 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 Grade Estimation

RMI constructed a 3D block model for the Johnny Lee LZ using MineSight® software. Table 14.13 summarizes the limits of the model and size of the blocks.

Table 14.13 Johnny Lee LZ Block Model Limits

Parameter	Minimum	Maximum	Extent (m)	Size (m)	Number
Easting (columns)	506,000	507,500	1,500	10	150
Northing (rows)	5,180,600	5,181,200	600	10	60
Elevation (levels)	1,100	1,500	400	2	200

RMI constructed separate inverse distance estimation plans for estimating "base" metals (copper, lead, zinc, iron, sulphur, barium, and arsenic) and "precious" metals (gold, silver, and cobalt). ID powers were 3 and 5 for base and precious metals, respectively. These powers were selected based on grade comparisons were made with NN models. 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 14.14 and Table 14.15 summarizes ID parameters for base and precious metals, respectively.

Table 14.14 Johnny Lee LZ Base Metal Estimation Parameters

Composite Selection			Ellipse Dimensions (m)			Ellipse Orientation		
Minimum	Maximum	Maximum/ Hole	Major Axis	Minor Axis	Vertical Axis[1]	ROTN[2]	DIPN[3]	DIPE[4]
1	8	2	250	250	100	280	0	12

Notes:
1. Vertical axis range is essentially truncated to the thickness of the wireframe.
2. Rotation about vertical axis using "left hand rule" generating a new major axis orientation.
3. Rotation about the new X-axis using "right hand rule"; in this case no rotation was done.
4. Rotation about the new Y-axis using the "left hand rule" causing the minor axis to plunge upward

Table 14.15 Johnny Lee LZ Precious Metal Estimation Parameters

Composite Selection			Ellipse Dimensions (m)			Ellipse Orientation		
Minimum	Maximum	Maximum/ Hole	Major Axis	Minor Axis	Vertical Axis[1]	ROTN[2]	DIPN[3]	DIPE[4]
1	3	1	250	250	100	280	0	12

Notes:
1. Vertical axis range is essentially truncated to the thickness of the wireframe.
2. Rotation about vertical axis using "left hand rule" generating a new major axis orientation.
3. Rotation about the new X-axis using "right hand rule"; in this case no rotation was done.
4. Rotation about the new Y-axis using the "left hand rule" causing the minor axis to plunge upward

14.20 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 14.21 and Figure 14.22 are cross sections through the block model showing composite and block copper grades. For reference, Figure 7.4 in Sectio 7.0 is a plan map showing the lines of section for Section D-D' and Section E-E'. Figure 7.6 in Section 7.0 is a geologic cross section through D-D'.

Figure 14.21 Johnny Lee LZ Block Model Cross Section D-D'

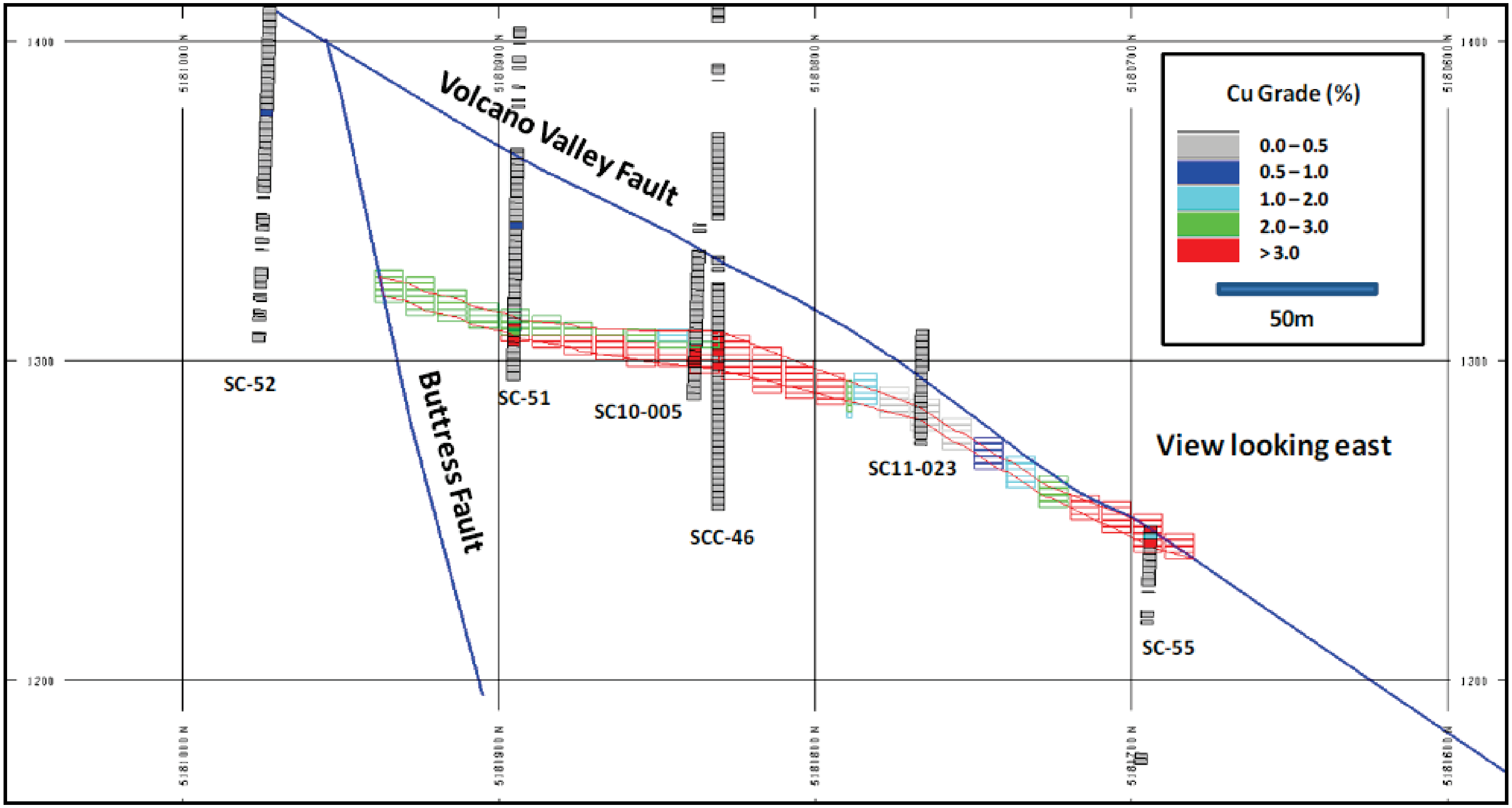
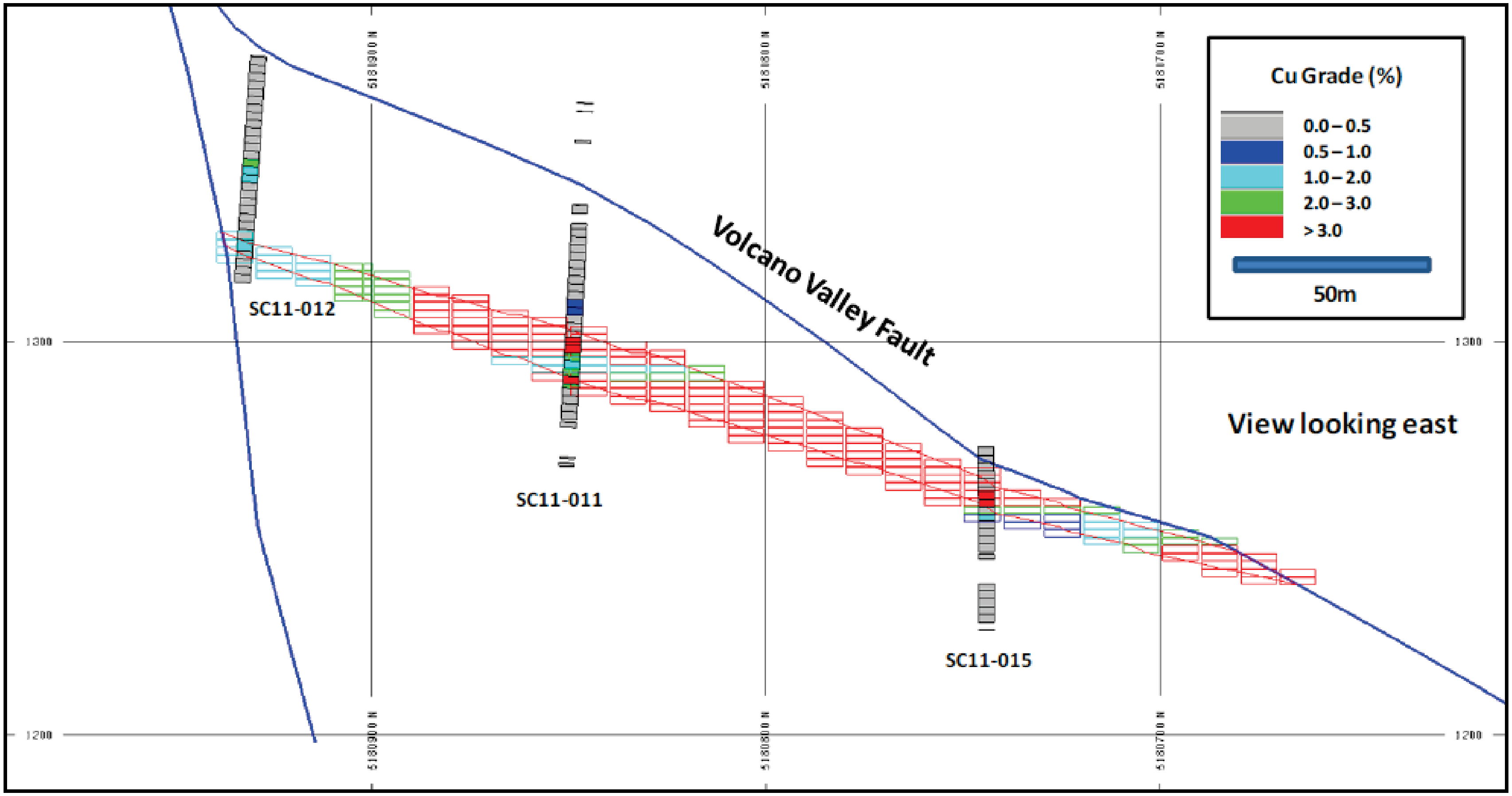


Figure 14.22 Johnny Lee LZ Block Model Cross Section E-E'



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 14.16 compares copper and cobalt grades estimated by ID and NN methods.

Table 14.16 Johnny Lee LZ Global Bias Check

Source of Estimate	Cu (%)	Co (ppm)	Au (g/t)	Ag (g/t)
IDW Estimate	4.4146	549.08	0.3379	5.0507
NN Estimate	4.2614	543.61	0.3277	4.9015
Percent Difference (%)	3.60	1.01	3.11	3.04

The data in Table 14.16 show a close comparison between the ID 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 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 14.23 through Figure 14.25 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 14.23 Johnny Lee LZ Copper Swath Plot – Easting

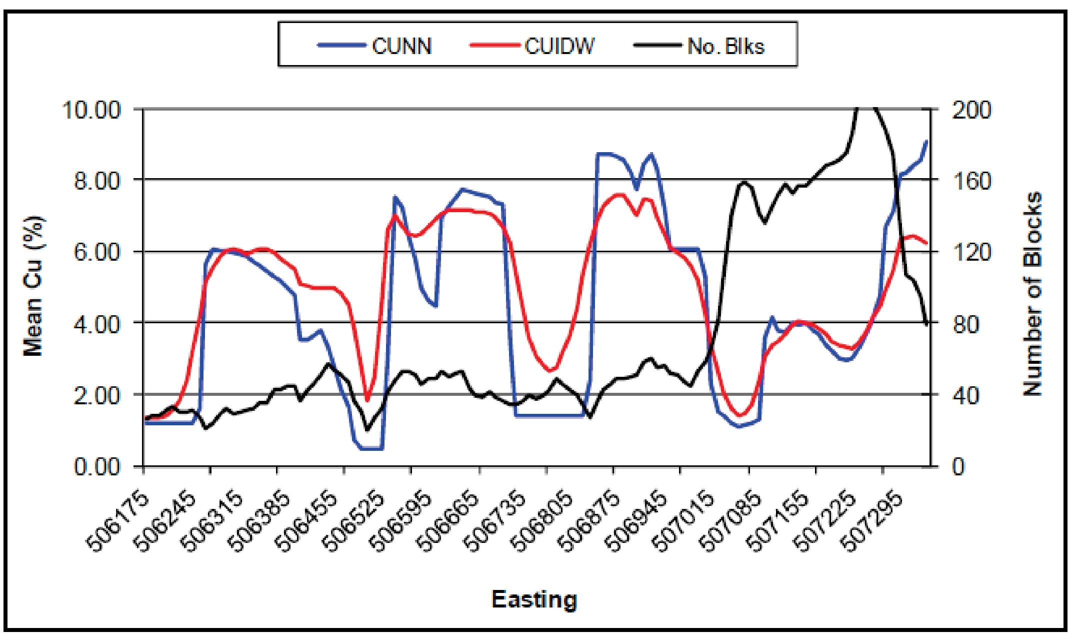


Figure 14.24 Johnny Lee LZ Copper Swath Plot – Northing

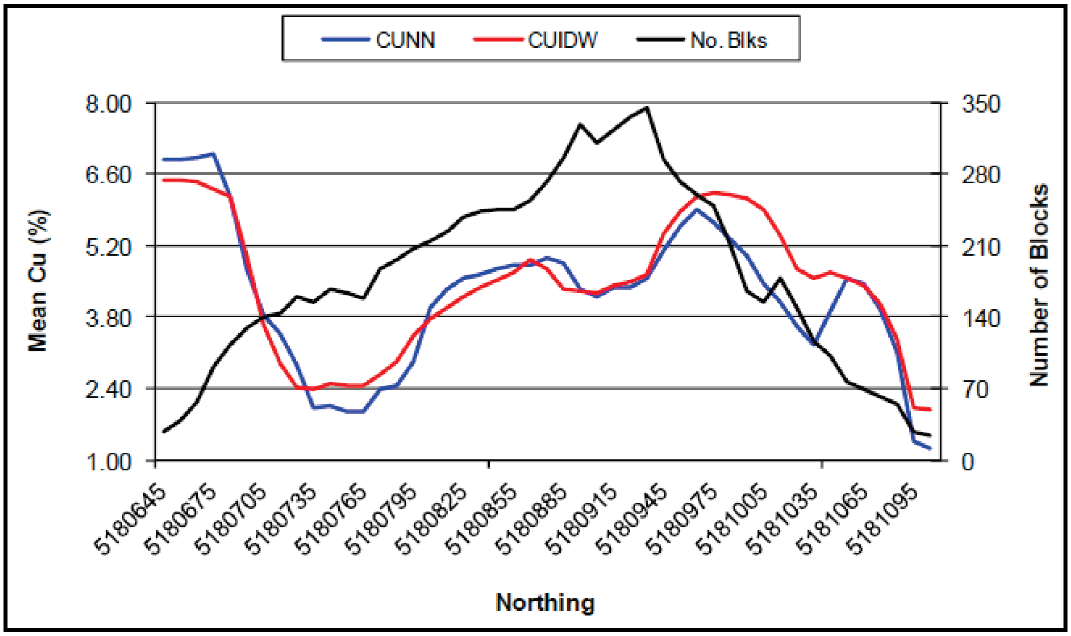
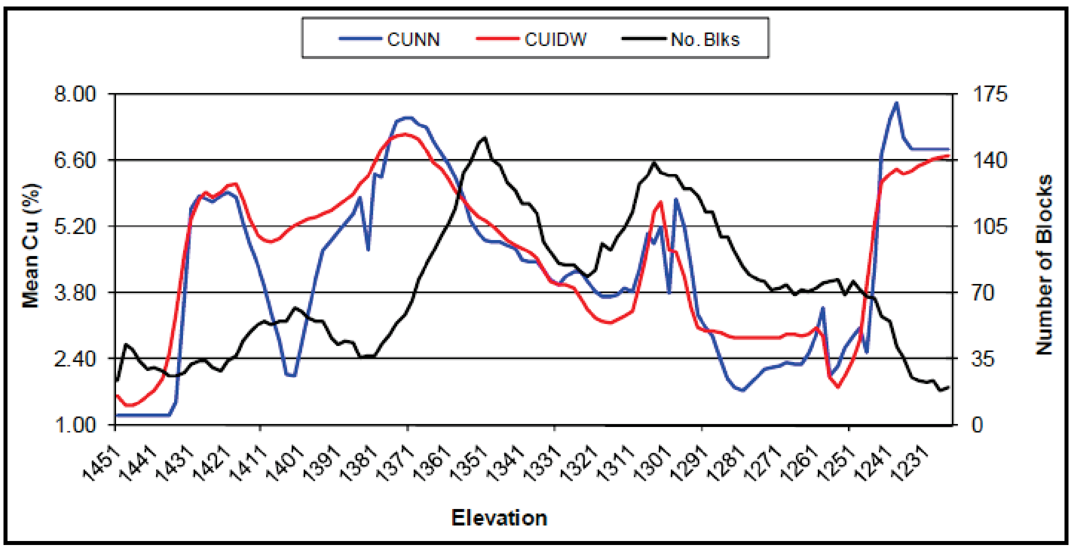


Figure 14.25 Johnny Lee LZ Copper Swath Plot – Elevation



The swath plots shown in Figure 14.23 through Figure 14.25 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 west to east (i.e. Figure 14.23). Similarly, copper grades tend to be relatively high-grade at the far south end of the zone with a dramatic drop in grade around 5,180,750 north. Copper then steadily increases in grade for about 250 to 300 m before decreasing.

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.

14.21 JOHNNY LEE LZ RESOURCE CLASSIFICATION

Blocks that were estimated by one or more drillholes within 75 m of a block were classified as Inferred Resources. RMI notes that no Inferred blocks were estimated by less than three drillholes and 83% of the Inferred Resource tonnage was estimated by five drillholes.

According to the CIM, an Inferred Mineral Resource is defined as: "*that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes*". Based on this definition, it is RMI's opinion that a portion of the Johnny Lee LZ Butte qualifies as an Inferred Resource. There is ample geologic evidence for the bedded massive sulphide zones and sufficient drilling to demonstrate the extent and geometry of the mineralized system.

14.22 JOHNNY LEE LZ DENSITY DATA

Tintina personnel obtained bulk density determinations from 17 pieces of drill core taken from their 2010 and 2011 drilling programs, as summarized in Table 14.17. These samples represent massive sulphide material from the Johnny Lee LZ. 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 14.17 Johnny Lee LZ Bulk Density Determinations

BHID	Depth (m)	Dry Weight (g)	Wet Weight (g)	SG	Description
SC11-012	386.40	1,365.00	1,024.20	4.01	Massive cp + py
SC11-029	437.50	1,786.10	1,309.10	3.74	Massive cp + py w/ carbonaceous residue
SC11-029	437.85	1,017.50	754.95	3.88	Massive py + cp w/ carbonaceous residue
SC11-029	439.40	1,184.50	888.30	4.00	Massive cp and pyrite
SC11-029	440.40	1,261.00	927.90	3.79	Massive py and cp
SC11-029	441.45	1,338.10	995.20	3.90	Silicified dol shale w/ banded fg pyrite
SC10-005	411.00	400.00	287.60	3.56	LSZ, cpy
SC11-031	429.08	1,191.10	883.60	3.87	Msv py

table continues…

BHID	Depth (m)	Dry Weight (g)	Wet Weight (g)	SG	Description
SC11-012	348.40	1,284.10	902.30	3.36	Laminated sulphide "mud"
SC11-012	372.30	1,840.00	1,374.10	3.95	Massive py w/ carbonaceous material
SC11-012	377.90	1,533.70	1,152.50	4.02	Massive py w/ VF and algal textures
SC11-031	350.14	2,748.60	2,002.60	3.68	Msv py + 1% cpy
SC11-031	397.65	1,799.10	1,240.20	3.22	Msv py
SC10-005	388.65	446.50	331.50	3.88	LSZ, mostly py
SC11-015	452.52	795.80	588.10	3.83	Cpy rich, slightly carb msv
SC11-015	446.84	1,347.50	977.85	3.65	Pyrite as "lumpy" wavy lams and clots
SC11-048	365.85	1,194.30	911.00	4.22	Cpy+ py
Average	407.25	1,325.46	973.59	3.80	N/A

Based on these results, RMI chose to use a bulk dry density of 3.80 for tabulating Inferred Resources for the Johnny Lee LZ.

14.23 Johnny Lee LZ Resource Summary

A cut-off grade of 1.5% copper was used to define an undiluted Inferred Mineral Resource for the LZ of 2,462,000 t with an average grade of 4.71% copper, 0.06% cobalt, 0.35 g/t gold, and 5.1 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 and refining costs of US$5.53/t. Table 14.18 summarizes resources at several cut-off grades. 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.

Figure 14.26 shows grade-tonnage curves from the data tabulated in Table 14.18.

Figure 14.26 Johnny Lee LZ Grade-tonnage Curves

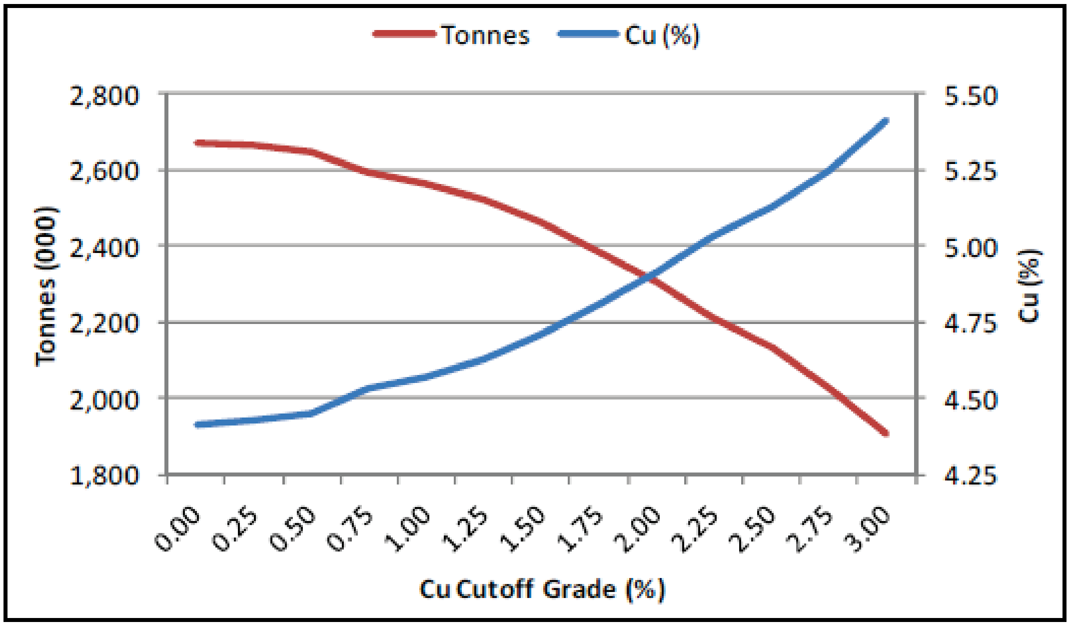


Table 14.18 Johnny Lee LZ Undiluted Inferred Mineral Resource

Cu Cutoff (%)	Tonnes (000)	Cu (%)	Co (ppm)	Au (g/t)	Ag (g/t)	Cu ('000 lb)	Co ('000 lb)	Au ('000 oz)	Ag ('000 oz)
0.00	2,671	4.41	549	0.34	5.1	259,612	3,232	29	438
0.25	2,663	4.43	549	0.34	5.1	260,008	3,222	29	437
0.50	2,649	4.45	546	0.34	5.1	259,809	3,188	29	434
0.75	2,592	4.53	549	0.34	5.0	258,788	3,136	28	417
1.00	2,563	4.57	550	0.34	5.0	258,153	3,107	28	412
1.25	2,520	4.63	547	0.34	5.0	257,154	3,038	28	405
1.50	**2,462**	**4.71**	**550**	**0.35**	**5.1**	**255,576**	**2,984**	**28**	**404**
1.75	2,387	4.81	558	0.35	5.1	253,052	2,936	27	391
2.00	2,304	4.91	557	0.34	5.1	249,331	2,828	25	378
2.25	2,212	5.03	555	0.34	5.1	245,225	2,706	24	363
2.50	2,131	5.13	557	0.33	5.1	240,942	2,616	23	349
2.75	2,028	5.25	551	0.33	5.1	234,660	2,463	22	333
3.00	1,905	5.41	566	0.33	5.2	227,145	2,376	20	318

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.

14.24 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 Inferred Mineral Resources discussed in this report.

A significant amount of 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.

14.25 Lowry Drilling Data

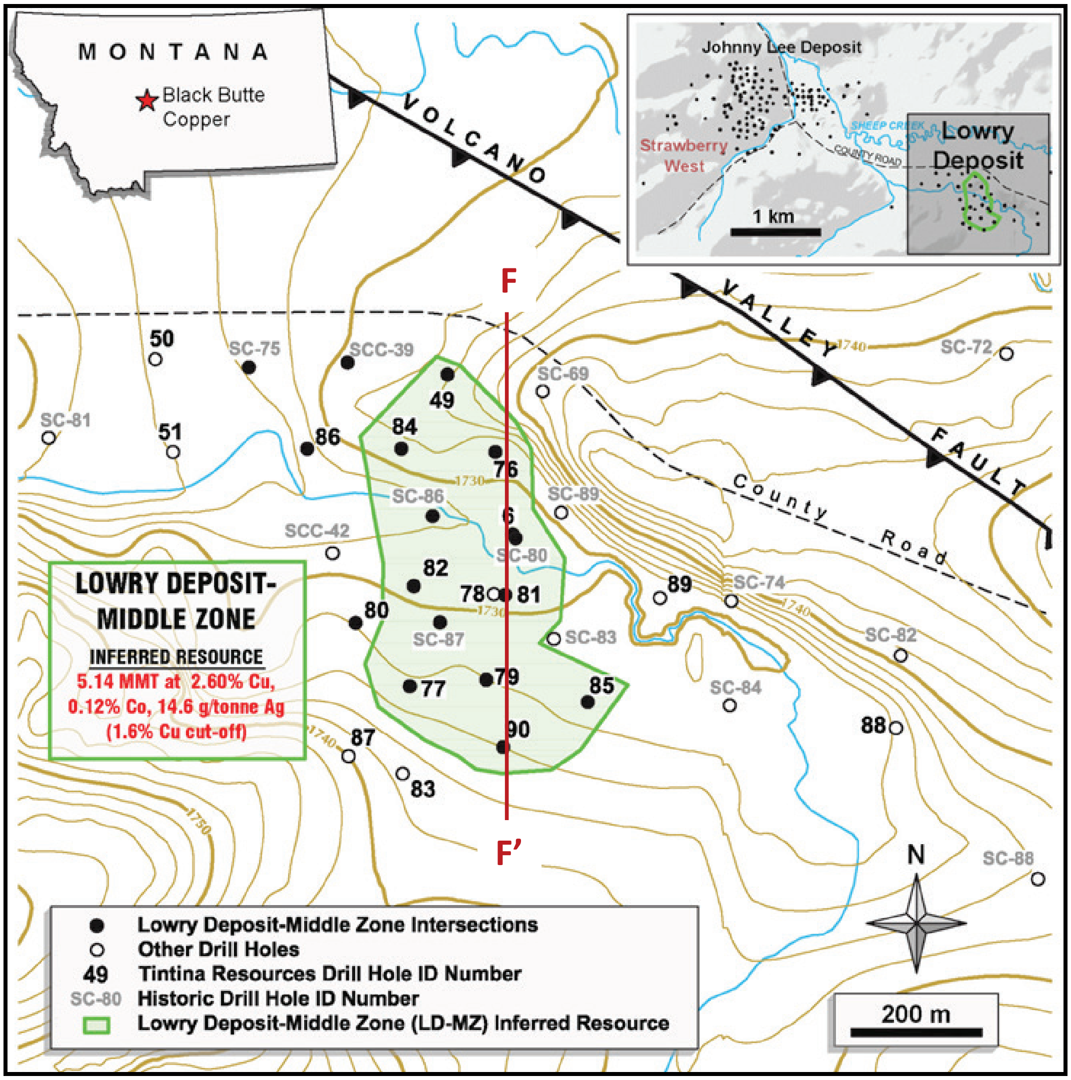
The Lowry Zone resource is based on a majority of drilling data that has been collected by Tintina. Data collected by CAI were used in conjunction with the newly acquired Tintina drilling data to estimate mineral resources for the Lowry Zone. Table 14.19 summarizes the drillhole data that were used by RMI to estimate mineral resources for the Lowry zone. The information in Table 14.19 includes the company that drilled the hole, beginning and ending depth of the horizon, the MZ intersection length and average copper, cobalt, silver, and gold grades for the MZ intersections. The grades shown in Table 14.19 are based on capped raw assays (see Section 14.27 for a discussion of grade capping limits).

Table 14.19 Lowry Zone Drillhole Data

Drillhole	Company	From Depth (m)	To Depth (m)	Length (m)	Lowry Zone Intersections			
					Cu (%)	Co (ppm)	Ag (g/t)	Au (g/t)
SC10-006	Tintina	384.02	430.64	46.62	2.57	1,159	13.27	0.01
SC11-049	Tintina	264.76	284.29	19.53	1.65	1,161	10.53	0.01
SC11-076	Tintina	319.36	356.87	37.51	2.38	1,257	13.06	0.00
SC11-077	Tintina	540.05	563.09	23.04	1.66	279	16.13	0.00
SC11-079	Tintina	550.00	593.75	43.75	1.83	1,067	18.72	0.02
SC11-081	Tintina	443.66	467.03	23.37	1.81	741	10.57	0.00
SC11-082	Tintina	455.00	460.14	5.14	1.47	588	20.21	0.00
SC11-084	Tintina	324.12	334.19	10.07	2.29	1,787	14.92	0.03
SC11-085	Tintina	659.98	668.13	8.15	3.31	1,069	12.61	0.01
SC11-090	Tintina	675.00	718.50	43.50	0.70	677	15.11	0.00
SC-80	CAI	393.50	444.40	50.90	2.80	1,118	11.25	0.00
SC-86	CAI	358.75	377.65	18.90	0.90	493	7.70	0.01
SC-87	CAI	472.14	524.87	52.73	0.82	500	11.59	0.02
SC-87W	CAI	471.37	527.15	55.78	0.81	420	10.33	0.07
Total	**N/A**	**N/A**	**N/A**	**438.99**	**1.68**	**842**	**12.92**	**0.02**

Figure 14.27 is a plan map that shows the distribution of drilling within the Lowry Zone. This map also shows a line of section (F-F') for Figure 14.28.

Figure 14.27 Lowry Zone Drillhole Map



14.26 LOWRY EXPLORATORY DATA ANALYSIS

The Lowry MZ was modelled as a single lens of massive sulphide mineralization using logged geologic information and drillhole copper grades. Basic assay statistics were tabulated at four different cut-off grades for uncapped copper, cobalt, silver, and gold in Table 14.20 (left portion of table). The data summarized in Table 14.20 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, 50% of the MZ intersections are above a 1% copper cut-off grade and 50% are less than that cut-off. Basic statistics are also summarized for grades after high-grade outlier values were capped (refer to Section 14.25 regarding grade capping).

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 (see Section 14.25) slightly reduced the CV for the key metals (copper and cobalt).

Table 14.20 Lowry Zone Assay Statistics

Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
Cu Cutoff (%)	Total (m)	Inc %	Mean Cu (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.00	439	50	1.68	738	11.8	2.11	1.25	1.65	723	12.0	1.93	1.17
1.00	221	21	2.94	651	17.4	2.35	0.80	2.87	636	17.7	2.07	0.72
2.00	130	15	4.02	523	22.9	2.55	0.64	3.90	508	23.4	2.16	0.55
3.00	62	14	5.68	354	47.9	2.87	0.51	5.44	339	46.9	2.27	0.42

Uncapped Co Statistics Above Cut-off								Capped Co Statistics Above Cut-off				
Co Cutoff (g/%)	Total (m)	Inc %	Mean Co (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV	Mean Co (%)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.00	439	13	0.08	37	0.6	0.09	1.12	0.08	37	0.6	0.09	1.08
0.01	382	34	0.10	37	12.7	0.10	0.99	0.10	36	12.8	0.09	0.95
0.05	233	25	0.14	32	21.9	0.10	0.74	0.14	32	22.1	0.10	0.70
0.10	122	28	0.20	24	64.8	0.11	0.57	0.19	24	64.5	0.10	0.53

Uncapped Ag Statistics Above Cut-off								Capped Ag Statistics Above Cut-off				
Ag Cutoff (g/t)	Total (m)	Inc %	Mean Ag (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV	Mean Ag (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0	439	13	12.92	5,672	2.4	8.27	0.64	12.89	5,658	2.4	8.15	5.12
5	380	27	14.58	5,538	15.5	7.63	0.52	14.54	5,524	15.5	7.48	0.70
10	260	23	17.94	4,661	21.5	6.95	0.39	17.89	4,647	21.6	6.75	0.28
15	157	36	21.94	3,439	60.6	6.19	0.28	21.86	3,425	60.5	5.86	0.22

table continues…

Uncapped Au Statistics Above Cut-off								Capped Au Statistics Above Cut-off				
Au Cutoff (g/t)	Total (m)	Inc %	Mean Au (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV	Mean Au (g/t)	Grd-Thk (%-m)	Inc %	Standard Deviation	CV
0.00	439	99	0.02	8	37.4	0.16	9.58	0.01	6	50.4	0.06	5.12
0.10	6	1	0.77	5	4.3	1.16	1.51	0.45	3	5.7	0.32	0.70
0.20	4	0	1.25	4	1.3	1.35	1.08	0.70	2	1.7	0.20	0.28
0.50	3	1	1.34	4	57.1	1.38	1.03	0.73	2	42.2	0.16	0.22

14.27 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 shows a copper probability plot for the Lowry Zone. The black circle in Figure 14.28 indicates the capping limit selected by RMI to minimize the potential for over estimating contained metal.

Figure 14.28 Lowry Zone Cumulative Probability Plot – Copper

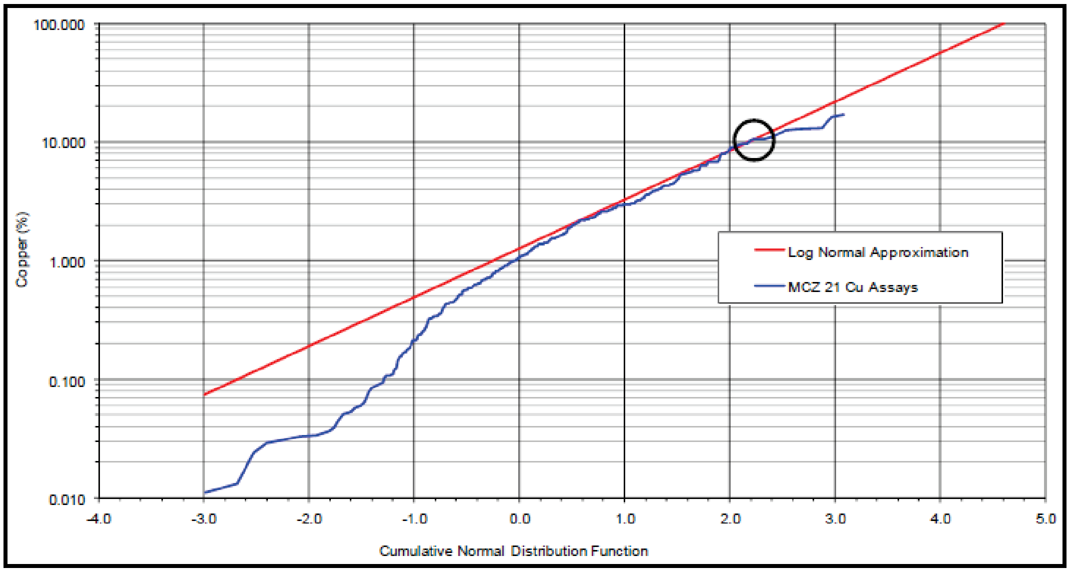


Table 14.21 summarizes high-grade outlier capping limits that were selected for copper, cobalt, silver, and gold for the Lowry Zone based on a review of probability plots. These limits were applied to the raw assays prior to creating drillhole composites.

Table 14.21 Lowry Grade Capping Limits

Metal	Capping Limit	No. Capped	Apparent Metal Reduction (%)
Copper	10.0%	11	2.00
Cobalt	0.50%	4	1.00
Silver	40 g/t	3	0.25
Gold	1 g/t	1	25.0

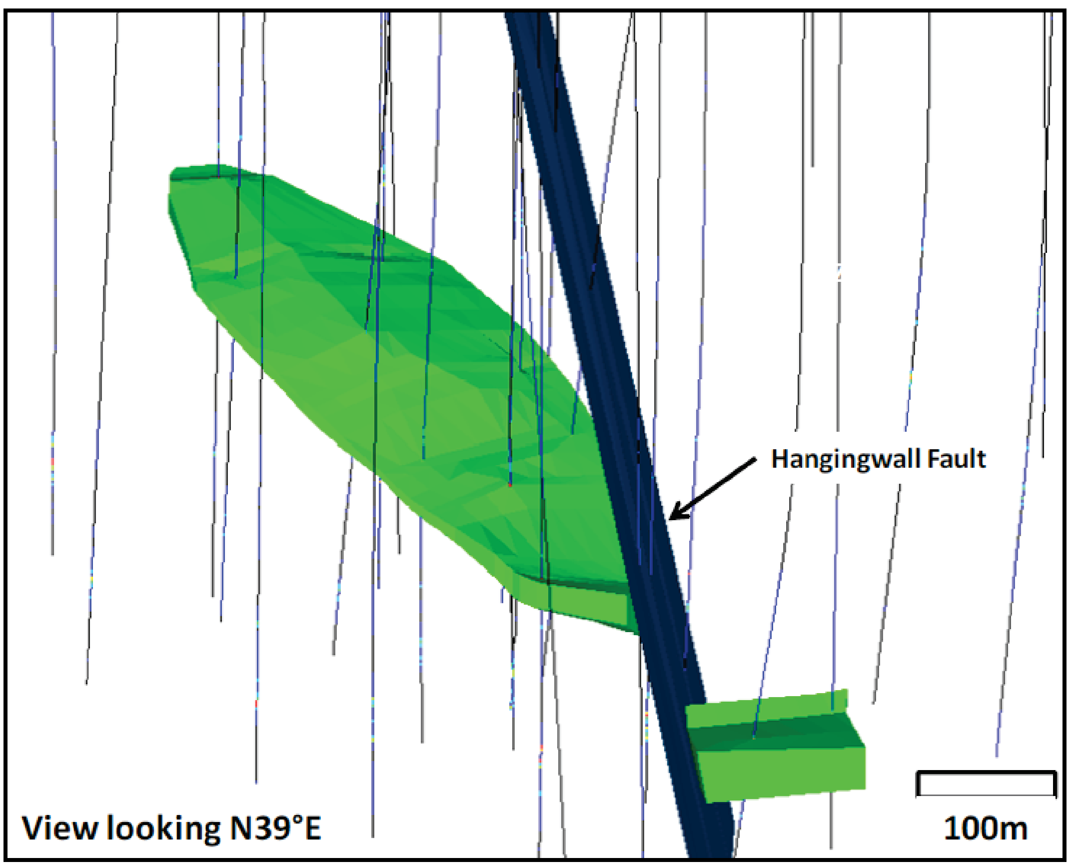
The apparent gold metal reduction of 25% for the Lowry Zone shows the affect of a single high-grade assay (4 g/t). The actual reduction of estimated gold metal is significantly less than Table 14.21 would suggest.

14.28 LOWRY DOMAINS

Mr. Vincent Scartozzi, a Senior Geologist with Tintina, constructed 3D wireframe to represent the Lowry MZ stratabound copper sulphide horizon. RMI reviewed the wireframe and believes 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 14.29 is a perspective view looking to the northeast showing a wireframe representing the Lowry mineralized copper zone. This wireframe was used to constrain the estimate of block grades.

Figure 14.29 Lowry MZ Wireframe Perspective



14.29 Lowry Grade Estimation

RMI constructed a 3D block model for the Lowry MZ using MineSight® software. Table 14.22 summarizes the limits of the model and size of the blocks.

Table 14.22 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

The capped drillhole assay data were composited into 2.5 m fixed lengths. The copper zone wireframe was used to control (start and stop) the compositing routine.

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 (refer to 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 14.23 summarizes the ID parameters that were used to estimate base and precious metals.

Table 14.23 Lowry MZ 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	6	1	200	200	200	10
2	3	2	6	2	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.

14.30 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 sample data. Figure 14.30 is cross section through the block model showing composite and block copper grades. For reference, Figure 14.29 is a plan map showing the line of section for Section F-F'.

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 14.24 compares copper, cobalt, silver, and gold grades estimated by ID and NN methods.

Table 14.24 Lowry MZ Global Bias Check

Metal	Resource Grade	NN Grade	Percent Difference (%)
Copper (%)	1.6923	1.6970	-0.28
Cobalt (ppm)	885.5	880.4	0.58
Silver (g/t)	12.69	12.84	-1.17
Gold (g/t)	0.0101	0.0090	12.22

The data in Table 14.24 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 14.30 through Figure 14.32 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 14.30 Lowry MZ Copper Swath Plot – Easting

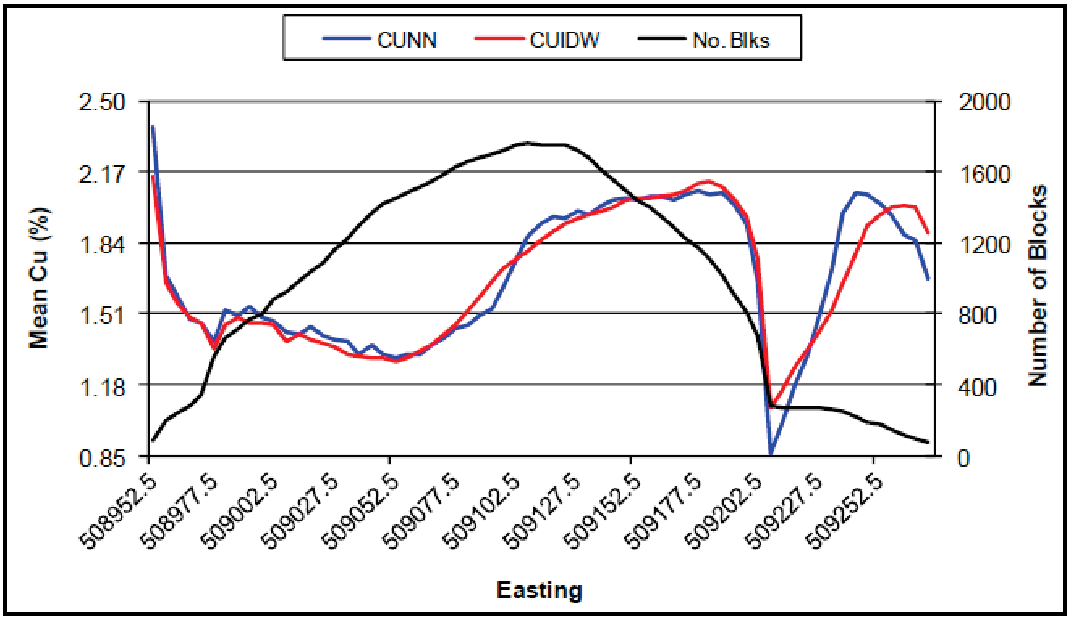


Figure 14.31 Lowry MZ Copper Swath Plot – Northing

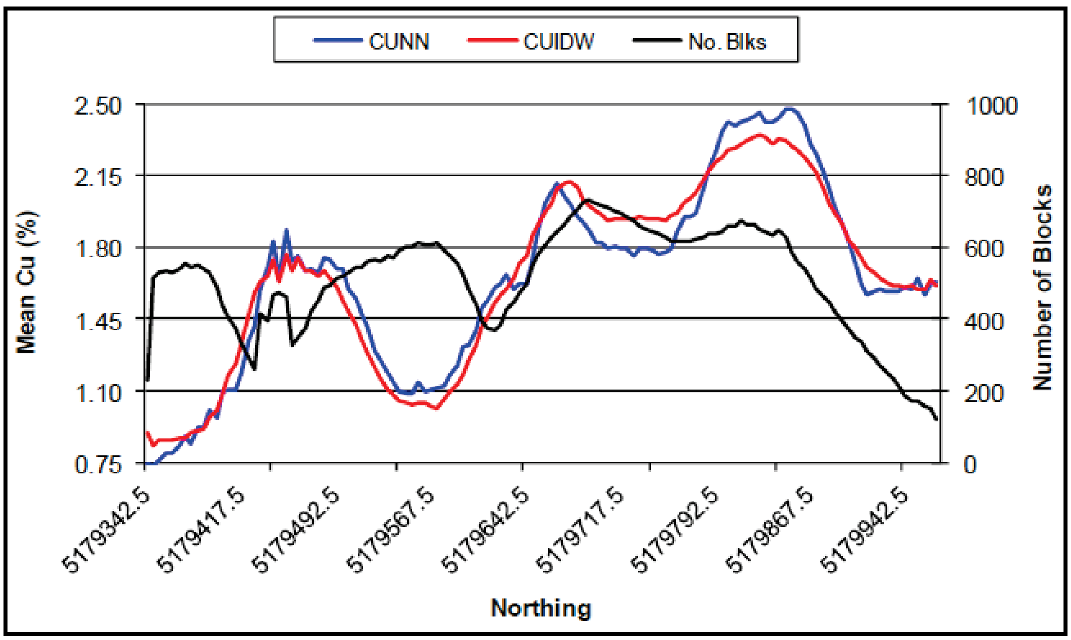
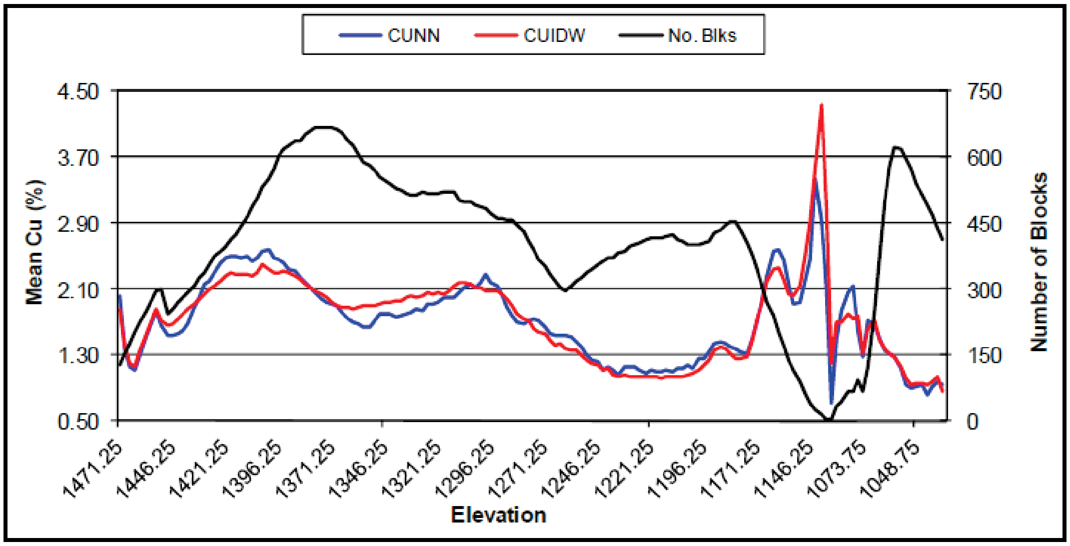


Figure 14.32 Lowry MZ Copper Swath Plot – Elevation



The swath plots shown in Figure 14.30 through Figure 14.32 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 14.31).

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.

14.31 Lowry Resource Classification

Blocks within the Lowry Zone wireframe that were estimated by one or more drillholes within 100 meters of a block were classified as Inferred Resources. RMI notes that 75% and 96% of the Inferred resource tonnage was estimated by holes within 60 and 75 meters of the blocks, respectively.

According to the CIM an Inferred Mineral Resource is defined as: "*that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes*". Based on this definition, it is RMI's opinion that a portion of the Lowry MZ qualifies as an Inferred Resource. There is ample geologic evidence for the bedded massive sulphide zones and sufficient drilling to demonstrate the extent and geometry of the mineralized system.

14.32 Lowry Density Data

A bulk density value of 3.35 g/cm^3 was used to tabulate resource tonnage. This value is based on the arithmetic average of 24 bulk density determinations obtained from recent Tintina diamond core hole samples.

14.33 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, and G&A costs of US$5.00/t. Table 14.18 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 14.25.

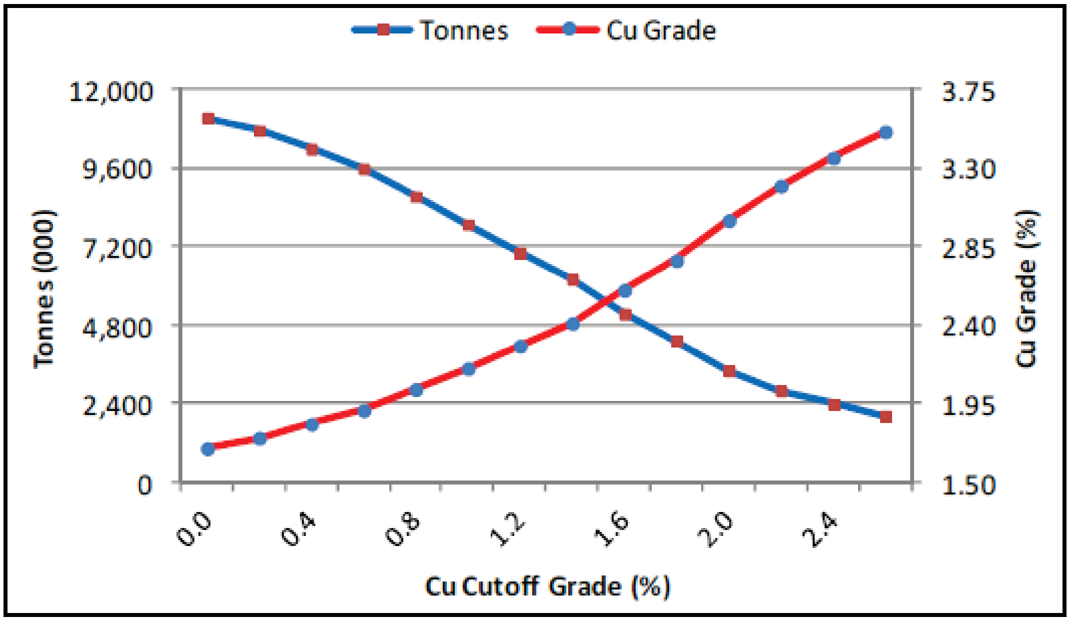
Table 14.25 Lowry MZ Undiluted Inferred Mineral Resource

Cu Cutoff (%)	Tonnes ('000)	Estimated Metal Grades				Estimated Contained Metal			
		Cu (%)	Co (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Au ('000 oz)	Ag ('000 oz)
0.00	11,157	1.69	0.09	0.010	12.7	416	22	3.6	4,556
0.20	10,753	1.75	0.09	0.010	12.9	415	21	3.5	4,460
0.40	10,191	1.83	0.09	0.011	13.1	411	20	3.6	4,292
0.60	9,595	1.91	0.10	0.011	13.2	404	21	3.4	4,072
0.80	8,732	2.03	0.10	0.011	13.2	391	19	3.1	3,706
1.00	7,876	2.15	0.11	0.011	13.4	373	19	2.8	3,393
1.20	7,015	2.28	0.11	0.011	13.7	353	17	2.5	3,090
1.40	6,196	2.41	0.11	0.010	14.0	329	15	2.0	2,789
1.60	**5,139**	**2.60**	**0.12**	**0.009**	**14.6**	**294**	**14**	**1.5**	**2,412**
1.80	4,327	2.77	0.12	0.009	14.9	264	11	1.3	2,073
2.00	3,398	3.00	0.12	0.010	15.4	225	9	1.1	1,682
2.20	2,791	3.20	0.12	0.010	15.9	197	7	0.9	1,427
2.40	2,382	3.36	0.13	0.010	16.4	176	7	0.8	1,256
2.60	2,024	3.51	0.13	0.011	17.0	157	6	0.7	1,106
2.80	1,762	3.63	0.13	0.010	17.2	141	5	0.6	974
3.00	1,530	3.74	0.14	0.010	17.4	126	5	0.5	856

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.

Figure 14.33 shows grade-tonnage curves from the data tabulated in Table 14.25.

Figure 14.33 Lowry MZ Grade-tonnage Curves



14.34 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.

A significant amount of 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 reader should be aware that no resources from the Lowry deposit have been included in the mine plan for purposes of assessing the project economics at a PEA level. The Lowry MZ was defined too late for inclusion in this PEA, but should be assessed in future studies.

15.0 MINERAL RESERVE ESTIMATES

There are no mineral reserves on the Property.

16.0 MINING METHODS

16.1 MINE ACCESS AND DEVELOPMENT

The UZ is flatly dipping (approximately 20°) and relatively shallow (approximately 105 to 230 m depth below local surface). The LZ also has an overall shallow dip, but somewhat more erratic geometry and extends from approximately 450 to 580 m depth below local surface.

For a lens thickness of typically only a few metres, Stantec determined that open pit mining would involve an extremely high stripping ratio that would render an open pit mining method impractical and uneconomic. In addition, local land owners will require a minimized site footprint. For these reasons, Stantec selected underground mining methods for both zones.

The underground mine workings will be accessed via a 15% gradient decline from surface. The decline will be used to transport all workforce and supplies into the mine, and to haul mineralized material and waste via truck to the surface. For the depth and scale of operations, shaft/hoisting facilities will not be warranted.

Figure 16.1 shows where the propose mine portal will be located. Provision is included for installation of a cover (culvert) for the immediate portal opening. The mine portal will be located on the footwall side of the mineralized material zone in order to avoid line-of-sight with the nearby state highway and to avoid proximity to local landowners' buildings to minimize noise and esthetic issues. The site was also chosen to minimize potentially acid generating (PAG) material in the underground development. The site is close to the proposed concentrator site to minimize re-handling of mineralized material on surface. Figure 16.2 provides a photograph of the area near the proposed portal, to illustrate general topography.

Figure 16.1 Surface Layout with Proposed Mine Openings

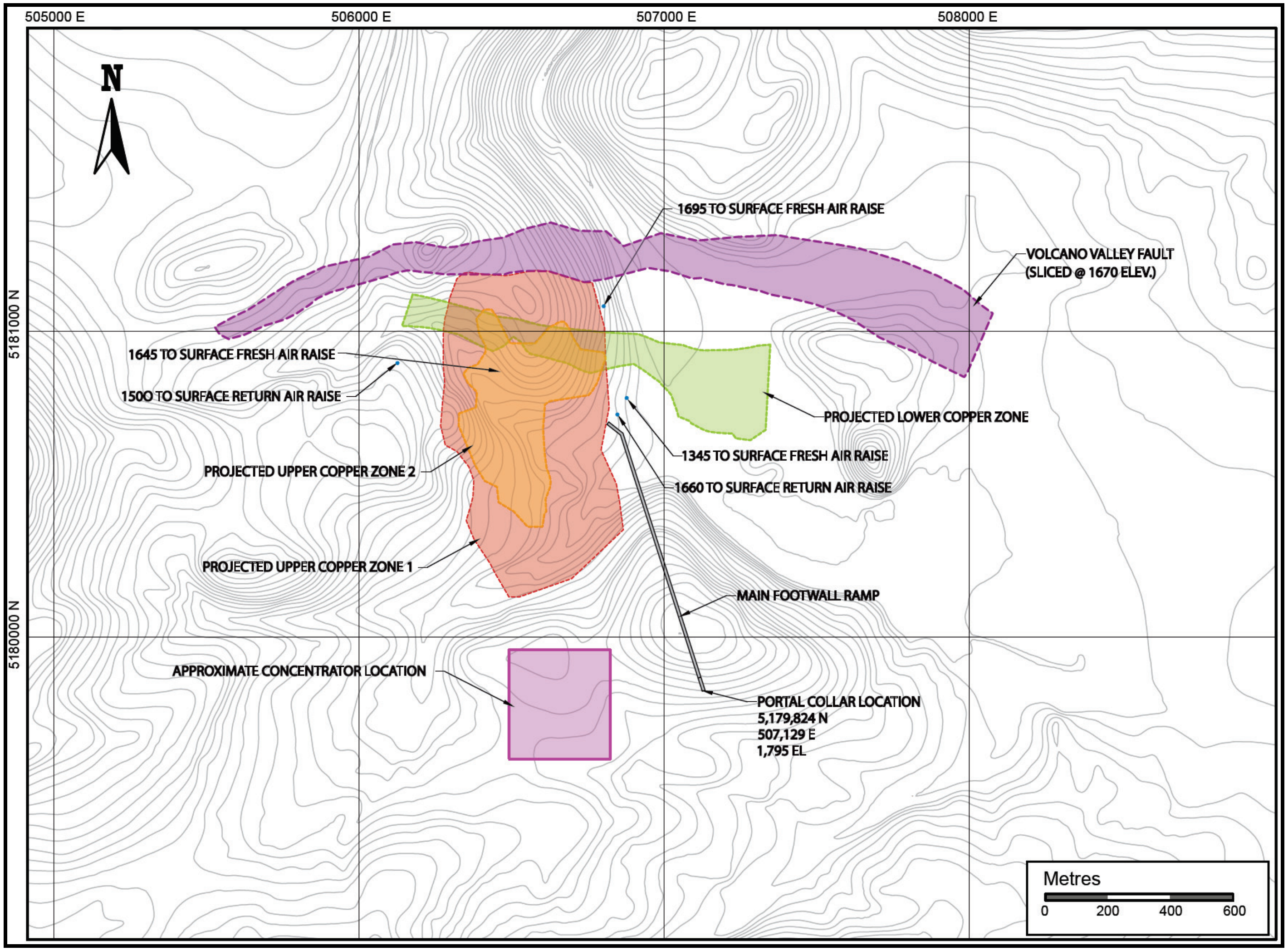


Figure 16.2 General Topography of the Area

To maximize longer term stability, the decline, or main access ramp, is maintained at a stand-off distance of typically 60 m or more from the mineralized material body. Development will continue following the mineralized material zone on the footwall side as further described in Section 16.2 (refer to Figure 16.8). Additional design considerations for the layout of the ramp from surface include:

- Avoid as much as practical, identified sulphide waste rock areas; any non-potentially acid generating (non-PAG)/metal leaching development waste (dolomite etc.) may be considered an asset for use as construction material rather than an environmental liability.
- Avoid developing repeatedly through the volcanic fault VVF which bounds the north end of the UZ.
- Prioritize development to specific areas on each production level, in order to access above average mineralized material grade resource as early as possible in the production schedule.
- Provide access to ventilation/escape raises with surface location acceptable to local landowners while avoiding developing the raises directly through known major geological structures.
- Locate the ramp to enable crosscuts to be driven roughly perpendicular to the strike of the mineralized material zone, roughly mid-strike length of the

UZ. Advance within each stope can then proceed both north and south of the crosscut access, thereby maximizing the period of time that multiple faces are available to mine, while avoiding excessive stope lengths to ventilate.

- Size the decline 5 m high by 6 m wide based on ventilation requirements. This will also accommodate underground diesel haulage trucks up to 40 to 50 t class with reasonable wall clearance.

16.2 Mining and Backfill Methods

Mineralized material zone geometry and geotechnical considerations are among key factors in determining a suitable mining method.

The UZ consists of a lower lens (UZ1), and a smaller upper lens (UZ2) situated immediately above UZ1. The zones are typically separated by a few metres of sulphide waste rock, but converge in some areas. The zone gently dips at approximately 20° southwest. There appears to be some local undulation of the mineralized material contacts. In most sections, total vertical thickness ranges from 2 to 8 m, therefore a blast hole method does not appear to be practical. Geological cross sections through the zones are shown in Figure 16.3 and Figure 16.4.

Potential mining methods considered included room-and-pillar, overhand post-pillar cut-and-fill, and drift/slash-and-fill.

Mining methods which employ backfill merit preference from the perspective of disposing as much sulphide tailings underground as practical. Underground backfilling could accommodate approximately half of the tailings produced over the LOM, hence, Room-and-Pillar mining is ranked lower.

Cut-and-fill and drift-and-fill methods would yield fairly comparable mining costs overall, but drift-and-fill is considered the preferred method as it provides opportunity to minimize the amount of mineralized material left unrecovered with patterned pillars.

For scheduling and cost estimation purposes, the mining method assumed for this PEA is drift/slash-and-fill. While this can be an extremely adaptive and flexible method, it is considered a high cost. As such, the operating cost estimates will tend to be conservative.

The particulars of the method are described below:

- Cross-cuts will be driven at minus 15% gradient from the ramp to the mineralized material at nominal 25 m vertical intervals.
- Within each stope horizon, primary “drifts” in mineralized material will be advanced along strike both north and south from the access. There is

considerable flexibility in the dimensions of these drifts. Where the mineralized material is thicker/flatter it is assumed the back height will be a nominal 5 m, and span up to 8 m (to be verified during future geotechnical assessment). Where the mineralized material is somewhat steeper, and/or narrower, both height and width may be reduced to 4 m by 5 m for example, in order to limit dilution.

- To further minimize dilution, "shanty-back" geometry is assumed whereby the corners of the drifts are profiled to follow the waste contact. Dilution within main mineralized material headings of the UZ would be expected to average slightly more than 10%. This compares with Stantec's database for dilution from actual operations with similar geometry. Sidewall slashes could more precisely follow contacts and potentially have less than 10% dilution; hence overall dilution of a nominal 10% is used for the UZ. The LZ has more erratic geometry hence has been assigned 20% dilution.
- Conventional electric hydraulic development drill jumbos will be used for face and slash drilling, and 6 yd^3 class load-haul-dump (LHD) trucks would typically be used to muck faces and load haulage trucks at the cross-cut intersection.
- Assuming there are no undue environmental sensitivities, ammonium nitrate/fuel oil (ANFO) would be used for most explosive charging using a mobile charging vehicle.
- For costing purposes, it is assumed that ground support will be installed in the mineralized material "drifts", consisting of pattern bolting (using a boom style bolter), and installation of steel mesh on the back and on walls to within 1.5 m of the floor.
- The next stage of mining will involve a long-wall slash, as shown in Figure 16.5, using the same development drill jumbo as the primary mineralized material drift. This first pass slash may be supported to enable a second pass slash. Due to the flat dip (flatter than angle of repose), slashes will need to be drilled such that the mineralized material will be ejected into the drift during the blast to the greatest extent practical.
- Once slashing has been completed along the entire length of the primary drift, a barricade will be constructed at the access and the heading backfilled. Backfill options could range from hydraulic to paste type fill, but it is assumed that an intermediate (dense) fill of 75% pulp density would be utilized to obtain desirable flow characteristics in relatively small headings, yet retain some of the operational advantages of true paste fill.

- The total number of on-strike drift/slashes will vary according to the mineralized material width. On some levels, a single pass will mine out the full width. Other areas will require backfill to set-up then drifting alongside the fill will begin anew. The widest areas (notably north end of the 1,645 m and 1,670 m levels of the UZ) will permit multiple drifts LOM concurrently driven parallel nominally 12 m apart. The intermediate panel is subsequently recovered between the two previously backfilled panels using the same methodology.

- Typical level plan for the UZ (1645 Level) is depicted in Figure 16.6 and for the LZ (1345L/1320L), in Figure 16.7. A longitudinal schematic of mine development is shown in Figure 16.8 and isometric views of mine development for the UZ and LZ are depicted in and Figure 16.10.

- While the lenses are generally too steep to permit trackless equipment to advance directly up-dip, a possible variation of this mining method may include driving the headings at some intermediate azimuth (along an apparent dip) rather than along strike at nominal zero gradient. While this may involve more complicated in-stope survey control, and possible additional difficulties ensuring tight backfill, there may be operational advantages in recovering the slashes. This may be further evaluated in a later feasibility phase once drilling has more precisely delineated the mineralized material geometry. This would be considered an optimization exercise; for purposes here, the impact on estimated operating cost per tonne are not likely to be significant.

- Once all available panels are mined/backfilled on a given horizon, the back/roof of the cross cut would be slashed down (nominal 5 m) to provide access to the next horizontal slice.

Figure 16.3 Sections through Upper and Lower Zones (Sections A-A' and B-B')

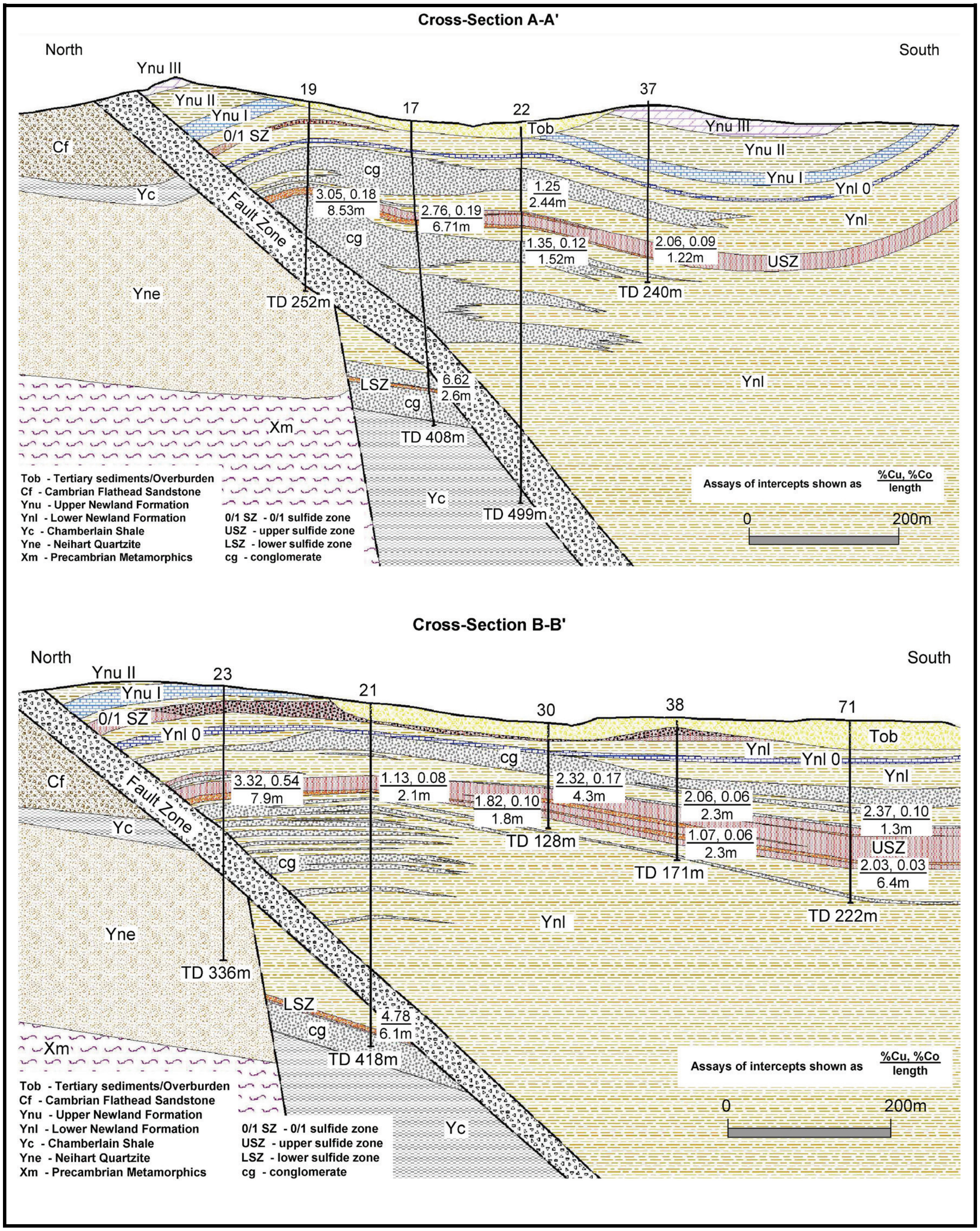


Figure 16.4 Sections through Upper and Lower Zones (Sections C-C' and D-D')

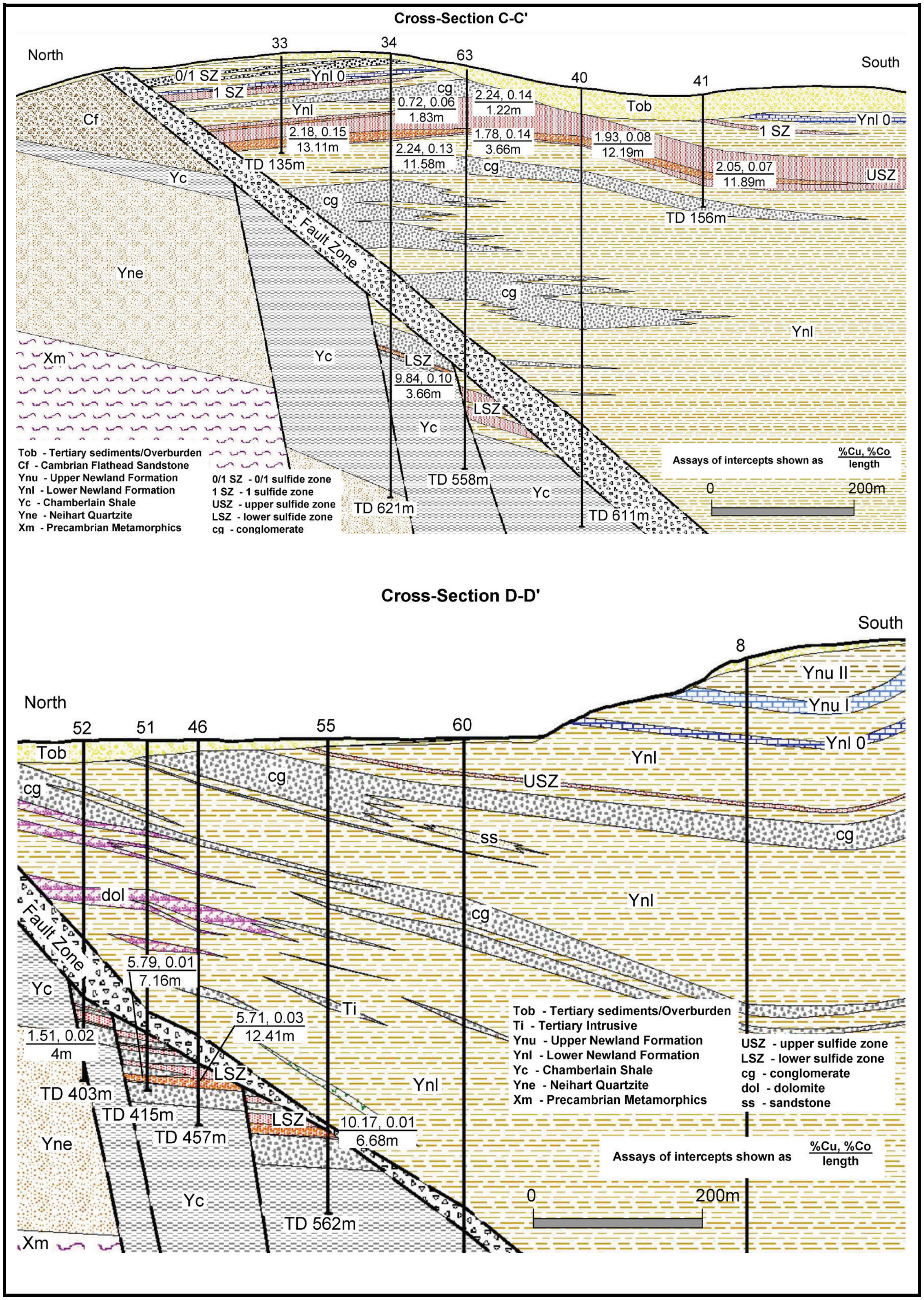


Figure 16.5 Cross Section Illustrating General Mining Concept

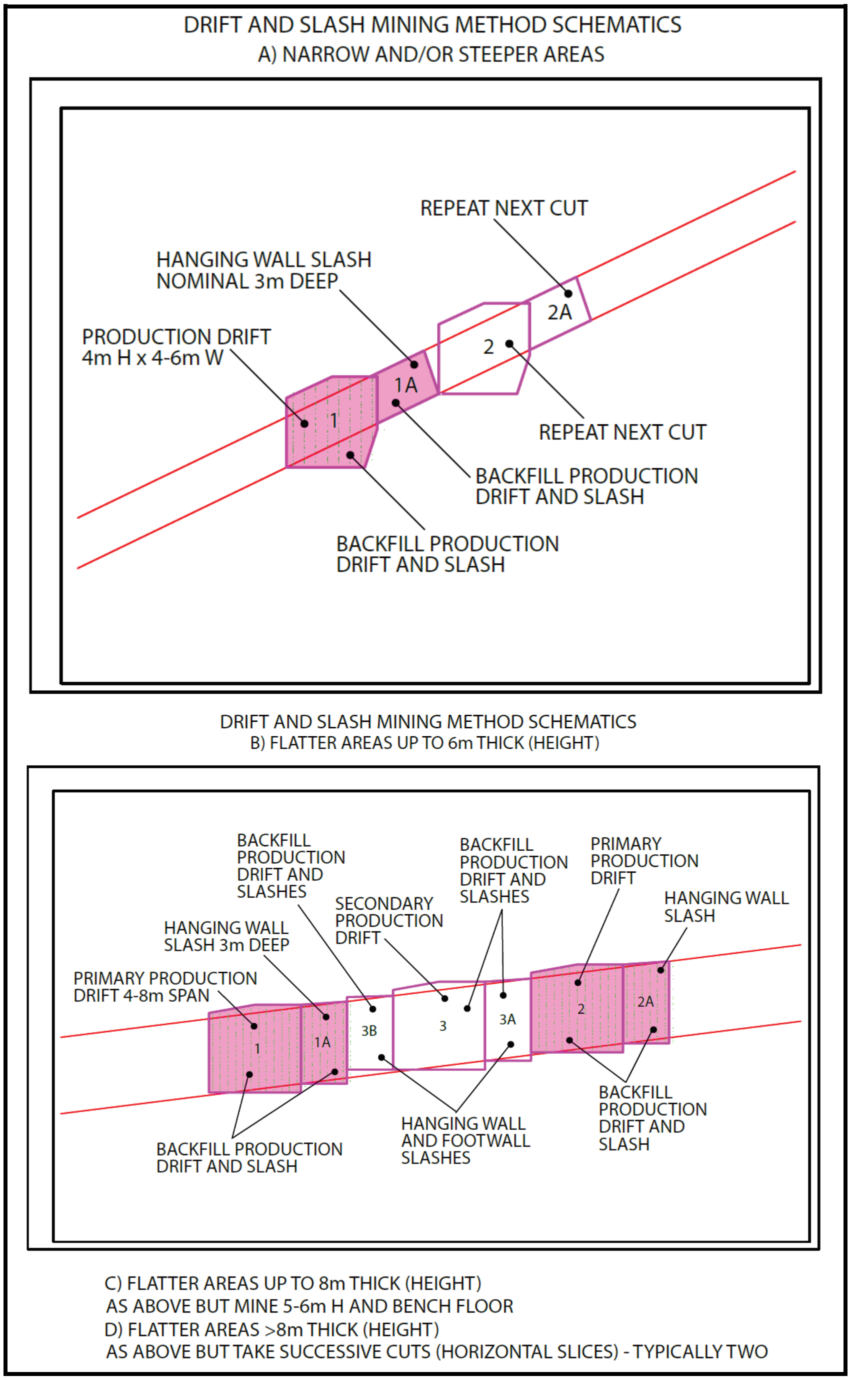


Figure 16.6 Level Plan – UZ 1645 Level

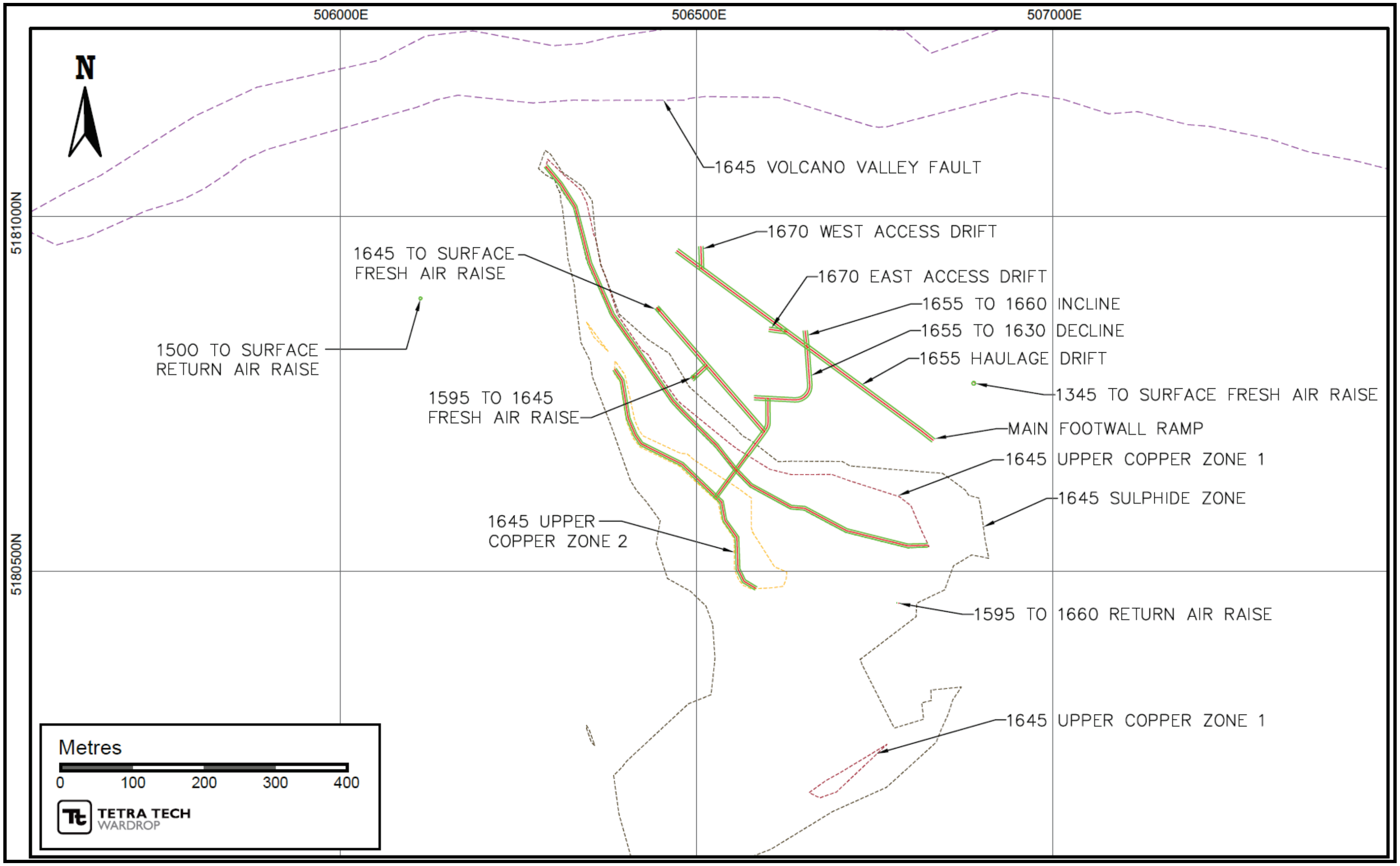


Figure 16.7 Level Plan – LZ 1345 and 1320 Levels

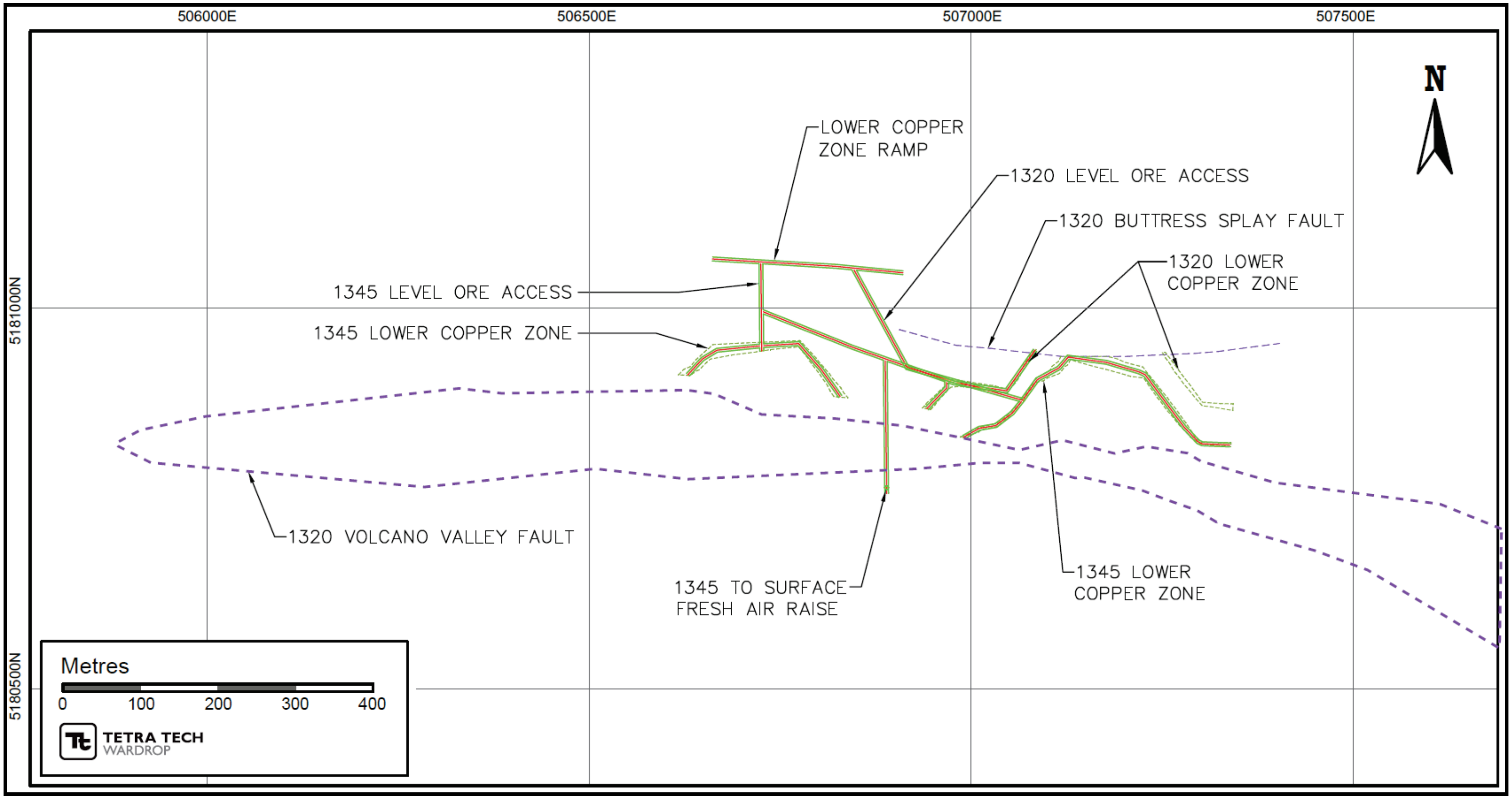


Figure 16.8 Longitudinal Section – Schematic

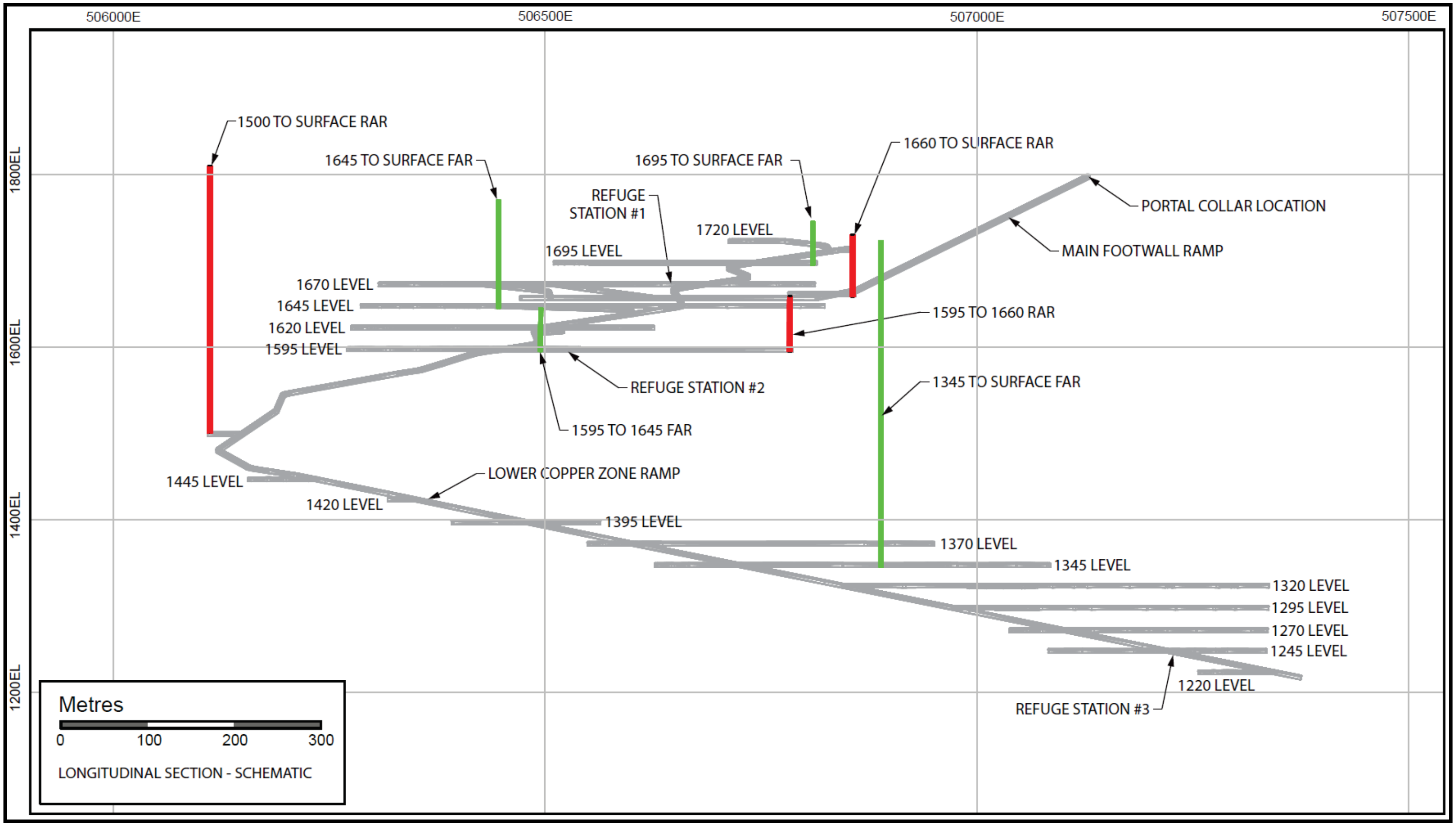


Figure 16.9 Isometric View of LOM Development – Looking West

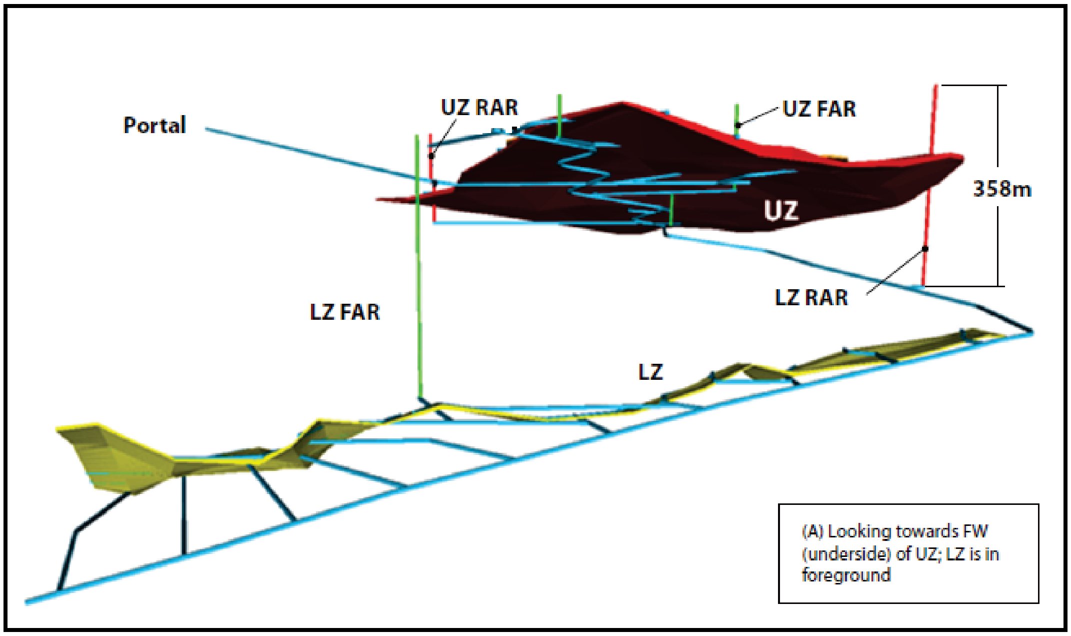
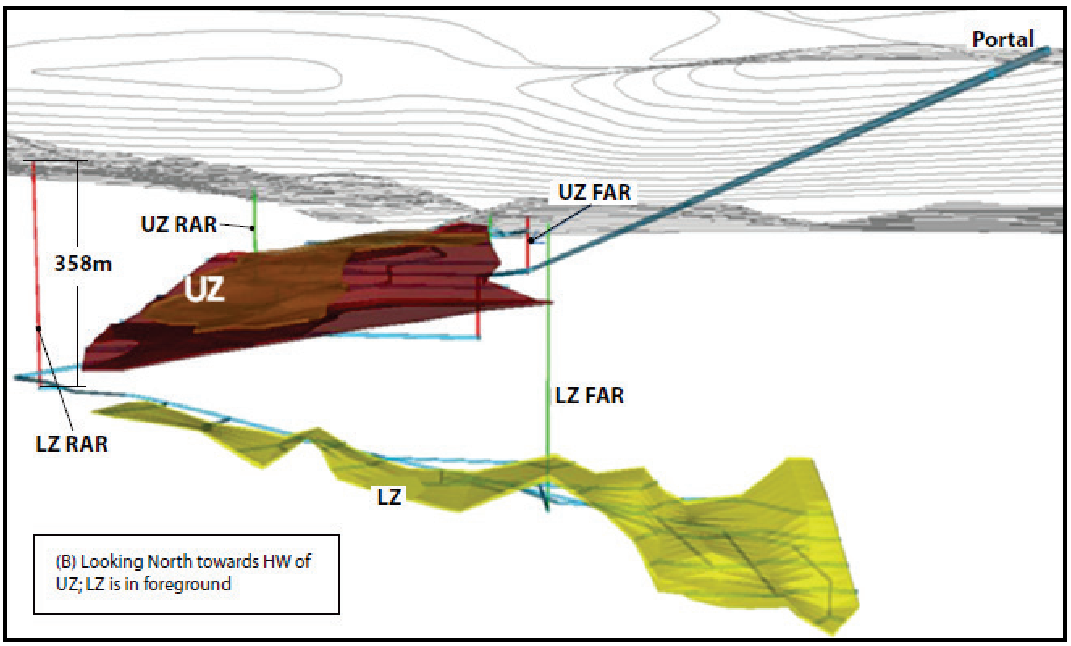


Figure 16.10 Isometric View of LOM Development – Looking North

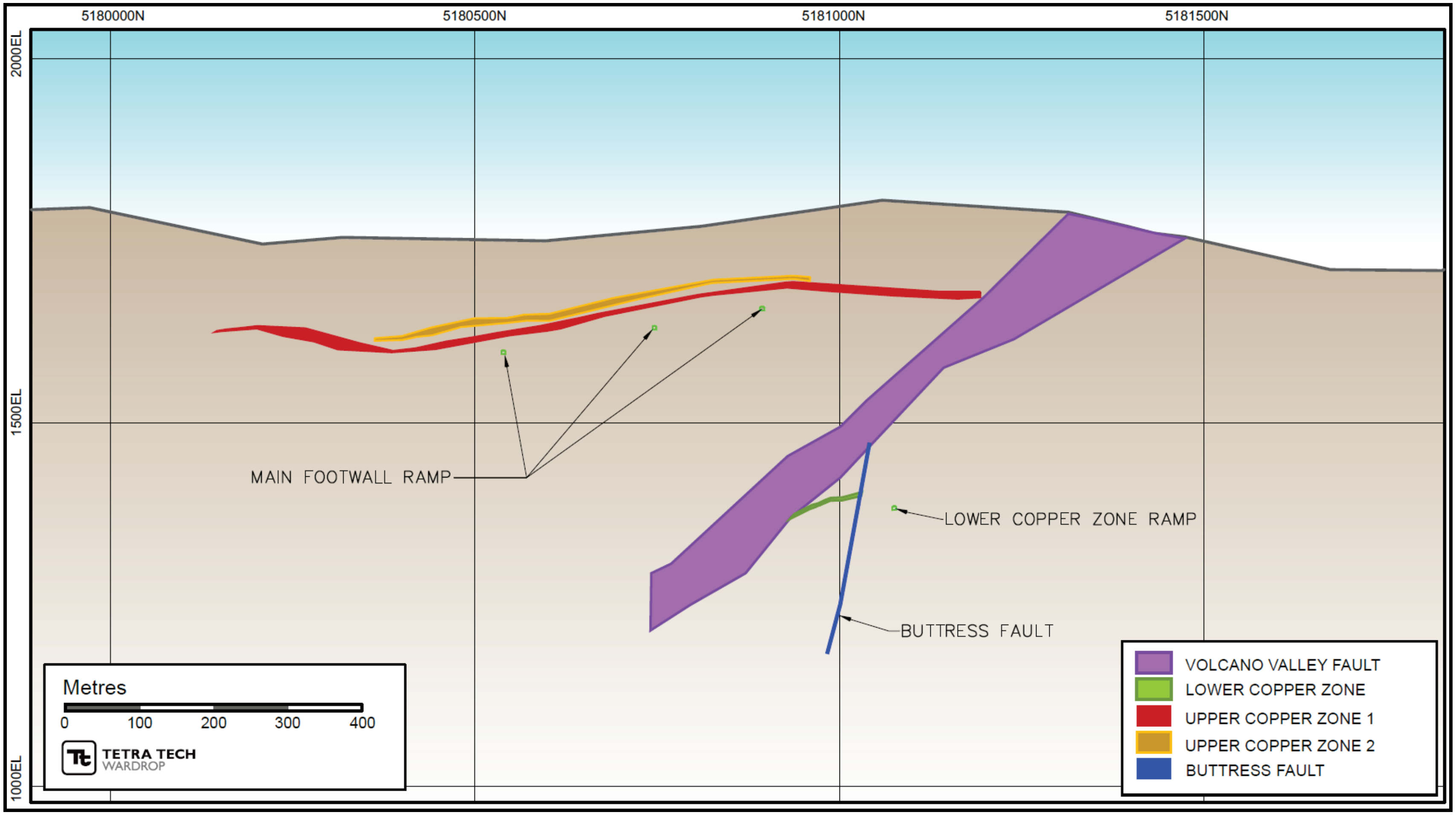


To date, a comprehensive geotechnical/geohydrological assessment has not been conducted. For the UZ, given the relatively shallow depth, high ground stresses are not expected to be a significant concern. Both the mineralized material zone and immediately adjacent wall rocks are massive sulphide, and no major faults or other significant planes of weakness within the mineralized material zone have been identified in the geological model. Drill core recovery is reported to be very good. Rock mass rating is therefore assumed to be generally “good”. The mining method described is exploratory/selective and provides considerable flexibility to adapt the span to variable ground conditions. Generally, the span of individual headings would be 5 to 6 m, but where the mineralized material is particularly flat and wide, an upper limit of an 8 m span is assumed and subject to future stability analyses.

It is presumed that the two lenses (UZ1 and UZ2) can generally be mined as separate headings concurrently at a given horizon (refer to Figure 16.11). Where UZ2 closely approaches intersecting the immediate hanging wall of UZ1, there may be restrictions to sequence UZ1 first to maximize ground stability. The potential to “mass-blast” the UZ1 and UZ2 together along with intervening waste lens does not appear to be economic. The intervening waste lens is typically as thick as, if not thicker than UZ2, therefore blasting together would dilute UZ2 to well below the cut-off grade (even allowing for incrementally lower mining cost per tonne).

The LZ is bounded on the south by the prominent VVF and along the north by a less-prominent Buttress Fault. For purposes of preliminary mine design, the main ramp pierces the VVF approximately perpendicular from south to north and thereafter sub-level access cross-cuts to the LZ are driven from the main ramp from north to south through the Buttress Fault at 25 m vertical intervals. A stand-off distance of nominally 60 m is maintained so the stability of the ramp is unlikely to be impacted by the Buttress Fault. Cross-cuts repeatedly pierce the Buttress Fault oriented at roughly right angles (refer to Figure 16.11). The ventilation raises for the LZ are sited, to avoid intersecting either of the faults: the entire raise length is either completely in the footwall or hanging wall of known significant structures.

Figure 16.11 Mining Cross Section



With a moderate quality backfill, the method is nominally pillar-less. However, some provision may also be made to selectively leave island pillars by not slashing along the full length of the mineralized material drift, in the event that ground conditions warrant.

Other than the opportunity to place development waste as rockfill into stopes, it is assumed that the bulk of backfill will be derived from mill tailings.

Crushing the waste rock on surface for inclusion within the backfill matrix may be a further option to dispose of the waste rock underground. This could be the subject of future trade-off analysis.

Provision has not been included in the cost estimate to haul a portion of backfill material from sources outside the Property. This is predicated on mill tailings having suitable physical characteristics (particle size distribution and rheology for paste or dense hydraulic fill) and suitable mineralogical characteristics (sulphide content does not create self-heating conditions nor degrade binder strength excessively). Once bench scale metallurgical tests have produced a quantity of sample tailings, a test program should be undertaken to confirm characteristics for backfill.

16.3 PRODUCTION RATE ESTIMATE

For scoping study purposes, an initial estimate of the production rate can be approximated by various rules of thumb. A reasonably reliable rule which relates production rate to size of resource is (Taylor 1986):

$$\textit{tonnes per year } (\pm 20\%) = 5 \times (\textit{resource tonnes})^{(3/4)}$$

This yields a median production capacity of approximately 3,300 t/d (±20%) based on a 12.2 Mt resource. This has been used as the initial, base case production rate, which will yield a LOM of nominally 14 years. While not considered “optimized”, there is good confidence in sustaining that production rate.

For drift/slash methodology, typical headings in mineralized material will yield 400 to 500 t per blast.

The expectation at this time is to derive approximately 75% of production (2,500 t/d) from the UZ and the remaining 800 t/d from the LZ. To achieve 2,500 t/d as the longer term steady-state rate from the UZ will require a shorter-term capacity of perhaps 20 to 25% greater than that to allow for variability in headings available, backfill cycle, and other variations in process. Therefore a capacity to blast eight headings per day will be required (four per shift). For efficient utilization of equipment and workforce, for each face charged and blasted it is desirable to have two other faces available for other mining cycle activities (mucking, installing ground support, drilling face, installing services, backfilling). This translates to nominally 12 active headings per shift. For the mine layout envisioned, maintaining at least four active mining horizons within the UZ concurrently with each providing two to four headings available would thus support the 2,500 t/d production rate. The LZ has a shorter strike length and more erratic dip and is thus scheduled for 800 t/d.

Although zone geometry is not amenable to blasthole/bulk mining methods, there are several factors which contribute towards the robustness of 3,300 t/d production rate:

- Relatively high tonnes per vertical metre of resource available, especially in the shallower UZ, requiring relatively low mine development per tonne of mineralized material.
- A very high specific gravity; for given labour and material inputs, the high density mineralized material yields over 35% more tonnes than a comparable heading in 2.8 specific gravity material.
- Relatively shallow depth for majority of the resource.
- Relatively simple lens geometry enabling multiple faces available and multiple mining horizons available concurrently.
- Simple mine infrastructure (no shaft).

An economic trade-off of higher capital expenditure versus a potentially higher production rate and optimization of production rate is recommended for future study phase work.

16.3.1 Estimation of Tonnes and Grades

The resources used in this assessment are exclusively from the electronic block model provided by Tintina in January 2012. Geological background and descriptions were also provided in an earlier report entitled “Sheep Creek Project – Upper Copper Zone Inferred Resource”, prepared by RMI, dated December 20, 2010.

Only the UZ and LZ are considered in this preliminary assessment. Further drilling results since that time are not considered. Datamine based files are used directly in Mine 2-4D mine planning software.

Preliminary cost estimates suggest a break-even cut-off grade of approximately 1.60% copper is “reasonable”. This corresponds with the resource estimate of 12.53 Mt grading 3.26% copper, 1,060 ppm cobalt, and 14.2 ppm silver used here for preliminary mine planning purposes, which is derived from Indicated and Inferred Resources as shown in Table 16.1.

Table 16.1 Indicated and Inferred Resources Used for Mine Planning

	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)
Indicated Resources					
UZ	8,482,855	2.96	1,229	0.008	16.9
LZ	0	-	-	-	-
UZ + LZ	8,482,855	2.96	1,229	0.008	16.9
Inferred Resources					
UZ	1,256,946	2.64	1,037	0.008	16.4
LZ	2,793,266	4.44	557	0.346	5.1
UZ + LZ	4,050,212	3.88	706	0.241	8.59

The tonnes and grades listed exclude allowances for mining recovery and dilution.

An estimate of tonnes and grade are provided in Table 16.2 with dilution and recovery factors applied. Since Inferred resource category material is included and based on the preliminary nature of the cost estimates, the resources are suitable for evaluation at the PEA level only. As such, the estimated tonnes and grade do not represent official "Reserves" that meet NI 43-101 standards.

For the proposed selective drift-and-fill mining method, 90% mining recovery is considered reasonable for the UZ, with 10% mining dilution. Immediate wall rocks are typically pyritic sulphides; for the purposes of estimating after dilution mineralized material grade, information provided by Tintina indicates that the immediate wall rocks are taken at a grade of 1.2% copper. The geometry of the LZ is more variable and confidence in geological information is somewhat less, hence 75% recovery factor and 20% mining dilution is applied to the LZ; grade of immediate diluting wall rocks is taken as 1.0% copper.

For the purposes of this PEA, the portion of the mineral resource is estimated at 12.16 Mt grading 2.99% copper, 965 ppm cobalt, and 13 ppm silver, after applying mining recovery and dilution factors. The amount of resource available is relatively sensitive to the cut-off grade. There may be opportunity to optimize cut-off grade once greater confidence is established for mining costs, other economic parameters (including metallurgical recovery), and for the mineralized material body itself.

Mining horizons have been proposed at 25 m vertical intervals. The level designations are approximate only, but represent approximate elevations in metres above sea level. Therefore, 1720 Level is the uppermost level, and 1595 Level is the deepest level for the UZ. The LZ extends from approximately 1445 Level (uppermost) to 1220 Level (deepest). For reference, the elevation of the portal collar is approximately 1,800 m.

Figure 16.12 graphically depicts the distribution of contained tonnes of copper metal by level. The largest proportion represented by the three biggest levels – 1670, 1645 and 1620, with 1620 Level being the largest contributor.

Figure 16.12 Distribution of Estimated Copper Metal (Tonnes)

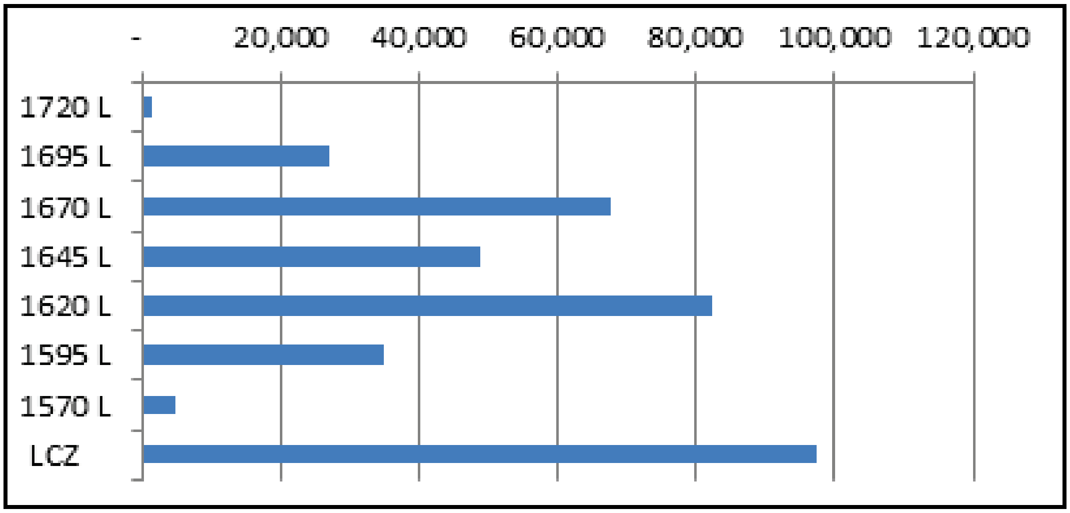


Table 16.2 Distribution of Estimated Tonnes and Grade

Level	Cut-off	In Situ					Dilution – 0.1					Recovered – 0.9				
		Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)
UZ																
1720	1.60	55,227	2.73	937	0.010	5.2	5,523	1.20	-	-	-	54,675	2.59	851	0.009	4.7
1695	1.60	988,747	2.92	1,313	0.013	12.0	98,875	1.20	-	-	-	978,859	2.76	1,193	0.012	10.9
1670	1.60	2,468,692	2.92	1,393	0.011	11.4	246,869	1.20	-	-	-	2,444,005	2.76	1,266	0.010	10.4
1645	1.60	1,683,731	3.10	1,586	0.005	16.8	168,373	1.20	-	-	-	1,666,893	2.93	1,442	0.005	15.3
1620	1.60	2,953,149	2.98	1,055	0.006	19.4	295,315	1.20	-	-	-	2,923,618	2.82	959	0.006	17.7
1595	1.60	1,389,946	2.66	769	0.007	23.8	138,995	1.20	-	-	-	1,376,046	2.53	699	0.006	21.6
1570	1.60	200,309	2.38	454	0.009	25.8	20,031	1.20	-	-	-	198,306	2.77	412	0.008	23.4
Total	**1.60**	**9,739,801**	**2.92**	**1,205**	**0.008**	**16.9**	**973,980**	**1.20**	**-**	**-**	**-**	**9,642,403**	**2.76**	**1,095**	**0.007**	**15.3**

Level	Cut-off	In Situ					Dilution – 0.2					Recovered – 0.75				
		Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)
LZ																
Total	**1.60**	**2,793,266**	**4.44**	**557**	**0.346**	**5.1**	**558,653.20**	**1.00**	**-**	**-**	**-**	**2,513,939**	**3.87**	**464**	**0.288**	**4.2**

Level	Cut-off	In Situ					Dilution					Recovered				
		Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)
Combined																
Total	**1.60**	**12,533,067**	**3.26**	**1,060**	**0.083**	**14.2**	**1,532,633**	**1.13**	**-**	**-**	**-**	**12,156,342**	**2.99**	**965**	**0.07**	**13.0**

16.4 LIFE-OF-MINE SCHEDULES

16.4.1 MINE DEVELOPMENT

The mine project is defined as starting (Year 1) when the mine contractor has (re)mobilized to site at some time after the mine portal has been collared and first 900 m of decline driven during the "Underground Exploration Program". An allowance for a portal cover (culvert) is included. It assumed that other project activities will have started, including surface preparation of roads and at least temporary, if not permanent facilities (camp/dry facilities, power connection, fuel storage etc.) that are required by the mine contractor.

During critical single heading development of the decline, the face advance is scheduled at 4.5 m/d. Given that the Underground Exploration Program will already have driven the initial 900 m of main ramp, it is expected that the construction of the concentrator and other surface infrastructure will be the "critical path" for the overall project schedule.

In that circumstance, the mine plan is to focus development of three additional production horizons and supporting ventilation raises during the next 15 months so that the ramp-up to full production is as compressed as possible. This will also avoid the requirement to prepare a large surface mineralized material stockpile/ containment area in advance of commissioning the concentrator. The total LOM development amounts to approximately 7.2 km of lateral development (excluding ramp development during exploration) and 1,230 m of vertical (bored raise) development as outlined in Table 16.3. Production can be derived from the UZ for the first two years while the ramp continues towards the LZ. While the LZ is at greater depth and has more erratic geometry, the average grade is higher than the UZ.

Table 16.3 LOM Development Summary

	LOM	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Total Lateral Development (m)	7,189	2,790	2,842	841	296	421	0	0
Total Vertical Development (m)	1,230	247	473	100	410	0	0	0
Estimated Development Waste Tonnes ('000)	738	276	291	84	47	40	0	0

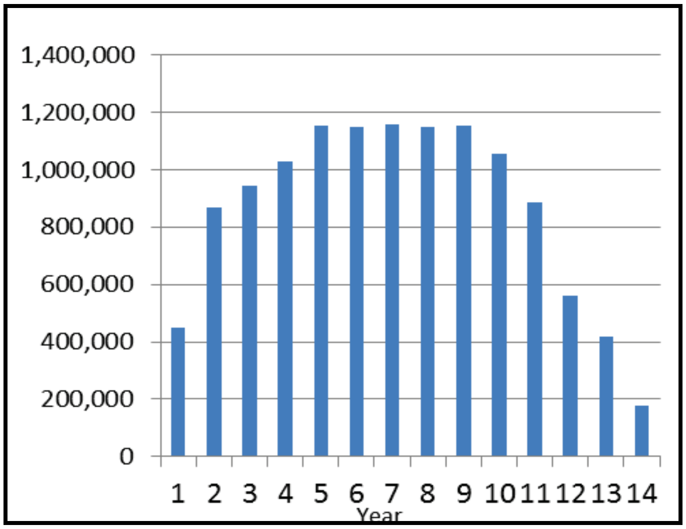
16.4.2 MINE PRODUCTION

Production will start by the second quarter of Year 2 of the mine project. Initial production scheduled from each set of horizons is initially limited to 500 t/d for several months to recognize the need to establish sufficient active faces in various stages of the mining cycle to achieve full efficiency. The production rate is thereafter scheduled at 800 t/d per set of headings.

The mineralized material grade profile assumes that higher-than-average grade areas on given mining horizons are selectively targeted as much as practical in earlier years. While this strategy is not sustained beyond the first couple of years, it enhances early cash flow. Then, as mineralized material grade from the UZ settles back to more average grade, the LZ begins contributing higher grade mineralized material.

Figure 16.13 depicts the production profile at a steady state rate of nominally 1,150,000 t/a.

Figure 16.13 LOM Production (Tonnes)



It is favourable to project economics to access higher grade areas earlier in the mine life. Since the four levels—1620 through 1695—within the UZ are the higher grade horizons, earliest in the production sequence and most heavily scheduled, by default, the mineralized material grade of early production will be somewhat higher than the overall average for UZ. However, by specifically targeting higher-grade areas within these levels, there may be opportunity to increase the grade further in first two to three years of production. Select areas of the UZ1 tend to have higher grade such as the north end of 1670 Level and south end of 1645 Level as shown in Figure 16.14 and Figure 16.15, respectively.

Figure 16.14 Illustration of High-Grade Distribution: 1670 Level

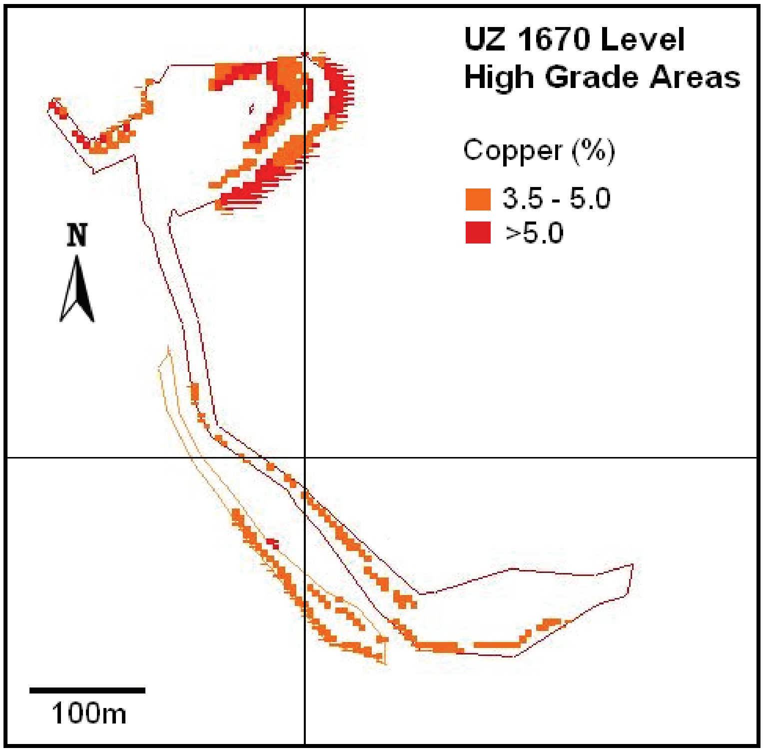
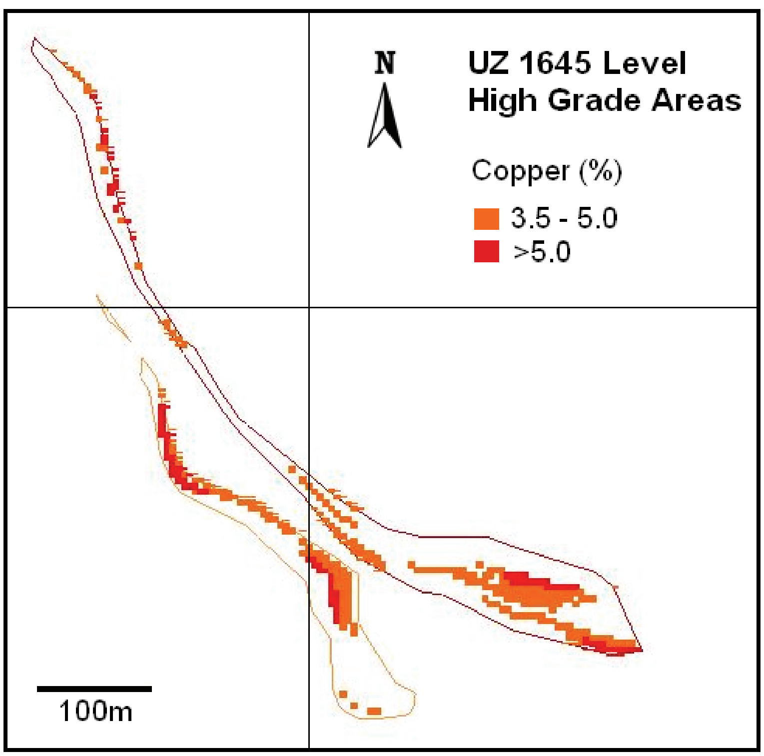
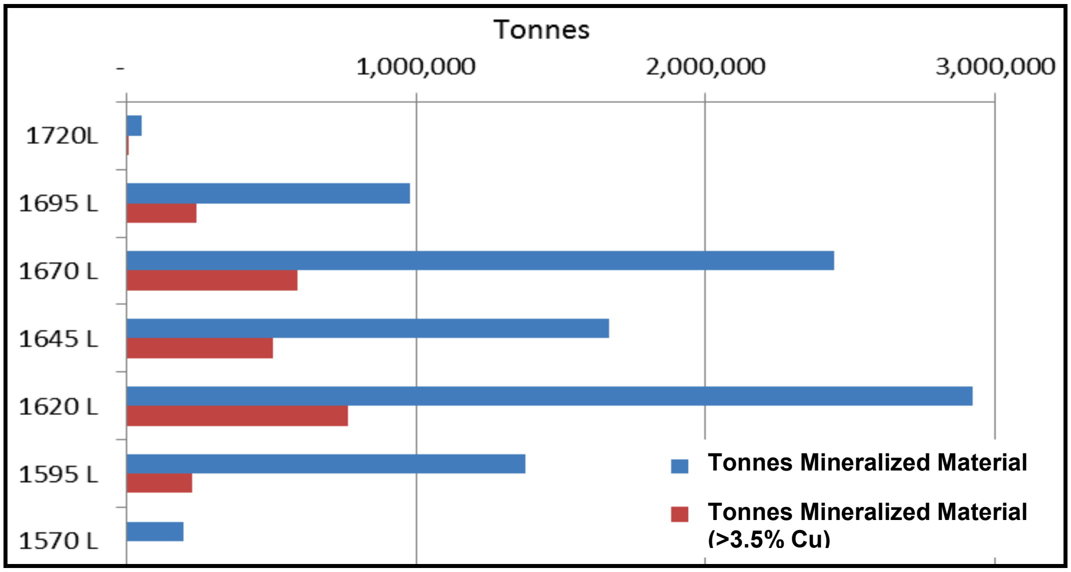


Figure 16.15 Illustration of High-Grade Distribution: 1645 Level



Approximately 26% of the tonnage on these levels is considered “high-grade”; this is determined by modifying the cut-off grade to 3.5% copper (refer to Table 16.4) to define the proportion that is considered high grade. The average of the high grade areas within UZ occurs within 1620 through 1670 Levels yield a grade of 4.4% copper grade after dilution (versus 2.76% copper grade for UZ overall). Figure 16.16 illustrates that most of the mineralized material grading over 3.5% copper is located on 1620 through 1670 Levels.

Figure 16.16 UZ Distribution of Total and High Grade Mineralized Material by Level



Based on the drill density to-date, the distribution of these higher-grade areas within individual levels is not known with precision at this time. Furthermore, the nature of drift-and-fill mining will to a large extent require mining through lower and average grade material to access higher grade material. It is assumed that there may be opportunity to selectively target higher copper grade areas such that the proportion of high-grade can be doubled for a two year period compared to the naturally-occurring proportion. This will equate to approximately 9% increase versus average grade for those levels as highlighted in Table 16.5. One consequence for mining early high grade is that slightly below-average copper grade will remain in subsequent years. The feasibility of high grade selection in early years would need to be further assessed once further drilling of the mineralized material body has increased confidence in grade distribution on individual levels.

The other element of the strategy to enhance early grade is to continue development to LZ without delay as it has an average grade more than 3% copper.

Table 16.4 High Grade Resource – UZ

Level	Cut-off	In Situ					Dilution – 0.1					Recovered – 0.9					Portion High Grade (%)
		Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	Tonnes	Cu (%)	Co (g/t)	Au (g/t)	Ag (g/t)	
1720	3.50	7,330	4.04	984	0.010	2.56	733	1.20	-	-	-	7,257	3.79	895	0.009	11.87	13
1695	3.50	246,860	4.75	1,931	0.009	13.06	24,686	1.20	-	-	-	244,391	4.43	1,756	0.008	2.32	25
1670	3.50	598,833	4.84	1,896	0.010	9.92	59,883	1.20	-	-	-	592,844	4.51	1,723	0.009	9.02	24
1645	3.50	512,534	4.60	1,934	0.006	16.94	51,253	1.20	-	-	-	507,409	4.29	1,758	0.006	15.40	30
1620	3.50	775,983	4.80	1,320	0.007	18.58	77,598	1.20	-	-	-	768,224	4.47	1,200	0.006	16.89	26
1595	3.50	230,217	4.68	1,007	0.005	23.31	23,022	1.20	-	-	-	227,915	4.36	9,16	0.005	21.19	17
1570	3.50	585	3.55	1,565	0.004	41.88	58	1.20	-	-	-	579	3.33	1,423	0.004	38.08	0
Total	**3.50**	**2,372,342**	**4.75**	**1,630**	**0.008**	**15.88**	**237,234**	**1.20**	**-**	**-**	**-**	**2,348,619**	**4.43**	**1,482**	**0.007**	**14.44**	**24**

Table 16.5 LOM Production Profile (with Target Early High Grade)

Total Mineralized Material Production	Value	Year														
		1	2	3**	4	5	6	7	8	9	10	11	12	13	14	15
1720 Level Total	**54,000**	-	-	-	20,000	20,000	14,000	-	-	-	-	-	-	-	-	-
Cu (%)	**2.59**	-	-	-	2.59	2.59	2.59	-	-	-	-	-	-	-	-	-
Co (ppm)	**851**	-	-	-	851	851	851	-	-	-	-	-	-	-	-	-
Ag (g/t)	**4.7**	-	-	-	4.7	4.7	4.7	-	-	-	-	-	-	-	-	-
1695 Level Total	**979,000**	-	70,000	175,000	175,000	175,000	175,000	175,000	34,000	-	-	-	-	-	-	-
Cu (%)	**2.76**	-	3.10	2.77	2.95	2.42	2.76	2.76	2.76	-	-	-	-	-	-	-
Co (ppm)	**1,193**	-	1,193	1,193	1,193	1,193	1,193	1,193	1,193	-	-	-	-	-	-	-
Ag (g/t)	**10.9**	-	10.9	10.9	10.9	10.9	10.9	10.9	10.9	-	-	-	-	-	-	-
1670 Level Total	**2,444,000**	-	150,000	260,000	260,000	260,000	260,000	260,000	260,000	260,000	240,000	134,000	100,000	-	-	-
Cu (%)	**2.76**	-	3.10	2.95	2.90	2.44	2.65	2.74	2.74	2.74	2.74	2.74	2.74	-	-	-
Co (ppm)	**1,266**	-	1,266	1,266	1,266	1,266	1,266	1,266	1,266	1,266	1,266	1,266	1,266	-	-	-
Ag (g/t)	**10.4**	-	10.4	10.4	10.4	10.4	10.4	10.4	10.4	10.4	10.4	10.4	10.4	-	-	-
1645 Level Total	**1,667,000**	-	130,000	175,000	175,000	175,000	175,000	175,000	175,000	175,000	175,000	137,000	-	-	-	-
Cu (%)	**2.93**	-	3.29	2.81	3.40	2.40	2.92	2.92	2.92	2.91	2.91	2.91	-	-	-	-
Co (ppm)	**1,442**	-	1,442	1,442	1,442	1,442	1,442	1,442	1,442	1,442	1,442	1,442	-	-	-	-
Ag (g/t)	**15.3**	-	15.3	15.3	15.3	15.3	15.3	15.3	15.3	15.3	15.3	15.3	-	-	-	-
1620 Level Total	**2,924,000**	-	100,000	260,000	260,000	260,000	260,000	260,000	260,000	260,000	260,000	260,000	260,000	150,000	74,000	-
Cu (%)	**2.82**	-	3.16	3.16	2.38	2.82	2.82	2.82	2.81	2.81	2.82	2.82	2.82	2.81	2.81	-
Co (ppm)	**959**	-	959	959	959	959	959	959	959	959	959	959	959	959	959	-
Ag (g/t)	**17.7**	-	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	-
1595 Level Total	**1,376,000**	-	-	-	-	-	-	-	151,000	175,000	175,000	175,000	175,000	175,000	175,000	175,000
Cu (%)	**2.53**	-	-	-	-	-	-	-	2.53	2.53	2.53	2.53	2.53	2.53	2.53	2.53
Co (ppm)	**699**	-	-	-	-	-	-	-	699	699	699	699	699	699	699	699
Ag (g/t)	**21.6**	-	-	-	-	-	-	-	21.6	21.6	21.6	21.6	21.6	21.6	21.6	21.6

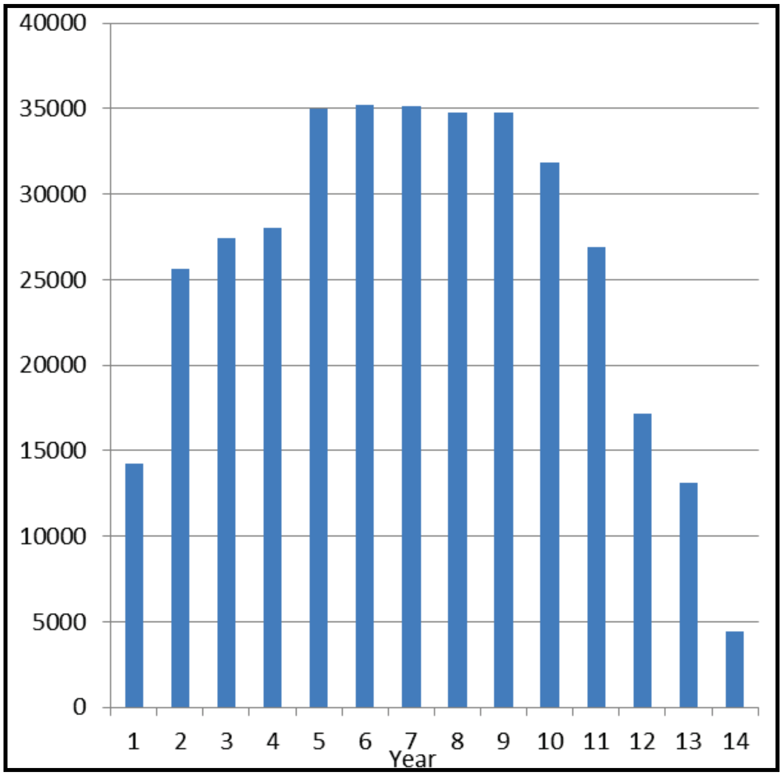
table continues…

Total Mineralized Material Production	Value	Year														
		1	2	3**	4	5	6	7	8	9	10	11	12	13	14	15
1570 Level Total	198,000	-	-	-	-	-	-	-	-	-	22,000	71,000	70,000	35,000	-	-
Cu (%)	2.27	-	-	-	-	-	-	-	-	-	2.27	2.27	2.27	2.27	-	-
Co (ppm)	412	-	-	-	-	-	-	-	-	-	412	412	412	412	-	-
Ag (g/t)	23.4	-	-	-	-	-	-	-	-	-	23.4	23.4	23.4	23.4	-	-
Subtotal UZ	9,642,000	-	450,000	870,000	890,000	890,000	884,000	870,000	880,000	870,000	872,000	777,000	605,000	360,000	249,000	175,000
Cu (%)	2.76	-	3.17	2.95	2.85	2.54	2.77	2.80	2.76	2.75	2.74	2.71	2.66	2.62	2.61	2.53
Co (ppm)	1,095	-	1,237	1,195	1,187	1,187	1,190	1,195	1,110	1,096	1,074	989	871	779	776	699
Ag (g/t)	15.3	-	13.5	13.7	13.5	13.5	13.5	13.7	15.5	15.8	16.1	17.4	18.3	20.1	20.4	21.6
t/d	-	-	1,286	2,486	2,543	2,543	2,526	2,486	2,514	2,486	2,491	2,220	1,729	1,029	711	500
Subtotal LZ	2,514,000	-	-	-	54,000	140,000	270,000	280,000	280,000	280,000	280,000	280,000	280,000	200,000	170,000	-
Cu (%)	3.87	-	-	-	3.87	3.87	3.87	3.87	3.87	3.87	3.87	3.87	3.87	3.87	3.87	-
Co (ppm)	464	-	-	-	464	464	464	464	464	464	464	464	464	464	464	-
Ag (g/t)	4.2	-	-	-	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	-
t/d	-	-	-	-	154	400	771	800	800	800	800	800	800	571	486	-
Total Mineralized Material Production (t)	12,156,000	-	450,000	870,000	944,000	1,030,000	1,154,000	1,150,000	1,160,000	1,150,000	1,152,000	1,057,000	885,000	560,000	419,000	175,000
t/d	-	-	1,286	2,486	2,697	2,943	3,297	3,286	3,314	3,286	3,291	3,020	2,529	1,600	1,197	500
Cu (%)	2.99	-	3.17	2.95	2.91	2.72	3.03	3.06	3.03	3.02	3.02	3.01	3.04	3.07	3.12	2.53
Co (ppm)	965	-	1,237	1,195	1,146	1,089	1,020	1,017	954	942	926	850	743	667	650	699
Ag (g/t)	13.0	-	13.5	13.7	12.9	12.2	11.3	11.4	12.7	13.0	13.2	13.9	13.8	14.5	13.9	21.6

The benefit of efforts to selectively mine higher grade material rather than average grade material from UZ in the first three years could bring forward 5,000 t of contained copper in the mine life schedule – with over US$30 million gross value.

Peak contained copper production of 35,000 t copper-in-mineralized material is sustained through Year 10 (refer to Figure 16.17).

Figure 16.17 Production Profile – Annual Tonnes Contained Copper



Note: Based on 3,300 t/d with selective higher grade in early years

16.4.3 MINE EQUIPMENT/FLEET

Assumptions on mine design and equipment fleet are predicated on the mine being suitably efficient and mechanized. It is assumed that hand-held (jackleg and stoper) drills would only be used occasionally for specialized purposes.

A leaky feeder radio communication system is included in the mine infrastructure, which will enable efficient dispatch of fleet and effective traffic control.

The fleet planned is considered fit-for-purpose without inclusion of highly advanced technologies (automation/computerized drilling/laser LHD guidance, etc.). Certain features may in fact have some application, but for preliminary analysis purposes, identification and economic justification is considered a future optimization exercise.

Table 16.6 summarizes the underground mobile fleet (including allowance for peak capacity). The equipment schedule allows for leasing primary equipment and outright purchase of ancillary vehicles during pre-production or capital period, and subsequently allows for further leases, and purchases equipment replacements as “sustaining capital” during the LOM. Sustaining capital for mobile equipment is factored for estimation purposes, but is based on most units receiving a major overhaul and/or replaced during the mine life. Higher utilized prime movers would typically be replaced two to three times over a 14-year mine life. Drills and other utility equipment typically are replaced once.

The opportunity to better optimize the mining fleet will be more feasible once the distribution of heading sizes becomes known with greater precision and confidence.

The mucking distances from the development face to re-muck bays or from the mineralized material face to truck loading areas will typically be less than 300 m. For the given development and production heading sizes, LHDs in the 6 to 8 yd^3 class are deemed suitable. The very high density of the mineralized material and sulphide waste will be significant in the required specifications. For example, it may be appropriate to equip a 6 yd^3 bucket on an otherwise 8 yd^3 chassis. Some mineralized material headings may vary in width according to local thickness and dip such that a mixed fleet of 6 yd^3 (or smaller) and some 8 yd^3 units may be considered. The operational advantages provided by multiple models would need to be compared to the advantages of standardization of parts inventories and maintenance skills required.

Ventilation requirements also need to be considered. The Mine Safety and Health Administration (MSHA)’s stringent limits on diesel particulates are specific to individual engine models: A number of smaller 6 yd^3 class units actually require higher ventilation than some 8 yd^3 class units. For cost estimate purposes, unit costs for 6 yd^3 LHDs are provided.

For the haulage requirements trucks in the 40 to 50 t class would be suitable (typically 1.5 km one-way from UZ at +15% gradient and 3 km from LZ).

For the heading sizes and methodology the jumbo drill fleet used in development headings are assumed to be standardized with units for production drifts; all two-boom electric-hydraulic drill jumbos with capacity to drill headings up to 6 m height by 6 m width.

Jacklegs/stopers/scissor trucks will not be used as primary means of installing ground support. The base case assumption is to utilize “boom style” mechanized bolters to pattern bolt/rebar headings up to 6 m high, install bolts in walls to within 1.5 m of the floor, and reach into slashed areas to install bolts.

There is no indication of particularly wet conditions underground, nor any other circumstances regularly requiring specialized explosives. It is assumed the majority of charging will be completed using ANFO; provision is made in the fleet for conventional mobile ANFO charging vehicles.

Table 16.6 Mine Equipment Fleet

Description	Year													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Lease Count														
Drill Jumbo	2	3	4	4	4	4	4	4	4	4	4	4	4	2
Bolter	2	4	5	5	5	5	5	5	5	5	5	5	5	3
LHD	3	4	5	5	5	5	5	5	5	5	5	5	5	4
Underground Haul Truck	2	2	3	4	4	5	5	5	5	5	5	4	3	2
Purchase Count														
Scissor Lift	2	2	3	3	3	3	3	3	3	3	3	3	3	2
Boom Truck	1	1	2	2	2	2	2	2	2	2	2	2	2	1
Explosive Loader	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Utility Vehicle/Man Carrier	4	5	6	6	6	6	6	6	6	6	6	6	6	5

16.4.4 *Mine Workforce*

The mine workforce has been estimated to include the functions of development, drilling, blasting, ground support, production mucking and trucking, backfilling, maintenance, technical services, and supervision personnel.

Whether the workforce would commute to centres within Montana, require accommodation facilities on site, or some combination, is outside the scope of the mine design and estimate. Provisions are not included in the estimated mine workforce for personnel related to providing room-and-board facilities or related services. However, the total number of mine personnel on payroll and number required on-site reflect the assumption that most mine production and maintenance functions would be on nominal 11 hour shifts, 7-days on/7-days off rotation or equivalent while most staff functions would be on a 5 d/wk work schedule.

Although the mining method will require relatively high skill sets (particularly jumbo operators and maintenance personnel for electric-hydraulic equipment), the labour availability in Montana is generally considered fair to good for such skills.

It is assumed that pre-production mine development and construction would be completed by experienced contractors and then as the mine enters the production phase, Tintina employees would have been suitably trained to take over.

For many underground mine projects in North America, availability of sufficiently experienced/skilled mine development contractors currently represents a risk to achieving the mine development schedule in a timely and cost effective manner.

Although the Project would not be a particularly development-intensive project, the ongoing production method will rely on development type skill sets. The mining labour market would need to be reassessed as the Project approaches the execution phase (or a mitigation plan brought forward).

Table 16.7 summarizes Tintina's and the Contractor's workforce and labour costs for direct mine personnel. These exclude maintenance personnel deemed part of surface G&A costs since the maintenance shop is on surface.

Table 16.8 provides build-up for indirect Tintina workforce and labour costs.

Table 16.9 provides build-up for indirect mine Contractors' workforce and labour costs.

Table 16.7 Estimated Contractor and Tintina Direct Mine Workforce and Cost

Description	Total (US$/h)	Wage (US$/h)	Burden 30% (US$)	Bonus (%)	Bonus (US$/h)	Schedule (days)	h/d	Payroll Count
Tintina Direct Labour								
Jumbo Operator – Level 4	52.50	25.00	7.50	80	20.00	7 on, 7 off	10.5	16
Bolter Operator – Level 3	48.30	23.00	6.90	80	18.40	7 on, 7 off	10.5	24
Services – Level 2	37.80	21.00	6.30	50	10.50	7 on, 7 off	10.5	16
LHD Operator – Level 2	37.80	21.00	6.30	50	10.50	7 on, 7 off	10.5	20
Truck Driver – Level 1	30.40	19.00	5.70	30	5.70	7 on, 7 off	10.5	20
Backfill Leader – Level 3	48.30	23.00	6.90	80	18.40	7 on, 7 off	10.5	4
Backfill Miner – Level 1	30.40	19.00	5.70	30	5.70	7 on, 7 off	10.5	12
Total Count								**112**
Contractor Direct Labour								
Jumbo Operator	70.90	-	-	-	-	-	10.5	4
Bolter Operator	69.45	-	-	-	-	-	10.5	4
Services	69.45	-	-	-	-	-	10.5	8
Total Count								**16**

Table 16.8 Build-up of Tintina's Direct Labour Cost Estimate Cost

Description Tintina Indirect Labour	Total (US$/a)	Wage (US$/a)	Burden 30% (US$)	Payroll Count	Comments
General Engineering					
Chief Mine Engineer	130,000	100,000	30,000	1	-
Mine Engineer	104,000	80,000	24,000	2	-
Surveyors	78,000	60,000	18,000	2	-
Technicians	78,000	60,000	18,000	1	-
Chief Mine Geologist	117,000	90,000	27,000	1	-
Geologist	97,000	75,000	22,500	2	-
Underground Geologist	97,500	75,000	22,500	4	-
Administration					
Mine Clerk	46,540	35,800	10,740	1	With surface G&A costs
Mine Supervision and Operating					
Mine Superintendent	156,000	120,000	36,000	1	-
Shift Supervisor	143,000	110,000	33,000	4	-
Trainers	130,000	100,000	30,000	3	-
Maintenance					
Mobile Equipment Mechanics	N/A	N/A	N/A	4	With surface G&A costs
Total Count				**20**	**Subtotal**
Total Indirect Labour Cost per Year				**25**	**Shown with surface G&A costs**

Table 16.9 Build-up of Contractor's Indirect Labour Cost Estimate Cost

Description Contractor's Staff (Indirect Labour)	Rate/Day (US$)	Capital Period
Supervision		
Project Superintendent	775.00	1.0
Rotation Supervisor	725.00	2.0
Safety Supervisor	725.00	1.0
Development Shift Bosses	675.00	-
Maintenance Personnel		
Lead Mechanics	760.00	-
Underground Mobile Equipment Mechanics	750.00	4.0
Welder	700.00	-
Lead Electricians	760.00	1.0
Electricians	750.00	-
Support Personnel		
Surface Operator	641.90	-
Nippers/Material Handling Underground	637.40	-

table continues…

Description Contractor's Staff (Indirect Labour)	Rate/Day (US$)	Capital Period
Grader Operator	701.50	-
Lube Truck Operator	701.50	-
Shotcrete Plant Operator	641.90	-
Transmixer Driver	637.40	-
Spare Miner/Trainee	543.90	-
Administration Personnel		
Project cost Engineer	521.30	-
Warehouse	388.30	-
Clerk/Administrator	388.30	-
Total Contractor Indirect Staff		**9.0**
Average Daily cost per Employee		**$745.56**
Contractor's Indirect Labour – Cost per Day		**$6,710**
Margin and Profit 18%	**18%**	**$1,207.80**
Total Contractor's Indirect Labour – Cost per Day		**$7,918**

16.4.5 MINERALIZED MATERIAL AND WASTE HANDLING

For fleet estimate purposes, primary mucking from underground development and stope workings is assumed using 6 yd^3 LHD units. There are select areas that would also be suitable for 8 yd^3 class (or even larger) units. However, the efficiency gained in some areas versus the additional costs associated with maintaining a mixed fleet would need to be subject to future trade-off analysis.

While developing the main ramp, allowance is included to excavate "remuck" bays every 150 m of ramp length. These will be slashed to enable haulage trucks to turn around, and be driven with sufficient overhead clearance to facilitate LHD units loading the trucks. Once mining operations commence, these will serve further functions, such as passing bays in the ramp, and the majority of remuck bays are also subsequently utilized as electrical stations, material storage areas, intermediate pump stations, etc.

Mineralized material mucked from stope headings will be trammed using LHD units (typical distance approximately 150 m, maximum approximately 300 m) to a truck loading area at each stope cross-cut. The truck loadout will include a temporary re-muck bay driven at the intersection of the cross-cut with the stope block (which may be driven in mineralized material). This will enable LHD units to load directly from the face into a truck whenever a truck is in position, and also continue to muck to a stockpile whenever a truck is in transit.

For fleet estimate purposes, haulage to surface is presumed to be using 40 t class diesel trucks. Alternately 50 t units are also possible, although consideration must be given to maintaining a matching LHD-truck combination. This should be the subject of future optimization using a simulation model.

A further option will be to equip at least part of the truck fleet with ejector boxes. While these will incur a cost premium and have somewhat lower net payload, this would enable trucks to backhaul sulphide development waste rock from surface for use as rockfill in select areas. This would minimize or potentially eliminate the operational issues and costs associated with disposing PAG waste rock on the surface.

Over the entire LOM, the equivalent of 7.2 km of waste mine development will be excavated, including operating development. During pre-production development and the period approaching full steady-state production (through to Year 3), the cumulative waste development amounts to approximately 560,000 t. However by that time, it is expected that a portion of waste remains underground as rockfill or is back-hauled underground as rockfill.

The following three stockpile pads are presumed to be required on surface (to be designed/costed by others):

- Clean Waste Storage: Non-PAG waste (dolomite etc.) hauled to surface would be stored, possibly crushed/screened and subsequently used as construction material (roads, tailings dams, concrete, etc.) for both surface and underground. A surface crusher/stacker/screen plant is not included in the underground mine cost estimate.
- PAG Waste Storage: As mine stope excavations become available, the intent would be to reclaim stockpiled PAG waste and backhaul it underground as operations permit. Material left on surface will be deposited in the TMF.
- Mineralized Material Stockpile: Mineralized material presumed to be PAG/metal leaching. A "working" stockpile or ROM pad of up to one week's production is suggested (approximately 6,000 m^3) to allow the mine to continue operation when there are planned or unplanned downtime in the crushing/milling plants.

A surface loader and operator will be required to reclaim from the stockpiles, feed the mine backfill plant, and as required, load waste rock into trucks for backhaul underground. Further duties may include loading miscellaneous bulk materials such as loading crushed roadbed ballast for delivery underground. These are not included in the underground mine capital cost estimates or mine workforce estimates.

16.4.6 MINE VENTILATION

Airflow requirements were estimated using a variety of approaches:

- overall airflow per annual production rate for based on the mining method
- total airflow per brake horsepower of diesel fleet

- airflow required for diluting diesel particulates to MSHA limits (diesel particulate matter limit of 0.16 mg/m^3).

Opening sizes and fan power requirements were initially modelled using the VnetPC 2007 network modelling software for a peak production rate 2,500 t/d from the UZ.

For a drift-and-fill mining method with a fleet of diesel powered haulage trucks and LHD units, the approximate air volume requirement for the UZ is 380 m^3/s (800,000 cfm), based on utilizing 50% diesel particulate filters (DPF). The openings required to extend the system to the LZ for incremental additional 800 t/d were estimated to add a further 95 m^3/s (200,000 cfm).

The primary ventilation system will be a "positive" or "push" system with fresh air (FA) supplied via two main intake raises or boreholes (initial raise at 3 m diameter enabling development of upper levels and one central raise at 4 m diameter to enable and sustain full production). Return air (RA) will free flow up the main 5 m by 6 m decline to surface and via an additional RA raise to surface so that velocity is maintained at 6 m/sec maximum in the main decline. As development is extended to the LZ, an additional fresh air raise (FAR) will be driven (which will double as the second egress route); exhaust air will free flow up the ramp and exhaust out the return air raise (RAR) near the bottom of the UZ.

Other ventilation dedicated opening sizes are typically based on limiting maximum air velocity to 18.5 m/sec. Optimization of major airway sizes, however, has not been undertaken. Diesel exhaust filtration systems should also be further investigated prior to finalizing the main ventilation system as there may be potential to reduce airflow and thus impact airway sizes as well as mine heating requirements.

Ventilation raises to surface are presumed to be developed as bored raises by contractors. A positive pressure system consisting of main fans installed on the fresh air intakes is presumed; a typical installation would consist of two intake fans operating at 2,000 Pa, 356 kW at 80% efficiency connected to 445 kW motors (600 HP). Local noise restrictions may require construction of silencer shrouds on surface, or, in more restrictive cases, the fan may be installed underground in a fan station excavated at the bottom of the raise. Modelling of the ventilation system to more precisely design the fans and size the mine openings is required.

Variable pitch fans and variable frequency drive (VFD) motors can ensure that flexibility of the ventilation system is maintained over the LOM. If less power is required or varies over the LOM, the system airflow (and power consumption) can be adjusted.

Meteorological data for a 30-year period (1961 to 1990) for White Sulphur Springs, Montana were considered in determining mine air heating requirements. Based on a maximum required temperature rise (-46°F to +40°F), the mine heaters are sized at 6 MW (20 MMBTUH) each and 7.5 MW (26 MMBTUH) for FAR #1 and FAR #2 respectively. Each direct propane fired system includes direct fired burners with a temperature rise of 86°F. The control room is complete with valve trains and electrics. A propane tank farm is required including two 30,000 gal tanks, vaporizers, pumps, cement slab and piers, piping, fencing, engineering, and commissioning. Operating costs include propane consumption based on 95% efficiency to heat ambient air from average temperatures recorded for five months per year (January, February, March, November, and December) to +40°F. The total propane consumption for an entire annual heating season is estimated at 2.3 ML.

Optimization of the main ventilation opening sizes, ventilations fans, and heating system are expected to be undertaken in future phase of engineering work.

The auxiliary ventilation system for the main levels is designed for 47 m^3/s (100,000 cfm) for simultaneous operation of both LHD and haulage truck. Within individual development or drift-and-fill stope headings, auxiliary ventilation is based on LHD only and provides a nominal 17 m^3/s (36,000 cfm with up to 20% leakage allowance). Fifty-six kilowatt (75 HP) fans are typically required for 1.21 m diameter (48") ventilation duct installations, although a trade-off may be considered for 75 kW (100 HP) fans paired with 1.07 m diameter (42") ventilation ducting.

16.4.7 Emergency Egress and Refuge

MSHA regulations require emergency egress be established before commencing the mine production phase. Where the length of such escapeways exceed 91 m (300 ft) vertical, which is the case for the Project, mechanical means of egress will be required per MSHA 30 CFR 57.11055. Provision is made in the mine design to equip a FAR with an emergency hoist (jib hoist arrangement with escape "pod"). The schedule is based on initial installation of the emergency hoist at the surface collar of FAR #1 (for the UZ). The unit will be skid mounted, thus is subsequently moved to surface collar of Intake Raise #2 (for the LZ) once mine development has advanced to lower levels.

Six refuge stations are provided in the mine plan. As the decline advances a portable refuge station can be initially located at the middle production horizon of the UZ (1645 L). Once mining is advanced up and down two horizons, additional refuge stations would be installed (pre-fab or underground-constructed).

Similarly, three refuge stations are planned for the LZ. This arrangement would ensure that throughout the LOM, workers are always within two levels of a refuge station, thus travel time from any point underground to the nearest refuge station should not exceed 30 min (per MSHA 30 CFR 57.11050).

Provision is made in the mine capital cost estimate for a complete set of mine rescue equipment (but not for surface storage facility).

16.4.8 BACKFILL PLAN

A nameplate backfill plant capacity of at least 3,300 t/d is considered prudent to support a steady-state average backfill rate of 1,700 t/d.

The cost estimate for the backfill plant is scaled from a recent installation of a pre-fabricated ARAN plant for paste or dense fill. Based on the shallowness and flat dip of the mineralized material body, relatively long horizontal runs will be required with relatively short vertical drops, which will necessitate booster pumps.

The backfill plant is assumed to be physically adjacent to the concentrator to take advantage of the following synergies:

- surface loader can be multi-purpose
- minimizes transfer distance of tailings from concentrator to backfill plant
- share building services
- supervisory, operations, and maintenance personnel.

The capital cost estimate is preliminary and optimized plant and distribution system will be subject to future work.

16.4.9 POWER SUPPLY REQUIREMENTS

The total connected load for the underground mine (including related surface ventilation fans) is estimated to be 4 to 5 MW. This estimate will be impacted by final capacity determined for dewatering (influenced by how much groundwater encountered and type of backfill selected – whether hydraulic or paste). Power related capital cost estimates for the mine are based on connecting to surface grid power which has been stepped down to nominal 4,160 V at the portal and at the various ventilation raises equipped with fans and/or the emergency hoist. Capital cost estimates include provision of underground transformers, cables, electrical starters, etc.

Surface substations, switch gear, distribution system, grounding beds etc. are excluded from the mine estimate.

16.4.10 MINE WATER SUPPLY REQUIREMENTS

It is assumed that wash water can continue to be supplied from well water on the property, and suitably treated for supply to showers, toilets, and sinks. The design and cost of supplying/treating wash water and related treatment is by others.

It is assumed that modest requirements for potable drinking water within the mine would be economically addressed by delivery of bottled water to underground lunch rooms/ refuge stations.

The mine's process water supply requirements are estimated at 1,000 to 1,500 m^3/d. More than half would be through the backfill plant as backfill transport water; the remainder would be underground primarily for drill water (for jumbos, bolters, underground diamond drills etc.), water sprays for dust suppression at muck piles, washing down blasted faces, etc.

A portion of the process water would remain in the backfill as residual moisture content, a smaller portion would be evaporated into mine exhaust air and while the bulk of the mine water would be captured in underground sumps to be pumped back to surface for treatment and potentially re-cycled to the concentrator and/or the mine.

16.4.11 DRAINAGE AND WATER DISCHARGE REQUIREMENTS

There are no reports of significant occurrences of either water loss or of making water from exploration drill holes. Although advanced hydrology and hydrogeological studies have not been conducted, the general opinion of site personnel is that the groundwater could be assumed to be modest (other than for seasonal run-off).

Key sources of water discharge from the mine will be from backfill transport water and mine process water.

Allowance is included for excavation of a main sump and pumping station underground with approximately 500 gpm (approximately 136 m^3/h) peak capacities. This is a median estimate for preliminary costing purposes only, and will require hydrogeological assessment of likely groundwater inflows.

To avoid requirement for underground settlers, slimes handling etc., the plan is based on dirty water pumps. Provision is not included for treatment of mine discharge water underground. All water treatment is assumed to be within the surface water treatment plant (to be designed/estimated by others).

16.4.12 UNDERGROUND FIRE PROTECTION

The bulk of re-fuelling is expected to take place on surface. It is assumed the surface diesel storage, including area for "Sat-Stat" system storage tanks and the dispensing unit will to be designed/estimated by others.

Haulage trucks will be the largest single consumer of diesel amongst the underground fleet and are expected to return to surface at the end of each shift. Slower moving vehicles that are impractical to return to surface each day would be fuelled at one or more underground "Sat-Stat" fuel stations. These prefabbed units are equipped with self-contained dry chemical fire suppression systems.

Although provisions can be made for minor in-field repairs and servicing of diesel equipment, there is no plan for excavation of underground service/welding shops or other such facilities requiring underground sprinkler/fire suppression systems. All major servicing, welding etc. for equipment will be in the surface shop (to be designed/estimated by others).

Additionally, all mobile equipment will be equipped with fire extinguishers, and where flammable liquid (fuel/hydraulic oils, etc.) exceeds 100 L, will also be equipped with on-board fire suppression systems.

16.4.13 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.

16.4.14 Communications

Allowance is made for leaky feeder radio system throughout the mine. Tie-in to surface communications, central control room facilities etc. are not included in the mining estimate.

16.4.15 Buildings and Ancillary Facilities

The estimates include provision for the following facilities:

- surface portal cover
- surface raise collars for fresh air and return air ventilation raises
- main surface fans
- mine intake air heaters with propane tank farm
- emergency egress hoist
- backfill plant and distribution system.

The following facilities are required for mine operations and are provided in the design and estimate by others:

- service utilities to be installed to the portal and raise collars/fans as required including electrical power, process water, and compressed air
- surface water treatment for mine discharge water
- sewage treatment facility
- surface mineralized material and waste stockpile areas
- concentrator and tailings disposal facility
- surface access roads/parking lot(s) and security facility/fencing

- surface maintenance shop facility
- compressors
- mine dry
- mine administrative building including “line up” areas for mine production and maintenance personnel
- central control room; supervisory/dispatch centre
- mine rescue equipment room/facility.

16.4.16 MINE WASTE DISPOSAL

The amount of development waste generated in pre-production and initial production period is in the order of 560,000 m^3. In the early stage of production, the ability of stopes to accept waste as rockfill will initially be limited; hence a stockpile area is recommended.

During steady state production, any ongoing development waste rock would be utilized directly as rockfill to the greatest extent practical; with the opportunity to backhaul waste from surface for use as rockfill (utilizing mineralized material haul trucks with ejector-boxes). Alternately, one or two rockfill raises could be bored from surface to enable waste to be readily tipped from surface dump and distributed using loaders underground. Such options would be the subject of a future trade-off study.

17.0 RECOVERY METHODS

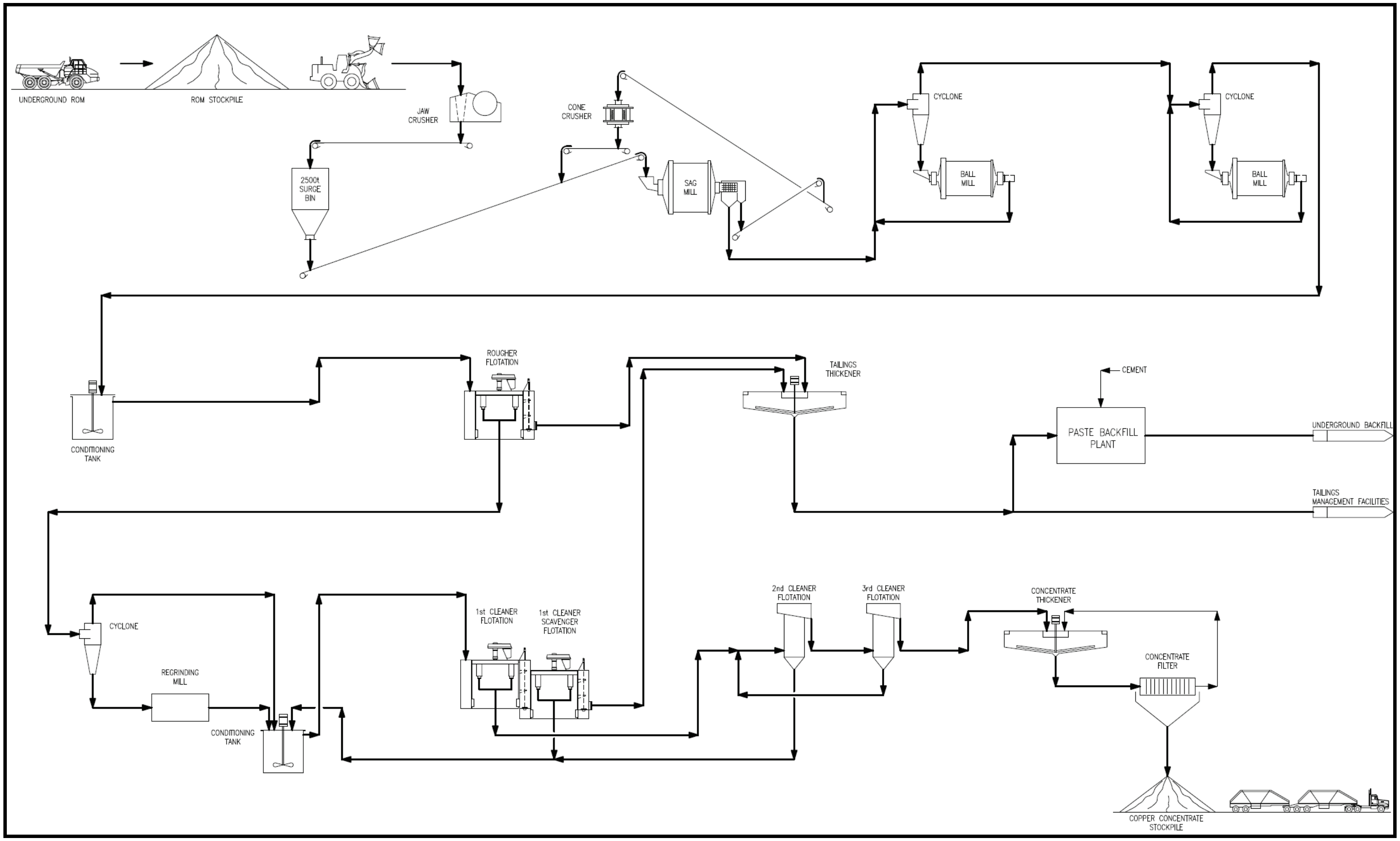
17.1 GENERAL DESCRIPTION

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, 350 d/a; the plant will process mineralized material at an annual rate of 1,155,000 t. The crushing plant availability will be 65% and grinding and flotation plant availability will be 94%.

The mill feed will be crushed by a jaw crusher to 80% passing 120 mm, and then ground to 80% passing 38 μm in a SAG/ball mill/SABC. 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% copper, will be thickened and then pressure-filtered before it is shipped to smelters.

A simplified process flowsheet is provided in Figure 17.1.

Figure 17.1 Locked Cycle Test Flowsheet



17.2 PROCESS DESIGN CRITERIA

Process design criteria were developed for the Project is based on a 3,300 t/d (1,155,000 t/a) plant design. 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	350
Plant Availability/Utilization		
Overall Plant Feed	t/a	1,155,000
Overall Plant Feed	t/d	3,300
Crusher Plant Availability	%	70.0
Grinding and Flotation Plant Availability	%	94.0
Crushing Rate	t/h	196
Grinding Rate	t/h	146
Flotation Rate	t/h	146
Design Factor	-	1.15
Head Grades (Average)	Cu %	2.99
Copper Recovery (Average)	%	86.4
Copper Concentrate Grade (Average)	Cu %	23.2
Copper Concentrate Mass Recovery (Average)	%	11.2
Copper Concentrate Production (Average)	t/a	129,000

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 950 x 1,250 mm primary jaw crusher driven by a 160 kW motor. The jaw crusher will crush the oversize material to approximately 80% passing 120 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 self-cleaning belt magnet and a metal detector installed above the surge bin feed conveyor will remove potential tramp steel going into the surge bin. A fogging system will be installed to minimize fugitive dust emissions.

17.3.2 Grinding Circuit Operation

A SABC grinding circuit consisting of one SAG mill, two ball mills, 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 two stages of ball mill 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 146 t/h, using two belt feeders, and then fed via a conveyor to a 6,710 mm by 2,740 mm effective grinding length (EGL) (22 ft by 9 ft) SAG mill, powered by a 1,600 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 900 µm. Lime will be added to the SAG mill to maintain a pulp pH at approximately 9.5 to suppress pyrite.

SECONDARY GRINDING – PRIMARY BALL MILL GRINDING CIRCUIT

The SAG mill screen undersize will discharge into the hydrocyclone feed pump box in the primary ball mill grinding circuit, together with the primary ball mill discharge. The combined slurry will be pumped to the hydrocyclone cluster consisting of three 500 mm diameter hydrocyclones. The hydrocyclone underflow, with a solid density of approximately 70 to 75%, will feed to a 4,120 mm diameter by 6,700 mm long (13.5 ft by 22 ft) ball mill powered by a 1,600 kW drive motor. The circulation load for this circuit will be approximately 250%. The hydrocyclone overflow will be directed to the hydrocyclone feed pump box in the secondary ball mill grinding circuit.

TERTIARY GRINDING – SECONADRY BALL MILL GRINDING CIRCUIT

The classification in the secondary ball grinding circuit will consist of four 250-mm hydrocyclones. The hydrocyclone underflow with a solid density of approximately 65 to 70% will feed to the secondary ball mill which has the same dimensions as the primary ball mill. The installed power for the ball mill will be 1,600 kW as well. 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 94%.

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 secondary ball mill grinding circuit will flow by gravity into the rougher flotation circuit, consisting of eight 50 m^3 flotation tank cells. The slurry pH at the rougher flotation will be maintained at 9.5 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 reused 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 20 100-mm hydrocyclones for classification. The hydrocyclone underflow will then be reground in a stirred mill with an installed power of 1,000 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 sodium cyanide and lime to depress pyrite. Lime will be added to adjust the pulp's pH up to 11. The conditioned pulp will then be fed to a bank of four 50 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 two 50 m^3 flotation cells. 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 pumpbox.

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

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 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 flocculent 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 12-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 180 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 30,000 mm. The flow rate reporting to the thickener will be approximately 114 t/h. Flocculent will be added to improve settling of the tailings. Thickener underflow slurry with a solid density of 55% 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 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

	Mill Feed		Copper Concentrate		
Year	Tonnage (t/a)	Grade Cu %	Recovery Cu %	Grade Cu %	Production (t/a)
1	450,000	3.17	82.2	21.730	53,919
2	870,000	2.95	82.2	21.730	97,011
3	944,000	2.91	83.6	22.213	103,274
4	1,030,000	2.72	85.3	22.290	107,350
5	1,154,000	3.03	87.4	23.564	129,724
6	1,150,000	3.06	87.4	23.564	130,682
7	1,160,000	3.03	87.4	23.564	130,321
8	1,150,000	3.02	87.4	23.564	129,028
9	1,152,000	3.02	87.4	23.564	128,944
10	1,057,000	3.01	87.4	23.564	118,194
11	885,000	3.04	87.4	23.564	99,865
12	560,000	3.07	87.4	23.564	63,710
13	419,000	3.12	87.4	23.564	48,535
14	175,000	2.53	87.4	23.564	16,424
Total	**12,156,000**	**2.99**	**86.4**	**23.157**	**1,356,981**

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 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 18.2 illustrates the location of the site buildings.

Figure 18.1 General Arrangement

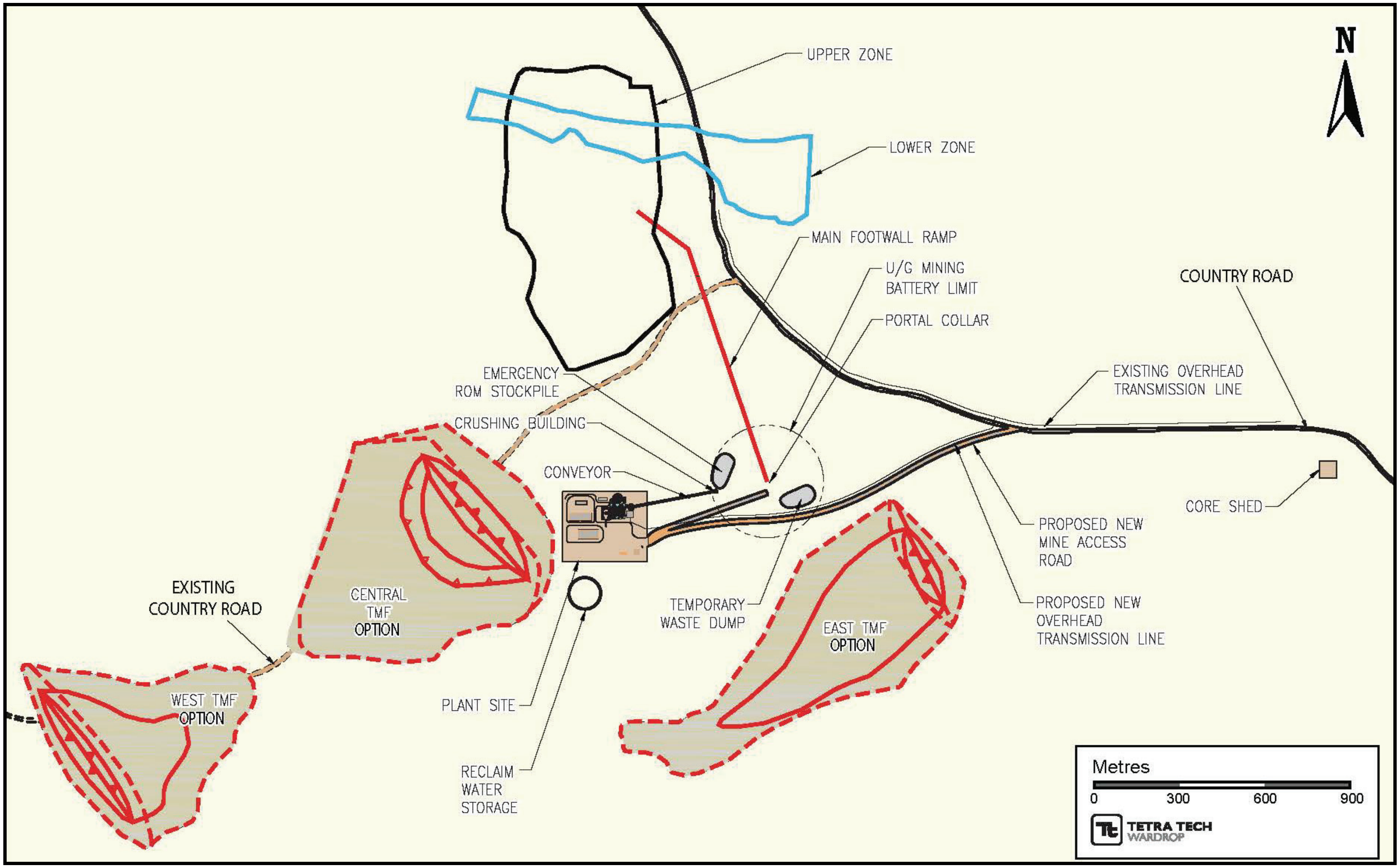
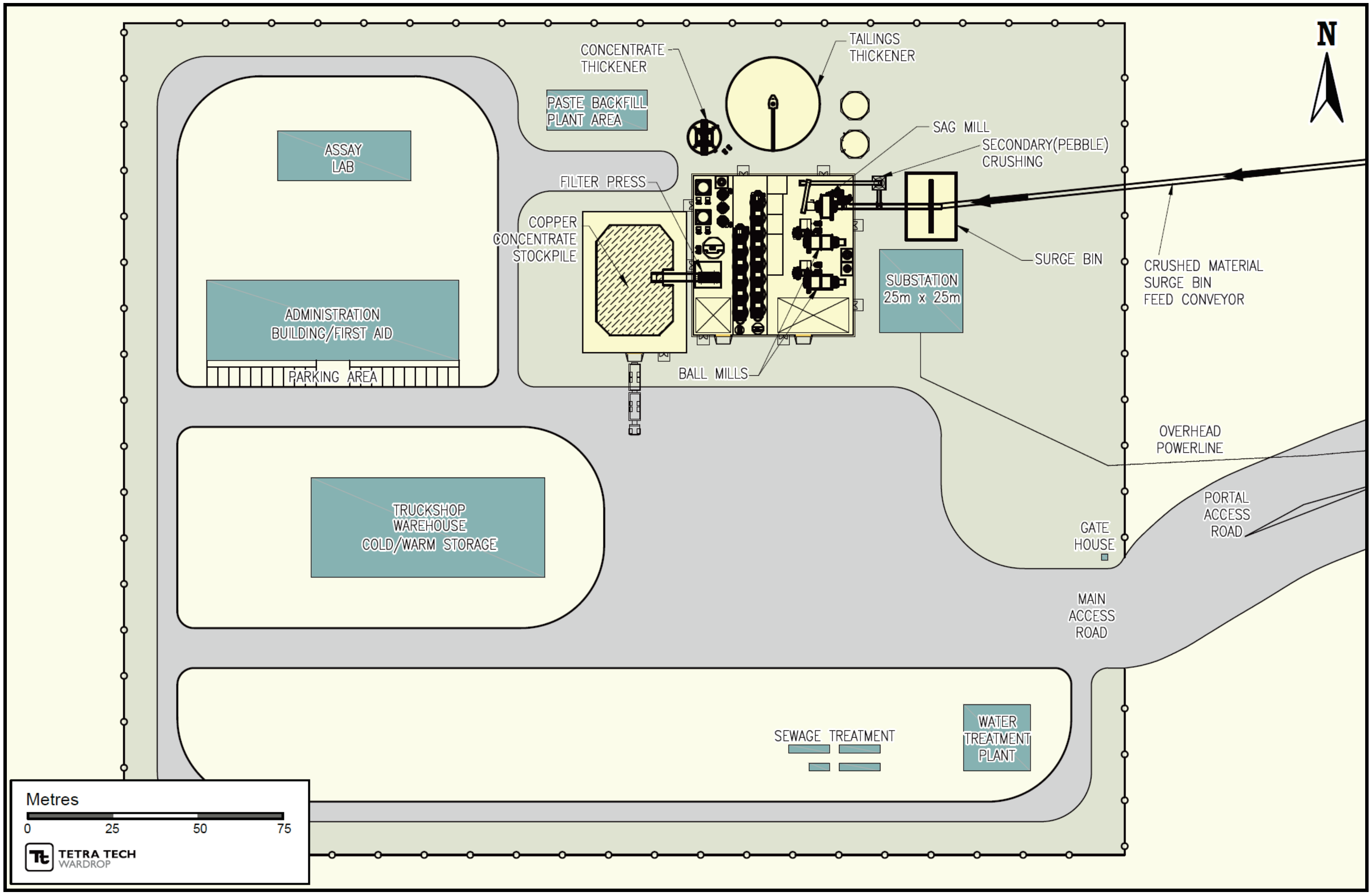


Figure 18.2 Plant and Ancillary Layout



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 loadout 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 collectioin 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 6.1 Mt of tailings (50% of total tailings production) 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 mm 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 from the first four years of operations, 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 12” diameter DR 7.3 HDPE 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 12" diameter DR 11 (Central Impoundment) or DR9 (East Impoundment) HDPE pipe. The barge will be positioned at the north end of the pond to minimize the pumping distance to the head tank.

18.9 Waste Dumps

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 100,000 t 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 dump 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 13.8kV for site wide power distribution.

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

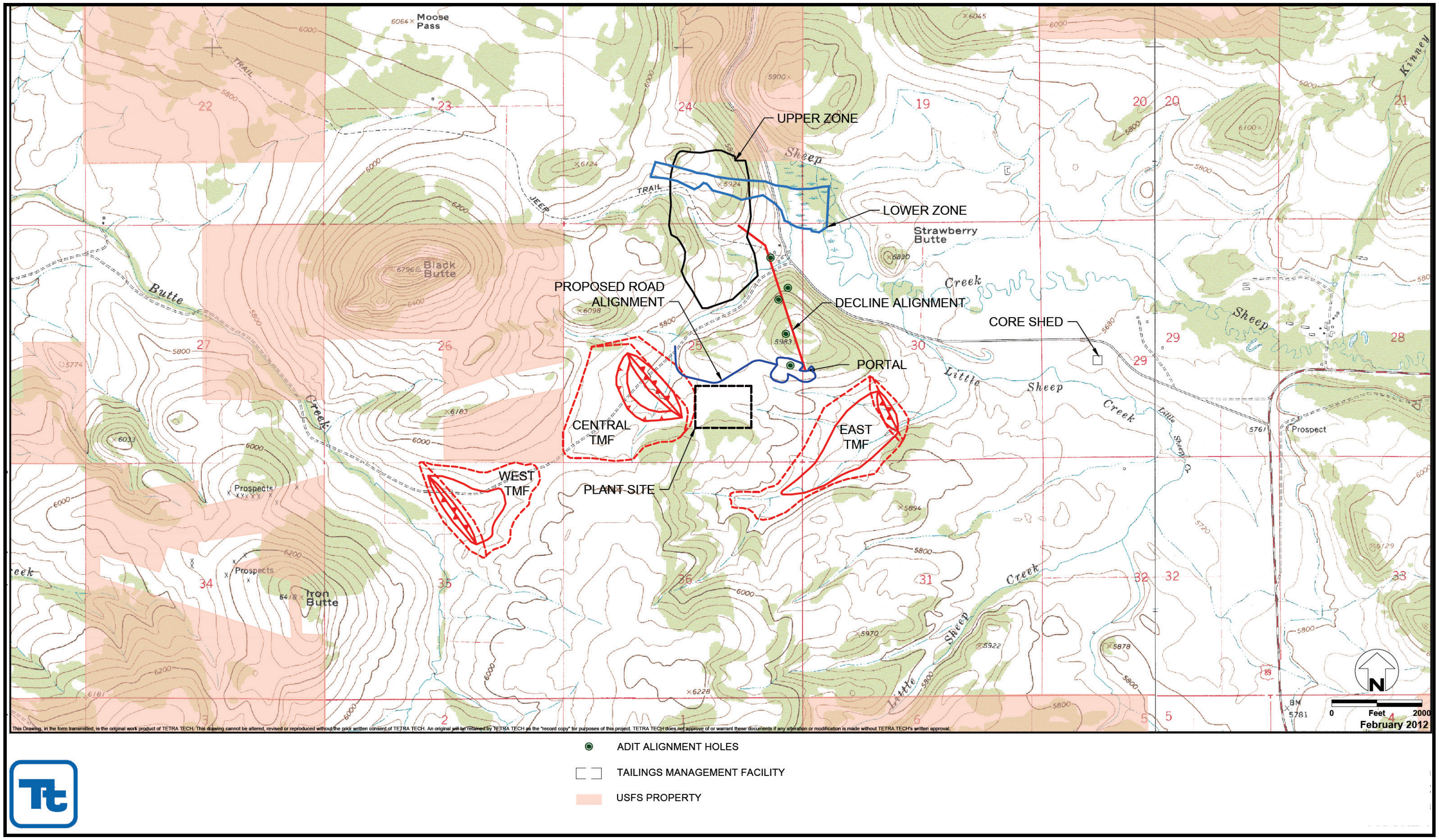
The Property is located on 4,700 acres (1,900 ha) of leased mineral rights on private ranch land, located in Meagher County, about 17 miles (24 km) north of the town of White Sulphur Springs, Montana. The Property abuts fragmented pieces of land west of the Property, whose surface is controlled by the USDA Forest Service, on the Lewis and Clark National Forest and other private owners. As illustrated in Figure 20.1, the Property can be accessed via gravel county and ranch roads located west of US Highway 89.

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 PEA describes:

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

Figure 20.1 Site Plan



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 owner who owns both the surface, mineral, and water rights. Because the Project is on private property and located in Montana, the Montana 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 meet DEQ's 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 which 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 MEPA and other public notice and participation statues.

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 is currently in the process of preparing an amendment to the exploration license to construct the exploration decline, and has 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 are 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; and 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, based upon a facilities total emission, is required. 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 landowner 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 center 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 to provide 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 in 2010 by Tintina Resources 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. The majority of studies, however, 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 late 2012.

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 a meteorological station in March 2012.

20.4.2 WATER RESOURCES

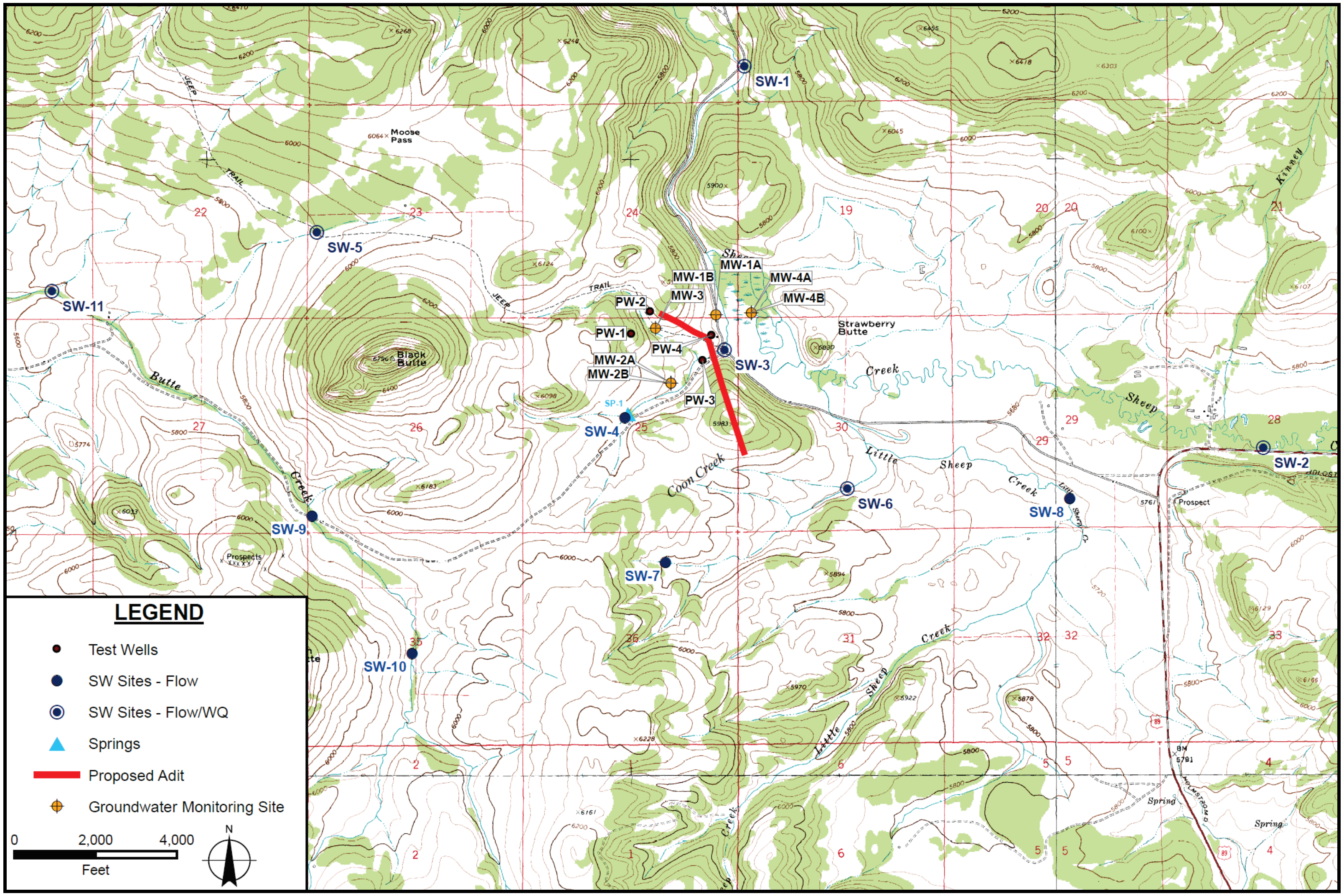
GENERAL HYDROLOGIC SETTING

The location of surface and groundwater quality monitoring sites (Figure 20.2), 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.

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 sampling events were completed in 2011 and two events have been completed in 2012. 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.

Figure 20.2 Surface and Groundwater Monitoring Sample Locations



GROUNDWATER MONITORING SITES AND WATER LEVEL DATA

Groundwater baseline studies were also initiated in 2011, and to date, nine groundwater monitoring installations have been completed (Figure 20.2). Groundwater quality and water level data has been collected from two pairs of monitoring wells (MW-1a/1b and MW2a/2b1b), which are located down-gradient of proposed mine activity and facility locations. These wells were completed in alluvium (1a/2a) and bedrock (1b/2b), 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 monitoring wells were installed in the vicinity of the proposed adit decline in 2012.

Groundwater sampling was conducted at one bedrock/colluvial well pair in August and November 2011, and again in June and August 2012. 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 some water samples from the colluvial well for dissolved thallium, and in the bedrock well for dissolved arsenic and thallium. The bedrock well sample also exceeded the narrative guidelines (based on secondary federal maximum contaminant levels (MCLs)) for iron and manganese. 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 with the exception of manganese and iron.

AQUIFER TESTING

Two aquifer testing programs have been completed for the Project which used open PQ and HQ core holes along with newly constructed pumping wells or 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, 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 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 most wetland areas. However, small wetlands do occur in areas initially proposed for possible tailings impoundment sites. 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
172E	Tripet, rubbley-Libeg complex	Fine-loamy, mixed, superactive Ustic Argicryolls	Poor
340D	Burnette-Lymanson-Adel loams	Fine, smectic, Pachic Argicryolls	Fair
372E	Parkview, extremely bouldery-Grafen, rubbly complex	Loamy-skeletal, mixed, superactive Ustollic Haplocryalfs	Poor
1176E	Kimpton-Zade families complex	Loamy-skeletal, mixed, superactive Ustic Haplocryalfs	Poor

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 in the vicinity of the mine portal and will be recommend for use in combination with various source control techniques (grouting and groundwater pumping) during initial dewatering of the mine decline and future dewatering prior to mining.

20.4.4 MINE WASTE GEOCHEMICAL CHARACTERIZATION

Baseline environmental geochemistry 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, 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.

STATIC TESTING OF ACID GENERATION POTENTIAL

Based on statistical analysis of whole rock exploration data (ICP analysis of four acid digestates), 8 to 10 samples of each material type have been identified, including dolomite, conglomerate, shale, rock containing 3 to 10% sulphide, and massive sulphide (greater than 10%). Analytical work is underway, which will be reported during Q3 2012.

The sulphur content of rock to be mined from the evaluation adit is variable, ranging from below detect to more than 40% by weight. Preliminary results of static tests indicate that portions of the dolomite, shale, conglomerate, and unclassified material have low potential to generate acid (NP:AP>3), with some samples that have an uncertain potential to generate acid, as indicated by NP:AP values that lie between 1 and 3. Samples of massive sulphide all fall into the acid generating field (NP:AP<1), along with most but not all of the 3 to 10% sulphide samples. These results need to be validated with the results of ongoing analyses, and correlated with block model estimates for the volume of each rock unit, before their significance can be determined.

Using the results of static tests, subsamples will be selected to develop one or more composites which represent the range of acid generation potential for each rock type. This composite will be split and analyzed to measure metal release potential using the Synthetic Precipitation Leachability Procedure (EPA Method 1312) and the NAG test with leachate analysis of metal concentrations.

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 which its operations introduce, or are likely to have introduced. Noxious weeds will be controlled using appropriate mechanical, biological and chemical treatments which meet the requirements of Montana and federal laws and a weed control plan will be developed between the landowners, 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, five mammals, one amphibian, and five 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 D EQ 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 1,210 ac 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 scaled 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 employs 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 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 and mineral rights owners vary, but the lthe 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 massive sulfide mineralogy of the deposit and the need to protect surface and groundwater resources from contamination. Oxidation of massive 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 Rainbird-type irrigation systems. A major component of this method of water disposal is through evaporation, so often the Rainbird-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 in the winter months 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.

A waste rock storage 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. This waste rock storage facility would be constructed using a composite compacted clay/geotextile bottom liner, with an internal waste rock seepage collection system that could be pumped to a water treatment facility for treatment as necessary. 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. Once the mine is in operation, wastes would either be stored in underground working voids or placed in the tailings impoundment. Tailings facilities 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. Treatment facilities being considered include lime treatment, reverse osmosis with thermal evaporation of brines, sulphide precipitation, and zero discharge strategies. Other methods 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.

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	210,635,778	-	±30%
Total Operating Costs	-	68.93	+35%/-15%

The estimate base date is Q2 2012; 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 Association for the Advancement of Cost Engineering (AACE) International Estimate Classification System with an expected accuracy range of ±30%.

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	2,611,844
Mine Capital (Stantec)	53,642,988
Processing	58,618,969
Water Management (Knight Piésold)	9,478,403
Utilities	5,194,203
Buildings	8,242,691
Off-site Infrastructure	4,066,207
Plant Mobile Equipment	2,063,212
Subtotal	**143,918,516**
Indirect Costs	30,346,106
Owner's Costs	6,294,657
Contingency	30,076,498
Total Capital Costs	**210,635,778**

21.1.3 Contributors to the Estimate

The estimate was developed by Tetra Tech, in conjunction with.

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

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 Q2 2012. No escalation beyond Q2 2012 was applied to the estimate.

The budget quotes used in this estimate were obtained in Q2 2012 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
				11120	Existing Access Road Improvement
				11130	Haul Roads
				11140	Site Roads at Mine
21	Mine Underground (Stantec)	211	Mine Development	21100	Mine Development
22	Mine Surface Facilities (Stantec)	223	Backfill Plant	22300	Backfill Plant
31	Process	311	Crushing	31110	Primary Crushing
				31120	Primary Crushing Conveyance
		312	Mineralized Material Stockpile and Conveying	31220	Crushed Mineralized Material 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	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	41320	Fresh Water Supply
				41330	Water Treatment Plant
				41340	Gland Water

table continues…

Major	Major Description	Area	Area Description	Sub-area	Sub-area Description
51	Utilities	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	Buildings	611	Ancillary Buildings	-	-
65	Off-site Infrastructure	651	Off-site Infrastructure	65110	Off-site Infrastructure
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)
				91180	Vendor Commissioning and Assistance
98	Owner's Costs	981	Owner's Costs	98100	Owner's Costs
99	Contingency	991	Contingency	91110	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.

The source and availability of labour should be verified in the next phase of the study.

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 Owner's costs.

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 as no geotechnical information was available at the time of the estimate preparation. 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 Stantec. An allowance for EPCM activities, as well as a contingency amount, were included with the construction indirects.

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. The average installed concrete unit rates for 30 MPa concrete used in the estimate are $1,125/m³. 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.

An average supply unit rate of $5,549/t for fabricated steel, based on quotations from recent similar projects, are used in the estimate. 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 $3 million has been included for spares.

21.1.13 EPCM

An allowance of 12% of the direct costs has been included for EPCM activities.

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% of the direct costs.

21.2 Mining Costs – Basis of Estimate

The mining cost estimates are based on Stantec's database of numerous recent mine development projects and mine operations. Where possible, recent budgetary quotes for mining equipment and supplies have been used, otherwise estimates have been derived from first principles with suitable adjustments for rates typical for Montana (such as power, fuel, labour, etc.).

Labour rates typical for Montana and for neighbouring underground operations at Stillwater were further compared and reconciled with recent budgetary quotes for another mining project provided by Tintina.

Scheduling was prepared using Oracle® Primavera scheduling software and linked to the Hard Dollar estimation software.

Depending on delivery times, equipment market, and other factors, initial sets of mining gear might be supplied by the mine development contractor in order to expedite the Project start. However, for costing purposes in this study, mining equipment required to develop and operate the mine are assumed to be procured through a combination of leasing (for prime movers – LHD units, haul trucks as well as drill jumbos and bolting rigs) and outright purchase by Tintina (for graders, forklifts, personnel vehicles, and other service units). During the pre-production period, the mine development contractors would operate and maintain the equipment but is it is presumed that they do not to supply the equipment. An exception to this would be the raise boring machine and drill rods that would be used to develop the ventilation/escape raise systems.

The models indicated are only to illustrate general type/class of equipment and are not a recommendation for specific manufacturer or model.

The mining capital estimate excludes a bottom line contingency allowance as this is expected to be applied to the overall project. "Working capital" allowance, first-fill inventories of supplies, and inventory of maintenance parts are not included.

The estimate does not purport to be an optimized procurement plan as there may be suitable used equipment on the market for select gear or there may be more capital effective arrangements for the contractor to supply the initial sets of gear, or otherwise potentially rent. However, for purposes of a PEA, these will suffice.

The fleet planned is considered fit-for-purpose without inclusion of highly advanced technologies (automation/computerized drilling/laser LHD guidance, etc.). Certain such features may in fact have some application, but for preliminary analysis purposes, identification and economic justification is considered a future optimization exercise.

Spares

Equipment spares and first fills are not included with the estimate. First fill costs have been included in the indirect costs.

Freight for mobile equipment is not included in the estimate. Freight for fixed equipment (surface fans, pumps, etc.) is included in the capital purchase price (procurement and install cost).

Initial Equipment Service

It is assumed that mobile equipment would be dropped off and ready to go underground and therefore initial equipment service is not required. Fixed equipment (surface fans, pumps, etc.) commissioning is included in the procurement and installation cost.

Construction Indirects – Mobilize and De-mobilize, etc.

Contractor indirects (labour) are included. A mobilization/de-mobilization cost is included for the contractor.

EPCM

EPCM is excluded from the mine cost estimate. Tintina's indirects are included and assumed to be sufficient to handle the construction period.

Contingency

Contingency not included specifically for the mine. It is assumed to be included in the contingency allowance for the overall Project.

Unit costs are summarized in Table 21.4.

All mine development and stopes are assumed to have ground support installed typical of North American mine industry standards: pattern bolting of backs and walls to within 1.5 m of the floor, generally with steel weld mesh applied. An allowance has been included for a proportion of the backs including intersections to have heavier support (super-swellex or cable bolts).

Backfill binder costs are based on Portland cement; fly ash may be available in Montana and represents an opportunity for some cost savings depending on pricing that Tintina can negotiate.

Table 21.4 Capital and Operating Cost Estimate Unit Cost Summary

Description	Unit Cost (Cdn$)	Unit
Development Cost		
Ramp Development	4,893	m
Lateral Development	3,672	m
Raisebore	5,500	m
Cut and Fill Advance		
Advance (Blended Advance and Slashing)	27.50	t
Haulage to Surface		
Haulage to Surface (UZ)	4.56	t
Haulage to Surface (LZ)	8.19	t
Paste Fill		
Backfill Crew and Equipment	3.18	t of Backfill
Backfill Material	7.25	t of Backfill
Equipment Operating	2.13	t of Backfill
Subtotal Pastefill	**12.56**	**-**

table continues…

Description	Unit Cost (Cdn$)	Unit
Contractor Indirect Costs	**7,918**	**day**
Tintina Indirect Costs		
Tintina Indirect Costs OPEX	1,131,000	year
Portland Cement	**175**	**t**
Propane	**0.60**	**L**
Electricity		
Electricity Consumption	0.052	kWh

21.3 MINING CAPITAL COST ESTIMATE

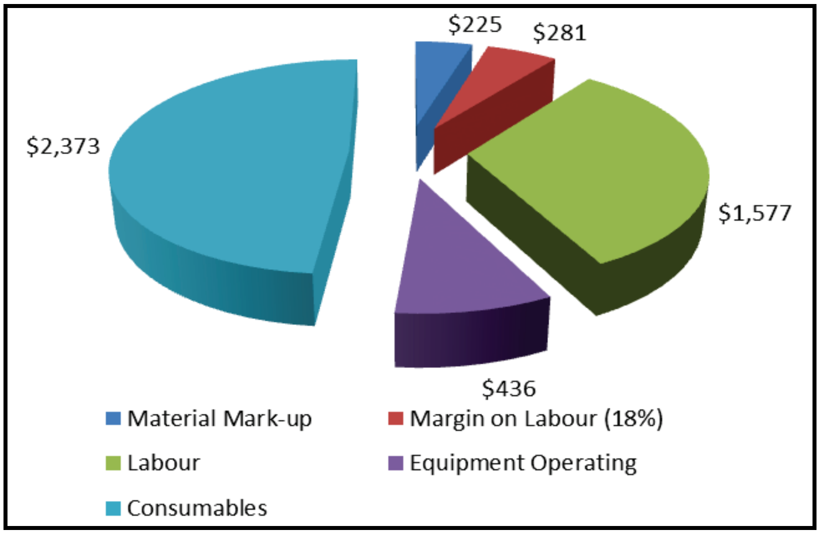
Total pre-production capital for the mine including underground mine development and related contractor and Tintina's indirect costs during the capital period is estimated at Cdn$53.8 million. The majority is for mine development but includes Cdn$6 million for the purchase of a portion of the underground mobile fleet that is not leased. LOM costs are summarized in Table 21.4 (assuming cost of collaring mine portal, any associated surface dewatering wells, and surface set-up to drive the main decline, and first 900 m of decline have already been included as part of the underground exploration program before commencement of the actual mine project).

A further Cdn$37.9 million over the remainder of the mine life is estimated as required sustaining capital in order to complete lateral development to the lower levels and related vertical development, and to replace equipment mobile equipment during the 14-year LOM.

Provision for closure costs are not included in the capital estimate for the underground mine, nor for any salvage value for fixed mine plant or mobile equipment. Closure of the mine portion of the operations would consist of essentially four concrete bulkheads: the decline and each of the three raise collars at surface. Removal of mobile equipment from the mine and stripping of various fixed plant gear may yield sufficient salvage value to break-even with cost of bulkhead construction.

Direct unit cost for main ramp development using mine contractors during the pre-production period is approximately Cdn$4,893/m of advance, which includes Cdn$2,373/m for consumables (air/water pipe, vent duct, electrical cable, drilling supplies, ground support supplies, explosives). The main decline unit cost estimate is shown in Figure 21.1.

Figure 21.1 Main Decline Unit Cost with Contractors



21.4 OPERATING COST ESTIMATE

21.4.1 SUMMARY

The total LOM operating cost for the proposed mine is estimated at $68.93/t milled on average. The estimate includes mining, processing, tailing management, G&A and surface service costs.

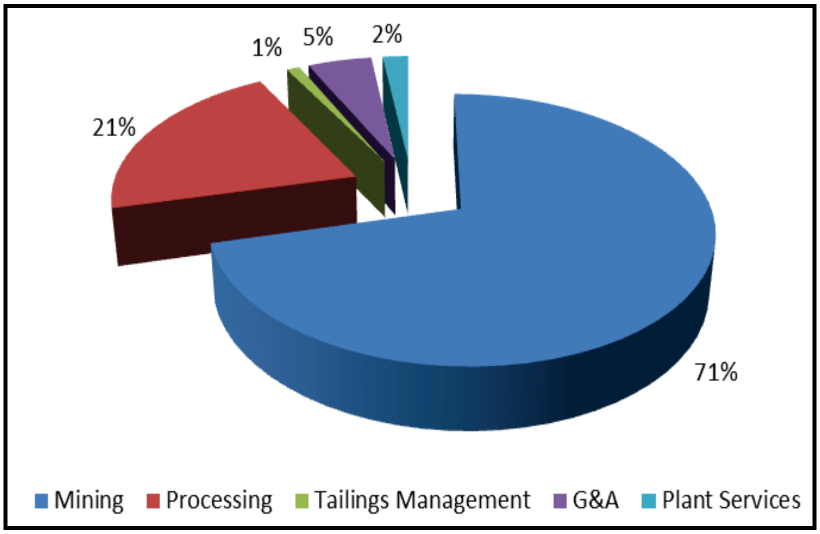
A total of 12,156,000 t mineralization from the underground mine will be processed during LOM based on the proposed mining schedule. On average annual process rate is approximately 960,000 t/a or 2,740 t/d at 350 d/a. The unit cost is estimated based on the LOM average mill feed rate. The accuracy for the estimate is expected to be within a range of +35% and -15%. The breakdown operating cost estimates are presented in Table 21.5. Figure 21.2 shows the cost breakdown at the LOM average throughput.

Table 21.5 Overall Operating Cost

Area	LOM Average Unit Operating Cost ($/t milled)
Mining*	48.96
Processing	14.63
Tailing Management	0.29
G&A	3.39
Plant Services	1.66
Total	**68.93**

Notes: *including backfill cost

Figure 21.2 LOM Average Operating Cost Distribution



21.4.2 *MINING OPERATING COST ESTIMATE*

Once the mine has been developed to the 1645 Level and is ready for initial production, it is assumed that Tintina's crews will have been adequately trained to take over from the contractors' crews for the mining of mineralized material. It is presumed that the contractors' crews continue with main ramp development to access the LZ.

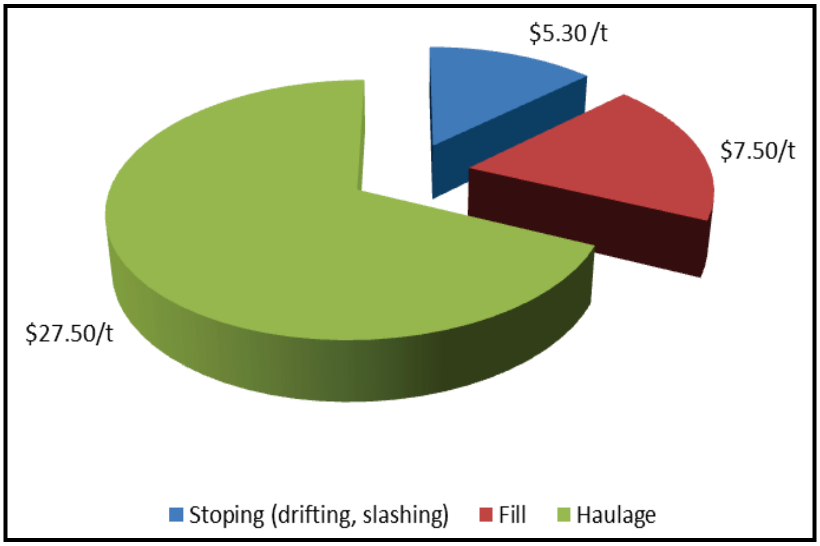
Direct stoping costs are estimated at Cdn$27.50/t based on a blend of 75% of production derived from "drifting" in mineralized material and 25% derived from "slashing".

In addition to direct stoping costs, other direct operating costs include:

- backfill at Cdn$7.54/t of mineralized material (including binder, labour, and equipment costs)
- haulage cost to surface ranging from Cdn$4.56/t of mineralized material from the UZ to Cdn$8.19/t of mineralized material for the deeper the LZ.

The total direct unit operating production cost estimate is Cdn$40.30/t mineralized material as summarized in Figure 21.3.

Figure 21.3 Drift-and-Fill Unit Costs (Average)



The total mine production costs include operating indirect costs (Cdn$3.58/t mineralized material average), including mine air heating (Cdn$1.41/t mineralized material average), leasing costs for portion of fleet leased (Cdn$3.67/t mineralized material average) amount to Cdn$49.01/t of mineralized material mined over the LOM which excludes mobile equipment maintenance and electrical power (covered in surface G&A).

An overall summary schedule of all expenditures for the LOM (initial capital, sustaining capital, and operating costs) for mine development, mine equipment and infrastructure, and mine operating costs is provided in Table 21.6.

Table 21.6 LOM Cash Expenditures – Black Butte Copper Project (Cdn$)

Description	LOM	Year -1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Advanced Exploration Program	**Excluded**																
Portal, Initial Ramp, U/G Drilling/ Bulk Sample																	
Upper Zone	**423,783,616**																
Surface Infrastructure	15,175,700	-	13,375,700	1,550,000	250,000	-	-	-	-	-	-	-	-	-	-	-	-
Underground Infrastructure	**9,653,471**	**-**	**3,305,275**	**4,658,903**	**599,918**	**137,250**	**136,875**	**136,875**	**136,875**	**137,250**	**136,875**	**136,875**	**130,500**	**-**	**-**	**-**	**-**
Mine Development (Ramps, Lat. Waste, Raises, Haulage)	17,169,813	-	11,429,031	5,740,782	-	-	-	-	-	-	-	-	-	-	-	-	-
Direct Mining																	
Stoping	265,155,000	-	-	12,375,000	23,925,000	24,475,000	24,475,000	24,310,000	23,925,000	24,200,000	23,925,000	23,980,000	21,367,500	16,637,500	9,900,000	6,847,500	4,812,500
Haulage	**43,967,520**	**-**	**-**	**2,052,000**	**3,967,200**	**4,058,400**	**4,058,400**	**4,031,040**	**3,967,200**	**4,012,800**	**3,967,200**	**3,976,320**	**3,543,120**	**2,758,800**	**1,641,600**	**1,135,440**	**798,000**
Backfill	72,662,112	-	-	3,391,200	6,556,320	6,707,040	6,707,040	6,661,824	6,556,320	6,631,680	6,556,320	6,571,392	5,855,472	4,559,280	2,712,960	1,876,464	1,318,800
Captial	40,059,692	-	28,110,006	11,949,686	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	**1,939,293**	**-**	**-**	**-**	**849,918**	**137,250**	**136,875**	**136,875**	**136,875**	**137,250**	**136,875**	**136,875**	**130,500**	**-**	**-**	**-**	**-**
Operating (Direct Mining Only)	381,784,632	-	-	17,818,200	34,448,520	35,240,440	35,240,440	35,002,864	34,448,520	34,844,480	34,448,520	34,527,712	30,766,092	23,955,580	14,254,560	9,859,404	6,929,300
Tonnages	9,642,000	-	-	450,000	870,000	890,000	890,000	884,000	870,000	880,000	870,000	872,000	777,000	605,000	360,000	249,000	175,000
Cost per tonne (Direct Mining Only)	39.60	-	-	39.60	39.60	39.60	39.60	39.60	39.60	39.60	39.60	39.60	39.60	39.60	39.60	39.60	39.60
Lower Zone	**145,296,876**																
Surface Infrastructure	3,885,700	-	-		3,760,700	125,000	-	-	-	-	-	-	-	-	-	-	-
Underground Infrastructure	**6,453,471**	**-**	**-**	**1,478,857**	**156,428**	**1,773,315**	**1,527,167**	**428,329**	**136,875**	**137,250**	**136,875**	**136,875**	**136,875**	**137,250**	**136,875**	**130,500**	**-**
Mine Development (Ramps, Lat. Waste, Raises, Haulage)	26,287,541	-	377,835	11,816,521	7,200,585	5,112,910	1,779,690	-	-	-	-	-	-	-	-	-	-
Direct Mining																	
Stoping	69,135,000	-	-	-	-	1,485,000	3,850,000	7,425,000	7,700,000	7,700,000	7,700,000	7,700,000	7,700,000	7,700,000	5,500,000	4,675,000	-
Haulage	**20,589,660**	**-**	**-**	**-**	**-**	**442,260**	**1,146,600**	**2,211,300**	**2,293,200**	**2,293,200**	**2,293,200**	**2,293,200**	**2,293,200**	**2,293,200**	**1,638,000**	**1,392,300**	**-**
Backfill	18,945,504	-	-	-	-	406,944	1,055,040	2,034,720	2,110,080	2,110,080	2,110,080	2,110,080	2,110,080	2,110,080	1,507,200	1,281,120	
Capital	3,701,679	-	377,835	3,323,845	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	**32,925,033**	**-**	**-**	**9,971,534**	**11,117,714**	**7,011,225**	**3,306,857**	**428,329**	**136,875**	**137,250**	**136,875**	**136,875**	**136,875**	**137,250**	**136,875**	**130,500**	
Operating (Direct Mining Only)	**108,670,164**	**-**	**-**	**-**	**-**	**2,334,204**	**6,051,640**	**11,671,020**	**12,103,280**	**12,103,280**	**12,103,280**	**12,103,280**	**12,103,280**	**12,103,280**	**8,645,200**	**7,348,420**	**-**
Tonnages	2,514,000	-	-	-	-	54,000	140,000	270,000	280,000	280,000	280,000	280,000	280,000	280,000	200,000	170,000	-
Cost per Tonne (Direct Mining Only)	43.23	-	-	-	-	43.23	43.23	43.23	43.23	43.23	43.23	43.23	43.23	43.23	43.23	43.23	-

table continues…

Description	LOM	Year															
		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Shared - Both UZ & LZ	**118,301,270**																
Capital Mobile Equipment Cost	5,992,500	-	1,997,500	998,750	998,750	-	-	-	-	665,833	665,833	665,833	-	-	-	-	-
Total Indirect Costs* **	**47,722,552**	-	**3,180,096**	**3,690,464**	**5,357,152**	**5,408,952**	**5,469,152**	**2,784,652**	**2,781,852**	**2,788,852**	**2,781,852**	**2,783,252**	**2,716,752**	**2,596,352**	**2,368,852**	**1,804,424**	**1,209,896**
Labour Owner	14,024,400	-			1,131,000	1,131,000	1,131,000	1,131,000	1,131,000	1,131,000	1,131,000	1,131,000	1,131,000	1,131,000	1,131,000	904,800	678,600
Labour Contractor	**13,856,500**	-	**2,771,300**	**2,771,300**	**2,771,300**	**2,771,300**	**2,771,300**										
Supplies & Services	2,288,340	-	146,940	148,380	154,140	154,140	154,140	154,140	154,140	154,140	154,140	154,140	154,140	154,140	154,140	150,540	146,940
Aux. Mobile Equipment Operating	9,044,112	-	261,856	455,784	691,712	691,712	691,712	691,712	691,712	691,712	691,712	691,712	691,712	691,712	691,712	455,784	261,856
Equipment Operating	**8,509,200**	-	-	**315,000**	**609,000**	**660,800**	**721,000**	**807,800**	**805,000**	**812,000**	**805,000**	**806,400**	**739,900**	**619,500**	**392,000**	**293,300**	**122,500**
Electricity - Carried in Surface G&A	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Air Heating***	17,642,218	-	367,546	735,092	1,102,639	1,470,185	1,470,185	1,470,185	1,470,185	1,470,185	1,470,185	1,470,185	1,470,185	1,470,185	1,102,639	735,092	367,546
Leased Equipment (Jumbo, Bolter, Truck & LHD)* **	**46,944,000**	-	**1,722,000**	**2,483,000**	**3,256,000**	**3,474,000**	**3,474,000**	**3,692,000**	**3,692,000**	**3,692,000**	**3,692,000**	**3,692,000**	**3,692,000**	**3,474,000**	**3,256,000**	**2,104,000**	**1,549,000**
***Year 1 - carried with capital, Year 2 - 25% (Q1) carried with capital 75% (Q2,3,4) with operating		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Totals	**687,381,762**																
Total Capital Costs (including Mobile Equipment)	53,754,402	-	35,754,983	17,999,419	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Sustaining Capital Costs (including Mobile Equipment)	37,860,576	-	-	9,971,534	12,966,382	7,148,475	3,443,732	565,204	273,750	940,333	939,583	939,583	267,375	137,250	136,875	130,500	-
Total Operating Costs (Direct Mining Only)	490,454,796	-	-	17,818,200	34,448,520	37,574,644	41,292,080	46,673,884	46,551,800	46,947,760	46,551,800	46,630,992	42,869,372	36,058,860	22,899,760	17,207,824	6,929,300
Total Operating Costs (Direct, Heating & Leasing)	**552,951,467**	-	-	**21,036,292**	**38,807,159**	**42,518,829**	**46,236,265**	**51,836,069**	**51,713,985**	**52,109,945**	**51,713,985**	**51,793,177**	**48,031,557**	**41,003,045**	**27,258,399**	**20,046,916**	**8,845,846**
Total Operating Costs (Direct, Heating, Leasing & Indirects)	595,766,784	-	-	22,999,617	44,164,311	47,927,781	51,705,417	54,620,721	54,495,837	54,898,797	54,495,837	54,576,429	50,748,309	43,599,397	29,627,251	21,851,340	10,055,742
Tonnages	12,156,000	-	-	450,000	870,000	944,000	1,030,000	1,154,000	1,150,000	1,160,000	1,150,000	1,152,000	1,057,000	885,000	560,000	419,000	175,000
Cost per Tonne (Direct Mining Only)	**40.35**	-	-	**39.60**	**39.60**	**39.80**	**40.09**	**40.45**	**40.48**	**40.47**	**40.48**	**40.48**	**40.56**	**40.74**	**40.89**	**41.07**	**39.60**
Cost per Tonne (Leasing)	3.67	-	-	4.14	3.74	3.68	3.37	3.20	3.21	3.18	3.21	3.20	3.49	3.93	5.81	5.02	8.85
Cost per Tonne (Heating)	1.41	-	-	1.23	1.27	1.56	1.43	1.27	1.28	1.27	1.28	1.28	1.39	1.66	1.97	1.75	2.10
Cost per Tonne (Indirects)	3.59	-	-	6.15	6.16	5.73	5.31	2.41	2.42	2.40	2.42	2.42	2.57	2.93	4.23	4.31	6.91
Total Operating Cost per Tonne	**49.01**	-	-	**51.11**	**50.76**	**50.77**	**50.20**	**47.33**	**47.39**	**47.33**	**47.39**	**47.38**	**48.01**	**49.26**	**52.91**	**52.15**	**57.46**

21.4.3 PROCESSING OPERATING COST

The estimated process operating cost is $14.63/t milled or $14.04 million per year. The estimate is based on an average annual process rate of 960,000 t or 2,740 t/d and 350 d/a.

A summary of the plant operation costs is shown in Table 21.7.

Table 21.7 Summary of Process Operating Cost

Description	Labour Force	Annual Cost ($)	Unit Cost ($/t milled)
Labour Force			
Operating Staff	15	1,669,707	1.74
Operating Labour	22	1,493,856	1.56
Maintenance	15	1,027,728	1.07
Subtotal Labour Force	**52**	**4,191,291**	**4.37**
Major Consumables			
Metal Consumables	-	4,722,000	4.92
Reagent Consumables	-	1,236,577	1.29
Supplies			
Maintenance Supplies	-	1,018,196	1.06
Operating Supplies	-	178,643	0.19
Subtotal Consumables and Supplies	**-**	**7,155,416**	**7.46**
Power Supply	-	2,690,439	2.80
Subtotal Power	**-**	**2,690,439**	**2.80**
Total (Process)	-	**14,037,146**	**14.63**

The estimated labour force cost is $4.37/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 350 d/a.

The operating cost for the major metal consumables is estimated to be $4.92/t milled. The metal consumables include mill and crusher liners and mill grinding media.

The estimated reagent cost is $1.29/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 $1.06/t milled. The power cost is estimated based on the average power requirement of 6.7 MW and a unit electric energy price of $0.05/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

21.4.4 General and Administrative Costs and Surface Services Costs

G&A costs are estimated to average $3.39/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 21.8.

Table 21.8 G&A Operating Costs

	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
G&A Labour Force			
G&A	13	1,498,500	1.56
G&A Hourly Personnel	4	306,000	0.32
Subtotal G&A Labour Force	**17**	**1,804,500**	**1.88**
G&A Expense			
General Office Expense	-	40,000	0.04
Computer Supplies including Software	-	50,000	0.05
Communications	-	40,000	0.04
Travel	-	30,000	0.03
Audit	-	100,000	0.11
Consulting/External Assays	-	50,000	0.05
Head Office Allowance: Marketing	-	100,000	0.11
Environmental	-	200,000	0.21
Insurance	-	433,125	0.45
Regional Taxes and Licenses Allowance	-	100,000	0.11
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	0.11
Subtotal G&A Expense	**-**	**1,443,125**	**1.51**
Total	**17**	**3,247,625**	**3.39**

The surface service costs were estimated at $1.66/t milled and are detailed in Table 21.9. 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 21.9 Surface Services Operating Costs

Surface Service	Labour Force	Total Cost ($/a)	Unit Cost ($/t milled)
Surface Service Personnel	12	797,472	0.83
Small Vehicles/Equipment	-	30,000	0.03
Potable Water and Waste Management	-	100,000	0.10
Building Maintenance	-	50,000	0.05
Building Heating	-	300,000	0.31
Electrical Power	-	120,000	0.13
Road Maintenance	-	200,000	0.21
Total	**12**	**1,597,472**	**1.66**

21.4.5 Tailings Management Cost

The estimated operating costs of the TMF is $0.29/t milled.

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

This 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.

Tetra Tech has prepared an economic evaluation of the Project based on a pre-tax financial model.

As of April 24, 2012, Tetra Tech's long-term consensus copper price used in the base case was US$2.97/lb.

The pre-tax financial results are:

- 20.4% IRR
- 5.5-year payback
- US$145.8 million NPV at an 8% discount rate.

Sensitivity analyses were conducted to analyze the sensitivity of the Project merit measures (NPV, IRR and payback periods) to the main inputs.

22.2 PRE-TAX MODEL

22.2.1 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)	12,156
Average Annual Tonnes to Mill ('000)	868
LOM (years)	14
Average Grade	
Copper (%)	2.99
Total Production	
Copper ('000 lb)	692,751
Average Annual Production	
Copper ('000 lb)	49,482

22.2.2 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 annual operating cash flow.

Initial capital costs as well as working capital have been incorporated on a year-by-year basis over the LOM. Salvage value and mine reclamation costs are applied to the capital expenditure in the last production year. 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.

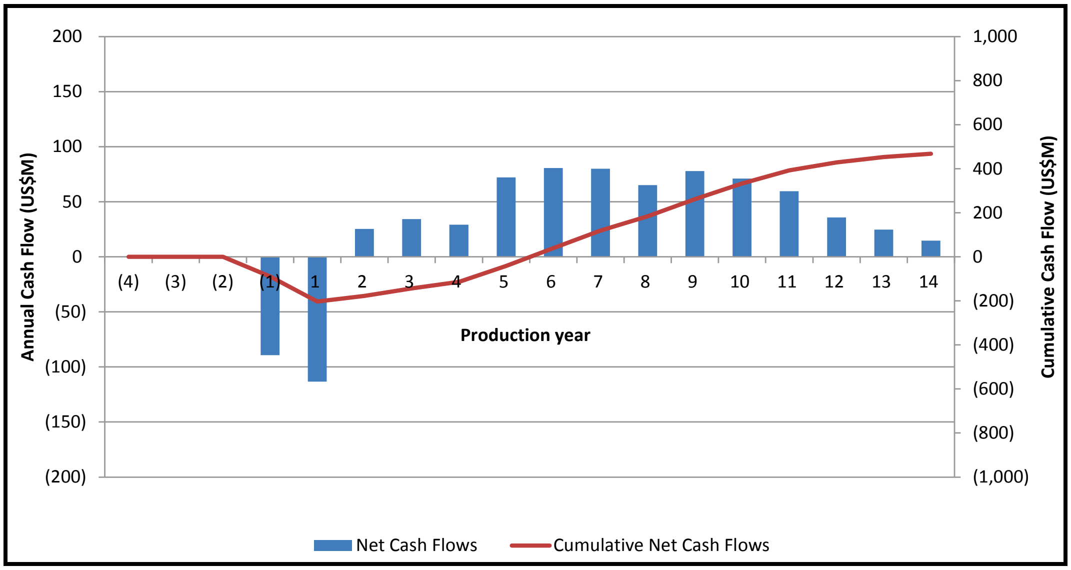
The pre-production period is assumed to be three years.

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.

The salvage value is assumed to be equal to the reclamation cost and both will occur at the end of the LOM.

The undiscounted annual net cash flow (NCF) and cumulative net cash flow (CNCF) are illustrated in Figure 22.1.

Figure 22.1 Undiscounted Annual and Cumulative Net Cash Flow



22.3 Summary of Financial Results

Tetra Tech evaluated the base case using a consensus copper price of US$2.97/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 Pre-tax Financial Results

Description	Base Case	Alternate Case					
		1	2	3	4	5	6
Copper Price (US$/lb)	2.97	2.50	2.75	3.00	3.25	3.50	4.00
Recovered Metal Value (US$ million)	2,057	1,732	1,905	2,078	2,251	2,425	2,771
Off-site Costs & Deductions (US$ million)	466	444	455	467	479	491	514
Operating Costs (US$ million)	838	838	838	838	838	838	838
Operating Cash Flow (US$ million)	754	450	612	773	935	1,096	1,419
Initial Capital Expenditure (US$ million)	210	210	210	210	210	210	210
Total Capital Expenditure (US$ million)	286	286	286	286	286	286	286
Net Cash Flow (US$ million)	468	164	326	487	649	810	1,133
Discounted Cash Flow NPV (US$ million) at 3%	307	80	201	321	442	563	804
Discounted Cash Flow NPV (US$ million) at 5%	230	41	142	242	342	442	643

table continues…

Description	Base Case	Alternate Case					
		1	2	3	4	5	6
Discounted Cash Flow NPV (US$ million) at 8%	146	1	78	155	232	309	463
Payback (years)	5.5	8.2	6.4	5.4	4.8	4.3	3.3
IRR (%)	20.4	8.1	14.9	21.1	27.0	32.6	43.6

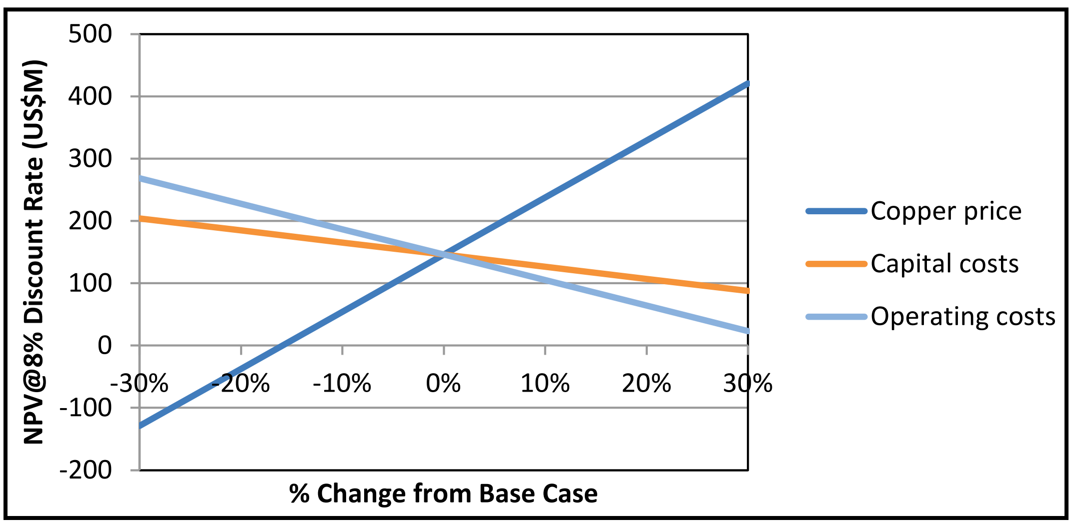
22.4 Sensitivity Analysis

Sensitivity of the Project's NPV, IRR to the Project key variables was investigated. Using the base case as a reference, each of key variables was changed between -30%/+30% at a 10% interval while holding the other variables constant. The following key variables were investigated:

- copper price
- capital costs
- operating costs.

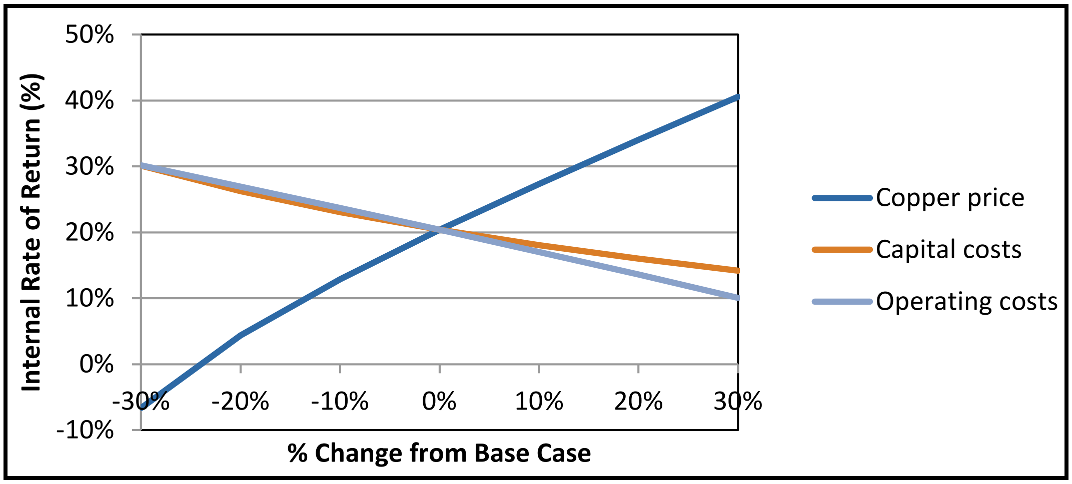
As shown in Figure 22.2, the Project NPV, calculated at an 8% discount, is most sensitive to the copper price and, in decreasing order, operating costs, and capital costs.

Figure 22.2 NPV Sensitivity Analysis



As shown in Figure 22.3, the Project IRR is most sensitive to the copper price followed by the operating costs and capital costs.

Figure 22.3 IRR Sensitivity Analysis



22.5 ROYALTIES

A 2% of NSR royalties was applied in the financial analysis.

22.6 SMELTER TERMS

Typical smelter terms for delivery of copper concentrate to an East Asian smelter to a range of roasters for similar projects have been applied.

Copper concentrate contracts will generally include payment terms as follows:

- copper – pay 100% of content less 1.0 unit at the LME price for Grade A copper less a refining charge of US$0.075/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 more than 0.2%.

22.7 TRANSPORTATION LOGISTICS

Transportation costs for the copper concentrate are listed below:

- trucking – US$33.00/wmt
- port storage and handling – US$18.00/wmt
- ocean transport to Asian port – US$68.00/wmt

- allowance – US$2.35/wmt
- moisture content – 9%.

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 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 sulfide 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

CAI developed preliminary "resource" estimates for the Johnny Lee UZ and LZ. These historic estimates are not considered to be compliant with NI 43-101 (CAI 1996). CAI ultimately terminated their interest in Black Butte due to relatively low copper prices (less than US$1.00/lb).

Using historic and more recent 2010-2011 estimates, Tintina drilling data have allowed for an estimate of mineral resources for the Johnny Lee UZ and LZ (Lechner 2010, 2011, 2012). The focus of this report is an updated estimate of mineral resources for the Lowry MZ. Table 25.1 summarizes the current Black Butte Mineral Resource inventory (only copper grades and contained metal are shown in this table).

Table 25.1 Undiluted Black Butte Mineral Resources

Zone	Tonnes ('000)	Cu (%)	Co (%)	Ag (g/t)	Cu (Mlb)	Co (Mlb)	Ag ('000 oz)
Indicated Mineral Resources							
Johnny Lee UZ[1]	8,483	2.96	0.12	16.9	553	22	4,609
Johnny Lee LZ	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Lowry MZ	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Total	**8,483**	**2.96**	**0.12**	**16.9**	**553**	**22**	**4,609**
Inferred Mineral Resources							
Johnny Lee UZ[1]	1,257	2.64	0.10	16.40	73	3.0	663
Johnny Lee LZ[2]	2,462	4.71	0.06	5.10	256	2.9	404
Lowry MZ[1]	5,139	2.60	0.12	14.60	294	14.0	2,412
Total	**8,858**	**2.60**	**0.12**	**14.60**	**623**	**19.9**	**3,479**

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.

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 indicates that 86% of the chalcopyrite is liberated at a primary grind of 53 µm P_{80}. The copper grade of the LZ composite is higher than the UZ but the cobalt and silver grades are much lower. The results of the locked cycle tests on the UZ composite are consistent with the conclusions of the mineralogy studies.

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 + nickel are slightly elevated and may incur minor penalties. The concentrate does not contain payable silver and cobalt values according to the current test results.

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 Environmental

The environmental baseline study process for the Project is well underway, and the 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.

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. By obtaining the assay certificates a significant portion of the older data could be verified and may minimize the amount of additional drilling that will need to be completed by Tintina.

Tintina should review their assay QA/QC protocols and institute procedures that will allow them to more closely track the performance of blanks and standards. Assay records imported into their database should be scanned for tolerance limits that will flag samples associated with blanks or standards that are under or over specified limits. The cost for this is insignificant.

Tintina should continue collecting more bulk density data from unmineralized hanging wall material, massive sulphide zones, and unmineralized footwall material. These determinations should be collected for all major rock types that may be mined. Confirmatory density determinations should be completed by a certified laboratory for representative rock types. These determinations will also provide an estimate of moisture content. The cost for these tasks is nominal provided that geologists logging core or geologic technicians perform the task. The costs associated with obtaining confirmatory bulk density determinations should not amount to more than US$5,000.

Results from recent metallurgical work suggest the presence of several different metallurgical types within the Johnny Lee UZ. Tintina's geologic staff should work with their metallurgical consultants to determine the characteristics of these possible metallurgical types. The goal should be to see if a metallurgical type model can be constructed given the available data. After various metallurgical types have been identified, representative samples (10 to 15 samples each) should be assembled from existing drill core and provided to Tintina's metallurgical team for test work. The costs for this activity are primarily associated with wages paid to Tintina's geologic staff members, although if insufficient core is available additional drilling maybe required to obtain sufficient material for test work.

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

Since the decision to use the Johnny Lee UZ Indicated and Inferred Resource for the PEA, Tintina has completed 4,691 m of drilling at a cost of approximately US$1.41 million. Tintina should update the resource based on this new drilling. In addition, Tintina should continue estimating mining and processing costs, continue with metallurgical recovery studies, continue preparation of a mine plan from the updated estimate of mineral resources, and continue with ongoing environmental data collection and appropriate permitting activities.

26.2.2 JOHNNY LEE LZ

Since the decision to use the Johnny Lee LZ Inferred Resource for the PEA, Tintina has completed an additional 9,086 m of drilling at a cost of approximately US$2.73 million. Tintina should update the Johnny Lee LZ resource based on this drilling and further improve the geologic interpretation of the Johnny Lee LZ using the recently obtained drilling results. In addition, Tintina should continue estimating mining and processing costs, continue with metallurgical recovery studies, prepare a mine plan from the updated estimate of mineral resources, and continue with ongoing environmental data collection and appropriate permitting activities 2011 drilling results.

26.2.3 LOWRY MZ

Tintina has recently collected drill core samples for metallurgical test work. These samples have been submitted to Inspectorate. Mr. Art Winckers will be overseeing the testing and interpreting the results. The estimated cost to complete this test work to about US$25,000.

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 two stage ball milling with 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. The two stage ball mill grinding circuit should be evaluated with a Verti Mill as the final primary grinding stage.

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

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$500,000.

26.5 Mining

For the next level of the study the following recommendations have been made:

- Geotechnical/hydrogeological assessments: Include drill program to confirm rock types/conditions along azimuth of proposed ramp route; assess rock mass ratings for mineralized material zone and wallrocks, and determine allowable stope spans; assess likely groundwater inflows to confirm dewatering capacity required and potential for negatively impacting on development rate and cost.
- Waste rock: A trade-off study may be appropriate to determine how to practically and economically transfer waste underground, whether via ejector-box trucks on back-haul or comparable alternative or via waste passes from surface.
- Production rate: Once resource categories have been upgraded and the mineralized material zone tonnage and geometry is known with greater confidence, undertake trade-off studies to determine optimal mine production rate as part of prefeasibility or final feasibility study; as confidence in distribution of higher mineralized material grade areas is increased, re-assess opportunity to selectively mine higher grade in early years of mine life.

- Key equipment selection: Trade-off studies for key equipment; truck simulation model to confirm number and size of haul trucks required, optimal frequency for laybyes.
- Backfill study: Determine particle size distribution of preliminary tailings product; mineralogical/chemical characterization to assess suitability of Black Butte Copper tailings for use as paste fill or dense fill; humidity cell tests to determine potential for self-heating of tailings if used as backfill; fill strength tests to determine if pyritic content of tailings may cause longer-term degradation of backfill strength and/or how to mitigate with binder type and content.
- Ventilation: Model the mine at various stages to optimize the size of the main ventilation drifts and raises, optimize main fan and auxiliary fan configurations such as centrifugal versus vane-axial fans on surface; determine strategy to maintain flexibility of ventilation system over the mine life.
- Overall risk assessment: Conduct risk assessment sessions for the mine development and operation plan to characterize key risks and prioritize follow-up engineering and other work to mitigate.

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 succeed with environmental baseline studies and permitting efforts by completing the thorough program they have initiated, in consultation with key stakeholders and regulatory agencies.

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28.0 CERTIFICATES OF QUALIFIED PERSON

ARTHUR H. WINCKERS, P.ENG.

I, Arthur H. Winckers, P.Eng., of North Vancouver, British Columbia, do hereby certify:

- I am a Consulting Mineral Processing Engineer and 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 Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (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. 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 28.0 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 30th day of August, 2012 at North Vancouver, British Columbia.

"Original document signed by Arthur H. Winckers, P.Eng."

Arthur H. Winckers, P.Eng.
President
Arthur H. Winckers & Associates Inc.

GEORGE DARLING, P.ENG.

I, George Darling, P.Eng., of Sudbury, Ontario, do hereby certify:

- I was a Senior Consultant with Stantec Consulting Ltd. with a business address at 1760 Regent Street, Sudbury, Ontario P3E 3Z8.
- This certificate applies to the technical report entitled Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (the "Technical Report").
- I am a graduate of Queen's University (Degree in Mining Engineering, 1976) and Sydney University (B.Sc. in Geology and Geophysics, 1970). I am a member in good standing of the Professional Engineers of Ontario (#10497014). I have practiced my profession for 35 years with experience in mine engineering, mine operations, and mine organization structure. My areas of expertise include NI 43-101 reports, mine feasibility studies, mine financial evaluations, life of mine planning, mining trade-off studies, mine construction management, EPCM, mine organizational structures and due diligence reports. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was June 8 to 9, 2011.
- I am responsible for Sections 1.10, 15.0, 16.0, 21.2, 21.3, 21.4.2, 26.5, 27.0 (mining only), and 28.0 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 30th day of August, 2012 at Sudbury, Ontario.

"Original document signed by George Darling, P.Eng."

George Darling, P.Eng.
Senior Consultant
Stantec Consulting Ltd.

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 Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (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 20 years in mine expansion, capital cost engineering for both green and brownfield construction, planning, costing and execution of mine/concentrate handling facilities including plant, road, rail and port and the preparation of studies. 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, 1.11 (except for 1.11.1), 1.14.1, 1.16, 1.17, 2.0, 3.0, 18.1 to 18.3, 18.11, 19.0, 21.0 (except 21.2 to 21.4), 24.0, 26.1, 26.6, and 28.0 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 30th day of August, 2012 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.

JIANHUI (JOHN) HUANG, P.ENG.

I, Jianhui (John) Huang, 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 Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (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 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 1.12, 1.14.2, 17.0, 21.4.1, 21.4.3, 21.4.4, 26.3.2, and 28.0 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 30th day of August, 2012 at Vancouver, British Columbia.

"Original document signed and sealed by Jianhui (John) Huang, P.Eng."

Jianhui (John) Huang, P.Eng.
Senior Metallurgist
Tetra Tech WEI Inc.

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 Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (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 is includes environmental baseline studies, water management planning, environmental monitoring, mine permitting and mine closure. 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 1.11.1, 18.4 to 18.10, 21.4.5, 26.4, and 28.0 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 30th day of August, 2012 at Vancouver, British Columbia.

"Original document signed and sealed by Ken Brouwer, P.Eng."

Ken Brouwer, P.Eng.
Managing Director
Knight Piésold Ltd.

LISA KIRK, P.G., PH.D.

I, Lisa Bithell Kirk, P.G., Ph.D., 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 Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (the "Technical Report").
- I am a graduate of University of Pennsylvania (B.A. in Geology/ Environmental Science, 1983), University of Colorado (M.S. in Geochemistry, 1990), Montana State University (PhD in Microbial Geochemistry, 2012). 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 mined materials. 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, 25.5, 26.7, 27.0 (environmental only), and 28.0 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 30th day of August, 2012 at Bozeman, Montana

"Original document signed and sealed by Lisa Bithell Kirk, P.G., Ph.D."

Lisa Bithell Kirk, P.G., Ph.D.
Principal Geochemist
Enviromin, Inc.

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 Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (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 (14 years). 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 28.0 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 30th day of August, 2012 at Stites, Idaho.

"Original document signed and sealed by Michael J. Lechner, P.Geo."

Michael J. Lechner, P.Geo.
President
Resource Modeling Inc.

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 Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana, dated August 30, 2012 (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 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, and 28.0 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 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 30th day of August, 2012 at Vancouver, British Columbia

"Original document signed and sealed by Sabry Abdel Hafez, P.Eng."

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.