Report to:

Silver Bear Resources Inc.

NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation

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SILVER BEAR RESOURCES INC.

NI 43-101 TECHNICAL REPORT, MANGAZEISKY SILVER PROJECT, REPUBLIC OF SAKHA (YAKUTIA), RUSSIAN FEDERATION

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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)	USD
dollar (Canadian)	CAD
dry metric ton	dmt
foot	ft
gallon	gal
gallons per minute (US)	gpm
Gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g

grams per litre	g/l
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilovolts	kV
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	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
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
Ruble (Russian)	RUB
second (plane angle)	"
second (time)	s
short ton (2,000 lb)	st
short tons per day	st/d
short tons per year	st/y
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
three-dimensional	3D
tonne (1,000 kg) (metric ton)	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

Abbreviations and Acronyms

AACE International	AACE
acid rock drainage	ARD
acid rock drainage	ARD
acid-base accounting	ABA
acid-generating potential	AP
acid-generating potential	AP
Administration for Subsoil Use in the Republic of Sakha-Yakutia	YAKUTNEDRA
all-terrain vehicle	ATV
ammonium nitrate fuel oil	ANFO
atomic absorption	AA
Bond Abrasion Index	BA_i
Bond Ball Mill Work Index	BW_i

calcium carbonate	$CaCO_3$
calcium hypochlorite	$Ca(OCl)_2$
calcium oxide	CaO
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian Security Administrators	CSA
Chemical Laboratory of the State Enterprise Aldangeologiya	Aldan Lab
chlorine	Cl
comma separated values file	.csv
cost breakdown structure	CBS
counter-current decantation	CCD
Crusher Work Index	CW_i
cumulative net cash flow	CNCF
cyanide weak acid dissociable	CN_{WAD}
Differential Global Positioning Systems	DGPS
engineering, procurement and construction management	EPCM
Enterprise resource plan	ERP
Environmental and Social Impact Assessment	ESIA
environmental protection plan	EPP
Environmental Resources Management Group Inc.	ERM
exploratory data analysis	EDA
Federal State Unitary Enterprise Central Institute of Science Research and Geological Surveying of Base and Precious Metals	TSNIGRI
Federal Subsoil Resources Management Agency	ROSNEDRA
Free Carrier	FCA
general and administration	G&A
General Seismic Zonation	GSZ
Geospark Core Database System™	Geospark
GEOVIA Surpac	Surpac
global positioning system	GPS
Gosudarstvennyy Standart	GOST
gravity recoverable silver	GRS
greenhouse gas	GHG
hazard and operability	HAZOP
Health, safety, security and environment	HSSE
heavy liquid separation	HLS
high-density polyethylene	HDPE
Important Bird and Biodiversity Area	IBA
inductively coupled plasma	ICP
input/output	I/O
Intensive Leach Reactor	ILR
internal rate of return	IRR
International Building Code	IBC
International Electrotechnical Commission	IEC
International Finance Committee	IFC
International Finance Corporation	IFC
International Organization for Standardization	ISO
inter-ramp angles	IRA
inverse distance weighting squared	IDW^2

ion exchange	IX
Key performance indicator	KPI
lead nitrate	$PbNO_3$
lead	Pb
life-of-mine	LOM
load-haul-dump	LHD
local area network	LAN
London Bullion Market Association	LBMA
material take-off	MTO
maximum permissible concentrations	MPC
Medvedev-Sponheur-Karnik	MSK
metal leaching	ML
Mineable Shape Optimizer	MSOw
Mineral Resources Extraction Tax	MRET
Modified Mercalli Intensity	MMI
motor control centre	MCC
multiple linear regression	MLR
National Institute of Non-ferrous Metals	GINTSVETMET
National Instrument 43-101	NI 43-101
National Oceanic and Atmospheric Administration	NOAA
Neryungristroiizyskaniya	NSI
net cash flow	NCF
net neutralization potential	NNP
net present value	NPV
neutralization potential	NP
neutralizing potential	NP
non-potentially acid generating	NPAG
ordinary kriging	OK
peak ground acceleration	PGA
personal protective equipment	PPE
polyvinyl chloride	PVC
potentially acid generating	PAG
preliminary economic assessment	PEA
PricewaterhouseCoopers	PwC
programmable logic controller	PLC
project execution plan	PEP
Qualified Person	QP
quality assurance	QA
quality control	QC
Quantitative Evaluation of Minerals by Scanning	QEMSCAN®
quantitative kriging neighbourhood analysis	QKNA
rock quality designation	RQD
run-of-mine	ROM
SGS Vostok Limited	SGS Vostok
Silver Bear Resources Inc.	Silver Bear
silver nitrate	$Pb(NO_3)_2$
silver	Ag
Special Protected Natural Area	SPNA

SRK Consulting (UK) Ltd.	SRK
standard reference material	SRM
State Unitary Enterprise	SUE
supervisory control and data acquisition	SCADA
System for Electronic Document Analysis and Retrieval	SEDAR
tailings management facility	TMF
TEO Conditsy Report	TEO
the Mangazeisky Project	the Project or the Property
TOMS Institute	TOMS
total cyanide	CN_{TOTAL}
total suspended solids	TSS
Ulyanovsk Automobile Plant	UAZ
uninterruptible power supply	UPS
Universal Transverse Mercator	UTM
Value Added Tax	VAT
variable speed drive	VSD
very-high frequency	VHF
volatile organic compound	VOG
waste rock dump	WRD
water balance model	WBM
wildlife management area	WMA
work breakdown structure	WBS
World Geodetic System	WGS
World Health Organisation	WHO
x-ray diffraction	XRD
x-ray fluorescence	XRF
ZAO Prognoz	Prognoz
zinc	Zn
ООО “НСИ”	NCI

1.0 SUMMARY

1.1 INTRODUCTION

Silver Bear Resources Inc. (Silver Bear) retained Tetra Tech to complete a Technical Report of the Mangazeisky Project, located in Far East Russia. Silver Bear is a public limited company incorporated under the *Business Corporations Act* of the Province of Ontario, Canada, on 8th April 2004. Silver Bear is listed on the Toronto Stock Exchange (TSX) under the symbol SBR. Effective June 30, 2017, Silver Bear completed a redomicilliation transaction whereby Silver Bear was acquired by Silver Bear Resources plc, an English company, which now continues as the parent company. All references to Silver Bear after June 30, 2017 are to Silver Bear Resources plc.

In compliance with National 43-101 (NI 43-101) guidelines, this Technical Report includes as of the effective date, all material scientific and technical information with respect to Silver Bear's Mangazeisky Property and therein presents the current status and economic outlook of the Vertikalny deposit, re-states the current Mineral Resources for the Nizhny Endybal and Mangazeisky North and South deposits, and includes a summary of the Mangazeisky North deposit economic opportunity (Section 24.4) at its potential economic viability, which would occur if its Mineral Resources (other than Inferred Mineral Resources), are integrated into the Vertikalny mine plan.

The effective date of this study is 20th August 2017, the effective date of the Vertikalny Mineral Resource estimate is 8th July 2016 and the effective date of the Mangazeisky North Mineral Resource estimate is 19th October 2016. All currency is reported in US dollars (USD), unless otherwise noted.

The key outcomes of the Project are summarized in Table 1.1.

Table 1.1 Vertikalny Deposit Summary

Description	Unit	Value
Commodity	-	Ag
Mining Area	-	Vertikalny Deposit
Mining Method	-	Open Pit/Underground
Average Head Grade (LOM)	g/t Ag	807
LOM	years	7.8
Processing Method	-	Cyanide Leach with Direct Electrowinning
Product	-	Silver Powder (99.8% Ag)
Metallurgical Recovery	%	Oxide – 78.0 Primary – 69.6
Total Initial Capital	$ million	65.0
Total LOM Capital	$ million	82.1

table continues...

Description	Unit	Value
Operating Cost	$/t processed	153.79
Silver Price (LOM Weighted Average)	$/t	19.76
Exchange Rate	RUB:USD	66.00:1.00
Pre-tax IRR	%	48.2
Pre-tax NPV (5%)	$ million	88
Pre-tax Payback (undiscounted)	years	1.9
Post-tax IRR (without the Far East Tax Incentives)	%	28.4
Post-tax NPV (5%) (without the Far East Tax Incentives)	$ million	46
Post-tax Payback (undiscounted) (without the Far East Tax Incentives)	years	2.4
Post-tax IRR (with the Far East Tax Incentives)	%	44.3
Post-tax NPV (5%) (with the Far East Tax Incentives)	$ million	77
Post-tax Payback (undiscounted) (with the Far East Tax Incentives)	years	2

Notes: Ag – silver; LOM = life-of-mine; IRR = internal rate of return; NPV = net present value; USD – US Dollar; RUB – Russian Ruble

Tetra Tech led a group of qualified consulting companies, commissioned by Silver Bear, to complete component studies for the purposes of this Technical Report. Table 1.2 outlines the responsibilities of each company.

Table 1.2 Consulting Company Responsibilities

Company	Responsibility
Tetra Tech	overall project management; property description and location; accessibility, climate, local resources; infrastructure and physiography; history; geological setting and mineralization; deposit type; exploration; drilling; sample preparation, analyses and security; data verification; adjacent properties; mineral resource estimates; mineral processing and metallurgical testing; recovery methods, logistical infrastructure; site layout; site access; site geotechnical; structural and architectural design basis; site roads; processing facility infrastructure; site facilities; remote site facilities; waste rock dumps, ore stockpiles and overburden storage area; site water management; waste and sewage; site storage; power supply and distribution; processing and general and administrative capital and operating cost estimates; economic analysis; project execution; mineral reserve estimates; mining methods; mining capital and operating costs
ERM	environmental studies, permitting, and social or community impact
SRK	mining geotechnical evaluation; TMF; tailings capital and operating costs; hydrology and hydrogeology

Note: ERM = Environmental Resources Management Group Inc.; SRK = SRK Consulting (UK) Ltd.; TMF – tailings management facility

1.2 PROPERTY DESCRIPTION AND LOCATION

The Property is located approximately 400 km north of Yakutsk, in the Kobyask district, within the Republic of Sakha (Yakutia) (Figure 1.1). The Property longitude and latitude is approximately 65°40' south and 130°07' east.

The Property is defined by the mineral rights to a single mining license, and is 100% held by Prognoz, a private Russian closed-joint stock company based in Yakutsk. Prognoz is a 100% held subsidiary of Silver Bear.

The mining license, number YaKU 03626 BE, covers the entire Vertikalny silver deposit over an area of 13.55 km^2. The coordinates of the mining licenses are shown in Table 1.3.

Figure 1.1 Property Location Map

Source: Silver Bear

Table 1.3 Mining License Coordinates

Mining License YaKU 03626 BE		
Corner No.	Northing Coordinate	Easting Coordinate
1	65°41'15.917"	130°01'55.380"
2	65°41'41.938"	130°03'23.150"
3	65°41'37.066"	130°04'59.859"
4	65°41'20.210"	130°06'27.196"
5	65°40'08.102"	130°08'20.361"
6	65°39'44.803"	130°08'11.742"
7	65°39'40.272"	130°07'17.802"
8	65°39'46.221"	130°05'22.190"
9	65°39'54.675"	130°03'29.389"
10	65°40'11.350"	130°01'57.673"
11	65°40'46.388"	130°01'42.001"

The mining license is situated in the central west portion of the Endybal exploration license (also known as the Mangazeisky exploration license), number YaKU 12692 BP, and covers several other exploration targets and silver deposits over an area of 570 km^2. The coordinates of the exploration licence are shown in Table 1.4.

Table 1.4 Exploration License Coordinates

Exploration License YaKU 12692 BP		
Corner No.	Northing Coordinate	Easting Coordinate
1	65°49'35"	130°00'00"
2	65°49'35"	130°19'20"
3	65°29'00"	130°22'00"
4	65°29'00"	130°00'00"

A summary of all existing licence information is detailed in Table 1.5.

Table 1.5 Property Mineral Licences

Licence Name	Licence ID	Type	Area (km^2)	Issue Date	Expiry Date	Annual Fees (RUB)
Endybal Area (Mangazeisky)	YaKU 12692 BP	Geological Allotment	570.00	28th September 2004	31st December 2023	150,242
Vertikalny Deposit	YaKU 03626 BE	Licence to Use Subsoil	13.55	31st August 2013	1st September 2033	110,771

1.3 GEOLOGY AND MINERALIZATION

1.3.1 REGIONAL GEOLOGY

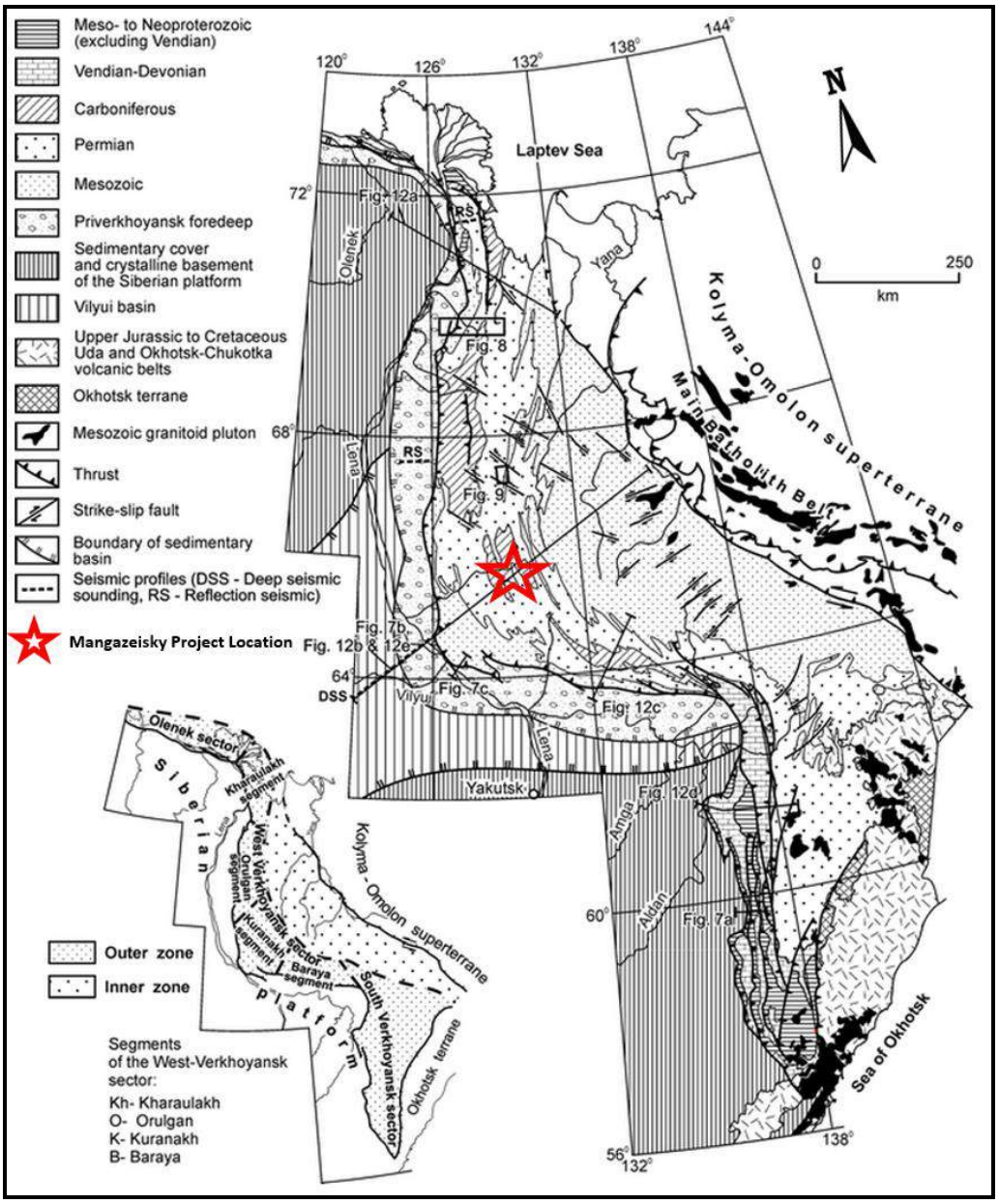
The Property is located within the Verkhoyansk mobile belt of north-eastern Yakutia. The fold-and-thrust belt forms part of a major orogenic system separating the Siberian North Asian Craton to the west from the immense expanse of accreted terranes that form most of the Russian Far East.

The belt extends for 2,000 km from the Laptev Sea to the Sea of Okhotsk. It is comprised of a greater than 7 km thick sequence of Late Precambrian to Triassic rocks deposited along the Palaeo-Pacific margin of the Siberian Craton, which developed because of rifting events during the Late Precambrian period and again during the Late Devonian to Early Mississippian periods.

Deformation during the Late Jurassic to Early Cretaceous occurred with low-grade metamorphism in the internal parts of the belt and the emplacement of high-level granitic bodies. Strike-slip faulting occurred during the Tertiary period.

The central part of the belt is dominantly comprised of a thick, monotonous succession of Carboniferous and Permian turbidites that are metamorphosed to lower-greenschist facies. Granodiorite and granite plutons have intruded the core of the belt and are associated with extensive gold-bearing quartz vein systems. Figure 1.2 presents the regional geology of the area.

Figure 1.2 Regional Geology Map



Source: Miller et al. (2002)

1.3.2 PROPERTY GEOLOGY

The mineral occurrences on the Property are located in the interfluve of the Nuektame and Arkachan rivers and range from the mid-channel of the Muhalkan River to the mouth of the Mysovoi Stream. The mineral deposits are confined to the eastern wing of the Kuranakhsky anticlinorium and are situated within the contours of the Endybal anticline.

Higher order asymmetric anticlinal and synclinal folds, that are 2 to 20 km in length, complicate the hinge portion of the Endybal anticline. The Mukhalkansky anticline with a north-northwest strike (340°) and the Mangazeisky syncline, which are adjacent on the western side, complicate the eastern wing of the Endybal anticline.

The dip of the rocks of the Endybal anticline in the area of the Mangazeisky deposits averages 20 to 45°.

Two north-south zones of faulting are distinguished. The most important is the Nuektaminsky shear zone, which limits the expansion of mineralisation to the east.

The movement along the Nuektaminsky fault is left-lateral. The next is the Granichny shear zone with right-lateral movement. The Granichny fault is a splay off the Nuektaminsky fault and together they form a V-shape, which represents a zone of tension. The Granichny shear zone hosts two vertical deposits (Semenovsky and Granichny) that cross-cut the stratigraphy.

The silver mineralisation is epigenetic. This is particularly clear at the Nizhny Endybal deposit, where the mineralised strata-bound zones exhibit structural deformation. The strata-bound veins and cross-cutting fracture systems in the region are associated with shearing. This is reflected in the common occurrence of parallel layers of slip planes along the boundaries of the sandstone beds and selvages of strata-bound veins, such as Mikhailovsky, Olgina, Vasilievsky, and Anglesite-Cerussite. The structural analysis of joints in the mineralised rocks shows the ubiquitous existence of overthrusting dislocations (Kostin et al. 1997).

The Endybal diatreme and quartz porphyry dykes postdate the overthrust event. The geological studies by Yangeologia (2000) and Kostin et al. (1997) have led to the following conclusions regarding the sequence of events:

1. mineralised layers of sandstone containing sulphide mineralisation
2. structural deformation
3. Endybal diatreme.

1.3.3 MINERALISATION

This region of Yakutia is known to contain many silver deposits. Figure 1.1 illustrates the position of the Property in relation to a number of the known silver projects in the region.

The Property contains several explored areas that host more than 100 occurrences of mineralisation. The mineral occurrences are concentrated within a 35 km long corridor. These main mineralised areas include: Yasny, Burny, Mukhalkansky, Zabyty, Zabyty-2, Mangazeisky, Strezhevoy, Vertikalny, Porfirovy-Borisovsky, Vostochny, Nizhny Endybal (NizhneEndybal), VerkhneEndybal, Jilny, Bezymyanny, Orogondia, and Kis-Kuel.

The Property hosts at least four different types of mineralisation:

- Strata-bound silver-bearing, quartz-carbonate-sulphide structures with average grades greater than 900 g/t silver, with lead and zinc by-products. These silver bearing structures were mapped and sampled over an area of 65 km^2. Examples of this are the Vasilievsky-Anglesite-Cerussite and Olgina-Mikhailovsky veins within the Mangazeisky North Zone.

- Thicker linear-type stockwork areas with carbonate-silver sulphosalt mineralisation. Examples of this occur in the Strezhevoy and Nizhny Endybal Zones.
- Narrow, steep dipping veins such as Vertikalny that cross-cut stratigraphy and feature grades in excess of 1,000 g/t silver over widths ranging from several centimetres to several metres. Vertikalny and possibly Zabytoe and Kis-Kuel are examples of this style of mineralisation.
- A porphyry area associated with quartz, quartz-carbonate and quartz-sulphide veins and veinlets, hosted by extrusive rhyolite porphyry. Porfirovy is an example of this.

1.4 EXPLORATION

Table 1.6 provides a summary of the exploration completed since 2007.

Early exploration focused upon the narrow, strata-bound silver mineralisation of the Vasilievsky and Mikhailovsky targets, with more of a focus on Vertikalny from 2007 onwards. Latterly, the exploration has incorporated the evaluation of the Sterzhnevoy, Nizhny Endybal, and Mangazeisky North deposits.

Table 1.6 Historic Exploration Activities at the Property

Year	Exploration Activities	Targets Explored
2004	No exploration was undertaken during 2004, other than a minor drilling campaign.	-
2005	9,641 m³ of trenching.	Vasilievsky, Mikhailovsky, Sterzhnevoy, Nizhny Endybal
2006	4,843 m³ of trenching and mapping.	Nizhny Endybal, Vostochny, Sterzhnevoy, Vertikalny
2007	8,000 m³ of trenching.	Vertikalny
2008	22,633 m³ of trenching. Mapping, lithochemical sampling, direct current induced polarisation/magnetotellurics and magnetic anomaly geophysical surveys.	Vertikalny, Zabyty, Zabyty-2, Kis-Kuel, Orogondia
2009	15,067 m³ of trenching. Lithochemical sampling, magnetic anomaly mapping.	Nizhny Endybal, Vertikalny, Kis-Kuel, Mukhalkan-Burny
2010	No exploration was undertaken during 2010.	-
2011 to 2012	1,600 m³ of trenching.	Nizhny Endybal
2013	11,200 m³ of trenching.	Mangazeisky North
2014	11,148 m³ of trenching.	Vertikalny, Mangazeisky, Porfirivoy, Sterzhnevoy.
2015	13,100 m³ of trenching	Vertikalny, Porfirivoy & Sterzhnevoy.

Tetra Tech is of the opinion that the exploration has generally been conducted using methods and procedures that are consistent with recognised industry best practices. Some bias was noted when comparing the trenches with nearby drillholes and as a result some of the trenches were re-sampled using a circular saw to cut channels. Following the re-sampling, the bias remained and it is therefore thought that this increase in grade is associated with some natural mechanical enrichment at surface. The trenches have been used for grade estimation in the Mineral Resource. All trenches are now channelled with circular saw.

1.5 DRILLING

Details about the drilling campaigns completed prior to 2010 have been disclosed fully in the Wardrop (2011). The 2011 to 2012 drilling campaigns have been disclosed fully in Tetra Tech (2013). The details of the drilling at Mangazeisky North completed during 2015 are disclosed in the 2016 Technical Report (Tetra Tech 2016). All of these reports are filed under Silver Bear's profile on SEDAR. Table 1.7 provides a summary of the drilling completed between 2004 and 2015. The 2015 drilling campaign at Vertikalny Central is described fully in Section 10.0.

Table 1.7 Summary of Drilling on the Property

Year	Metres Drilled	Targets Drilled
2004	151	Vertikalny
2005	3,370	Vasilievsky, Mikhailovsky, Nizhny Endybal, Strezhevoy
2006	732	Vertikalny
2007	3,100	Vertikalny
2008	11,663	Vertikalny, Semenovsky, Vasilievsky
2009	12,373	Vertikalny
2010	-	No drilling undertaken
2011 to 2012	13,445	Vertikalny, Nizhny Endybal
2013 to 2014	9,153	Mangazeisky, Vertikalny
2015	4,427	Vertikalny North West, Mangazeisky North, Porphivory, Strezhevoy

In 2015, 12 holes were drilled at Mangazeisky North for a total drill length of 480 m. The 2015 drilling campaign at Mangazeisky North is described fully in Section 10.0.

Tetra Tech is of the opinion that drilling has generally been completed using methods and procedures that are consistent with recognised industry practices and that the data is adequate for Mineral Resource estimation.

1.6 MINERAL PROCESSING AND METALLURGICAL TESTING

As part of the feasibility study, mineralogical and metallurgical test work was completed at two commercial laboratories; SGS Vostok Limited (SGS Vostok) (Chita, Russia) and TOMS Institute (TOMS) (Irkutsk, Russia).

SGS Vostok tested a composite sample, which comprised samples collected from dedicated metallurgical drillholes within the Central Zone of the orebody representing the oxide ore. The SGS Vostok test work was completed between January 2014 and December 2014.

TOMS completed two separate test work programs in 2015. TOMS initially completed a leach variability test program in February 2015, followed by leach optimisation and comminution test work for a primary ore composite. The primary ore testing was completed between November 2015 and December 2015.

Tetra Tech analysed all the metallurgical test work results with the objective of identifying the optimal process flowsheet. The metallurgical studies indicate that the oxide ore is amenable to gravity concentration and cyanidation and can be efficiently processed using a gravity and agitated tank leach based flowsheet. However, the process flowsheet has been simplified by adopting a ROM leach flowsheet[1]. The studies also indicate that the primary ore is amenable to cyanidation and can be effectively processed by a whole ore leach method.

Based on the metallurgical and mineralogical test work, the average silver recovery is expected to be 78.1% for oxide ore and 69.6% for primary ore.

1.7 MINERAL RESOURCE ESTIMATES

1.7.1 SUMMARY

Tetra Tech completed a new Mineral Resource estimate for the Mangazeisky North deposit, with an effective date of 19th October 2016. The most recent data included in the estimate was received on 19th October 2016. Mr. Joseph Hirst, BSc (Hons), MSc, EurGeol, CGeol, an independent QP as defined by NI 43-101, estimated the Mineral Resources.

Tetra Tech completed a new Mineral Resource estimate for the Vertikalny deposits, with an effective date of 8th July 2016. The most recent data included in the estimate was received on 1st July 2016. Mr. Robert Davies, BSc, EurGeol, CGeol, an independent QP as defined by NI 43-101, estimated the Mineral Resources.

This section also presents a restatement of the other Mineral Resources within the Property. Table 1.8 presents a summary status of the Mineral Resource estimates.

[1]the installation of gravity concentration circuit has been deferred as a result of logistical issues

Table 1.8 Summary of Resource Estimates on the Mangazeisky Property

Deposit	Qualified Person	Effective Date
Nizhny Endybal	Mr. Robert Davies, BSc, EurGeol, Cgeol	11th September 2012
Mangazeisky North	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	19th October 2016
Mangazeisky South	Mr. Robert Davies, BSc, EurGeol, Cgeol	3rd June 2015
Vertikalny Central	Mr. Robert Davies, BSc, EurGeol, Cgeol	8th July 2016
Vertikalny Northwest	Mr. Robert Davies, BSc, EurGeol, Cgeol	8th July 2016
Sterzhnevoy	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	27th August 2016
Pophirovy	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	27th August 2016

Further details of the Project Mineral Resource estimates are publically available in Technical Reports prepared by Tetra Tech and filed under the Silver Bear profile on the CSA SEDAR website.

A summary of the current Mineral Resources for the Property are presented in Table 1.9.

Table 1.9 Total Resources for the Mangazeisky Property

Zone	Indicated Resource			Inferred Resource		
	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)
Vertikalny Central	700,000	1,227	27,700,000	350,000	786	8,900,000
Vertikalny Northwest	-	-	-	200,000	476	3,100,000
Nizhny Endybal	-	-	-	710,000	316	7,200,000
Mangazeisky North	334,000	770	8,250,000	127,000	560	2,300,000
Mangazeisky South	-	-	-	60,000	246	500,000
Sterzhnevoy	-	-	-	48,000	1,530	2,360,000
Total	**1,034,000**	**1,079**	**35,950,000**	**1,495,000**	**504**	**24,360,000**

Zone	Inferred Resource						
	Tonnes (t)	Grade Au (g/t)	Grade Ag (g/t)	Grade Cu (%)	Contained Metal Au (troy oz)	Contained Metal Ag (troy oz)	Contained Metal Cu (lb)
Porphirovy	80,000	1.7	46	0.6	4,000	115,000	1,000,000

Notes: Vertikalny deposits have a higher cut-off grade as they are assumed to be worked predominantly via underground mining methods.

1.7.2 SUMMARY OF ESTIMATION TECHNIQUES FOR THE VERTIKALNY DEPOSIT

Tetra Tech created wireframe models using Leapfrog Geo (version 3.1). Wireframes were created to represent lithologies, oxidation state, and mineralised bodies. Wireframes were developed for strongly mineralised core zones and where present, less strongly mineralised wall rock alteration (as described in Section 14.2.1). Variography, block modelling and Mineral Resource estimation were completed in GEOVIA Surpac™ software (version 6.7).

The silver grades for the Vertikalny Central and Northwest zones were estimated using Ordinary Kriging (OK) interpolation methodology. Density for the Central Zone was estimated using Inverse Distance Weighted Squared (IDW^2), while density was assigned for the Northwest Zone.

Statistical and grade continuity analyses were completed in order to characterise the mineralisation, and were subsequently used to develop grade interpolation parameters. Grade estimation was completed using OK. The search ellipsoid dimensions and orientations were chosen to reflect the continuity revealed by geostatistical studies. Block size, discretisation, search size, and sample numbers were optimised using Quantitative Kriging Neighbourhood Analysis (QKNA).

Tetra Tech adopted the definition of Mineral Resources as outlined within the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014) in order to classify the Mineral Resources.

In order to demonstrate that the deposit has reasonable prospects for economic extraction, a cut-off grade of 200 /t silver has been applied for Mineral Resources contained within an open pit shell at Vertikalny Central, or within 80 m of surface at Vertikalny Northwest. A 350 g/t silver cut-off has been applied for the underground Mineral Resource, based upon the following assumptions:

- silver price of USD17.00/troy oz
- mining cost USD2.53/t open pit and USD70.00/t underground
- processing cost of USD52.00/t
- general and administrative (G&A) costs of USD40.60/t.

At Vertikalny Central, a pit shell wireframe was used to constrain the open pit resource in order to demonstrate that the resource has reasonable prospects for economic extraction. The pit shell was developed using the following assumptions:

- silver price of USD19.00/troy oz
- mining recovery 95%
- waste dilution 30%
- mining cost of USD2.53/t
- processing cost of USD52.00/t
- G&A costs of USD40.60/t

- royalty of 6.5%
- overall silver recovery of 88%.

1.7.3 SUMMARY OF ESTIMATION TECHNIQUES FOR THE MANGAZEISKY NORTH DEPOSIT

Tetra Tech created wireframe models using Datamine Studio 3 (version 3.24) software (Datamine) to represent mineralised bodies. Block modelling and Mineral Resource estimation were also completed in Datamine.

The metal grades for the Mangazeisky North deposit were estimated using Ordinary Kriging (OK) interpolation method. Density was estimated using and Inverse Distance Weighting Squared (IDW^2) interpolation methodology.

Statistical and grade continuity analyses were completed in order to characterise the mineralisation and were subsequently used to develop grade interpolation parameters. The search ellipsoid dimensions and orientations were chosen to reflect the continuity revealed by geostatistical studies. Block size, discretisation, search size, and sample numbers were optimised using Quantitative Kriging Neighbourhood Analysis (QKNA).

Tetra Tech adopted the definition of Mineral Resources as outlined within the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014) in order to classify the Mineral Resources for the Mangazeisky North deposit.

In order to demonstrate that the deposit has reasonable prospects for eventual economic extraction, a cut-off grade of 150 g/t silver was applied for Mineral Resources within 60 m of surface, and was based on the following assumptions:

- silver price of $17.00/troy oz
- mining recovery 95%
- waste dilution 30%
- mining cost of $2.53/t
- processing cost of $52.00/t
- general and administration costs of $40.60/t
- royalty of 6.5%
- overall metal recovery of 88%.

1.8 MINERAL RESERVE ESTIMATES

The Mineral Reserve was estimated in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (prepared by the CIM Standing Committee on Reserve Definitions, Adopted by CIM Council on November 27, 2014). According to the CIM, a Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a prefeasibility study. A Mineral Reserve includes diluting materials and allowances for losses that

may occur when the material is mined. The Mineral Reserve is inclusive of the reported Mineral Resource.

Proven and Probable Mineral Reserves for the Vertikalny deposit are presented in Table 1.10.

Table 1.10 Vertikalny Deposit Mineral Reserve Statement – April 30th 2017

Category	Cut-off Grade (g/t Ag)	Ore (t)	Ag Grade (g/t)	Ag Metal Content (koz)
Probable – Open Pit	240	400	1,083	13,934
Probable – Underground	450	458	569	8,375
Total Mineral Reserves	-	**858**	**809**	**22,309**

1.9 MINING METHODS

1.9.1 OPEN PIT

GEOVIA Whittle™ software was used to carry out the pit optimisation and to define the optimal economic boundary for the open pit. GEOVIA GEMS™ software was also used for the pit and waste rock dump (WRD) designs and for preparation of the end-of-period pit plans and rock inventory estimation.

In both the optimisation and pit design the SRK geotechnical study and slope configuration results (Appendix B) were applied. Note that whereas the North Pit was designed using two phases, the South Pit has only one phase.

A sensitivity analysis on the selected pit shell for the base case shows that the orebody within the selected shell is most sensitive to process recovery and silver price, while the sensitivity of the cash flow to the mining cost and processing cost is minimal.

The total mineralised rock above cut-off grade (240 g/t silver) within the designed pits is 400 kt, at an average silver grade of 1,083 g/t. The total low-grade material and waste rock is 10.1 Mt and the total rock is 10.4 Mt. The North Pit contains 70% of total rock and 80% of silver in the open pits. The average grade of the orebody, within the designed North Pit, is approximately 25% higher than the average grade in the South Pit.

The LOM production schedule was developed by preparing approximate pit plans for the end of each monthly period for the first 20 months from May 2017 to the end of 2018, plus the quarterly pit plans from 2019 to 2020. The final mining period in 2020 will deplete the remaining rock in the North Pit. Once the rock inventory within the prepared plans was estimated, mining dilution and losses were applied, and the schedule was prepared until the end of 2020. From the start of 2021 the remaining low-grade stockpile (65 kt at 316 g/t silver) is drawn down as required.

The mining operation is scheduled to start in May 2017 with seven months of pre-stripping prior to full operation in January 2018. The ramp-up schedule for the

process plant production starts in December 2017 and is assumed to be a two month period, where a total of 14 kt will be delivered to the process plant. From January 2018, the process plant will reach the nominal capacity of 110 kt/a or 27.5 kt per quarter. The process plant throughput will be provided from direct haulage of run-of-mine (ROM) material, or from a combination of ROM and reclaimed stockpiled material. For the updated production schedule, the higher grade stockpiled ore will be utilised to delay the delivery of the lower grade ROM ore to the process plant, or to supplement feed when there is, insufficient exposed ore in the working benches during any given period.

The mining and ancillary fleet comprise 1.65- to 2.5-m^3 backhoe excavators, with a mix of 20-t and 40-t trucks, and a downhole drill capable of drilling 110-mm blastholes. Other earthmoving and support equipment have been estimated as ancillary equipment, requiring two drills, two excavators, up to 12 trucks, 3 dozers, and 2 road graders as the main mining equipment selected for this operation.

1.9.2 Underground Mining

Tetra Tech recommends mechanised sub-level open stoping, with remote stope cleaning, as the safest and most productive underground mining method for the Vertikalny deposit. This method is preferred as it is particularly suited for narrow vein, steeply dipping, hard rock mineralisation geometries. The implementation of this method has become more practical with the advent of mining equipment technologies that have are compact and can be operated remotely. Mine simulations show that slightly higher dilution can be expected in in sub-level open stoping when compared to shrinkage stoping, but that this drawback is overshadowed by the benefit of lower underground unit operational costs.

All lateral development will be completed using an electrically-powered, single boom jumbo drill rig, with ventilation raises developed using a long-hole drill rig (stoping drill rig). The underground Mineral Reserve will be exploited in two distinct underground mining areas in the north and south, each with each own independent ramp access. This takes into account the stopes' proximity to surface and the open pit, and the operational flexibility provided by mining multiple deposits with a single fleet of mobile equipment. For the South Zone, adit accesses were designed for some of the sub-levels, with access from a surface road and from within the South Pit. As the underground access ramps and adits will be tied into the open pit designs, the timing of their development will be contingent on the open pit mining schedule.

1.10 Recovery Methods

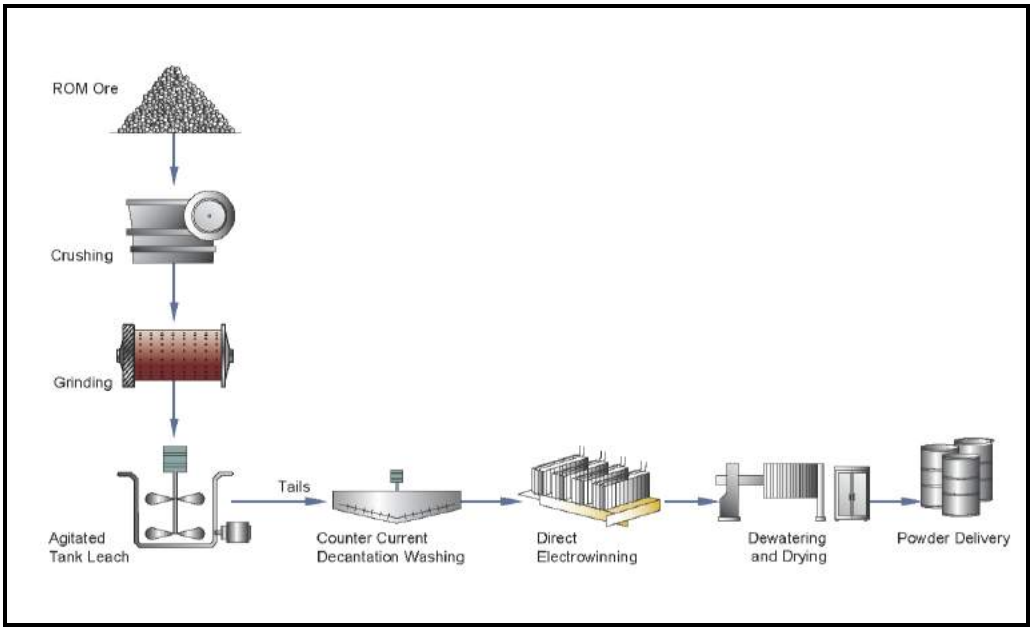
The process design was based on the completed metallurgical test work, with the objective of developing a flowsheet that is simple as possible to supply, operate, and maintain, while still maximising silver recovery and minimising power requirements.

The process plant is designed to treat 110,000 t/a of ore with an average LOM silver grade of 772 g/t. The plant is expected to achieve a 78.1% silver recovery oxide ore and 69.6% of fresh ore. The treatment of fresh ore will be deferred to the last year of the production schedule and not processed in combination with the oxidised material.

The process flowsheet consists of a standard crushing and milling circuit, followed by agitated tank leach of the ground slurry. The leached slurry from the tank leach will go through a simple counter current decantation washing system and the pregnant solution, at a LOM average silver grade of approximately 600 ppm, will be processed by direct electrowinning to recover silver metals.

A conceptual block flow diagram of the proposed process plant is shown in Figure 1.3.

Figure 1.3 Conceptual Process Block Flow Diagram



Source: Tetra Tech

Tailings from the process plant will comprise filtered solids and a blead stream from the depleted electrowinning solution, and will be disposed of in the TMF.

1.11 PROJECT INFRASTRUCTURE

1.11.1 GENERAL

The Project will require the development of a number of infrastructure components. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and ensure efficient and convenient operation of the mine haul fleet.

Project infrastructure will include:

- airstrip
- off-site, on-site, and service road access
- mine haul roads
- TMF
- WRD, crushed ore stockpile, and ROM pad

- camp flood protection berms
- borehole water supply and distribution system
- water treatment and distribution
- sewage collection and management
- surface water management
- process and ancillary facilities, including:
  - process plant
  - crushing facility
  - reagent storage and handling
  - facilities for administration and an assay laboratory
  - heavy and light vehicle workshops
  - fuel storage and distribution
  - explosives storage and handling
  - emergency helipad
  - camp accommodation
  - mess and laundry facilities
  - medical centre
- power supply and local distribution network
- communications infrastructure.

The site layout (Figure 1.4) displays the location and orientation of the three main mine site components:

- the Vertikalny open pit area and underground mine
- the process plant facility
- the supporting site infrastructure which encompasses all roads, buildings, facilities, and structures that support—but do not directly contribute to—the primary purpose of the mining operation.

Figure 1.5 shows the layout of the process plant complex.

1.11.2 TAILINGS MANAGEMENT FACILITY

The TMF will consist of a dry stack facility, contained within a fully-lined pad, surrounded by a series of containment bunds. A clarification pond to store all process affected fluids before retreatment is included in the design.

SRK undertook a TMF site selection study, in which a series of four alternative locations were assessed for development. A waste rock and tailings co-disposal option was also considered as part of this assessment.

The TMF will be constructed 0.2 km northeast of the plant site, and will cover an area of 7.69 ha (Figure 1.5). Approximately 0.8 Mt of tailings material will require storage over the LOM.

Figure 1.4 Site Infrastructure Layout

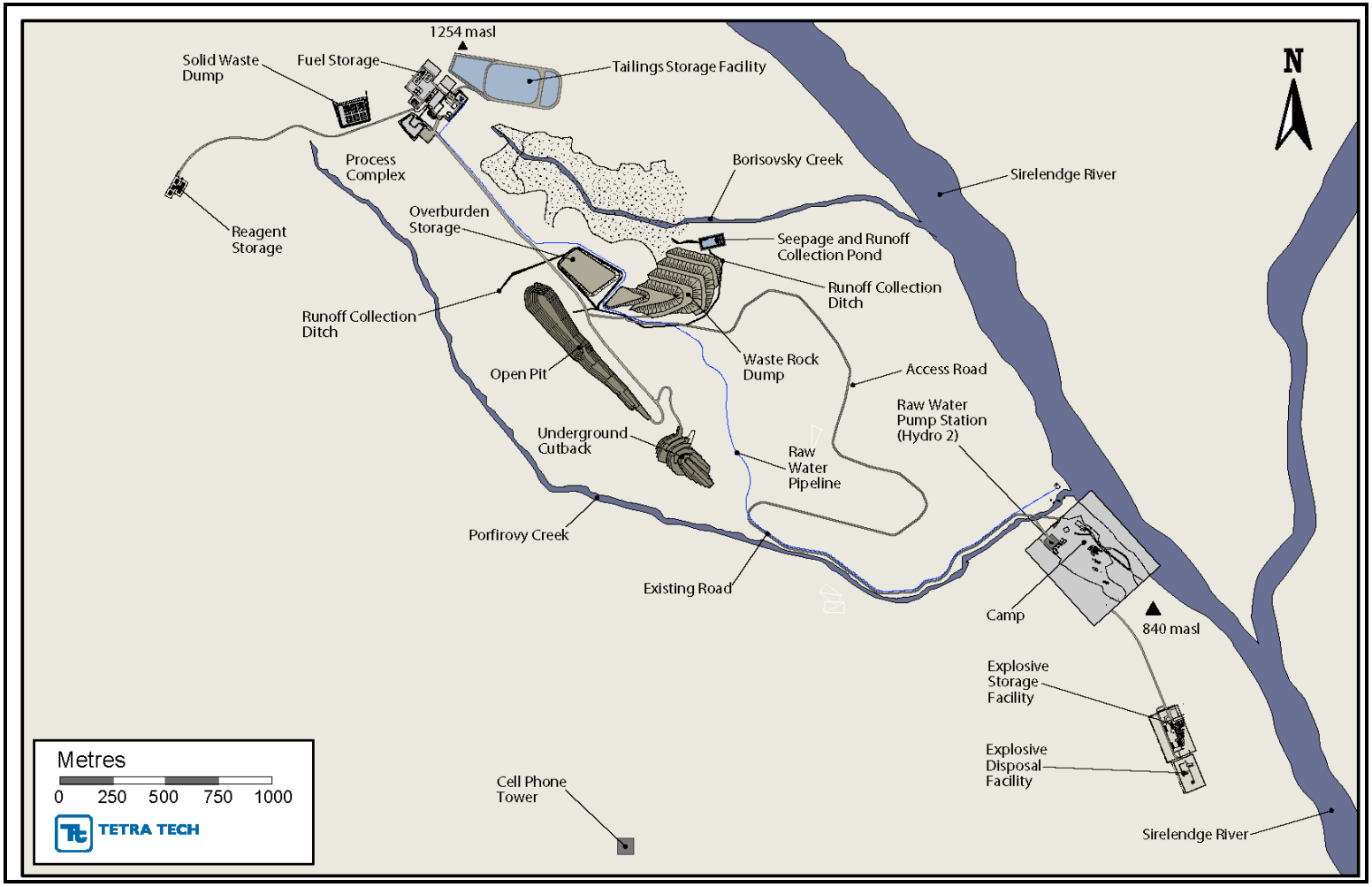
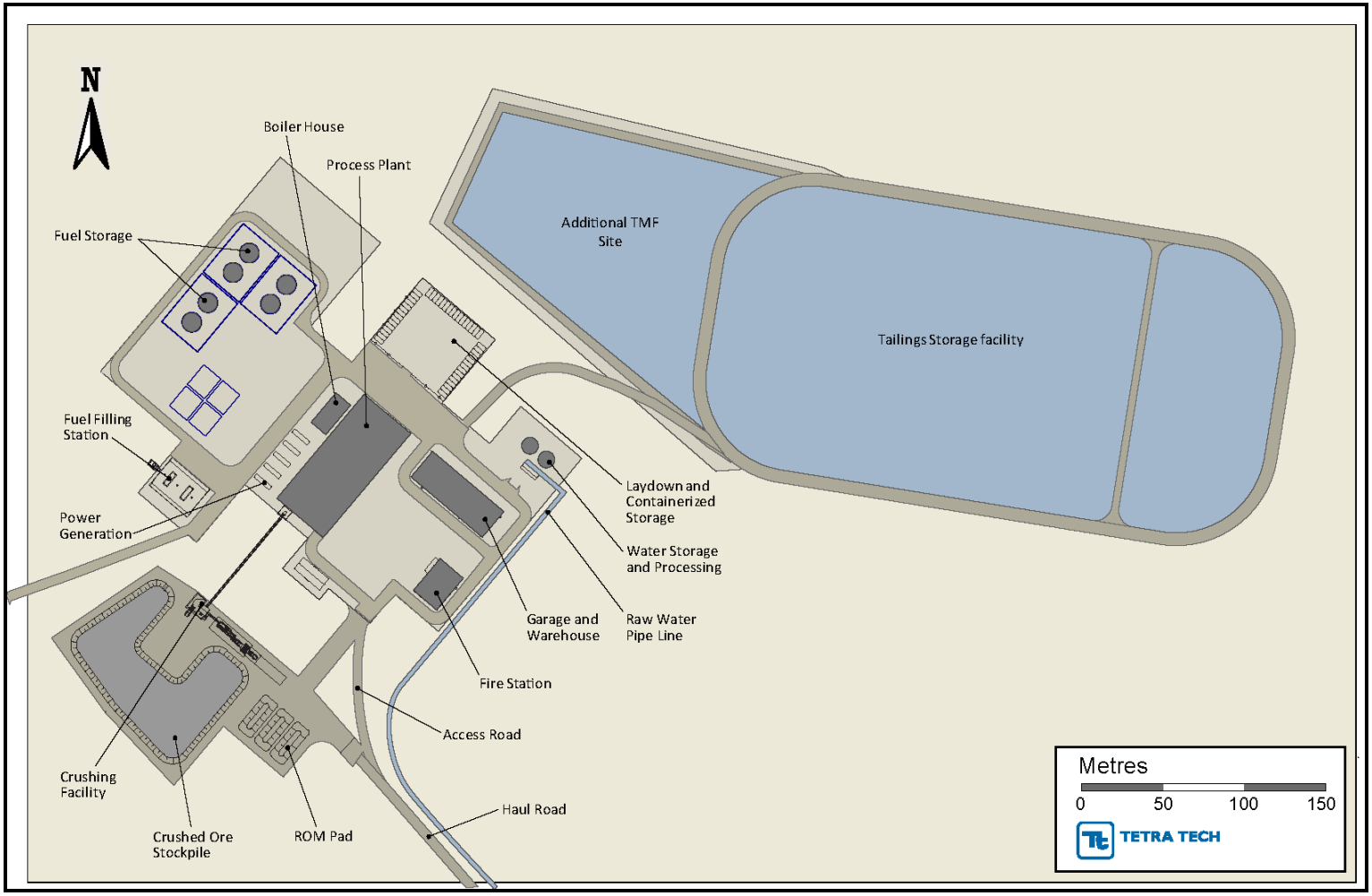


Figure 1.5 Process Plant Facility Layout



1.11.3 WINTER ROAD

The area is serviced by a number of public winter roads serving villages on route to site. The current haulage route is 570 km long. Based on the last five years of moving consumables and heavy equipment to site, a one-way trip takes, on average, two days.

The local Sakha authority is responsible for determining when the road is open and granting permission for the road to be used. Road maintenance is undertaken by specialist contractors under the direction of Prognoz, with the maintenance burden increasing in line with road usage. Maintenance begins in September to prepare for the official opening of the road on 1st January.

Early in the season the road opens with a 5-t weight limit and, as the ice thickens, weight limits are raised to 10 t, and finally 20 t, although heavier loads can be transported by special arrangement. For general site deliveries, a size limit of 12 m (a 40 ft container) and a load weight limit of 14 t is recommended before special measures are taken.

Historically, Silver Bear has moved up to 50-t payloads on the winter roads for one-off special loads, such as a Caterpillar D9 bulldozer. For the 2015/2016 winter road season, the road was upgraded to provide a reliable route for process equipment delivery, in addition to regular bulk deliveries of construction material, spares, and consumables.

As part of the logistics supply line, upgrading of the Batamai Port on the northern bank of the Lena River will enable containerised consumables such as reagents, along with fuel, to be stockpiled during the summer months for haulage along the winter road. This will reduce the current haulage distance of 570 km to 370 km, reduce turnaround time from approximately six days to three days, and extend the transport window.

1.12 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Project is in progress with the Russian review and approval process. As part of this process, the environmental and social aspects of the Project are under evaluation by the Russian regulatory authorities. Currently, the mining part of the Project has received positive conclusions by the State Environmental Review; however, infrastructure and the processing facilities are being revised based comments from the regulators. There can be no guarantees that the Project will be approved under Russian law, the timeline for any decision, and if the Project is approved, that there won't be terms and conditions that will have a material impact on the feasibility of the Project.

Silver Bear currently accesses the camp and the mine site via land from the existing airstrip. A portion of the route taken traverses alongside the Sirelendge River and crosses it in multiple locations. This is not a permitted road or activity. The Sirelendge River has a status of high fisheries significance.

An all-season access road to the mine from the existing airstrip forms part of the Project as contemplated by the Feasibility Study, but does not form part of the Project Design Documentation submitted for permitting purposes. Due to the sensitivity of the adjacent river, all-season access is not likely to be permitted by the fisheries authorities. ERM is of the view that all-season access to the mine represents a potentially material risk to the Project that still needs to be addressed.

The assessment of potential effects of the Project, the development of mitigating and management measures, and the establishment of monitoring programs embedded in the Technical Report are preliminary and are expected to evolve over time. The results of ongoing work, including ongoing environmental studies, stakeholder engagement, and permitting have the potential to influence the design, operations, and economic results of the Project.

The Project execution plan and schedule has been de-coupled from permitting activities. Construction activities are being undertaken without the necessary permits in place. Further, the commencement of mining requires additional permits that may not be in place to support the currently planned schedule upon which the Feasibility Study is based.

In addition to real or perceived risks to safety and/or the environment, conducting these current and future activities without the necessary permits presents procedural risks that could result in regulatory action, including fines, Project re-design and/or schedule delays to the Project that could be material.

1.13 CAPITAL AND OPERATING COST ESTIMATES

1.13.1 CAPITAL COST ESTIMATE

The total estimated initial capital cost for the design, construction, installation, and commissioning of all facilities and equipment is $65.0 million. A summary breakdown of the capital cost is provided in Table 1.11.

Table 1.11 Capital Cost Summary

Area	Initial ($)	Sustaining ($)	Total ($)
Mining	4,151,164	13102,368	17,253,532
Processing	15,383,403	700,000	16,083,403
Infrastructure	4,584,021	-	4,584,021
Utilities	2,708,651	-	2,708,651
TMF	1,350,566	1,325,776	2,676,342
Site Facilities	7,221,388	-	7,221,388
Off-site Facilities	43,327	-	43,327
Project Indirects	16,388,012	1,016,409	17,404,421
EPCM	5,439,947	20,248	5,460,195
Owner's Cost	7,241,830	100,000	7,341,830
Allowances	462,031	831,232	1,293,263
Total	**64,974 340**	**17,096,033**	**82,070,373**

Note: EPCM = engineering, procurement, and construction management

This estimate includes direct field costs required to execute the Project, plus indirect costs associated with design, construction and commissioning. This estimate is based on pricing as of Q3 2015, with no allowances for inflation or escalation. Elements of actual expenditures until June 2017 have been included, which further reduce the exposure to inflation or escalation. All currencies in this capital cost estimate is expressed in US dollars, unless otherwise noted.

1.13.2 Operating Cost Estimate

The LOM operating cost estimate for the Project consists of mining, processing, and G&A costs (which includes TMF and site water management costs) and is estimated at $156.21/t processed. Table 1.12 summarises the unit operating costs of each area.

Table 1.12 LOM Averaged Operating Cost Summary

Area	Unit Cost ($/t processed)
Mining	53.17
Processing	60.58
G&A	41.92
Total	**156.21**

1.14 Economic Analysis

An economic evaluation was prepared for the Project based on a pre-tax financial model. The base case uses a variable silver price of $19.65/oz, $18.57/oz, $19.62/oz, $19.79/oz, $19.72/oz, and $19.92/oz during Q1 2018, Q2 2018, Q3/4 2018, 2019, 2020, and 2021, as well as the remaining project life, respectively, with a life-of-mine (LOM) weighted average silver price of US$19.76/oz. The exchange rate applied in the base case is RUB66.00/USD1.00. All currency units are US dollars unless otherwise specified. For the 8-year LOM and 858,000 t Mineral Reserve, the following pre-tax financial parameters were calculated:

- 48.2% internal rate of return (IRR)
- 1.9-year payback on the $65 million initial capital
- $88 million net present value (NPV) at a 5% discount rate.

Silver Bear commissioned PricewaterhouseCoopers (PwC) in Toronto, Canada, to prepare tax calculations for use in the post-tax economic evaluation of the Project with the inclusion of applicable Russian income and mining taxes. PwC estimated the applicable taxes in two cases. The first case does not account for the Far East Tax Incentives and the second case takes into account the Far East Tax Incentives (refer to Section 22.4 for further details). The following post-tax financial results were calculated for the base case without considering the Far East Tax Incentives:

- 28.4% IRR
- 2.4-year payback on the $65 million initial capital
- $46 million NPV at a 5% discount rate.

The following post-tax financial results were calculated for the base case when considering the Far East Tax Incentives:

- 44.3% IRR
- 2-year payback on the $65 million initial capital
- $77 million NPV at a 5% discount rate.

Sensitivity analyses were conducted to analyse the sensitivity of the Project merit measures (NPV, IRR, and payback periods) to the main inputs such as silver price, exchange rate, capital costs, and on-site operating costs. The Project's pre-tax NPV, calculated at a 5% discount rate, is most sensitive to silver price followed by exchange rate, on-site operating costs, and capital costs. The Project's pre-tax IRR is most sensitive to silver price followed by exchange rate, capital costs, and on-site operating costs. The payback period is most sensitive to silver price and exchange rate, followed by capital costs and on-site operating costs.

1.15 PROJECT EXECUTION PLAN

The execution strategy for the successful monitoring and control of the Project reflects a modification to the traditional EPCM approach to project execution. Silver Bear will manage the completion of the Project with the input from various engineering companies, including Russian-based companies, who will complete the bulk of the detailed engineering, and international engineering firms, who will provide engineering review and specialist metallurgical input. Silver Bear will be responsible for the project management, procurement, and construction management using in-house resources. Short lines of decision making will be used to execute this fast-track project.

The Silver Bear execution strategy will be built on a lean, fit-for-purpose approach that favours self-sufficiency in execution and operation.

1.16 RISKS AND OPPORTUNITIES

1.16.1 RISKS

The most significant risks related to project execution (identified as critical risks) are shown in the risk matrix with their mitigation strategies in Table 1.13.

Table 1.13 Critical Risks

Description	Impact	Mitigation
Mishandling of sodium cyanide during processing.	Contamination of environment (land and rivers) subject to exposure during a spill.	Only let trained personnel handle sodium cyanide. Train medical personnel on site in sodium cyanide emergency care. Medical evacuation plan.
Insufficient water supply due to inadequate season data collection.	Stop operation until water supply is secured.	Conduct full hydrogeological program in the spring. Find alternate winter source of water.
Data from metallurgical test work done on the samples provided by Silver Bear in 2014 does not reflect mine life ore behaviour.	Predicted recoveries may not be accurate. Lower revenue possible.	Complete metallurgical test work representative of life of mine (see Section 13.0)
There is a possible levy on the deposition of tailings in the TMF by the local authorities that relates to the classification of that material. Discussions with the local authorities are ongoing, however there is a possibility that a charge of $7.88/t of tailings may be payable.	An increase in the operating cost of the mine.	According to Silver Bear, the payment of this levy is negotiable, subject to ongoing test work. Silver Bear has committed to ongoing test work and ongoing discussions with the local authorities.
Constructing the mine prior to final approvals and permits to construct means that the terms of the approvals may differ from the current design.	Parts of the design and/or constructed facilities may have to be changed leading to extra cost and possible delays.	Employ experienced Russian design agency to design in strict accordance with regulations. Keep the authorities informed and appoint a dedicated team in Yakutsk to manage the design documentation and permit approval process.
The approvals and permits for operation may be later than expected.	Without permits to operate the hot commissioning of the plant will be delayed accordingly.	Employ experienced Russian design agency to design in strict accordance with regulations. Keep the authorities informed and appoint a dedicated team in Yakutsk to manage the design documentation and permit approval process.
Inability to install the synthetic liner for TMF in low temperatures.	Delay of TMF commissioning.	Proper work scheduling in warmer weather periods.

Note: ESIA – environmental and social impact assessment

1.16.2 OPPORTUNITIES

MANGAZEISKY NORTH

Silver Bear's stated objective is to become a world-class producer of silver from the wholly-owned Prognoz Vertikalny Resource whilst continuing strategic exploration works and evaluating and developing additional resources for Silver Bear's continuous growth at optimised capital and operating costs. The sunk capital cost associated with the infrastructure development for the Project may lower the cut-off grade for subsequent deposits and subsequently the hurdle to entry. On inspection it was clear from the outset that the Mangazeisky North deposit contains a large enough Indicated Resource at a sufficient average silver grade to warrant a

preliminary economic assessment in order to examine its potentially economic viability.

Although the scale of the deposit did not allow it to carry significant capital cost, its proximity to the Vertikalny mine site made it ideal as a satellite mine that could be mined by the same equipment and processed using the same plant as the Vertikalny operation, matching the required production rate. Mangazeisky North offers an opportunity to increase the overall mine life and extend the useful life of the capital equipment on the Mangazeisky property.

A preliminary economic assessment for the Mangazeisky North opportunity outlined in Section 24.0, which includes Mineral Resources (other than Inferred Mineral Resources) into the Vertikalny mine schedule, indicates a potential increase of the overall project NPV (5%) from $133 million to $143 million, for an initial capital investment of $1.5 million.

This PEA of the Mangazeisky North deposit should not be considered to be a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. There is no certainty that the PEA will be realized. Note that Inferred material in the Mangazeisky North opportunity is treated as waste and is given zero value in the pit optimisation, mine schedule and financial modelling.

PROCESSING

The advantage of the inclusion of gravity separation combined with intensive leaching should be considered as a way of increasing metallurgical recovery and reducing cyanide consumption.

Tetra Tech believes that it may be possible to identify a more efficient operating point for the (oversized) ball mill to operate during the treatment of relatively soft oxide ore.

1.17 RECOMMENDATIONS

Recommendations for the Project are highlighted in Table 1.14. Further details and additional recommendations can be found in Section 26.0.

Table 1.14 Project Recommendations

Area	Recommendations
Strategic	• Develop the Mangazeisky North deposit by completing the next level of feasibility study.
Geology	• adopt an industry standard Geological Database Management System
	• complete infill drilling in areas where there is presently insufficient information to predict the grade of mineralization
	• conduct a small exploration drilling program in the Mangazeisky South mineralization

table continues...

Area	Recommendations
Mining	• increase material mined during the warmer weather periods, allowing a relaxed operation in the winter months without any interruption in supplying the ore tonnage to the process plant
	• implement operational readiness six months before the start of the underground operation to ensure well trained personnel for sub-level open stoping
	• adopt an accurate and practical grade control operation with standard sampling and testing protocols with short turnarounds to enable the geology and mining departments to properly plan for the short term mining schedule
	• prepare a detailed topography contour map of the open pit, WRD, and the road to the crusher be prepared that can be used in the detail design and scheduling phase during the mining operation
Mineral Processing and Metallurgical Testing	• conduct ongoing metallurgical sampling and test work on the oxide ore
	• complete a QEMSCAN® mineralogical investigation and silver deportment study for the primary ore
	• complete detailed metallurgical test work for the primary ore, including comminution and leach optimization tests along with intensive leach of flotation concentrates with appropriate pre-treatment
	• revisit the processing options for the primary ore based on the outcomes of the mineralogical and metallurgical studies and the size of the primary ore reserve
	• optimise the ball mill operating duty for soft ore
Project Infrastructure	• implement a logistics plan
	• prepare a contingency logistics plan
	• set up a maintenance camp along the winter road to provide first aid, vehicle breakdown and accident assistance, as well as provision of accommodation and stores for maintenance equipment
	• measure fuel quality at every step of supply chain
	• complete value engineering study to consider increase cost associated with a future increase in storage against the risk of insufficient supply during the winter road season
	• implement standardization on the selection of equipment and material
TMF	• investigate methods to minimise the volumes of the surface runoff water reporting to the clarification pond
	• model the thermal equilibrium of the dry stack to verify that the dry stack will be frozen with only the top 2-m thick layer being active
	• refine the proposed closure plant to consider progressive rehabilitation of the dry stack
	• consider the impact of future Russian classification and permitting of the tailings material on the design and associated levy on the placement of tailings material

table continues...

Area	Recommendations
Environmental studies, permitting and social or community impact	• continue to actively engage with government authorities and community stakeholders as part of the Russian review and approval process
	• complete a detailed compliance assessment and work with regulatory authorities to address current permit gaps; update project design documentation and include the access road to the mine as part of project permitting; cease any work, including construction, for which permits are not in place
	• complete an updated gap assessment against the Equator Principles and IFC Performance Standards to assess the current level of alignment of the Project as defined by the feasibility study
	• further engagement with local indigenous communities with regard to land and resources restoration to ensure that social requirements of the IFC Performance Standards are met
	• complete further environmental studies to enhance the dataset characterizing the aquatic environment and confirm the source of elevated metals found in the local and regional waterways, including the potential impact of exploration and development activities that have already taken place
	• expand the current water balance to include all site-wide sources and to incorporate water quality characterisation; validate model predictions and confirm that environmental mitigation measures are effective and discharge can occur in accordance with anticipated permit conditions without further treatment
	• further characterization of the tailings material and the potential for ARD/ML leaching is required; the ARD/ML work completed to date is preliminary
	• further study to characterize the predicted quality and quantity of water from the waste rock seepage and surface water runoff collection pond to confirm the assumed acceptability of release without additional management or treatment
	• establish more detailed management and monitoring programs to validate that the Project will need to be designed and implemented to support construction, operations, and closure phases

Note: QEMSCAN® – Quantitative Evaluation of Minerals by Scanning; IFC – International Finance Committee; ARD – acid rock drainage; ML – metal leaching

2.0 INTRODUCTION

Silver Bear retained Tetra Tech to complete a Technical Report of the Mangazeisky Project, located in Far East Russia. Silver Bear is a public limited company incorporated under the *Business Corporations Act* of the Province of Ontario, Canada, on 8th April 2004. Silver Bear is listed on the TSX under the symbol SBR. Effective June 30, 2017, Silver Bear completed a redomicilliation transaction whereby Silver Bear was acquired by Silver Bear Resources plc, an English company, which now continues as the parent company. All references to Silver Bear after June 30, 2017 are to Silver Bear Resources plc.

In compliance with NI 43-101 guidelines this Technical Report includes as of the effective date all material scientific and technical information in respect of Silver Bear's Mangazeisky Property and therein presents the current status and economic outlook of the Vertikalny deposit, re-states the current Mineral Resources for the Nizhny Endybal and Mangazeisky North and South deposits, and includes as summary of the Mangazeisky North deposit economic opportunity (Section 24.4) as its potential economic viability, which would occur if its Mineral Resources (other than Inferred Mineral Resources), are integrated into the Vertikalny mine plan.

The effective date of this report is 21st August 2017 and the effective date of the Vertikalny Mineral Resource estimate is 8th July 2016.

Tetra Tech worked with additional qualified consulting companies, commissioned by Silver Bear, to complete component studies for the purposes of the Technical Report. Table 2.1 outlines the responsibilities of each company.

Table 2.1 Consulting Company Responsibilities

Company	Responsibility
Tetra Tech	overall project management; property description and location; accessibility, climate, local resources; infrastructure and physiography; history; geological setting and mineralization; deposit type; exploration; drilling; sample preparation, analyses and security; data verification; adjacent properties; mineral resource estimates; mineral processing and metallurgical testing; recovery methods, logistical infrastructure; site layout; site access; site geotechnical; structural and architectural design basis; site roads; processing facility infrastructure; site facilities; remote site facilities; waste rock dumps, ore stockpiles and overburden storage area; site water management; waste and sewage; site storage; power supply and distribution; processing and general and administrative capital and operating cost estimates; economic analysis; project execution; mineral reserve estimates; mining methods; mining capital and operating costs
ERM	environmental studies, permitting, and social or community impact
SRK	mining geotechnical evaluation; TMF; tailings capital and operating costs; hydrology and hydrogeology

2.1 QP SITE VISITS

A summary of the QPs responsible for this report is provided in Table 2.2. The following QPs conducted site visits of the Property:

- Jacques du Toit, CEng, PrEng, MScEng, PMP, visited the site on the 21st July 2014, for one day.
- Robert Davies, BSc (Hons), CGeol, EurGeol, FGS, visited the site from the 12th to 15th September 2016 and between 26th June and 9th July 2016.
- Laszlo Bodi, P.Eng., visited the site from the 11th to 13th October 2016.
- Houcyne El Idrysy, PhD, CGeol, FGS visited the site on 15th September 2015.

Table 2.2 Summary of QPs

Report Section		Company	QP
1.0	Summary	All	Sign-off by Section
2.0	Introduction	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
3.0	Reliance on Other Experts	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS Sabry Abdel Hafez, PhD, PEng. Jacques du Toit, CEng, PrEng, MScEng, PMP
4.0	Property Description and Location	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
6.0	History	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
7.0	Geological Setting and Mineralisation	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
8.0	Deposit Types	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
9.0	Exploration	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
10.0	Drilling	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
11.0	Sample Preparation, Analyses and Security	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
12.0	Data Verification	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Arunasalam Vathavooran, PhD, CEng, MIMMM
14.0	Mineral Resource Estimates	Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol, FGS Joseph Hirst Bsc (Hons), Msc, CGeol, EurGeol
15.0	Mineral Reserve Estimates	Tetra Tech	Anton von Wielligh, BSc (Hons), PrEng, MSAIMM/ Matthew Randall, BSc (Hons), PhD, CEng, MIMMM

table continues...

Report Section		Company	QP
16.0	Mining Methods	Tetra Tech/ SRK	Anton von Wielligh, BSc (Hons), PrEng, MSAIMM Rene Gharapetian, P.Eng. Matthew Randall, BSc (Hons), PhD, CEng, MIMMM Max Brown, BSc, MSc, CEng, MIMMM
17.0	Recovery Methods	Tetra Tech	Arunasalam Vathavooran, PhD, CEng, MIMMM
18.0	Infrastructure	-	-
18.1	Overview	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.2	Logistical Infrastructure	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.3	Site Layout	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.4	Site Geotechnical	Tetra Tech	Laszlo Bodi, MSc, CEng, P.Eng.
18.5	Design Basis	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP Saunjay Duggal,MSc, P.Eng., PE Laszlo Bodi, MSc, CEng, P.Eng.
18.6	Site Roads	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.7	Processing Facility	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.8	Site Facilities	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.9	Remote Site Facilities	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.10	Tailings Management Facility	SRK	Krysztof Czajewski, BSc, P.Eng.
18.11	Waste Rock Dump, Ore Stockpiles and Overburden Storage	Tetra Tech	Laszlo Bodi, MSc, CEng, P.Eng.
18.12	Site Water Management	Tetra Tech	Guy Roemer, PE
18.13	Waste and Sewage	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.14	Site Storage	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.15	Site Services	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
18.16	Power Supply and Distribution	Tetra Tech	Damian Hicks, MIEAust, CPEng
19.0	Market Studies and Contracts	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
20.0	Environmental Studies, Permitting and Social or Community Impact	-	-
20.1	Introduction	ERM	Derek Chubb, P.Eng.
20.2	Permitting Requirements	ERM	Derek Chubb, P.Eng.
20.3	Environmental and Social Studies	ERM	Derek Chubb, P.Eng.
20.4	Environmental and Social Impact Assessment, Mitigation and Management Plans	ERM	Derek Chubb, P.Eng.
20.5	Water and Waste Management	ERM	Derek Chubb, P.Eng.
20.6	ERM Comments	ERM	Derek Chubb, P.Eng.
20.7	Closure and Reclamation	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP

table continues...

Report Section			Company	QP
21.0	Capital and Operating Costs		-	-
	21.1	Capital Costs	-	-
	21.1.1	Summary	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	21.1.2	Basis of Estimate	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	21.1.3	Direct Costs	Tetra Tech SRK	Jacques du Toit, CEng, PrEng, MScEng, PMP Kris Czajewski, P.Eng.
	21.1.4	Indirect Costs	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	21.1.5	Qualifications, Assumptions and Exclusions	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	21.2	Operating Costs	-	-
	21.2.1	Summary	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	21.2.2	Basis of Estimate	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	21.2.3	Mining	Tetra Tech	Anton von Wielligh, BSc (Hons), PrEng, MSAIMM/ Jacques du Toit, CEng, PrEng, MScEng, PMP
	21.2.4	Processing	Tetra Tech	Arunasalam Vathavooran, PhD, CEng, MIMMM
	21.2.5	Tailings Management Facility	SRK	Krysztof Czajewski, BSc, P.Eng.
	21.2.5	General and Administrative	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
22.0	Economic Analysis		Tetra Tech	Sabry Abdel Hafez, PhD, P.Eng.
23.0	Adjacent Properties		Tetra Tech	Robert Davies, BSc (Hons), CGeol, EurGeol., FGS
24.0	Other Relevant Data and Information		-	-
	24.1	Project Execution Plan	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	24.2	Risks and Opportunities	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP
	24.3	Hydrogeology and Water Supply	SRK	Houcyne El Idrysy, PhD, CGeol, FGS
	24.4	Mangazeisky North Open Pit Opportunity	Tetra Tech	Jacques du Toit, CEng, PrEng, MScEng, PMP Laszlo Bodi, MSc, CEng, P.Eng. Arunasalam Vathavooran, PhD, CEng, MIMMM
25.0	Interpretation and Conclusions		All	Sign-off by Section
26.0	Recommendations		All	Sign-off by Section
27.0	References		All	Sign-off by Section
28.0	Certificates of Qualified Person		All	Sign-off by Section

2.2 SOURCES OF INFORMATION

All sources of information for this study are located in Section 27.0.

2.3 UNITS OF MEASUREMENT AND CURRENCY

All units of measurement used in this technical report are in metric.

All currency is in US Dollars, unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

3.1 GEOLOGICAL RELIANCE

Tetra Tech has reviewed and analysed data provided by Silver Bear, its consultants and previous operators of the Property, and has drawn its own conclusions, augmented by its direct field examination.

While exercising all reasonable diligence in checking, confirming, and testing, Tetra Tech has relied upon Silver Bear's presentation of the data for the Project in formulating its opinion and estimates.

The various agreements under which Silver Bear holds title to the mineral lands for the Project have not been reviewed by Tetra Tech, and Tetra Tech offers no opinion as to the validity of the mineral title claimed. Tetra Tech is relying on legal opinions regarding Property title prepared by experts who are not QPs as defined by NI 43-101. A legal title opinion was authored by Ivan A. Trifonov, Managing Partner of Squire, Sanders & Dempsey (Moscow) L.L.C. in Appendix 2 of the 2004 Technical Report by Roscoe Postle Associates Inc. (RPA 2004). A description of the Property and ownership is provided for general information purposes only in Section 4.0.

The description of geology, mineralisation, and exploration methodology used in this report are based on direct field examination, and on data available in published and unpublished reports supplied by Silver Bear. In turn, Silver Bear has relied on various reports and data supplied by state organisations and others that have previously conducted exploration on the Property. The information provided to Silver Bear was supplied by reputable state organisations and Tetra Tech has no reason to doubt its validity.

3.2 COST ESTIMATE

Jacques du Toit, CEng, PrEng, MScEng, PMP relied on Graham Hill, Chief Executive Officer of Silver Bear, to provide the following inputs used in the capital cost estimate section (Section 21.1):

- capital spent and committed to date
- estimate to complete budget
- list of equipment and facilities on site.

3.3 ECONOMIC RELIANCE

Sabry Abdel Hafez, PhD, P.Eng. relied on Graham Hill, Chief Executive Officer of Silver Bear, to provide the following inputs used in the economic analysis section (Section 22.0):

- royalties applicable to the Project
- smelting and refining quotation
- silver off-site transportation and insurance costs.

Mr. Abdel Hafez also relied on PwC concerning tax matters relevant to this report. The reliance is based on a letter to Silver Bear titled "Assistance with the calculation and review of the Russian income and mining tax portion (the 'tax section') of the Feasibility Study Model (the 'Model') in connection with the NI 43-101 Technical Report (the 'Report') for the Mangazeisky project (the 'Project') prepared by an external consultant engaged by Silver Bear Resources Inc. ('Silver Bear' or the 'Company' or the 'Client')" and dated 18th August 2017.

Mr. Jacques du Toit, CEng, PrEng, MScEng, PMP relied on PwC concerning tax matters relevant to Section 24.4.7. The reliance is based on a letter to Silver Bear titled "Assistance with the calculation and review of the Russian income and mining tax portion (the 'tax section') of the Feasibility Study Model (the 'Model') in connection with the NI 43-101 Technical Report (the 'Report') for the Mangazeisky project (the 'Project') prepared by an external consultant engaged by Silver Bear Resources Inc. ('Silver Bear' or the 'Company' or the 'Client')" and dated 18th August 2017.

3.4 ENVIRONMENTAL RELIANCE

Mr. Jacques du Toit, CEng, PrEng, MScEng, PMP relied on Ms. Nicky Spooner, MSc, for environmental and social impact information provided in Section 24.4.5 Ms. Spooner has over 20 years' experience, having conducted over 150 environmental and social due diligence assignments, as well as having led multiple ESIAs in Africa, the Middle East, Asia, and the Commonwealth of Independent States. She has extensive experience in environmental and social due diligence, management training and capacity building, environmental and social impact assessment, international lender compliance, and environmental and social management plans.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Property is defined by the mineral rights to a single mining license, and is 100% held by Prognoz, a private Russian closed-joint stock company based in Yakutsk. Prognoz is a 100% held subsidiary of Silver Bear.

4.1 LOCATION

The Property is located:

- at approximately 590,960 m east and 7,234,750 m north (Zone 36J; World Geodetic System (WGS 84)
- at approximately 65°40' south and 130°07' east; in central Sakha Republic (Yakutia), Kubyaysky District of Russia
- approximately 400 km north of Yakutsk, capital city of the Sakha Republic
- approximately 300 km southwest of Batagai in northern Sakha Republic
- approximately 230 km north of Sangary, a river port on the right bank of the Lena River, and administrative centre for the Kubyaysky District.

The Property is situated as shown in Figure 4.1.

Figure 4.1 Property Location Map

Source: Silver Bear

4.2 DESCRIPTION

Silver Bear holds the mineral rights to the Property through its 100% interest in Prognoz. Silver Bear purchased Prognoz in 2004 from the National Resource Company.

The Property is located in the north of Kobyaysky District, in central Sakha Republic (Yakutia), and is comprised of one mining licence within a larger exploration licence. The mining license, number YaKU 03626 BE, covers the entire Vertikalny silver deposit over an area of 13.55 km^2. The coordinates of the mining license are shown in Table 4.1.

Table 4.1 Mining License Coordinates

Mining License YaKU 03626 BE		
Corner No.	Northing Coordinate	Easting Coordinate
1	65°41'15.917"	130°01'55.380"
2	65°41'41.938"	130°03'23.150"
3	65°41'37.066"	130°04'59.859"
4	65°41'20.210"	130°06'27.196"
5	65°40'08.102"	130°08'20.361"
6	65°39'44.803"	130°08'11.742"
7	65°39'40.272"	130°07'17.802"
8	65°39'46.221"	130°05'22.190"
9	65°39'54.675"	130°03'29.389"
10	65°40'11.350"	130°01'57.673"
11	65°40'46.388"	130°01'42.001"

The mining license is situated in the central west portion of the larger Endybal exploration license (also known as the Mangazeisky exploration license), number YaKU 12692 BP, and covers several other exploration targets and silver deposits over an area of 570 km^2. The coordinates of the exploration licence are shown in Table 4.2.

Table 4.2 Exploration License Coordinates

Exploration License YaKU 12692 BP		
Corner No.	Northing Coordinate	Easting Coordinate
1	65°49'35"	130°00'00"
2	65°49'35"	130°19'20"
3	65°29'00"	130°22'00"
4	65°29'00"	130°00'00"

4.3 HISTORY OF PROPERTY ACQUISITION

On 21st October 2004, Silver Bear completed an acquisition of all of the outstanding shares of Prognoz. Pursuant to the transaction, Silver Bear acquired 100% of the issued and outstanding common shares of Prognoz for RUB10,000,000 or CAD331,000 and assumed certain bank indebtedness and other liabilities of Prognoz. The parties to the transaction agreed that the value of the exploration licences held by Prognoz closely approximated the indebtedness assumed and accordingly, a value of RUB20,585,221 or CAD890,310 was attributed to the licences. Tetra Tech is not aware of any liability in the form of royalties or other financial encumbrances, but these may be applicable.

The exploration licence YaKU 12692 BP was granted to Prognoz on 24th September 2004 by the Federal Subsoil Resources Management Agency (ROSNEDRA), and was

valid for an initial term of five years. On 18th February 2009, the exploration licence was extended by ROSNEDRA to 31st December 2011. Another extension was subsequently granted for one year until 31st December 2012. On 27th December 2012, a third extension was granted for four years until 31st December 2016.

The exploration licences give the recipient the authority to use the subsoil for the purposes of geological investigation within the licence area, for exploration, and appraisal of the gold and silver deposits. The licence area has the status of a "geological allotment" with the preliminary borders outlined and an unlimited licenced depth for investigation. According to the English translations of the exploration licences, there are no specially protected natural territories within the limits of the licence.

In September 2013, Silver Bear received its mining licence YaKU 03626 BE for the Vertikalny deposit. The term of the licence is approximately 20 years (to 2033). The licence requirements include:

- completion of 15,000 m of drilling and 15,000 m^3 of trenching by or before December 2017
- initiation of drilling and trenching no later than March 2015
- mine must be operational within the next nine years (2023), inclusive of permitting and report approvals
- mine output must be greater than 180,000 t/a by the year 2023.

A summary of all of the existing licence information is detailed in Table 4.3.

Table 4.3 Property Mineral Licences

Licence Name	Licence ID	Type	Area (km^2)	Issue Date	Expiry Date	Annual Fees (RUB)
Endybal Area (Mangazeisky)	YaKU 12692 BP	Geological Allotment	570.00	28th September 2004	31st December 2023	150,242
Vertikalny Deposit	YaKU 03626 BE	Licence to Use Subsoil	13.55	31st August 2013	1st September 2033	110,771

4.4 SURFACE RIGHTS

Silver Bear does not hold any surface rights on the Property, but has access to the Property for the purposes of mineral exploration. Construction activities have commenced on the Property but are not supported by having all of the necessary permits in place. Silver Bear acknowledges this is a risk to the Project and is working with the regulatory authorities to obtain the necessary construction and operations permits.

4.5 ENVIRONMENTAL LIABILITIES

According to the licence granted by ROSNEDRA, the main environmental requirements during exploration activities for the exploration licence are as follows:

- observance of the established requirements for protection of the environment and subsoil
- performance of necessary measures to reduce or prevent pollution caused by the execution of the exploratory work
- prevention of water facility pollution and clogging and observance of the regime of water protection zones
- reclamation of land disturbed during exploratory work, restoring it to a state suitable for further use and in compliance with landscape and recreational specifics of the territory
- timely payment of compensation for damages caused to hunting, agriculture, fisheries, and forestry
- coordination with and approval by the Administration for Subsoil Use in the Republic of Sakha-Yakutia (YAKUTNEDRA) for the terms and types of work to be carried out within the water protection zones.

Preliminary laboratory tests indicate that concentrations of minerals in some surface water and sediment samples exceed local regulatory standards. These elevated concentrations are attributed to natural weathering processes across the Project affecting regional watersheds and to exploration activities in local waterways near the Vertikalny deposit area. The Technical Report does not include an allowance to remediate any historical impacts that may be caused to date by Project activities.

5.0 ACCESSIBILITY, CLIMATE, INFRASTRUCTURE, LOCAL RESOURCES AND PHYSIOGRAPHY

The Property is located in the Kobyask district within the Republic of Sakha (Yakutia) of the Russian Federation. Yakutia covers an area of 3.1 Mkm^2, makes up one fifth of Russia's landmass, and has a population of approximately 1.0 million people. The region is rich in natural resources and 80% of its industrial production originates from the non-ferrous metals industry (diamonds, gold, tin) and antimony and fossil fuel energy (coal, gas, and oil). Over 10% of the available work force of 0.5 million people are involved in mining.

The major population centre for the region of Yakutsk has approximately 300,000 inhabitants. Yakutsk is located approximately 400 km south of the Property.

Yakutia recognises the present and long-term predicted demand for mineral resources in the region, and is implementing a strategic plan for the social and economic development of the region ("Scheme of Complex Development of Productive Forces, Transport and Power Industry of the Sakha Republic [Yakutia] by 2020"), which is supported by the Russian Federation Government. It is hoped that the Project will create new transportation corridors, increase export opportunities, and create employment and subsequent population inflow. It is also hoped that the Project will increase national security and assist integration with the Asia-Pacific regions. The majority of the infrastructure improvements will not have a direct impact on the Project especially in its early stages. Opening of supply lines etc. will have a positive impact in terms of supply of consumables and labour.

5.1 ACCESSIBILITY

The Property is only accessible from Yakutsk by air, either by fixed wing aircraft or by helicopter. There is an airstrip on the Property, at the confluence of the Endybal and Arkachan Rivers, approximately 10 km from the base camp. There are several charter companies based in and around Yakutsk for both types of aircraft. A flight by AN-2 is typically two hours.

The Property may also be accessed via Batagai, located approximately 300 km northeast of the Property. There are regular scheduled flights to Batagai as well as aircraft available for charter.

There is also a winter road that is open for several months during the winter for transport of all freight and supplies to the Property.

5.2 CLIMATE

The climate of northeast Russia is Continental subarctic to Tundra Climate zones (Dfd to ET; Köppen climate classification), and is characterized extreme cold dry winters and cool summer seasons. There is no climatic station at the Project site, but data is available from Verhojansk (National Oceanic and Atmospheric Administration (NOAA) Station ID RA24266; 67°33' north, 133°23' east, 137 m) and Yakutsk (NOAA Station; ID RA24959; 62°01' north, 129°43' east, 101 m).

The annual precipitation averages 200 mm with the majority occurring as rain during the summer months. Average temperatures range from +25°C in July to −40°C in December and January; although during the winter the temperatures can decrease to a low of −65°C for short periods. Snow cover is formed around the end of September and will have usually melted by the second half of May. The thickness of the snow varies from 40 to 50 cm. The rivers are usually ice-bound at the end of September and spring break-up is completed by the end of May. The region experiences light winds throughout the year.

The area has a permafrost layer of 300 to 400 m in thickness, or more.

Exploration activities may take place most of the year; however, the field season typically extends from April/May to September/October.

5.3 INFRASTRUCTURE

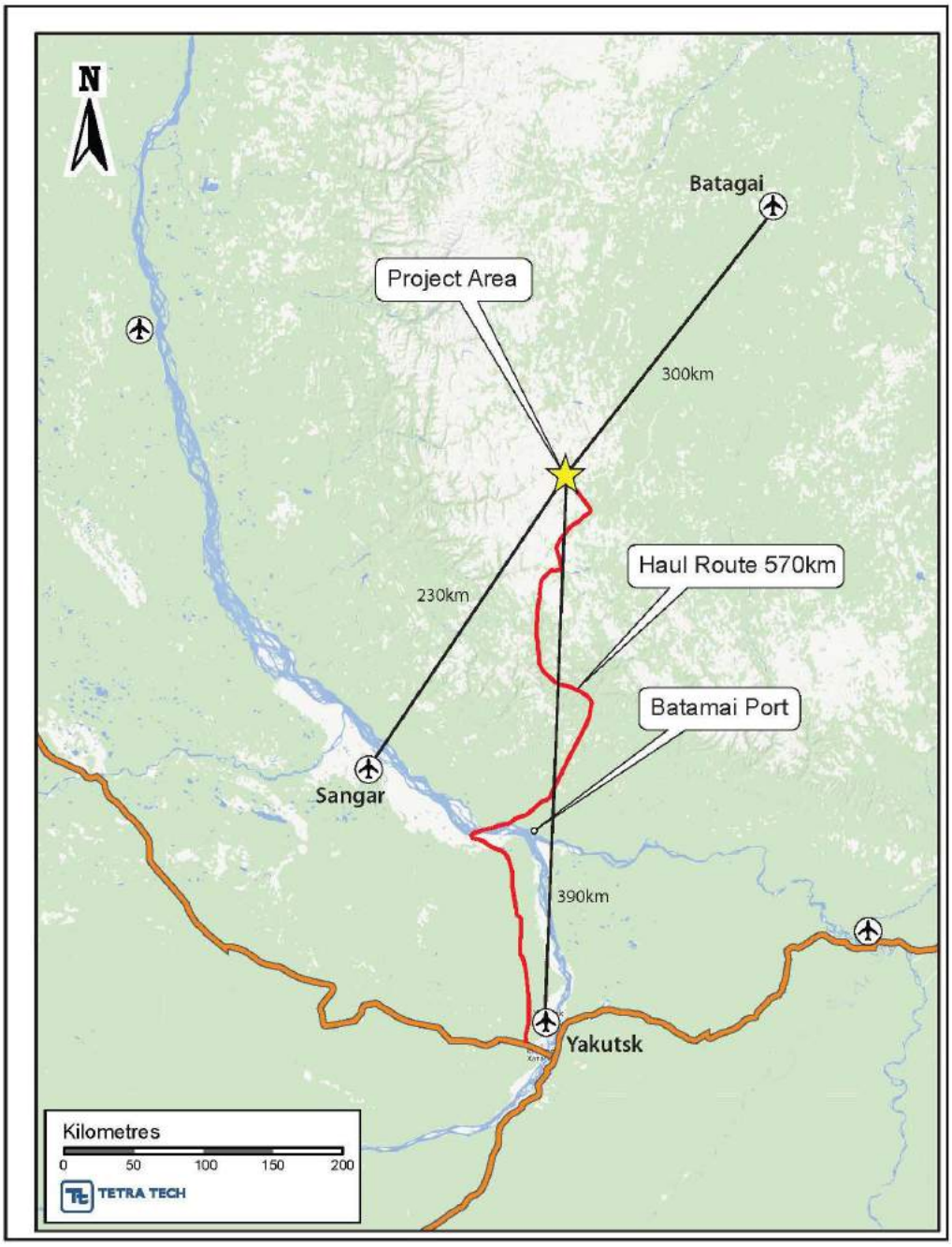
5.3.1 ROADS

Yakutia has a poorly developed road network and only 8% of the republic's territory is accessible all-year-round; the remaining 92% is seasonally accessible. Through rapid expansion it is hoped that the road infrastructure can be developed such that 91.5% of the population will have access to the republican centre and that 47.5% of the republic's territory will be provided by year-round motorways. Currently over 70% of passenger traffic is by air.

The Project area is isolated and can be accessed by a winter road that is usable from mid-January until mid-April, depending on the depth of snow fall. All-terrain vehicles (ATVs) are used for transporting workers and materials to site. The vehicles have a capacity of 7 t and a round trip from Yakutsk to the Property takes six days. Outside of this three month window, the roads are impassable due to surface thaw of the permafrost and subsequent boggy conditions and swollen river crossings.

The haul route currently used is shown in Figure 5.1. From Yakutsk to the Lena River crossing, there is a year-round road (approximately 200 km). The Lena River is navigable from June until September and has a barge capacity of up to 3,000 t at Batagai, as well as at Sangar. During the winter haulage season, the Lena River may be driven on by trucks up to 20 t. An additional 370 km along the winter road makes a total haulage distance of 570 km.

Figure 5.1 Airports and Current Haul Route



Source: Tetra Tech

Batagai is the administration centre for the Verkhoyanskiy district with a population of approximately 2,000 inhabitants and is situated approximately 280 km to the northeast of the Property. The closest accommodation is located in this town, which has two hotels and restaurants.

Batagai has a river port, along with aviation and transportation companies, a telegraph, and post office. There is an all-weather dirt road between the villages of

Batagai and Tokuma, and from Tokuma there is a link to the Property via a winter road, which is open for only a short period.

The closest village to the Property is Sebyan-Kuel which is located approximately 50 km south. The village of Sangar, located 240 km southwest of the Project, is the administrative centre for the district of Kobyask Ulas, which includes the area encompassed by the Project.

5.3.2 AIR TRANSPORT

During most of the year the Property is accessible primarily by helicopter or light fixed wing aircraft from Yakutsk, Batagai, or Sangar. From Yakutsk, the weather may interfere with flight schedules due to the intervening Verkhoyanskiy mountain range and delays of days and occasionally weeks are not uncommon. From Batagai, the helicopter service is more frequent, but delays due to weather conditions may occur. At the Mangazeisky exploration camp there is a helicopter pad, along with a small airstrip upon which to land small planes. The airstrip was extended to 800 m in 2009 and there is the possibility to extend further to around 1,200 m.

Currently, AN-2 and AN-3 fixed wing aircraft are being used for small loads (800 to 900 kg); MI-8 MTV and MI-26 helicopters are available for heavier loads (up to 1,800 kg).

5.3.3 RAIL AND PORTS

The Berkakit-Tommot-Yakutsk rail link is reportedly near completion. The rail head will be located on the east side of the River Lena; it is not known if a bridge is planned. This spur will link Yakutsk to the Trans-Siberian, Amur-Yakutia Railroad and the Northern Sea Route. Journey times will be significantly reduced.

Further rail links are planned that will eventually connect Yakutsk to Magadan and the Sea of Okhotsk.

The River Lena is navigable from April until late September. The river ports of Yakutsk, Batamai, and Sangar may be used to reduce the haulage distance to site by approximately 200 km.

5.3.4 POWER

There is no access to the main power grid on the Property. The Kobyask district is located in the decentralised power supply zone, and has a power generating capacity of 16 MW coming from 12 diesel generating sites. The nearest power generator set to the Project site is at Sebyan-Kuel (375 kW).

The main power supply company for Yakutia is Sakha Energy. The current power supply system incorporates three isolated power networks; the southern, western, and eastern regions and a decentralised region. Power is generated by both thermal and hydro-electric methods in the main regions. The primary source of electrical power in the decentralised region is diesel generation and alternative sources of energy such as wind power are being introduced.

By 2020, the electrical generating capacity of Yakutia will be supplemented with a further 8,500 MW from seven new power stations. The distribution network will be expanded to join the current isolated regions and connect to the unified power grid of Central and Western Russia.

5.4 LOCAL RESOURCES

The Property is relatively isolated with limited local resources. The base camp, Hogan Camp, is comprised of one to two room cabins, huts and accommodation containers. There are several permanent structures for kitchen, bath, and offices. There are also buildings for core logging and sampling, sample preparation and sample storage, as well as sheltered core box storage.

Potential water sources include the Arkachan River, located 10 km from the Project, and the Endybal, Sirelendge, Fedor-Yuryage, and Mangazeisky creeks, which flow through the licence area. Water resources may also become available from underground exploration and development. Silver Bear is working with regulatory authorities to obtain relevant permitting for the use of water resources.

5.5 PHYSIOGRAPHY

The Property lies in a mountainous region with elevations ranging from 800 to 1,400 masl. The main ridges have steep slopes (25 to 30° and rounded crests that are 200 to 500 m above the valley floors).

The vegetation surrounding the Property is composed of primarily aspen, birch and fir trees in the lower parts of the valleys with generally only grass and wildflowers at higher elevations. There are also widespread lichens and mosses. The region is well known for species such as bear, ermine, wolverine, fox, deer, elk, squirrel, and wolf. There is an abundance of fresh water fish in the streams and lakes, including grayling, lenok, pickerel, taimen, and perch.

6.0 HISTORY

6.1 EXPLORATION HISTORY

6.1.1 GENERAL EXPLORATION HISTORY TO 2004

Discovery of the mineralisation within the Property is attributed to the Russian Cossacks in 1764. Intermittent exploration by individuals continued until the formation of the Soviet Union in 1917. Upon nationalisation of all assets, exploration and mining became the responsibility of various government departments or agencies. Sporadic exploration was then undertaken during the 1930s and 1940s.

During 1952 and 1953, work focused on two previously identified showings identified as the Mikhailovsky and Kuzminsky zones, which are located 7.5 km and 10 km to the north of Vertikalny, respectively (Figure 7.4 presents a map of the explored mineralisation within the licence area). This work included geological mapping (1:50,000), trenching with systematic mapping and sampling, and the establishment of two short adits (32 m) beneath the trenches. Work also included a topographic survey (1:2,000, 3 km^2) and an induced polarisation (IP) survey (1:5,000, 1.7 km^2). By 1960, the exploration work completed in the licence area had identified more than 160 anomalies, within a north-south trend up to 20 km in length. This trend is 2 km wide in the north (Nuektame River) and up to 4.5 to 5.0 km wide in the south (Endybal River).

In 1989, systematic prospecting and exploration resumed, and has continued to this day.

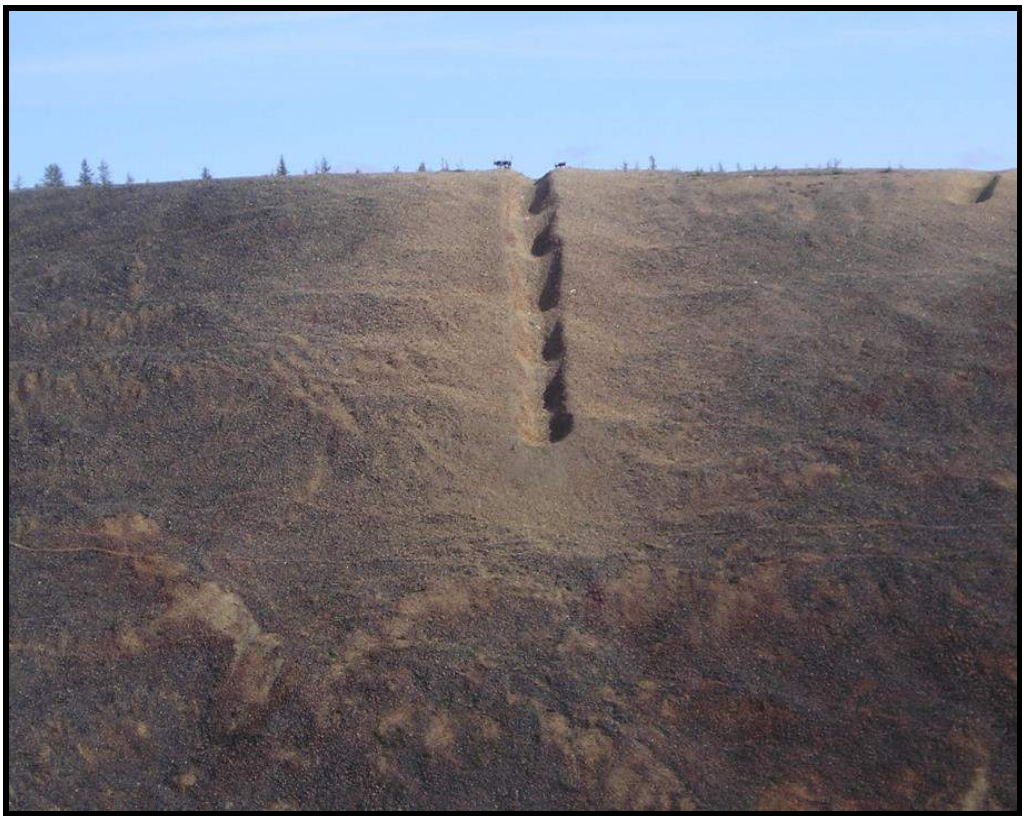
From 1991 to 2003 JSC Yangeologia completed 151,452 m^3 of trenching, 10.2 line km of magnetic surveys, detailed geological mapping, soil geochemical surveys, and 10 diamond drillholes totalling 1,303 m. This exploration work covered more than 15 principal vein systems.

From 1989, exploration was primarily located within the Vasilievsky, Sterznhevoy, and Nizhne-Endybalsky mineralised zones, bringing the total number of discretely mineralised structures containing potentially economic grades to more than 30.

Figure 6.1 shows an example of a historical trench on the Property.

Prognoz acquired the Mangazeisky exploration licence in September 2004. On 21st October 2004, Silver Bear acquired 100% of Prognoz by purchasing the company from National Resource Company.

Figure 6.1 Historical Trenches on the Property; Porphyrovy Zone



Source: Silver Bear

6.1.2 METALLURGICAL TEST WORK AND RECOVERABILITY

Initial metallurgical results were summarised in a report by Western Services (2004). The Western Services (2004) report describes metallurgical test work that was conducted on two samples from the Property. One sample, from the Vasilievsky vein, was composed of un-oxidised primary silver mineralisation, and contained minor components of lead, zinc, and copper along with minor stibnite. The silver was primarily concentrated within sulphosalts and present as native silver.

Single step flotation tests resulted in recovering 95% of the contained silver into a concentrate. The concentrate grade was 48 kg/t silver. Cyanidation of the flotation product recovered 99.1% silver from the concentrate. The total recovery was projected to be in the order of 94% for the material contained in the southern block of Vasilievsky.

A gravitational recovery circuit was also investigated, in which 57% silver was recovered into a concentrate with an average grade of 20.15 kg/t silver. Cyanidation of the gravity concentrate recovered 99.2% silver from the concentrate. The total recovery for this process option was 57%.

The second metallurgical sample collected was from the silver-lead Granichny mineralised zone. It consisted of partially oxidised mineralisation composed mainly of galena, sphalerite, and anglesite (0.1 to 2.0 to 3.0 mm in size). The silver occurred as freibergite, pyrargyrite, and argentite (0.02 to 0.50 mm in size) as well as extremely

fine-grained native silver. The second sample weighed 17.1 kg with an average grade of 320 g/t silver, 12% lead, and 1.4% zinc.

Based on the metallurgical test work, a single stage flotation circuit recovered 80.9% lead and 89.2% silver into a concentrate. Cyanidation results for the concentrate were not reported.

In 2008, Silver Bear collected two bulk samples from the Vertikalny vein. The samples were collected from the mineralisation intersected by surface trenching. The samples were submitted to the Federal State Unitary Enterprise Central Institute of Science Research and Geological Surveying of Base and Precious Metals (TSNIGRI) for metallurgical analysis. The results of this work are described in the 2009 technical report (Wardrop 2009a) and reproduced in Section 13.0 for convenience.

6.2 HISTORICAL RESOURCE ESTIMATES

6.2.1 HISTORICAL NON-COMPLIANT RESOURCE ESTIMATES

The Property has historical Mineral Reserve and Resource estimates on numerous deposits prepared according to the Russian classification system. Further clarification of the Russian classification system can be found in Ministry of Natural Resources of the Russian Federation (MNRRF) Order of 11th December 2006 No. 278, Classification Approval and Predicted Resources Solid Minerals (MNRRF 2006).

The historical Mineral Reserve estimate was reviewed by JSC Yangeologia (2000) and is based entirely on chip samples from within trenches and outcrop exposures within the river banks. Most of the silver assays were established using the gamma-activation method with external control by fire assay. Lead, zinc, tin, antimony, and copper were determined by using the x-ray radiometric method. The reported methodology (Young 2004) is outlined in the following paragraphs.

An average silver composite grade within each trench was calculated using a length weighted averaging for the intervals assaying in excess of 150 g/t silver.

The composited trench results were plotted in plan view, and the limits of the mineralisation within and between the trenches were linked together. Based on the trenches contained within the mineralised intercept outline, an average mineralised thickness was determined. The average contained grade of the mineralisation was calculated for all mineralised trenches and was also completed on a length weighted basis.

Vertical longitudinal projections were then constructed for each zone. The projection distance downward was 100 m below the surface for almost all of the mineralised zones. The depth of the projected trench values from the surface was based on the methodologies employed at the nearby Prognoz silver project, which has similar drill results as well as a similar geological genesis, mineralisation type, and overall character.

The projected area was multiplied by the weighted average thickness to estimate a mineralised volume. This volume was multiplied by one of several specific gravity

values, which were established from 56 samples collected from the major mineralised zones. These specific gravity values were determined by the Yakutsk Nuclear Physics Laboratory in 1992.

The tonnage, average metal grades and thickness for all discrete mineralised zones or subzones were then summarised into a tabular format for the final reporting of the estimated inventory. This mineralisation is contained the part of the deposit located nearest to the surface and was classified in the P1 category according to the Russian reserve/resource classification system. This classification was selected as there was very little sub-surface sampling (drilling or underground exploration) available to establish the down dip continuity of the mineralisation.

The mineralisation located at greater depth was then compiled and categorised in the P2 class. This material was derived by projecting the mineralisation approximately another 100 m below the P1 lower datum limit. Typically, this resulted in a doubling of the total tonnage over that represented by the P1 material

A total of 36 mineralised structures have been identified as including P1 and/or P2 Russian classified “reserves”. Table 6.1 summarises the distribution of the historical Russian “reserves” by mineralised zone with the top 15 zones reported individually and the remaining zones combined.

Table 6.1 Summary of the Historical Russian Inventory by Mineralised Zone for the Property

Mineralised Zone	Tonnes (kt)	Horizontal Width (m)	Average Grade				Contained Metal			
			Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)	Ag (t)	Au (t)	Pb (t)	Zn (t)
Vasilievsky	9,076	2.00	1,944	-	7.89	0.31	17,646	-	716,049	28,265
Sterznhevoy	153	0.90	2,652	0.80	0.31	0.10	405	0.12	473	153
Mikhailovsky	233	0.45	2,384	-	24.64	1.19	555	-	57,345	2,769
Kuzminsky	61	0.22	1,492	-	13.53	0.47	91	-	8,285	286
Burnoe	268	0.67	2,756	-	30.10	1.00	739	-	80,668	2,680
Mukhalkansky	1,048	0.00	1,509	-	19.58	1.05	1,581	-	205,205	11,039
Semenovsky	503	0.6	2,278	-	24.00	1.20	1,145	-	120,600	6,030
Zabyty	484	0.61	1,299	-	14.72	6.45	629	-	71,228	31,202
Granichny	209	1.25	954	-	11.02	1.38	200	-	23,073	2,889
Vostochny	80	0.95	1,272	-	12.80	0.27	101	-	10,184	215
Nizhny Endybal	2,020	3.00	874	-	3.06	1.38	1,766	-	61,868	27,897
Privet	115	1.28	994	0.50	4.00	0.83	114	0.49	4,608	956
Mysovoe	194	0.29	2,435	-	36.08	0.70	472	-	70,000	1,358
Bezymyanny	661	0.76	1,965	-	0.51	0.17	1,300	-	3,367	1,143
Vertikalny	322	1.10	1,529	-	5.90	6.70	493	-	18,977	21,550
Olgina	121	0.62	2,640	-	1.65	0.68	318	-	1,988	817
All Other Mineralisation	1,216	1.00	2,281	0.15	15.20	1.60	2,774	0.22	184,840	19,457
Total/Weighted Average	16,764	-	1,809	0.24	9.78	0.95	30,329	1	1,638,758	158,706

Source: Young (2004)

Table 6.2 summarises the historical Russian inventory by category according to the Russian classification system.

Table 6.2 Summary of the Historical Russian P1 and P2 Inventory for the Property

Russian Reserve Category	Tonnes (kt)	Average Width (m)	Average Grade			Contained Metal		
			Ag (g/t)	Pb (%)	Zn (%)	Ag (t)	Pb (t)	Zn (t)
Total P1 Reserves	7,881	1.34	1,717	8.28	0.80	13,529	652,524	62,707
Total P2 Reserves	8,883	0.93	1,891	11.10	1.08	16,799	986,233	95,999
Total	**16,764**	**1.09**	**1,809**	**9.78**	**0.95**	**30,328**	**1,638,757**	**158,706**

Source: Young (2004)

All inventory calculated in 2000 were reviewed by JSC Yangeologia, a federal geological agency within Russia.

Historical C class inventory have also been estimated for the Property using the Russian classification system (see Table 6.3).

Table 6.3 Historical Resources for the Property

Mineralised Zone	Russian Category	Contained Ag (Moz)	Au Grade (g/t)
Vertikalny	C1	14.17	729
Vertikalny	C2	22.72	588
Mangazeisky North	C2	10.86	1,656

The C1 and C2 inventories are based on a Russian TEO Conditsy Report (TEO) (Russian prefeasibility study) of December 2011 by Oreall Engineering and confirmed by Russian State Commission on Mineral Reserves (GKZ) in Protocol document No. 2754 (GKZ ROSNEDRA) of 13th April 2012.

The cut-off grade used for the C1 and C2 material was 75 g/t silver. Based on protocol document No. 2754, Silver Bear received its Certificate of First Discovery from ROSNEDRA on 2nd August 2012. The C2 Resource (Mangazeisky North) was based on a 2007 JSC Yangeologia annual property report filing.

The historical Russian Reserve and Resource estimates do not conform to the CIM Standards and Definitions for Resource Estimates. A QP has not carried out sufficient work to verify the historical estimates as current Mineral Reserves and Mineral Resources and therefore, Silver Bear is treating the numbers as historical only.

6.2.2 HISTORIC NI 43-101 COMPLIANT RESOURCE ESTIMATES

In March 2009, Tetra Tech (formerly Wardrop Engineering Inc.) published an initial NI 43-101 Mineral Resource estimate for the Vertikalny structure (Wardrop 2009a) that was later revised in December 2009 (Wardrop 2009b). The Mineral Resource was further updated in the September 2011 PEA (Wardrop 2011) and February 2015 (Tetra Tech 2015a).

6.3 HISTORICAL PRODUCTION

There has been no historical production on the Property. Due to the remote location and lack of infrastructure there has been very little mining in this region.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

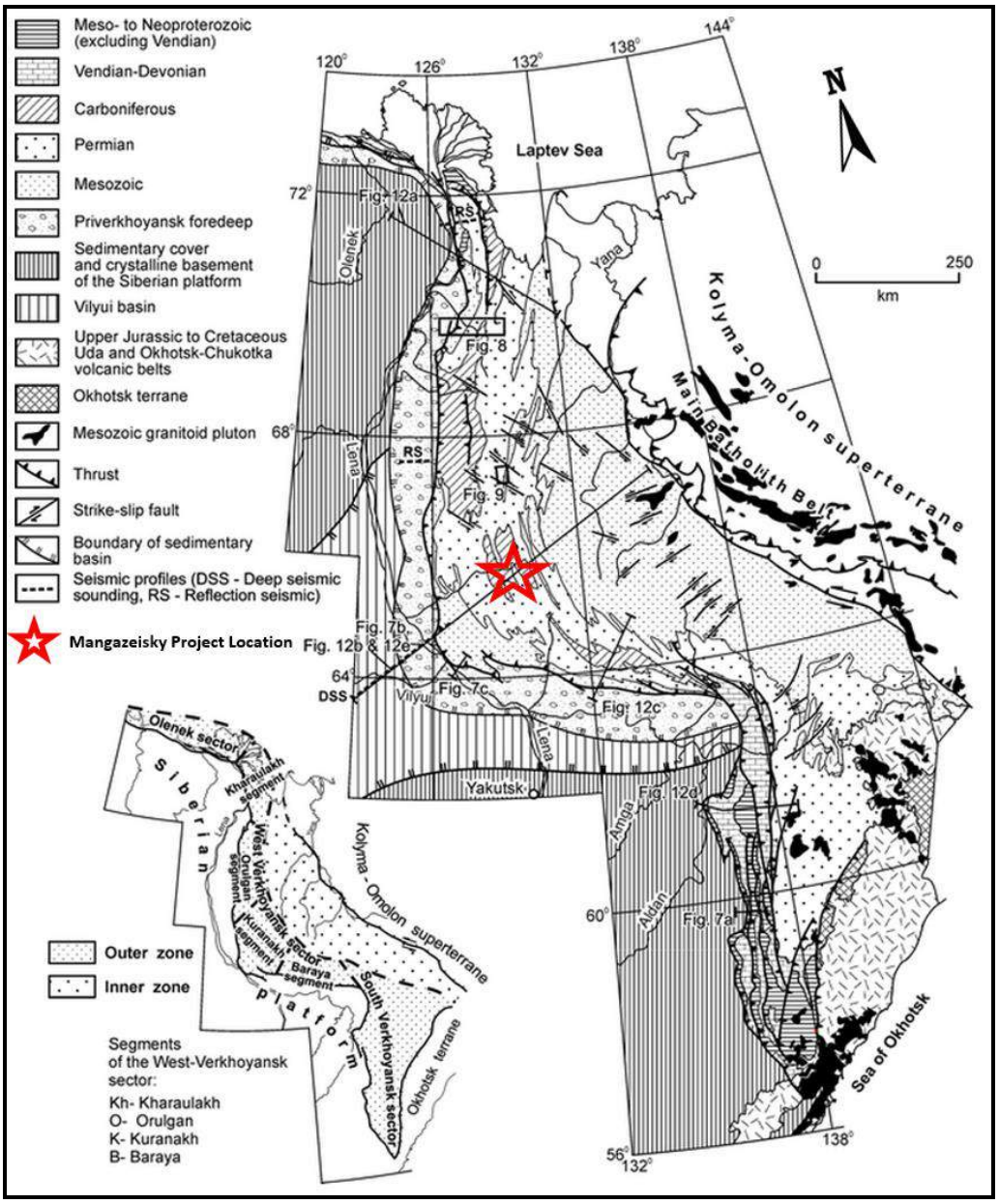
7.1 REGIONAL GEOLOGY

The Property is located within the Verkhoyansk mobile belt of north-eastern Yakutia. The fold-and-thrust belt forms part of a major orogenic system separating the Siberian North Asian Craton to the west from the immense expanse of accreted terranes that form most of Far East Russia. The belt extends for 2,000 km from the Laptev Sea to the Sea of Okhotsk (Figure 7.1). It is comprised of a greater than 7 km thick sequence of Late Precambrian to Triassic rocks deposited along the Palaeo-Pacific margin of the Siberian Craton, which developed because of rifting events during the Late Precambrian period and again during the Late Devonian to Early Mississippian periods.

Deformation during the Late Jurassic to Early Cretaceous occurred with low-grade metamorphism in the internal parts of the belt and the emplacement of high-level granitic bodies. Strike-slip faulting occurred during the Tertiary period.

The central part of the belt is dominantly comprised of a thick, monotonous succession of Carboniferous and Permian turbidites that are metamorphosed to lower-greenschist facies. Granodiorite and granite plutons have intruded the core of the belt and are associated with extensive gold-bearing quartz vein systems.

Figure 7.1 Regional Geology of the Property



Source: Miller et al. (2002)

7.2 PROPERTY GEOLOGY

The mineral occurrences on the Property are located in the interfluve of the Nuektame and Arkachan rivers and ranges from the mid-channel of the Muhalkan River to the mouth of the Mysovoi Stream. The mineral deposits are confined to the eastern wing of the Kuranakhsky anticlinorium and are situated within the contours of the Endybal anticline.

Higher order asymmetric anticlinal and synclinal folds that are 2 to 20 km in length complicate the hinge portion of the Endybal anticline. The Mukhalkansky anticline with a north-northwest strike (340°) and the Mangazeisky syncline, which are

adjacent on the western side, complicate the eastern wing of the Endybal anticline. The dip of the rocks of the Endybal anticline in the area of the Mangazeisky deposits averages 20 to 45°.

Two north-south zones of faulting are distinguished. The most important is the Nuektaminsky shear zone which limits the expansion of mineralisation to the east.

The movement along the Nuektaminsky fault is left-lateral. The next is the Granichny shear zone with right-lateral movement. The Granichny fault is a splay off the Nuektaminsky fault and together they form a V-shape, which represents a zone of tension. The Granichny shear zone hosts two vertical deposits (Semenovsky and Granichny) that cross-cut the stratigraphy.

The silver mineralisation is epigenetic. This is particularly clear at the Nizhny Endybal deposit, where the mineralised strata-bound zones exhibit structural deformation. The strata-bound veins and cross-cutting fracture systems in the region are associated with shearing. This is reflected in the common occurrence of parallel layers of slip planes along the boundaries of the sandstone beds and selvages of strata-bound veins, such as Mikhailovsky, Olgina, Vasilievsky and Anglesite-Cerussite. The structural analysis of joints in the mineralised rocks shows the ubiquitous existence of overthrusting dislocations (Kostin et. al.1997).

The Endybal diatreme and quartz porphyry dykes postdate the overthrust event. The geological studies by Yangeologia and by A.V. Kostin et al. have led to the following conclusions regarding the sequence of events:

1. mineralised layers of sandstone containing sulphide mineralisation
2. structural deformation
3. Endybal diatreme.

7.3 MINERALISATION

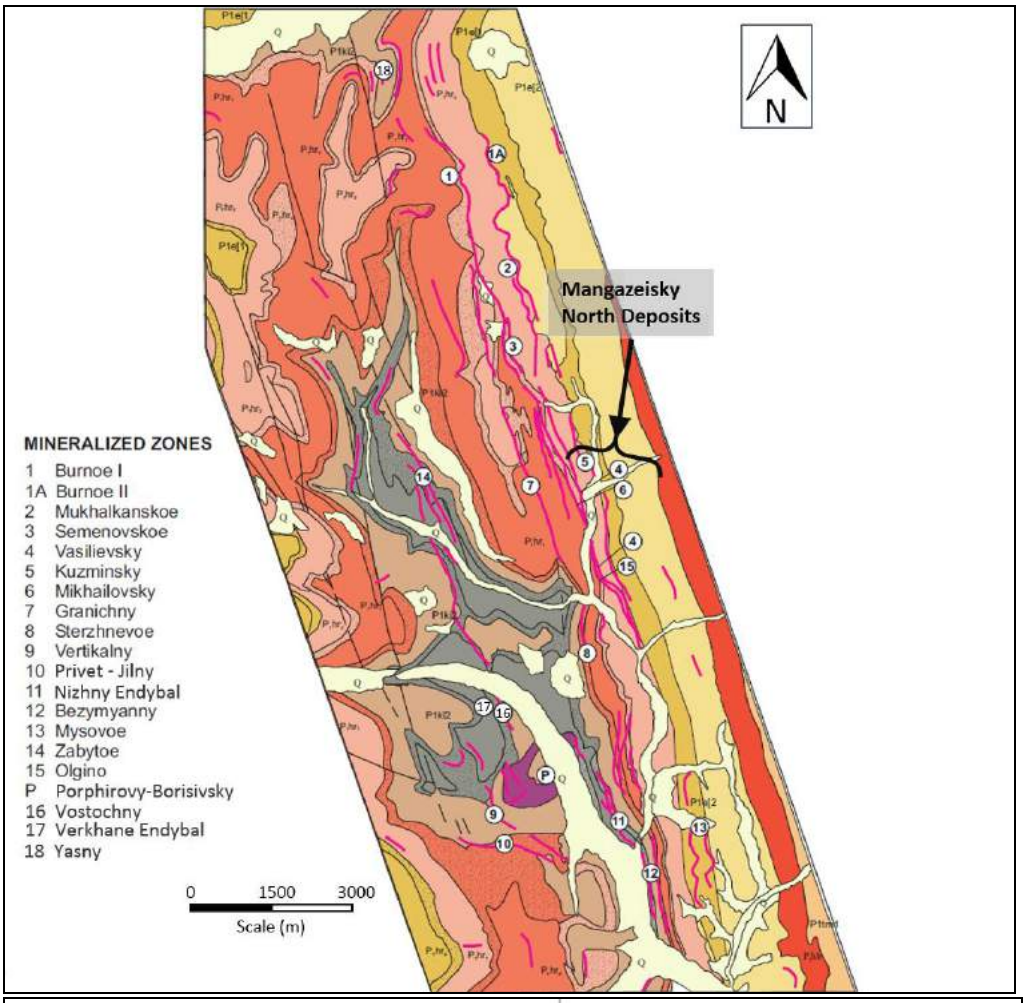
This region of Yakutia is known to contain many silver deposits; Figure 7.2 illustrates the position of the Property in relation to a number of the known silver projects in the region.

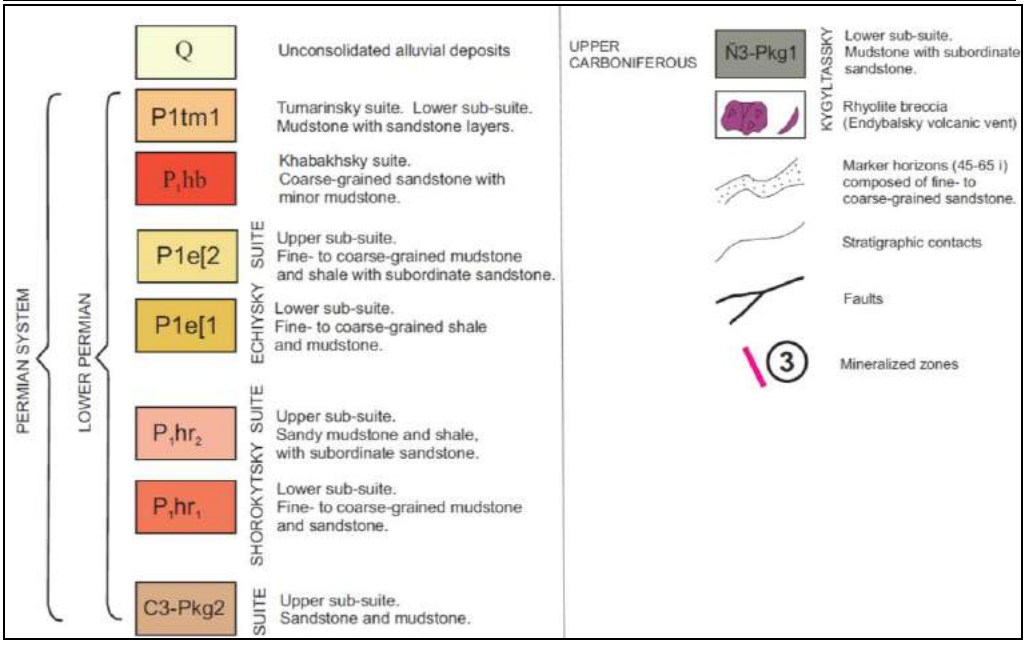
Figure 7.2 Silver Projects within the Sakha Republic of Yakutia

Source: Silver Bear

The Property contains several explored areas that host more than 100 occurrences of mineralisation. The mineral occurrences are concentrated within a 35 km long corridor (Figure 7.3).

Figure 7.3 Exploration Targets and Property Geology





Source: Tetra Tech
Notes: Orogondia and Kis-Kuel located further south than then southern extent of Figure 7.3.

The Property hosts at least four different types of mineralisation:

- Strata-bound silver-bearing, quartz-carbonate-sulphide structures with average grades greater than 900 g/t silver, with lead and zinc by-products. These silver bearing structures were mapped and sampled over an area of 65 km^2. Examples of this are the Vasilievsky—Anglesite-Cerussite and Olgina—Mikhailovsky veins within the Mangazeisky North zone.
- Thicker linear-type stockwork areas with carbonate-silver sulphosalt mineralisation. Examples of this occur in the Strezhevoy and Nizhny Endybal Zones.
- Narrow, steep dipping veins such as Vertikalny that cross-cut stratigraphy and feature grades in excess of 1,000 g/t silver over widths ranging from several centimetres to several metres. Vertikalny and possibly Zabytoe and Kis-Kuel are examples of this style of mineralisation.
- A porphyry area associated with quartz, quartz-carbonate and quartz-sulphide veins and veinlets, hosted by extrusive rhyolite porphyry. Porfirovy is an example of this.

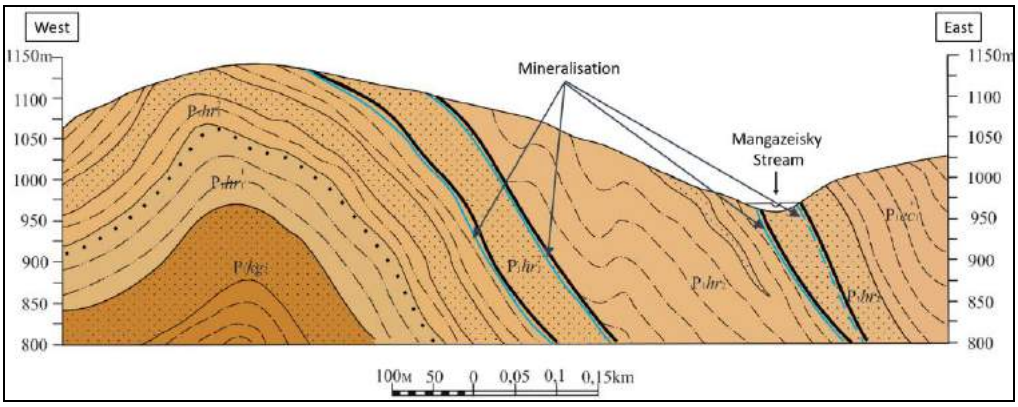
Each mineralised area is described in the following sections.

7.3.1 MANGAZEISKY

The Mangazeisky North deposits comprise of a series of sub-parallel shallowly dipping stratiform epithermal veins, hosted within a package of sediments belonging to the Echinyskaya, Khorokiskaya and Kigiltasskaya suites. The host rocks comprise a series of interbedded siltstones, sandstones and argillites,

The area is dominated by a north-north west/ south-south east striking anticlinal fold, Figure 7.4. Bedding planes and mineralisation dips at between 20 and 40° towards the East.

Figure 7.4 West to East Cross Section through Mangazeisky North



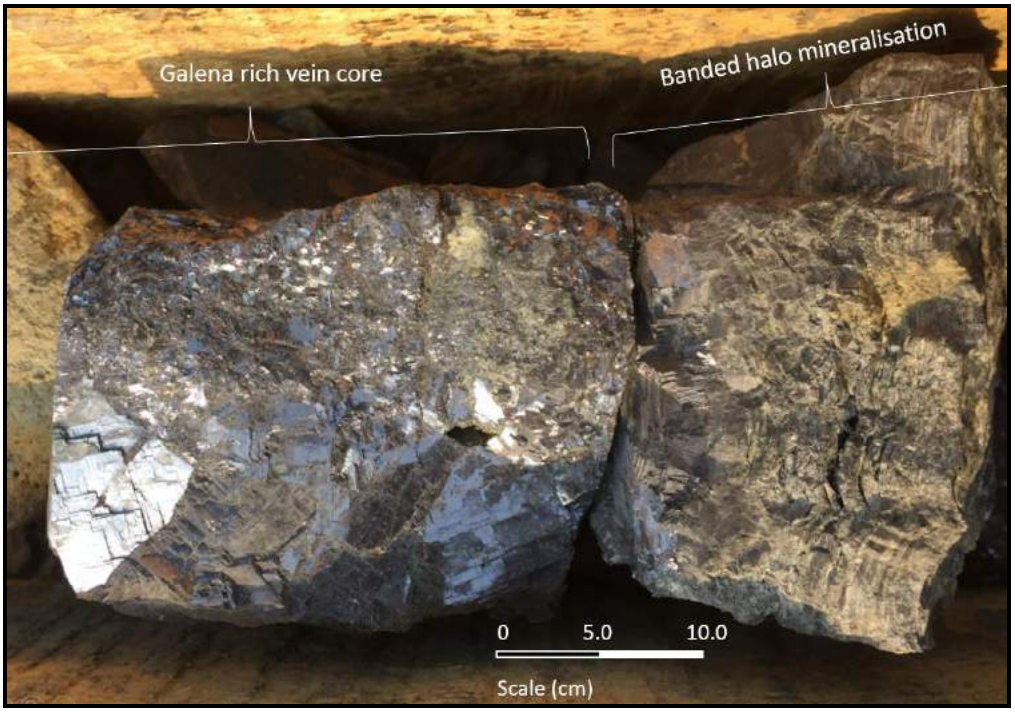
Source: Silver Bear

The mineralization of Mangazeisky North is mainly associated with banded brecciated textures. The following minerals are present: Native silver, galena, acanthite, stephanite, polybasite, canfieldite, pyrargyrite, miargyrite, diaphorite, tetrahedrite, owyheeite, freieslebenite, boulangerite, arsenopyrite, sphalerite, pyrite, and chalcopyrite.

The veins are strata-bound, occurring on contacts between strata, particularly around sandstone units, and tend to be dilated towards the hinge of the local fold structure. It is noted that the strongest silver mineralisation occurs within the central core of the veins, which tends to be comprised of massive galena. The veins have a halo of mineralisation occurring on the footwall and hanging wall. The halo mineralisation takes the form of brecciated and banded material comprising fine grained light grey siderite and quartz (Figure 7.5).

The brecciated sandstones associated with the veins have a thickness up to 0.5 m with the contacts of the brecciated zones being sharp and defined by early shearing. The shear commonly exhibits slickensides with horizontal striations. The interpretation of tectonic stress fields for the southern flank of the Vasilievsky Zone (Kostin et al. 1997) indicates that mineralised shoots formed in areas where both vertical and horizontal displacement occurred.

Figure 7.5 Core Samples from Mangazeisky North Showing Stratiform Mineralisation



Source: Tetra Tech

A third form of mineralisation is also observed at the Mangazeisky deposits, taking the form of very narrow sub-vertical oxidised veins (Figure 7.6). It is noted that this form of mineralisation is locally high grade, but extremely narrow and therefore does not form part of the Mineral Resource. This mineralisation is likely to be later phase and structurally controlled.

Figure 7.6 Core Sample from Mangazeisky North Showing Narrow Sub-Vertical Vein

Source: Tetra Tech

7.3.2 VERTIKALNY

The Vertikalny vein has been traced on surface for 5.1 km. The under-explored southeast portion, identified in 1990 by Yangeologia, possibly extends to the Endybal River valley. To the northeast, along the areas where strongly oxide material is identified, the vein swells from 3 to 15 m. Further northeast, the zone splits into two. The eastern branch, which is obscured by a siltstone bed, was traced by individual grab samples in erosional windows.

The mineralisation occurs as sandstone breccias with quartz or siderite cement, abundant hydrous ferric oxides that results from the oxidation of sphalerite and galena. In the Northwest Zone, potential mineralisation was traced in float and frost heaves up to the head of Krainy Creek. Further along, the mineral zone enters the right side of the Serilenge River, where it is tested by trenches and found to contain galena-siderite mineralisation. Along the entire length, mineralisation is expressed as breccias with siderite-sphalerite-galena and various quantities of silver sulphosalts. Mineralisation is usually associated with the presence of dykes of intermediate to basic composition.

The Vertikalny Zone is not a single linear feature, but rather a combination of conjugated faults and breccias with various mineral compositions and quality. Structurally, the Vertikalny Zone is associated with the right-lateral strike-slip fault zone, which can be readily identified on IKONOS satellite images. Exploration of the structure has revealed sub-parallel branches with potentially prospective mineralisation similar to that found in the Central Zone. The intersections of the main structure and vein splays can result in the development of thicker intersections of mineralisation.

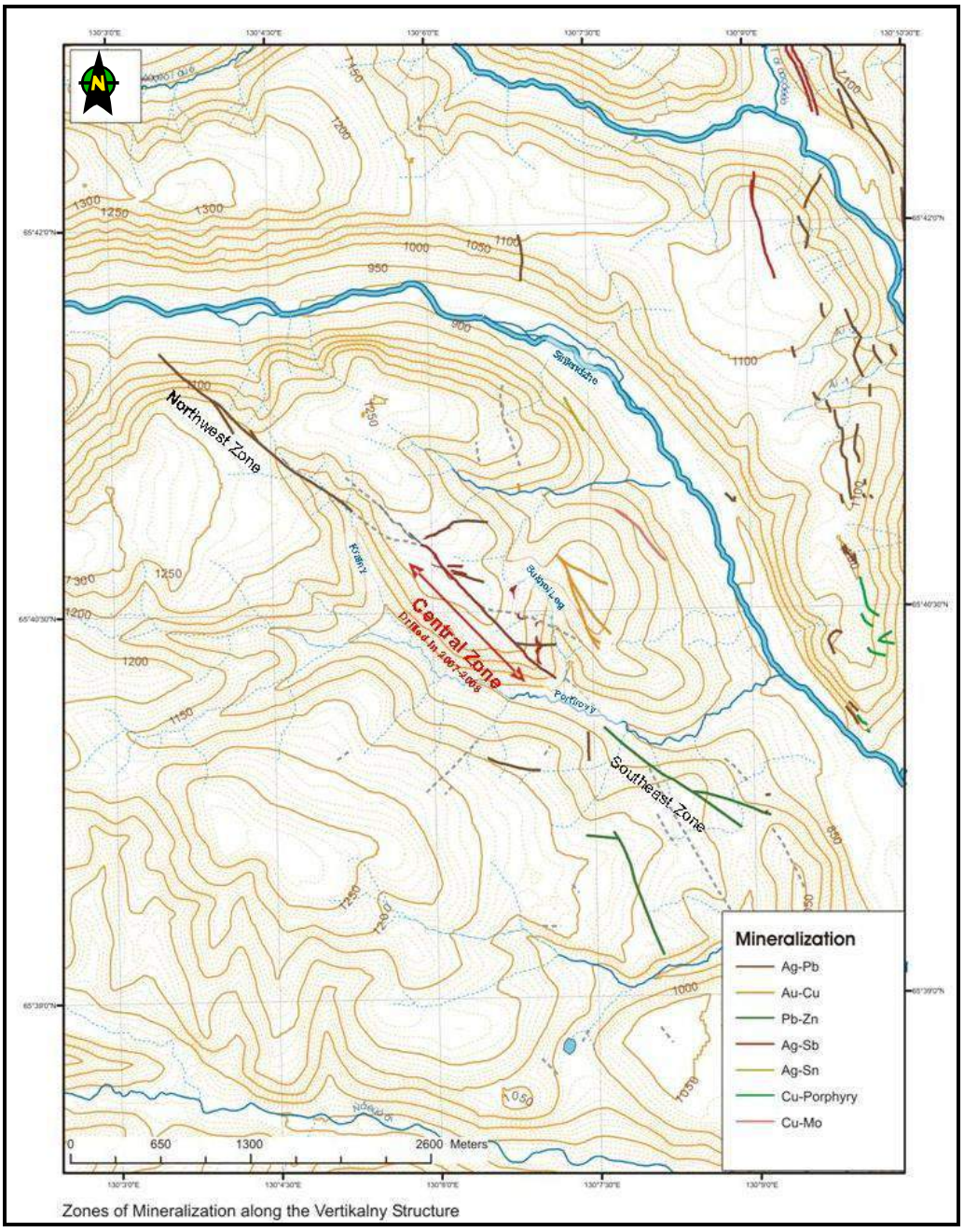
The mineral composition along the length of the Vertikalny structure is represented by galena and sphalerite, with subordinate values of other sulphides. However, early geochemical association analysis has shown three mineralisation types, which

characterise the zones: The Northwest Zone (silver-lead-zinc), the Central Zone (silver-lead-zinc-tin) and the Southeast Zone (lead-zinc).

The three zones of the Vertikalny structure are presented in Figure 7.7.

The total drilled strike length of the Vertikalny structure is some 2,400 m. Drilling has tested portions of the structure to a maximum depth of 500 m from surface. Figure 7.8 shows a cored mineralised intersection from drillhole V13-020.

Figure 7.7 Mineral Zonation of the Vertikalny Structure



Zones of Mineralization along the Vertikalny Structure

Source: Silver Bear

Figure 7.8 Beginning of the Mineralised Intersection from Drillhole V13-020

Source: Silver Bear

7.3.3 ZABYTOE AND ZABYTOE-2

The Zabytoe occurrence was discovered in 1934 and further explored in 1992 to 1994. Mineral occurrences are concentrated on the walls and in the watershed of the Fedor-Yuryage River (refer to Figure 7.3).

This area is referred to as the 'up-structure' of the Endybal anticline and is composed of nearly horizontal beds of sandstones and siltstones (Kygyltassky Suite). The stratigraphy is cut by sub-parallel; sub-vertical felsic dykes and later intermediate to basic dykes, of up to 5 m thick, which occur along a strike length of 2.5 km. The host rocks are sheared and broken by a number of fractures, often gouge-bearing, which control the branching, sub-parallel and steep-dipping zones of mineralisation. The structural disturbance is related to the Granichny fault system.

Mineralisation occurs as a structurally disturbed zone that intersects stratigraphy under a gravelite layer in the same stratigraphic level as the Strezhevoy mineralised body. The total width of the steeply dipping zone is 50 to 60 m (up to 2 km), striking at 315 to 340°. A single trench, developed by Yangeologia in 1991, returned 1,906.5 g/t silver, 59.3% lead and 1.85% zinc from sandstone breccia cemented with sulphides.

The following minerals are present in the Zabytoe occurrence: galena, sphalerite, tetrahedrite, chalcopyrite, pyrite, boulangerite and arsenopyrite. The association is silver-galena-sphalerite-siderite. Mineralisation occurs with galena, sphalerite and siderite-silver sulphosalt aggregates.

There are two parallel zones of mineralisation located within 200 to 300 m from each other – the Western and Eastern (A.I. Nekrasov et. al. 2004).

The mineralisation potential is promising, given its structural and mineralogical similarity with Vertikalny. It is evident from the IKONOS satellite image that the potentially mineralised areas of Zabytoe can be traced south toward the Porfirovy site.

7.3.4 KIS-KUEL

The Kis-Kuel site is located approximate 20 km southeast of Vertikalny, in the western limb of the Endybal anticline (this is some 10 km to the south of the southern limit of Figure 7.3).

The geology at Kis-Kuel consists of granitoid intrusive bodies beneath sedimentary units. The sediments above the stocks contain veinlets of sphalerite. The stock itself hosts quartz-galena-chalcopyrite veinlets, brecciated intermediate dykes, brecciated sandstone with argentiferous healed with galena-quartz cements, oxidised intrusive with disseminated sulphides. In most cases the stock is obscured by the overlying sediments.

The mineralisation is controlled by linear structures associated with the Nuektaminsky fault as well as structures formed during the crystallisation of the Kis-Kuel pluton.

This style of mineralisation has no similarities to others seen on the exploration licence and therefore, warrants further investigation. Lead and zinc mineralisation, previously only seen in the margins of intrusive bodies, occurs in the centre of the Kis-Kuel pluton, which might indicate the presence of an undiscovered copper–molybdenum or gold-copper porphyry. There are also several occurrences of arsenopyrite-quartz veining, which indicates the potential for gold mineralisation.

7.3.5 OROGONDIA

The Orogondia deposit is located some 20 km to the south of the Mangazeisky North deposits and 8 km to the south of the Endybal airstrip.

Based on data by V.V. Shoshin (Nekrasov A.I. et. al. 1997) mineralisation at Orogondia is connected with a near conformable mineralised zone of shearing with a strike of 325 to 345° that can be traced for up to 4 km. Most of the mineralisation is concentrated on the surface of medium-grained sandstones as sulphide-quartz-siderite veinlets, of 0.5 to 2 m wide (average thickness is 0.8 m). The reported length of the mineralised body is circa 1 km.

The grade varies from 138 to 886 g/t silver, 4.83 to 25.44% lead, and 2.12 to 7.78% zinc. The following minerals are present: galena, sphalerite, pyrite, tetrahedrite, arsenopyrite, chalcopyrite, stannite, owyheeite, diaphorite, boulangerite. Based on the data from Yangeologia, the Orogondia showing was evaluated from nine grab samples, selected from float and subcrop.

Mapping and prospecting in 2008 led to the discovery of a strata-bound zone of oxidised quartz-carbonate which measures 0.8 m wide, with a strike length of 200 to 1,200 m. This occurrence is described as potentially non-argentiferous as it contains only sparse quantities of disseminated sphalerite and galena.

The work in 2008 failed to locate the occurrences of the mineralisation as reported by Yangeologia within 4 km of its proposed location. It is likely that there were several different occurrences, which combined and appeared to be a single feature.

7.3.6 STERZHNEVOE

This zone is a complex vein system of approximately 600 m in length with a thickness of up to 5 m, but averaging approximately 0.9 m. The thickness of the individual veins within the larger vein system varies from 10 to 20 cm. In addition, parallel mineral zones are associated with the main deposit. The average grade of silver of the zone is 2,652 g/t. The mineralised veins are situated within medium and coarse-grained sandstones and siltstones, overlying a lower conglomerate unit. A significant amount of a black organic substance (shungite) is present in the cement, binding the particles of the sedimentary units together. In the areas where there has been hydrothermal reworking of the sandstones, shungite forms lenses and flakes in association with coarse-crystalline drusy quartz. The underlying layers of sandstone contain disseminated silver mineralisation and are within the same stratigraphy as the Nizhny Endybal deposit.

The Sterzhnevoe Zone was formed as a result of remobilised mineralisation from underlying silver rich sulphide-bearing sandstones, into structures formed in a shear zone with numerous secondary downthrown blocks and overthrusts (Kostin et. al., 1997). There is a downthrown layer with a thick zone of boudinage and jointing situated parallel to the main zone. The Sterzhnevoe Zone can be traced from the Bezymyanny deposit to the Zabytoe mineral occurrence and is characterised by barren hydrothermal activity, expressed in remobilisation of an organic substance (shungite) and the formation of quartz druses from sandstone.

Tracing the hydrothermal rock alterations through the boudinage zone along the strike of the siltstone layer to the north of the Sterzhnevoe Zone as indicated by the preservation of the vein and rock alterations (quartz crystals druses, lenses and shungite flakes) shows that silver mineralisation is absent. The assumption is that the mineral components for the Sterzhnevoe Zone were taken from stratigraphically underlying rocks of the Nizhny Endybal deposit and hydrothermal activity channelled their remobilisation into more favourable structures.

7.3.7 MUKHALKANSKY

The Mukhalkansky mineralisation is represented by two galena bearing veins. The first vein, which cross-cuts the stratigraphy has been traced for 800 m and has an average width of 0.2 m and contains an average silver content of 1,295.4 g/t. The vein is confined to a tectonic fault that was accompanied by brecciation of the enclosing rocks during its formation. The vein is orientated parallel to the Nuektaminsky fault and is believed to be its mineralisation supplying fragment. The second vein is a concordant vein in its mode of occurrence and is confined to a stratum of thin layer sandstones. The second vein branch can be traced for approximately 500 m and has a width of between 0.2 and 0.4 m. The average silver content of the second vein is 2,328.7 g/t.

7.3.8 BEZYMYANNY

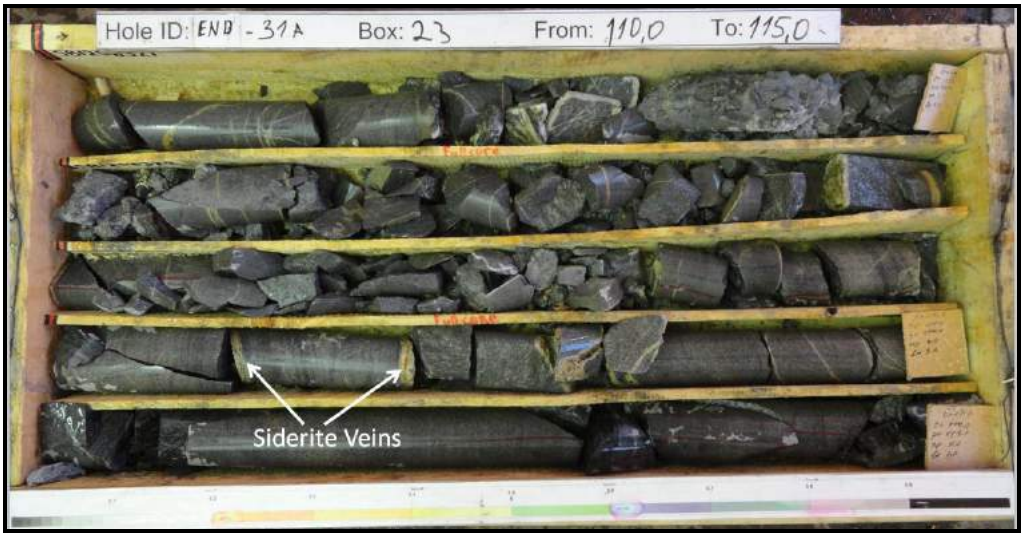
The Bezymyanny mineralisation is composed of several zones confined to inter-stratal ruptures. The mineral veins are located along the contacts of medium-grained and fine-grained sandstone strata. Veins are traced for 500 m with a thickness ranging

from between 2 to 5 cm and 34 to 40 cm. The mineralisation content varies from 756 to 24,433 g/t silver, 0.4 to 1.5 g/t gold, 0.1 to 7.81% lead and 0.1 to 0.8% zinc.

7.3.9 NIZHNY ENDYBAL

The Nizhny Endybal deposit has been known since 1774 under the name "Kokovin Silver Mine". The style of the mineralisation at Nizhny Endybal takes the form of a stockwork style network of thin veins. Concentrations of the stockwork are accumulated into lens shaped bodies centred on the hinge of the anticline as shown in Figure 7.9. The mineralisation is associated with siderite veins, which are principally parallel or sub-parallel to bedding. However, occasional sub-vertical veins are present, which cross-cut the bedding planes. The veins are predominantly 1 to 2 cm in thickness, but occasionally up to 10 to 15 cm. The thicker veins include veinlets of fahlore and vugs filled with fahlore crystals (refer to Figure 7.5 and Figure 7.6). Galena may also be present as separate veins, but the silver mineralisation does not appear to be directly related to the presence of galena.

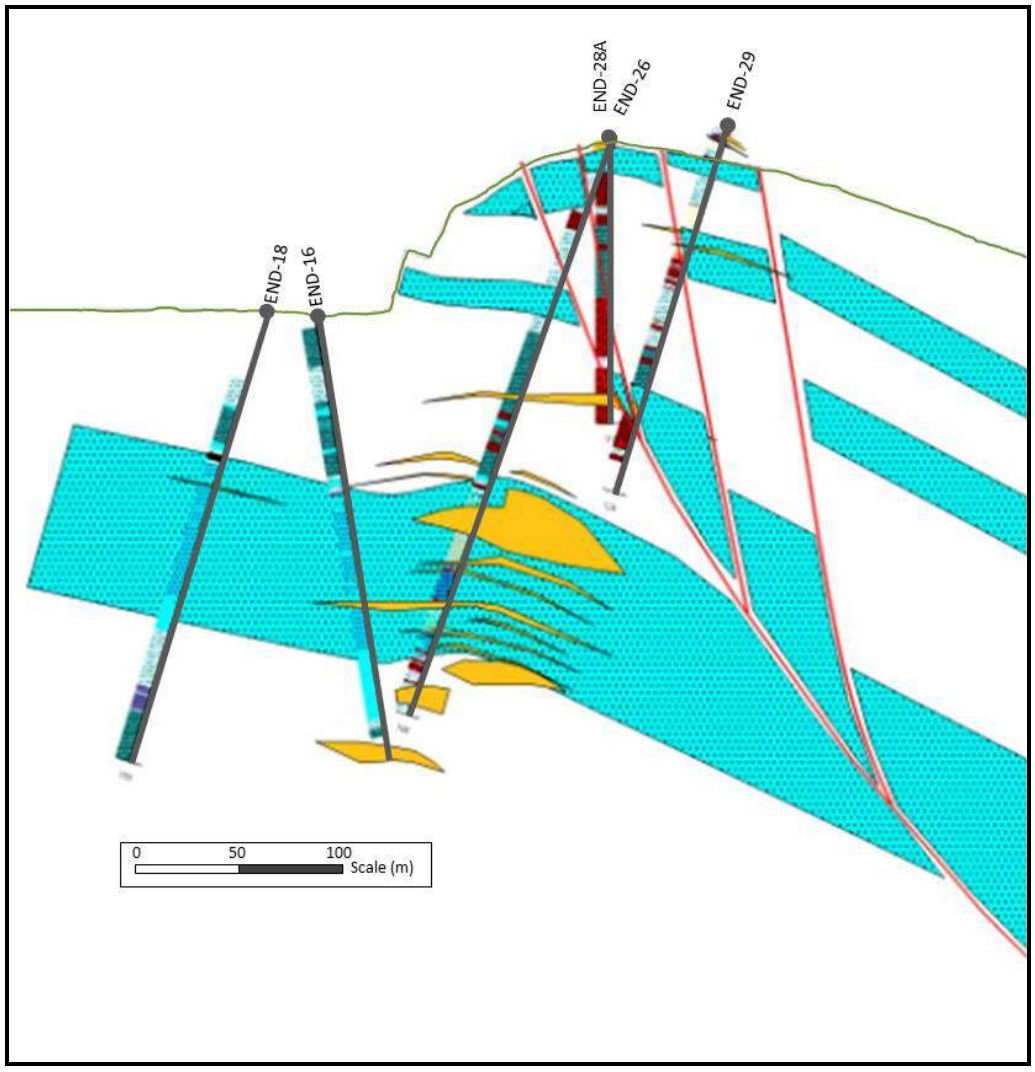
Figure 7.9 Stratiform Siderite Veining in Drillhole END-31A



Source: Silver Bear 2012.

The mineralisation is concentrated within the fold axis of the anticline and decreases with distance along the fold limbs from the axis. If a transect is taken approximately east-west, perpendicular to the fold axis, the thickness of the intersections and mean silver grades decrease to negligible at a distance of circa 40 to 50 m from the centre of the fold axis (illustrated in cross-section in Figure 7.10). In general, the width of the mineralised zones west to east (across strike) is approximately 80 to 100 m and has a continuous strike length of approximately 700 m. Discontinuous mineralisation has been traced in recent drilling to a distance of approximately 1,300 m along strike.

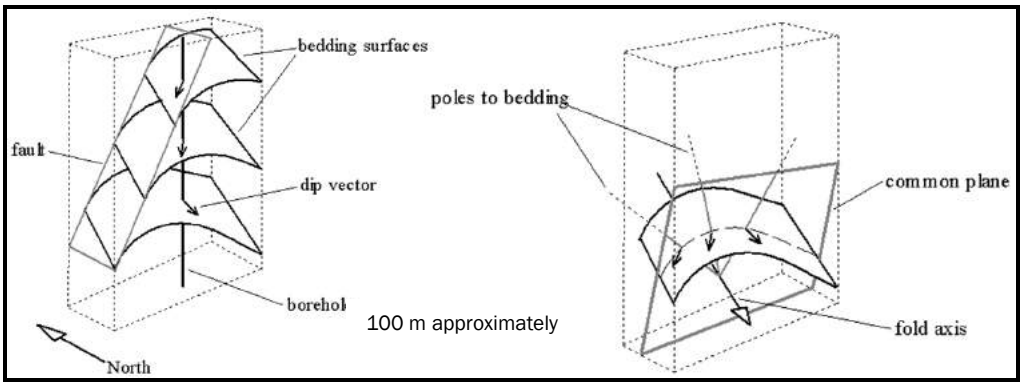
Figure 7.10 **Vertical Cross-Section through Drillholes END-16, END-18, END-26, END-28A and END-29 Looking North**



Source: Silver Bear (2012)
Note: Yellow = Lenses of mineralisation developed in antiform; Blue = sandstone units.

Structurally, the mineralisation at Nizhny Endybal is thought to relate to an anitiformal thrust zone with associated anticline system. The structure is interpreted to include a series of listric faults and is illustrated in Figure 7.11.

Figure 7.11 Representation of the Geometry of the Interpreted Structural Features



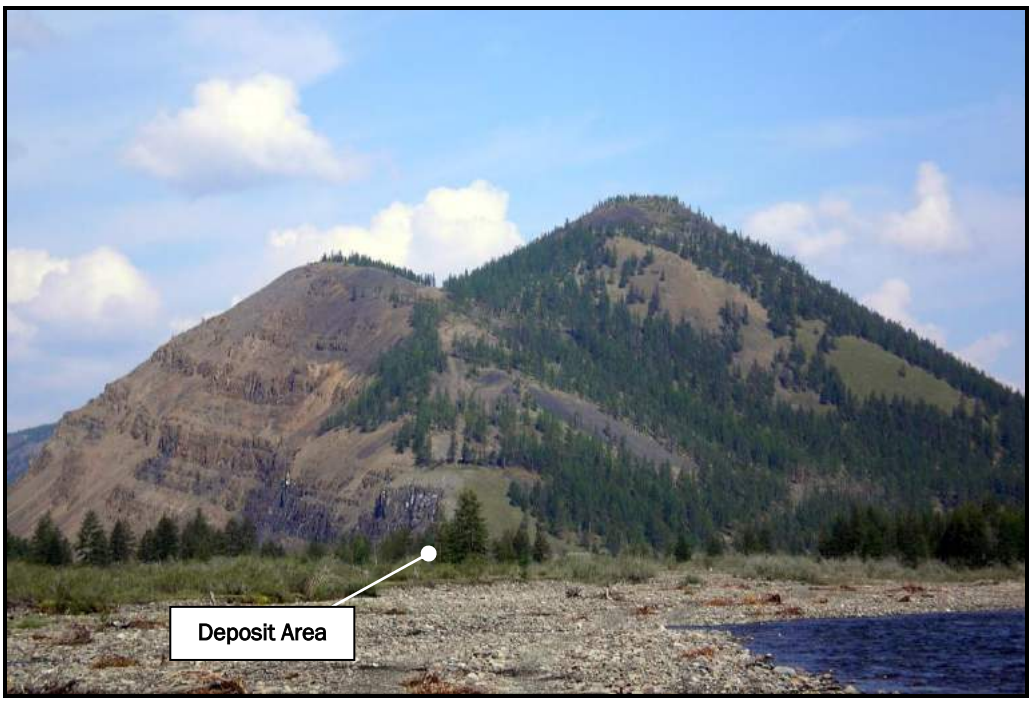
Source: Silver Bear

The mineralisation at Nizhny Endybal is contained within a series of interbedded sandstone and silty-sandstone units. It is thought that the silty-sandstone units have a lower permeability than the adjacent sandstone units and have, therefore, inhibited the flow of fluids. The sandstones are anticipated to have higher permeability and are more readily fractured and, as such, have provided preferential pathways for fluid flow and associated mineralisation.

Three major sandstone units have been highlighted within the interbedded sequence at Nizhny Endybal. The average thicknesses of the units are 17, 22 and 60 m. The upper and middle units are 17 and 22 m in thickness and comprise fine and medium-grained sandstones. The lower unit is 60 m in thickness and comprises coarse-grained sandstones. The significant mineralised intersections are associated with layers of medium-grained sandstones.

Figure 7.12 shows a general view of the prospect area, with the deposit prominent in the cliff face.

Figure 7.12 General View of Nizhny Endybal; looking north



Source: Tetra Tech

Figure 7.13 Siderite Vein with Vugs filled with Fahlore Crystals, Galena, Ankerite and Quartz

Source: Silver Bear

Figure 7.14 Siderite Vein with Tetrahedrite Crystal

Source: Silver Bear

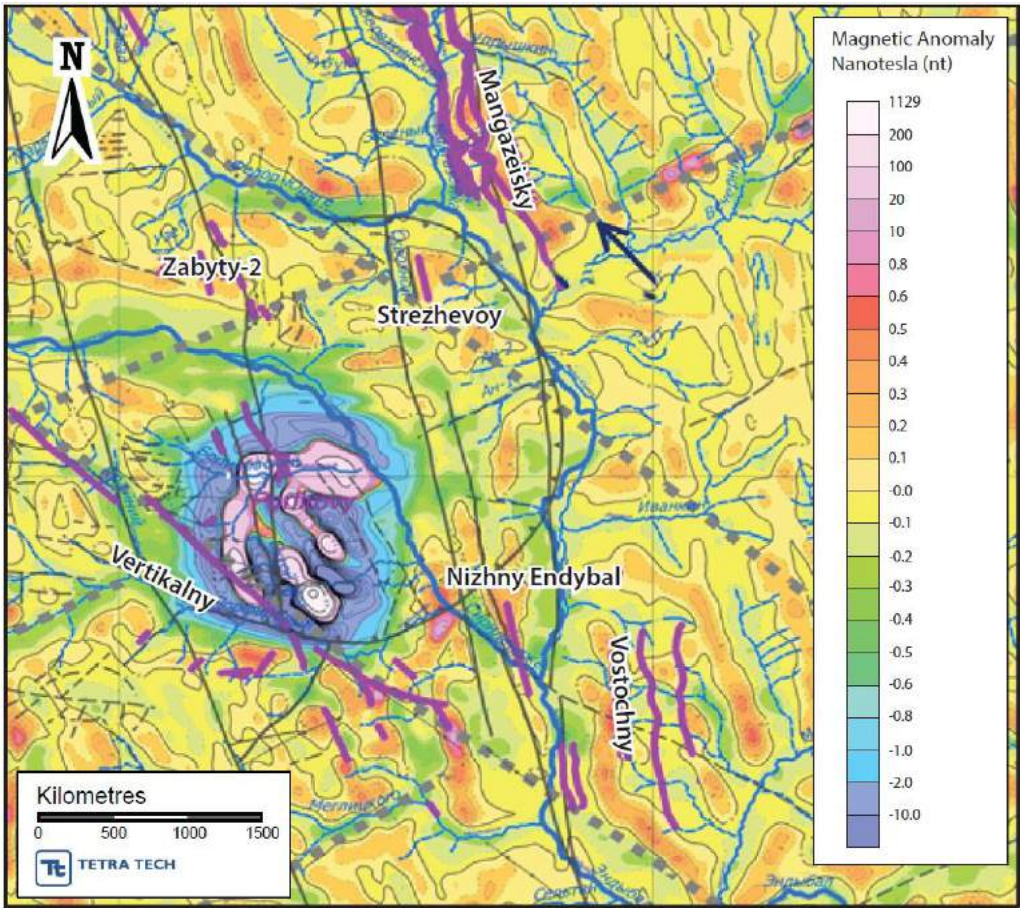
7.3.10 PORFIROVY

Mineralisation in the Porfirovy area is associated with quartz, quartz-carbonate and quartz-sulphide veins and veinlets hosted by rhyolite and dacite porphyry.

The copper and gold potential of the area has only recently been recognised and the copper-silver soil geochemical results indicate an anomalous area that corresponds with the intrusive complex. Locally the rhyolite has been silicified and the feldspars exhibit argillic alteration.

The porphyry is well defined by the magnetic anomaly survey (Figure 7.15).

Figure 7.15 Magnetic Anomaly Map



Source: Silver Bear

7.3.11 BORISOVSKY

Borisovsky occurs on the terrace along the northeast wall of the Serilenge River in the mouth of the Borisovsky Creek to the northwest. Outcrop with fragments of quartz-carbonate breccias with dispersed sulphides (mostly pyrite) as pockets and veinlets were found within the zone. The outcrops are thought to be correlated with along a north-western trend (330°), around Vostochny.

At Zapadny, 1.5 mm quartz-filled fractures, quartz-carbonate breccias and quartz-siderite formations with dispersed pyrite veinlets are hosted within fine and medium-grained sandstones. The structures general trend is 075°.

The Vostochny fault zone was traced for 400 m to the northwest along the wall of the northeastern terrace of the Serilenge River by sporadic fragments of yellow-white quartz and quartz-siderite with sulphide (pyrite, rare galena) mineralisation. In the north-western part of this zone, grab samples are of pyrite and galena that occur as disseminated inclusions in yellow-white quartz with pockets of siderite.

8.0 DEPOSIT TYPES

The Property contains two main deposit types: epigenetic polymetallic silver-lead-zinc veins (metasediment hosted), and porphyry (copper, gold, silver). The majority of the exploration, and the existing resources, are focused upon the polymetallic silver-lead-zinc veins. The veins are sulphide-rich, containing sphalerite, galena, silver, and sulphosalt minerals in a carbonate and quartz gangue.

At Vertikalny the veins are emplaced in and along fractures and faults in a sedimentary basin dominated by clastic rocks that have been deformed, metamorphosed, and intruded by igneous rocks. The veins are steeply dipping and narrow.

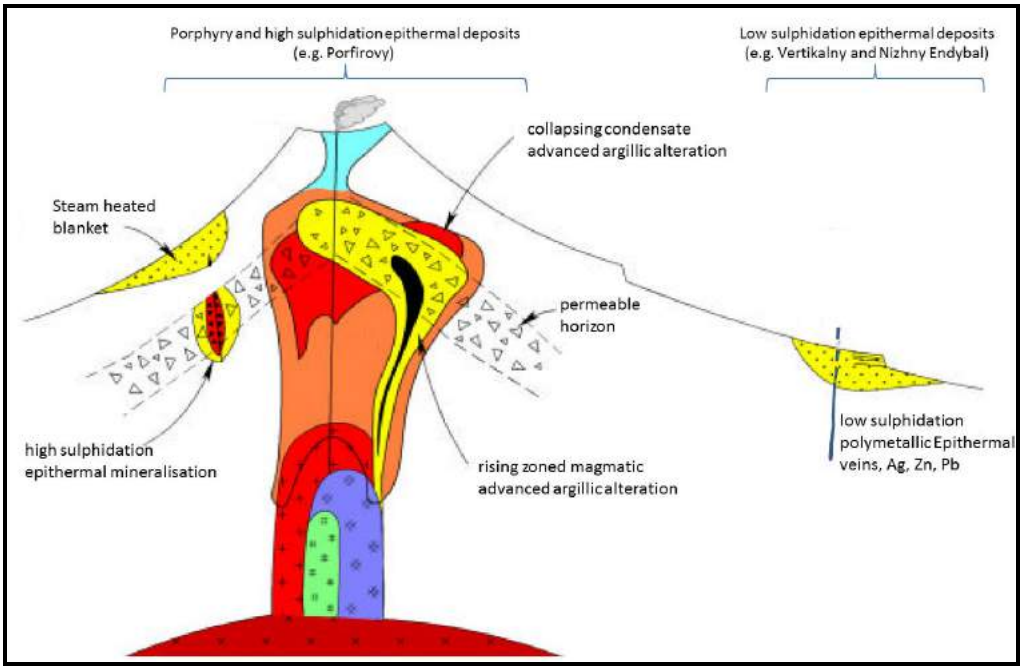
The Mangazeisky North deposit comprise of a series of sub-parallel shallowly dipping stratiform epithermal veins. The mineralization of Mangazeisky North is mainly associated with banded breccias texture and represented with siderite-sphalerite-galena, quartz-pyrite-arsenopyrite, and ankerite-sulphosalts forming a silver rich polymetallic mineralisation.

At Nizhny Endybal a stockwork of cross-cutting polymetallic veins are concentrated within strataform lenses of sandstone and siltstone. The mineralisation is concentrated to the centre of the Endybal anticline fold axis.

In many cases, the geology and morphology of the veins appear to be similar to the Prognoz property, which Nordgold NV is developing some 160 km to the east of the Property. The Porfirovy area of the Property represents a potential copper-gold-silver porphyry deposit, related to a sub-volcanic intrusive body.

Figure 8.1 presents a section through a conceptual porphyry and epithermal deposit model that could be applied to parts of the Mangazeisky licence.

Figure 8.1 Idealised Section through a Porphyry and Epithermal Deposit Model



Source: Corbett (2009)

8.1 GEOLOGICAL CONCEPTS BEHIND EXPLORATION PROGRAMME

At Vertikalny and Nizhny Endybal, structural geology influences the extent and distribution of the mineralisation. Surface trenching and diamond drilling campaigns have been targeted to systematically explore profiles along the strike length of the interpreted structural features (i.e. along the fractures and faults at Vertikalny, and along the fold axis of the Endybal anticline).

The Mangazeisky North deposits are known to be associated with lithological boundaries; therefore the focus of the trenching has been around outcropping lithological contacts. The strike and dip of the lithological units was taken into account when designing the drilling campaigns to ensure that the intercepts are normal to the strike and dip of the veins.

Geological mapping, historical geophysical surveys, and soil sampling campaigns guided the initial drilling and trenching programmes.

9.0 EXPLORATION

9.1 2004 TO 2010 EXPLORATION PROGRAMME SUMMARY

The exploration campaigns completed prior to 2010 have been disclosed in Wardrop (2011) completed by Tetra Tech (formerly Wardrop Engineering Inc). Table 9.1 provides a summary of the exploration completed prior to 2015.

Early exploration by Prognoz, Silver Bear's subsidiary, was focused upon the narrow, strata-bound silver mineralisation of the Vasilievsky and Mikhailovsky veins at Mangazeisky North. From 2007, the focus shifted to the development of the Vertikalny deposit and included the exploration activities on the thicker, linear, stockworks at Nizhny Endybal.

Table 9.1 Historic Exploration Activities at the Property

Year	Exploration Activities	Targets Explored
2004	No trench exploration was undertaken during 2004.	-
2005	9,641 m³ of trenching.	Vasilievsky, Mikhailovsky, Sterzhnevoy, Nizhny Endybal
2006	4,843 m³ of trenching and mapping.	Nizhny Endybal, Vostochny, Sterzhnevoy, Vertikalny
2007	8,000 m³ of trenching.	Vertikalny
2008	22,633 m³ of trenching. Mapping, lithochemical sampling, direct current induced polarisation/magnetotellurics and magnetic anomaly geophysical surveys.	Vertikalny, Zabyty, Zabyty-2, Kis-Kuel, Orogondia
2009	15,067 m³ of trenching. Lithochemical sampling, magnetic anomaly mapping.	Nizhny Endybal, Vertikalny, Kis-Kuel, Mukhalkan-Burny
2010	No exploration was undertaken during 2010.	-
2011-2012	1,600 m³ of trenching.	Nizhny Endybal
2013	52 trenches at regular intervals with 474 m of sampling.	Mangazeisky North and South
2014	19 trenches across multiple exploration targets.	Vertikalny, Mangazeisky South, Porfirovy and Sterzhnevoy
2015	8 trenches for a total length of 598 m.	Porfirovy and Sterzhnevoy

9.2 2011 AND 2012 EXPLORATION PROGRAMME

Silver Bear re-commenced exploration work during the 2011 and 2012 field seasons, which included a trenching programme. A further six trenches were excavated within the central part of the Nizhny Endybal deposit, totalling 1,600 m³.

Table 9.2 presents a summary of the significant mineralised intersections from the trenches excavated at Nizhny Endybal during in 2011 and 2012.

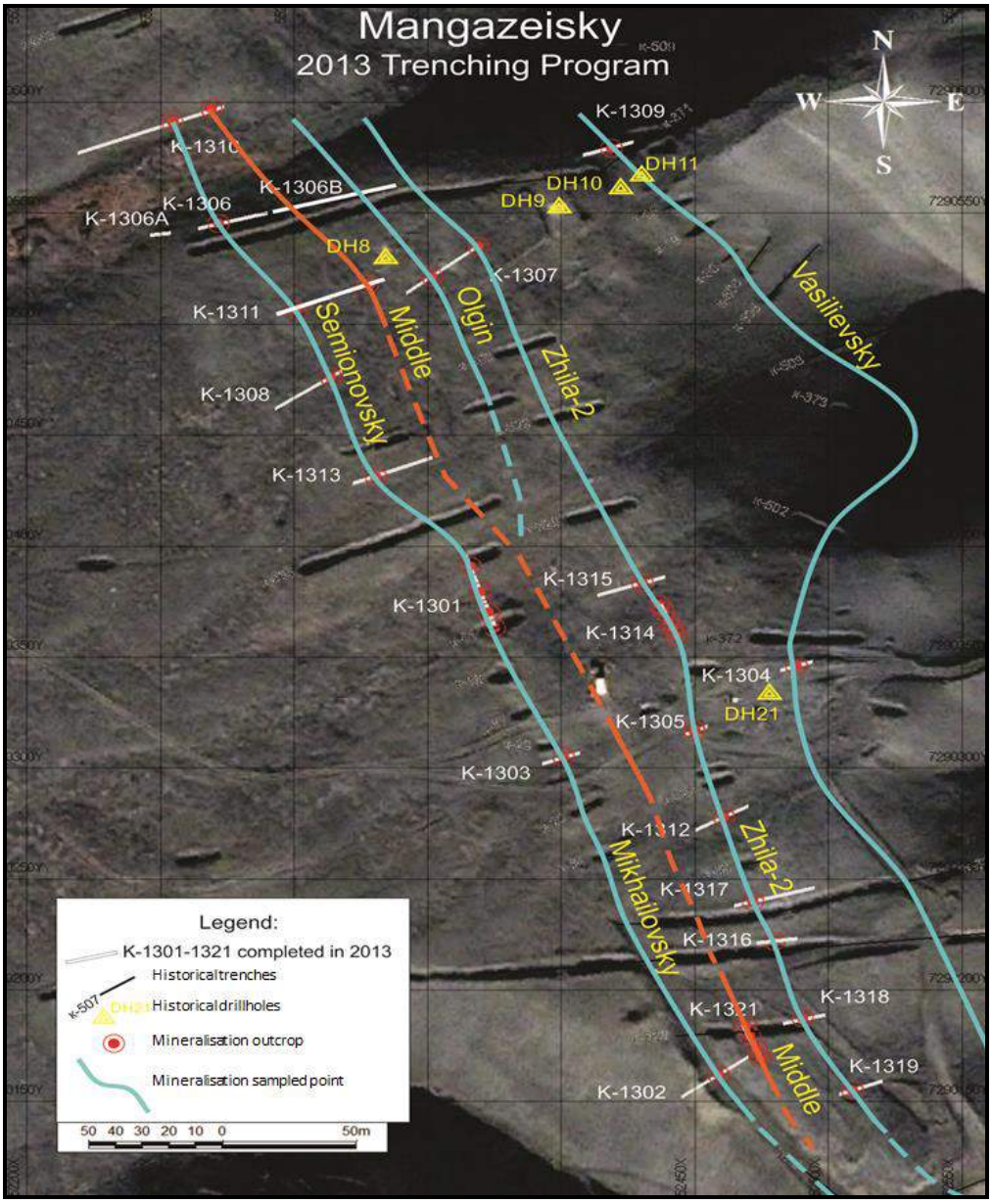
Table 9.2 Summary of Intersections from 2011 to 2012 Trenches at Nizhny Endybal with a Silver Grade of >75 g/t

Trenches	From (m)	To (m)	Intersection Length (m)	Ag (g/t)
K-6049	19.0	43.0	24.0	162.0
including	19.0	20.0	1.0	807.0
including	32.0	33.0	1.0	1,420.0
K-6051	47.5	51.8	4.1	82.3
including	51.6	51.8	0.2	175.0
K-6053	16.0	18.0	2.0	486.0
including	16.1	17.1	1.0	683.0
K-6053	37.0	41.0	4.0	136.0
including	37.0	38.0	1.0	225.0

9.3 2013 EXPLORATION PROGRAMME

During the summer of 2013, exploration shifted focused on Mangazeisky North and South, where 52 trenches were excavated at regular intervals normal to the mineralised structure. A total of 474 m of sampled material was collected. Figure 9.1 illustrates the location of the 2013 trenches in relation to the mineralised veins at Mangazeisky North.

Figure 9.1 Plan Illustrating 2013 Trench Locations on Mangazeisky North Zone



Source: Silver Bear, Tetra Tech

Table 9.3 presents a summary of significant trench intersections on the Mangazeisky North deposit.

Table 9.3 Summary of Intersections from 2013 Trenches on Mangazeisky North; with a Silver Grade of >75 g/t Ag

Trench ID	Sample Profile No.	Sample Length Across Strike (m)	Ag Grade (g/t)	Mangazeisky North Target Vein
k13-01	P1	1.9	1,705.0	Mikhailovsky
k13-01	P2	2.1	1,590.0	Mikhailovsky
k13-01	P3	2.0	1,700.0	Mikhailovsky
k13-01	P4	1.6	1,780.0	Mikhailovsky
k13-01	P5	2.0	2,032.0	Mikhailovsky
k13-01	P6	0.5	2,123.0	Mikhailovsky
k13-03	p1	1.0	129.0	Mikhailovsky
k13-03	p10	1.0	264.0	Mikhailovsky
k13-03	p7	1.0	302.0	Mikhailovsky
k13-03	p8	1.0	102.0	Mikhailovsky
k13-11	P1	1.0	267.0	Mikhailovsky
k13-11	P2	1.0	343.0	Mikhailovsky
k13-11	P3	1.0	99.6	Mikhailovsky
k13-13	P1	0.4	506.0	Mikhailovsky
k13-13	P2	0.8	157.0	Mikhailovsky
k13-22	P1	1.7	2,280.0	Mikhailovsky
k13-22	P2	1.6	2,575.0	Mikhailovsky
k13-22	P3	1.6	2,017.0	Mikhailovsky
k13-22	P4	1.2	2,090.0	Mikhailovsky
k13-22	P5	1.0	874.0	Mikhailovsky
k13-22	P6	1.2	1,262.0	Mikhailovsky
k13-22	P7	1.1	932.0	Mikhailovsky
k13-23	P3	1.0	90.5	Mikhailovsky
k13-24	P1	0.7	2,774.0	Mikhailovsky
k13-24	P2	0.8	191.0	Mikhailovsky
k13-04	P1	1.0	221.0	Vasilievsky
k13-04	P2	1.0	79.6	Vasilievsky
k13-04	P3	1.0	334.0	Vasilievsky
k13-04	P4	1.0	308.0	Vasilievsky
k13-05	P1	1.0	484.0	Zhila-2
k13-05	P2	1.0	186.0	Zhila-2
k13-05	P3	1.0	1,224.0	Zhila-2
k13-05	P4	1.0	1,232.0	Zhila-2
k13-05	P5	1.0	370.0	Zhila-2
k13-12	P1	0.6	90.0	Zhila-2
k13-12	P2	0.4	1,364.0	Zhila-2
k13-14	P1	1.5	1,504.0	Zhila-2
k13-14	P2	0.3	2,543.0	Zhila-2
k13-14	P3	0.9	1,730.0	Zhila-2
k13-14	P4	1.4	1,026.0	Zhila-2
k13-14	P5	2.2	768.0	Zhila-2
k13-15	P1	0.5	2,166.0	Zhila-2

table continues...

Trench ID	Sample Profile No.	Sample Length Across Strike (m)	Ag Grade (g/t)	Mangazeisky North Target Vein
k13-16	P1	0.25	218.0	Zhila-2
k13-16	P2	1.2	2,665.0	Zhila-2
k13-16	P3	0.25	80.6	Zhila-2
k13-17	P1	1.0	375.0	Zhila-2
k13-17	P2	0.5	2,144.0	Zhila-2
k13-18	P1	0.3	1,964.0	Zhila-2
k13-18	P2	0.8	105.0	Zhila-2
k13-18	P3	0.3	1804.0	Zhila-2

9.4 2014 EXPLORATION PROGRAMME

In 2014, a trenching programme was carried out on the Vertikalny, Mangazeisky South, Porfirovy and Sterzhnevoy exploration targets (Table 9.4).

Table 9.4 Summary of Significant Intersections from 2014 Trenching Campaigns; Silver Grade of >100 g/t or Gold Grade >0.5 g/t

Trenches	From (m)	To (m)	Intersection Length (m)	Ag (g/t)	Au (g/t)	Deposit
MNS14-02	0.9	0.98	0.08	3,150	-	Mangazeisky North
MNS14-03	0.9	0.98	0.08	3,066	-	Mangazeisky North
KV14-03/1	8.7	11.1	2.4	912	-	Vertikalny
including	10.1	11.1	1	912	-	Vertikalny
KV14-05	10.1	11.1	1	853	-	Vertikalny
KS14-01	68.2	75	6.8	1,589	-	Sterzhnevoy
including	68.2	68.5	0.3	4,916	-	Sterzhnevoy
KS14-02	78.1	101.2	23.1	3,234	-	Sterzhnevoy
including	78.1	78.6	0.5	7,019	-	Sterzhnevoy
and	99.9	100.2	0.3	13,938	-	Sterzhnevoy
KS14-02	83.2	86.7	3.5	3,273	-	Sterzhnevoy
including	83.2	84	0.8	9,688	-	Sterzhnevoy
KS14-02	93.7	94.3	0.6	1,005	-	Sterzhnevoy
KS14-03	49.5	51.5	2	850	-	Sterzhnevoy
including	49.5	50.5	1	1,439	-	Sterzhnevoy
KS14-03	56.1	58.1	2	2,373	-	Sterzhnevoy
including	56.1	57.1	1	4,507	-	Sterzhnevoy
KS14-04	49.5	61.9	12.4	4,584	-	Sterzhnevoy
including	49.5	51.7	2.2	9,196	-	Sterzhnevoy
and	61.7	61.9	0.2	4,129	-	Sterzhnevoy
TV14-04 ПР II	2.2	4.9	2.7	602	-	Vertikalny
including	4.0	4.9	0.9	1,132	-	Vertikalny
TV14-04 ПР IV	2.5	3.2	0.7	269	-	Vertikalny
TV14-04 ПР VI	1.9	2.7	0.8	1,020	-	Vertikalny
TV14-06 ПР VII	4.1	4.4	0.3	1,203	-	Vertikalny

table continues...

Trenches	From (m)	To (m)	Intersection Length (m)	Ag (g/t)	Au (g/t)	Deposit
TV14-04 пр 2	4.0	4.9	0.9	1,141	-	Vertikalny
KP14-08	0.0	6	6	-	0.71	Porphyry
including	0.0	1	1	-	2.44	Porphyry
KP14-08	17.0	19	2	-	0.52	Porphyry
including	17.0	18	1	-	0.86	Porphyry
KP14-08	34.0	35	1	-	0.79	Porphyry

9.5 2015 EXPLORATION PROGRAMME

The 2015 exploration campaigns focused on trenching at Sterzhnevoy and Porphirivoy. In total, six trenches were excavated at Sterzhnevoy, with a total length of 395 m. At Porphirivoy, three trenches were excavated, with a total length of 203 m.

One notable intercept was noted at Sterzhnevoy, with a grade of 283 g/t silver over a 1 m intercept in trench number SK15-03.

At Porphirivoy notable samples included 4.59 g/t gold over a 1 m intercept in trench number PR15-01 and 2.17 g/t gold over a 1 m intercept in trench number PR15-06.

9.6 TRENCH SAMPLING PROCEDURES

The current sampling methods and approaches are summarised as follows:

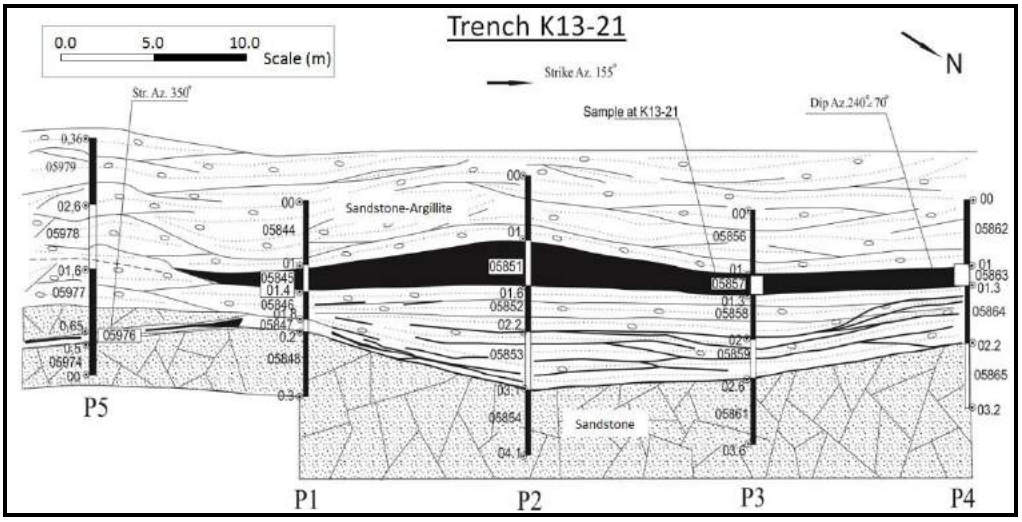
- A main trench was excavated over and along the strike of the mineralised zones to a depth of up to 4 m to expose the mineralised zones. The width of the trenches was 1 to 4 m depending on the thickness of the overburden.
- Where possible sampling and drilling is conducted on a grid of parallel lines (profiles) oriented perpendicular to the strike of the main mineralised zones. All trenches, road cuttings and other exposures were mapped in detail and sampled as required, Figure 9.2.
- Trench channel sampling was conducted perpendicular to the strike of the mineralised zones, the cross-section of which was 3 cm by 10 cm. The length of the samples varies from 0.1 to 6.0 m and averages 1 m. The entire trench exposure was sampled. Sample weights vary from 5 to 10 kg and average 7.5 kg.

All initial trench sampling was conducted on a grid of parallel profiles oriented perpendicular to the strike of the main mineralised zones. The sampled profiles were initially established at widely spaced intervals, which have been in-filled as exploration continues.

The initial trenches were dug along the strike of the mineralised zones to a depth of up to 4 m to expose the mineralised zones. The width of the trenches was generally

between 4 m and 5 m to allow full intersections to be taken as profiles orientated perpendicular to the strike of the mineralisation (Figure 9.2 and Figure 9.3).

Figure 9.2 Example of Trench Mapping and Sample Profiling from K13-21



Source: Silver Bear
Note: Az – azimuth

Samples were bagged, tagged and given a unique reference number, before being returned to camp on a daily basis, where the material is dried, crushed, split and assayed using the methods outlined in Section 11.0. As with the drill core samples, standard and blank samples are submitted every 20 samples, however, no field duplicates are collected.

Figure 9.3 Sampling Profiles within Trench K13-21

Source: Silver Bear

9.1 TETRA TECH OPINION

Tetra Tech is of the opinion that the exploration has generally been conducted using methods and procedures that are consistent with recognised industry best practices. However, the trench sampling has not been included in the grade interpolation due to statistically significant bias that is noted when comparing the trenches with nearby drillholes.

The bias may be due to natural mechanical enrichment at surface, or may have been introduced as a result of the manual channelling methods used to sample the trenches. The trenching also lacked QA/QC samples, which prevented full data verification to be undertaken.

Tetra Tech recommends further work to resample the trenches using a rock saw to cut a sample, where appropriate and a full QA/QC programme.

10.0 DRILLING

Details about the drilling campaigns completed prior to 2010 have been disclosed fully in the Wardrop (2011). The 2011 to 2012 drilling campaigns have been disclosed fully in Tetra Tech (2013). The details of the drilling at Mangazeisky North completed during 2015 are disclosed in the 2016 Technical Report (Tetra Tech 2016). All of these reports are filed under Silver Bear's profile on the CSAs SEDAR. Table 10.1 provides a summary of the drilling completed between 2004 and 2015. The 2015 drilling campaign at Vertikalny Central is described fully in Section 10.0.

Table 10.1 Summary of Drilling on the Property

Year	Metres Drilled	Targets Drilled
2004	151	Vertikalny
2005	3,370	Vasilievsky, Mikhailovsky, Nizhny Endybal, Strezhevoy
2006	732	Vertikalny
2007	3,100	Vertikalny
2008	11,663	Vertikalny, Semenovsky, Vasilievsky
2009	12,373	Vertikalny
2010	-	No drilling undertaken
2011 to 2012	13,445	Vertikalny, Nizhny Endybal
2013 to 2014	9,153	Mangazeisky, Vertikalny
2015	4,427	Vertikalny North West, Mangazeisky North, Porphivory, Strezhevoy

The most recent drilling campaign at Vertikalny main deposit was completed in 2015 and is detailed below. The most recent drilling campaign at the Mangazeisky North deposit was completed during the 2016 season and is detailed in Section 10.2.

10.1 DRILLING PROGRAMME 2015

10.1.1 VERTIKALNY DEPOSITS

Silver Bear re-commenced drilling during the 2015 field season. During this period, diamond drilling added a further six intersections of the Vertikalny Central vein and a further eight intersections through the Vertikalny Northwest mineralised structures.

The 2015 in-fill drilling programme was designed with the intention of increasing knowledge of the short-range variability of the mineralisation and to build confidence in the grade continuity. The drilling was focused on mineralisation at depth, within the Vertikalny Central Zone and Northwest Zone.

The 2015 drilling is inclined at between 55 and 70°, Vertikalny Central drilling is orientated towards the southwest (generally with a strike of 210 to 230°). The

Vertikalny Northwest drilling is orientated towards the northeast (with a strike of 40°) As a consequence of the drilling angle and the steeply dipping mineralisation, the true thickness of the mineralisation is between 40 and 50% of the drilled intersection lengths (see Table 10.2).

In order to provide representative mineralised samples of sufficient quantity for metallurgical testing, drillholes V13-35MET to V13-45MET were drilled at 80°, approaching the dip of the mineralized zone, which resulted in a greater intersection of mineralization. For these drillholes, the true mineralisation thickness is approximately 17% of the reported intersection length in Table 10.2.

Table 10.2 provides summary information relating to the 2015 drilling campaign.

Table 10.2 Summary of 2015 Drillholes

Drillhole ID	Length Drilled (m)	Easting	Northing	Elevation (m)	Azimuth (°)	Dip (°)	Mineralised Intercept Length (m)	True Mineralisation Thickness (m)
VNW15-01	110.00	549210.16	7285405.14	1,233	40	-60.00	1.0; 1.0	0.5; 0.5
VNW15-02	105.00	549143.24	7285448.08	1,232	40	-60.00	1.25	0.625
VNW15-03	185.00	549108.08	7285444.11	1,233	40	-60.00	-	-
VNW15-04	188.00	549049.75	7285454.75	1,237	40	-60.00	-	-
VNW15-05	184.00	549024.01	7285484.53	1,236	40	-60.00	-	-
VNW15-06	313.00	548903.45	7285539.02	1,235	40	-60.00	-	-
VNW15-07	80.00	549251.19	7285377.23	1,233	40	-60.00	1.00	0.5
VNW15-08	204.00	549214.04	7285321.11	1,240	40	-60.00	1.40	0.7
V15-01	218.00	551218.10	7283974.47	1,116	210	-65.00	1.00	0.5
V15-02	257.00	551217.45	7283973.85	1,116	215	-70.00	1.3; 2.9	0.65; 1.45
V15-03	278.00	551268.89	7283959.53	1,081	220	-66.00	1.1; 1.0	0.55; 0.5
V15-04	221.00	551280.88	7283898.45	1,059	206	-70.00	1.1; 1.7	0.55; 0.85
V15-05	311.00	551320.88	7283945.43	1,048	230	-63.00	-	-
V15-06	230.00	551356.66	7283892.29	1,016	224	-55.00	1.80	0.9

Significant intersections are tabulated in Table 10.3. Intersections are considered significant where silver has a length weighted grade of more than 100 g/t over a length of more than 1.0 m.

Table 10.3 2013/2014 Drill Programme Significant Intersections; >1 m at >100 g/t Silver

Drillhole ID	From (m)	To (m)	Thickness (m)	Ag (g/t)
VNW15-01	81.5	82.5	1.00	385
VNW15-01	83.5	84.5	1.00	203
VNW15-02	98.2	99.45	1.25	187
VNW15-07	56.0	57.0	1.00	154
VNW15-08	197.2	198.7	1.50	1,816
V15-01	208.5	209.5	1.00	210
V15-02	96.0	97.3	1.30	123
V15-02	244.5	247.4	2.90	596
V15-03	266.0	267.1	1.10	137
V15-03	269.5	270.5	1.00	110
V15-04	211.7	212.8	1.10	176
V15-04	213.1	214.8	1.70	344
V15-06	217.6	219.4	1.80	153

The average and range of true thickness of mineralisation are outlined in Table 10.4.

Table 10.4 Mineralisation Thickness Observed in 2013 Drilling

Measure	True Mineralisation Thickness (m)
Mean	0.68
Maximum	1.45
Minimum	0.50

The average and range of mineralised widths observed in the 2015 drilling are consistent with those observed in previous drilling conducted in the same areas.

The approximate average intercept spacing in the upper parts of the Vertikalny Central Zone is now 30 m, reducing to circa 50 m at depth.

10.2 MANGAZEISKY NORTH DRILLING – 2016

Silver Bear commenced drilling during the 2016 field season to further explore the known structures at Mangazeisky North. During 2016, 11 short diamond drill holes were completed exploring for mineralised structures, an additional 29 holes were drilled to collect samples for metallurgical test work requirements, as well as 3

technical holes for collecting thermal and hydrogeological data. The metallurgical and technical holes were not included in the Mineral Resource estimation sample dataset. The additional Mineral Resource drilling increased the meters drilled for Mangazeisky North by 134 m to a total of 1,642 m.

The 2016 drilling is inclined at 60°, and is orientated towards the west with a dip direction of 240°. The drilling intercepts the mineralisation at a normal angle and therefore the apparent thickness of the sample intersections are broadly representative of true thickness.

Table 10.2 provides summary information relating to the 2016 drilling campaign.

Table 10.5 Summary of 2016 Drillholes

Drillhole ID	Length Drilled (m)	Easting	Northing	Elevation (m)	Azimuth (°)	Dip (°)
MN16-01	35.00	552331.12	7290629.43	1179.05	240	-60
MN16-02	25.00	552250.31	7290703.12	1194.93	240	-60
MN16-03	23.00	552277.60	7290661.35	1195.28	240	-60
MN16-04	33.30	552302.19	7290675.67	1190.78	240	-60
MN16-05	30.50	552301.88	7290616.60	1189.35	240	-60
MN16-06	32.00	552522.29	7290169.05	1157.95	240	-60
MN16-07	17.00	552494.80	7290155.93	1161.07	240	-60
MN16-08	36.00	552551.18	7290126.00	1143.70	240	-60
MN16-09	23.00	552534.07	7290110.19	1141.77	240	-60
MN16-10	30.00	552233.25	7290754.49	1191.95	240	-60
MN16-11	31.00	552203.53	7290794.32	1195.83	240	-60

Significant intersections are tabulated in Table 10.3. Intersections are considered significant where silver has a length weighted grade of more than 100 g/t over 1 m. As the mineralisation is very narrow, samples over 100 g/t silver are typically only 20 to 30 cm wide.

Table 10.6 2016 Drill Programme Significant Intersections at >100 g/t Silver

Drillhole ID	From (m)	To (m)	Thickness (m)	Ag (g/t)	Ag (g/t/m)
MN16-01	26.00	26.20	0.20	1190.00	238.0
MN16-03	14.80	15.50	0.70	176.00	123.2
MN16-03	16.20	16.50	0.30	2210.00	663
MN16-04	22.55	23.00	0.45	1230.00	553.5
MN16-06	12.30	12.60	0.30	364.00	109.2
MN16-06	26.55	26.85	0.30	1160.00	348
MN16-08	30.20	30.40	0.20	678.00	135.6
MN16-08	32.40	32.60	0.20	1030.00	206
MN16-09	13.30	13.50	0.20	893.00	178.6
MN16-09	18.50	18.70	0.20	927.00	185.4

The average and range of mineralised widths observed in the 2016 drilling are less than the trench intersections, but consistent with other similar mineralisation on the Property. It is probable that surface mechanical processes are making the mineralisation appear thicker at surface.

10.3 DRILLING METHODS

All of the 2015-16 drilling was completed using diamond coring techniques, with the majority completed using Boart Longyear rigs, and the remainder using a Russian SKB 5 rig.

Figure 10.1 and Figure 10.2 show typical drill setups. The Boart Longyear rigs collar at HQ (63.5 mm) diameter and reduce to NQ (47.6 mm) at around 40 m. The Russian drill rig collars with 112 mm diameter core reducing at a depth of 6 m to 93 mm and then to 76 mm where required. The Russian rig drills dry. The Boart Longyear rigs have an established water line and drilling mud is used to maintain core recovery. Core recovery is measured and recorded by Silver Bear geologists and was found to be excellent for both the Boart Longyear and the Russian SKB 5 rig.

Figure 10.1 Drilling at Vertikalny in 2011

Source: Silver Bear

Figure 10.2 Drilling at Vertikalny in November 2013

Source: Tetra Tech

For the drilling programmes at Vertikalny, the core containing mineralised intersections were split in half by core saw or trowel as appropriate and then sampled. The weight of the core samples range from approximately 2 to 4 kg.

10.4 DOWN HOLE DIRECTIONAL SURVEY METHODOLOGY

For the Boart Longyear drill rig, surveys were undertaken using a Reflex 'EZ-shot'™ single shot down hole survey tool that is operated by the driller. Measurements were taken every 25 m for the first 100 m and at 50 m intervals thereafter. Readings are recorded on paper slips that are delivered to the Project geologist on a daily basis. The azimuth is recorded as magnetic and this is corrected by the appropriate declination. The measurements are recorded in the drillhole database and the drill logs. Copies are kept by the contractor.

For the Russian drill rig, down hole surveys are conducted using an inclinometer at 20 m intervals.

10.5 DRILLHOLE COLLAR SURVEY METHODOLOGY

The Project geologist and drill foreman co-operate to prepare all drill sites prior to the drill moves. The surveyor is directly responsible to the Project geologist.

From 2011 onwards, the previous method of chain survey was replaced by the use of Precision Point Differential Global Positioning Systems (DGPS), with the establishment of control points and a survey network. All drillhole collars and trench

locations have been re-surveyed using the DGPS. Correlation with the previous survey was very good. All surveys are reported in Universal Transverse Mercator (UTM) Wideband Global SATCOM (WGS) 84, Northern Zone 52.

10.6 DRILL RIG SETUP

Silver Bear geologist are responsible for setting the rig for a new hole at the correct coordinates, with the correct azimuth and dip for all holes. The azimuth and dips are checked using the drillhole orientation tool or compass clinometer. The decision to commence and stop drilling should be made in agreement with the chief geologist.

The geologist should also ensure that the core trays are ready prior to drilling, with partitions nailed in at the correct spacing according to the core diameter (HQ = 63.5 mm, NQ = 47.6 mm).

10.7 DRILLING, CORE HANDLING AND CORE BOX MARK-UP

Silver Bear geologists supervise the drillers to ensure that they remove the core from the core barrel and onto the core racks or directly into the core tray with sufficient care, without loss of fragments, without disrupting the order, and noting carefully the shallowest and deepest ends of the core run.

Silver Bear ensures that the core tray is marked before starting to fill it with core. The correct markings include:

- the direction of increasing depth, from the shallowest depth to the deepest, is to be shown by arrows—from left to right
- the hole number, tray number, start and end depth (from and to) are to be marked on the shallowest end of the core tray
- end depth is marked after the core tray has been filled.

Figure 10.3 Example of Core Tray Markings

10.8 GEOTECHNCIAL LOGGING OF CORE SAMPLES

Geotechnical logging is currently completed in the core logging facility at the Hogen camp. Tetra Tech recommends that the geotechnical logging is completed at the rig side in future, to avoid artificial damage to the core being accounted for as natural fractures. The following information is recorded:

Core size: This refer to the type of core drilled (HQ, HQ2, NQ, NQ2, etc.).

Total Core Recovery (TCR): The total length of core recovered (including broken zones).

Solid Core Recovery (SCR): The total length of the solid core (excluding pieces of core smaller than the core diameter). However, if a single joint parallel to the core axis intersects the core piece, the pieces are counted as a solid core.

Rock Quality Designation (RQD): The total length of solid core cylinders of greater than 100 mm in length, expressed as a percentage.

$$\text{RQD} = \frac{\text{sum of solid core peices} > 100\text{mm in length}}{\text{core run length}} \text{ x } 100$$

Other information collected in the geotechnical log include:

- lithology
- weathering classification
- strength.

Figure 10.4 Core Logging

10.9 CORE PHOTOGRAPHY

Every box of core is photographed at the Hogen camp. Each photo is labelled with a card showing drillhole number. The photos also include a metric scale bar, and a standard colour chart.

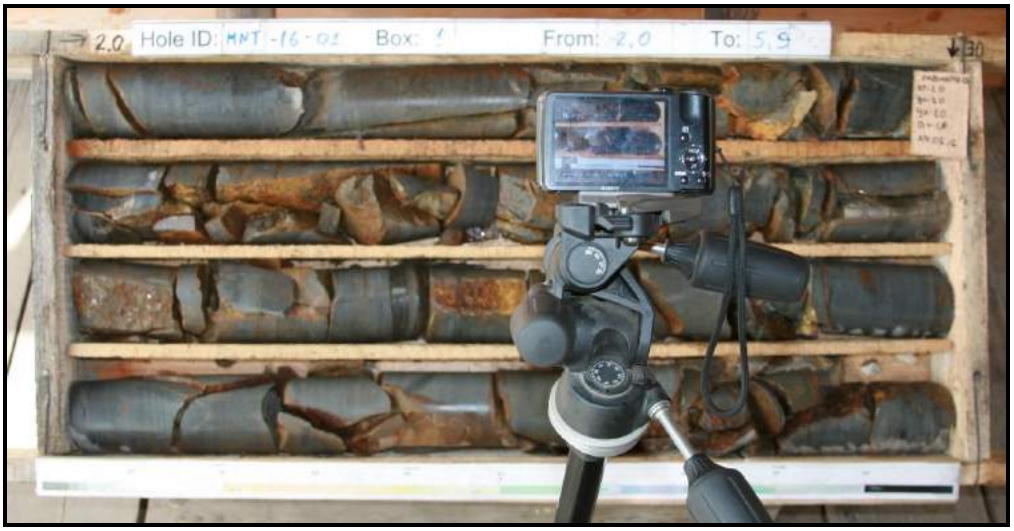
During low-light conditions, the core boxes are illuminated by two spot lights, arranged so that there are little-to-no visible shadows in the photograph and the light conditions are the same for each photo.

The camera is positioned in the same place each time using a tripod so that the angle of the photo is perpendicular to the core box. The core is photographed wet and dry.

Photos are stored in digital format (*.jpg), on the computer in the geology office, and backed up to servers and hard drives.

Figure 10.5 Core Photography

Figure 10.6 Example of How to Set up a Photograph of the Core Box



10.10 GEOLOGICAL LOGGING OF CORE SAMPLES

For the 2015 campaign, the core was logged into standard hard copy logging sheets and the transcribed into excel. Tetra Tech recommends that the logging is completed directly into excel or into proprietary logging software in future to minimise the risk of transcription errors.

In general, the minimum logging interval is 0.2 m, unless specifically required to capture a significant feature (e.g. narrow mineralised vein or structure).

Lithological and oxidation logging is completed using a standardised legend of codes.

Other information recorded includes the following:

- colour
- grain/crystal size
- oxidation state
- textures
- alteration minerals
- logged by (name of geologist).

Figure 10.7 Geological Logging of Core Samples

10.11 SAMPLING DISCUSSION

10.11.1 VERTIKALNY

Prior to sampling, a sampling plan created by the geologists and reviewed by the chief geologist.

At many of the target areas, large sections of the drill core will contain obviously barren sedimentary rocks; therefore, sampling will be restricted to mineralised zones, with a number of waste samples taken either side of the mineralisation to ensure that full intercepts are sampled and that contacts are defined in the subsequent assay data.

The zones for sampling are selected based upon the lithological logging, with reference to the following important observations:

- visible alteration
- sulphide minerals
- breccia textures
- Intrusive (dyke) intervals
- faulted or disrupted zones.

Standard sample lengths are 1 m, but are reduced to a minimum of 0.1 m to ensure that samples do not cross lithological boundaries.

A minimum of two, 1 m samples of visually non-mineralised material are taken above and below the mineralised intercepts. If the angle of intercept of the mineralised zone is acute (less than 45° to the core axis) then additional 'barren' samples are selected.

Visually un-mineralised inter-burden between mineralised samples should be continuously sampled in lengths of up to 1 m, if the interval between mineralised zones is less than 5 m.

On the porphivory deposit, the mineralised material is more difficult to identify visually; therefore, continuous sampling is undertaken from the start to end of the drillhole. As with other deposits, the standard sample length is 1 m, but is reduced to a minimum of 0.1 m to honour lithological contacts.

Sampling is recorded on the sampling register. Sample numbers are continuous and consistent with the sample tags. Sample tags are stapled to the core box in the relevant location, with the duplicate sample tag retained. The sample register lists the sample numbers, the intended analytical techniques and mass of sample(s). It also shows which samples should have a duplicate sample weighed and bagged, and which sample numbers to be assigned to blanks and standards.

Samples are indicated by marking the end of each sample with a black marker on both the core and on the core box divider walls. The intervals are also recorded on the sample tags that are stapled to the core box.

Samples marked for cutting are marked with a ruled line marking the cutting line. This cutting line bisects the symmetry present in the mineralisation (e.g. dip of structures or ore zone boundaries) or if this is absent, is drawn along the axis of bedding symmetry of the surrounding host rocks.

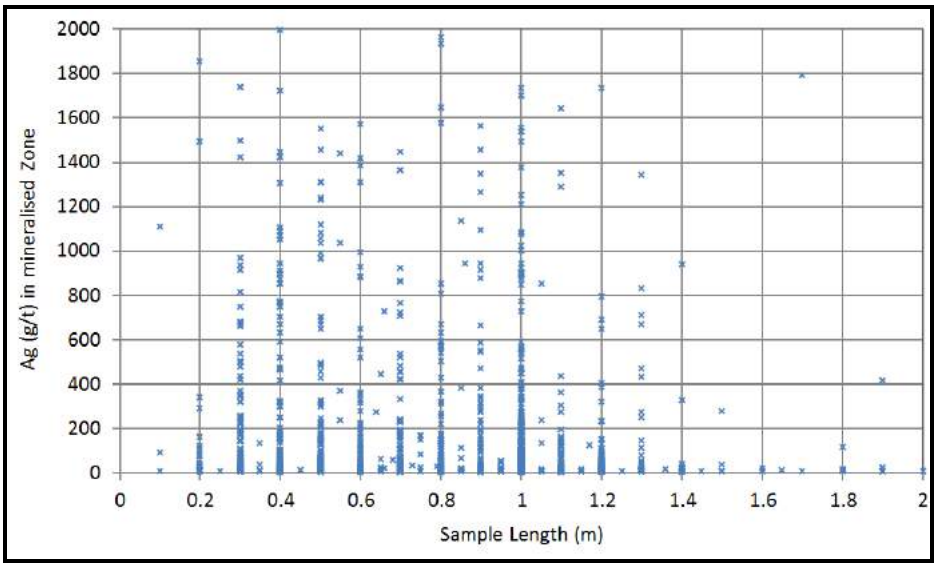
Samples are generally cut with a diamond saw. Where the core is too fragile or broken for cutting, no line is drawn and half the width of the channel holding the core in the core tray is to be taken with a spatula.

Following cutting, the core box then is to be taken to the sample preparation room where the core samples are taken, placed in bags, dried and weighed. The geologist manages the sample preparation team to ensure that sampling is undertaken in accordance with the sample register.

Figure 10.8 illustrates that there is no correlation between sample length and grade.

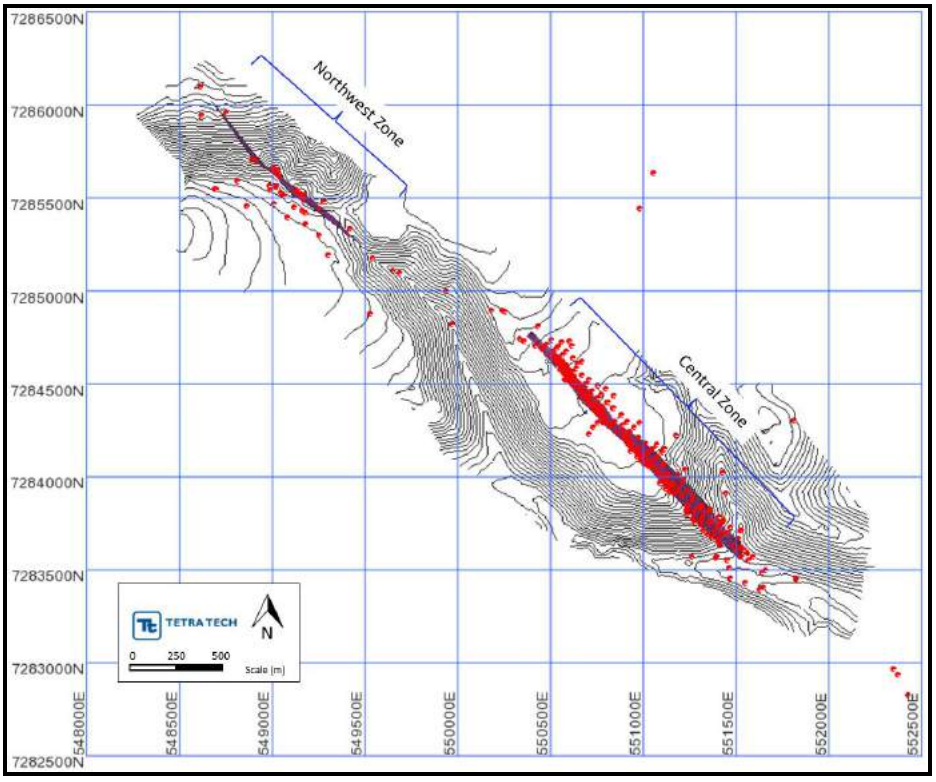
The Vertikalny (vein) structure can be identified during logging, therefore any intersection of the structure may contain a variety of lower and higher grade intervals. During the 2011 and 2012 campaigns, sampling was undertaken according to a nominal 1 m standard sample length with non-mineralised material sampled at either end of the mineralised intersections. From 2013 onwards, the sampling was based upon logging intervals, however, systematic 1 m samples of non-mineralised material was sampled at either end of the mineralised intersections as in previous campaigns.

Figure 10.8 Silver Grade vs. Sampled Length



Source: Tetra Tech

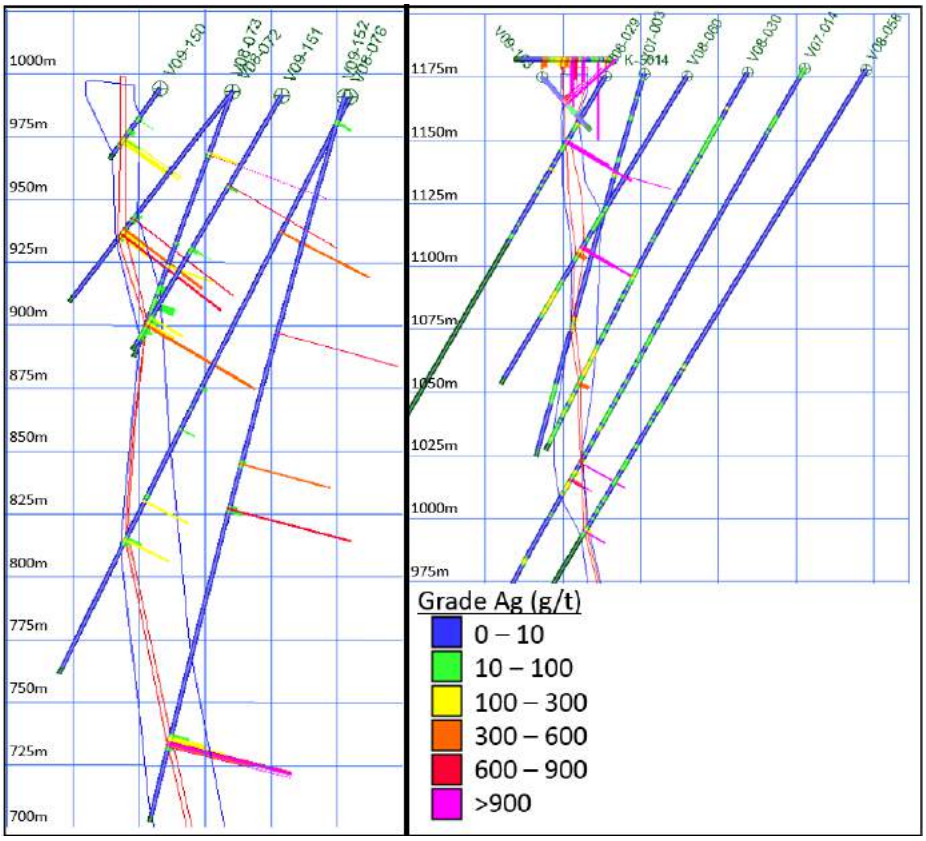
Figure 10.9 Drillhole Collar Locations and Mineralisation



Source: Tetra Tech

Figure 10.10 presents a representative drill section through the Central Zone of the Vertikalny deposit.

Figure 10.10 Section through the Central Zone of Vertikalny



Source: Tetra Tech

The Vertikalny mineralised zone is dominated by a series of brecciation events and probable subsequent fault movement within the host rocks. As a result, the presentation of the mineralised zone varies from solid cylinders of core through to highly broken, soft gouge type material.

The core recovery is generally very good, averaging approximately 95%. Decreased recovery has occurred in a small number of mineralised intervals. Drilling muds and polymers were used to enhance recovery. There is no correlation between recovery and silver grade.

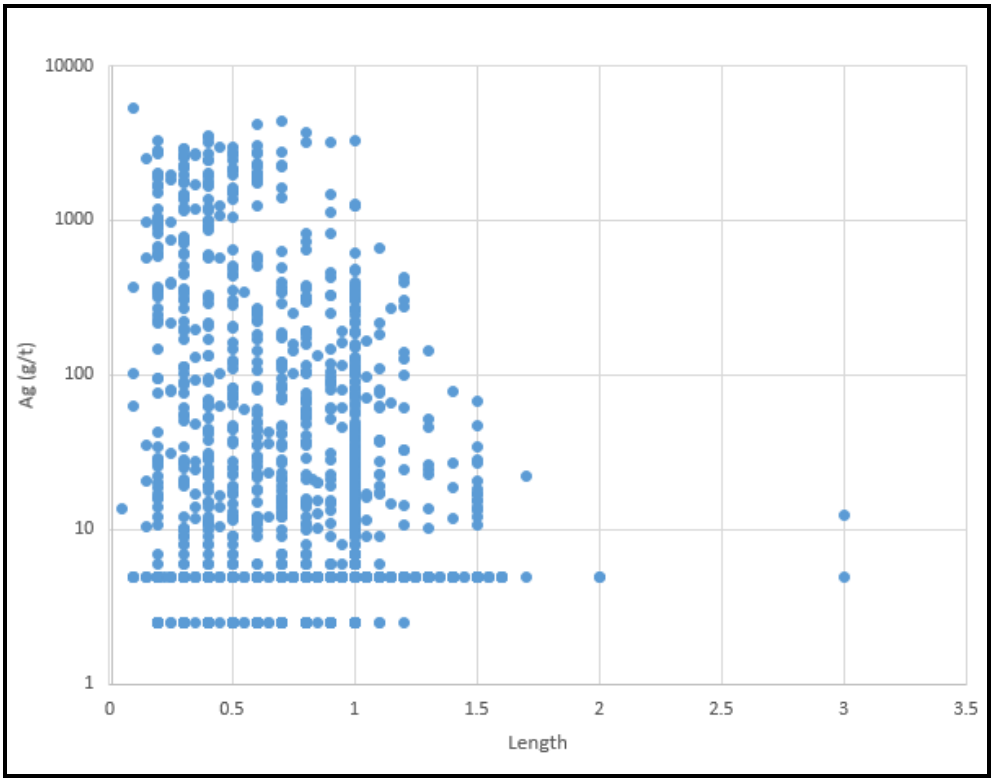
10.11.2 Sampling Discussion Mangazeisky North

Sampling for Mangazeisky North follows the same procedures as Vertikalny as described in section 10.11.1

The vein structure can be identified during logging, therefore any intersection of the structure may contain a variety of lower- and higher-grade intervals. Additionally, some lithologies logged as non-mineralised contain significant grade, such is the style of mineralisation. During the 2011 and 2012 campaigns, sampling was undertaken according to a nominal 1 m standard sample length with non-mineralised material sampled at either end of the mineralised intersections. From 2013 onwards, the

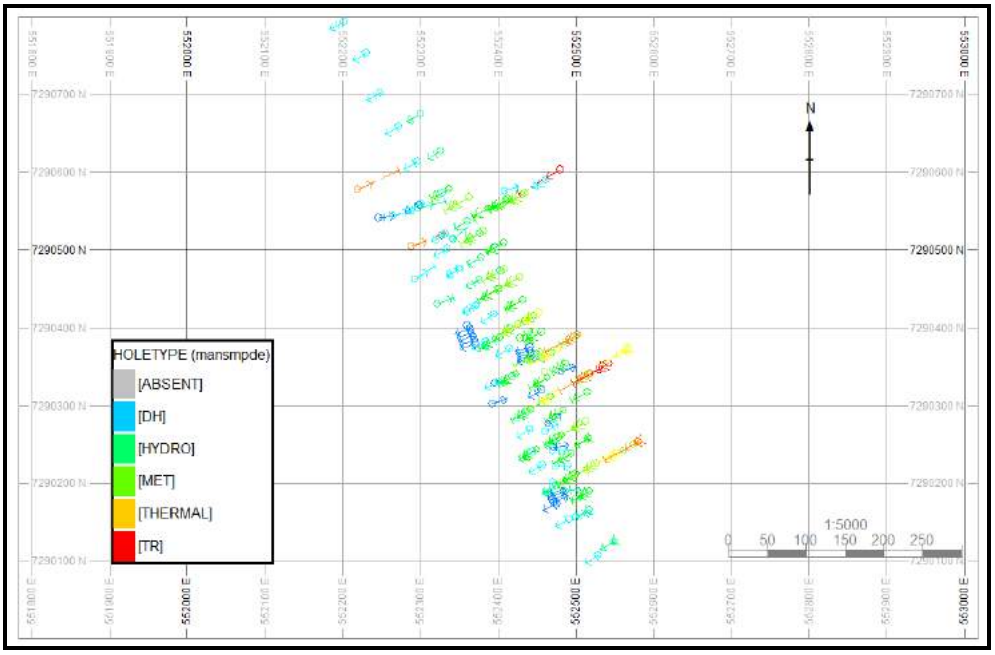
sampling was based upon logging intervals, however, systematic 1 m samples of non-mineralised material was sampled at either end of the mineralised intersections as in previous campaigns. Figure 10.8 illustrates that there is no correlation between sample length and grade.

Figure 10.11 Silver Grade vs. Sampled Length –Mangazeisky North



Source: Tetra Tech

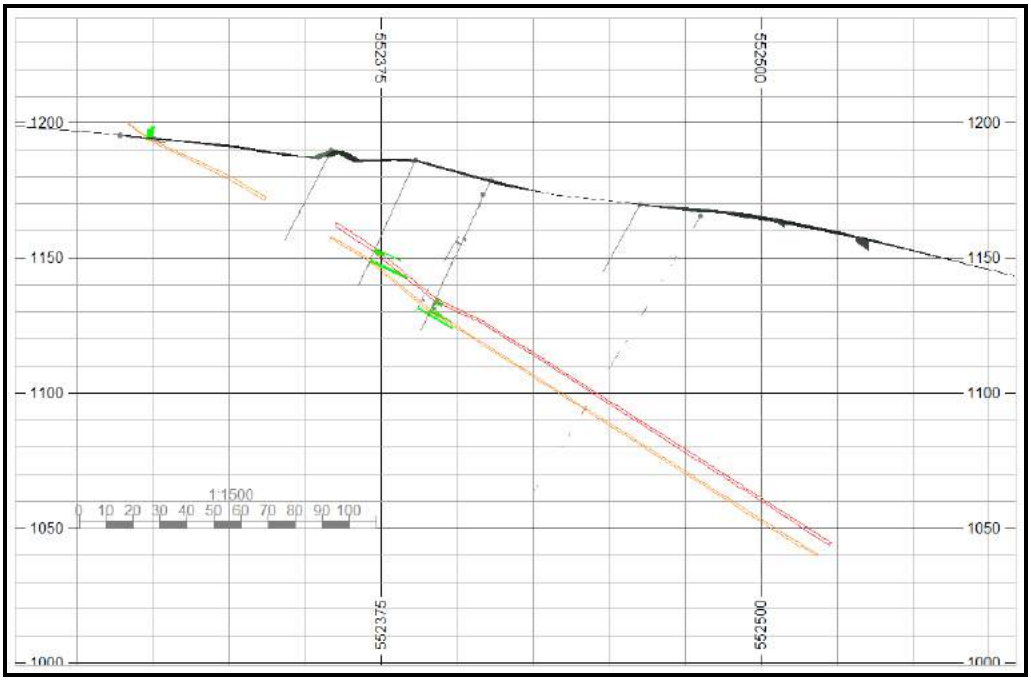
Figure 10.12 Drillhole Traces - Mangazeisky North



Source: Tetra Tech

Figure 10.10 presents a representative drill section through the Mangazeisky North deposit.

Figure 10.13 West-east section through Mangazeisky North – 7290571N



Source: Tetra Tech

The core recovery is generally very good, averaging approximately 95%. Decreased recovery has occurred in a small number of mineralised intervals. Drilling muds and polymers were used to enhance recovery. There is no correlation between recovery and silver grade.

10.12 TETRA TECH OPINION

Some inconsistencies were noted within the drillhole database, relating to the survey of the technical, non-resource, holes at Mangazeisky North. This relates to how drill pads are reused, but have to be re-cut and levelled to provide a suitable area for the rig. This has resulted in some of the metallurgical holes, which were twinned with Mineral Resource holes, to have inconsistent depths of mineralised intersections. As these holes were not drilled for Mineral Resource estimation purposes, and that the mineralisation was not necessarily sampled and assayed, these holes have not been included in the Resource estimation database. In Tetra Tech's opinion, the inconsistencies do not represent a material risk to the robustness of the Mineral Resource estimate.

Tetra Tech is of the opinion that the drilling has generally been completed using methods and procedures that are consistent with recognised industry practices and that the data is adequate for Mineral Resource estimation.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section summarises the sampling and sample preparation methodology adopted by Silver Bear during their various exploration programmes since 2007. Prior to 2007, the sample preparation, analyses, and security was conducted according to Russian State Reserve Committee (GKZ) standards.

The general approaches and methodologies are consistent for all of the targets across the Property, with the exception of specific components which are described separately for each area in Sections 11.1.4 and 11.1.5.

Training of Russian geologists and support staff in sampling procedures is carried out by Silver Bear.

11.1 SAMPLE PREPARATION AND ANALYSES

Upon completion of logging and sampling, paperwork outlining the regime for analysis was prepared from the geological log. Silver Bear operates its own sample preparation laboratory where samples are crushed and milled (Figure 11.1).

Figure 11.1 Crushing Equipment in Silver Bear's On-site Sample Preparation Facilities

Source: Tetra Tech

Samples are crushed and milled to greater than 85% passing 1 mm, before quartering, to produce a 1 kg sample. Final milling is carried out at the assaying laboratory ALS Chemex in Chita, Russia, where the material is reduced to 85%

passing 75 µm. This sizing is based on industry standard sizing for homogeneity in the preparation of the sample aliquot.

Sample trays are vacuum cleaned post jaw crushing. The jaw crusher operator clears all sample material from the crusher and brushes off the crusher apron and top of the jaws. After the jaw crushing, the sample tray is thoroughly brushed out. No crushing of samples was conducted during the site visit; therefore, no vacuuming of this tray was observed. Blank material is run through the crushers after every 10 samples, or after every sample if designated for fire assay in order to wash through the equipment to help prevent cross contamination of sample. Currently, there is no compressed air available to blow out dust from the crushers. It is strongly recommended that an air compressor be introduced to thoroughly clean all sample preparation equipment.

The crushed sample is systematically mixed on a metal-surfaced table using flat blades, and then manually quartered using a steel quadrant device (Figure 11.2).

Figure 11.2 Sample Splitting in Silver Bear's On-site Sample Preparation Facilities

Source: Tetra Tech

The operator then takes an approximately equal scoop from each quarter to fill the required sample bags. The sample bags are weighed by the operator to ensure that a sample of more than 100 g of sample is taken, greater than the 50 g typically required for assay plus spare pulp. The remaining pulp sample is retained if the sample is designated for fire assay so that in addition to the duplicates additional assay can be run from the homogenised sample if required. Coarse reject material is also retained as shown in Figure 11.3. The table is thoroughly brushed off between samples. Compressed air is also recommended to clean the table for this stage of preparation.

Figure 11.3 Coarse Reject Store Shed

Source: Tetra Tech

The samples are sealed and accurately weighed, and the weight is recorded on the sampling list (Figure 11.4). Samples for analysis are boxed and sealed, usually in batches by drillhole, for despatch to the laboratory concerned.

Figure 11.4 Weighing Scales in Silver Bear's On-site Sample Preparation Facilities

Source: Tetra Tech

Boxed samples are checked against the sample list and a despatch note. The exact sequence of sample numbers is listed as to the order in which the samples are to be processed at the laboratory. This includes how the samples are to be processed, even if the numbers are not sequential. Chain of custody documentation and a copy of the instructions is sent to the laboratory.

11.1.1 LABORATORIES

Prior to 2011, analysis was carried out at Russian certified Chemical Laboratory of the State Enterprise Aldangeologiya (Aldan Lab), located in Yakutia, Russia.

Analysis for 2012, 2013, 2014, and 2015 campaigns were completed by International Organization for Standardization (ISO)/International Electrotechnical Commission (IEC) 17025 accredited laboratory ALS Chemex in Chita, Russia.

Both laboratories are internationally accredited and wholly-independent of Silver Bear.

11.1.2 ASSAYING

Prior to 2011, the samples sent for fire assay were analysed in duplicate for silver. All samples were sent for fire assay. Samples with significant silver grades, determined from spectral analysis were also analysed for silver, copper, lead, and zinc using atomic absorption (AA). Samples sent for spectral analysis were analysed for 36 elements, including tin, lithium, titanium, cobalt, mercury, and vanadium.

From 2011 onwards, analyses were completed using a four acid sample digestion of 0.25 g, followed by inductively coupled plasma (ICP) finish and reporting of 33 elements (laboratory code ME-ICP62). Where values of silver, lead or zinc exceeded the respective upper detection limits, further four acid digestion analyses were carried out of 0.4 g, followed by ICP finish (laboratory code ME-OG62). Where values of silver exceeded the upper detection limit for ME-OG62 (1,500 g/t), a 50 g sample was taken for fire assay analyses with a gravimetric finish (laboratory code Ag-GRA22).

A selection of the samples was identified by the Prognoz geologists for gold assaying. This was undertaken via fire assaying with an AA finish using a 50 g sample (laboratory code Au-AA24).

Retained samples (duplicates and rejects) are boxed by drillhole number and stored in a locked container at the Endybal site. Analytical results are received electronically and updated in the drillhole database.

Assay certificates are submitted in electronic and hard copy form to the Project geologists.

11.1.3 QUALITY ASSURANCE/QUALITY CONTROL

Since 2007, Silver Bear has employed an industry standard quality assurance (QA)/quality control (QC) programme, which includes chain of custody protocols, as well as systematic submittals of standards, duplicates, and blank samples into the flow of samples sent for assaying. Prior to 2007, the sampling procedures were

consistent with the requirements of the Russian GKZ system. No sample results prior to 2007 have been used in estimating the current Mineral Resource for Verticalny, due to the lack of data validation and QA/QC.

A continuous sample numbering system has been adopted by Silver Bear, which does not differentiate control samples so that the QA/QC samples are sequential in the packing list in order that they not be recognised by the lab. Sample numbering is defined by the geologist for each hole as part of the sampling plan. ALS Chemex also reported its own standards, duplicates, and blanks as part of their own QA/QC protocol.

11.1.4 VERTIKALNY QA/QC

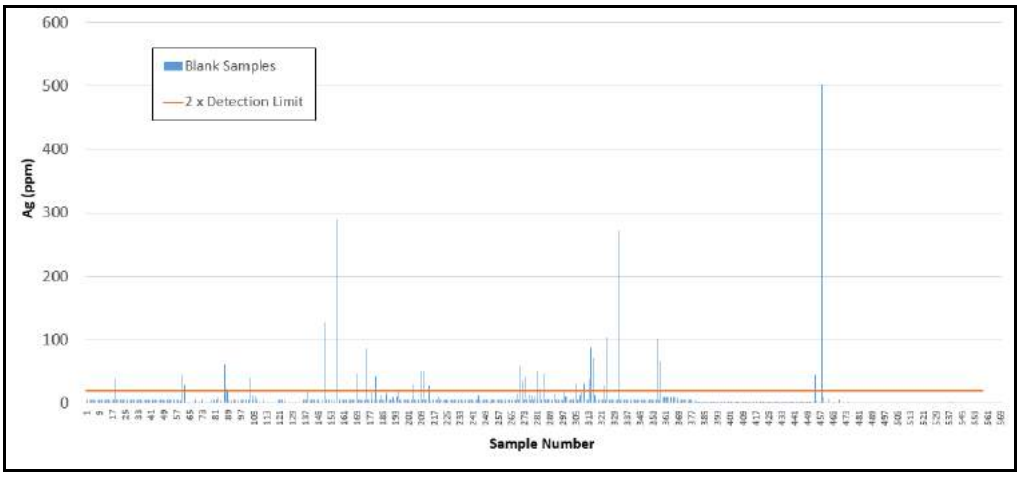
Since 2007, the Vertikalny exploration programmes have included the submission of 1,654 QA/QC samples, along with 13,892 regular samples. For the purposes of QA/QC and Resource Estimation, Spectral Analysis (SA) results have not been included due to their unreliability in this study.

VERTIKALNY BLANKS

Blank samples are created from local non-mineralised rock and processed in the on-site sample preparation laboratory. Blank material is stored on the ground outside of the sample preparation laboratory.

Blank material is broken up with a hammer and placed into a sample bag. Blank samples are then inserted in numerical order into the sequence for preparation. Since 2007, 558 blank samples have been analysed at Vertikalny (Figure 11.5). Of those 558 samples, 37 were above two times detection limit of which 10 are significant .The blank failure is not considered material to the Estimate as the majority of the recorded values are significantly lower than any economic mineralisation.

Figure 11.5 Blanks Chart for Vertikalny Exploration Data



Further analysis has shown that the performance of the blanks was poorer within higher-grade intersections (greater than 100 g/t silver), which could indicate that there was some contamination. It is noted, however, that the number of blanks submitted within the higher-grade intersections is low; therefore, statistical significance cannot be established.

Further analysis was carried out to determine whether blank sample failure was due to contamination, or because of substandard original blank material. The analysis was completed by comparing the poorly performing blanks against the three samples that preceded them through the sample preparation process. No obvious indication was found that could link the poorly performing blanks to the samples that were processed immediately before. Therefore, it was concluded that the blank material probably contained minor mineralisation, and it is recommended that the material used for blank samples is re-examined or replaced by material known not to carry any target mineralisation. Commercially-sourced certified blank material would be recommended to eliminate any possibility of minor mineralisation. It is also recommended that the sample preparation apparatus is flushed with a quartz sand wash between samples.

Blank samples were also independently assessed for each campaign, and were found to be well within acceptable limits.

VERTIKALNY STANDARDS

Since 2007, a total of 563 standard reference samples have been submitted for analysis as part of the Vertikalny exploration. The standard reference materials (SRMs) were sourced from Geostats Pty Ltd, who are based in Perth.

The SRMs used and the number of times they were submitted are outlined in Table 11.1, with all SRM plots shown below.

Table 11.1 SRMs used in the Vertikalny Exploration

Standard	No. of Submissions	Ag Value (g/t)	Standard Deviation Ag	Number of Failures*	Pass Rate (%)
GBM 906-6	194	389.7	21.1	6	96.91
GBM 303-1	11	1419.6	73.5	3	72.73
GBM 909-11	25	25.5	1.7	4	84
GBM 909-13	25	127.3	6.8	4	84
GBM 310-16	43	314.3	14.9	3	93.02
GBM 997-4	116	287.9	38.2	4	96.55
GBM 998-9	133	101.2	4.8	10	92.48
GBM 913-13	16	74.1	3.9	0	100
Total	**563**	-	-	**35**	-

Note: *failure defined as assay result falling outside of two standard deviations.

Good standard performance is reported for the following standards:

- GBM997-4
- GBM906-6
- GBM310-16
- GBM998-9.

No failures were reported for the following standards:

- GBM 913-13.

Figure 11.6 GBM913-13 Chart for Vertikalny Exploration Data

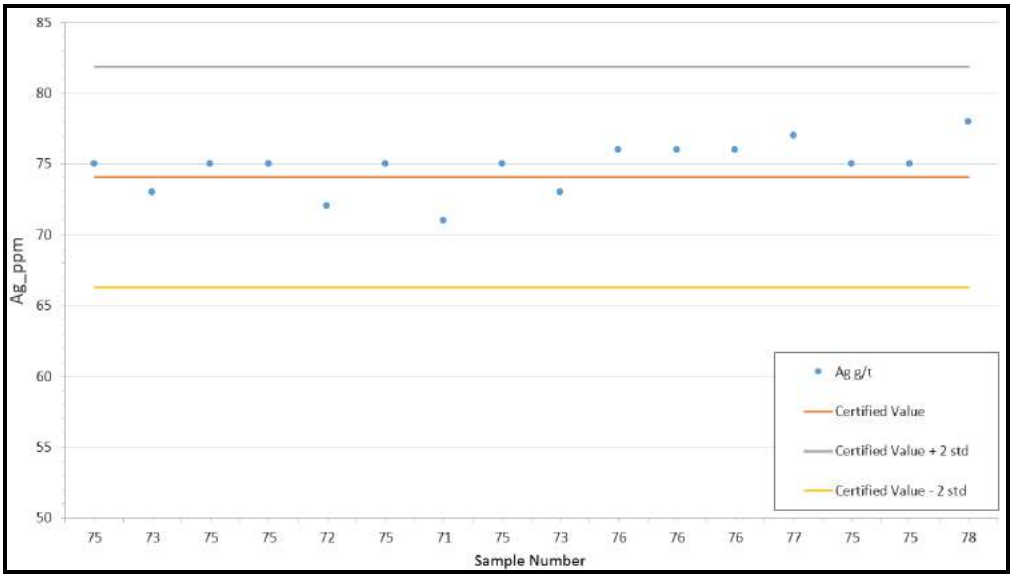


Figure 11.7 GBM997-4 Chart for Vertikalny Exploration Data

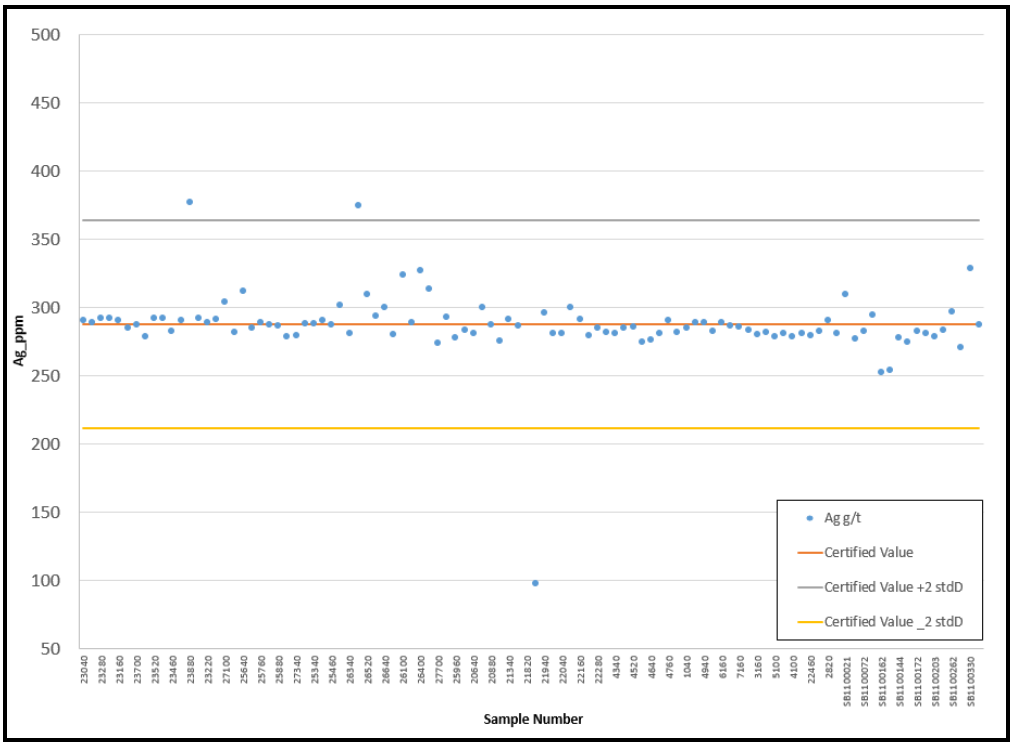


Figure 11.8 GBM906-6 Chart for Vertikalny Exploration Data

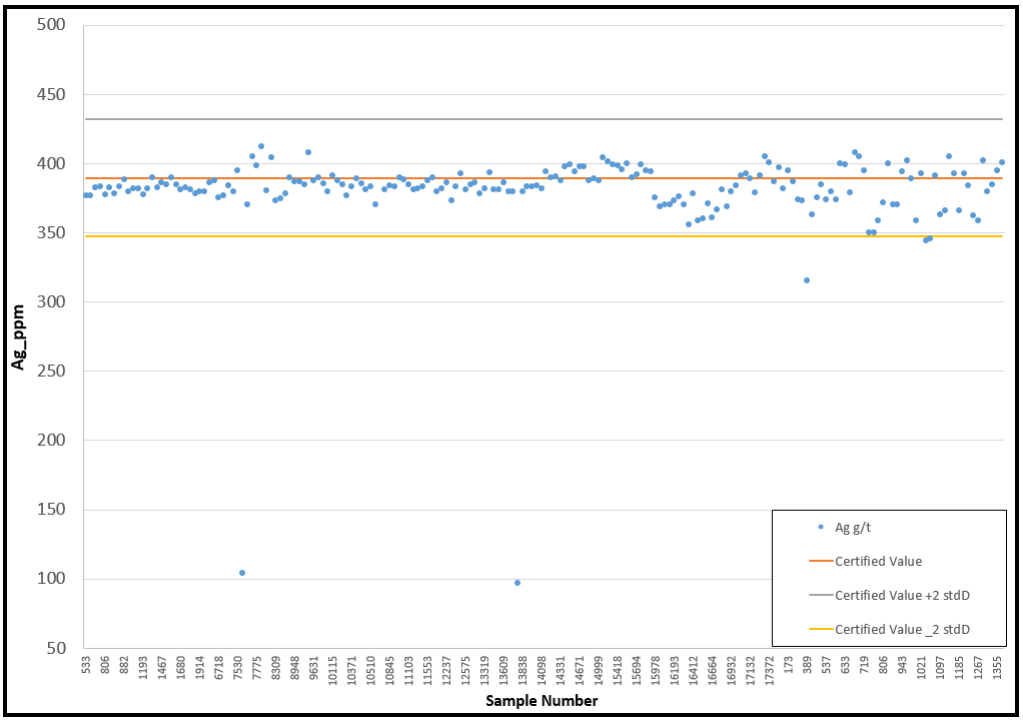
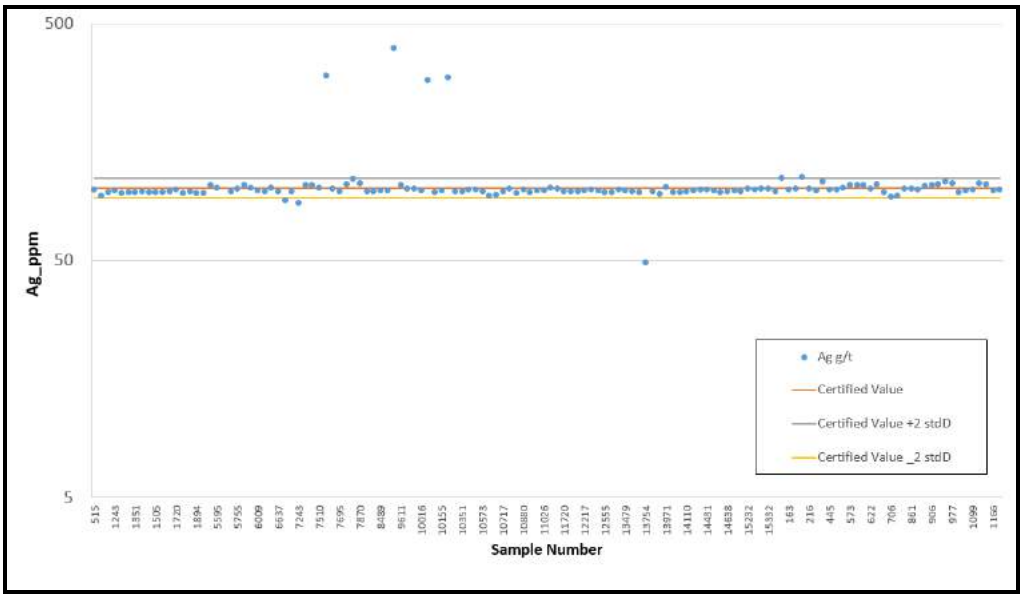
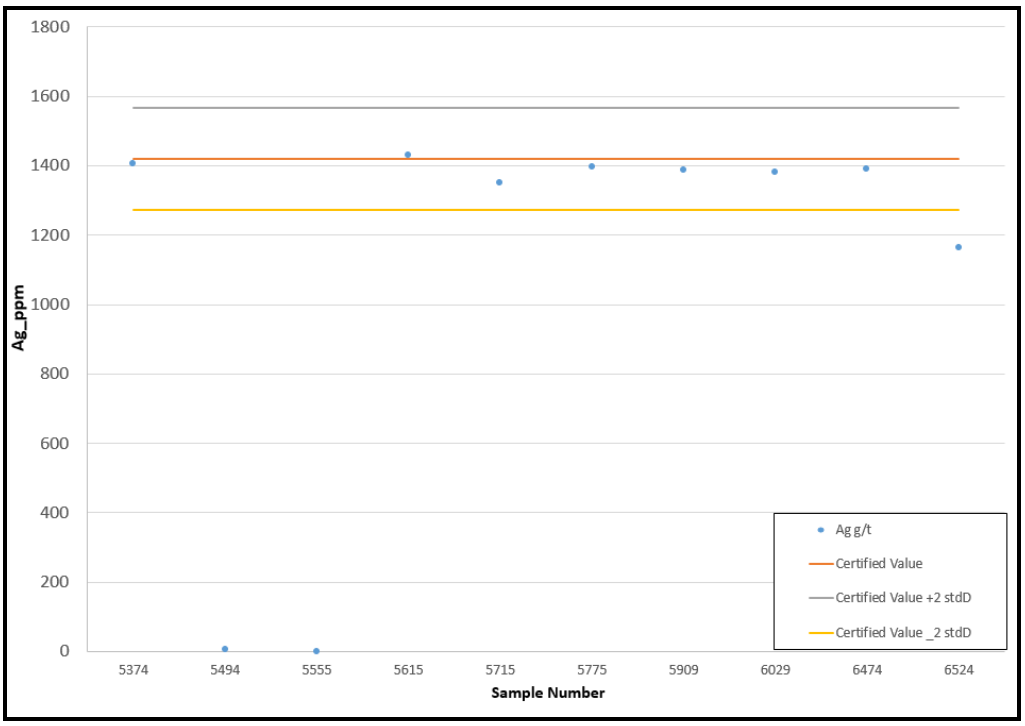


Figure 11.9 GBM998-9 Chart for Vertikalny Exploration Data



It is noted that the only high-grade silver standards used was GBM303-1, and this standard was only submitted eleven times (Figure 11.10). The GBM303-1 plot highlights two outliers which may have been miss labelled due to the near detection limit values recorded.

Figure 11.10 GBM303-1 Chart for Vertikalny Exploration Data



Tetra Tech note that standard performance across the 2012 campaign was poor, leading to outliers shown in GBM310-16, GBM909-13 and GBM909-11 (Figure 11.11 to Figure 11.13). These outliers are near to the silver detection limit so could have been miss labelled as blank material or another factor. Interrogation of the assay results across the five 2012 drill holes shows the data is all below cut-off grade. Thus there would be little impact on the overall Vertikalny resource as standard performance across these SRMs is adequate.

Figure 11.11 GBM310-16 Chart for Vertikalny Exploration Data

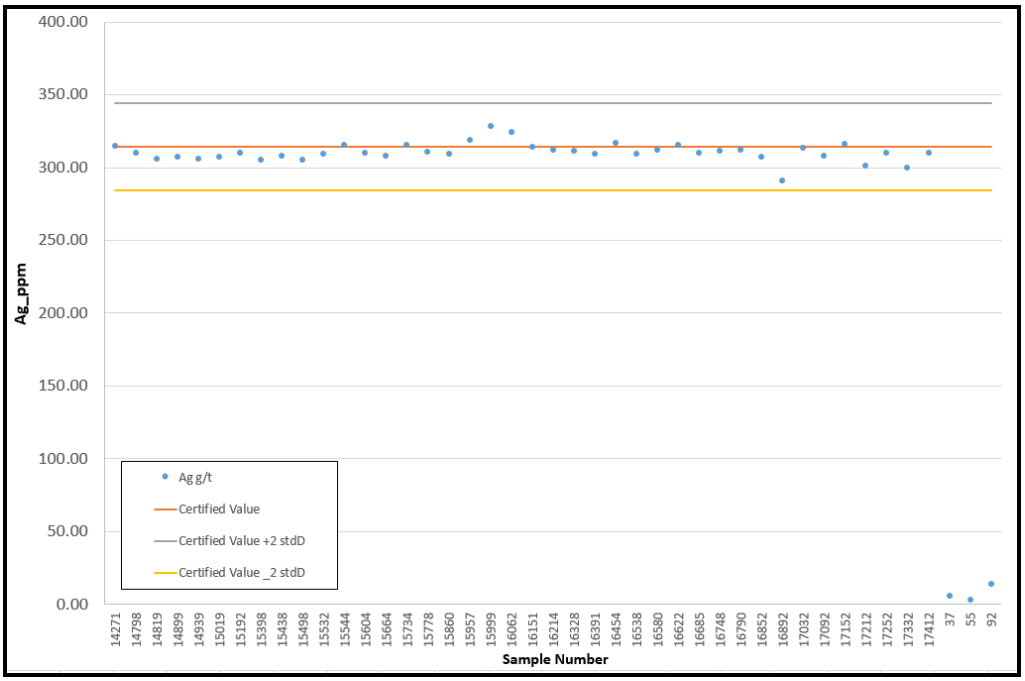


Figure 11.12 GBM909-13 Chart for Vertikalny Exploration Data

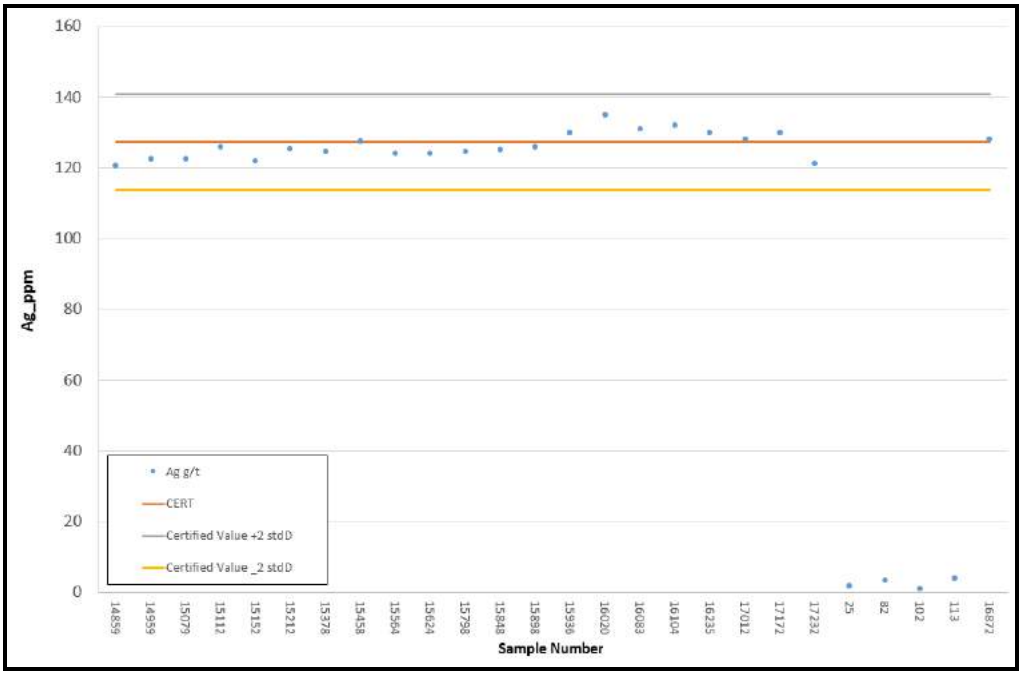
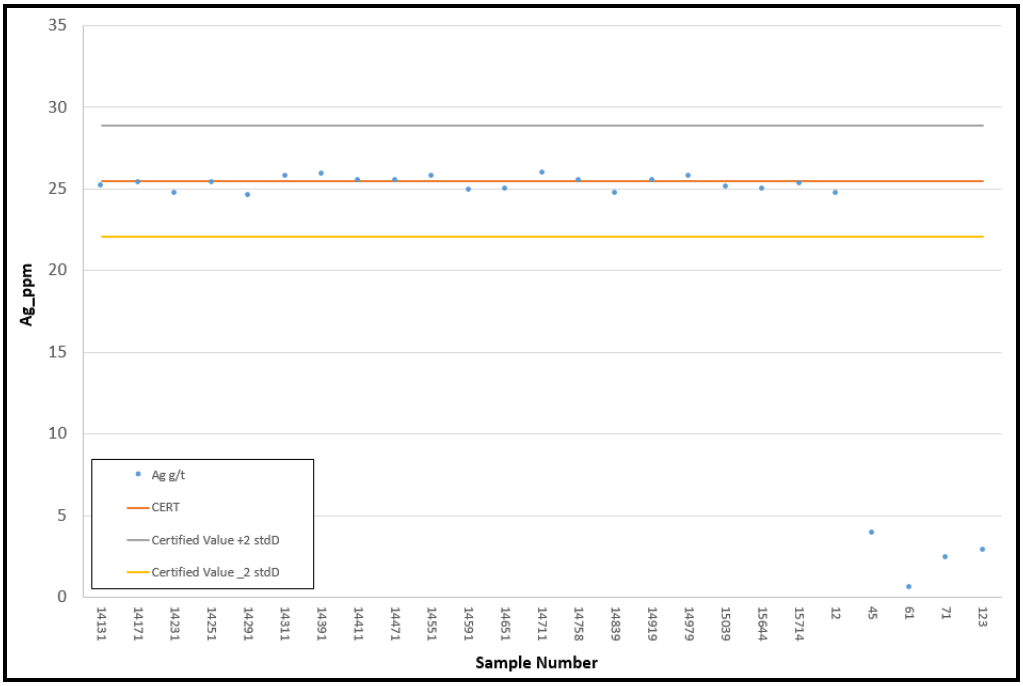


Figure 11.13 GBM909-11 Chart for Vertikalny Exploration Data

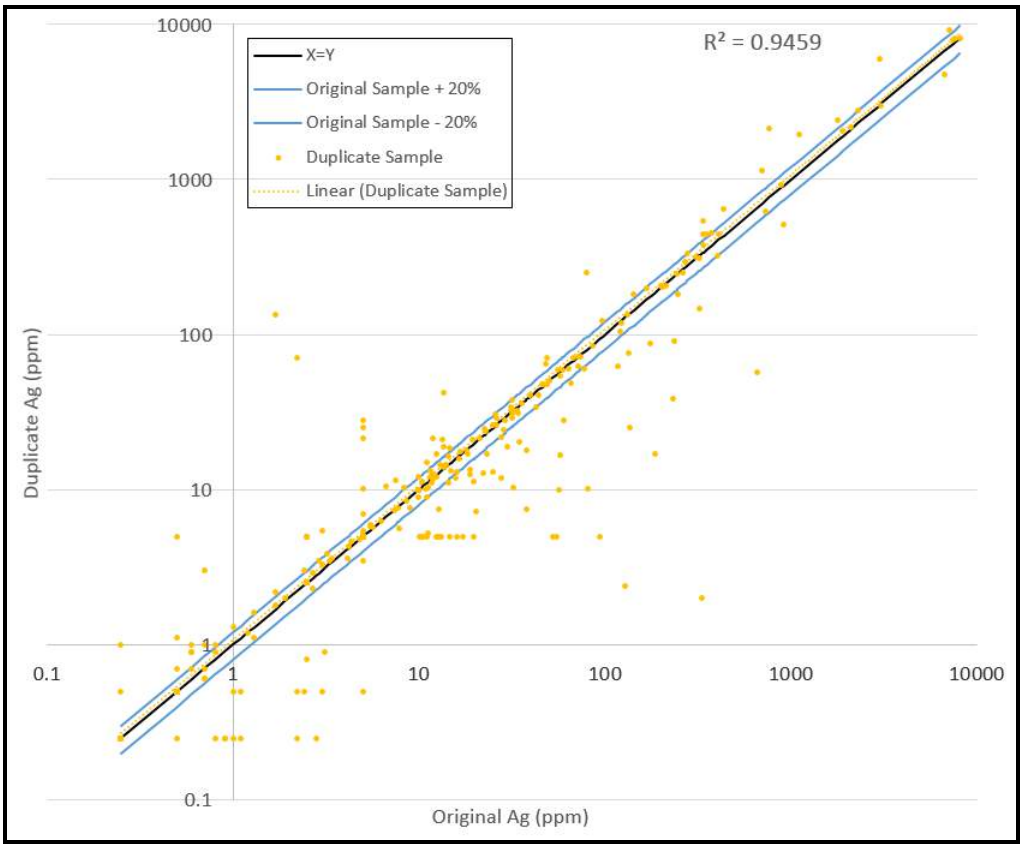


Standard reference samples were independently assessed for each campaign, and were found to be well within acceptable limits.

PROJECT DUPLICATES

Between 2007 and 2015, 533 duplicates were submitted as part of the Vertikalny exploration programs. The data ranges from less than 10 g/t through to greater than 8,000 g/t. Overall, the data show good levels of precision and repeatability, with a slope of regression of 0.95. The data suggest an improvement as the grade increases. Many of the duplicates falling outside of ±20% difference envelope are at lower grades less than 100 g/t. There are also indications of duplicate swapping within the 7 to 50 g/t range (Figure 11.14) where duplicates were used in error and have had the results recorded as a different sample.

Figure 11.14 Duplicate Chart for Vertikalny Exploration Data



Field duplicates have also been examined on a campaign by campaign basis, and show similar results with repect to precision and repeatability.

ALS Chemex also submit laboratoroy duplicates as part of their internal QA/QC programme. The analysis of the laboratory duplicate data shows good compliance, with excellent precision and repeatbility.

11.1.5 NIZHNY ENDYBAL QA/QC

The Nizhny Endybal exploration campaigns were completed during the 2005, 2011, and 2012 field seasons. The QA/QC data associated with the 2005 campaign are no longer available; however, drillholes completed during the 2011 and 2012 campaigns were twinned with 2005 holes to validate the information. The 2005, 2011, and 2012 drillhole data was used in estimating the current Inferred Mineral Resource. As part of the 2011/2012 exploration campaigns 607 QA/QC samples were submitted, along with 2,752 regular samples.

NIZHNY ENDYBAL BLANKS

Blank samples are created from local non-mineralised rock and processed in the on-site sample preparation laboratory. Blank material is stored on the ground outside of the sample preparation laboratory.

Blank material was broken up with a hammer and placed into a sample bag. Blank samples were then inserted in numerical order into the sequence for preparation.

During the 2011/2012, 107 blank samples were analysed at Nizhny Endybal. Of those 107 samples, 3 were found to be out of range.

The three failures presented low silver grades of 1.9, 3.3, and 6.2 g/t. Overall, the blank performance was found to be acceptable.

Blank samples have also been independently assessed for each campaign, and were found to be well within acceptable limits.

Nizhny Endybal Standards

As part of the Nizhny Endybal exploration during 2011 and 2012, 193 standards were submitted for analysis. The standard reference materials used and the number of times they were submitted are outlined in Table 11.2.

Table 11.2 SRMs used in the Nizhny Endybal Exploration during 2011 and 2012

Standard	No. of Submissions	Ag Value (g/t)	Standard Deviation Ag	Number of Failures*	Failure Rate (%)
GBM 906-6	81	389.7	21.1	5	6
GBM 909-11	46	25.5	1.7	0	0
GBM 997-4	2	287.9	38.2	0	0
GBM 998-9	64	101.2	4.8	2	3
Total	193	-	-	7	4

Note: *failure defined as assay result falling outside of two standard deviations.

It is noted that the overall average silver grades are lower at Nizhny Endybal than Vertikalny, and that the range of standard samples used are appropriate.

No failures were reported for the following standards:

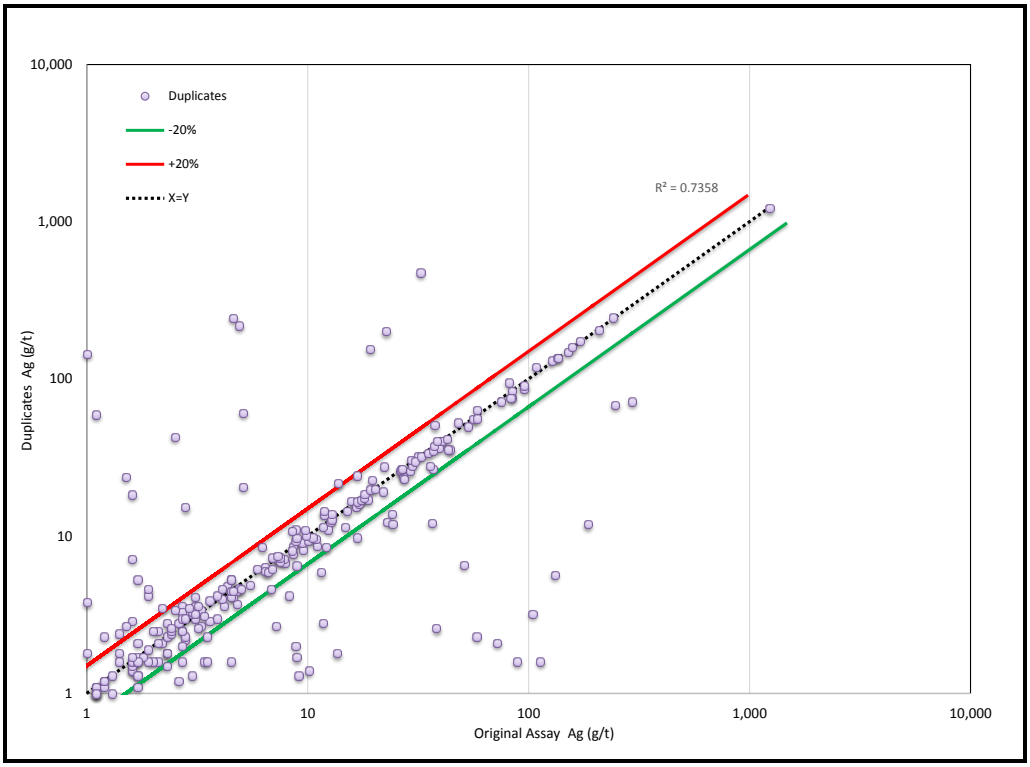
- GBM 909-11
- GBM 997-4.

Standard samples have also been independently assessed for each campaign and were found to be well within acceptable limits.

Project Duplicates

As part of the Nizhny Endybal exploration campaigns in 2011 and 2012, 307 duplicates were submitted. The data ranges from less than 10 g/t through to greater than 1,200 g/t. Overall, the data shows adequate levels of precision and repeatability, with a slope of regression of 0.74 (Figure 11.15).

Figure 11.15 Nizhny Endybal Duplicate Chart



Field duplicates have also been examined on a campaign by campaign basis and show similar results with repect to precision and repeatability.

ALS Chemex also submit laboratoroy duplicates as part of their internal QA/QC programme. The analysis of the laboratory duplicate data shows good compliance, with excellent precision and repeatbility.

11.1.6 MANGAZEISKY NORTH DEPOSITS QA/QC

Presented below is a summary of the QA/QC programme adopted for the samples collected since 2011 at the Mangazeisky North deposits.

BLANKS

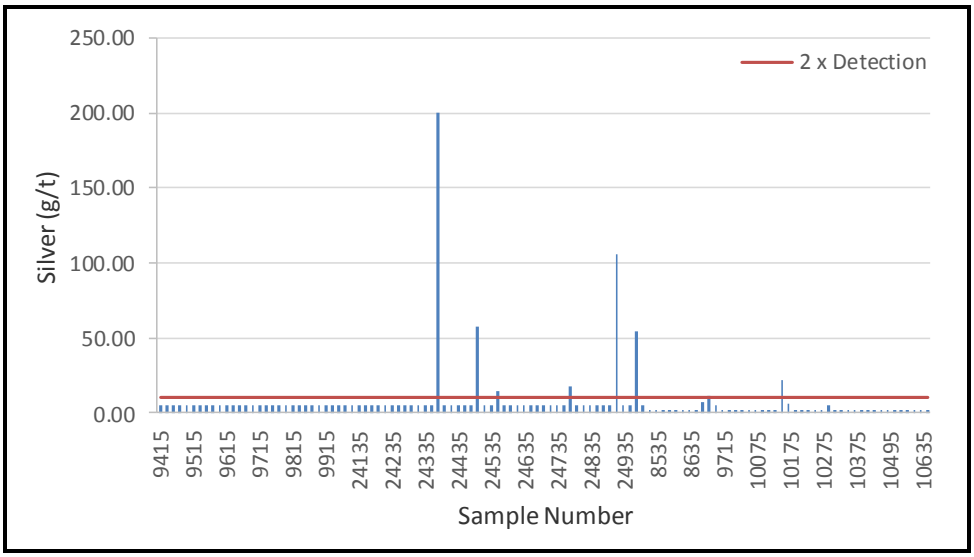
Blanks have been inserted into the sample stream every 20th sample.

Blank samples are created from local non-mineralised rock and processed in the on-site sample preparation laboratory. Blank material is stored on the ground outside of the sample preparation facility.

A total of 117 blank samples were analyses between 2011 and 2016 of which eight failed, which is a 7% failure rate. This includes 32 blank samples from the 2016 drilling program of which 1 failed, representing a 3% failure rate for 2016. Half of the failed blanks are low grade (less than 50 ppm) while the remaining failures are greater than 50 ppm (maximum 200 ppm). These failures may be due to minor mineralization within the blank material, rather than poor practice and do not represent material risk to the project. Tetra Tech recommends sourcing a true blank material to reduce the failure rate to zero.

Figure 11.16 shows the Tetra Tech blank analysis for silver grades.

Figure 11.16 Blank Sample Performance



Source: Tetra Tech

STANDARDS

Standard reference materials are inserted on site into the sample stream, every 20th sample. The standard reference materials were purchased from Geostats Pty Ltd.

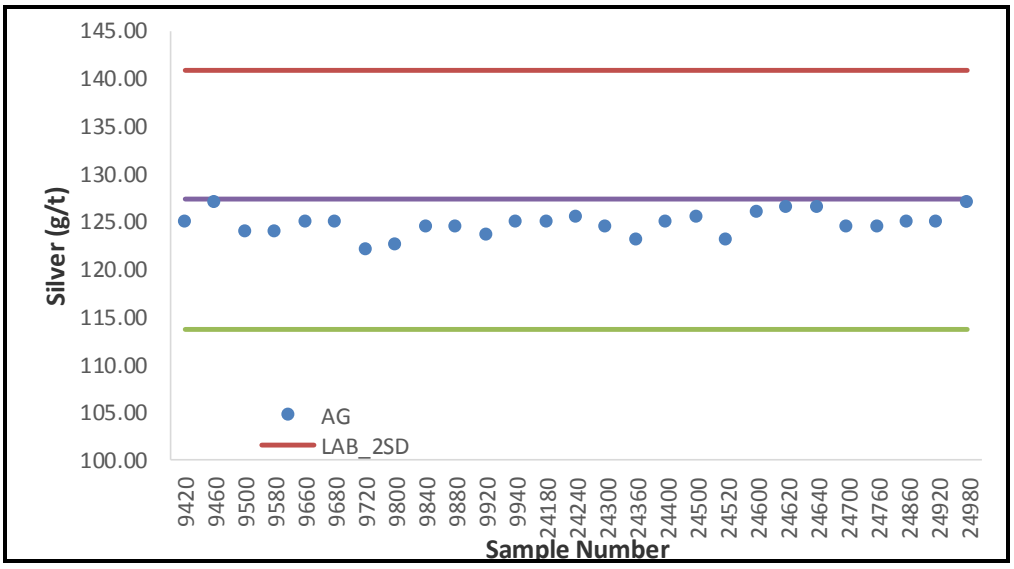
Four standard reference materials were used between the 2011 to 2016 exploration campaigns, GBM906-6, GBM909-13, GBM310-16 and GBM919-13. GBM909-13 and GBM913-13 performed well, with all of the results being within 2 standard deviations of the stated mean.

One sample from GBM906-6 did not fall within two standard deviations of the mean (Table 11.3) with a value below detection limit (less than 5 ppm). Tetra Tech recommend this sample is re-analyses as well as checked to determine if there has been any sample swapping. Five samples of GBM310-16 submitted to the laboratory failed to return results within two standard deviations of the stated mean (Table 11.3). Three of the results were consistently 30% lower than the expected value. The failure was queried with the laboratory (ALS), who commented that the failure was due to the standard having very high sulphur concentrations, which meant that the ICP analysis was not able to detect all of the contained silver. ALS commented that the precision and repeatability of their testing was within acceptable limits based upon internal duplicates. The sulphur content of GBM310-16 is significantly high than the content of the Mangazeisky mineralisation and therefore is not representative of the deposit. ICP is an appropriate method for the mineralisation, therefore GBM310-16 will be replaced by a different standard (with lower sulphur) for future campaigns.

Table 11.3 Silver Standard Performance

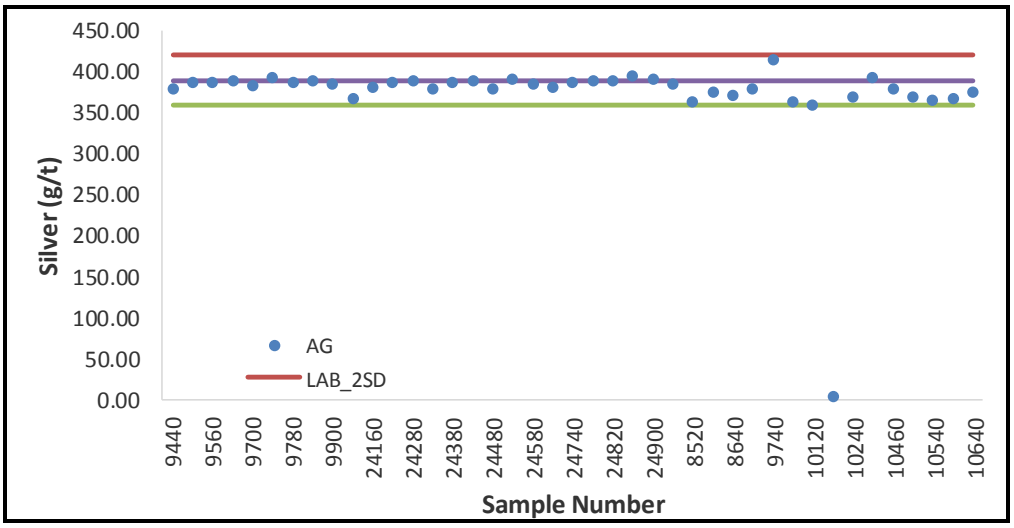
Name	Count	Certified Value	Ag Mean (g/t)	Ag Lab Standard Deviation (g/t)	Difference (%)	Ag Fail >2 Standard Deviations
GBM906-6	41	389.7	372.4	14.9	-4%	1
GBM909-13	27	127.3	124.8	6.8	-2%	0
GBM913-13	16	74.1	75.2	3.9	1%	0
GMB310-16	31	314.3	299.1	14.9	-5%	5

Figure 11.17 Standard GBM906-6 Performance



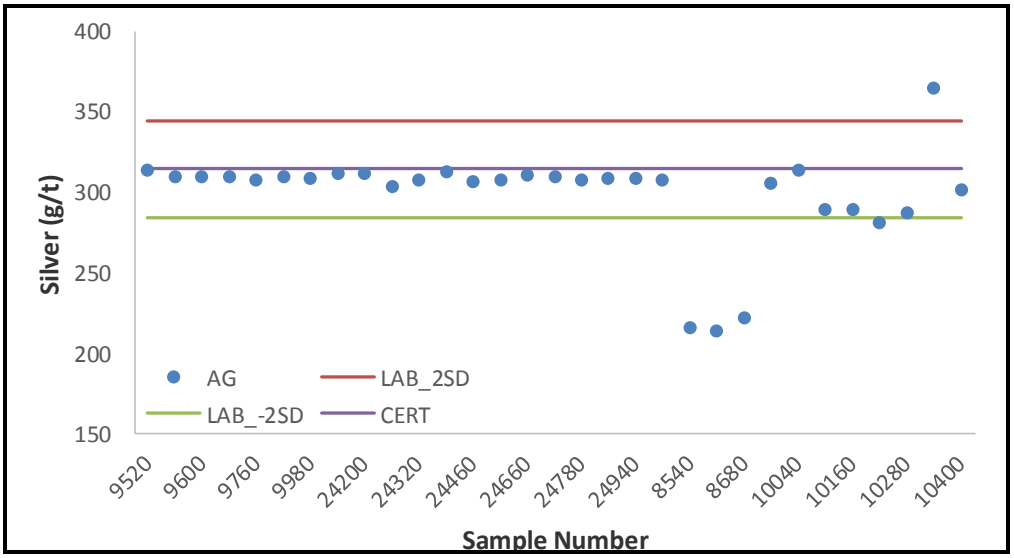
Source: Tetra Tech

Figure 11.18 Standard GBM909-13 Performance



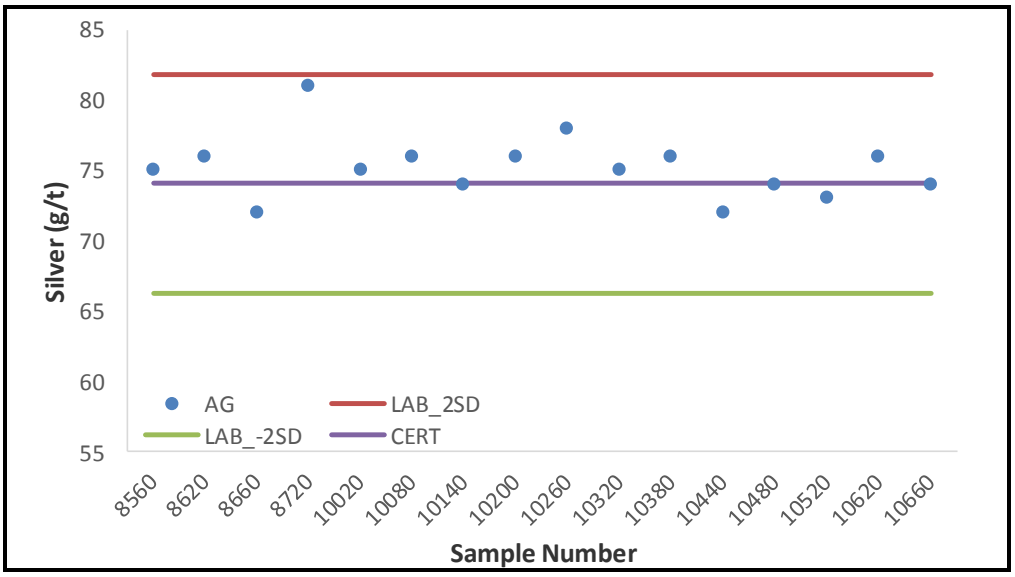
Source: Tetra Tech

Figure 11.19 Standard GBM310-16 Performance



Source: Tetra Tech

Figure 11.20 Standard GBM913-13 Performance



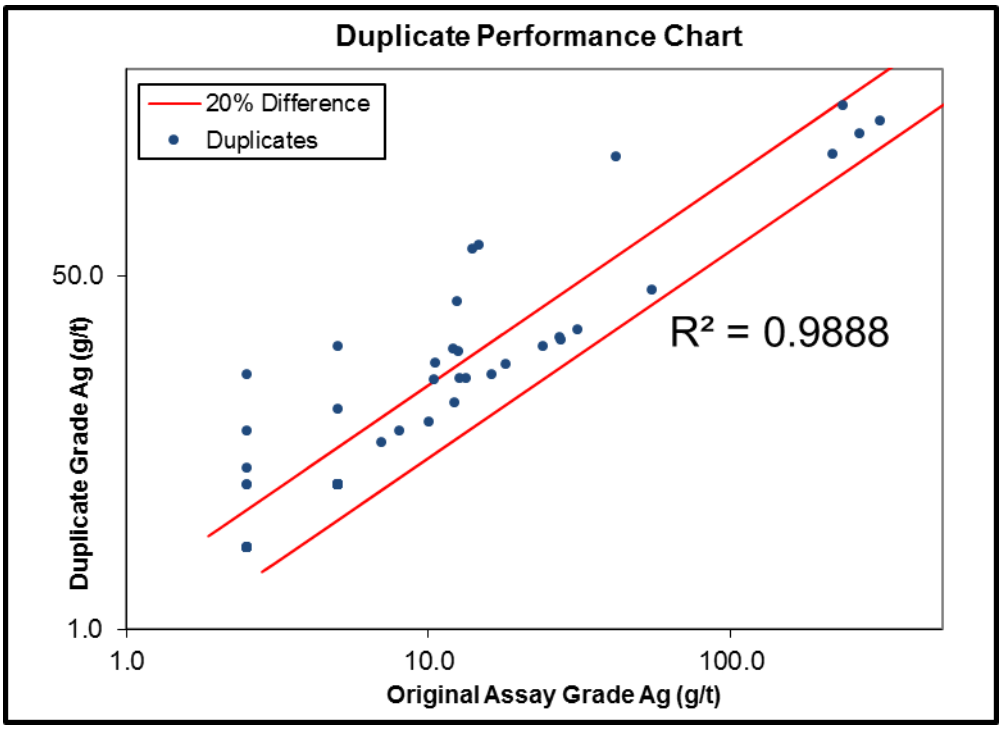
Source: Tetra Tech

In the context of the previous good standard performance, Tetra Tech considers the standard assessment for the Project has been robust. Tetra Tech recommend the replacement of GBM310-16 with a lower sulphur standard and a continuing of the monitoring of the standard performance over time to identify any drift.

DUPLICATES

Duplicate samples were inserted into the sample stream every 20 samples from the pulps. No field duplicates were analysed. In total 104 duplicates were assessed across the 2011 to 2016 exploration campaigns.

Figure 11.21 Pulp Duplicate Performance (Log Scale)



Source: Tetra Tech

Eighteen duplicates failed to fall within ±20% of the original sample, representing a 17% failure rate. The majority of the failed samples are found within the low grade proportion of the graph within waste rock duplicate samples. Tetra Tech deems the duplicate samples across the exploration programs adequate for Mineral Resource estimation.

Tetra Tech recommends changing the sampling of duplicates to focus on the mineralised intersections, where duplicate analyses will be more meaningful than analysing the waste that is inevitable on a set 20 sample interval.

11.2 SECURITY

Silver Bear has implemented adequate systems in relation to sample security during transport and on-site storage and handling, with satisfactory chain of custody documentation and procedures.

Database security is maintained through a limited number of personnel having access to the database. One staff member is responsible for the transfer of assay results to the drillhole database in the office in Yakutsk. The database is password protected on a backed-up network drive and is verified by the chief geologist for the Project.

11.3 TETRA TECH OPINION

Tetra Tech considers that the QA/QC programmes for drill core are suitable to support Mineral Resource estimation. The number of samples assessed for QA/QC purposes is to industry best practice. Tetra Tech recommends focusing the QA/QC programme

toward the mineralised intersections in future campaigns. Tetra Tech further recommends that core sample duplicates should be made by splitting two samples from the initial crushing cycle, as well as by splitting two samples from the pulped material.

The trenching campaign completed in 2013 did not contain QA/QC samples, it is recommended that the trenches are re-sampled with the inclusion of QA/QC samples.

12.0 DATA VERIFICATION

12.1 TETRA TECH SITE VISIT

Tetra Tech has conducted several site inspections in recent years. The most recent site visit to the property was performed by Mr. Robert Davies, CGeol, EurGeol, Senior Resource Geologist for Tetra Tech in July 2016.

12.1.1 TETRA TECH JULY 2016 SITE VISIT

Mr. Davies conducted a site visit to the Property between 26th June and 9th July.2016. At the time of the visit, drilling was being completed on Mangazeisky North deposit. No active drilling or exploration was taking place in the Vertikalny area. The visit included an inspection of the Mangazeisky North and South, Porfirovy, Vertikalny, Sterzhnevoy, Vostotchny and Kis Kuel deposits.

The site visit also included an inspection of the base camp, core logging and core storage facilities, drill core sampling preparation and sample rejects storage facilities.

Mr. Davies was accompanied on the site visit by the following personnel:

- Sergey Kraushkin, Chief Geologist for Prognoz
- Sergey Ivanov, Translator for Prognoz
- Vladimir Stepanov, Exploration Geologist for Prognoz.

12.1.2 DRILLHOLE LOCATIONS

The locations of several drillholes were measured for comparison with coordinates provided by Silver Bear. Drillhole collar locations were verified using a hand-held global positioning system (GPS); Garmin™ GPSmap 60Cx. The collar locations were found to be consistent with the drillhole database survey data, given GPS unit accuracy, the X and Y coordinates are within ±6 m in X and Y (Table 12.1).

Drillholes were marked by logs stuck into the drillhole. The logs were marked with an aluminium tag bearing the drillhole number.

At the time of the site visit, an infill drill programme was in progress on the Vertikalny deposit, with one drill rig in operation. Drill sites were left tidy and clear of debris.

Table 12.1 Verified Drillhole Collar Coordinates (UTM WGS 84)

Drillhole/ Trench Name	Silver Bear		Tetra Tech		Δ X (m)	Δ Y (m)
	X (m)	Y (m)	X (m)	Y (m)		
Mangazeisky North						
MN14-13	552398	7290384	552402.5	7290387.9	4.5	3.9
MN14-14	552424	7290400	552425.8	7290404.5	1.8	4.5
MN14-17	552454	7290366	552454.1	7290368.2	0.1	2.2
MN14-18	552477	7290368	552476.5	7290372.1	-0.5	4.1
MN14-24	552450	7290240	552449.6	7290243.2	-0.4	3.2
MN14-27	552474	7290194	552475.7	7290197.4	1.7	3.4
MN14-37	552454	7290392	552456	7290394.7	2.0	2.7
MN14-13	552398	7290384	552402.5	7290387.9	4.5	3.9
Mangazeisky South						
MS14-02	552656	7289761	552655.6	7289767	-0.4	5.7
MS14-03	552638	7289785	552639.2	7289789	1.2	3.9
MS14-04	552633	7289804	552632	7289807	-1	3.2
MS14-05	552621	7289828	552621.1	7289831	0.1	2.8
MS14-07	552598	7289877	552597.9	7289879	-0.1	1.8
MS14-09	552608	7289915	552608.9	7289918	0.9	2.7
MS14-10	552609	7289910	552609.3	7289914	0.3	4.2
MS14-15	552684	7289777	552683.4	7289782	-0.6	4.9
MS14-16	552606	7289792	552605.4	7289796	-0.6	4.2
MS14-17	552581	7289839	552581.9	7289840	0.9	1.1
Vertikalny Central						
V08-080	551473	7283730	551471.3	7283730.9	-1.7	0.9
V09-145	551357	7283824	551356.4	7283822.5	-0.6	-1.5
V13-48	551085	7284015	551085	7284016	0	1
V13-48	551083	7284016	551085	7284016	2	0
V14-05	551185	7283932	551186.7	7283930.6	1.7	-1.4
V14-06	551181	7283924	551180.4	7283923.6	-0.6	-0.4
Sterzhnevoy						
ST15-08	552940	7287072	552938	7287071	-1.9	-0.9
ST15-04	552895	7287106	552894	7287106	-1.4	0.4
ST15-02	552913	7287167	552911	7287167	-1.7	0
ST15-05	552943	7287115	552940	7287115	-2.7	-0.4
Vertikalny Northwest						
V09-116	549129	7285542	549130.9	7285540.9	1.9	-1.1
VNW14-01	548980	7285572	548984.1	7285570.3	4.1	-1.7
VNW14-02	549019	7285563	549020.8	7285560.7	1.8	-2.3
VNW14-07	549145	7285525	549144	7285525.8	-1	0.8
VNW14-08	549171	7285515	549171.1	7285517	0.1	2
VNW14-09	549159	7285504	549160.4	7285505.9	1.4	1.9
VNW14-10	549171	7285487	549169.7	7285492.3	-1.3	5.3
VNW14-11	549155	7285433	549156.8	7285434.1	1.8	1.1
VNW14-12	549176	7285421	549178.3	7285423.9	2.3	2.9

table continues...

Drillhole/ Trench Name	Silver Bear		Tetra Tech		Δ X (m)	Δ Y (m)
	X (m)	Y (m)	X (m)	Y (m)		
VNW14-13	549060	7285515	549062.2	7285514.5	2.2	-0.5
VNW14-14	549037	7285535	549040.8	7285532.9	3.8	-2.1

Note: Δ – delta

12.1.3 INDEPENDENT CHECK SAMPLES

Independent samples of the drill core were not collected by Tetra Tech on the most recent site visit, due to restrictions in Russia on travelling with core samples. However, on previous site visits Tetra Tech collected one grab sample of massive galena from Trench K-1322 at Mangazeisky North and sent in to Actlabs in Ancaster, Ontario, for analysis. The assay report returned a value of 4,170 g/t silver. This sample is show below in (Figure 12.1).

Figure 12.1 Grab Sample from Trench K-1322 at Mangazeisky North

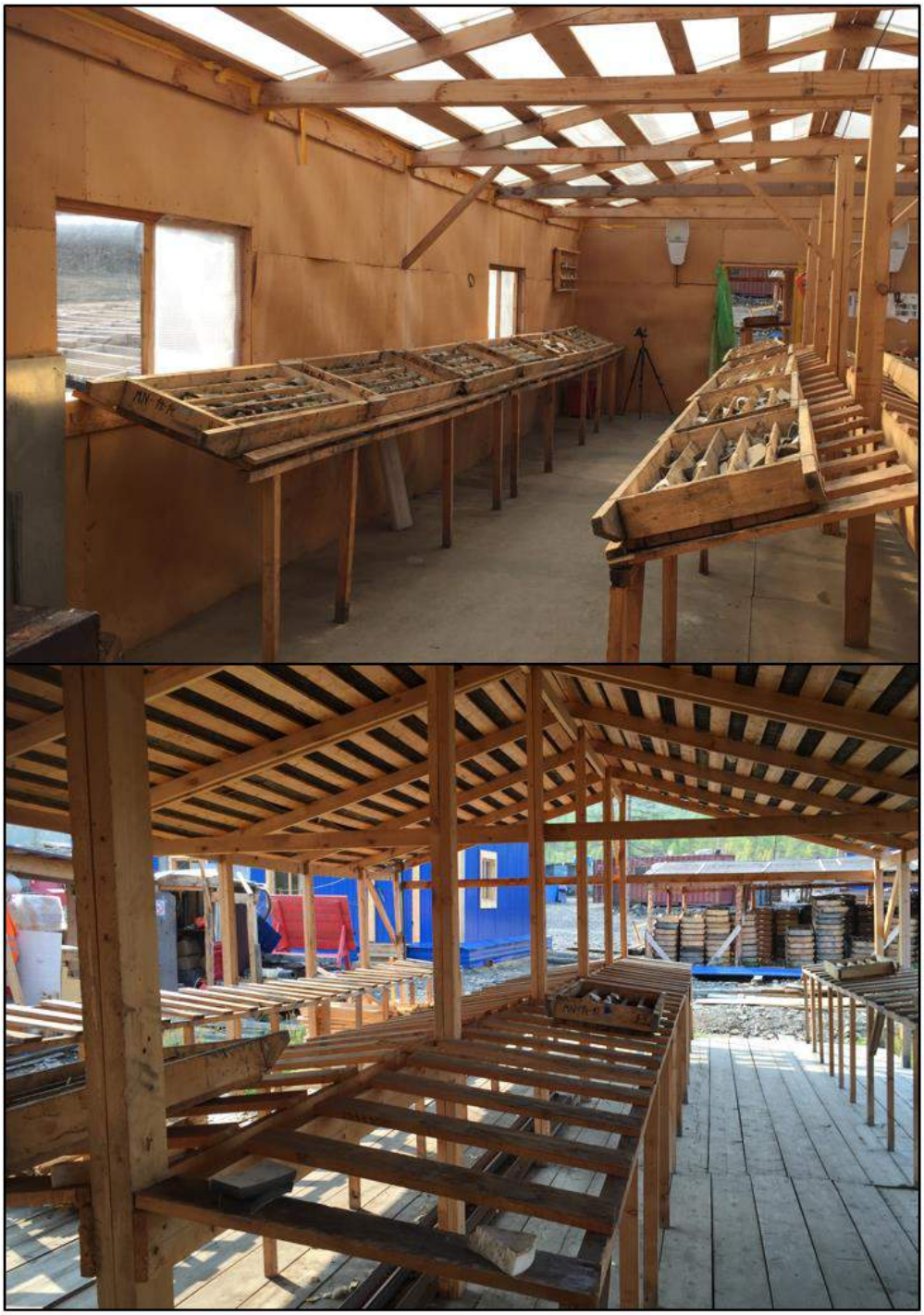
Source: Tetra Tech

12.1.4 CORE LOGGING AND SAMPLING

Upon receipt of all core boxes from any one drillhole, the core boxes were brought in and laid out on the core logging tables. Drill core was then aligned; geotechnical logging was carried out first, followed by geological core logging. Core logging was carried out or supervised by Mr. Kraushkin. Logging procedures are described in Section 10.0.

The logging facility was kept clean and tidy with sufficient lighting (Figure 12.2).

Figure 12.2 Core Logging Facilities; Indoor and Outdoor



Source: Tetra Tech

CORE SAMPLES

Core splitting and sampling was carried out in batches as each drillholes or series of drillholes was completed. Sample intervals were selected based upon observations of lithology/mineralogy through mineralized intercepts, generally with three systematic samples taken before and after mineralization. Systematic samples were 1 m in length.

Sampling and QA/QC procedures were in place. Standards, duplicates and blanks were inserted at regular intervals. Drill core was split historically by pneumatic splitter and for recent campaigns by diamond saw, where one half was placed in the sample bag with the sample tag and the other half was returned to the core box for storage.

Core Sample Preparation Facilities

The following equipment was inspected (Figure 12.3):

- pneumatic splitter
- two diamond saws
- two jaw crushers
- roller crusher
- sample dryer
- sample splitter.

No sample preparation activities were carried out during the site visit. Sample preparation and equipment was kept reasonably clean and tidy.

Figure 12.3 Sample Preparation Equipment; (left to right) Pneumatic Splitter, Sample Dryer (Kiln), Samples Being Crushed and Jaw Crusher

Source: Tetra Tech

Storage of Core and Sample Pulps

All storage facilities at the camp were inspected. Drill core from previous drill programs are stored under covered shelters adjacent to the core logging and sample preparation facilities (Figure 12.4). The core boxes are stacked by drillhole.

Figure 12.4 Core Storage Facilities; Current and New Core Storage Facilities

Source: Tetra Tech

All sample pulps from the previous drill programmes are stored on site at the camp. All sample pulps are stacked on sturdy wooden shelves by drillhole number (Figure 12.5).

Figure 12.5 Sample Pulps Storage

Source: Tetra Tech

Tetra Tech observed outcrops in trenches at Mangazeisky North & South, Porfirovy, Vertikalny, Vostotchny, Kis kuel and Sterzhnevoy, most of which were obviously mineralised. Native silver and malachite were seen in hands specimen at Sterzhnevoy.

Figure 12.6 Mineralised Veins in Outcrop at Verticalny Central (Left) and Mangazeisky North (Right)

Source: Tetra Tech

12.2 TETRA TECH OPINION

It is Tetra Tech's opinion that the current drill program, logging and sampling procedures are consistent with recognised industry best practices and are considered adequate for this type of deposit.

12.3 OFFICE BASED DATA VERIFICATION

12.3.1 ASSAY CERTIFICATES

Silver Bear provided Tetra Tech with signed portable document format (.pdf) assay certificates for the drilling campaigns. Tetra Tech checked 10% of the assay certificates against the values contained within the database. No errors were encountered.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 SUMMARY

This section details the mineralogical and metallurgical test work completed to date on the Mangazeisky ore samples.

Historical investigations were conducted by two organisations: TSNIGRI (2008) and the National Institute of Non-ferrous Metals (GINTSVETMET) (2011). A high-level summary of these investigations is presented in this section.

As part of the feasibility study, additional mineralogical and metallurgical test work was completed by two commercial laboratories: SGS Vostok (Chita, Russia) (2014) and TOMS (Irkutsk, Russia) (2015a; 2015b).

SGS Vostok tested a composite sample, which comprised samples collected from dedicated metallurgical drillholes within the Central Zone of the orebody, representing the oxide ore. The SGS Vostok test work was completed between January 2014 and December 2014.

TOMS completed two separate test work programs in 2015 as part of the feasibility study. TOMS initially completed a leach variability test program in February 2015, followed by leach optimisation and comminution test work for a primary ore composite. The primary ore testing was completed between November 2015 and December 2015.

In addition, various laboratory research programs were completed by TOMS in order to satisfy the Russian regulatory approval requirements. Tetra Tech was neither involved, nor had access to those laboratory research programs, apart from translated sections made available in June 2017. A high-level summary of the TOMS research work made available to Tetra Tech is included in this section.

Tetra Tech analysed all the metallurgical test work results with the objective of identifying the optimal process flowsheet. The metallurgical studies indicate that the oxide ore is amenable to gravity concentration and cyanidation and can be efficiently processed using a gravity and agitated tank leach based flowsheet. The study also indicates that the primary ore is amenable to cyanidation and can be effectively processed by a whole ore leach method.

Based on the metallurgical test work results, the LOM recovery for silver is estimated to be 78% for oxide ore and 69.6% for primary ore.

13.2 HISTORICAL TEST WORK REVIEW

13.2.1 TSNIGRI TEST WORK RESULTS (2008)

BACKGROUND

Two samples collected from trenches were used for the 2008 test work programme, which consisted of:

- elemental analysis
- phase analysis
- mineralogical characterisation
- chemical analysis of crushed mineral by size
- gravity concentration of the feed mineral
- flotation of gravity tailings
- cyanidation of flotation tailings.

SAMPLE MINERALOGY

This section summarizes the main conclusions of TSNIGRI (20089).

Two trench samples were tested; the key minerals found are detailed in Table 13.1.

Table 13.1 2008 Sample Chemical Analysis

Sample No.	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)
Sample 1	340	1.63	1.14	-
Sample 2	575	6.74	2.35	0.16

Both samples were analysed for minerals associated with silver (Table 13.2). It is evident that the silver association varies significantly between the two samples; however, for both samples the majority of silver is found within the mineral grain boundaries.

Table 13.2 Silver Association

Silver Characteristics	Sample 1		Sample 2	
	Ag (g/t)	Distribution (%)	Ag (g/t)	Distribution (%)
Free with Clear Surface	-	-	-	-
Covered with Film and Enclosed in Acid-soluble Minerals	112.3	17.55	0.1	0.01
In Joints	487.0	76.09	273.0	48.75
Enclosed in Galena	26.2	4.09	156.7	27.98
Enclosed in Sphalerite	7.2	1.13	82.2	14.68
Enclosed in Copper Minerals and Pyrite	7.3	1.14	48.0	8.57
In Acid-insoluble Residue	-	-	-	-
Total	**640.0**	**100.00**	**560.0**	**100.00**

In Sample 1, the majority of silver was present in open joints, with the balance present as free silver with film coatings. There was a poor association with sulphides. However, with Sample 2, there were no coated silver or acid-soluble minerals, and approximately half of the silver content occurred in open joints. The remainder of the silver was associated with sulphide minerals, mainly galena, but also sphalerite, with some pyrite and copper minerals. This highlights the variability in the samples taken.

Gravity Concentration

The focus of the test work programme was to maximise recovery of all metals into saleable gravity and flotation concentrates. Gravity separation was attempted first through a two-stage gravity table concentration, and second via a single table, followed by Knelson concentration of the tailings. The concentrates from the two gravity processes were combined to predict a total silver product grade and recovery.

On tabling, Sample 1 produced silver recoveries of 26.76% and Knelson concentration of the table tails achieved 24.32% silver recovery. The silver concentrate grades achieved were over 9,000 g/t for both products.

Sample 2 achieved better results, with the first test achieving a silver recovery of 45.40% and the second 49.99%. However, the silver grades were lower at approximately 2,500 g/t for both products.

The gravity test work indicated the highly-variable nature of the samples tested and suggested that additional upgrading of both the gravity concentrate and tailings is necessary to achieve an acceptable silver concentrate grade and recovery.

Flotation

Flotation recovery was evaluated on the first-stage gravity tailings for both Sample 1 and Sample 2. Flotation was undertaken at a number of grind sizes, which indicated a particle size (P_{85}) of 71 µm was optimum for Sample 1, and a particle size (P_{80}) of 71 µm was optimum for Sample 2. This once again indicates the difference between the two samples. Additional flotation tests were carried out to optimise metal recovery at varied reagent additions. The optimum results achieved are highlighted in Table 13.3.

Table 13.3 Gravity and Flotation Test Work Overview

Products	Mass (%)	Metal Content			Recovery		
		Ag (g/t)	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Zn (%)
Sample No. 1							
Gravity Concentrate of Stages I and II	1.68	9,265.67	59.89	2.07	24.32	60.64	2.81
Gravity Tails	98.32	491.26	0.66	1.22	75.68	39.36	97.19
Feed	100.00	638.23	1.65	1.24	100.00	100.00	100.00
Flotation of Gravity Tails							
Flotation Concentrate	3.42	12,320.00	6.55	17.57	66.02	13.54	47.45
Flotation Tailings	94.90	65.00	0.45	0.70	9.66	25.82	49.74
Total Product Recovery	-	-	-	-	90.34	74.18	50.26
Sample No. 2							
Gravity Concentrate of Stages I and II	11.42	2,517.90	44.68	4.79	49.99	77.28	20.79
Gravity Tails	88.58	324.75	1.69	2.36	50.01	22.71	79.21
Feed	100.00	575.30	6.60	2.63	100.00	100.00	100.00
Flotation of Gravity Tails							
Flotation Concentrate	5.31	4,680.00	15.20	20.00	43.23	12.23	40.35
Flotation Tailings	83.27	46.80	0.83	1.23	6.78	10.48	38.86
Total Product Recovery	-	-	-	-	**93.22**	**89.51**	**61.14**

Note: *The tailings grade was adjusted by Tetra Tech to ensure correct silver balance.

CYANIDATION

Flotation tailings were leached with cyanide to assess if this would increase silver recovery. A silver recovery of 55 to 71% was achieved, representing an additional overall recovery increase of 5 to 6%. The test work conditions were undertaken at high-cyanide consumptions; however, lower cyanide dosages were not trialled during these tests, and therefore the optimum cyanide addition was not determined.

13.2.2 GINTSVETMET TEST WORK RESULTS (2011)

BACKGROUND

The test work carried out by GINTSVETMET (2011) investigated the following aspects of the ore:

- Chemical composition of a composite sample and the drillhole samples were analysed.
- Two composite samples were made up from two drill zones labelled SB-1 and SB-2. These two samples were analysed for chemical composition and mineralogy. Elemental analysis and the association of the silver were determined for sample SB-2:
  - SB-1 comprised of material collected from surface and oxidised material

- SB-2 was a composite sample made up from drill core samples (the laboratory report states that this sample is the most representative of the mineral deposit).

Samples were used for beneficiation test work, which included:

- milling parameters on SB-2
- laboratory flotation tests on sample SB-1
- gravity concentration on feed mineral
- gravity concentration on flotation tailings
- separate flotation of fine and coarse material
- locked flotation test work
- cyanidation of ROM mineral and flotation concentrate.

Sample Mineralogy

The chemical composition of the samples is summarised in Table 13.4.

Table 13.4 Sample Chemical Composition

Element	Unit	SB-1	SB-2
Silver	g/t	695	710
Lead	%	1.3	4.8
Zinc	%	1.3	1.7
Copper	%	0.1	0.1
Iron	%	10.9	18.9
Antimony	%	0.1	0.1
Arsenic	%	0.6	0.3
Sulphur	%	1.5	3.6
Calcium Oxide	%	0.3	0.5
Sodium Oxide	%	0.2	0.1
Magnesium Oxide	%	0.6	1.2
Silicon Oxide	%	52.6	34.0
Aluminium Oxide	%	17.2	11.2

SB-2 is understood to be the most representative of the mineral deposit, and was analysed for silver mineral association. Table 13.5 indicates that the majority of the silver is free (47.4%), with the remainder associated with galena, sphalerite, other sulphides, oxides/sulphosalts or rocks, and carbonates. It is anticipated that the silver recovery on cyanidation of these minerals will be 98% free/open; 80% lead sulphides; 80% zinc and iron sulphides; 90% other sulphides, oxides, and sulphosalts; and 85% rocks and carbonates. Based on these estimations, the cyanide recovery will be approximately 91%; however, this requires confirmation via test work.

Table 13.5 SB-2 Silver Association

Silver Form	Grade (g/t)	Distribution (%)
Free/Open	336.2	47.4
Associations with:		
Galena	115.9	16.3
Sphalerite	78.5	11.1
Other Sulphides, Oxides and Sulphosalts	131.6	18.6
Rock and Carbonates	46.7	6.6
Total	**708.9**	**100.0**

Milling Parameters

The key milling parameters outlined in GINTSVETMET (2011) include:

- Protodyakonov hardness: 11 to 23
- Bond Work Index: 12 to 13 kWh/t.

Flotation Test Work

Stage flotation of lead and zinc was undertaken at various grind sizes. The optimum grind size was found to be 96.5% -71 µm. The results are summarised in Table 13.6.

Table 13.6 SB-1 Flotation Test Work Results at 96.5% -71 µm

Products	Mass (%)	Metal Content			Recovery		
		Ag (g/t)	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Zn (%)
Lead Concentrate – Intermediate	2.55	2,750.00	4.83	1.10	10.17	7.93	3.01
Lead Concentrate – Main	2.32	1,700.00	2.08	1.10	5.72	3.11	2.74
Lead Concentrate – Oxidised	4.88	1,400.00	2.43	2.05	9.90	7.63	10.75
Zinc Product	1.95	1,350.00	2.29	1.80	3.82	2.88	3.77
Tailings	88.30	550.00	1.38	0.84	70.40	78.45	79.72
Feed	100.00	689.86	1.55	0.93	100.00	100.00	100.00

Note: *The tailings grade was adjusted by Tetra Tech to ensure correct silver balance.

Gravity Concentration

Gravity concentration was trialled on the raw mineral sample SB-1 at a number of grind sizes, as well as flotation tails. Gravity concentration was attempted on the feed mineral at 55.7%, 79.1%, and 96.5% -71 µm. A maximum silver recovery of 17.63% was achieved at a grind size of 79.1% -71 µm. This indicates additional processing is required to recover silver from the gravity tailings.

Gravity concentration of flotation tailings showed 5.26% silver recovery to gravity concentrate. This indicates poor upgrade of flotation tailings by gravity.

Locked Cycle Flotation Test Work

Sodium carbonate, butyl xanthate, sodium sulphide, and frother reagents were used as flotation reagents in the locked cycle test work. Sample SB-1 showed poor silver, zinc, and lead recoveries to the lead concentrate, as outlined in Table 13.7.

Table 13.7 SB-2 Locked Cycle Flotation Test Results

Products	Mass (%)	Metal Content			Recovery		
		Ag (g/t)	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Zn (%)
Lead Concentrate	1.59	8,320	18.80	1.00	18.56	19.93	1.57
Tailings	98.41	590	1.22	1.01	81.44	80.07	98.43
Feed	100.00	713	1.50	1.01	100.00	100.0	100.00

Note: The tailings grade was adjusted by Tetra Tech to ensure the correct silver balance.

Sample SB-2 was therefore the focus of the locked cycle flotation test work. Zinc sulphate, dimethyl dithiocarbamate, and frother were used to recover a lead concentrate, whilst lime, copper sulphate, butyl xanthate, and frother agent were used for zinc recovery. The results from the locked cycle tests are shown in Table 13.8.

Table 13.8 SB-2 Locked Cycle Flotation Test Results

Products	Mass (%)	Metal Content			Recovery		
		Ag (g/t)	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Zn (%)
Lead Concentrate (Rougher)	4.15	5,500.00	58.00	4.84	33.06	47.90	10.75
Lead Gravity Concentrate	2.39	7,400.00	52.00	5.58	25.62	24.73	7.14
Lead Flotation Concentrate	1.24	9,525.00	55.30	7.65	17.11	13.65	5.08
Total Lead Concentrate	7.78	6,725.19	55.73	5.52	75.79	86.28	22.96
Zinc Concentrate	2.21	2,550.00	8.30	42.10	8.16	3.65	49.79
Zinc Flotation Tailings	3.03	570.00	1.20	1.30	2.50	0.72	2.11
Tailings	86.98	107.50	0.54	0.54	13.54	9.35	25.14
Feed	100.00	690.35	5.03	1.87	100.00	100.00	100.00

Note: *The tailings grade was adjusted by Tetra Tech to ensure the correct silver balance.

The zinc grade of the zinc concentrate appears to be low when compared with typical smelter concentrate requirements.

Cyanidation Test Work

Cyanidation was carried out at 44 µm on feed mineral and flotation concentrate, in order to assess the amenability of the mineral to cyanidation. Silver recovery during leaching of the feed mineral ranged from 19.3 to 51.3%. Flotation concentrate cyanidation achieved silver recoveries from 55.2 to 90.8%.

The wide range of recoveries achieved shows the variability in the samples used for the test work. The key to ensuring accurate predictor test work is the collection of representative samples on which to carry out the work. Tetra Tech is not aware of the sample selection method, or from which mineralised zone the samples originate, therefore, an accurate assessment of the mineral's amenability to cyanidation is not

possible. Subsequent test work was aimed at addressing this and assessed the potential for cyanidation leaching of the mineral deposit.

13.2.3 Conclusions of Historical Work

Silver is by far the most significant economic component within the ore and as a result, the recovery of the silver is the focus of this study. The Project's processing plant is in an isolated environment where access conditions are difficult, and the production of a concentrate product would be impractical in terms of transport requirements and supply and availability of flotation reagents. As a result, Tetra Tech proposes that a simple process flowsheet that produces a final silver product would be the most practical solution for this ore.

The test work carried out during the historical studies indicates that highly variable samples were taken and tested for gravity, flotation, and cyanide recovery. The test work did not simulate the processing scheme that Tetra Tech proposes. It was not possible to effectively evaluate the performance of a Knelson gravity concentrator followed by intensive leaching of the gravity concentrate and standard cyanide leaching of the gravity tails. As the test work did not provide adequate predictions of actual process performance, the mineralogy of feed sample SB-2 was used to determine the expected gravity and cyanidation recovery achievable from this mineral. This was based on the fact that the SB-2 was stated as being most representative of the actual mineralisation in the GINTSVETMET (2011) test work report.

If Silver Bear were to implement a gravity and flotation process scheme, a number of site-specific problems would arise. The flotation test work indicates that the zinc concentrate produced did not achieve a saleable zinc grade. Additionally, owing to the isolated plant location and difficult transport logistics, gravity and lead concentrate would require leaching to recover the silver prior to shipment off site, thereby reducing the value of the lead concentrate. Tetra Tech, therefore, proposes a less complex process design that will make certain that all of the ROM feed is leached, ensuring maximum opportunity for silver recovery.

13.3 Feasibility Study Test Work (SGS Vostok 2014)

13.3.1 Metallurgical Sampling

Sample Background

Tetra Tech was not involved with the sample selection for the metallurgical test work completed at SGS Vostok, but did review the sample lists provided by Silver Bear and SGS Vostok. It is understood that the test work samples were collected from nine metallurgical drillholes (Figure 13.1) within the Central Zone (Figure 13.2) of the mineralised areas. It appears that the drillholes covered the high-grade areas of the Central Zone, which could be mined during the early years of operation.

Figure 13.1 Location of Metallurgical Drillholes

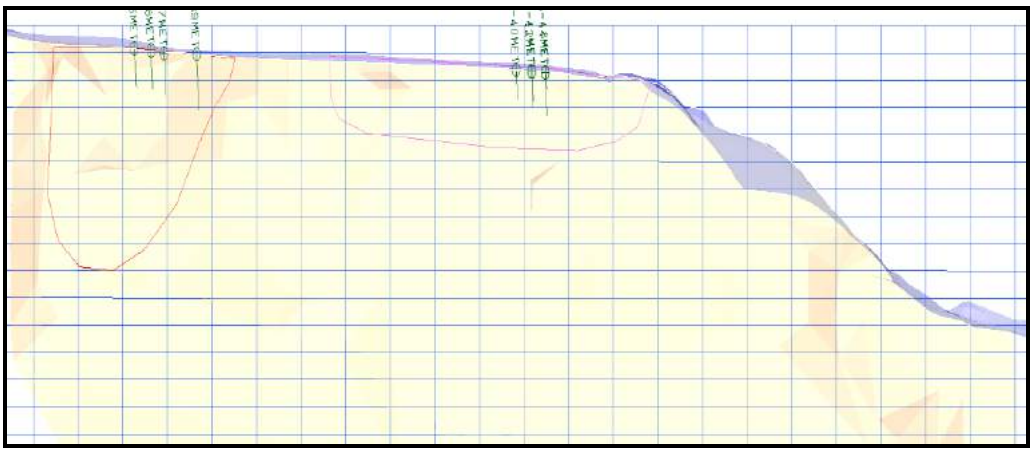
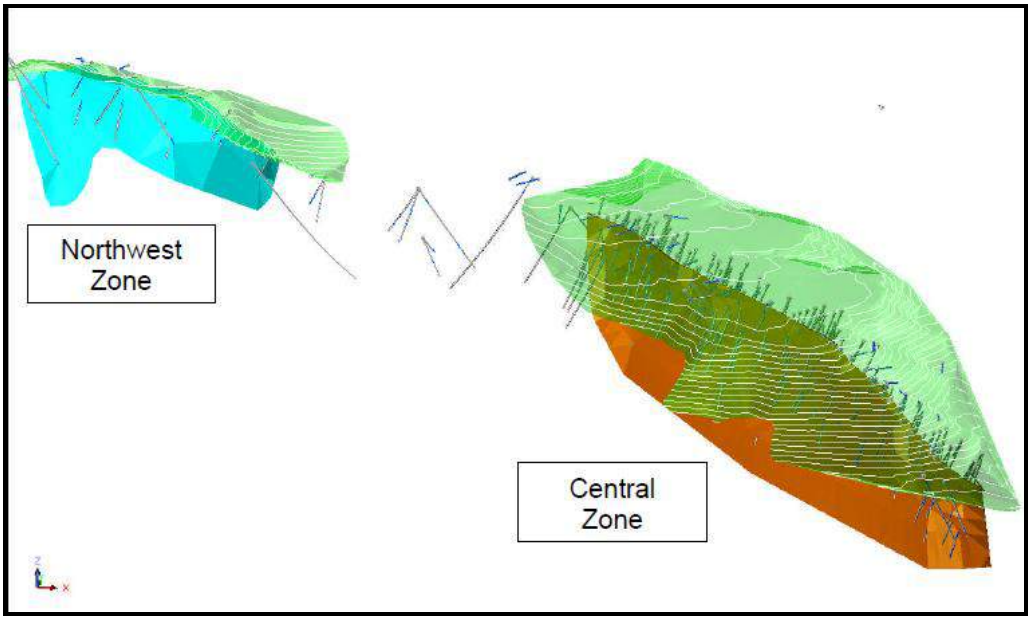


Figure 13.2 Mineralised Zones within the Orebody



Metallurgical samples were shipped to SGS Vostok in nine boxes as shown in Table 13.9

Table 13.9 Samples Sent to SGS Vostok

Pit Location	Hole ID	Test Work Stage
Zone 1	V13-35	Stage 1
	V13-36	
	V13-37	
	V13-38	
	V13-39	
Zone 2	V13-40	Stage 2
	V13-41	
	V13-42	
	V13-43	

Sampling Program

Tetra Tech was not directly involved with the selection or QA/QC aspects of the sampling programme. However, it is understood that trained personnel were involved in all stages of the sampling and adhered to industry best practices of sampling. Satisfactory evidence to support this was shown to Tetra Tech during a site visit to the SGS Vostok laboratories in July 2014.

A list of the samples used in the metallurgical test work programs is shown in Table 13.9.

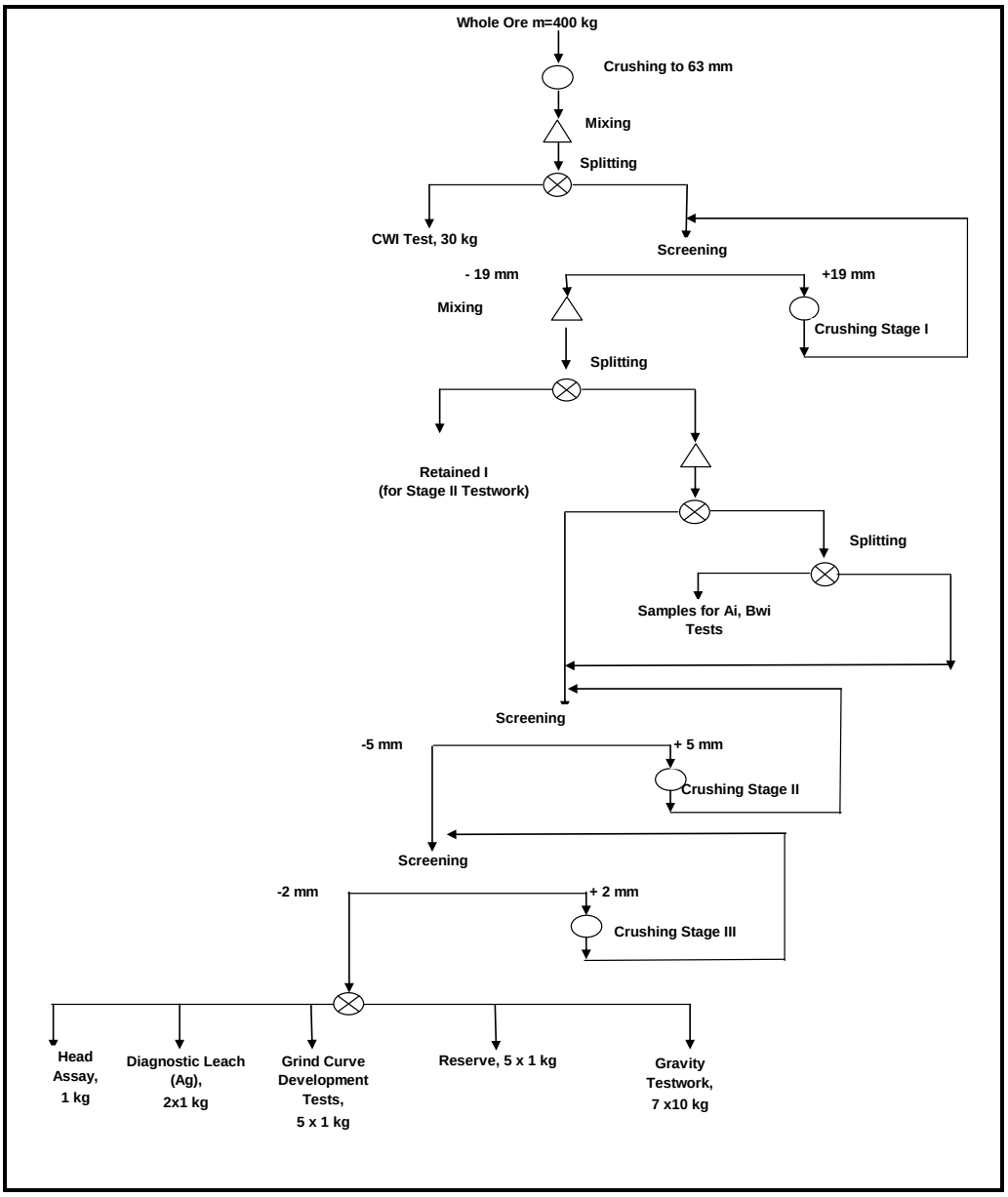
Sample Preparation

The samples were delivered to SGS Vostok in two wooden crates (Figure 13.3) as whole HQ core (Figure 13.4). SGS Vostok blended samples belonging to Zone 1 together in order to make a homogenised composite. Samples were then taken for various tests as shown in Figure 13.5.

Figure 13.3 Sample Delivered to SGS Vostok

Figure 13.4 One of the Delivered Core Boxes at SGS Vostok

Figure 13.5 Sample Preparation Flowsheet



ORE CHARACTERISTICS

Vertikalny ore is characterized as a polymetallic silver-lead-zinc, partially oxidized deposit. Silver is the most significant economic metal, with the presence of lead and zinc at less significant quantities.

Most of the silver is present in open joints with other minerals (metallic silver and acanthite), with the balance as free silver with film coatings. There is a very low association of silver with sulphides is observed in the sample tested by SGS Vostok.

The most abundant silver mineral is acanthine with the presence of galena and sphalerite, with subordinate values of other sulphides.

PHYSICAL PROPERTIES

The specific gravity of the samples was determined in the SGS Vostok laboratory. A summary of the specific gravity results is shown in Table 13.10. No other physical properties of the ore were determined during the current test work program.

Table 13.10 Specific Gravity of the Ore

Description	Specific Gravity (g/cm³)	
	Sample 1	Sample 2
Maximum Value	2.66	3.01
Minimum Value	2.28	2.34
Mean	2.50	2.65
Standard Deviation	0.10	0.12

Chemical Analysis

SGS Vostok analysed the head sample for silver, gold, copper, lead, and zinc assays, and conducted a whole rock analysis for all other major elements. A summary of the assay and whole rock analyses are shown in Table 13.11 and Table 13.12.

Table 13.11 Summary of Head Assays

Element	Unit	Assay
Silver	g/t	1,112.50
Gold	g/t	0.07
Copper	%	0.14
Lead	%	1.99
Zinc	%	1.80

Note: *Silver assays ranged from 1,020 g/t to 1,150 g/t with an average of 1,112.5 g/t.

Table 13.12 Summary of Whole Rock Analysis

Element	Unit	Assay	Element	Unit	Assay
Carbon (Organic)	%	0.06	Cadmium	ppm	130
Sulphur (Sulphides)	%	0.3	Chromium	ppm	30
Carbon (Total)	%	0.98	Iron	%	34.2
Sulphur (Total)	%	1.52	Potassium	%	1.01
Sulphur (elemental)	%	0.28	Magnesium	%	0.67
Beryllium	ppm	<0.5	Manganese	ppm	40,100
Bismuth	ppm	<5	Nickel	%	<0.001
Cobalt	ppm	7	Phosphorus	%	0.03
Mercury	ppm	<1	Antimony	%	0.119
Lanthanum	ppm	0.9	Scandium	ppm	<5
Lithium	ppm	<1	Tin	ppm	350
Molybdenum	ppm	2	Strontium	ppm	30
Sodium	%	0.01	Titanium	%	0.12
Aluminium	%	2	Vanadium	ppm	20
Arsenic	%	0.194	Tungsten	ppm	<50
Barium	ppm	90	Yttrium	ppm	12
Calcium	%	0.2	Silicon Dioxide	%	19.7

13.3.2 MINERALOGICAL TEST WORK

SGS Lakefield in Canada completed a mineralogical characterisation and silver deportment study on the Mangazeisky sample prepared and sent by SGS Vostok. The mineralogical investigation and the key results are summarised in this section. Further details can be found in Appendix A.

The mineralogical characterization of the sample was carried out with heavy liquid separation (HLS) followed by QEMSCAN® and chemical analysis.

The purpose of the mineralogical characterization was to determine the overall mineral assemblage and the liberation and association of the silver minerals.

SAMPLE PREPARATION

The sample was stage-crushed to 150 µm. Sub-samples were riffled for assays by ICP, whole rock analysis, lead, zinc, sulphur, silver, and for mineralogical analysis. The remainder of the sample was screened at 25 µm to generate a +25 µm and a -25 µm fraction. Both fractions were subjected to HLS at 3.1 specific gravity. The sink products were submitted for whole rock analysis by ICP, zinc, lead, and barium by x-ray fluorescence (XRF), sulphur by Leco, and silver by fire assay. A micro-riffled sub-sample from the +25 µm float fraction was submitted for silver analysis.

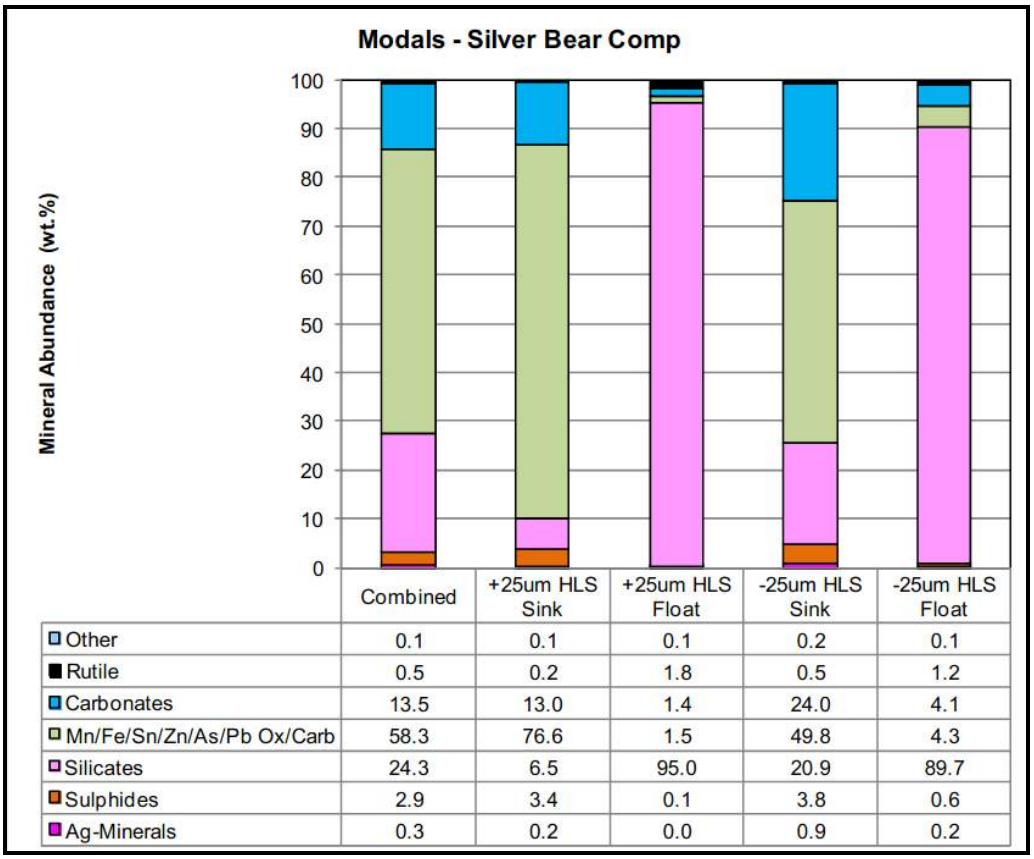
Modal Mineralogy

The mineralogical characterization indicated that the sample consists mainly of manganese-iron oxides/carbonates (mainly goethite and hematite and account for 58% total), followed by quartz (15%), siderite (14%), and muscovite (6%), and traces of K-feldspar, biotite, chlorite, clays, amphibole, zinc/manganese oxide, rutile, and other minerals.

It was also indicated that the silver minerals include silver-chlorides, silver sulphide, silver-rich tetrahedrite, and silver-lead-sulfosalts. The other sulphides included trace amounts of pyrite, chalcopyrite, stannite, covellite/chalcocite, sphalerite, galena and arsenopyrite.

A summary of the modal mineralogy is shown in Figure 13.6.

Figure 13.6 Summary of Modal Mineralogy



	Combined	+25um HLS Sink	+25um HLS Float	-25um HLS Sink	-25um HLS Float
Other	0.1	0.1	0.1	0.2	0.1
Rutile	0.5	0.2	1.8	0.5	1.2
Carbonates	13.5	13.0	1.4	24.0	4.1
Mn/Fe/Sn/Zn/As/Pb Ox/Carb	58.3	76.6	1.5	49.8	4.3
Silicates	24.3	6.5	95.0	20.9	89.7
Sulphides	2.9	3.4	0.1	3.8	0.6
Ag-Minerals	0.3	0.2	0.0	0.9	0.2

The x-ray diffraction (XRD) analysis indicated that the sample consists mainly of quartz, goethite, hematite, and mica, minor galena, K-feldspar, siderite, montmorillonite, illite, hausmannite, ferrihydrite, and akaganeite, and traces of sphalerite, ramsdellite, bixbyite, and magnetite.

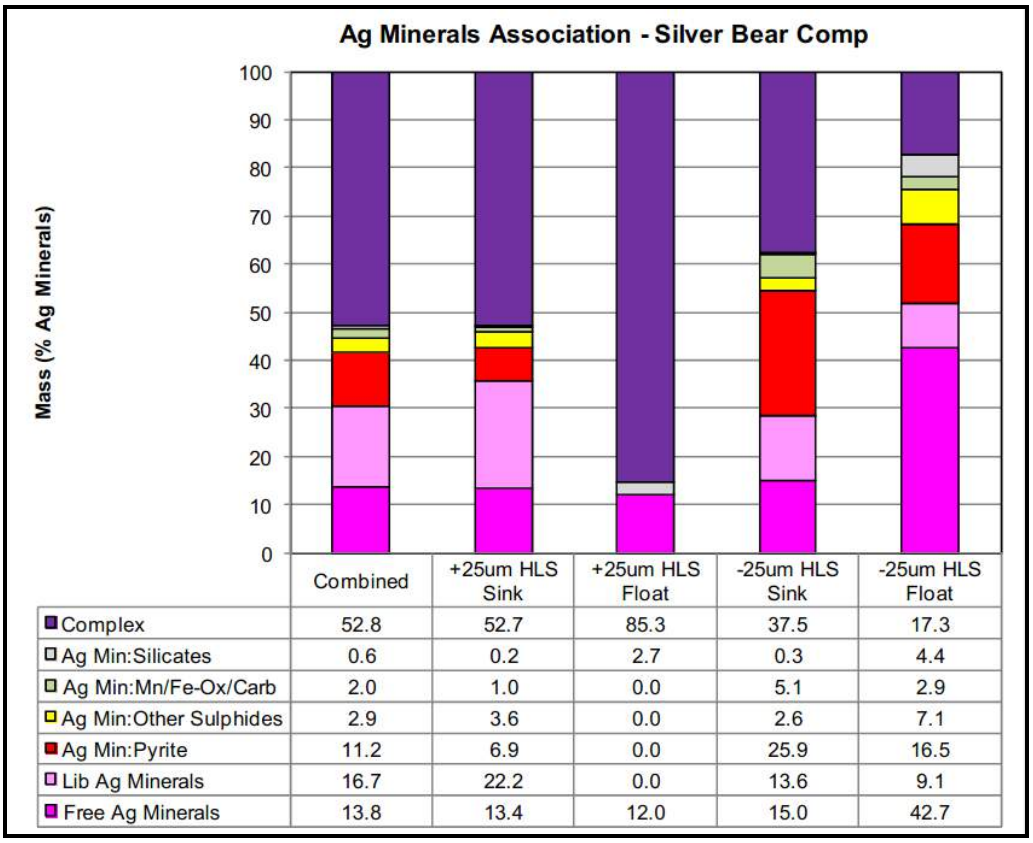
Liberation, Association and Exposure of Silver Minerals

The liberation and association of silver minerals is shown in Figure 13.7, which illustrates that they are poorly liberated (at approximately 31%). A significant amount occurs as complex particles (53% as ternary and quaternary middlings), associations with pyrite (11%), various sulphides (3%), and carbonates/oxides and sulfosalts (2%).

Liberation is poor in both the +25 µm sink and float fractions at 36% and 12%, respectively. Liberation is 29% and 52% in the -25 µm sink and float, respectively. Note that the -25 µm float fraction accounts for only 0.3% of the total mass of the sample. Therefore, its contribution to the overall liberation is negligible.

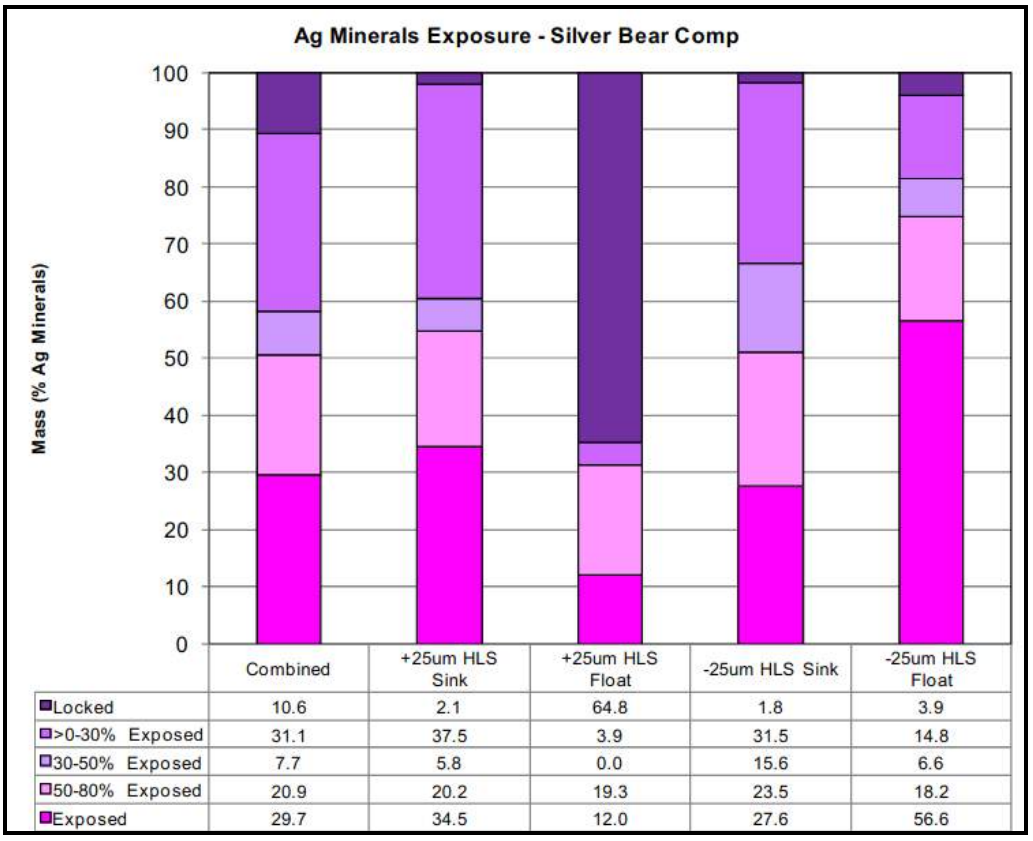
The exposure of silver minerals is shown in Figure 13.8. For the overall sample, the silver minerals that are well exposed (greater than 30% exposure surface) account for 59%, moderately exposed (greater than 0 – less than 30%) for 31%, and locked for 11%. The exposure of silver minerals is similar in the +25 µm sink fraction (61% well exposed, 37% moderately exposed and 2% locked), and -25 µm sink fraction (66%, 32% and 2%, respectively). However, 65% of the silver minerals are not exposed in the -25 µm float fraction.

Figure 13.7 Liberation and Association of Silver Minerals



	Combined	+25um HLS Sink	+25um HLS Float	-25um HLS Sink	-25um HLS Float
Complex	52.8	52.7	85.3	37.5	17.3
Ag Min:Silicates	0.6	0.2	2.7	0.3	4.4
Ag Min:Mn/Fe-Ox/Carb	2.0	1.0	0.0	5.1	2.9
Ag Min:Other Sulphides	2.9	3.6	0.0	2.6	7.1
Ag Min:Pyrite	11.2	6.9	0.0	25.9	16.5
Lib Ag Minerals	16.7	22.2	0.0	13.6	9.1
Free Ag Minerals	13.8	13.4	12.0	15.0	42.7

Figure 13.8 Exposure of Silver Minerals



	Combined	+25um HLS Sink	+25um HLS Float	-25um HLS Sink	-25um HLS Float
Locked	10.6	2.1	64.8	1.8	3.9
>0-30% Exposed	31.1	37.5	3.9	31.5	14.8
30-50% Exposed	7.7	5.8	0.0	15.6	6.6
50-80% Exposed	20.9	20.2	19.3	23.5	18.2
Exposed	29.7	34.5	12.0	27.6	56.6

Based on the mineralogical characterization the following primary conclusions can be drawn.

- The SGS Vostok sample consisted mainly of manganese-iron oxides/carbonates (58%), followed by quartz (15%), siderite (14%), and muscovite (6%), and traces of K-feldspar, biotite, chlorite, clays, amphibole, zinc/manganese-oxide, rutile, and other minerals.
- Silver minerals included chlorargyrite, silver sulphide and silver-antimony-lead sulfosalts, silver-rich tetrahedrite, and silver-lead sulfosalts.
- Silver minerals are poorly liberated (approximately 31%), and mainly occur as various middling particles and complex particles with the gangue minerals in the sample. Liberation is poor in both the +25 µm sink and float fractions at 36% and 12%, respectively, and 29% in the -25 µm sink fraction.
- Silver minerals with well exposed surface area (greater than 30% exposure) account for 59%, moderately exposed (greater than 0 – less than 30%) for 31% and locked for 11%. Therefore, chemical recovery of silver at 100% passing 150 µm grind would likely yield better grades and recoveries.

Diagnostic Leach Tests

Silver diagnostic leach tests were conducted on a representative sample at a grind of P_{80} = 75 µm according to two different standard procedures. The summary of the results are shown in Table 13.13. Details of the test procedures can be seen in Appendix A.

Table 13.13 Summary of Diagnostic Leach Results

Silver Deportment	Distribution (%)	
	Method 1	Method 2
Free Silver	0.10	0.10
Open Intergrowths	89.60	89.70
Amenable to Cyanidation	89.70	89.80
Locked in/Associated with Sulphides	3.39	5.00
Associated With Rock Forming Minerals	1.41	1.00
Other	5.50	4.30

The results indicates that approximately 90% of the silver is amenable to cyanidation at a P_{80} = 75 µm grind.

13.3.3 METALLURGICAL TEST WORK

COMMINUTION TEST WORK

SGS Vostok and TOMS completed comminution test work on a composite sample in order to determine the parameters required for the design of the comminution circuit.

The following tests were conducted:

- Bond Ball Mill Work Index (BW_i) (SGS Vostok)
- Bond Abrasion Index (BA_i) (SGS Vostok)
- Crusher Work Index (CW_i) (TOMS).

A summary of the comminution test results are shown in Table 13.14.

Table 13.14 Summary of Comminution Results

Description	Unit	Test 1	Test 2
Bond Ball Mill Index			
Limiting Screen Size	µm	75	106
F_{80}	µm	2,540	2,540
P_{80}	µm	58	84
Work Index, BW_i	kWh/t	14.6	13.9
Bond Abrasion Index			
Abrasion Index	g	0.0392	
Crusher Work Index			
Mean	kWh/t	4.41	
Standard Deviation		1.86	
Minimum		1.68	
Maximum		7.73	

The Bond Ball Mill Indices (BW_i = 13.9 kWh/t and 14.6 kWh/t) indicate that the ore is categorised among the moderately hard range of the BW_i scale. This means that fine grinding by ball milling could be an energy intensive operation.

The abrasion index of the Mangazeisky ore is moderately low and indicates a low wear rate of mill liners and media compared to similar silver projects.

The crusher work index is also categorised as moderately low, and indicates a lower crushing power requirement.

Gravity Concentration Tests

Effect of Grind on Gravity Concentration

The gravity concentration tests were carried out on a representative sample using the batch laboratory-scale Knelson KC-MD3 (Figure 13.9) test unit and Mozley Laboratory Separator. The Knelson concentrator acted as a primary gravity concentrator while the Mozely concentrator was used to clean the primary concentrates. These gravity tests were conducted at various grinds (P_{80} = 212 µm, 106 µm, 75 µm, and 53 µm) in order to estimate the silver recovery at various grind sizes. A summary of the effect of grind gravity test results are shown in Table 13.15.

Figure 13.9 Knelson KC-MD3 Laboratory Batch Test Unit

Table 13.15 Effect of Grind on Gravity Concentration

Feed Size (P_{80} – µm)	Product	Mass (%)	Ag (g/t)	Ag Recovery (%)
212	Concentrate	0.12	6,960	0.77
	Middlings	1.15	3,890	3.97
	Tailings	98.72	1,090	95.26
106	Concentrate	0.12	6,910	0.75
	Middlings	1.01	5,580	4.95
	Tailings	98.86	1,090	94.30
75	Concentrate	0.14	9,220	1.15
	Middlings	1.06	7,650	7.32
	Tailings	98.8	1,030	91.53
53	Concentrate	0.12	11,400	1.19
	Middlings	1.01	8,210	7.11
	Tailings	98.87	1,080	91.70

The results shown in Table 13.15 indicate that silver recovery gradually increased with finer grind up to P_{80} = 75 µm grind size and remained unchanged with further grinding.

Gravity Recoverable Silver Test

Gravity recoverable silver (GRS) tests were conducted using a centrifugal Knelson concentrator (KC-MD3) with stage grinding. The stage grinds included:

- Stage 1 – P_{80} = 850 µm
- Stage 2 – P_{80} = 150 µm
- Stage 3 – P_{80} = 75 µm.

The GRS test was conducted on 20 kg ore charge with wash water added at 3.5 l/min. The results of the GRS tests are summarised in Table 13.16.

Table 13.16 Summary of GRS test results

Feed Size (P_{80} – µm)	Product	Mass (%)	Ag (g/t)	Ag Recovery (%)
850	Concentrate 1	0.55	2,460	1.13
150	Concentrate 2	0.67	5,710	3.20
75	Concentrate 3	0.59	7,700	3.78
N/A	Total Concentrate	1.81	5,368	8.10
N/A	Tailings	98.19	1,120	91.90

The results presented in Table 13.16 indicate that the overall gravity recoverable silver is approximately 8%. This is much lower than the gravity silver recovery reported in the historical test work completed by both laboratories.

These results suggest that gravity concentration as a pre-concentration step is not an attractive flowsheet solution based on the gravity silver recoveries. However, this aspect should be evaluated by taking in to account the overall silver recoveries and the capital and operating costs. Tetra Tech subsequently completed a high-level economic trade-off study, in order to evaluate the gravity concentration against ROM leach.

Bulk Gravity Concentration Tests

Bulk gravity concentration tests were conducted in a laboratory Knelson concentrator, at a grind of P_{80} = 75 µm, in order to prepare a feed sample for the leach tests.

The gravity concentrates were treated by intensive cyanidation in order to simulate the intensive leach reactor.

The gravity tails were then leached according to standard bottle roll cyanide leach procedures, with the objective of optimising the leach parameters (i.e. grind size, cyanide strength, pulp density and the addition of leach aids).

The summary of the bulk gravity concentration results are shown in Table 13.17.

Table 13.17 Bulk Gravity Concentration Results

Product	Mass (%)	Ag (g/t)	Ag Recovery (%)
Concentrate	1.1	8,550.0	7.9
Tailings	98.9	1,110.0	92.1
Feed	100.0	1,191.8	100.0

Intensive Cyanidation of Gravity Concentrates

The gravity concentrates were tested for intensive cyanidation at two different sets of conditions as shown in Table 13.18.

Table 13.18 Test Condition for Intensive Cyanidation

Parameter	Unit	Test 1	Test 2
Cyanide Concentration	g/l	5	15
pH	-	10.5-11.0	10.5-11.0
Grind Size (P_{80})	µm	75	25
Leach Aid ($PbNO_3$)	g/t	-	150
Pre-aeration	h	8	8
Leach Time	h	96	96

Note: $PbNO_3$ – lead nitrate

The summary of the silver recovery to solution are shown in Table 13.19.

Table 13.19 Silver Recovery to Solution by Intensive Cyanidation

Test Reference	Recovery (%)
Test 1	72.9
Test 2	98.0

The results indicate that a concentrate regrind of P_{80} = 25 µm, a higher cyanide concentration of 15 g/l and the addition of leach aid has provided excellent silver extraction into solution.

These results clearly indicate that intensive cyanidation of gravity concentrates is an effective way of processing silver sulphide minerals which normally show very slow leach kinetics.

Cyanidation of Gravity Tails

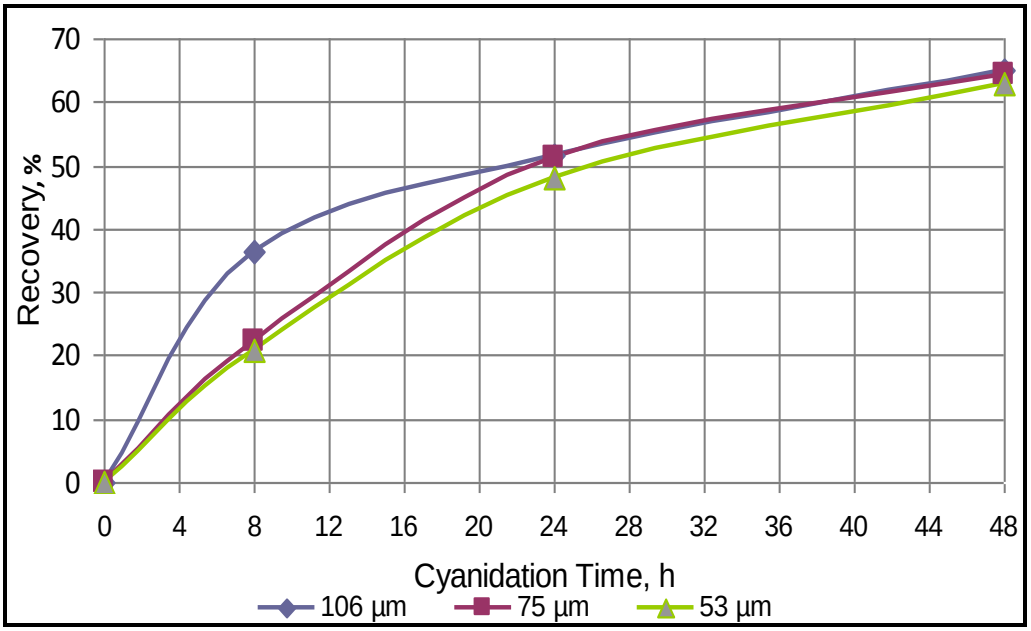
Effect of Grind Tests

Batch dissolution tests were carried out on the gravity tailings in order to evaluate the effect of grind on gold dissolution. Three grinds were tested: P_{80} = 106 µm, 75 µm, and 53 µm.

The effect of grind leaches were run at standard conditions (i.e. 40% solids, 8 hour pre-conditioning period, 1.0 g/l sodium cyanide addition, and a pH range of 10.5 to 11.0). The slurries were leached for 48 hours at an ambient temperature.

A summary of the effect of grind leach tests are shown in Figure 13.10.

Figure 13.10 Effect of Grind Leach Test Results



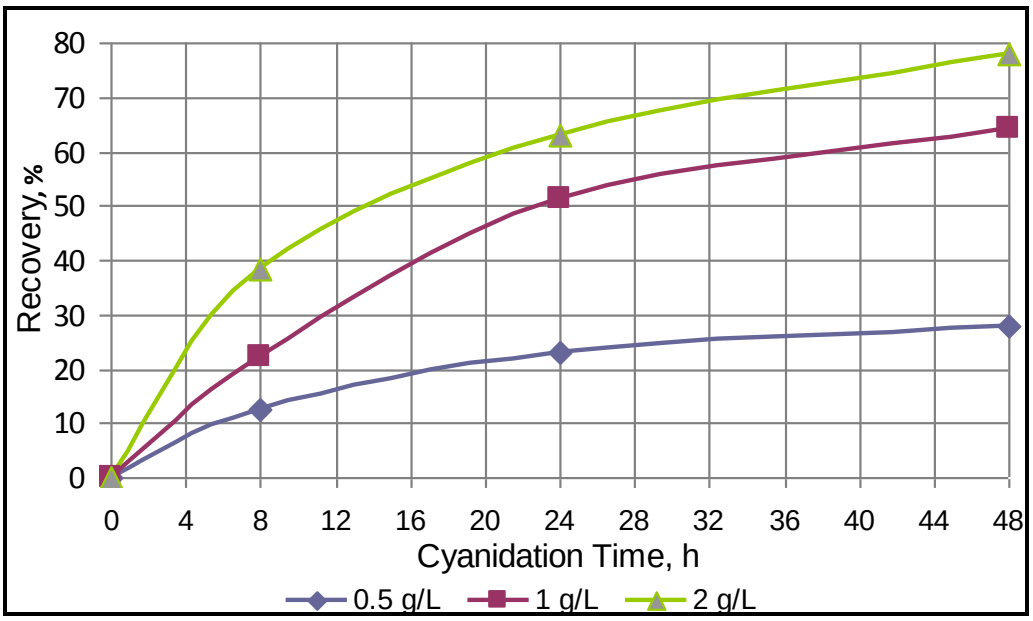
The results indicate that leach dissolution after 48 hours is almost identical for all three tests, but the leach kinetics are better for the coarser grind (P_{80} = 106 µm) than the other two tests. This trend is unusual, but this may have happened as a result of cyanide insufficiency at finer grinds (i.e. at finer grinds liberated sulphide minerals might have competed with silver particles for cyanide).

The results also show that the leach did not reach equilibrium after 48 hours and the leach extraction curve continued to increase. In this context, firm conclusions about the effect of grind on the leach performance cannot be made based on these results. Thus, a grind size of P_{80} = 75 µm was selected for the rest of the test work based on the diagnostic leach and wider industrial practice.

Effect of Cyanide Addition

The effect of cyanide addition test was used to establish the optimum cyanide addition required for each sample in order to obtain maximum silver dissolution. The test was conducted at a P_{80} = 75 µm grind using three different cyanide additions (0.5 g/l, 1.0 g/l, and 2.0 g/l). The summary of the results are presented in Figure 13.11.

Figure 13.11 Effect of Cyanide Addition Leach Results

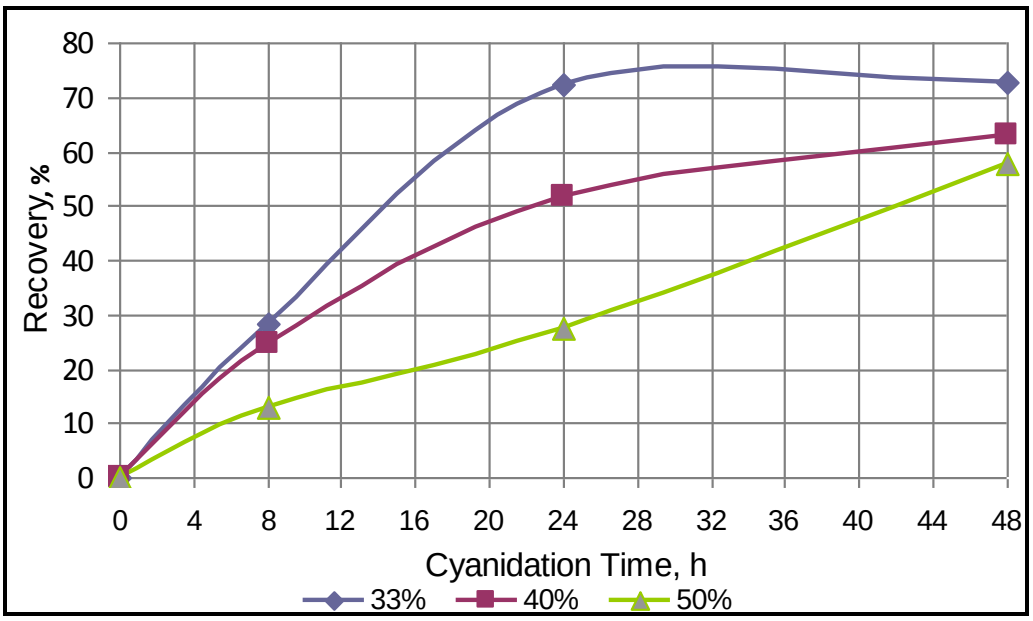


The results clearly indicate that both leach recovery and kinetics are better with higher cyanide concentration, but the leach has not stabilised after 48 hours. Based on these results, the cyanide concentration for gravity tails leach was selected as 2 g/l; however, this initial selection needs to be confirmed with further tests.

Effect of Pulp Density

Three different pulp densities (33%, 40%, and 50%) were tested in order to establish the optimal pulp density for leaching. All other conditions remained as per standard conditions (i.e. P_{80} = 75 µm grind, 8 hour pre-conditioning period, 1.0 g/l sodium cyanide addition, and a pH range of 10.5 to 11.0). The slurries were leached for 48 hours at an ambient temperature. A summary of the results is shown in Figure 13.12.

Figure 13.12 Effect of Pulp Density Results



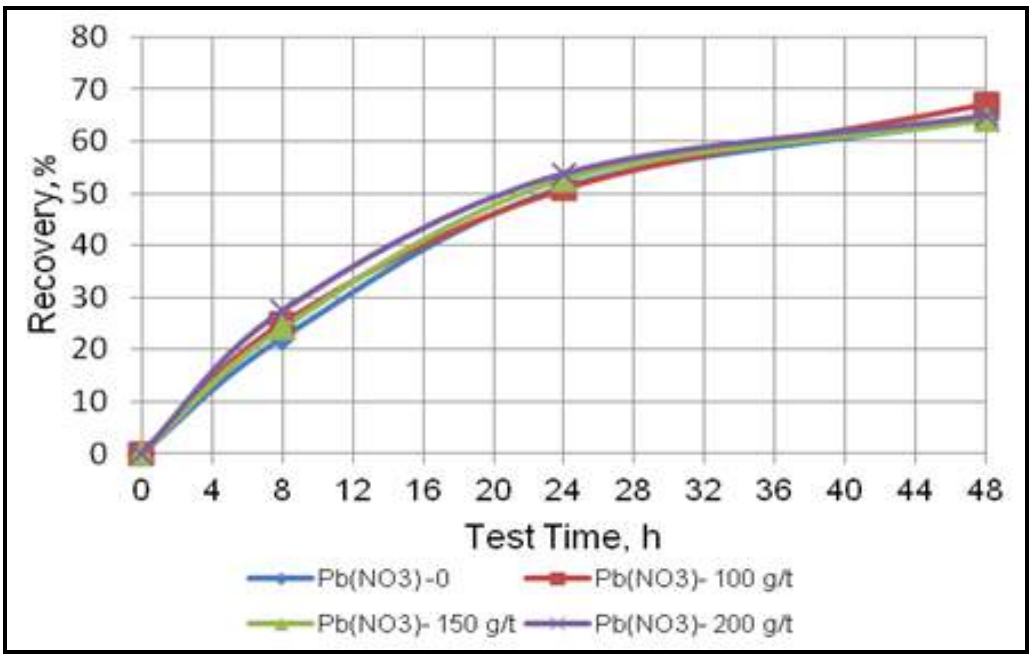
The results clearly indicate that both recovery and leach kinetics are better with lower pulp densities; however, the incremental recovery as a result of lower pulp density becomes smaller as the leach progresses.

Pulp density and cyanide concentration are interrelated to each other, and therefore no conclusions about pulp density can be made at this point.

Effect of Leach Aid (Silver Nitrate) Addition

The effect of leach aid test was conducted at standard conditions (i.e. P_{80} = 75 µm grind, 8 hour pre-conditioning period, 1.0 g/l sodium cyanide addition, 40% pulp density, and a pH range of 10.5 to 11.0) but with the addition of silver nitrate [$Pb(NO_3)_2$] during the pre-conditioning period as a leach aid. Three different rates (100 g/t, 150 g/t, and 200 g/t) of silver nitrate additions were tested along with a baseline test.

Figure 13.13 Effect of Silver Nitrate Addition

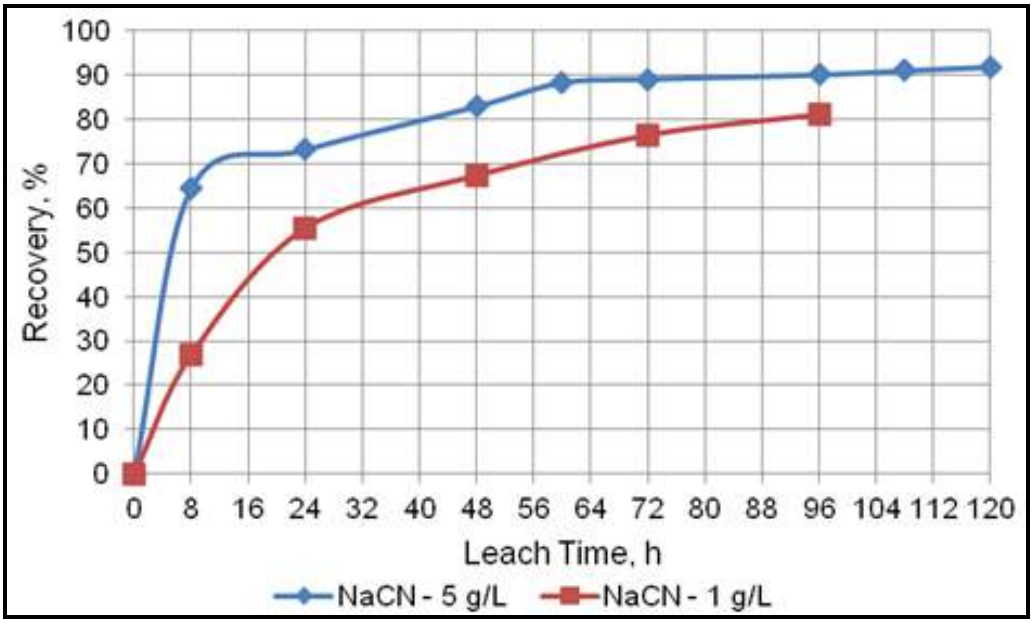


The results indicate that the silver nitrate addition has not influenced the overall recovery or leach kinetics.

Effect of Time/Leach Kinetics

The effect of time test was conducted at two different cyanide concentrations (1 g/l and 5 g/l) for 96 hours and 120 hours in order to determine the optimal leach time. All other leach parameters remained as standard (i.e. P_{80} = 75 µm grind, 8 hour pre-conditioning period, 40% pulp density, and a pH range of 10.5 to 11.0). The summary of the leach kinetics results are shown in Figure 13.14.

Figure 13.14 Effect of Time Test/Leach Kinetics



The effect of time results indicate that the leach performed at 5 g/l cyanide concentration reached equilibrium at approximately 56 to 72 h leach time. The final silver recovery to solution (after 120 hour leach) was reported at 91.85%.

The leach performed at 1 g/l of cyanide concentration had not reached equilibrium even after 96 hours of leach time. A silver recovery of 81.1% was achieved after this time.

In summary, the following observations can be made from the cyanidation of gravity tails:

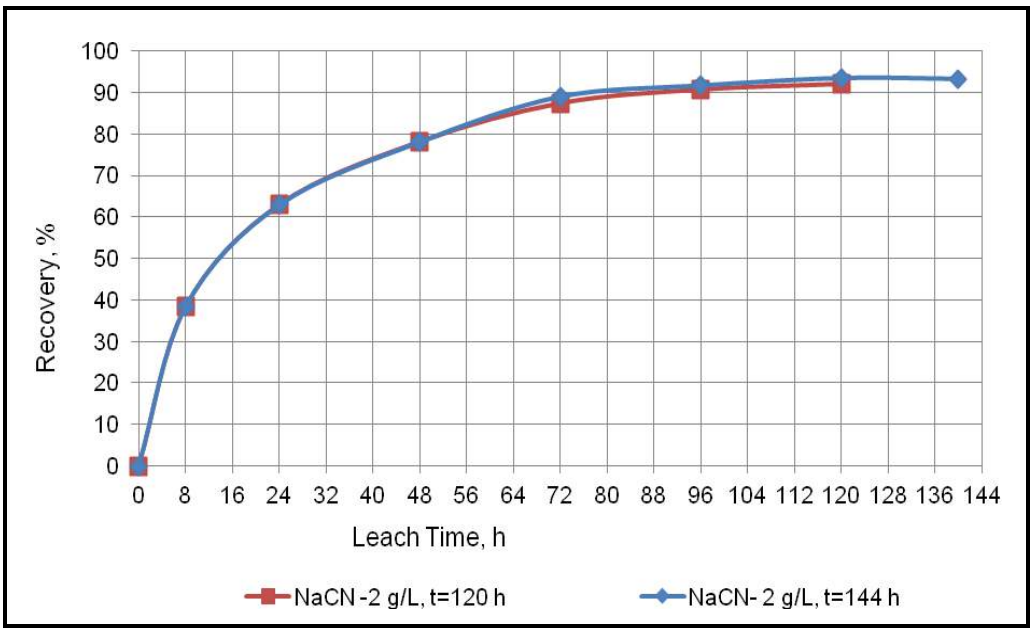
- The leach is not significantly influenced by the tested particle size distributions.
- The cyanide concentration has significant influence on the leach kinetics and silver recoveries. It is understood that 1 g/l is not sufficient and 5 g/l is more than sufficient. It appears 2 g/l is adequate cyanide concentration for leaching gravity tails.
- Leach kinetics is very slow and 48 hours is not sufficient in order for the leach to reach equilibrium.
- Pulp density has influenced the leach kinetics but the recovery after 48 hours of leach is not significantly different for the tested pulp densities.
- Silver nitrate addition has not influenced the leach kinetics or recoveries.

Re-leaching of Leach Tails

Tetra Tech has proposed to conduct additional leach tests, where the dry solids residue from the 48 hour leach can be re-pulped and leached for extended leach time in order to reach higher silver recoveries. These tests were conducted from the sample retained from 2 g/l cyanide leach, all leach conditions other than retention time were as per standard (i.e. P_{80} = 75 µm grind, cyanide concentration 2 g/l, 40% pulp density, and a pH range of 10.5 to 11.0). Two leach tests were conducted with increased leach retention time of an additional 72 hours and 96 hours. The overall leach retention time of these tests are equivalent to 120 hours and 144 h as a result of the additional leach time. The summary of the leach kinetics for increased retention time is shown in Figure 13.15.

The results shown in Figure 13.15 clearly indicate that the leach kinetics reach equilibrium at around 72 to 96 hours overall leach time. The silver recoveries for 72 hour and 96 hour leach were reported as 89%, 87.3% and 91.7%, 90.6% respectively.

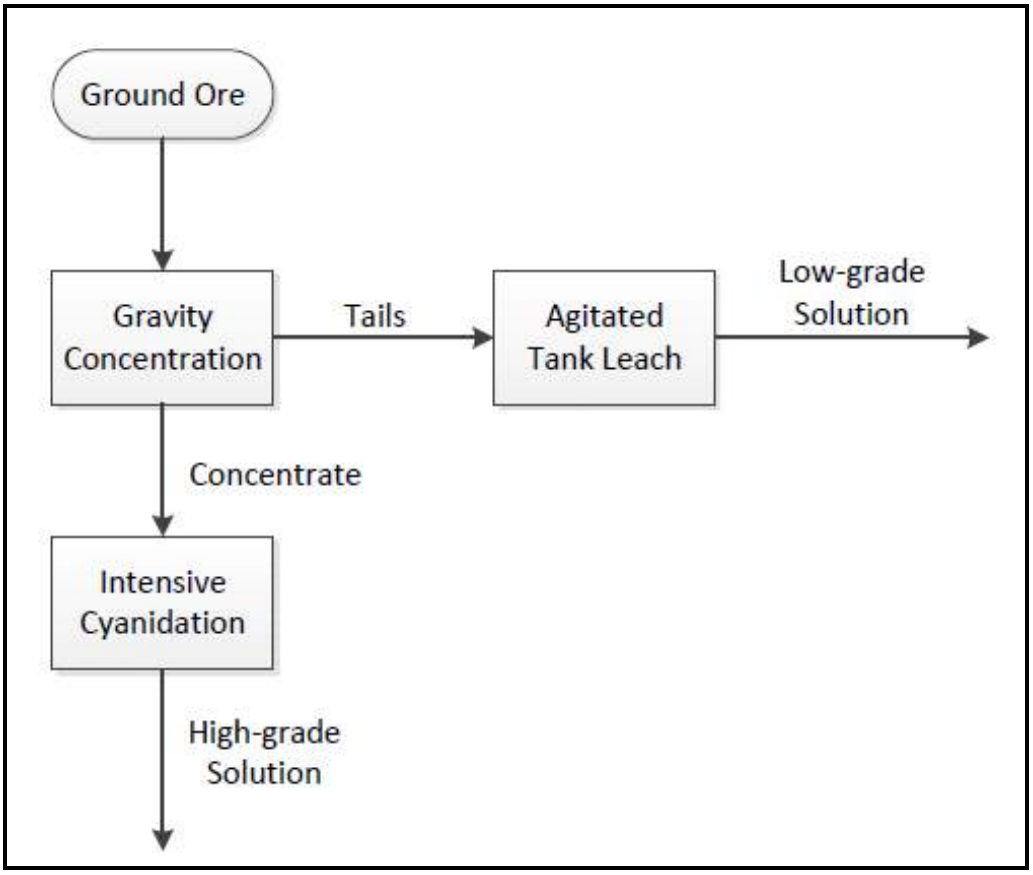
Figure 13.15 Leach Kinetics for Re-leaching the Leach Tails



Overall Recovery of Gravity Plus Leach Based Option

The overall silver recovery of the gravity and leach based option was estimated based on a conceptual process block flow diagram as shown in Figure 13.16.

Figure 13.16 Conceptual Block Flow Diagram for Gravity and Leach Based Option



The total silver reported to both high grade and low grade solutions were estimated based on the test work results shown in this report and shown in Table 13.20.

Table 13.20 Overall Silver Recovery Calculation for Gravity Based Flowsheet

Gravity Concentrates	Option A	Option B
Silver Recovery to Gravity Concentrate (%)	7.9	7.9
Silver Dissolution at Intensive Cyanidation (%)	98.0	98.0
Silver Reported to Intensive Leach Solution (%)	7.7	7.7
Gravity Tails		
Silver Recovery to Gravity Tails (%)	92.1	92.1
Standard Leach Silver Dissolution* (%)	88.2	91.2
Silver Recovery to Standard Leach Solution (%)	81.2	84.0
Overall Recovery		
Overall Silver Recovery to Solution	88.9	91.7

Note: *Standard leach retention times of 72 hours and 96 hours assumed for Options A and B, respectively.

Flotation Test Work

The flotation test work included initial optimisation tests followed by locked cycle flotation test work. In addition bulk flotation tests were conducted in order to prepare a sample for intensive cyanidation tests. All of the flotation tests were conducted in a laboratory bench scale Denver flotation cell (Figure 13.17).

Figure 13.17 Laboratory Flotation Cell

Grind Size Optimisation

Effect of grind and reagent scouting tests were conducted as the primary step in flotation in order to establish the optimal flotation conditions. A summary of the effect of grind tests are shown in Table 13.21. Full details of the test conditions can be seen in the Appendix A.

Table 13.21 Effect of Grind Flotation Test Results

Grind	Product	Mass (%)	Assay			Recovery		
			Ag (g/t)	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Zn (%)
P_{80} = 53 µm	Concentrate	1.67	37,900	45.75	14.5	59.16	40.48	13.98
	Middling 1	0.66	14,300	9.54	3.1	8.86	3.35	1.19
	Middling 2	4.67	3,080	2.5	1.63	13.47	6.2	4.41
	Scavenger Concentration 1	2.29	1,270	1.68	1.42	2.72	2.04	1.88
	Scavenger Concentration 2	2.76	792	1.51	1.42	2.04	2.21	2.26
	Tailings	87.95	167	0.98	1.5	13.75	45.73	76.28
P_{80} = 44 µm	Concentrate	1.98	40,300	45.98	14	71.11	45.27	15.03
	Middling 1	0.64	14,600	7.5	2.28	8.33	2.39	0.79
	Middling 2	6.93	1,120	1.41	1.53	6.94	4.87	5.77
	Scavenger Concentration	3.84	568	1.35	1.49	1.95	2.58	3.11
	Tailings	86.61	151	1.04	1.6	11.68	44.89	75.3

The results shown in Table 13.21 indicate that higher silver recoveries can be expected at a finer grind size of P_{80} = 44 µm. Based on this result, this grind size was considered for the locked cycle test work.

Locked Cycle Test Work

The summary of the locked cycle flotation test results are shown in Table 13.22.

Table 13.22 Summary of Locked Cycle Test Results (Selective Flotation)

Product	Mass (%)	Assay			Recovery		
		Ag (g/t)	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Zn (%)
Concentrate	2.82	37,091	36.67	10.66	85.74	49.33	16.47
Tailings	97.18	179	1.09	1.57	14.26	50.67	83.53
Feed	100.00	1,220	2.10	1.83	100.00	100.00	100.00

The locked cycle test work results presented in Table 13.22 indicate that a flotation concentrate at 2.8% mass pull could potentially recover 85.7% of silver and 49.3% of lead.

It is important to note that not only lead and zinc recoveries are poor, but also the concentrates grades are much lower than required the minimum specifications for saleable lead and zinc concentrates.

Bulk Flotation Tests

Bulk flotation tests were conducted at a grind of P_{80} = 44 µm in order to prepare a feed sample for the intensive cyanidation tests.

The summary of the bulk flotation test results are shown in Table 13.23.

Table 13.23 Summary of Bulk Flotation Results

Product	Mass (%)	Ag (g/t)	Ag Recovery (%)
Concentrate	2.1	40,100.0	74.5
Middlings	1.1	7,378.1	6.6
Tailings	96.9	223.0	18.9
Feed	100.0	1,141.5	100.0

Intensive Cyanidation of Flotation Concentrates

The flotation concentrates were tested for intensive cyanidation at two different sets of conditions (same conditions as intensive cyanidation of gravity concentrates) as shown in Table 13.18.

The summary of the silver recovery to solution are shown in Table 13.24.

Table 13.24 Silver Recovery to Solution by Intensive Cyanidation

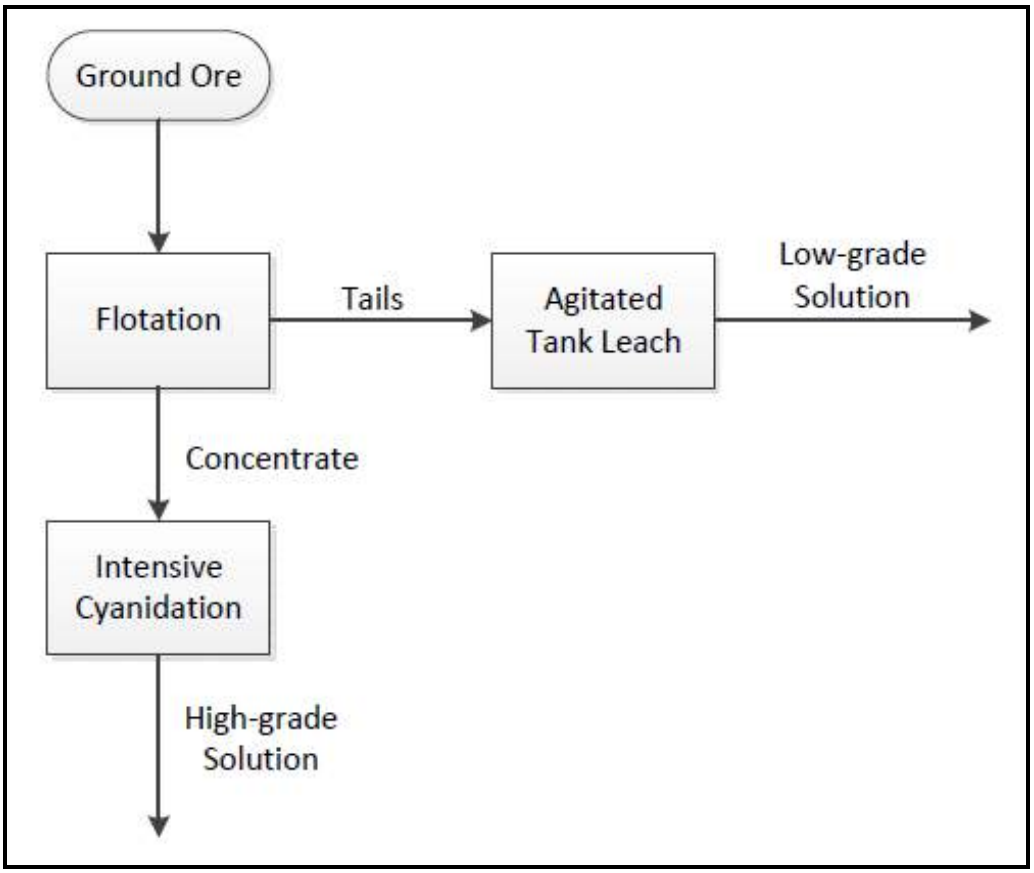
Test Reference	Recovery (%)
Test 1	83.4
Test 2	87.2

The results indicate that a concentrate regrind to P_{80} = 25 µm, a higher cyanide concentration of 15 g/l and the addition of leach aid has not significantly influenced the silver recoveries into solution.

Overall Recovery by Flotation

The overall silver recovery of the flotation plus leach based option was estimated based on a conceptual process block flow diagram as shown in Figure 13.18.

Figure 13.18 Conceptual Block Flow Diagram for Flotation Option



It is important to note that the flotation tails has not been tested for their leach behaviour. However, the leach recoveries of gravity tailings were assumed for the flotation tails, in order to make comparison between processing options.

The total silver reported to both high-grade and low-grade solutions was estimated based on the test work results shown in this report and as shown in Table 13.25.

Table 13.25 Overall Silver Recovery of Flotation Based Option

	Calculation (%)
Flotation Concentrates	
Silver Recovery to Flotation Concentrate	74.5
Silver Dissolution on Intensive Cyanidation	87.2
Silver Reported to Intensive Leach Solution	64.9
Gravity Tails	
Silver Recovery to Gravity Tails	25.6
Standard Leach Silver Dissolution	88.2
Silver Recovery to Standard Leach Solution	22.5
Overall Recovery	
Overall Silver Recovery to Solution	87.4

HEAP LEACH TESTS

Heap Leach Predictor Tests

Coarse ore bottle roll leach tests were conducted as heap leach predictor tests at three different crush sizes (P_{80} = 20 mm, 10 mm, and 5 mm). The summary of the results are shown in Table 13.26.

Table 13.26 Summary of Heap Leach Predictor Results

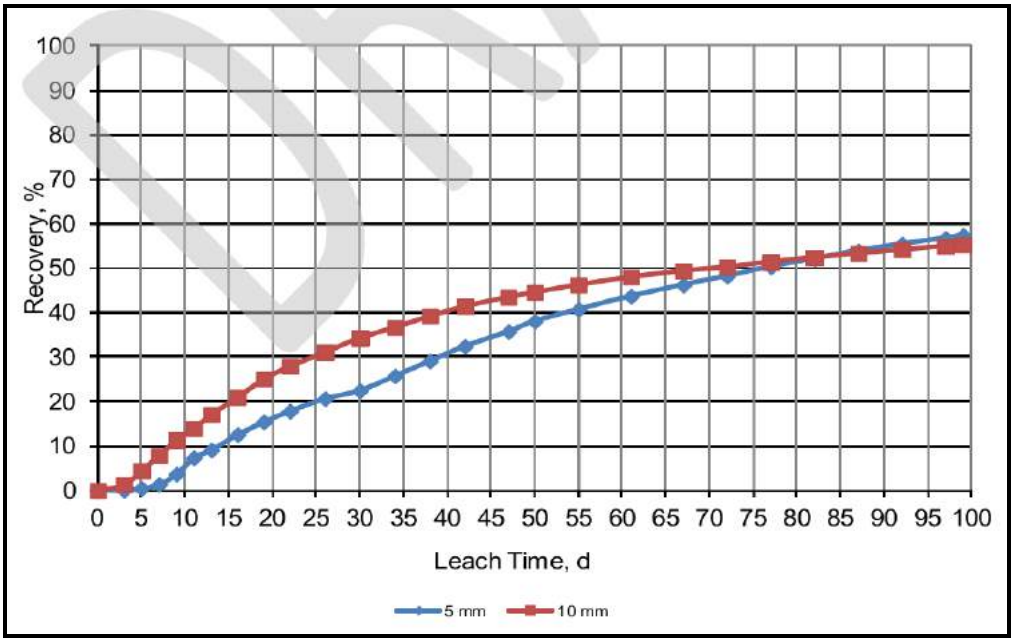
Size (P_{80}-mm)	Recovery (%)
20	39.12
10	47.83
5	61.19

The results clearly indicate that the leach recovery increases with finer particle size distribution. Based on this results, P_{80} = 10 mm and 5 mm sizes were selected for column heap leach test work.

Column Heap Leach Test

Colum heap leach tests were conducted in percolation columns with 110 mm diameter and 2,000 mm height according to standard procedure. Two different particle size distribution (P_{80} = 10 mm and 5 mm) were considered for this test based on the heap leach predictor tests. The solution flow rate was kept at 0.003 l/s/m^2 throughout the test. The cyanide concentration of the solution and the pH of the column were kept at 0.2% and 10.5 to 11.5. The column heap leach tests were conducted for an extended duration of 100 days as a result of slower leach kinetics. The summary of the results are shown in Figure 13.19. Further details can be seen in the Appendix A.

Figure 13.19 Heap Leach Test Results



The results indicated that the expected silver recovery for heap leach after 100 days is approximately 56.5% and 59.4% for the P_{80} = 10 mm and 5 mm grinds, respectively.

ROM Leach Tests

Preliminary ROM Leach Tests

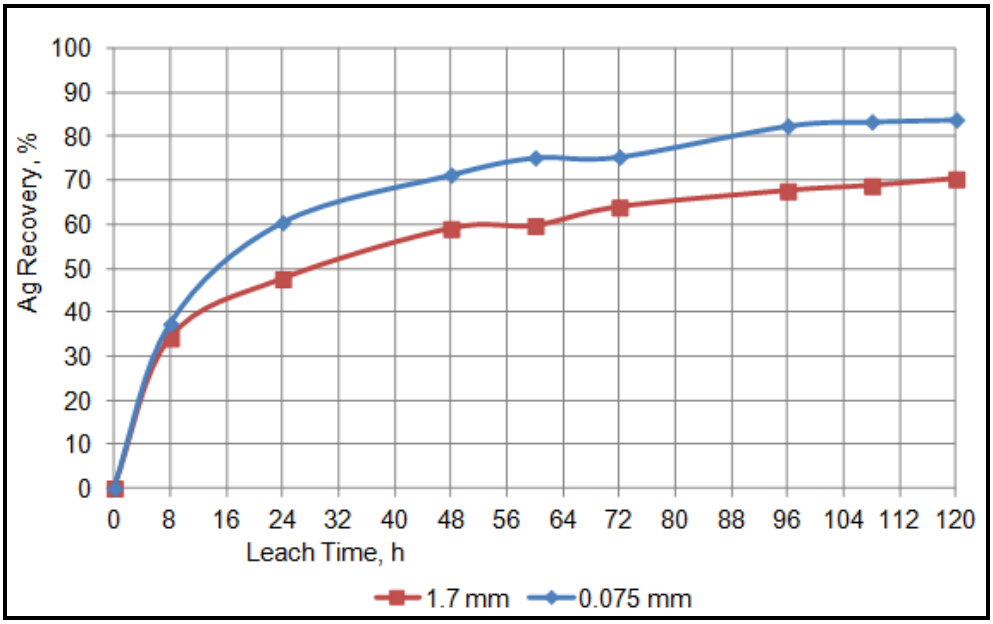
Preliminary ROM leach tests were conducted on two different particle size distributions (P_{80} = 1.7 mm and 75 µm). The tests were conducted according to the conditions shown in Table 13.27.

Table 13.27 ROM Leach Test Conditions

Parameter	Unit	Value
Cyanide Concentration	g/l	2
pH	-	10.5 to 11.0
Pre-aeration	h	8
Pulp Density	w/w %	33
Leach Time	h	120

The summary of the preliminary ROM leach results are shown in Figure 13.20.

Figure 13.20 Leach Kinetics of the Preliminary ROM Leach Test



The results indicate that leach kinetics reach equilibrium after 96 hours for both particle size distributions.

SGS Vostok has reported final silver recoveries of 70.2% and 83.7% for the 1.7 mm and 75 µm tests, respectively.

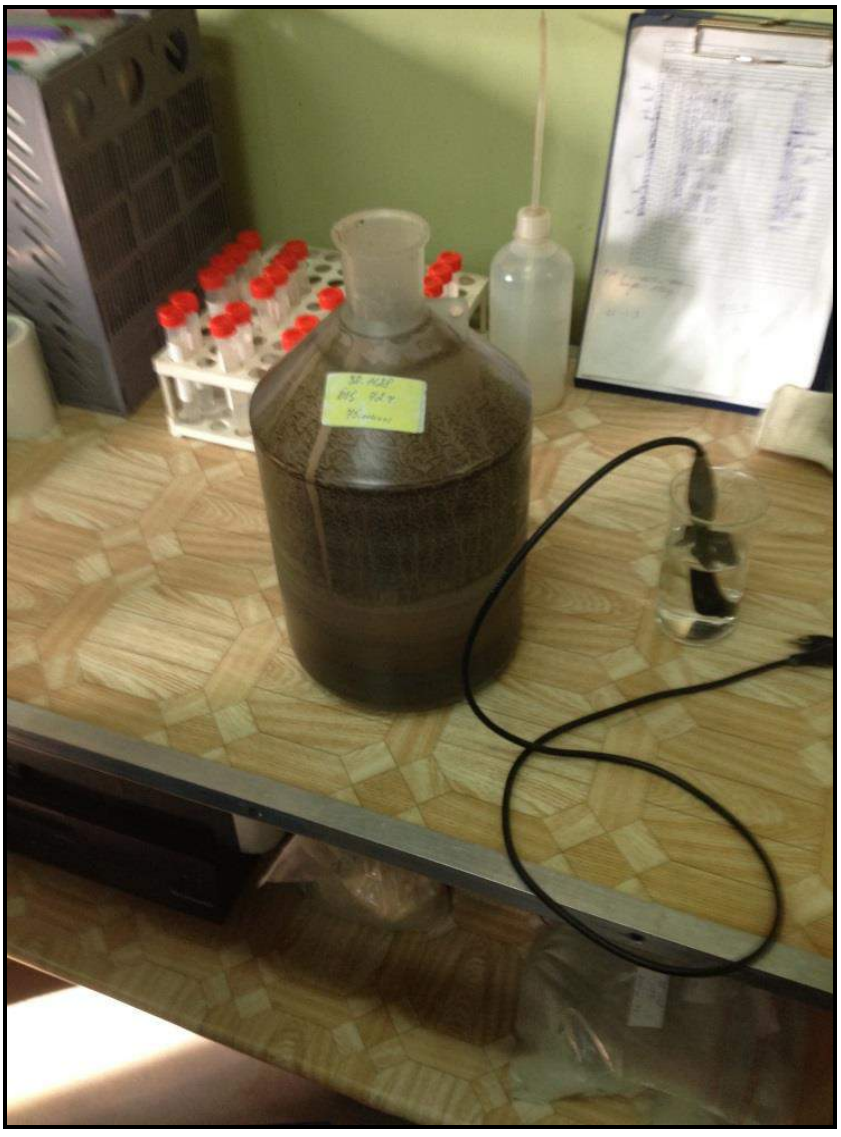
Reference Cyanide Leach Test

In addition to the preliminary ROM leach test, a third leach was conducted as a reference test during a site visit by the author to the SGS Vostok laboratories. A 2 l test bottle (Figure 13.21) was used in this test work and the test was conducted according to conditions shown in Table 13.28.

Table 13.28 Test Conditions for Reference Leach Test

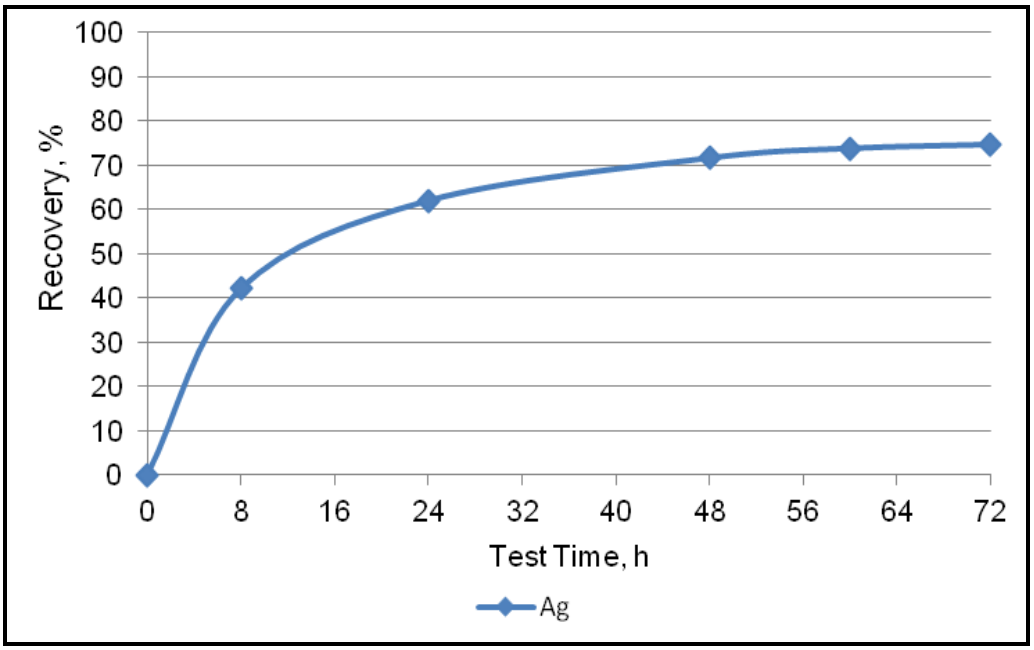
Parameter	Unit	Value
Pre-aeration	h	8
pH	-	10.5
Cyanide Concentration	g/l	2
Pulp Density	w/w %	40
Leach Time	h	72

Figure 13.21 Bottle Roll Cyanide Leach Test Sample Preparation



Summary of the reference leach test results are shown in Figure 13.22.

Figure 13.22 Reference Leach Test Results



SGS Vostok has reported the final silver recovery to solution as 74.8%.

The 74.8% recovery is comparable to the 72 hours recovery of the preliminary leach test of 75.2%.

These results, indicates that the optimal leach conditions for ROM ore and gravity tailings are not same despite the lower silver recovery to gravity concentrates. It appears some silver minerals and base metal sulphides that reported to the gravity concentrates have slowed the leach kinetics significantly during the ROM ore leach tests.

As a result of lower silver recoveries, a fresh set of ROM leach tests were conducted in order to optimise the leach conditions for the ROM ore.

ROM Leach Optimisation Tests

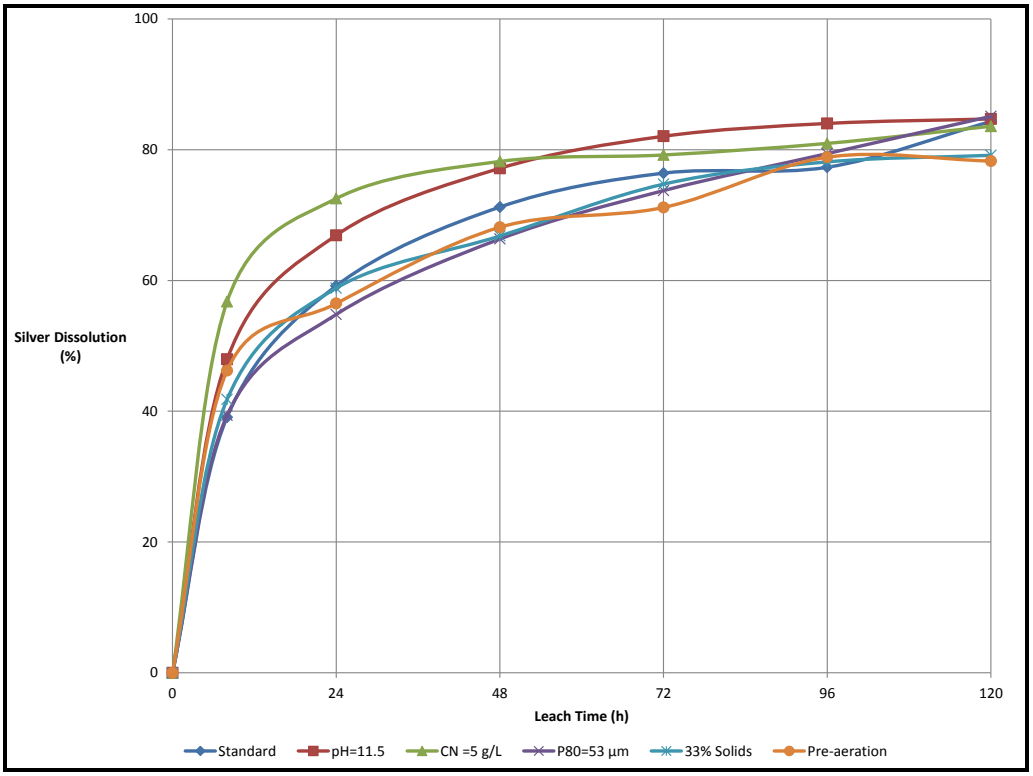
Six ROM leach tests were conducted in order to identify the leach parameters that increase the leach kinetics. The leach retention time for these tests was 120 hours. Other test conditions are shown in Table 13.29.

Table 13.29 Test Conditions for ROM Leach Optimisation Tests

Test	Reference	pH	Cyanide Concentration (g/l)	Grind (P_{80} -µm)	Pulp Density (w/w%)	Pre-aeration (h)	Leach Aid
1	Baseline	10.5	2	75	40	-	no
2	pH	11.5	2	75	40	-	no
3	Cyanide	10.5	5	75	40	-	no
4	Grind	10.5	2	53	40	-	no
5	Pulp Density	10.5	2	75	33	-	no
6	Pre-aeration	10.5	2	75	40	8	Peroxide

The summary of the leach kinetics curves for the ROM leach optimisation tests are shown in Figure 13.23.

Figure 13.23 Leach Kinetic Curves for the Optimisation Tests



It appears that a higher cyanide concentration and higher pH has raised the leach kinetics significantly compared to the other tests. This could be possibly as a result of higher availability of free cyanide at those conditions. This pattern is in line with the general expectation of increased leach kinetics at higher availability of cyanide.

Direct Electrowinning Test Work

Lab scale direct electrowinning test work was conducted by Electrometals LLC (2014) at the SGS Vostok premises with the following objectives:

- test the suitability of direct electro-winning technology for processing Mangazeisky and Vertikalny ores
- identify the typical tailings silver tenor and determine the operating parameters by depleting the solution
- obtain design parameters for large scale plant.

The sample for the test work was prepared by SGS Vostok by leaching a representative ore sample obtained by riffling. The silver concentration of the solution was reported as 798 ppm. Full details of the solution analysis can be seen in Appendix A

The bench scale direct electro-winning test equipment set up at SGS Vostok is shown in Figure 13.24.

Figure 13.24 Direct Electrowinning Test in Progress at SGS Vostok

The summary of the direct electro-winning results are shown in Table 13.30.

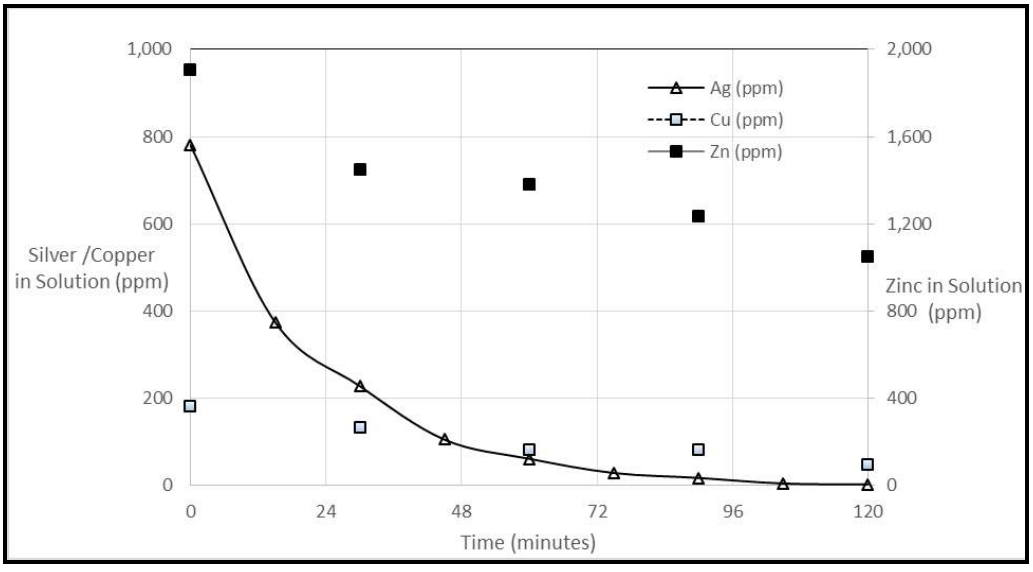
Table 13.30 Summary of the Results from Direct Electrowinning Test Work

Time (h)	Trial 1	Trial 2	Trial 4	Trial 5	Trial 6
	Ag (ppm)				
0.00	798	729	798	798	798
0.25	396	353	N/A	N/A	N/A
0.50	196	206	226	388	285
0.75	90	121	N/A	N/A	N/A
0.92	51	59	N/A	N/A	N/A
1.00	N/A	29	79	183	47
1.25	N/A	13	N/A	136	N/A
1.50	N/A	5	31	136	9.3
1.75	N/A	2		53	5.2
2.00	N/A	N/A	N/A	31	3.6

The results shown in Table 13.30 demonstrate that the direct electro-winning depletes the silver from the solution to below 5 ppm in approximately 2 hours of time. Full details of the results can be seen in Appendix A.

Based on this results Tetra Tech plotted the depletion curves for silver, copper and zinc (Figure 13.25)

Figure 13.25 Depletion Curves for Silver, Copper and Zinc



THICKENING TESTS

Thickening tests were performed on the cyanide leach residue slurry of the gravity tailings at a P_{80}=75 µm grind and 45% solids (w/w) pulp density. The test work included flocculent screening and optimization followed with further tests at the optimized conditions according to standard procedure.

A summary of the thickening test results with the specific thickener areas calculated by SGS Vostok according to Talmage and Fitch method are shown in Table 13.31.

Results and details of all other tests can be seen in Appendix A.

Table 13.31 Thickening Test Results

Reagent Addition (g/t)	Final Percent Solids (%)	Thickener Underflow Unit Area ($m^2/t/d$)	Thickener Hydraulic Unit Area Overflow Capacity ($m^2/t/d$)	Settling Rate ($m^3/m^2/d$)	Total Suspended Solids (Overflow) (mg/l)
0	69	0.309	0.314	1.4	24.0
10	69	0.258	0.111	4.5	22.3
20	69	0.205	0.050	15.3	19.8
30	69	0.191	0.026	23.5	18.0

Based on these results a unit thickener area of 0.191 $m^2/t/d$ was taken for thickener sizing along with a flocculent consumption of 30 g/t.

FILTRATION TESTS

Standard vacuum filtration tests were conducted on the cyanide leach residue slurry of the gravity tailings at a P_{80} = 75 µm grind and 45% solids (w/w) pulp density. These tests were conducted with and without flocculants and the filter cake thickness and moisture content were determined. The summary of the filtration tests results are shown in Table 13.32. Additional details of the filtration tests can be seen in the Appendix A.

Table 13.32 Filtration Tests Results

Reagent Added (g/t)	Time (s)			Residue Thickness (mm)	Residue Moisture (%)	Specific Solids Throughput ($kg/m^2/h$)	Specific Filtrate Throughput ($l/m^2/h$)
	Residue Sampling	Residue Drying	Cycle				
0	60	420	1,080	19	32.28	115.79	71.29
30	320	130	450	19	32.16	251.92	184.97

It is clear from the data shown in Table 13.32 that flocculants addition helps to reduce the filtration time substantially as expected.

CYANIDE DESTRUCTION TESTS

Batch Cyanide destruction tests were conducted with calcium hypochlorite as the detoxifying reagent. The summary of the results are shown in Table 13.33. Full details of the test conditions and results can be seen in the Appendix A.

Table 13.33 Summary of Cyanide Destruction Test Results

Stage	pH	Parameters			Assay (mg/l)			
		Eh, mV	Added (kg/m³)					
		Initial	$Ca(OCl)_2$	CaO	CN_{TOTAL}	CN_{WAD}	CNS	Cl
Feed Pulp	10.27	124	-	-	9649.9	427.34	334.2	<50
Stage I – 10 min	10.63	330	12.5	6.14	476.1	11.9	69.8	178
Stage II – 1 h	10.5	530	8.6	0.82	5.54	0.67	<1	213
Stager III – 1 h	10.7	610	5.0	0.74	2. 5	<0.1	<1	533

Notes: $Ca(OCl)_2$ – calcium hypochlorite; CaO – calcium oxide; CN_{TOTAL} – total cyanide CN_{WAD} – cyanide weak acid dissociable; Cl – chlorine

The test results indicated that hypochlorite treatment of the cyanide solution allows effective destruction of CN_{TOTAL}, C_{WAD}, and CNS.

Acid Base Accounting Tests

Acid-base accounting (ABA) test was conducted on cyanide tailings to determine the acid-generating potential (AP) and neutralization potential (NP), and subsequently calculate the net neutralization potential (NNP). The results of the test work is expressed as the required mass of calcium carbonate ($CaCO_3$) for the neutralization of 1,000 tons of tailings. A summary of the ABA test results are shown in Table 13.34.

Table 13.34 ABA Results

Sample	S total (%)	Potential (t of $CaCO_3$/1,000 tons)		
		AP	NP	NNP
CN Residue	0.82	25.1	53.9	28.8

The results indicates that the tested sample is non-acid generating (NAG), but expressed a positive NNP of 28.8 t of calcium carbonate per 1,000 ton of samples.

13.4 Leach Variability Test Work (TOMS)

13.4.1 Sample Background and Selection

Tetra Tech has reviewed the sample inventory list provided by Silver Bear and selected 18 leach variability samples in order to provide variations of silver grade and oxidation state as reported in the geological Mineral Resource model. The preliminary selection was then reviewed and modified based on the sample delivery list to the laboratory. Tetra Tech's recommendation for sample selection is shown in Table 13.35. The sample inventory list provided by Silver Bear is located in Appendix A.

Table 13.35 Tetra Tech Sample Recommendations for Leach Variability Test Work

	Hole ID	Indicated Silver Grade (g/t)
Oxides		
Low Grade	V13-22	216
	V13-24	221
Medium Grade	V13-10	652
	V13-14	759
High Grade	V07-004	971
	V13-11	1,323
	Hole ID	**Head Grade (g/t)**
Transition		
Low Grade	V09-150	232
	V07-015	239
Medium Grade	V13-11	525
	V09-152	756
High Grade	V09-156	1,214
	V13-22-2	3,072
Primary		
Low Grade	V13-11-3	11.9
	V08-075	338.5
Medium Grade	V08-080	671
	V13-10-2	1488
High Grade	V08-087	933.5

Note: *The oxidation status of some of the selected samples were reassigned based on the laboratory sulphur analysis, details of this analysis can be seen in Appendix A.

13.4.2 Sample Preparation

The selected variability leach samples were prepared by two stage closed circuit crushing and riffling in the laboratory in order to produce a representative sub-sample of -0.5 mm. The subsample of -0.5 mm size was the ground to P_{80} 75 µm mixed, and then representative samples were taken for silver assays, sulphur phase analysis, and leach tests.

13.4.3 Head Assays

The silver head assays as reported by the laboratory are shown in Table 13.36.

Table 13.36 Silver Head Assays

Sample No.	Sample Marking	Ag Grade (g/t)
1	V07-004	337
2	V07-015	669
3	V08-075	504
4	V08-087	266
5	V09-150	149
6	V13-14	1,144
7	V13-24	132
8	V08-080	442
9	V09-152	1,060
10	V09-156	2,219
11	V13-10-1	89.
12	V13-10-2	1,488
13	V13-11-3	11.9
14	V13-11-1	498
15	V13-11-2	1,830
16	V13-22-1	695
17	V13-22-2	3,072

Note: It was reported by TOMS that one of the recommended sample was not available for test work due to sample shipping errors.

13.4.4 LEACH TESTS

Agitated leach tests were conducted in the laboratory (Figure 13.26) according to the conditions specified in Table 13.37. These conditions were selected based on the leach optimisation test work completed in SGS Vostok in 2014.

Figure 13.26 Agitated Leach Test in Progress

Table 13.37 Leach Test Conditions

Parameter	Units	Value
NaCN Concentration	ppm	2,000
pH	–	11.5
Slurry Density at Cyanidation	% solids	40
Leach Retention Time	h	120
Grind (P_{80})	µm	75

13.4.5 RESULTS

The leach results for the oxidized, transition and primary ore samples are shown in Table 13.38, Table 13.39, and Table 13.40.

Table 13.38 Leach Results for Oxidized Samples

Sample Reference	Ag Grade (g/t)		Ag Dissolution (%)
	Head	Tails	
V07-004	337	96.1	71.48
V13-14	1,144	174	84.79
V13-24	132	8.74	93.38
V13-10-1	89.2	24.5	72.53
V13-10-2	1,488	215	85.55
V13-22-2	3,072	394	87.17
V13-11-2	1,830	416	77.27
V13-11-3	11.9	1.51	87.31
Mean			**82.4**
Standard Deviation			**7.8**

Table 13.39 Leach Results for Transition Samples

Sample Reference	Ag Grade (g/t)		Ag Dissolution (%)
	Head	Tails	
V07-015	669	538	19.58
V09-150	149	58.6	60.67
V09-152	1,060	874	17.55
V09-156	2,219	1769	20.28
V13-22-1	695	169	75.68
V13-11-1	498	135	72.89
Mean			**44.4**
Standard Deviation			**28.2**

Table 13.40 Leach Results for Primary Samples

Sample Reference	Ag Grade (g/t)		Ag Dissolution (%)
	Head	Tails	
V08-075	504	292	42.06
V08-087	266	231	13.16
V08-080	442	303	31.45
Mean			**28.9**
Standard Deviation			**14.6**

The results clearly indicate that the amenability to cyanidation varies significantly between the tested samples. The silver dissolution in cyanide ranges from 13.16% (primary ore, Table 13.40) to 93.38% (oxide ore, Table 13.38). The average silver dissolution for oxide ore (82.4%) is comparable to the silver dissolutions reported by SGS Vostok test work in 2014. However the average dissolutions for transition and primary ore types are 44.4% and 28.9%. The lower dissolutions clearly indicate that

the leaching conditions for the transition and primary ore types are not similar to the oxide ore. It can be concluded from this test work that the primary ore leach requires additional optimization work.

13.5 PRIMARY ORE TEST WORK (TOMS)

13.5.1 SAMPLE BACKGROUND

Tetra Tech was not involved with the sample selection for the primary ore metallurgical test work completed by TOMS, but did review the sample lists provided by Silver Bear. It is understood that the 27 test work samples were collected from 11 drillholes within the mineralization to represent the primary ore. It appears that the drillholes have covered the entire mineralization which could be mined over the LOM. The selected samples were sent to TOMS as a single composite sample weighing approximately 89 kg with an average silver head grade of 368.5 g/t. The sample selection list provided by Silver Bear is shown in Table 13.41.

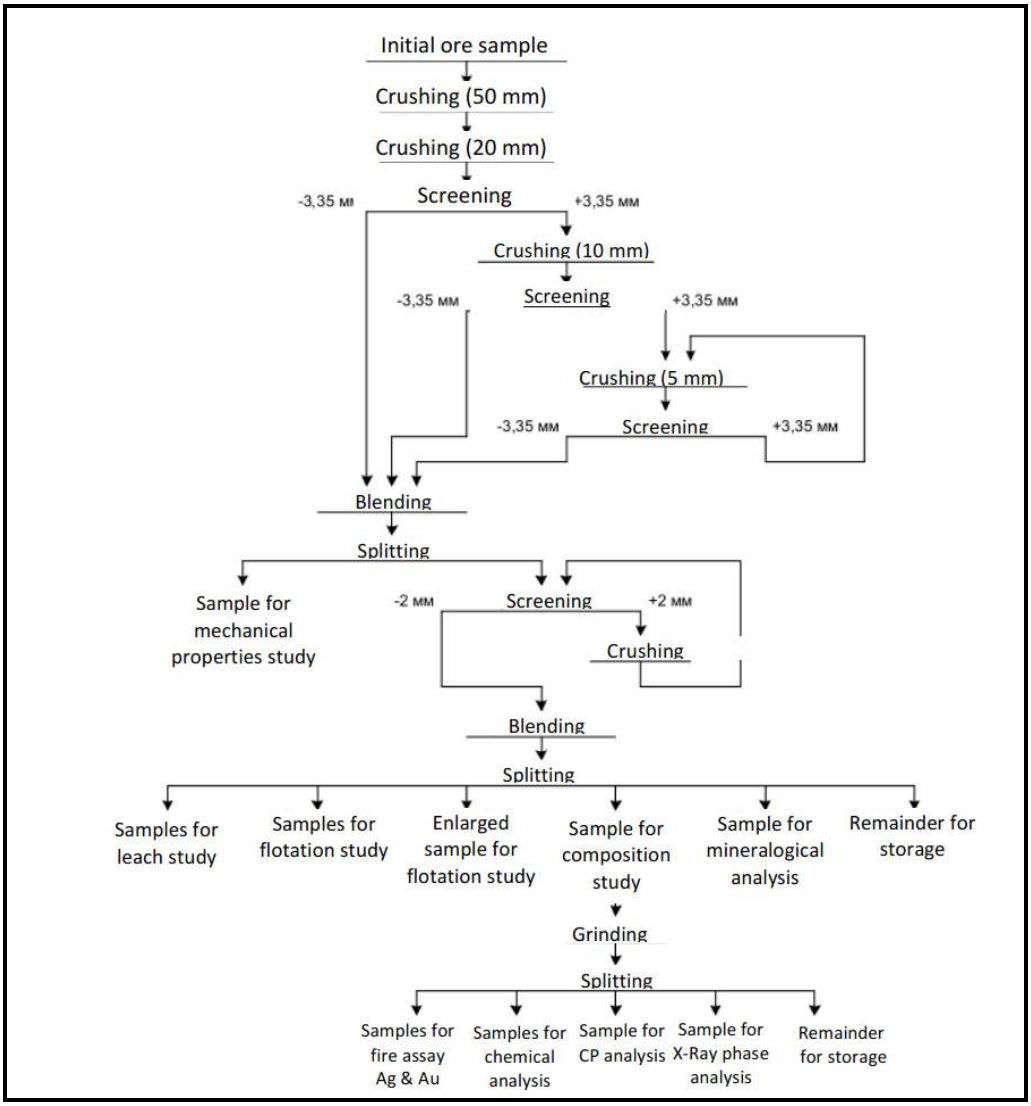
Table 13.41 Sample List for Primary Ore Composite

Sample ID	Hole ID	From (m)	To (m)	Ag (g/t)
9675	V14-02	85.30	86.40	220
9675	V14-03	68.60	69.40	366
9650	V14-06	52.50	53.15	77.2
9650	V14-06	53.15	54.00	152
9650	V14-06	54.00	54.95	218
9650	V14-06	54.95	55.50	880
9650	V14-06	55.50	56.20	116
9175	V14-09	107.40	108.05	222
9175	V14-09	108.05	109.10	191
9175	V14-09	109.10	110.10	147
9175	V14-09	110.10	111.10	177
9150	V14-14	42.00	43.00	203
9150	V14-16	169.00	169.25	718
10050	V14-19	177.00	177.70	1594
10050	V14-19	177.70	178.40	361
10050	V14-19	178.40	179.20	136
10050	V14-19	179.20	179.40	226
9950	V14-21	145.40	145.80	431
9950	V14-21	145.80	146.00	208
9950	V14-21	146.00	146.30	757
9950	V14-21	146.30	147.30	122
9350	V09-150БД	26.50	27.50	265
9350	V09-150БД	27.50	28.50	527
9350	V09-150БД	28.50	29.00	205

13.5.2 SAMPLE PREPARATION

The composite sample was prepared by crushing, screening and riffling for various tests as shown in Figure 13.27.

Figure 13.27 Sample Preparation Flowsheet



13.5.3 HEAD ASSAYS

TOMS analysed the head sample for silver, gold, copper, lead, and zinc assays, and conducted a whole rock analysis for all other major elements. A summary of the assay and whole rock analyses are shown in Table 13.42.

Table 13.42 Head Assays of the Primary Ore Composite

Element	Sample Grade (g/t)				Average Grade (g/t)
	1	2	3	4	
Silver	366	379	361	379	371
Gold	0.141	0.115	-	-	0.128

Table 13.43 Whole Rock Analysis of the Primary Ore Composite

Element, Component	Grade (%)	Element, Component	Grade (%)
SiO_2	33.1	Zn	3.56
AL_2O_3	7.33	Pb	3.25
CaO	0.56	Cи	0.089
K_2O	2.18	Ni	0.0013
Na_2O	<1.0	Sb	0.048
MgO	1.27	As	0.94
MnO	3.32	Cd	0.023
P_2O_5	0.089	Cr	0.006
TiO_2	0.35	Co	0.0015
C_{gen}	4.05	Mo	<0.001
CO_2	12.3	Ba	0.022
Fe_{gen}	19.6	Sn	0.031
Fe(S)	2.53	Sr	0.0067
Fe_{oxide}	17.07	W	<0.005
$Fe_{hydroxide}$	1.18	Bi	<0.005
S_{gen}	4.59	Au (g/t)	0.128
$S_{sulphate}$	<0.25	Ag, (g/t)	371

13.5.4 MINERALOGICAL ANALYSIS

TOMS has undertaken a mineralogical analysis by optical microscope analysis, and estimated the quantities of each mineral by basic calculations of chemical composition of the sample. The summary of TOMS mineralogical analysis is shown in Table 13.44.

Table 13.44 Mineralogical Analysis

Mineral	Estimated Mass Fraction (%)
Rock Forming Minerals	
Quartz	25.0
Siderite	34.0
Mica	20.0
Ore Forming minerals	
Galena	4.0
Sphalerite	5.0
Arsenopyrite	2.0
Pyrite	4.0
Iron Hydroxides	2.0
Pyrolusite, Psilomelane	4.0
Total	**100.0**

13.5.5 BOND BALL MIL INDEX TEST

TOMS completed a Bond ball mill work index test in order to determine the parameters required for the design of the comminution circuit. The summary of the results are shown in Table 13.45.

Table 13.45 Bond Ball Mill Index Test Results

Description	Unit	Results
Limiting Screen Size	µm	38
F_{80}	µm	2,434
P_{80}	µm	33
Work Index, BW_i	kWh/t	19.0

The Bond ball mill index (BW_i = 19.0 kWh/t) indicate that the ore is categorised amongst the hardest range of the Bond ball mill expert scale. This means that fine grinding of primary ore by ball milling could be an energy intensive operation.

13.5.6 CYANIDE LEACH TESTS

TOMS has conducted a series of agitated tank leach tests (Figure 13.28) in order to optimize the leach conditions for primary ore.

Figure 13.28 Agitated Tank Leach Tests

Baseline Test

The first leach test was conducted as a baseline under the optimized conditions for oxides ore but with saturated dissolved oxygen levels. The summary of the baseline leach test conditions are shown in Table 13.46.

Table 13.46 Baseline Leach Test Conditions

Parameter	Unit	Value
Grinding size P_{80}	µm	75
pH	-	11.5
Concentration Cyanide-ions	mg/l	2000
Concentration DO	mg/l	Around 8.4-8.6
Pulp Aeration	-	Compressed air
Pulp Density	% solids	40
Leaching Duration	h	120
Solution Sampling	h (from start of test)	4, 8, 24, 48, 72, 96, 120

The summary of the baseline leach test results are shown in Table 13.47.

Table 13.47 Baseline Leach Test Results

Test Reference	Head Grade (Ag g/t)	Tails Grade, (Ag g/t)	Dissolution (Ag %)
Baseline	371	262	29.38

The poor leach dissolution (29.4%) demonstrates that the leach conditions need further optimization.

Grind Size Optimization

Grind size optimization tests were performed on three grinds (P_{80} 75 µm, P_{80} 53 µm, and P_{80} 38 µm). Cyanide concentration was maintained at 10,000 mg/l throughout the testing. All other test parameters were similar to the base line test parameters (Table 13.46).

The summary of the grind size optimization leach test results are shown in Table 13.48.

Table 13.48 Grind Size Optimization Results

Grind Size (P_{80}-µm)	Head Grade (Ag g/t)	Tails Grade (Ag g/t)	Dissolution (Ag %)
75	371	199.5	46.23
53		179.5	51.62
38		167	54.99

The results indicate that the silver dissolution steadily increases with fine grinds, and based on this, the grind size for the next stage of testing was selected as P_{80} = 25 µm in order to maximize the silver dissolution.

Cyanide Concentration Optimization

Cyanide concentrate optimization tests were performed on three different cyanide concentrations (2,000 mg/l, 5,000 mg/l, and 10,000 mg/l). The grind size was kept constant at P_{80} = 25 µm for all three tests. All other test parameters used similar baseline test parameters.

The summary of the cyanide concentration leach test results are shown in Table 13.49.

Table 13.49 Cyanide Concentration Optimization Results

Cyanide Concentration (mg/l)	Head Grade (Ag g/t)	Tails Grade (Ag g/t)	Dissolution (Ag %)
2,000	371	211	43.13
5,000		143.5	61.32
10,000		107.5	71.02

The results indicate that the leach dissolution increases with the increase in cyanide concentration. Based on this results a cyanide concentration of 10,000 mg/l was selected for the final leach test under optimized conditions.

Leach Test Under Optimized Conditions

The leach results indicate that the leach dissolution steadily increases with both finer grind and higher cyanide concentration. The results also demonstrated that the primary ore leach kinetics are extremely slow and the leach curves steadily progress even after 120 hours. It was clear that further work is required to adequately optimise the leach conditions. However the project schedule and the lack of sample availability of sample for additional test work has prevented additional optimisation work. Therefore, the leach test was conducted under the optimised test conditions shown in Table 13.50.

Table 13.50 Leach Test Condition for Optimized Condition Test

Parameter	Unit	Value
Grinding size P_{80}	µm	25
pH	-	11.5
Concentration Cyanide-ions	mg/l	10,000
Concentration DO	mg/l	Around 8.4-8.8
Pulp Aeration	-	Compressed air
Pulp Density	% solids	30
Leaching Duration	h	120
Solution Sampling	h (from start of test)	2, 4, 8, 24, 48, 72, 96, 120

The summary of the leach under optimized conditions are shown in Table 13.51.

Table 13.51 Results of the Leach under Optimized Conditions

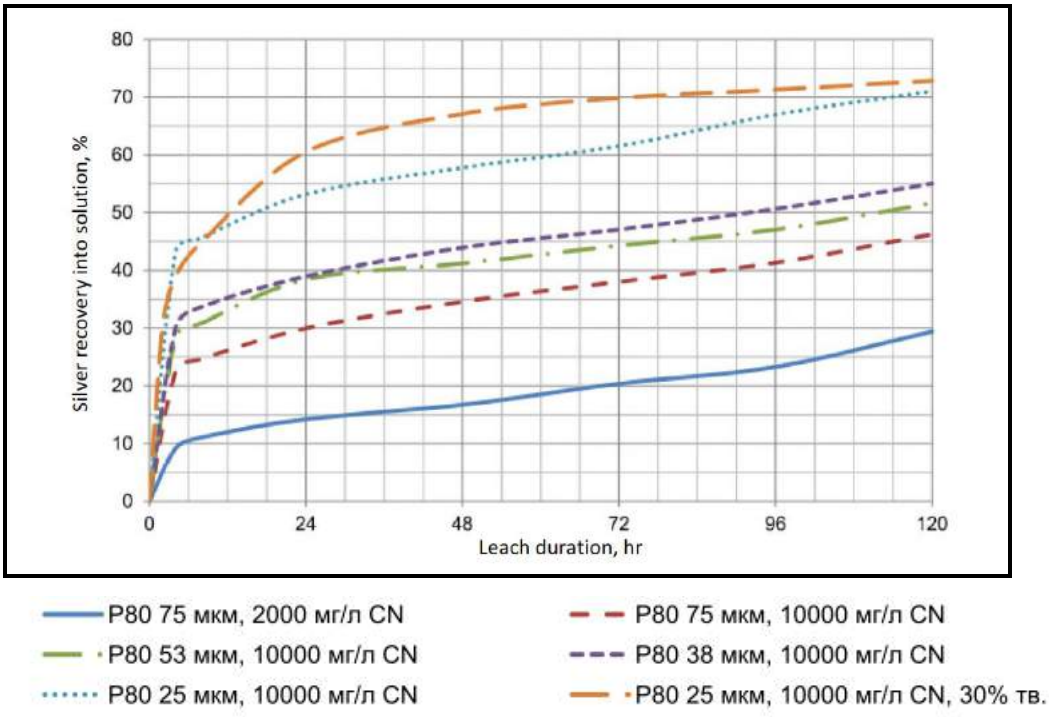
Test Reference	Head Grade (Ag g/t)	Tails Grade (Ag g/t)	Dissolution (Ag %)
Optimised Conditions	371	100.8	72.83

The reduction in leach pulp density from 40% solids to 30% solids resulted in a marginal increase of overall silver dissolution from 71 to 72.8%. However the overall leach kinetics significantly increased from this pulp density reduction (Figure 13.29).

Summary of the Leach Test Results

The leach optimization program indicated that leach kinetics are extremely slow for the primary ore when the dissolution is increased beyond 120 hours (Figure 13.29).

Figure 13.29 Leach Kinetics Curves



As shown in Figure 13.29, the test with the lowest pulp density (30% solids), finer particle size distribution (P_{80} = 25 µm), and higher cyanide concentration (10,000 mg/l) has shown that better kinetics, with approximate silver dissolution of 70%, can be attained within 72 hours of leach time.

It is also clear from the leach optimisation program that the overall silver dissolution significantly increased with the finer grind and higher cyanide concentration as shown Figure 13.30 and Figure 13.31.

Figure 13.30 Variation of Leach Dissolution with Grind Size

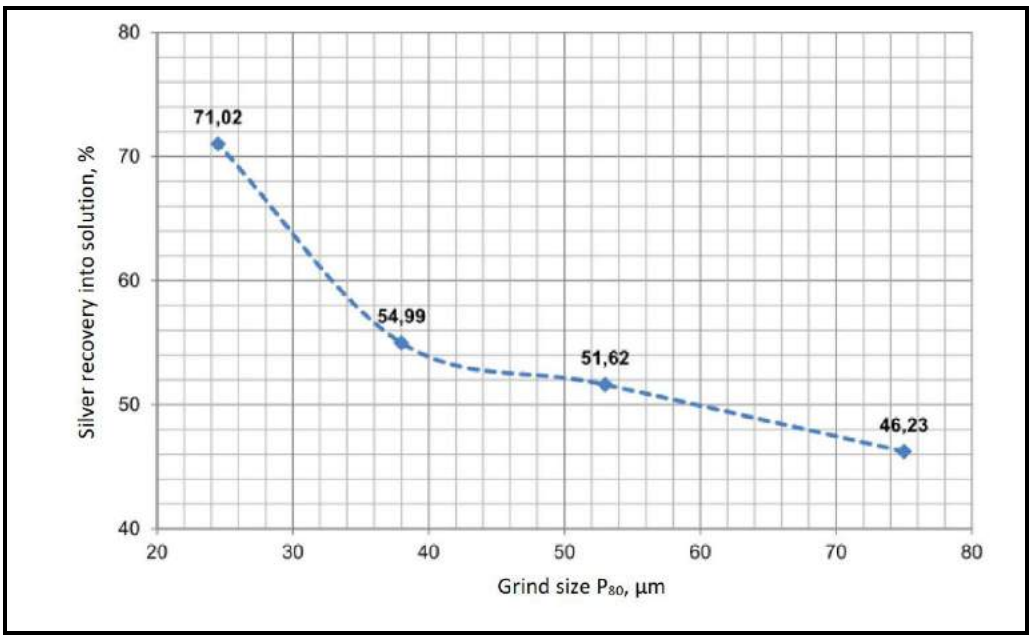
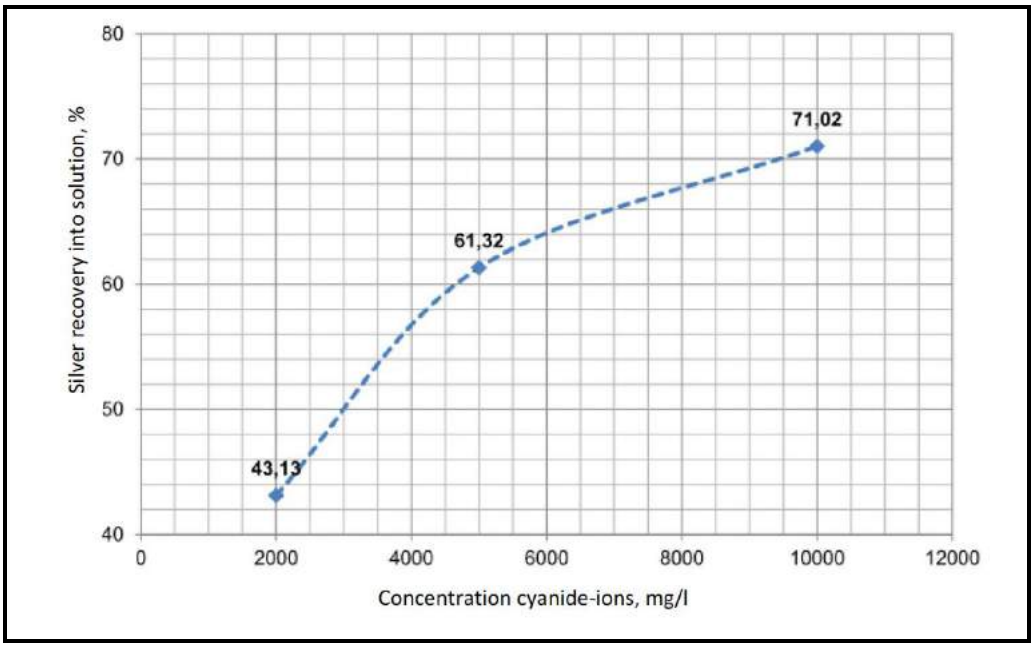


Figure 13.31 Variation of Leach Dissolution with Cyanide Concentration



Multiple Linear Regression Analysis

Tetra Tech has undertaken a multiple linear regression (MLR) analysis of the primary ore leach test results in order to identify the most significant and least significant factors for silver dissolution. The summary of MLR analysis is shown in Table 13.52 and Table 13.53.

Table 13.52 Summary of MLR Analysis

Ag Rec %	X_1 (P_{80} µm)	X_2 (% Solids)	X_3 (CN Conc' mg/l)	X_4 (CN Cons' kg/t)	X_5 (CaO Cons' kg/t)
29.38	75	40	2000	13.4	6
46.23	75	40	1000	37.8	6
51.62	53	40	1000	41.6	6.3
54.99	38	40	1000	43.7	6.2
43.13	25	40	2000	9.8	9.9
61.32	25	40	5000	21.7	10.6
71.02	25	40	10000	49.3	9.8
72.83	25	30	2000	58	14.1

Table 13.53 MLR Parameters

X_5	X_4	X_3	X_2	X_1	b
4.7102	0.7371	-0.0003	2.8802	-0.0381	-121.162

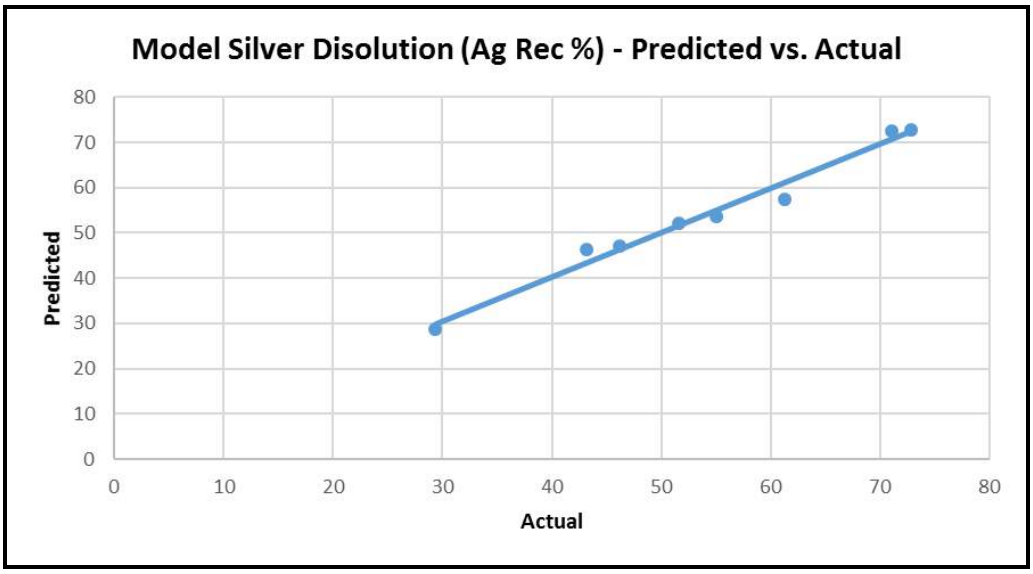
The MLR results were interpreted by expressing the constants in the form of a dissolution deviation equation.

$$Ag\ Rec\ \% = X_1.(P_{80}\ um) + X_2.(\%\ Solids) + X_3.(CN\ Conc') + X_4.(CN\ Cons') + X_5.(CaO\ Cons') + b$$

The dissolution deviation equation suggests that the silver dissolution is predominantly influenced by the lime consumption, followed by slurry density. However the bigger constant indicates that there are unknown parameters which has not been monitored during the test work that has a significant influence in the overall silver dissolution.

The dissolution deviation equation has been used to predict the silver dissolution in order to validate the MLR analysis and an excellent correlation has be observed between predicted and actual dissolutions (Figure 13.32).

Figure 13.32 Validation of MLR Constants



13.5.7 FLOTATION TEST WORK

The flotation test work included initial optimisation tests followed by an open cycle bulk flotation test in order to prepare a sample for intensive cyanidation tests. All of the flotation tests were conducted in a laboratory bench scale flotation cell (Figure 13.33).

Figure 13.33 Laboratory Flotation Test

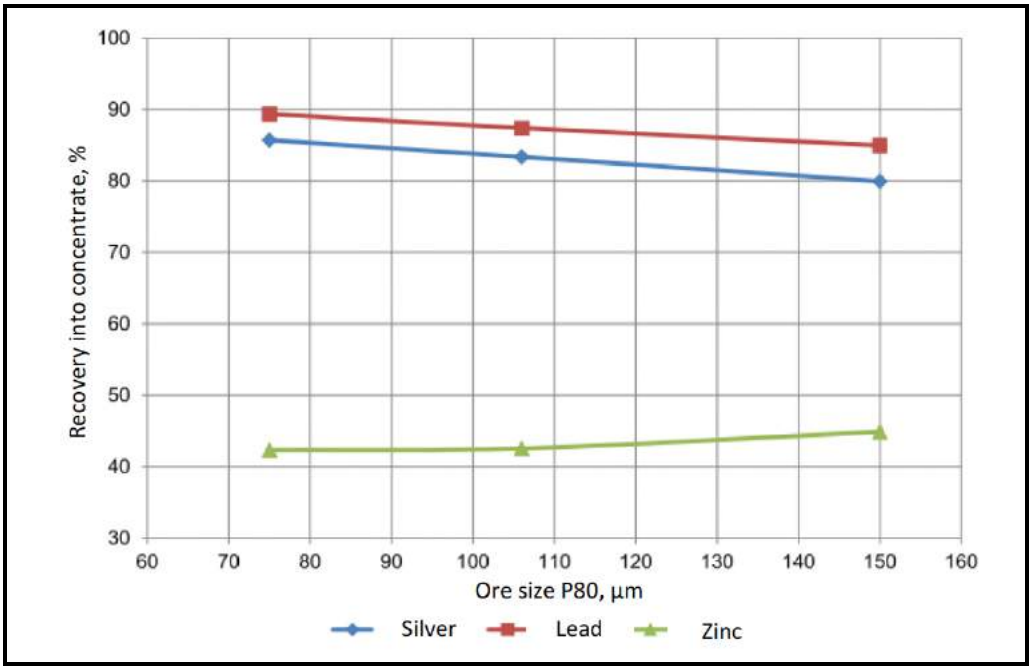
Flotation Optimization Tests

Effect of grind and reagent scouting tests were conducted as the primary step in flotation in order to establish the optimal flotation conditions. A summary of the effect of these optimisation tests are shown in Table 13.54 and Figure 13.34 . Full details of the test conditions and detail results are shown in the Appendix A.

Table 13.54 Effect of Grind Results

Grind Size (P_{80}-µm)	Product	Masspull (%)	Grade (Ag g/t)	Recovery (Ag %)	Grade (Pb %)	Recovery (Pb %)	Grade (Zn %)	Recovery (Zn %)
150	Concentrate	11.35	2,667	79.94	22	84.97	14.4	44.88
	Middlings scavenger	18.2	307	14.76	1.85	11.47	5.81	29.05
	Middlings control	11.75	61.4	1.91	0.344	1.38	1.78	5.75
	Flotation tails	58.7	21.9	3.4	0.109	2.18	1.26	20.32
	Head ore	100	379	100	2.935	100	3.64	100
106	Concentrate	12.3	2,452	83.35	18.91	87.4	12.03	42.55
	Middlings scavenger	21.8	209	12.59	1.17	9.58	5.2	32.61
	Middlings control	14.55	36.6	1.47	0.237	1.3	1.7	7.11
	Flotation tails	51.35	18.2	2.58	0.089	1.72	1.2	17.72
	Head ore	100	362	100	2.66	100	3.48	100
	Concentrate	13.25	2,401	85.71	19.16	89.37	11.32	42.3
75	Middlings scavenger	24.2	166	10.82	0.94	8.01	5.17	35.3
	Middlings control	14.95	31.2	1.26	0.215	1.13	1.68	7.09
	Flotation tails	47.6	17.2	2.21	0.089	1.49	1.14	15.31
	Head ore	100	371	100	2.84	100	3.54	100

Figure 13.34 Summary of Effect of Grind Test Results



The results indicate that the P_{80} = 75 µm grind have provided the highest silver (85.7%) and lead recoveries with lowest zinc recoveries. At this grind (P_{80} = 75 µm) the concentrate silver grade and mass pull are expected at approximately 2,400 g/t and 13%, respectively.

The direct relationship between silver and lead recoveries and inverse relationship between silver and zinc recoveries could be interpreted as the primary silver minerals are mostly associated with galena rather than sphalerite.

Bulk Flotation Tests

Bulk flotation tests were conducted at a grind of P_{80} = 75 µm in order to prepare a feed sample for the intensive cyanidation tests.

The summary of the bulk flotation test results are shown in Table 13.55.

Table 13.55 Bulk Flotation Results

Product	Masspull (%)	Grade (Ag g/t)	Recovery (Ag %)	Grade (Pb %)	Recovery (Pb %)	Grade (Zn %)	Recovery (Zn %)
concentrate	14.84	2,333	93.59	19.69	96.03	11.59	44.7
Tailings	85.16	27.83	6.41	0.142	3.97	2.5	55.3
Head ore	100	370	100	3.04	100	3.85	100

The bulk flotation results confirm that approximately 93.6% of silver can be concentrated in to a polymetallic concentrate with approximately 15% masspull. The expected concentrate silver grade is 2,333 g/t.

Intensive Leach Tests

The flotation concentrates were tested for intensive cyanidation using three different sets of conditions, as shown in Table 13.56.

Table 13.56 Test Conditions for Intensive Cyanidation Tests

Parameter	Unit	Value		
		1	2	3
Grinding Size P_{80}	µm	25		15
pH	-	11.5		
Concentration Cyanide-ions	mg/L	20,000	30,000	
Concentration DO	mg/L	Around 8.4-8.8		
Pulp Aeration	-	Pure oxygen		
Pulp Density	% solids	25		
Leaching Duration	h	144		
Solution Sampling	h (from start of test)	2, 4, 8, 24, 48, 72, 96, 120, 144		

The results of the intensive cyanidation tests are summarized in Table 13.57.

Table 13.57 Intensive Cyanidation Results

Test Reference	Grind Size (P_{80} µm)	Cyanide Concentration (mg/l)	Head Grade (g/t)	Tails Grade (g/t)	Dissolution (%)
Test 1	25	20,000	2333	1940	16.85
Test 2		30,000		1711	26.66
Test 3	15			1776	23.87

It appears that the intensive cyanidation of flotation concentrate resulted in very poor dissolutions. The best silver dissolution (26.7%) was obtained at a P_{80} = 25 µm grind and a cyanide concentration of 30,000 ppm. This is a clear indication that the concentrate is not amenable to cyanidation at the tested grinds and selected conditions suggest that silver mineral might be refractory. Based on this test, it can be concluded that the intensive cyanidation of flotation concentrates is not a viable process option for the primary ore.

13.6 LABORATORY RESEARCH PROGRAM (TOMS)

The objective of the laboratory research program was to demonstrate/justify the socio-economic benefits of the Project to the Russian regulatory authorities. Tetra Tech was not involved in any aspect of this test work; however, sections of this test work report were translated and made available to Tetra Tech in June 2017 for review. The translated work consist of the following research;

- gravity concentration tests
- leach tests
- direct electrowinning tests.

A high-level summary of this work is presented in this section and additional details can be found in the Appendix A.

13.6.1 GRAVITY CONCENTRATION TESTS

TOMS completed two batch gravity concentration tests using a Knelson concentrator. One of the tests was conducted under standard conditions, while the second test was conducted with lower concentrate masspull.

The tests were conducted on a sample with an approximately 960 g/t silver head grade, using the batch laboratory-scale Knelson KC-MD3 test unit. The grind size for the test was fixed at (P_{76} = 71 µm). A summary of the batch gravity test results are shown in Table 13.58 and Table 13.59.

Table 13.58 Batch Gravity Concentration Test Results (Standard Conditions)

Product	Mass (%)	Ag Grade (g/t)	Ag Recovery (%)
Concentrate	1.08	5,814.0	6.50
Tailings	98.92	911.5	93.50
Feed	100.00	964.4	100.00

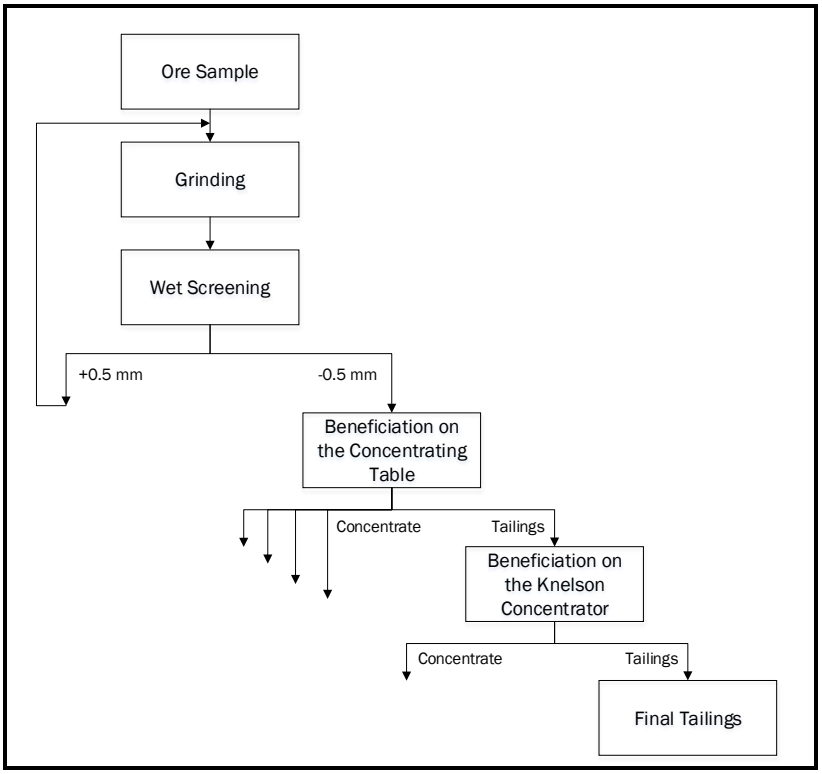
Table 13.59 Batch Gravity Concentration Test Results (Lower Masspull)

Product	Output (%)	Ag Grade (g/t)	Ag Recovery (%)
Concentrate	0.13	5,060	0.71
Tailings	99.87	953	99.29
Feed	100.00	958.5	100.00

The results of the standard condition test are comparable to the feasibility study test work completed by SGS Vostok (SGS 2014). However, the results of the lower masspull test indicates much lower silver recovery; the silver recovery has reduced significantly with the reduction in concentrate masspull.

Based on the batch test results, TOMS has undertaken a series of gravity tests using tables and Knelson concentrator with the objective of understanding the relationship between masspull and silver recovery. These tests were conducted according to the flowsheet shown in Figure 13.35.

Figure 13.35 Flowsheet for TOMS Gravity Test Work



The results of the gravity concentration by tabling and Knelson concentrator is shown in Table 13.60.

Table 13.60 Gravity Concentration Results

Product	Output (%)	Grade of Ag (g/t)	Recovery of Ag (%)
Table Concentrate №1	0.85	5,668.0	4.95
Table Concentrate №2	2.02	7,125.0	14.85
Table Concentrate №3	3.25	3,850.0	12.89
Table Concentrate №4	3.39	1,810.0	6.32
Knelson Concentrate	1.69	1,594.0	2.78
Total Gravity	11.20	3,620.7	41.78
Concentrate			
Tailings	88.80	636.0	58.22
Feed	100.00	970.1	100.00

The results indicates that silver recovery continues to increase with the concentrate masspull. The results also indicates that up to 42% silver recovery could be achieved by gravity concentration.

13.6.2 LEACH TESTS

ROM LEACH TEST

TOMS completed a series of leach optimisation tests on the ROM sample and reported a set of parameters (Table 13.61) as the optimal conditions.

Table 13.61 Optimal Parameters for ROM Leach

Parameter	Unit	Value
Ore Grinding Size P_{80}	µm	71
NaCN Concentration	%	0.5
pH	–	10.5-11.5
Pulp Density	% -solids	40
Leach Retention Time	h	120

Based on this work, TOMS completed three ROM leach tests (using the conditions specified in Table 13.61) and reported that silver dissolutions ranging from 79.45 to 81.13% were observed. Tetra Tech notes that these dissolutions are comparable to the feasibility study leach test work conducted in SGS Vostok and TOMS variability leach test work on the oxide samples. However, Tetra Tech has not received further details, such as the leach kinetics curves, and is therefore unable to comment further on these results.

GRAVITY TAILS LEACH TESTS

TOMS completed leach tests on a gravity tails sample with a silver head grade of 912 g/t. The leach tests were conducted with varying particle size distribution and

cyanide concentration, similar to the ROM leach tests. TOMS reported that silver dissolution of the gravity tails ranged from 70.86 to 80.82% for 120 h leach retention time. A summary of gravity tails leach results are shown in Table 13.62.

Table 13.62 Leach Results for Gravity Tails Leach Tests

Particle Size (P80 µm)	Concentration (NaCN %)	Ag Grade (g/t)		Dissolution (Ag %)
		Head	Tails	
71	0.1	912	278	70.86
50			246	74.21
38			238	75.05
71	0.2		196	79.45
			194	79.66
50			186	80.50
			192	79.87
38			183	80.82
			188	80.29
71	0.5		184	80.71
50			189	80.19
38			184	80.71

Tetra Tech notes that the reported silver dissolutions are lower than the silver dissolutions reported by SGS Vostok (SGS 2014). The average silver dissolution for the gravity tails leach was reported as approximately 88% by SGS.

Direct Electrowinning Tests

Lab scale direct electrowinning tests were conducted by Electrometals at the TOMS premises in 2015. The tests were conducted in a similar setup to the direct electrowinning tests completed at SGS Vostok in 2014. The results confirmed that direct electrowinning can be applied to recover silver from the leach solution. Additional details include the results of this work can be seen in the Appendix A.

13.7 Process Selection and Flowsheet Development

The completed test work indicates that the oxide and primary ores are mineralogically similar but their amenability to various metallurgical processes have significant differences. It is understood that both oxide ore and primary ore can be separated at the mining stage and processed in separate campaigns through the processing plant. In this context, two different process approaches are proposed to economically extract the silver.

13.7.1 Oxide Ore Processing

The oxide ore from the Mangazeisky deposit is free milling, amenable to gravity concentration and cyanidation.

The comminution test work indicates that the oxide ore is generally considered as moderately hard. Based on the test work a comminution circuit that contains two stage crushing followed by ball mill grinding in order to produce a feed with P_{80} = 75 µm is recommended.

Gravity concentration followed with intensive leaching of the concentrates and agitated tank leach of the gravity tails has shown to be the most economical approach to get maximum silver dissolution. Hence, a gravity circuit and an intensive leach circuit are recommended for the oxide ore processing.

The leach optimisation study indicates a 72-hour leach retention time along with a pH of 11.5 and cyanide concentration of 2 g/l would provide better economic returns on gravity tails. Thus, agitated tank leaching with a 72-hour retention time is recommended.

The direct electrowinning test work indicates that this technology has significant advantages for this type of ore. Therefore direct electrowinning is recommended for the silver extraction method from cyanide leach solutions.

In summary, the conceptual process flowsheet reported in the PEA has been validated with some modifications in order to reduce costs and maximise the silvery recovery from the oxide ore.

13.7.2 PRIMARY ORE PROCESSING

It is understood that the primary ore only constitutes a minor proportion (approximately 10%) of the feasibility study ore reserves and mining schedule. Thus, utilizing the oxide ore process plant with required modifications is proposed as an approach for the feasibility study.

The comminution test work indicates that the primary ore is generally considered as extremely hard. The leach optimization work indicated that a finer grind at P_{80} = 25 µm is required to achieve reasonable silver dissolution. The increased hardness and reduced grind size require more grinding energy. However the same flowsheet used for the oxide ore (two-stage crushing followed by ball mill grinding) can be utilized for the primary ore. Hence, Tetra Tech recommends the same comminution circuit but an increased milling capacity.

The test work indicates that the primary ore is amenable to flotation; however, the intensive leaching of flotation concentrates have resulted in very poor silver dissolutions. In addition, it must be recognized that the site location is very remote with challenging environmental and logistical problems. This works against the flotation process which has demanding requirements, for example the supply and availability of reagents as well as the production and transportation of the concentrate to a suitable smelter. In this context flotation is not recommended as a process flowsheet option for processing the primary ore.

The test work also indicates that the primary ore is amenable to direct cyanidation at finer grinds and higher cyanide concentrations with extremely slow leach kinetics.

The leach optimisation study indicates a 120 hour leach retention time along with a pH of 11.5 and cyanide concentration of 10 g/l would provide maximum silver

dissolution. Thus, increasing the number of tanks in order to achieve a leach retention time of 120 hours is recommended.

In terms of electrowinning, utilising the same direct electrowinning circuit is recommended.

In summary, the utilising the oxide ore flow sheet with minimal modifications in order to managed the ore hardness, finer grind and longer leach time requirements is recommended for the processing of the primary ore.

13.8 METALLURGICAL RECOVERY

Metallurgical recovery for the oxide ore and the primary ore estimated based on the reported test work and proposed plant design.

13.8.1 OXIDE ORE RECOVERY

The oxide ore recovery is estimated based on ROM leach flowsheet incorporating the variability leach test work. It is important to note that the results of the laboratory research work (TOMS 2015) are comparable to variability leach results, but are excluded from the recovery estimation in order to avoid any bias as a result of different operating conditions than the process design[1].

The summary of the overall silver recovery calculation is shown in Table 13.63.

Table 13.63 Overall Recovery Calculation for Oxides

	Amount (%)
Overall Silver Dissolution to Solution	79.7
Electrowinning Efficiency	99.0
Dissolved Silver Losses	1.0
Overall Silver Recovery	78.1

Based on the data presented in Table 13.63, the overall silver recovery for oxide ore is estimated as 78.1%.

13.8.2 PRIMARY ORE RECOVERY

The primary ore recovery is estimated based on the leach test work completed in TOMS.

Based on a leach dissolution 71.02% (120-hour leach retention time) the recovery is calculated as 69.6% assuming 99% electrowinning efficiency and 1% dissolved silver losses.

[1]The process design is based on 72 h leach retention time; TOMS work is based on 120 h leach retention time. Tetra Tech does not have information relating to silver dissolution after 72 h from TOMS research work.

14.0 MINERAL RESOURCE ESTIMATES

14.1 SUMMARY

Tetra Tech completed a new Mineral Resource estimate for the Mangazeisky North deposit, with an effective date of 19th October 2016. The most recent data included in the estimate was received on 19th October 2016. Mr. Joseph Hirst, BSc (Hons), MSc, EurGeol, CGeol, an independent QP as defined by NI 43-101, estimated the Mineral Resources.

Tetra Tech completed a new Mineral Resource estimate for the Vertikalny deposits, with an effective date of 8th July 2016. The most recent data included in the estimate was received on 1st July 2016. Mr. Robert Davies, BSc, EurGeol, CGeol, an independent QP as defined by NI 43-101, estimated the Mineral Resources.

This section also presents a restatement of the other Mineral Resources within the Property. Table 14.1 presents a summary status of the Mineral Resource estimates.

Table 14.1 Summary of Resource Estimates on the Mangazeisky Property

Deposit	Qualified Person	Effective Date
Nizhny Endybal	Mr. Robert Davies, BSc, EurGeol, Cgeol	11th September 2012
Mangazeisky North	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	19th October 2016
Mangazeisky South	Mr. Robert Davies, BSc, EurGeol, Cgeol	3rd June 2015
Vertikalny Central	Mr. Robert Davies, BSc, EurGeol, Cgeol	8th July 2016
Vertikalny Northwest	Mr. Robert Davies, BSc, EurGeol, Cgeol	8th July 2016
Sterzhnevoy	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	27th August 2016
Pophirovy	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	27th August 2016

Further details of the Project Mineral Resource estimates are publically available in Technical Reports prepared by Tetra Tech and filed under the Silver Bear profile on the CSA SEDAR website.

14.2 MANGAZEISKY NORTH MINERAL RESOURCE ESTIMATE

14.2.1 SUMMARY OF ESTIMATION TECHNIQUES

Tetra Tech created wireframe models using Datamine Studio 3 (version 3.24). Wireframes were created to represent mineralised bodies. Block modelling and Mineral Resource estimation were completed in Datamine.

The metal grades for the Mangazeisky North deposit were estimated using the OK interpolation method. Density was estimated using the IDW^2 interpolation methodology.

Statistical and grade continuity analyses were completed in order to characterise the mineralisation, and were subsequently used to develop grade interpolation parameters. The search ellipsoid dimensions and orientations were chosen to reflect the continuity revealed by geostatistical studies. Block size, discretisation, search size, and sample numbers were optimised using QKNA.

Tetra Tech adopted the definition of Mineral Resources as outlined within the CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014) in order to classify the Mineral Resources.

In order to demonstrate that the deposit has reasonable prospects for eventual economic extraction, a cut-off grade of 150 g/t silver was applied for Mineral Resources within 60 m of surface based on similar depths of the open pits at the other deposits on the Property.

14.2.2 DATABASE

Silver Bear provided Tetra Tech with its exploration database, in the form of an Microsoft® Excel spreadsheet containing the following tabs:

- collar
- survey
- assay
- geotechnical
- lithology
- density.

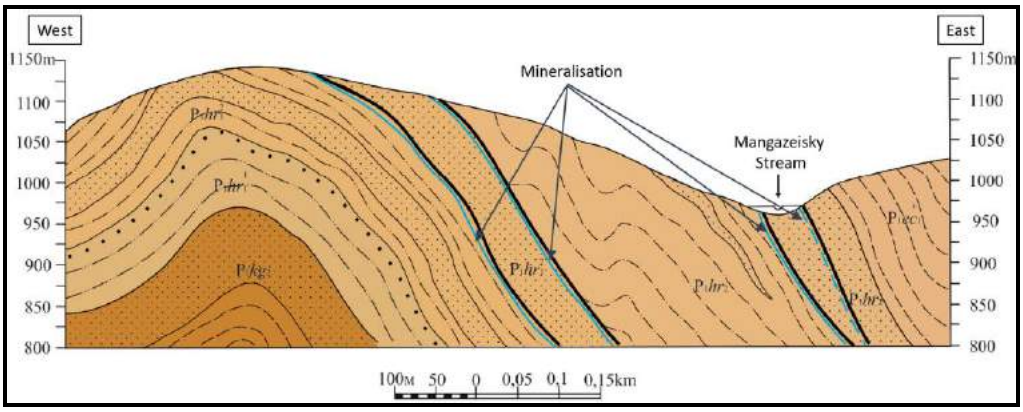
The database includes information on trenches and drillholes sampled during the period from 2013 through to the summer 2016.

The updated Mineral Resource for Mangazeisky North is based on additional drilling completed in the 2016 season, to test the strike extent of the mineralisation to the north and to infill some areas where additional secondary structures were identified. No additional surface trenches were cut and sampled in 2016.

14.2.3 GEOLOGICAL INTERPRETATION

The area is dominated by a north-northwest/south-southeast striking anticlinal fold (Figure 14.1). Bedding planes and mineralisation dips between 20 and 40° towards the east.

Figure 14.1 West to East Cross Section through Mangazeisky North



Source: Silver Bear

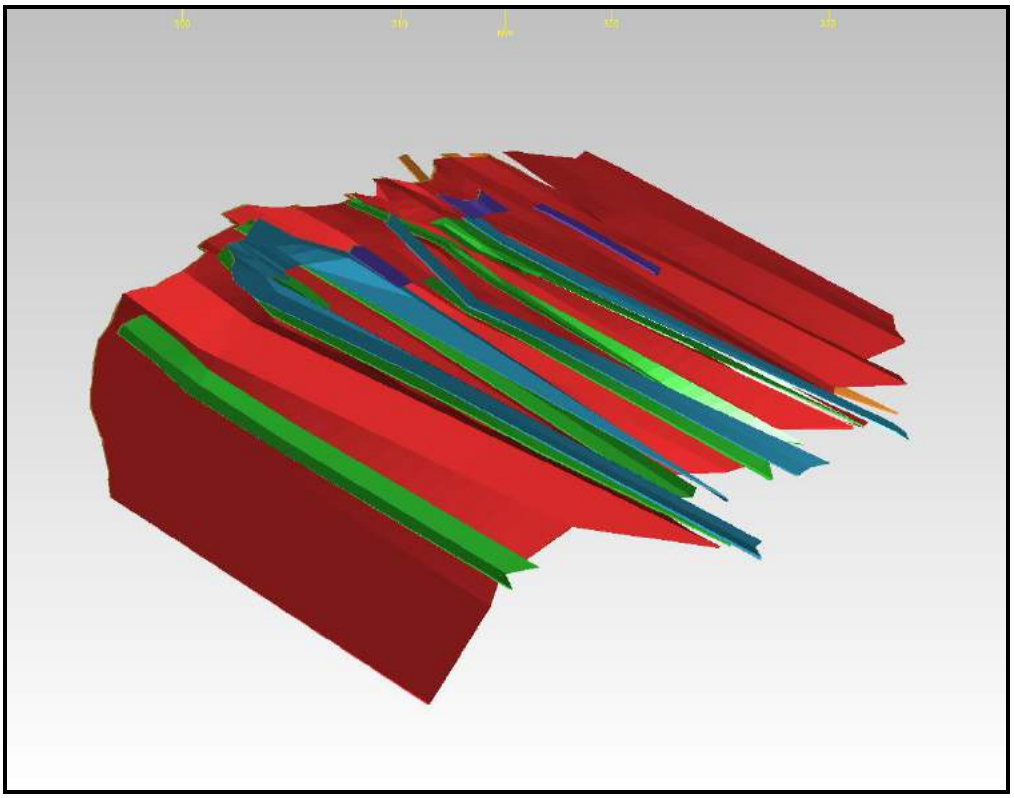
The mineralization of Mangazeisky North is mainly associated with banded brecciated textures. The following minerals are present: native silver, acanthite, stephanite, polybasite, canfieldite, pyrargyrite, miargyrite, diaphorite, tetrahedrite, owyheeite, freieslebenite, boulangerite, arsenopyrite, sphalerite, pyrite, galena, and chalcopyrite.

The veins are strata-bound, occurring on contacts between strata, particularly around sandstone units, and tend to be dilated towards the hinge of the local fold structure. It is noted that the strongest silver mineralisation occurs within the central core of the veins, this also correlates with the zones of strongest oxidation.

Brecciated sandstones have a thickness up to 0.5 m with the contacts of the brecciated zones being sharp and defined by early shearing. The shear commonly exhibits slickensides with horizontal striations. Brecciation was also accompanied by silicification of the finely laminated fine-grained sandstones. The cement is represented by medium grained quartz and fine-grained light grey siderite. The interpretation of tectonic stress fields for the southern flank of the Vasilievsky Zone (Kostin et al. 1997) indicates that mineralised shoots formed in areas where both vertical and horizontal displacement occurred.

Figure 14.2 illustrates the mineralised domain wireframes that have been modelled for Mangazeisky North.

Figure 14.2 Isometric View of Mineralised Domains at Mangazeisky North – Looking Northwest

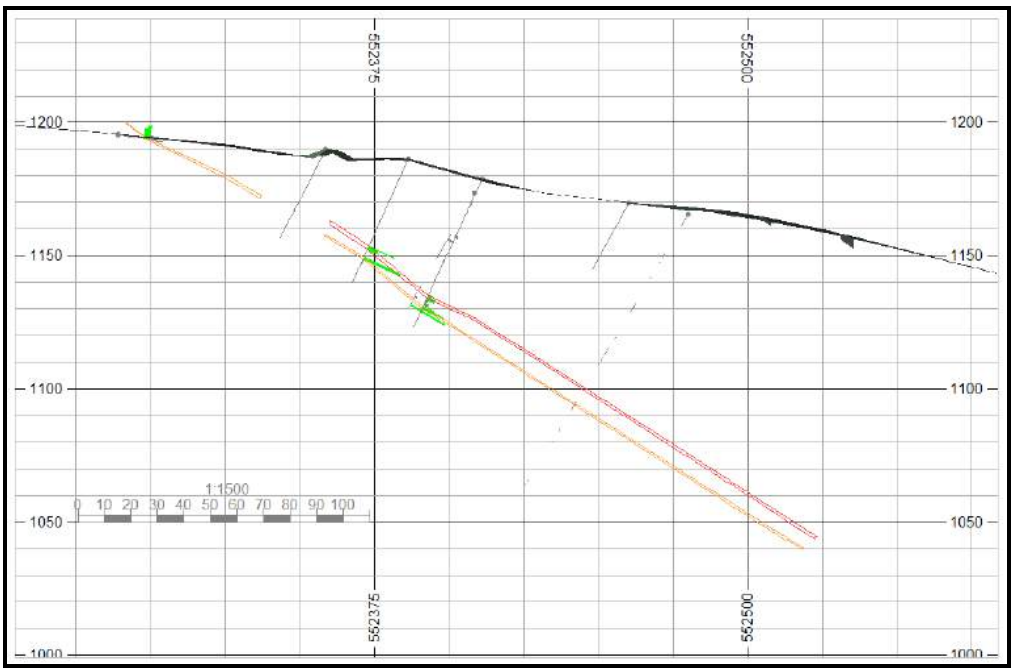


Note: Not to scale
Source: Tetra Tech

14.2.4 WIREFRAME MODELLING

Wireframe models were created by linking sectional interpretations in Datamine. The models were created based upon interval selections that referenced the silver grades, lithological descriptions and structural interpretation. Grades greater than 150 g/t silver were linked together between each drill-section. Where continuity could not be established between sections the strike extrapolation was limited to a half section spacing. The structures were all projected down dip as the data suggests that the mineralisation is open at depth. The wireframes were created with large extrapolations so that the grade interpolation could be limited by the search parameters and the wireframe would not over constrain the estimate. The continuity of the various structures is reflected in the Mineral Resource classification. Figure 14.3 shows a typical cross section of the mineralisation.

Figure 14.3 West-east Cross Section through Mangazeisky North



14.2.5 Exploratory Data Analysis/Domaining

The mineralisation at Mangazeisky North has been treated as a single domain for each correlated vein. For some of the veins that do not have good interpreted continuity, these have been aggregated into a single domain to ensure sufficient sample numbers to interpolate the grade. This has been reflected in the Mineral Resource classification. The exploratory data analysis indicated that there are differences in mineralogy encountered in the mineralisation zones, which control the different grade populations. The veins were separated in order that the grade interpolation does not cross-populated the veins.

Various statistical analyses of the data were performed, and are documented in this section.

Raw Drillhole Statistics

Tetra Tech received a total of 2,127 metal assay results from a series of drilling and trenching campaigns between 2013 and 2016.

Analysis for base metals, including copper, lead, and zinc, was also completed for most of these samples. The base metals have been estimated into the block model for information, but are not currently included within the Mineral Resource as they do not have reasonable prospects for economic extraction, as required for compliant disclosure.

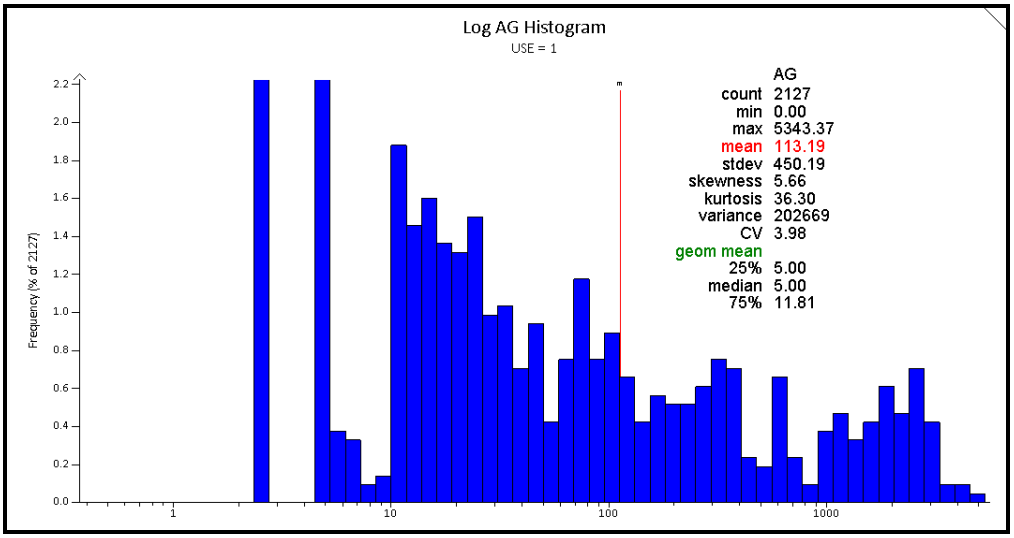
Table 14.2 presents the statistics for all raw silver assays.

Table 14.2 Raw Drillhole Statistics

Field	Ag Grade (g/t)
Count	2,127
Minimum	0
Maximum	5,343.37
Mean	113.19
Variance	202,669
Standard Deviation	450.19
Coefficient of Variation	3.98
Skewness	5.66
Kurtosis	36.3
Geometric Mean	N/A

Statistical analysis of raw samples presents multiple grade populations (Figure 14.4). For robust grade interpolation the populations need to be differentiated and estimated independently.

Figure 14.4 Log Histogram of all Raw Samples



Note: Below detection limit samples are recorded as 2.5 or 5 g/t silver depending on the equipment used.

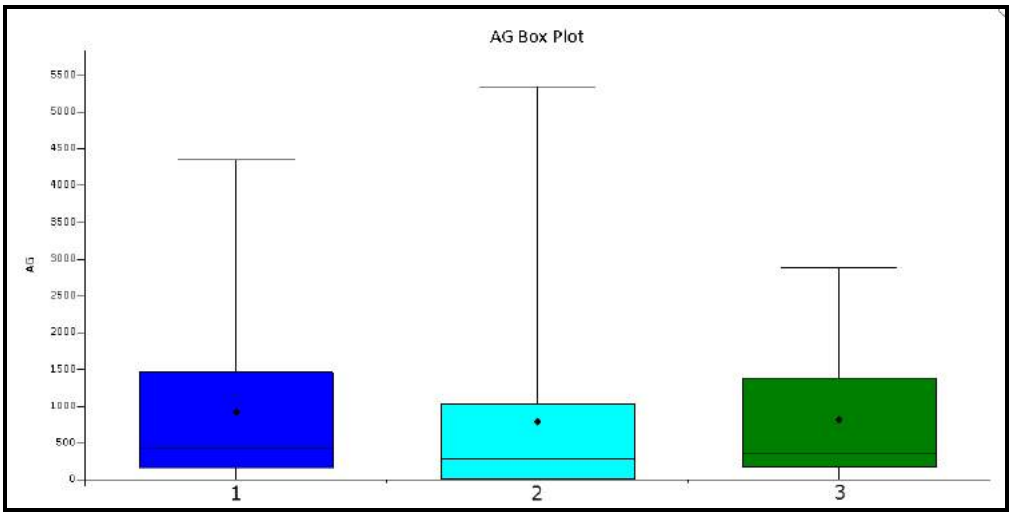
A series of domains constrained by vein interpretation have been identified based upon exploratory data analysis. The domains are described in Table 14.3.

The wireframe models presented in Section 14.3.3, successfully differentiate the high-grade, low-grade, and waste populations. Statistics for the Mangazeisky North domains are presented in the box and whisker plot in Figure 14.5 and descriptive statistics in Table 14.3.

Table 14.3 Descriptive Statistics for Selected Samples

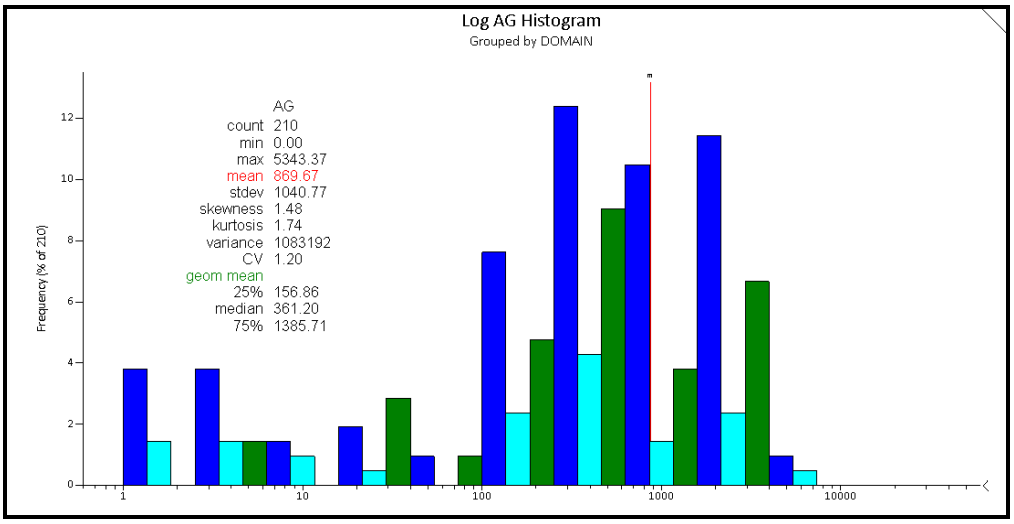
Domain	Count	Minimum	Maximum	Mean	Std Dev	CV	Skewness	Kurtosis	Geom Mean	Median
All	210	0	5,343.37	869.67	1,040.77	1.20	1.48	1.74	NaN	210
1	116	0	4,350.00	921.27	1,066.68	1.16	1.32	0.82	NaN	116
2	32	2.5	5,343.37	788.48	1,178.05	1.49	2.45	6.73	163.55	32
3	62	5	2,881.00	815.04	921.63	1.13	1.17	-0.19	327.05	62

Figure 14.5 Box and Whisker Plot of Selected Raw Samples



The grade distribution in the log histograms and probability plots associated with the selected raw data are reasonably well developed. Figure 14.6 presents a log histogram for all of the selected raw silver assays

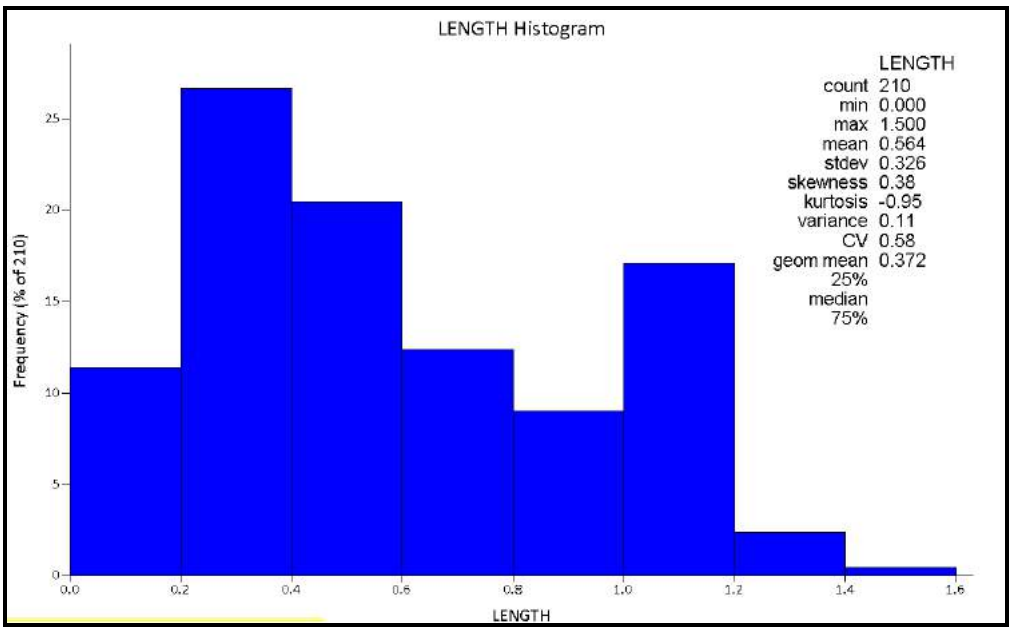
Figure 14.6 Clustered Log Histogram of Selected Raw Samples (All Veins)



SAMPLE LENGTH AND COMPOSITING

Statistics on the sample lengths were analysed using histograms (Figure 14.7). The high variation in length reflects the narrow, but pinching and swelling mineralisation intersections. A length of 0.4 m was chosen as it is close to the geometric mean in the mineralised zone and best preserved the variance and mean grade through the compositing process. Compositing was completed in Datamine using a 0.4 m best fit routine, applying hard domain boundaries.

Figure 14.7 Length Histogram for Selected Raw Samples



The compositing routines have been reasonably effective in preserving the grade distribution, a number of different lengths and methods were tested, but some reduction in grade was unavoidable in this stage (Table 14.4).

Table 14.4 Table Statistics of Selected Raw Samples and 0.4 m Composites

Data	Domain	Count	Minimum	Maximum	Mean	Standard Deviation	Coefficient of Variation	Skewness	Kurtosis	Geom Mean
Selected	All	210	0	5,343.37	869.67	1,040.77	1.20	1.48	1.74	NaN
Composites	All	251	2.5	4,350.00	738.39	869.73	1.18	1.64	2.03	288.16
Selected	1	116	0	4,350.00	921.27	1,066.68	1.16	1.32	0.82	NaN
Composites	1	140	5	4,350.00	858.67	964.41	1.12	1.46	1.34	343.75
Selected	2	32	2.5	5,343.37	788.48	1,178.05	1.49	2.45	6.73	163.55
Composites	2	32	2.5	2,697.80	540.18	805.23	1.49	1.88	1.92	109.63
Selected	3	62	5	2,881.00	815.04	921.63	1.13	1.17	-0.19	327.05
Composites	3	79	5	2,876.00	605.52	670.32	1.11	1.83	2.70	311.81

Top Cuts

The Parrish method of quantile analysis was performed on the composited samples to assess what proportion of the total metal was represented in the top 10% of the data. Once the Parrish method was completed, log probability plots for the composites were referenced to select a suitable value to top-cut the data and reduce the potential bias of the over-representation of the highest-grade samples.

The Mangazeisky North data set did not have undue bias in the high grade bins.

14.2.6 Density

Sixty-two specific gravity measurements were provided by Silver Bear for the Mangazeisky North deposit. The density was interpolated into the block model by IDW^2 method for Mangazeisky North. A mean value of 3.18 g/cm^3 was used for all mineralised blocks that did not received an estimate grade due to the sample numbers.

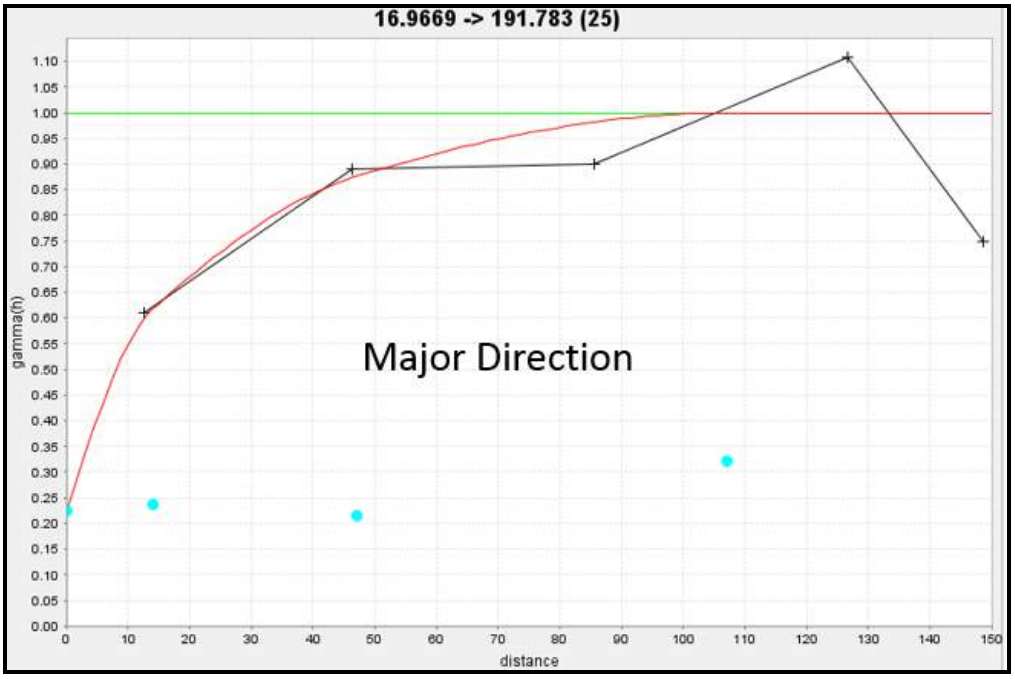
14.2.7 Variography

Variography was completed for the Mangazeisky North data. The updated dataset did not add any further confidence on previous estimates.

A good variogram model fit was achieved for downhole, major, semi-major directions.

The major direction is shown in Figure 14.8.

Figure 14.8 Major Direction Variogram Model Fit



The variogram parameters for the model grade interpolation are shown in Table 14.5.

Table 14.5 Variogram Parameters

Parameter	Value
Angle 1	71
Angle 2	17
Angle 3	25
Axis 1	Z
Axis 2	Y
Axis 3	X
Nugget	0.22
Structure	Spherical
Structure 1	
Range X	12
Range Y	5.7
Range Z	5
C	0.4
Structure 2	
Range X	46
Range Y	21.8
Range Z	10
C	0.3
Structure 3	
Range X	95
Range Y	45
Range Z	15
C	0.08

14.2.8 Resource Block Models

The block model was constructed in Datamine. The rotated block model parameters are given in Table 14.6.

Table 14.6 Mangazeisky North Block Model Parameters

Type	X	Y	Z
Minimum Coordinates	551980	7290650	1000
User Block Size (m)	20	20	5
Minimum Block Size (m)	1	1	0.25
Rotation (degrees)	n/a	n/a	60 (azimuth)

Standardised sub-cell splitting to the minimum block sizes presented in Table 14.6 was employed. Sub-cells received parent cell grades during estimation. The larger parent cell is selected for best kriging performance during the grade interpolation, whilst the smaller sub-cell allows the narrow wireframes to fill with blocks and help to maintain consistency between the final block volumes and the wireframe volume. A

smaller Z axis cell was chosen to as the model will be used for mine design and consistency with bench height was preferable.

INTERPOLATION STRATEGY

Grades were estimated using OK, adopting a multi-pass methodology. A summary of the estimation strategy is show in Table 14.7.

Table 14.7 Estimation Parameters

Pass Number	Samples Numbers		Search (m)	Descritisation	Maximum Samples per ID
	Minimum	Maximum			
1	6	20	60	x3,y3,z2	2
2	6	20	90	x3,y3,z2	2
3	1	20	240	x3,y3,z2	2

14.2.9 BLOCK MODEL VALIDATION

Block model validation was completed using graphical and statistical methods, to confirm that the estimated block model grades appropriately reflect the local composite grades. This is completed primarily by statistical and swath plot methods.

QKNA methods were employed for the kriged parts of the model estimate to test parameters such as block size, discretisation, and informing sample numbers.

The visual inspection demonstrated reasonable correlation between composite and block grades. Table 14.8 presents table statistics comparing the informing composites with the block.

A number of the measures indicate a reduction in variance. This is a result of the change of support associated with the estimation process and the kriging interpolation. There is a reduction in the mean silver grade from composites to block estimate due to the skewness in the data, as the composite data does not conform to a log-normal distribution. Domain 2 shows an increase in grade and over-estimation from the fewest number of informing samples. Overall, for all zones combined, the statistics present reasonable conformance for the stage of the project.

Table 14.8 Statistics Comparing Block Estimate and Composite Grades

Data	Domain	Count	Minimum	Maximum	Mean	Standard Deviation	Coefficient of Variation	Skewness	Kurtosis	Geom Mean	Median
Block Model	All	251	2.5	4350	738.39	869.73	1.18	1.64	2.03	288.16	355
Composites	All	209326	0	2697.8	692.27	437.83	0.63	1.58	4.09	NaN	624.05
Block Model	1	140	5	4350	858.67	964.41	1.12	1.46	1.34	343.75	395.07
Composites	1	128325	5	2011.23	698.96	298.35	0.43	0.77	1.33	604.49	655.7
Block Model	2	32	2.5	2697.8	540.18	805.23	1.49	1.88	1.92	109.63	266.19
Composites	2	33466	61.06	2697.8	999.03	733.84	0.73	0.8	-0.59	740.43	826.1
Block Model	3	79	5	2876	605.52	670.32	1.11	1.83	2.7	311.81	358.4
Composites	3	251	2.5	4350	738.39	869.73	1.18	1.64	2.03	288.16	355

SWATH PLOTS

Swath plots have been used to assess the differences and similarities between the block estimate silver grade and the informing composite grades (Figure 14.9 and Figure 14.10).

Figure 14.9 Easting Swath Plot Comparing the Informing Composite and the OK Estimated Silver Grades for Mangazeisky North

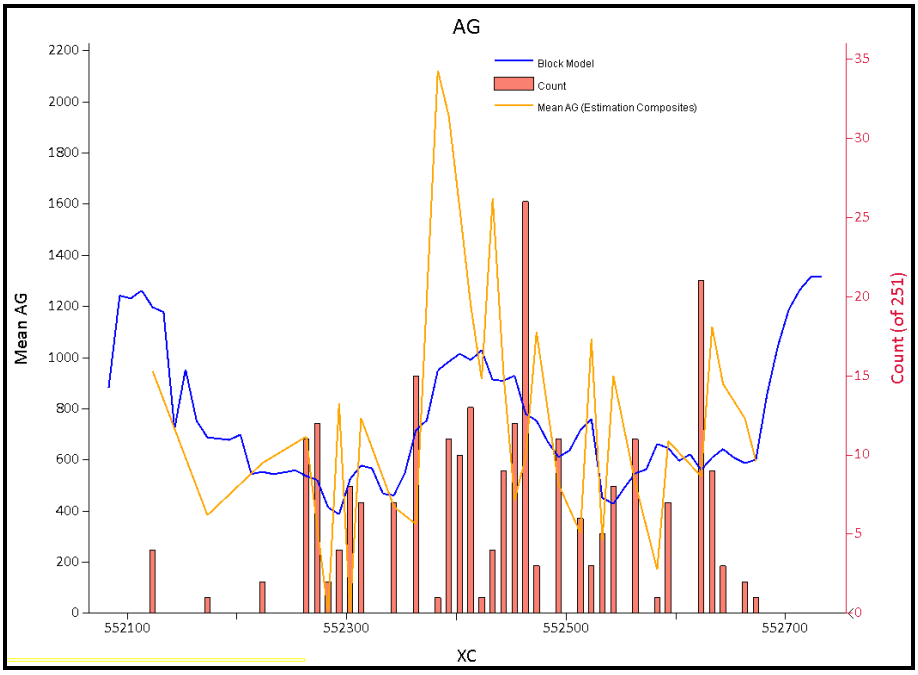
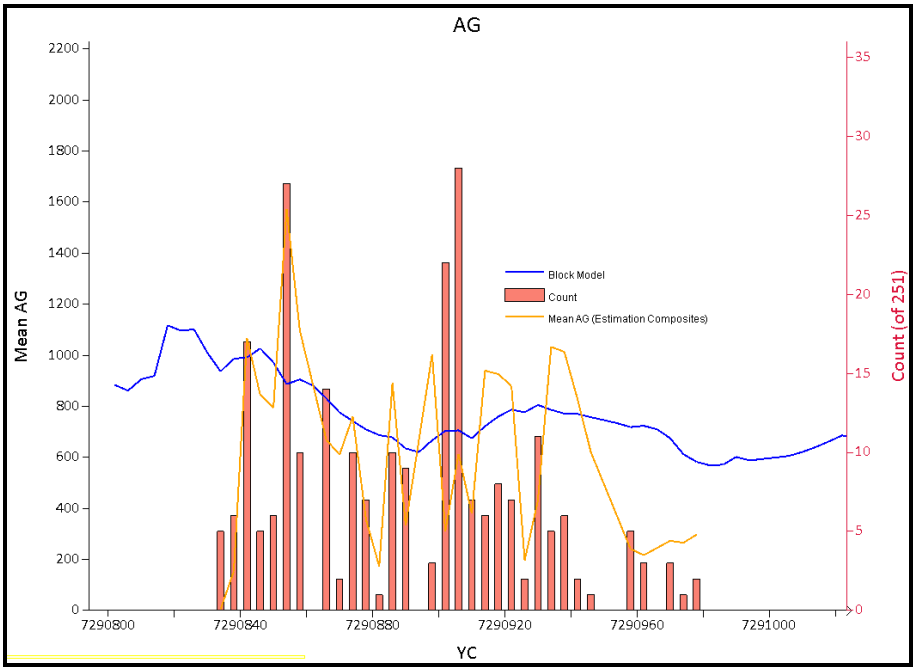


Figure 14.10 Northing Swath Plot Comparing the Informing Composite and the OK Estimated Silver Grades for Mangazeisky North



The swath plots for Mangazeisky North present good conformance between informing composites and estimated block grades. Several iterations of the interpolation were run to achieve optimal conformance for the amount of data available.

Conclusion

The various comparators described in the foregoing subsections serve to illustrate that the block model estimates are robust and satisfactorily models the distribution and variability of the informing sample grades without undue bias or smoothing. The models are suitable for the current level of development.

14.2.10 Mineral Resource Classification

The Mineral Resource model was classified according to CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014).

The Mineral Resources Mangazeisky North are classified as Indicated and Inferred. The mineralisation has been identified and measurements and sampling completed to a reasonable degree of confidence for two of the structures present, but the data is insufficient to allow the geological and grade continuity to be confidently interpreted for the remaining structures. It would be reasonable to expect that some of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration; however, due to the uncertainty of Inferred Mineral Resources it should not be assumed that such upgrading will always occur.

Confidence in the estimate of Inferred Mineral Resources is not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning in Prefeasibility or Feasibility studies.

14.2.11 Mineral Resource Tabulation

Cut-off Grades

In order to demonstrate that the deposits have reasonable prospects for economic extraction a cut-off grade of 150 g/t silver was applied for Mineral Resources at Mangazeisky North.

The parameters considered for cut-off grade derivation, based in part on the parameters established during studies on the Property's other mineral deposits, use the following assumptions:

- silver price of $17.00/troy oz
- mining recovery 95%
- waste dilution 30%
- mining cost of $2.53/t
- processing cost of $52.00/t
- general and administration costs of $40.60/t
- royalty of 6.5%
- overall metal recovery of 88%.

Mineral Resource Tabulation

The updated Mineral Resource for Mangazeisky North deposit is summarised in Table 14.9. The effective date of the updated Mineral Resource is 19th October 2016.

Total Mineral Resources for the Property are detailed in Table 14.10.

Table 14.9 Summary of Mineral Resources – Mangazeisky North

Zone	Indicated Resource			Inferred Resource		
	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)
Mangazeisky North	334,000	770	8,250,000	127,000	560	2,300,000

Table 14.10 Summary of the Mangazeisky Property Resources

Zone	Indicated Resource			Inferred Resource		
	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)
Vertikalny Central	700,000	1,227	27,700,000	350,000	786	8,900,000
Vertikalny Northwest	-	-	-	200,000	476	3,100,000
Nizhny Endybal	-	-	-	710,000	316	7,200,000
Mangazeisky North	334,000	770	8,250,000	127,000	560	2,300,000
Mangazeisky South	-	-	-	60,000	246	500,000
Sterzhnevoy	-	-	-	48,000	1,530	2,360,000
Total	**1,034,000**	**1,079**	**35,950,000**	**1,495,000**	**504**	**24,360,000**

Zone	Inferred Resource						
	Tonnes (t)	Grade Au (g/t)	Grade Ag (g/t)	Grade Cu (%)	Contained Metal Au (troy oz)	Contained Metal Ag (troy oz)	Contained Metal Cu (lb)
Porphirovy	80,000	1.7	46	0.6	4,000	115,000	1,000,000

Notes: The effective date of the Sterzhnevoy and Porphirovy maiden resources is August 27, 2016. The effective date of the Vertikalny Central resource is July 8, 2016 and the Vertikalny Northwest Resource is February 10, 2015. The effective date of the original Nizhny Endybal Resource estimate was September 11, 2012, this resource was re-stated with a higher cut-off grade on the June 10, 2015. The effective date of the Mangazeisky North is October 19, 2016 and the Mangazeisky South Mineral Resource is June 10, 2015.
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Although Silver Bear and Tetra Tech are not aware of any material barrier to eventual economic extraction.

Geological and Sampling Procedures

A quality assurance/quality control program is part of the drilling program on the Mangazeisky deposits. This program includes chain of custody protocols as well as systematic submittals of standards, duplicates, and blank samples into the flow of samples produced by the drilling. Prior to 2011, analysis was carried out at Russian certified laboratories in Yakutsk and Aldan, Russia. Analysis for 2011, 2012, 2013, 2014, and 2015 campaigns were completed by ALS Chemex in Chita, Russia.

Table 14.11 Mineral Resource Estimation Assumptions and Methods

Prospect Area	Key Assumptions
Vertikalny Central and North West	• The Mineral Resources were estimated into two separate 3D block models comprising the following parameters: Vertikalny Central: 10 m x 10 m x 1 m (x, y, z), with minimum sub-block dimensions of 2.5 m x 2.5 m x 0.25 m (x, y, z). Vertikalny Northwest: 10 m x 10 m x 1 m (x, y, z), with minimum sub-block dimensions of 2.5 m x 2.5 m x 0.25 m (x, y, z). • The estimation was constrained to the mineralised zone using wireframe solid models. The wireframes were sub-domained to isolate the strongly mineralised main zone from the silver mineralisation associated with wall rock alteration. • Grade estimates were based on 0.7 m composited assay data. • The interpolation of the metal grades was undertaken using OK. • In order to demonstrate that the deposit has reasonable prospects for economic extraction, a cut-off grade of 200 g/t silver was applied for Mineral resources contained within an open pit shell at Vertikalny Central or within 80m of surface at Vertikalny Northwest. A 350 g/t silver cut-off has been applied for the underground Mineral Rresource. • At Vertikalny Central, a pit shell wireframe was used to constrain the open pit Mineral Resource in order to demonstrate that the Mineral Resource has reasonable prospects for economic extraction. The pit shell uses the following assumptions: • silver price of $19.00/troy oz • mining recovery 95% • waste dilution 30% • mining cost of $2.53/t • processing cost of $52 /t • G&A cost of $40.6 /t • royalty of 6.5% • overall silver recovery of 88%.
Nizhny Endybal	• In order to maintain the Nizhny Endybal Resource as a current Mineral Resource, Tetra Tech has restated the Mineral Resource with the following assumptions: • cut-off grade of 150 g/t silver • spot metal prices effective February 10, 2015 of $17.00/oz troy oz • mining cost of $3.16/t • processing cost of $50/t • G&A cost of $40/t • overall silver recovery of 90%.

table continues...

Prospect Area	Key Assumptions
Mangazeisky North & South	• The Mineral Resources were estimated into two separate 3D block models each comprising the following parameters: 20 m x 20 m x 5 m (x, y, z), with minimum sub-block dimensions of 1 m x 1 m x 0.5 m (x, y, z). • The estimation was constrained to the mineralised zone using wireframe solid models. The wireframes were sub-domained to isolate each mineralised structure. • Grade estimates were based on 0.5 m composited assay data. • The interpolation of the metal grades was undertaken using OK. • In order to demonstrate that the deposit has reasonable prospects for economic extraction a cut-off grade of 150 g/t was applied for Mangazeisky North, based upon the following assumptions: • silver price of $17.00/troy oz • mining recovery 95% • waste dilution 30% • mining cost of $2.53/t • processing cost of $52.00/t • G&A cost of $40.60/t • royalty of 6.5% • overall metal recovery of 88%.
Sterzhnevoy and Porphirovy	• The Mineral Resources were estimated into two separate 3D block models each comprising the following parameters: 20 m x 20 m x 10 m (x, y, z), with minimum sub-block dimensions of 1 m x 1 m x 0.5 m (x, y, z). • The estimation was constrained to the mineralised zone using wireframe solid models. The wireframes were sub-domained to isolate each mineralised structure. • Grade estimates were based on 0.5 m composited assay data. • The interpolation of the metal grades was undertaken using IDW. • In order to demonstrate that the deposit has reasonable prospects for economic extraction a cut-off grade of 150 g/t was applied for Sterzhnevoy, based upon the following assumptions: • silver price of $17.00/troy oz • mining cost of $2.50 /t • processing cost of$50 /t • G&A cost of $40/t • overall silver recovery of 90% • a cut-off grade of $77.00 equivalent based on the same assumptions for Porphirovy; dollar equivalent calculated with the formula = Au ozt x $1250 + Ag ozt x $17 + Cu lbs x $2.2

Note: Assumes that the Mangazeisky deposits will be worked with Vertikalny and will therefore share G&A costs.

GRADE TONNAGE SENSITIVITY

The Mangazeisky North block model has been reported at a range of cut-offs to provide an indication of the grade-tonnage sensitivity Table 14.12 shows the sensitivity for the combined Indicated and Inferred Resources.

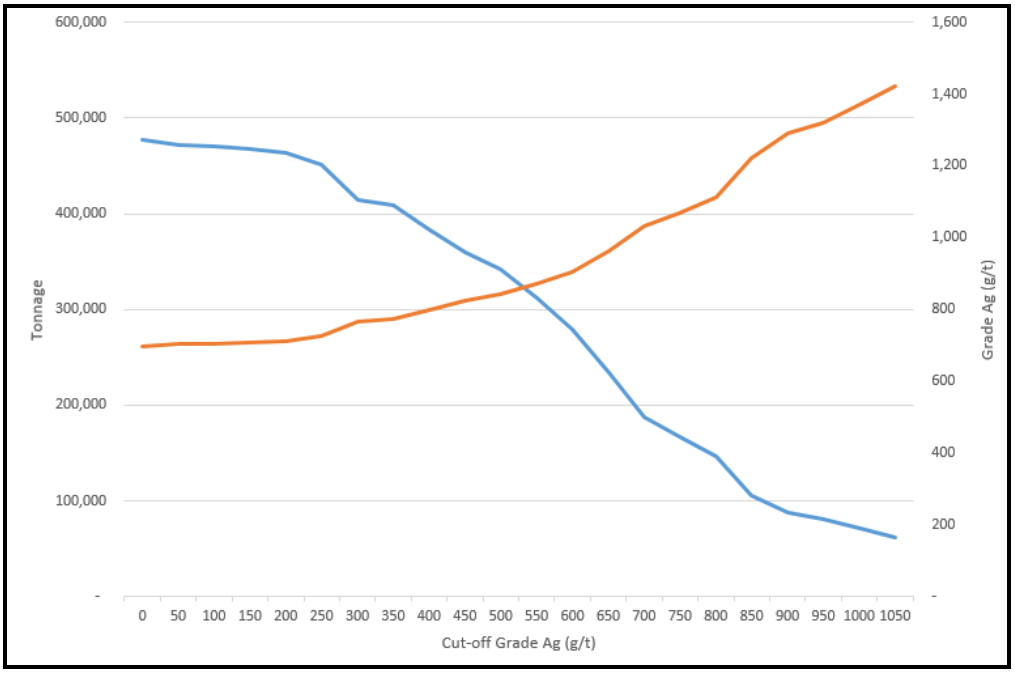
Table 14.12 Mangazeisky North – Grade-Tonnage Sensitivity

Cut-off Ag Grade (g/t)	Tonnage (kt)	Ag Grade (g/t)	Contained Metal Ag (koz)
100	470,700	704	10,660,000
125	469,500	706	10,660,000
150	461,000	709	10,640,000
175	464,600	711	10,630,000
200	463,700	712	10,630,000
225	455,300	722	10,570,000
250	451,000	726	10,540,000
275	431,300	748	10,370,000
300	414,700	766	10,210,000

Grade-Tonnage Curves

Figure 14.11 depicts the grade-tonnage curve for the Mangazeisky North deposit's Indicated and Inferred Mineral Resources.

Figure 14.11 Grade-Tonnage Curve for Indicated and Inferred Resources at Mangazeisky North



Tetra Tech is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that could materially affect the Mineral Resource.

14.3 VERTIKALNY MINERAL RESOURCE ESTIMATE

14.3.1 SUMMARY OF ESTIMATION TECHNIQUES

Tetra Tech created wireframe models using Leapfrog Geo (Version 3.1). Wireframes were created to represent lithologies, oxidation state and mineralised bodies. Wireframes were developed for strongly mineralised core zones and where present less strongly mineralised wall rock alteration (as described in Section 14.2.1). Variography, block modelling and resource estimation were completed in GEOVIA Surpac software (version 6.7).

The silver grades for the Vertikalny Central and Northwest zones were estimated using OK interpolation methodology. Density for the Central Zone was estimated using IDW^2, while density was assigned for the Northwest Zone.

Statistical and grade continuity analyses were completed in order to characterise the mineralisation, and were subsequently used to develop grade interpolation parameters. Grade estimation was completed using OK. The search ellipsoid dimensions and orientations were chosen to reflect the continuity revealed by geostatistical studies. Block size, discretisation, search size, and sample numbers were optimised using QKNA.

Tetra Tech adopted the definition of Mineral Resources as outlined within the CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014) in order to classify the resources.

In order to demonstrate that the deposit has reasonable prospects for economic extraction, a cut-off grade of 200 g/t silver has been applied for resources contained within an open pit shell at Vertikalny Central or within 80 m of surface at Vertikalny Northwest. A 350 g/t silver cut-off has been applied for the underground resource, based upon the following assumptions:

- silver price of $17.00/troy oz
- mining cost $2.53/t open pit and $70.00/t underground
- processing cost of US$52.00/t
- general and administration costs of US$40.60/t.

At Vertikalny Central, a pit shell wireframe was used to constrain the open pit resource in order to demonstrate that the resource has reasonable prospects for economic extraction. The pit shell was developed using the following assumptions:

- silver price of $19.00/troy oz
- mining recovery 95%
- waste dilution 30%
- mining cost of US$2.53/t
- processing cost of US $52.00/t
- general and administration costs of US$40.60/t

- royalty of 6.5%
- overall silver recovery of 88%.

In order to demonstrate that the deposit has reasonable prospects for economic extraction, a cut-off grade of 150 g/t was applied for the Mangazeisky North and Nizhny Endybal deposits, based upon the following assumptions:

- forecasted silver price of $17.00/troy oz open pit mining cost of $3.16/t
- processing cost of $50/t
- general and administrative costs of $12.50/t
- transport costs, from pit to process plant, of $3.60/t
- overall silver recovery of 80%.

14.3.2 DATABASE

Silver Bear provided Tetra Tech with its exploration database, in the form of excel spreadsheet containing the following tabs:

- collar
- survey
- assay
- density
- geotechnical
- lithology.

The database include information on trenches and drillholes sampled during the period from 2007 through to the summer 2015.

The resource update for Vertikalny Central benefited from an additional 15 diamond holes drilled in 2014, but assayed in 2015, 6 diamond holes drilled and assayed in 2015 and 21 trenches that were excavated in 2014 but assayed in 2015. An additional eight holes were drilled on the Vertikalny Northwest deposit in 2015.

14.3.3 GEOLOGICAL INTERPRETATION

The Vertikalny deposit is interpreted to be a breccia and vein hosted, epithermal silver deposit, hosted within a relatively vertical, northwest-southeast trending fault zone. The mineralisation is associated with vertical dykes of dacitic composition. Based on the surrounding deposits and the host rock, it is interpreted that the Vertikalny deposit formed through the re-mobilisation of strata-bound silver-lead-zinc mineralisation, into breccia and vein-hosted high-grade deposits of variably oxidized manganosiderite-sphalerite-galena mineralisation, with silver sulphosalts and native silver.

It is interpreted that one of the controls on mineralisation is the interaction between oblique subordinate faults, with the main mineralised structure. Whilst the main structure at Vertikalny Central is mineralised along its entire strike length, the highest

grade zones appear to be coincident with the intersection between the mineralised body and oblique fault plane.

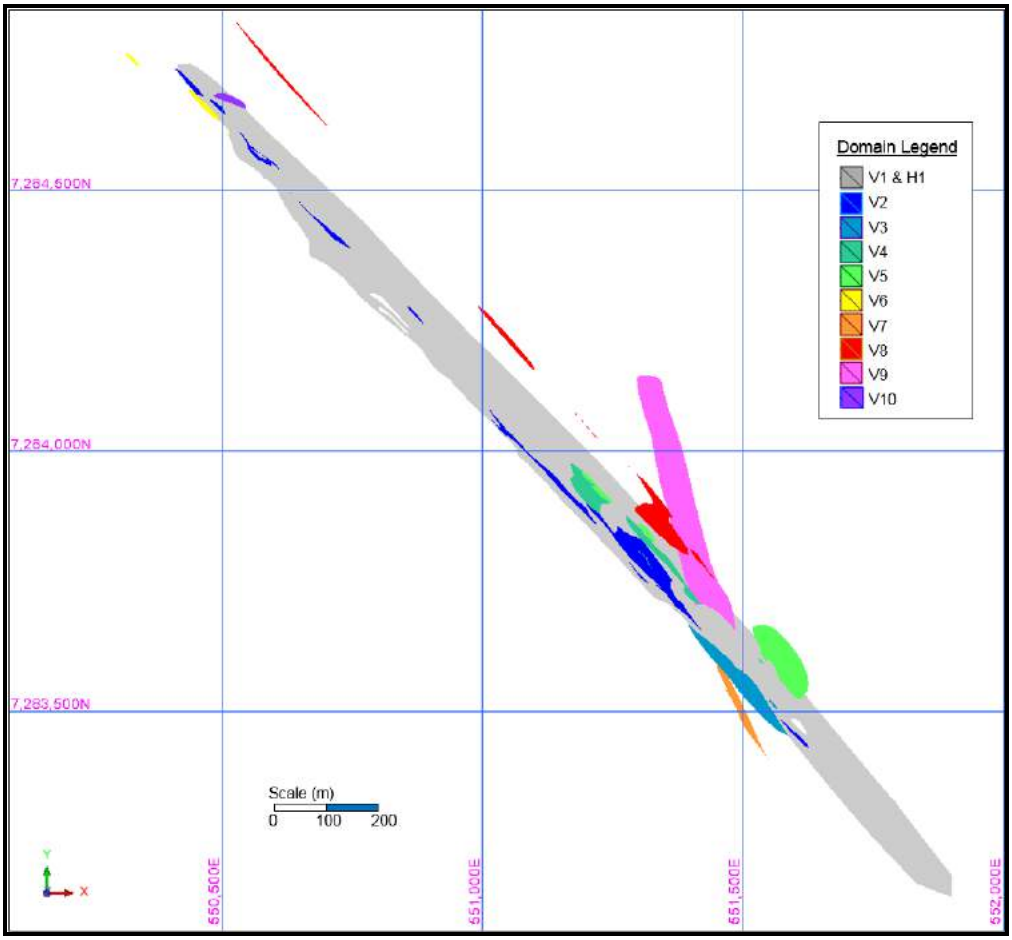
Strongly Mineralised Zone

During the logging of the 2013-2015 seasons, it was noted that within the brecciated/mineralised zone there existed a subset of more strongly mineralised, and hence higher-grade mineralisation. These subset intervals were compared to the assay results and were shown to have a good correlation with the highest-grade intervals in each logged hole. These observations were applied to drillholes and trenches from earlier campaigns by using silver grades to define the strongly mineralised core within the mineralised zone. Less strongly mineralised material was grouped into a wall rock alteration model. The strongly mineralised domain was named V1, with the alteration halo named H1.

Subordinate parallel and Oblique Veins

At Vertikalny Central a series of subordinate parallel and oblique veins have been modelled, in addition to the main V1/H1 domain. The veins have limited continuity, but do contain some pockets of strong silver mineralisation, Figure 14.12.

Figure 14.12 Mineralised Domains at Vertikalny Central



Source: Tetra Tech

WIREFRAME MODELLING

Implicit wireframe models were generated in Leapfrog Geo Version 3.1. The models were created based upon interval selections that referenced both the silver and lead grades, lithological descriptions and structural interpretation. The lead grades provided an indication of the style of mineralisation and abundance of galena in the vein.

The models were created as a series of tightly constrained sub-vertical veins that could pinch and swell according to the informing drillhole data, Figure 14.13 and Figure 14.14.

Figure 14.13 Plan and Cross Section through Vertikalny Central Wireframe Models

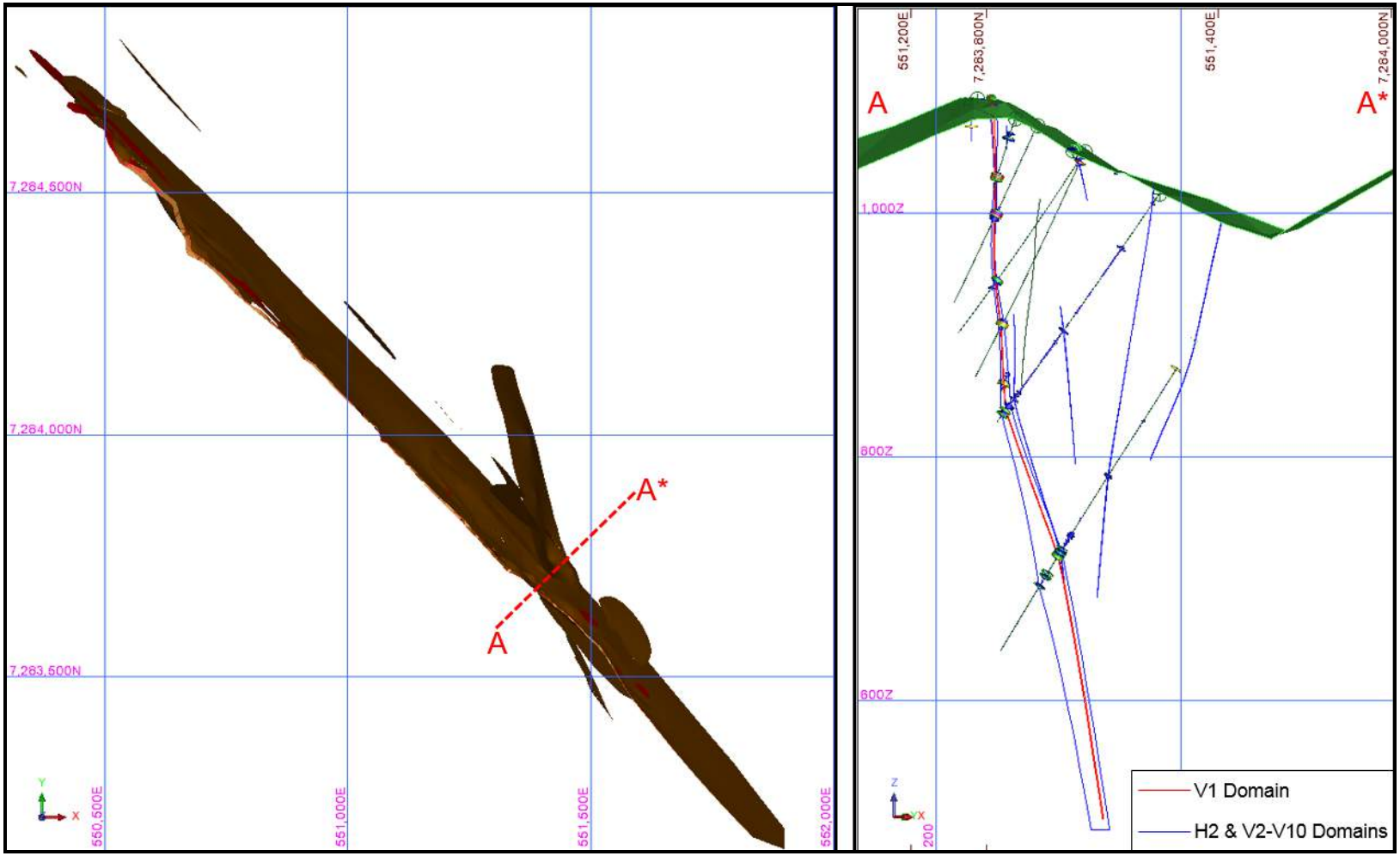
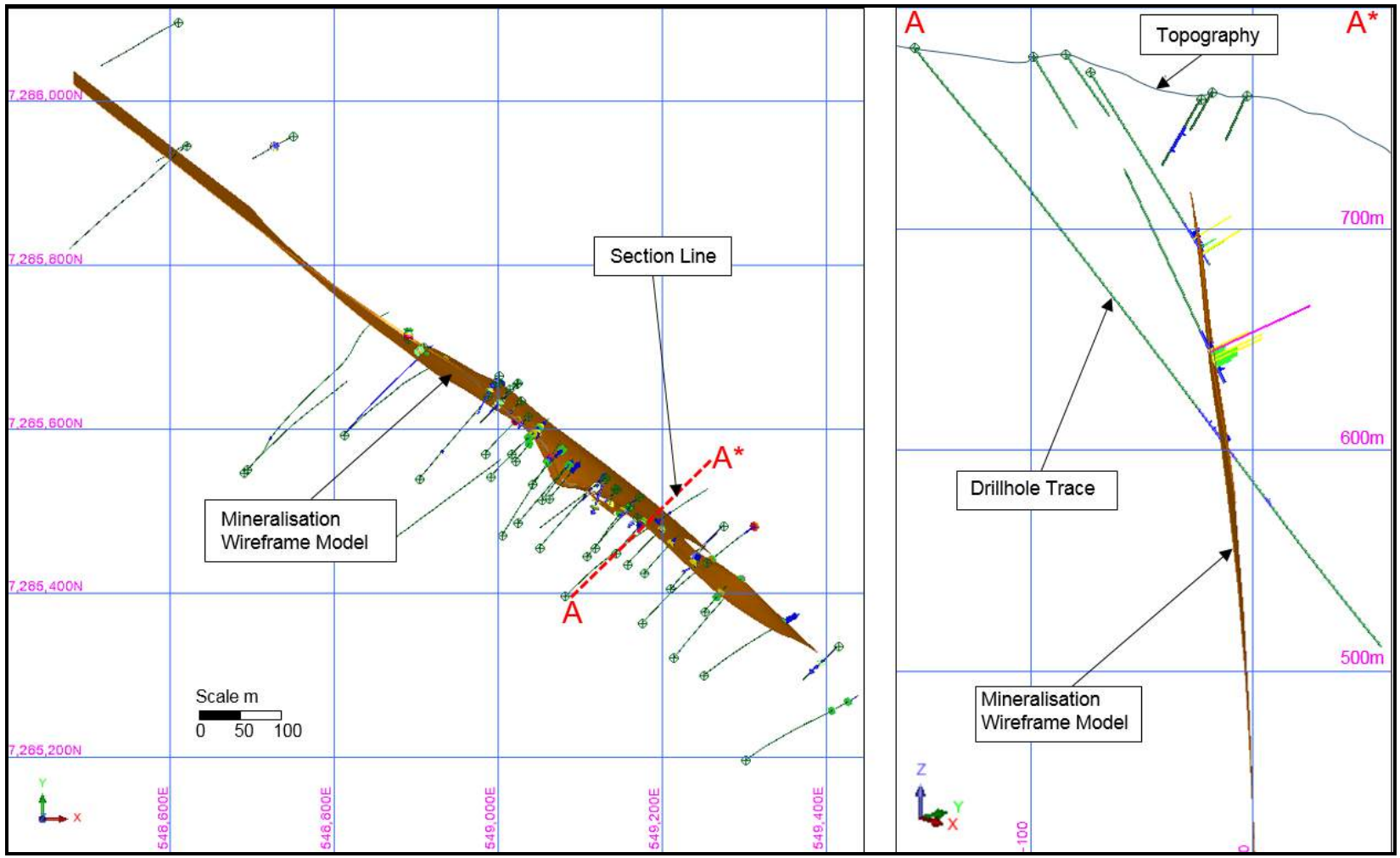


Figure 14.14 Plan and Cross Section through Vertikalny Northwest Wireframe Models



14.3.4 EXPLORATORY DATA ANALYSIS/DOMAINING

The main zone at Vertikalny central has been subdivided into two domains, a high-grade core (domain V1) and a low-grade halo (domain H1), based upon exploratory data analysis (EDA) described herein. The remaining domains at Vertikalny Central include V2, V3, V4, V5, V6, V7, V8, V9 and V10, which represent the subordinate parallel and oblique veins. There is a single domain at Vertiklany Northwest, referred to as VNW1. The EDA indicated that there are differences in mineralogy encountered in the mineralisation zones, which control the different grade populations.

The subordinate parallel/ oblique veins and the Vertikalny Northwest area do not have sufficient sample numbers to allow division into sub-domains.

Various statistical analyses of the data were performed, and are documented in this section.

RAW DRILLHOLE STATISTICS

Tetra Tech received a total of 13,879 silver assay results from a series of drilling and trenching campaigns between 2007 and 2015.

Analysis for base metals, including copper, lead and zinc was also completed for 8,529 samples. The base metals have been estimated into the block model for information, but are not currently included within the mineral resource due to them not having reasonable prospects for economic extraction, as required for compliant disclosure.

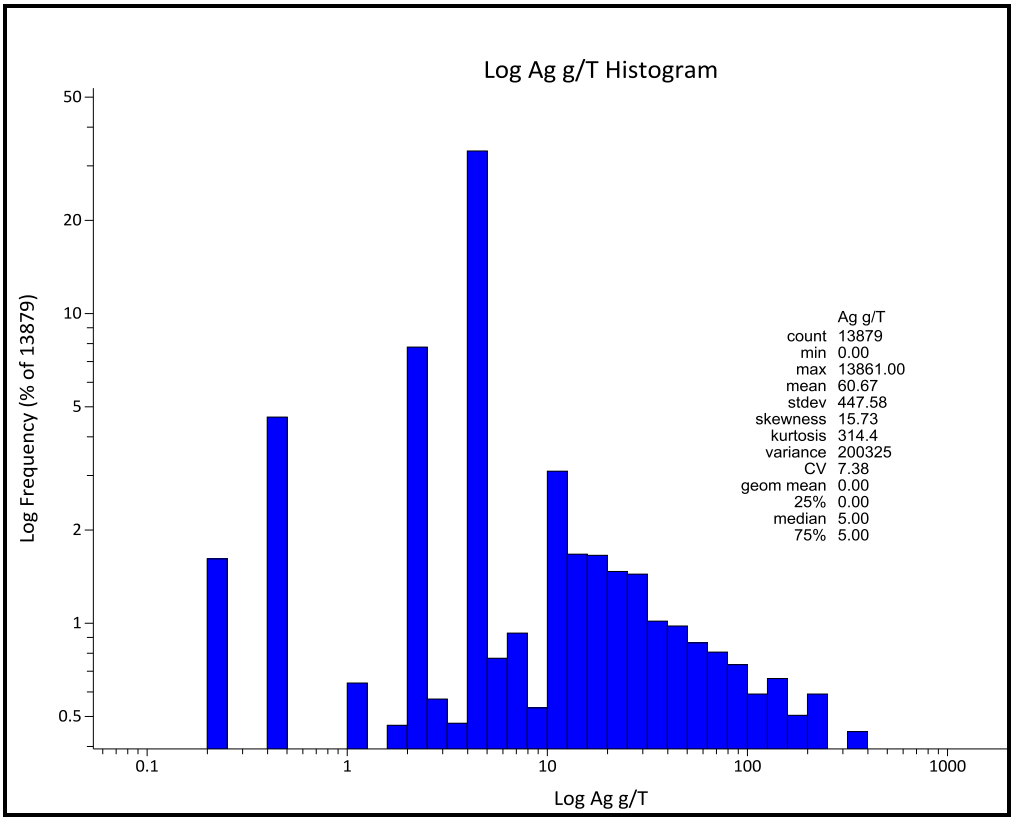
Table 14.13 presents the statistics for all raw silver and density data.

Table 14.13 Raw Drillhole Statistics

Field	Ag Grade (g/t)	Density (t/m³)
No. of Samples	13,879	393
Minimum	0	2.11
Maximum	13,861	6.23
Range	13,861	4.12
Mean	61	3.13
Variance	200,325	0.39
Standard Deviation	448	0.62
Coefficient of Variation	7	0.20

Statistical analysis of raw samples presents multiple grade populations (Figure 14.15). For robust grade interpolation the populations need to be differentiated and estimated independently.

Figure 14.15 Log Histogram of all Raw Samples



A series of domains have been identified based upon exploratory data analysis. The domains are described in Table 14.14.

Table 14.14 Summary of Mineralisation Domains

Domain Name	Area	Description
V1	Vertikalny Central	Sub-vertical persistent strongly mineralised vein at Verticalny central. Constitutes majority of the resource.
H1	Vertikalny Central	Weakly mineralised alteration halo surrounding V1
V2-V10	Vertikalny Central	Subordinate parallel and oblique veins. Generally small narrow veins with poor continuity.
VNW	Vertikalny Northwest	Sub-vertical vein with strong mineralisation in places. Main structure at Vertikalny northwest.

The wireframe models presented in Section 14.3.3, successfully differentiate the high-grade, low-grade, and waste populations. Statistics for the Vertikalny domains are presented in the box and whisker plot in Figure 14.16 and descriptive statistics in Table 14.15.

Figure 14.16 Box and Whisker Plot of Selected Raw Samples

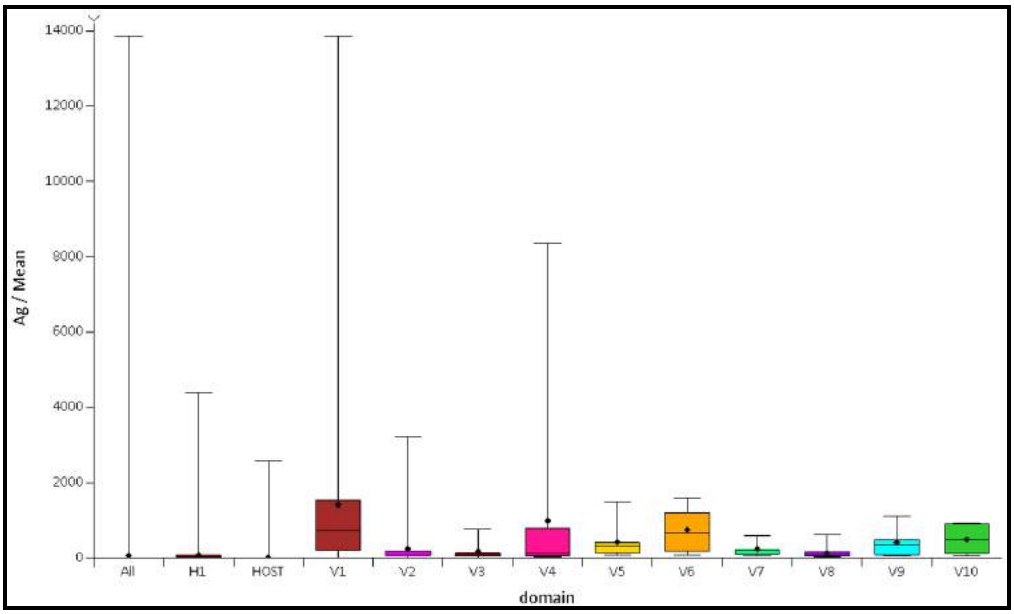


Table 14.15 Descriptive Statistics for Selected Raw Samples

Domain	Count	Minimum	Maximum	Mean	Std Dev	CV	Skewness	Kurtosis	Geom Mean	Median
HOST	11,455	0	2,578	8	56	7.3	24.0	747.5	-	2.5
H1	1,447	0.25	4,407	76	185	2.5	12.3	240.0	28.2	24.6
V1	413	6.8	13,861	1,413	2,076	1.5	2.9	9.4	536.5	722.0
V10	5	61	929	503	413	0.8	0.1	-4.6	318.1	485.0
V2	55	5	3,229	246	541	2.2	4.5	21.2	77.6	76.8
V3	8	7.1	770	180	246	1.4	3.0	8.6	79.0	134.2
V4	14	32.5	8,369	1,001	2,199	2.2	3.7	13.5	221.7	113.5
V5	7	88.36	1,503	432	491	1.1	2.8	7.3	282.3	324.5
V6	4	76.2	1,590	747	718	1.0	0.6	-6.3	417.9	660.5
V7	5	59	589	243	208	0.9	2.1	4.0	181.5	218.0
V8	9	26	621	140	196	1.4	2.7	6.7	72.8	45.0
V9	9	53.6	1,114	410	373	0.9	1.3	0.2	262.1	349.0
VNW	102	5	4,030	369	612	1.7	3.6	15.7	154.7	156.4

The differentiated datasets approximate single log normal populations are shown in Figure 14.17, Figure 14.18, Figure 14.19 and Figure 14.20.

Figure 14.17 Log Histogram of Selected Raw Samples from Domain V1

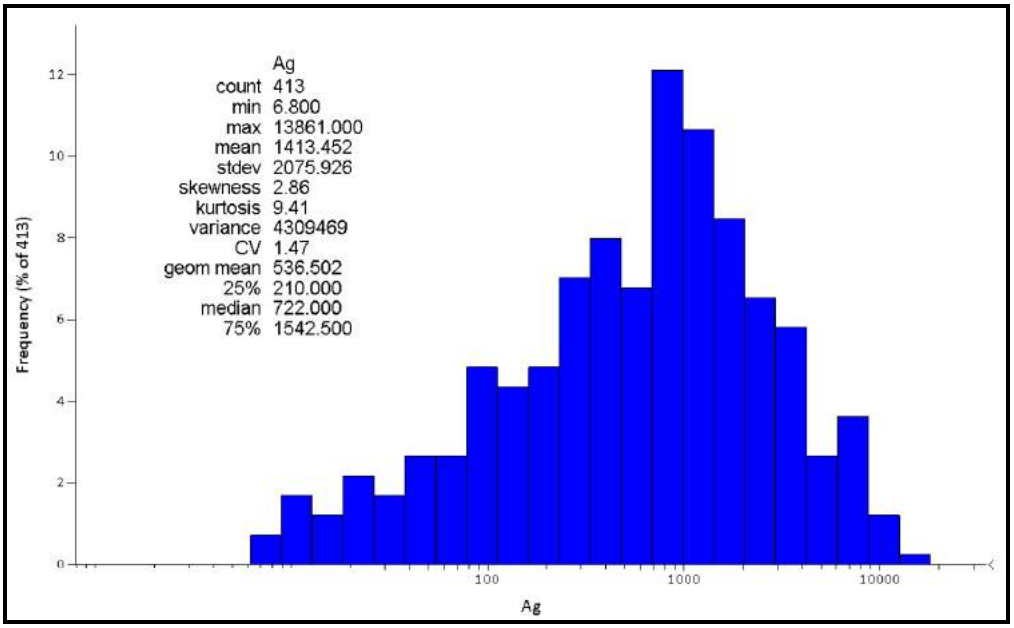
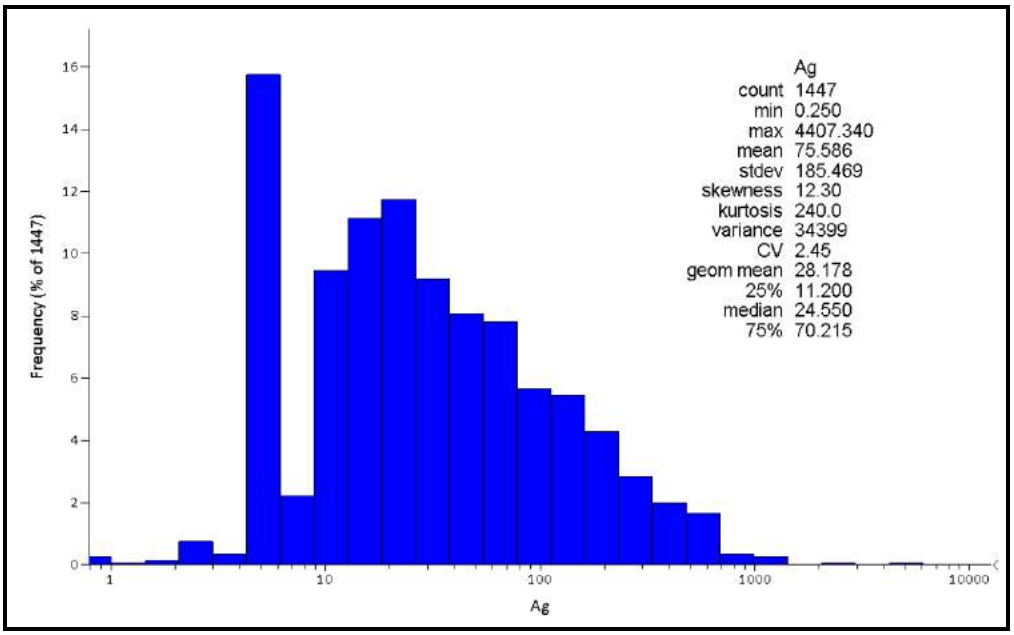


Figure 14.18 Log Histogram of Selected Raw Samples from Domain H1



Notes: For the majority of the samples the analytical detection limit is 5 g/t, this causes a spike in the histogram at this grade.

Figure 14.19 Log Histogram of Selected Raw Samples from Domains V2-V10

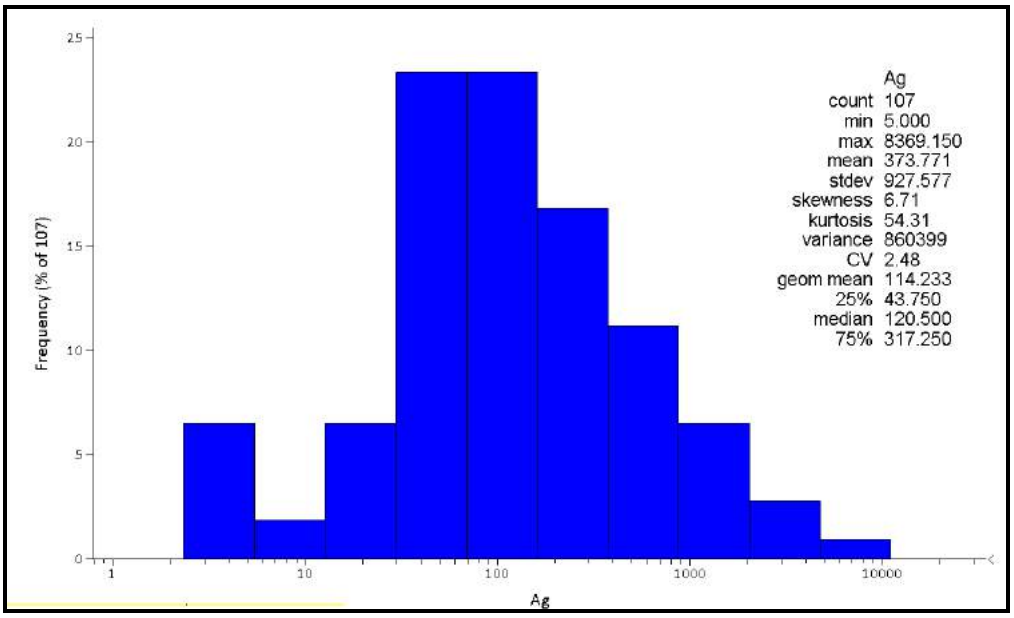
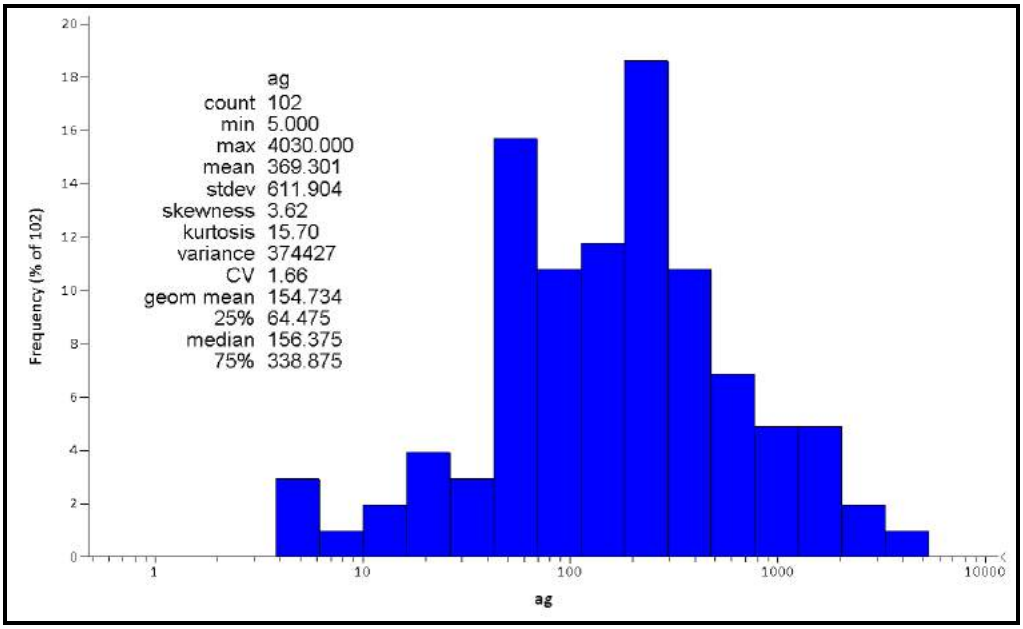


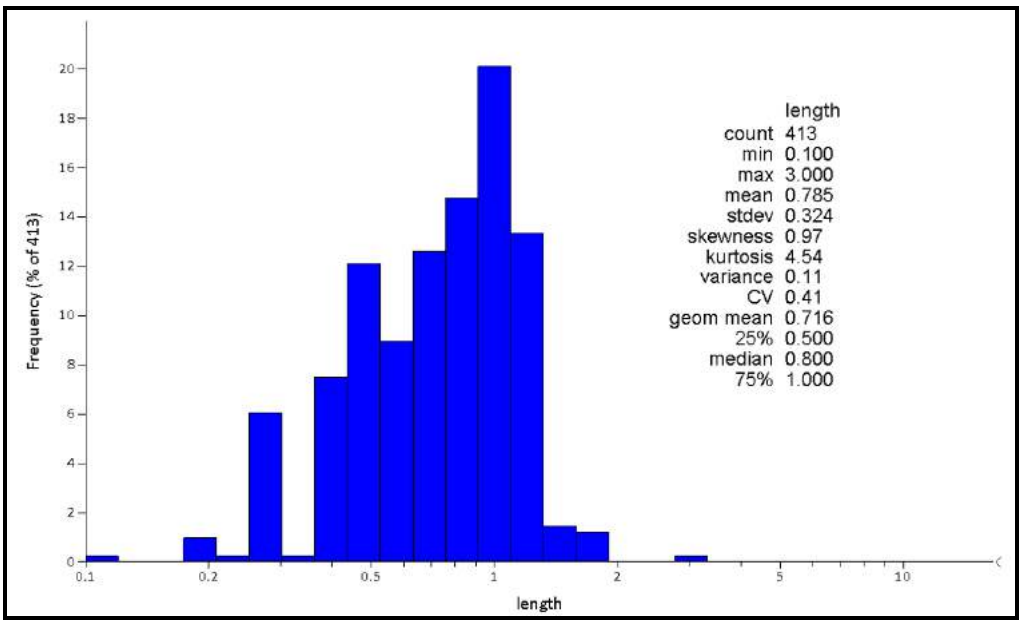
Figure 14.20 Log Histogram of Selected Raw Samples from Domains VNW



Sample Length and Compositing

Statistics on the sample lengths were analysed using histograms (Figure 14.21), and a length of 0.7 m was chosen as it is close to the mean and geometric mean. Compositing was completed in Surpac using a 0.7 m best fit routine, applying hard domain boundaries.

Figure 14.21 Length Histogram for Selected Raw Samples with V1 Domain



The compositing routines have been effective in preserving the grade distribution (Figure 14.21). However, the process has reduced the mean silver grade by 11% from 758 to 672 g/t. The reduction in grade is due to the presence of a number of very high-grade samples that have a length of less than 1 m (Figure 14.22). When the high-grade short samples are combined with lower-grade neighbouring samples to create a length weighted average grade over a standard 1-m length, the result is a small overall reduction in silver grade (Table 14.16).

Figure 14.22 Sample Length Plotted Against Silver Grade

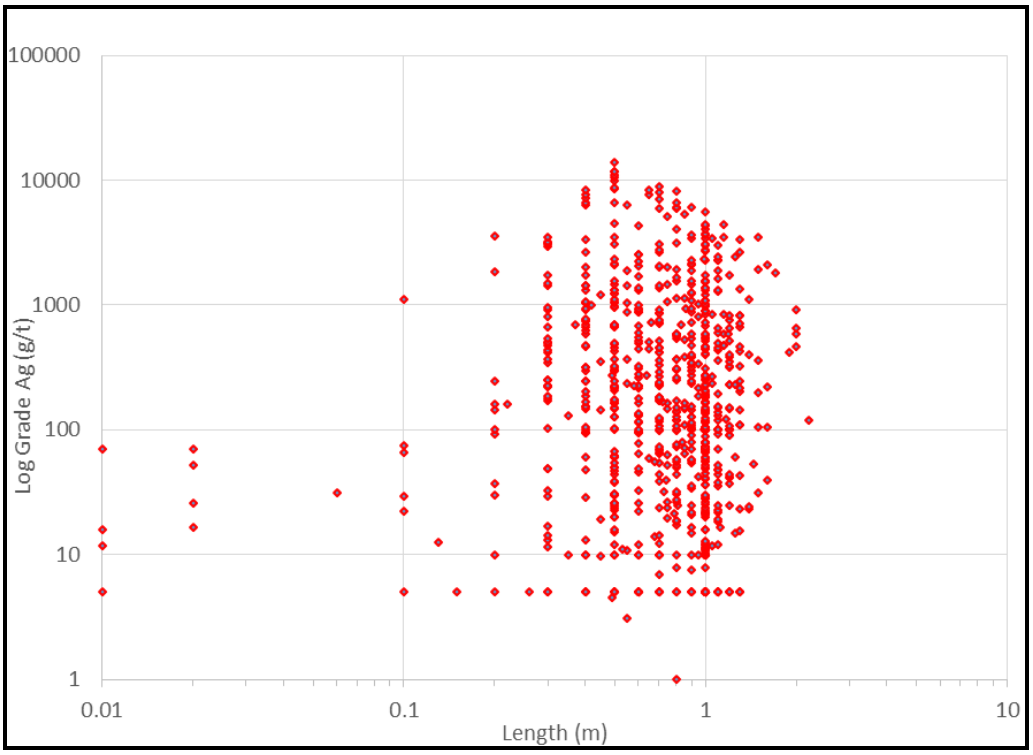


Figure 14.23 Log Histogram of Silver Grades from Selected Raw Samples and 0.7 m Composites for Vertikalny Central Domains

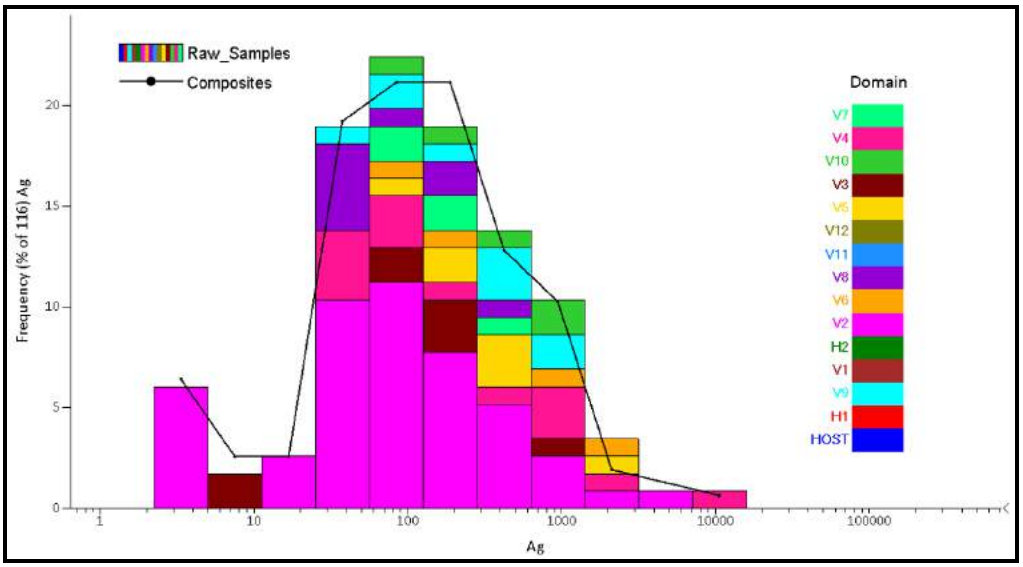


Table 14.16 Table Statistics of Selected Raw Samples and 0.7 m Composites

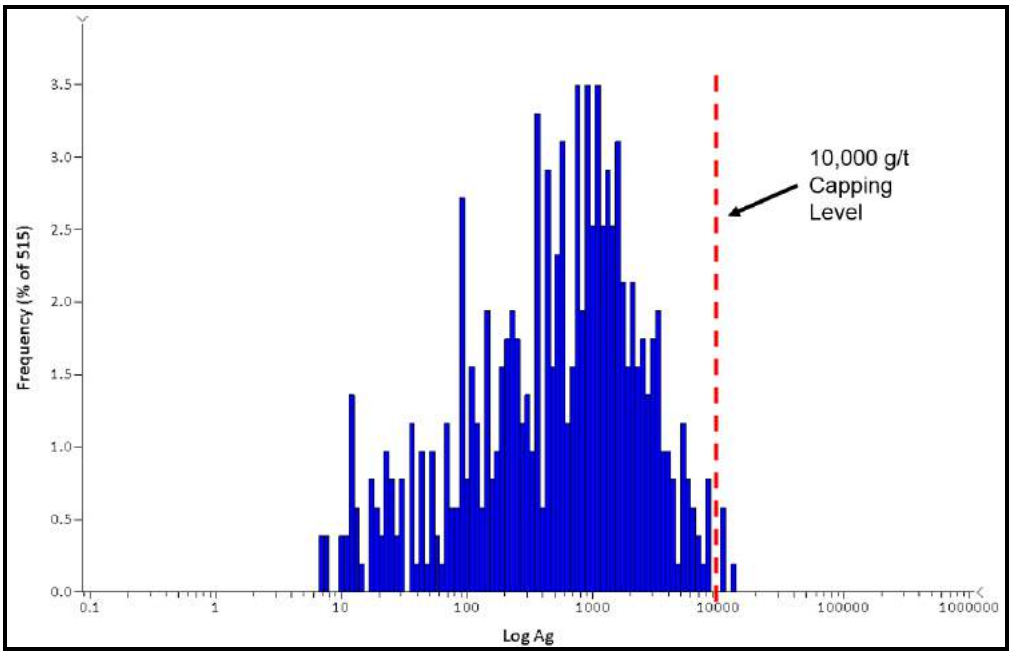
Data	Domain	Count	Min	Max	Mean	Std Dev	CV	Skewness	Kurtosis	Geom Mean
Raw	H1	1447	0.25	4,407	76	185	2.45	12.30	240.00	28.18
Cmps	H1	1770	0.25	4,393	72	177	2.44	13.19	262.50	30.23
Raw	V1	413	6.8	13,861	1,413	2,076	1.47	2.86	9.41	536.50
Cmps	V1	499	11.5	13,054	1,256	1,727	1.38	3.02	11.94	533.77
Raw	V10	5	61	929	503	413	0.82	0.05	-4.58	318.11
Cmps	V10	5	61	929	503	413	0.82	0.05	-4.58	318.11
Raw	V2	55	5	3,229	246	541	2.20	4.45	21.18	77.64
Cmps	V2	75	5	3,229	238	581	2.44	4.49	20.30	70.00
Raw	V3	8	7.1	770	180	246	1.36	3.02	8.59	78.95
Cmps	V3	12	7.1	770	171	226	1.32	2.37	5.17	65.11
Raw	V4	14	32.5	8,369	1,001	2,199	2.20	3.72	13.47	221.70
Cmps	V4	18	32.5	8,369	809	1,920	2.37	4.36	18.55	223.59
Raw	V5	7	88.36	1,503	432	491	1.14	2.82	7.30	282.31
Cmps	V5	9	88.36	1,503	362	448	1.24	3.02	8.65	230.68
Raw	V6	4	76.2	1,590	747	718	0.96	0.56	-6.28	417.92
Cmps	V6	7	76.2	1,590	662	653	0.99	0.66	-2.85	355.64
Raw	V7	5	59	589	243	208	0.85	2.11	4.03	181.51
Cmps	V7	7	59	589	289	215	0.75	1.03	-1.40	220.45
Raw	V8	9	26	621	140	196	1.40	2.69	6.73	72.84
Cmps	V8	12	26	621	154	170	1.10	2.38	6.25	92.82
Raw	V9	9	53.6	1,114	410	373	0.91	1.26	0.17	262.12
Cmps	V9	11	53.6	894	358	255	0.71	0.88	0.72	261.56
Raw	VNW	102	5	4,030	369	612	1.7	3.6	15.7	154.7
Cmps	VNW	112	5	3,361.897	360.661	540.246	1.5	3.31	12.91	169.247

TOP CUTS

Due to the high-grade nature of the deposit, it was preferable to limit top cutting to avoid smoothing silver values within the true high-grade pockets of mineralisation. Very high grades form a genuine part of the sample population.

Log probability plots were analysed for the Vertikalny central and Vertikalny northwest domains (Figure 14.25 and Figure 14.26). . Two inflection points were noted in the Vertikalny central domains: one at approximately 6,000 g/t, and one at approximately 10,000 g/t. Compositing was performed prior to top cutting in order to reduce the high-grade burden organically. The 10,000 g/t threshold was chosen and applied to the composited data, Figure 14.24.

Figure 14.24 Log Histogram for Domain V1 Illustrating Capping Level



This top cut only affected five samples. No capping was applied to the Vertikalny northwest domains.

Figure 14.25 Log Probability Plot for Vertikalny Central Domains

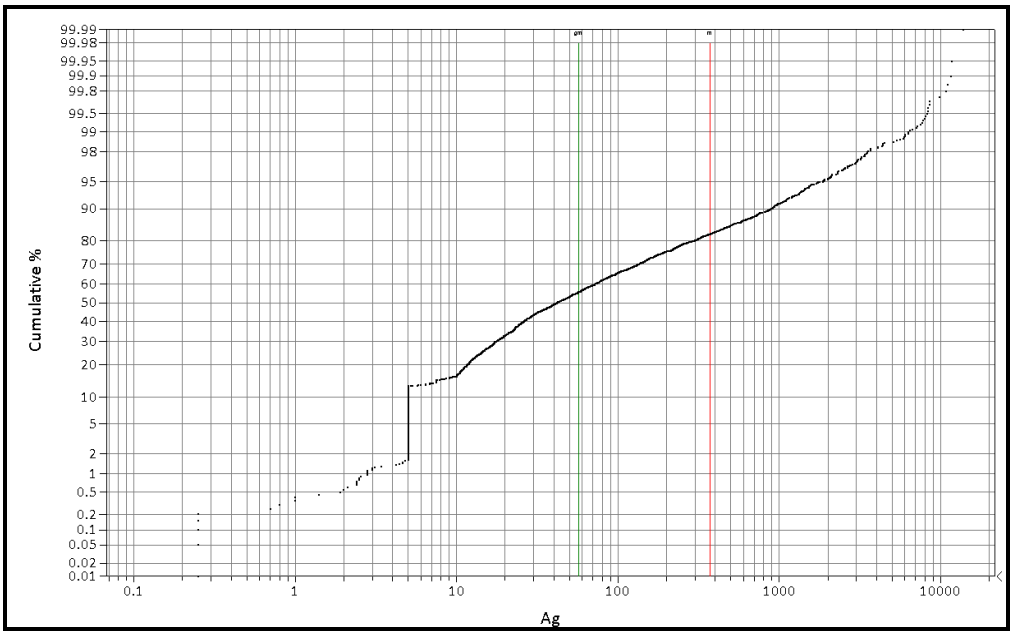
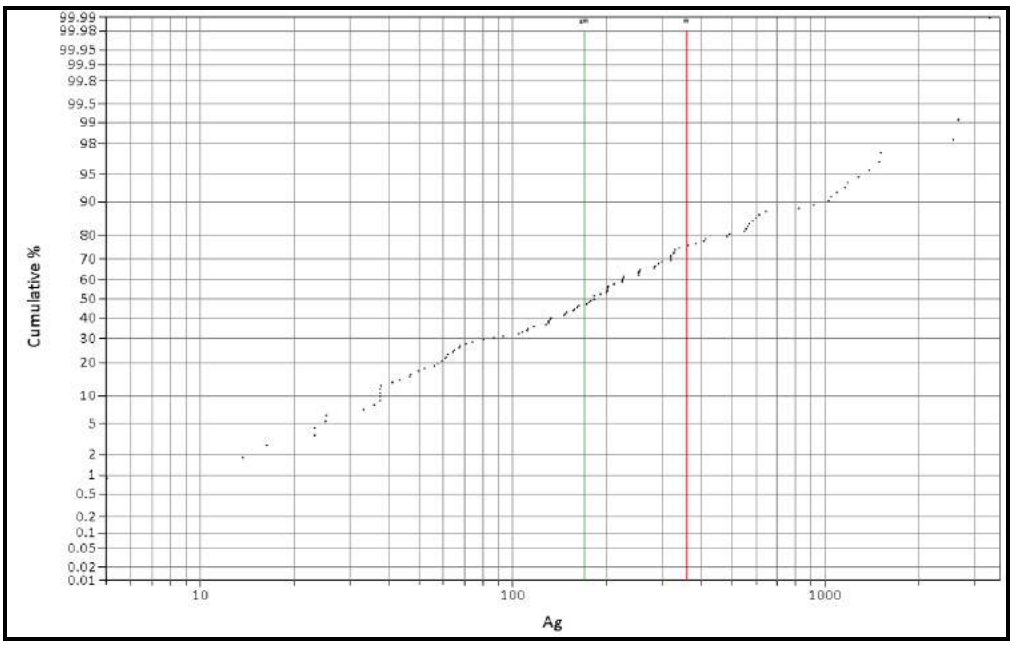


Figure 14.26 Log Probability Plot for VNW Domain



14.3.5 DENSITY

Specific gravity measurements from selected holes from the 2007 through 2015 drilling programs were provided by Silver Bear. In total, 375 measurements were included in the database. Density results per lithology group are presented in Table 14.17.

Density was estimated into the mineralised domains within the Vertikalny central area utilizing the IDW^2 method. Due to the low number of samples in the Northwest

Zone, density for that zone was assigned at the mean value for the density measurements within the mineralised domain 3.2 t/m^3. The volume weighted average estimated density within the Vertikalny central domains is 3.65 t/m^3.

Mean density values for each lithology were assigned to the waste blocks using geological wireframe models as a constraints.

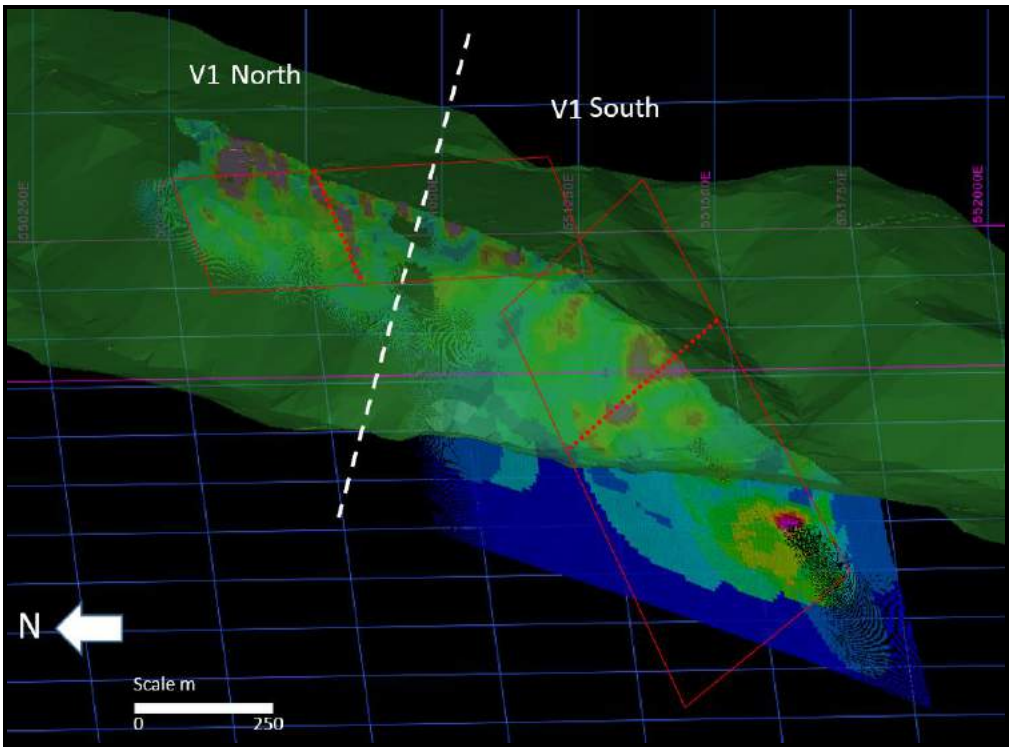
Table 14.17 Mean Density Values per Lithology Group

Lithology	Mean Density Value (t/^{m3})
Breccia	3.7
Igneous	3.0
Mineralisation	3.5
Sediments	2.7
Mean	3.2

14.3.6 VARIOGRAPHY

Variography was completed in Surpac Version 6.7. Experimental variograms were produced using drillhole composite data subjected to a normal scores transformation. Variograms were developed in plan view initially, with the strike, dip and plunge established independently thereafter. The nugget value has been established from a downhole variogram. For the main mineralised domain at Vertikalny central (V1), it was interpreted that the distribution of high grade mineralisation is controlled by oblique structures intersecting the mineralised vein. This interaction of structures produces different grade distributions in the northern part of the structure compared to the southern area. For variography the V1 domain was sub-divided into 'V1 North' and 'V1 south', Figure 14.27.

Figure 14.27 3D Isometric View Illustrating the Theoretical Interaction between the Mineralised Vein and the Oblique Structure

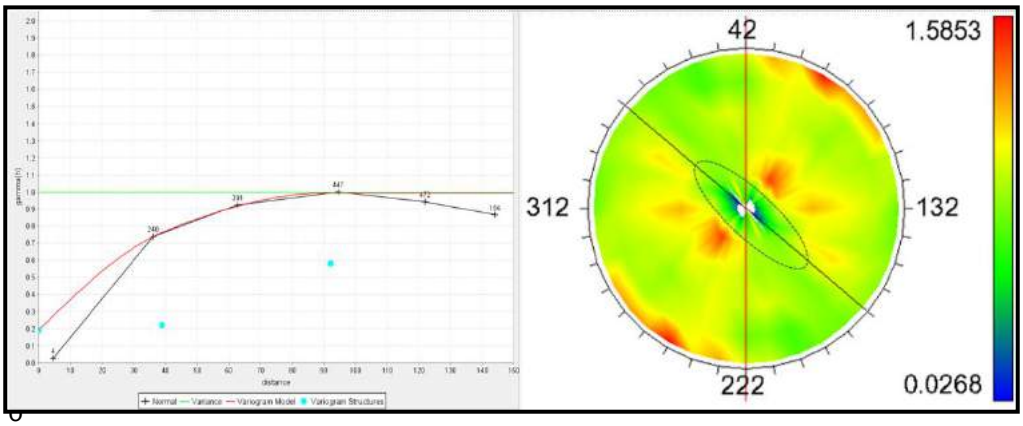


Notes: Red lines show a schematic representation of the oblique structures, with the red dashed line representing the theorectical intersection between the structures and the mineralised vein.
White dashed line illustrates where the V1 domain was sub-divided for variography.

The modelled structures were used to establish the variogram structures to be used within the kriging estimation.

Figure 14.28 illustrates an example semi-variogram and variogram map for the V1 domain.

Figure 14.28 Semi Variogram and Associated Variogram Map for V1 North – Major D



derately

Strong directional control has been established within the silver grade distribution. Table 14.18 presents the variogram parameters used for the estimation.

Table 14.18 Directional Variogram Parameters

Zone	Major Direction	Plunge	Dip	Nugget	Range 1	Sill 1	Range 2	Sill 2	Semi-major Ratio
V1 South	139	39.4	0	0.2	35	0.22	92	0.58	1.67
V1 North	121	-49	0	0.2	5	0.015	50	0.65	1.42
H1	108	-68	0	0.2	130	N/A	N/A	N/A	2
V2-V10	108	-68	0	0.2	130	N/A	N/A	N/A	2

14.3.7 Resource Block Models

Two separate block models were constructed in Surpac, one for the Vertikalny Northwest Zone and a second for the Central Zone. The block model parameters are given in Table 14.19 and Table 14.20.

Table 14.19 Vertikalny Central Block Model Parameters

Type	Y	X	Z
Minimum Coordinates	7282628	551717	450
Maximum Coordinates	7286008	552850	1250
User Block Size	10	1	10
Minimum Block Size	2.5	0.25	2.5
Rotation	-45	0	0

Table 14.20 Vertikalny Northwest Block Model Parameters

Type	Y	X	Z
Minimum Coordinates	7285000	549400	600
Maximum Coordinates	7286600	549900	1400
User Block Size	10	1	10
Minimum Block Size	2.5	0.25	2.5
Rotation	-53	0	0

Standardised sub-cell splitting to the minimum block sizes presented in Table 14.19 and Table 14.20 was employed. Sub-cells received parent cell grades during estimation. Table 14.21 presents a comparison of the block model and wireframe volumes. The large parent cell is selected for best Kriging performance during the grade interpolation, whilst the smaller sub-cell maintains consistency between the final block volumes and the wireframe volume.

Table 14.21 Comparison of Block Model and Wireframe Volume

Domain	Block Model Volume (m^3)	Wireframe Volume (m^3)	Percentage Difference in Volume (%)
H1	4,544,650	5,018,370	100.0
V1	473,722		
V10	1,158	1,163	100.4
V2	38,303	38,437	100.3
V3	25,708	25,716	100.0
V4	5,178	5,118	98.8
V5	5,480	5,699	104.0
V6	2,942	2,925	99.4
V7	12,245	12,208	99.7
V8	15,805	15,749	99.6
V9	29,927	29,867	99.8
VNW	193,669	193,927	99.87

INTERPOLATION STRATEGY

Grades were estimated using OK, adopting a multi-pass methodology. The kriging employed the variogram parameters as presented in Section 14.6. Each of the domains was estimated independently.

QKNA was used to optimise the block size, the number of informing samples, discretisation, and search distances used in the estimation.

Quantitative measures of the kriging performance (e.g. slope of regression, kriging efficiency, kriging variance, block variance, proportion of negative weights) were used to test the appropriateness and optimise the kriging parameters. The analysis was carried out on large representative portions of the block model, as well as on isolated test blocks. Where test blocks were used, these were chosen as examples of one well informed block and one poorly informed block.

A summary of the estimation strategy is show in Table 14.22.

Table 14.22 Estimation Parameters

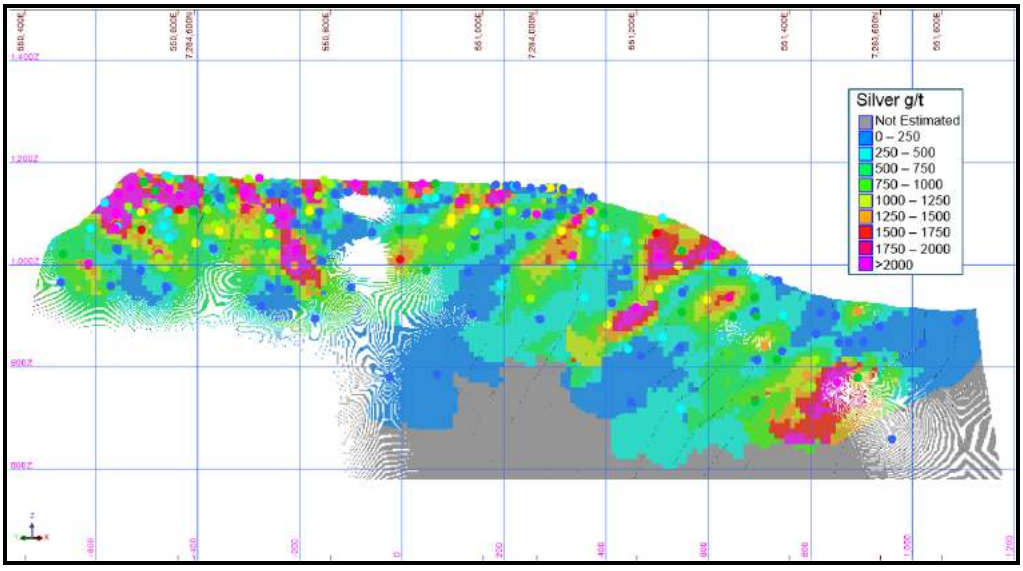
Domain	Pass Number	Samples Numbers		Search (m)	Descritisation	Maximum Samples per ID	Variogram Model
		Minimum	Maximum				
V1 North	1	3	8	160	x1,y3,z3	2	V1 North Model
V1 North	2	3	8	75	x1,y3,z3	2	V1 North Model
V1 North	3	3	8	35	x1,y3,z3	2	V1 North Model
V1 South	1	3	8	160	x1,y3,z3	2	V1 South Model
V1 South	2	3	8	92	x1,y3,z3	2	V1 South Model
V1 South	3	3	8	60	x1,y3,z3	2	V1 South Model
H1	1	3	8	195	x1,y3,z3	2	H1 Model
H1	2	3	8	130	x1,y3,z3	2	H1 Model
H1	3	3	8	85	x1,y3,z3	2	H1 Model
V2-V10	1	3	10	130	x1,y3,z3	2	H1 Model
VNW	1	3	8	160	x1,y3,z3	2	V1 North Model
VNW	2	3	8	75	x1,y3,z3	2	V1 North Model
VNW	3	3	8	35	x1,y3,z3	2	V1 North Model

14.3.8 BLOCK MODEL VALIDATION

Block model validation was completed using graphical and statistical methods, to confirm that the estimated block model grades appropriately reflect the local composite grades.

Graphical analysis of the informing samples versus estimated block grades was completed using long sections (Figure 14.29 and Figure 14.30).

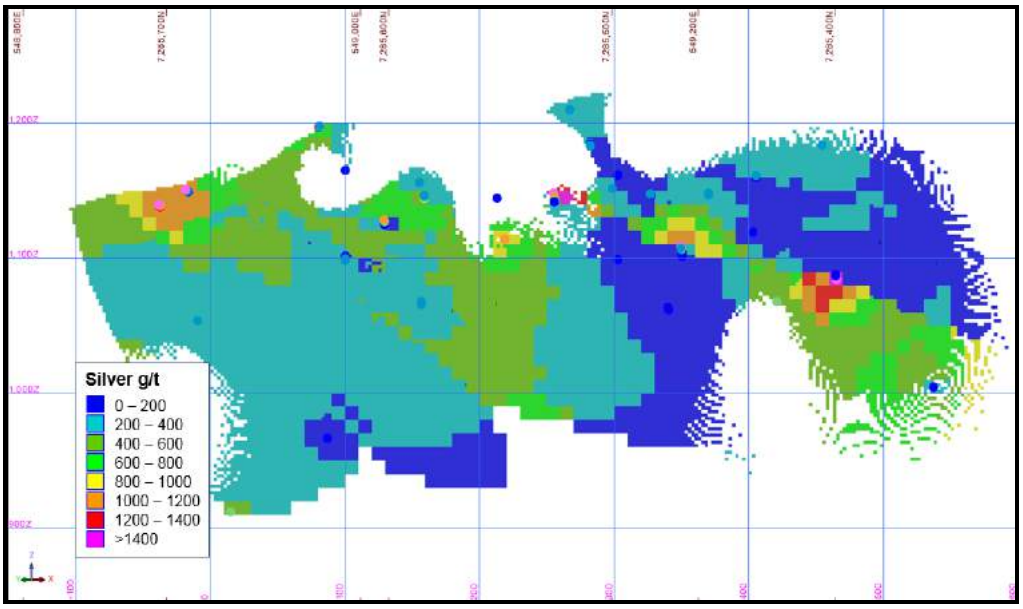
Figure 14.29 Long Section Showing Composites and Estimated Block Grades for Vertikalny Central



Notes: Vertical and horzontal grid lines are at 200m intervals.

The visual inspection demonstrated excellent correlation between composite and block grades. The anisotropic directionality noted in the variography can be clearly shown in Figure 14.30 where the anisotropy is reflected in the grade distribution.

Figure 14.30 Long Section Showing Composites and Estimated Block Grades for Vertikalny Northwest



Note: Vertical and horizontal gridlines at 100 m
Looking northeast

A comparison was made between the informing composites and the estimated block grades using a range of statistical measures. Table 14.23 presents a comparison between the informing composites and the estimated block grades for the Indicated Resource blocks within Vertikalny Central.

Table 14.23 Statistics Comparing Block Estimate and Composite Grades

Data	Domain	Count	Min	Max	Mean	Std Dev	CV	Skewness	Kurtosis	Geom Mean	Median
Block Model	H1	528,760	1	1,744	59	62	1.05	6.22	92	42	43
Composites	H1	1,770	0	4,393	72	177	2.44	13.19	263	30	28
Block Model	V1	171,852	12	7,129	842	774	0.92	2.32	9	562	646
Composites	V1	499	12	13,054	1,256	1,727	1.38	3.02	12	534	687
Block Model	V2-V10	35,591	16	2,818	354	173	0.90	2.75	16	153	160
Composites	V2-V10	153	5	8,369	440	626	1.87	3.40	13	132	128
Block Model	VNW	52,933	42	1,619	352	202	0.57	1.71	5	302	295
Composites	VNW	87	5	3,362	436	590	1.35	2.95	10	216	225

Notes: Table 14.23 assesses Vertikalny Central Indicated Resource blocks only.

A number of the measures indicate a reduction in variance. This is a result of the change of support associated with the estimation process and the kriging interpolation. There is a reduction in the mean silver grade from composites to block estimate due to the de-clustering effect of the OK. Overall, the statistics present good conformance.

Swath Plots

Swath plots have been used to assess the differences and similarities between the OK block estimate silver grade and the informing composite grades (Figure 14.31 and Figure 14.32).

Figure 14.31 Northing Swath Plot Comparing the Informing Composite and the OK Estimated Silver Grades for Vertikalny Central

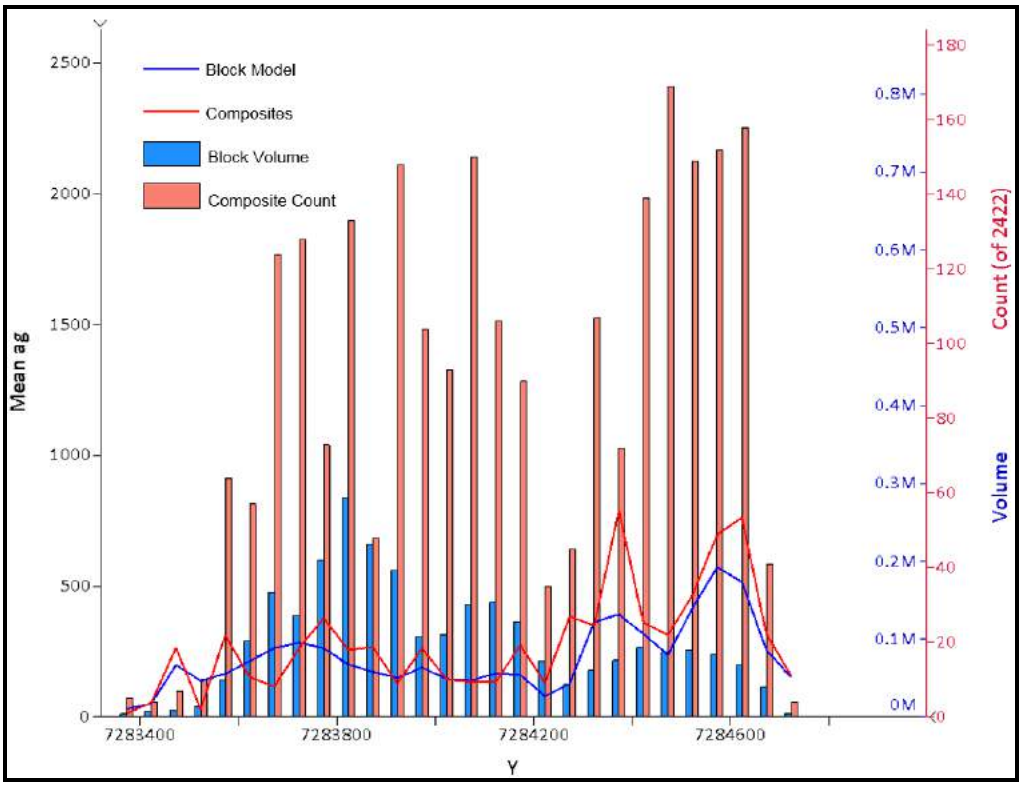


Figure 14.32 Easting Swath Plot Comparing the Informing Composite and the OK Estimated Silver Grades for Vertikalny Central

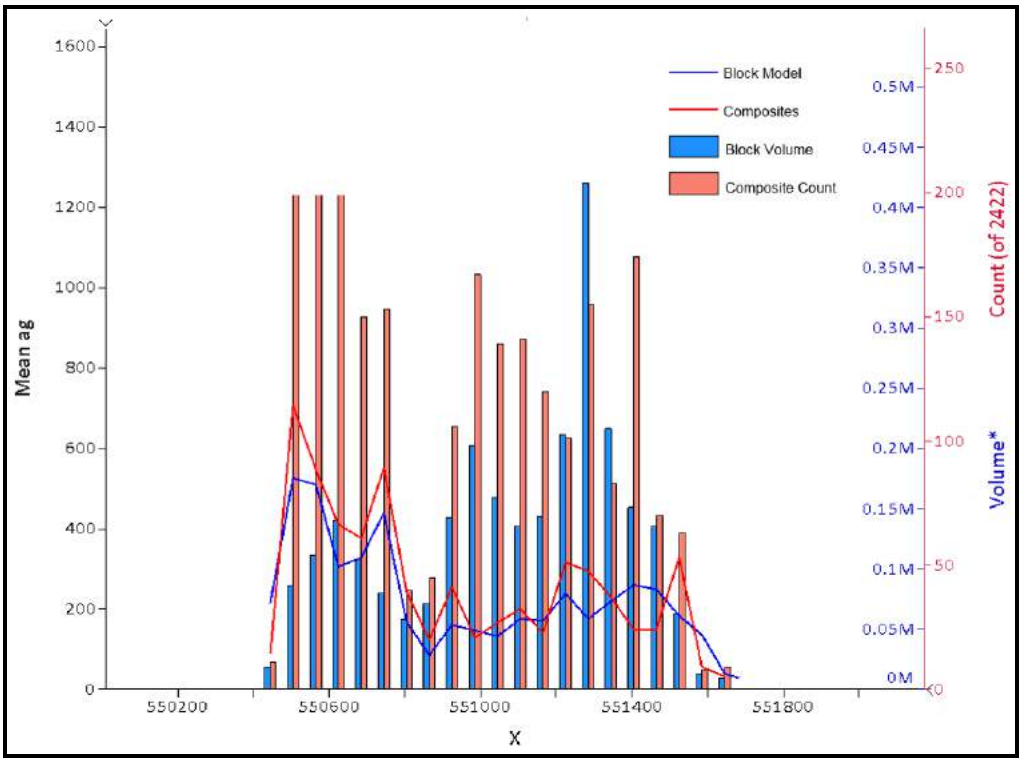
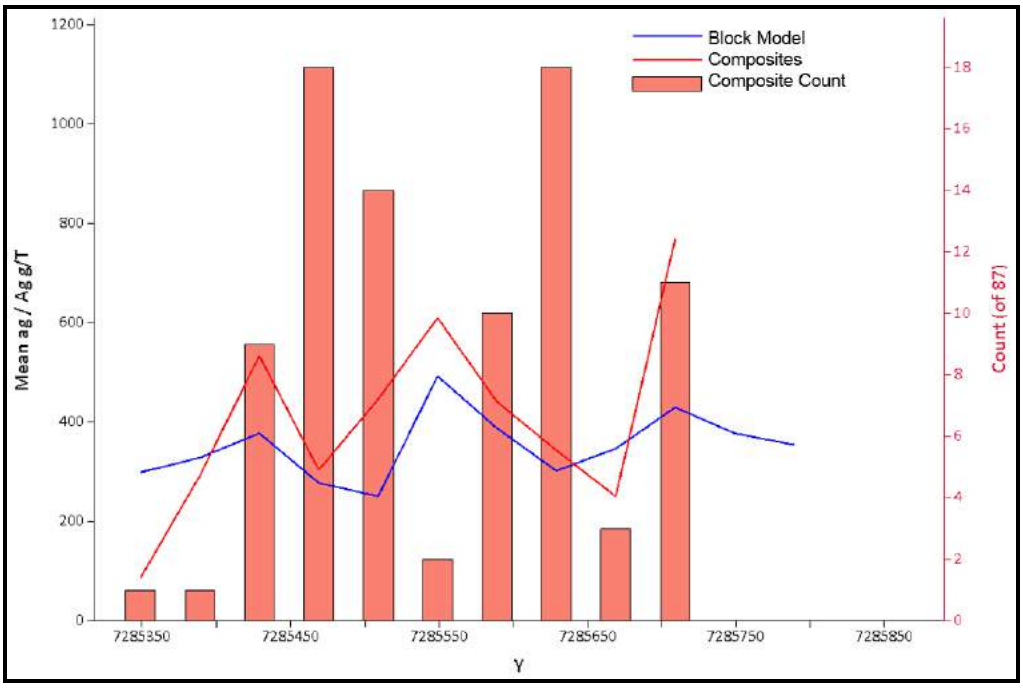


Figure 14.33 Northing Swath Plot Comparing the Informing Composite and the OK Estimated Silver Grades for Vertikalny Northwest



The swath plots for Vertikalny Central present good conformance between informing composites and estimated block grades. The swath plots for Vertikalny Northwest showed poorer conformance due to the impertinent mineralisation and paucity of data.

CONCLUSION

The various comparators described in the foregoing subsections serve to illustrate that the block model estimates are robust and satisfactorily models the distribution and variability of the informing sample grades without undue bias or smoothing. The models are suitable for the current level of development.

14.3.9 MINERAL RESOURCE CLASSIFICATION

The resource model was classified according to CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014).

A large proportion of the Mineral Resource within Vertikalny Central has been classified as Indicated. In this part of the deposit the nature, quality, amount, and distribution of data are such as to allow confident interpretation of the geological framework, and to assume continuity of mineralisation. Confidence in the estimate is sufficient to allow application of modifying factors within future technical and economic studies.

The remaining Mineral Resource, and all of the Vertikalny Northwest Mineral Resource are classified as Inferred. The mineralisation has been identified and limited measurements and sampling completed, but the data is insufficient to allow the geological and grade continuity to be confidently interpreted. It would be reasonable to expect that the majority of Inferred Mineral Resources would be upgraded to Indicated Mineral Resources with continued exploration; however, due to the uncertainty of Inferred Mineral Resources it should not be assumed that such upgrading will always occur.

Confidence in the estimate of Inferred Mineral Resources is not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning in prefeasibility or feasibility studies.

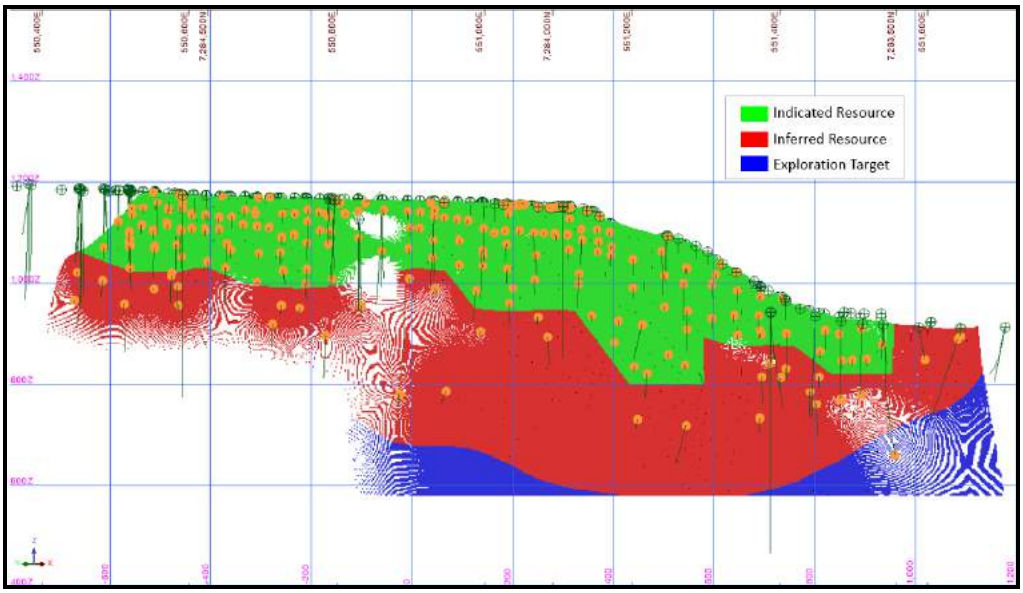
To classify the Mineral Resources, wireframe models have been generated to represent the limit of the Indicated and Inferred Resources. The models consider the following criteria:

- confidence in the sampling data and geological interpretation
- analysis of variogram parameters
- the data distribution (based upon graphical analysis and quantitative assessment of nearest and average distances to informing composites)
- kriging efficiency and slope of regression.

The wireframe models reflect the trends in the classification parameters as presented on a block by block basis, while ensuring that the classification resulted in appropriately coherent units. Figure 14.34 presents the scheme of Mineral Resource Classification for the main structure at Vertikalny Central. The subordinate structures

(domains V2-V10) and Vertikalny northwest are classified in the Inferred Resource category.

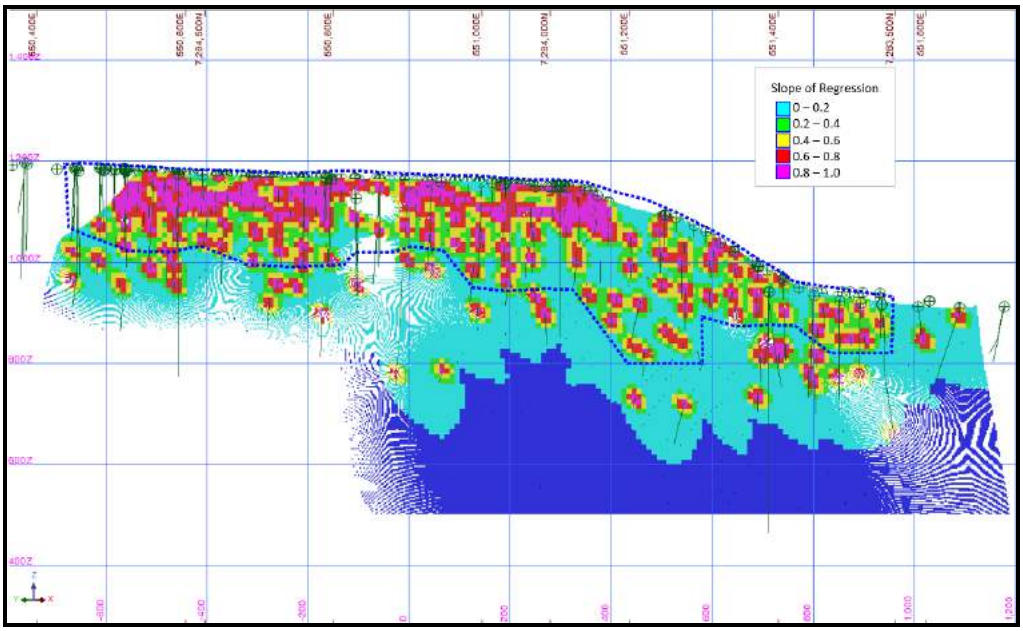
Figure 14.34 Long Section of the Mineral Resource Classification for Vertikalny Central



Note: Looking northeast

Figure 14.35 presents a long section illustrating the adopted resource classification and the slope of regression.

Figure 14.35 Long Section of the Mineral Resource Classification and Slope of Regression



Notes: Dashed blue line shows the limit of the Indicated Resource. Looking Northeast

14.3.10 MINERAL RESOURCE TABULATION

CUT-OFF GRADES

In order to demonstrate that the deposit has reasonable prospects for economic extraction a cut-off grade of 200 g/t silver has been applied for resources contained within an open pit shell at Vertikalny Central or within 80 m of surface at Vertikalny Northwest. A 350 g/t silver cut-off has been applied for the underground resource

The resource pit shell used to constrain the open pit resource at Vertikalny Central uses the following assumptions:

- silver price of $19.00/troy oz
- mining recovery 95%
- waste dilution 30%
- mining cost of US $2.53/t
- processing cost of US $52.00/t
- general and administration costs of US $40.60/t
- royalty of 6.5%
- overall silver recovery of 88%.

MINERAL RESOURCE TABULATION

The updated Mineral Resource for the Vertikalny deposit is summarised in Table 14.24. The effective date of the updated Mineral Resource is 8th July 2016.

Table 14.24 Summary of Mineral Resources – Vertikalny Deposit

Zone	Assumed Mining Method	Cut-off Grade Ag (g/t)	Indicated Resources			Inferred Resource		
			Tonnes	Grade Ag (g/t)	Contained Metal Ag (troy oz)	Tonnes	Grade Ag (g/t)	Contained Metal Ag (troy oz)
Vertikalny Central	Open Pit	200	360,000	1,482	17,100,000	4,000	496	100,000
	Underground	350	340,000	959	10,600,000	350,000	789	8,800,000
	All	-	700,000	1,227	27,700,000	350,000	786	8,900,000
Vertikalny Northwest	Open Pit	200	-	-	-	110,000	430	1,600,000
	Underground	350	-	-	-	90,000	535	1,500,000
	All	-	-	-	-	200,000	476	3,100,000
Total	**All**	**-**	**700,000**	**1,227**	**27,700,000**	**550,000**	**672**	**12,000,000**

GRADE TONNAGE SENSITIVITY

The Vertikalny block models have been reported at a range of cut-offs to provide an indication of the grade-tonnage sensitivity (Table 14.25, Table 14.26, and Table 14.27).

Table 14.25 Vertikalny Central Indicated Blocks – Grade-Tonnage Sensitivity

Cut-off Ag Grade (g/t)	Tonnage (kt)	Ag Grade (g/t)	Contained Metal Ag (koz)
150	1,046,000	892	30,010,000
200	852,000	1,057	28,940,000
250	773,000	1,142	28,370,000
300	707,000	1,222	27,790,000
350	661,000	1,285	27,310,000
400	627,000	1,334	26,900,000
450	595,000	1,383	26,470,000

Notes: Sensitivity numbers cannot be compared directly to the Mineral Resource tabulation in Table 14.24 because different cut-offs have been applied for open pit and underground resource blocks.

Table 14.26 Vertikalny Central Inferred Blocks – Grade-Tonnage Sensitivity

Cut-off Ag Grade (g/t)	Tonnage (kt)	Ag Grade (g/t)	Contained Metal Ag (koz)
150	628,000	539	10,880,000
200	518,000	617	10,270,000
250	460,000	667	9,860,000
300	378,000	754	9,160,000
350	351,000	786	8,880,000
400	333,000	809	8,650,000
450	312,000	835	8,370,000

Notes: Sensitivity numbers cannot be compared directly to the Resource tabulation in Table 14.24 because different cut-offs have been applied for open pit and underground resource blocks.

Table 14.27 Vertikalny North West Inferred Blocks - Grade Tonnage Sensitivity

Cut-off Ag Grade (g/t)	Tonnage (kt)	Ag Grade (g/t)	Contained Metal Ag (koz)
150	255,000	465	3,810,000
200	237,000	487	3,710,000
250	209,000	522	3,510,000
300	170,000	579	3,160,000
350	154,000	606	2,990,000
400	134,000	638	2,750,000
450	114,000	677	2,470,000

Notes: Sensitivity numbers cannot be compared directly to the Resource tabulation in Table 14.24 because different cut-offs have been applied for open pit and underground resource blocks.

In September 2012, June 2015 and March 2016, Tetra Tech estimated the Mineral Resource for the Nizhny Endybal Mangazeisky South and Mangazeisky North deposits, which are shown along with Vertikalny Central and Vertikalny Northwest estimates in Table 14.28 as a summary of the total Mineral Resources for the Property.

Table 14.28 Total Resources for the Mangazeisky Property

Zone	Indicated Resource			Inferred Resource		
	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)	Tonnes (t)	Grade Ag (g/t)	Contained Metal Ag (troy oz)
Vertikalny Central	700,000	1,227	27,700,000	350,000	786	8,900,000
Vertikalny Northwest	-	-	-	200,000	476	3,100,000
Nizhny Endybal	-	-	-	710,000	316	7,200,000
Mangazeisky North	334,000	770	8,250,000	127,000	560	2,300,000
Mangazeisky South	-	-	-	60,000	246	500,000
Sterzhnevoy	-	-	-	48,000	1,530	2,360,000
Total	**1,034,000**	**1,079**	**35,950,000**	**1,495,000**	**504**	**24,360,000**

Zone	Inferred Resource						
	Tonnes (t)	Grade Au (g/t)	Grade Ag (g/t)	Grade Cu (%)	Contained Metal Au (troy oz)	Contained Metal Ag (troy oz)	Contained Metal Cu (lb)
Porphirovy	80,000	1.7	46	0.6	4,000	115,000	1,000,000

Notes: The effective date of the Sterzhnevoy and Porphirovy maiden resources is August 27, 2016. The effective date of the Vertikalny Central resource is July 8, 2016 and the Vertikalny Northwest Resource is February 10, 2015. The effective date of the original Nizhny Endybal Resource estimate was September 11, 2012, this resource was re-stated with a higher cut-off grade on the June 10, 2015. The effective date of the Mangazeisky North is October 19, 2016 and the Mangazeisky South Mineral Resource is June 10, 2015. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Although Silver Bear and Tetra Tech are not aware of any material barrier to eventual economic extraction.

Table 14.29 presents a summary of the Mineral Resource Estimation assumptions and methods

Table 14.29 Mineral Resource Estimation Assumptions and Methods

Prospect Area	Key Assumptions
Vertikalny Central and North West	• The Mineral Resources were estimated into two separate 3D block models comprising the following parameters: Vertikalny Central: 10 m x 10 m x 1 m (x, y, z), with minimum sub-block dimensions of 2.5 m x 2.5 m x 0.25 m (x, y, z). Vertikalny Northwest: 10 m x 10 m x 1 m (x, y, z), with minimum sub-block dimensions of 2.5 m x 2.5 m x 0.25 m (x, y, z). • The estimation was constrained to the mineralised zone using wireframe solid models. The wireframes were sub-domained to isolate the strongly mineralised main zone from the silver mineralisation associated with wall rock alteration. • Grade estimates were based on 0.7 m composited assay data. • The interpolation of the metal grades was undertaken using OK. • In order to demonstrate that the deposit has reasonable prospects for economic extraction, a cut-off grade of 200 g/t silver was applied for Mineral resources contained within an open pit shell at Vertikalny Central or within 80m of surface at Vertikalny Northwest. A 350 g/t silver cut-off has been applied for the underground Mineral Rresource. • At Vertikalny Central, a pit shell wireframe was used to constrain the open pit Mineral Resource in order to demonstrate that the Mineral Resource has reasonable prospects for economic extraction. The pit shell uses the following assumptions: • silver price of $19.00/troy oz • mining recovery 95% • waste dilution 30% • mining cost of $2.53/t • processing cost of $52 /t • G&A cost of $40.6 /t • royalty of 6.5% • overall silver recovery of 88%.
Nizhny Endybal	• In order to maintain the Nizhny Endybal Resource as a current Mineral Resource, Tetra Tech has restated the Mineral Resource with the following assumptions: • cut-off grade of 150 g/t silver • spot metal prices effective February 10, 2015 of $17.00/oz troy oz • mining cost of $3.16/t • processing cost of $50/t • G&A cost of $40/t • overall silver recovery of 90%.

table continues...

Prospect Area	Key Assumptions
Mangazeisky North & South	• The Mineral Resources were estimated into two separate 3D block models each comprising the following parameters: 20 m x 20 m x 5 m (x, y, z), with minimum sub-block dimensions of 1 m x 1 m x 0.5 m (x, y, z). • The estimation was constrained to the mineralised zone using wireframe solid models. The wireframes were sub-domained to isolate each mineralised structure. • Grade estimates were based on 0.5 m composited assay data. • The interpolation of the metal grades was undertaken using OK. • In order to demonstrate that the deposit has reasonable prospects for economic extraction a cut-off grade of 150 g/t was applied for Mangazeisky North, based upon the following assumptions: • silver price of $17.00/troy oz • mining recovery 95% • waste dilution 30% • mining cost of $2.53/t • processing cost of $52.00/t • G&A cost of $40.60/t • royalty of 6.5% • overall metal recovery of 88%.
Sterzhnevoy and Porphirovy	• The Mineral Resources were estimated into two separate 3D block models each comprising the following parameters: 20 m x 20 m x 10 m (x, y, z), with minimum sub-block dimensions of 1 m x 1 m x 0.5 m (x, y, z). • The estimation was constrained to the mineralised zone using wireframe solid models. The wireframes were sub-domained to isolate each mineralised structure. • Grade estimates were based on 0.5 m composited assay data. • The interpolation of the metal grades was undertaken using IDW. • In order to demonstrate that the deposit has reasonable prospects for economic extraction a cut-off grade of 150 g/t was applied for Sterzhnevoy, based upon the following assumptions: • silver price of $17.00/troy oz • mining cost of $2.50 /t • processing cost of$50 /t • G&A cost of $40/t • overall silver recovery of 90% • a cut-off grade of $77.00 equivalent based on the same assumptions for Porphirovy; dollar equivalent calculated with the formula = Au ozt x $1250 + Ag ozt x $17 + Cu lbs x $2.2

Note: Assumes that the Mangazeisky deposits will be worked with Vertikalny and will therefore share G&A costs.

Table 14.30 presents a summary status of the Mineral Resource estimates.

Table 14.30 Summary of Resource Estimates on the Mangazeisky Property

Deposit	Qualified Person	Effective Date
Nizhny Endybal	Mr. Robert Davies, BSc, EurGeol, Cgeol	11th September 2012
Mangazeisky North	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	19th October 2016
Mangazeisky South	Mr. Robert Davies, BSc, EurGeol, Cgeol	3rd June 2015
Vertikalny Central	Mr. Robert Davies, BSc, EurGeol, Cgeol	8th July 2016
Vertikalny Northwest	Mr. Robert Davies, BSc, EurGeol, Cgeol	8th July 2016
Sterzhnevoy	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	27th August 2016
Pophirovy	Mr. Joe Hirst, BSc, MSc, EurGeol, CGeol	27th August 2016

14.3.11 GRADE-TONNAGE CURVES

Figure 14.36 and Figure 14.37 depict grade-tonnage curves for the Vertikalny Central Zone and Northwest Zone Mineral Resources.

Figure 14.36 Grade-Tonnage Curve for Indicated & Inferred Resources at Vertikalny Central Zone

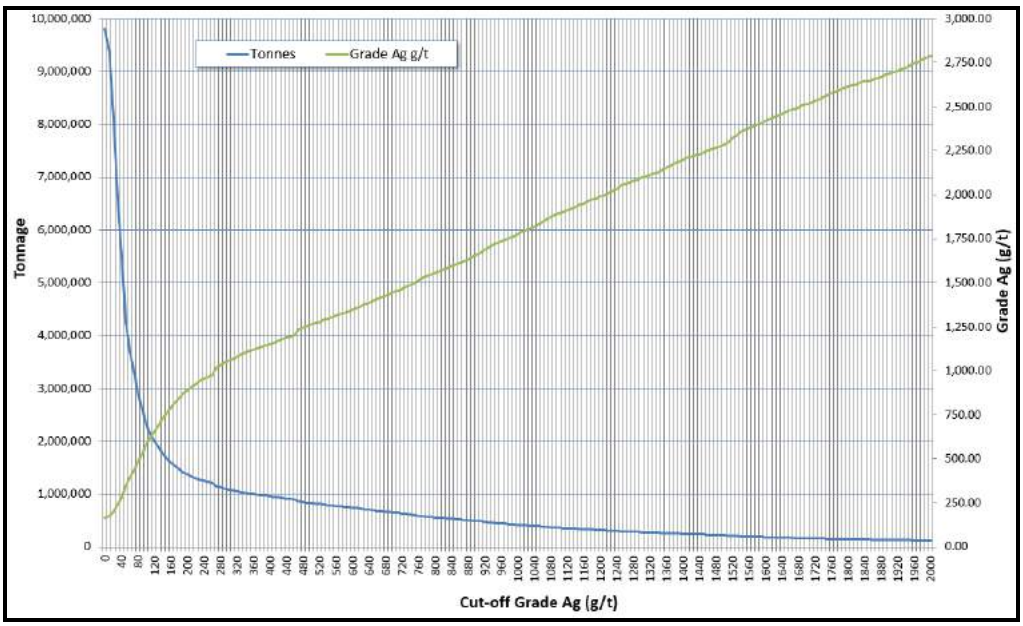
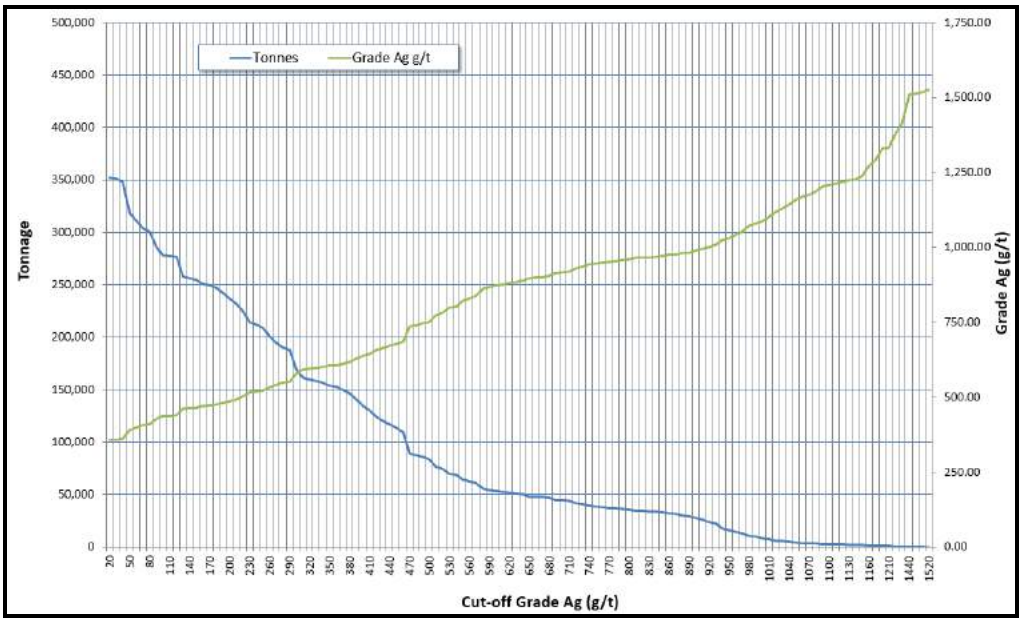


Figure 14.37 Grade-Tonnage Curve for Inferred Resource at Vertikalny Northwest Zone



Tetra Tech is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that could materially affect the Mineral Resource.

15.0 MINERAL RESERVE ESTIMATES

15.1 CUT-OFF GRADE PARAMETERS

The input parameters used to calculate the open pit and underground cut-off grades are shown in Table 15.1 and Table 15.2, respectively.

Table 15.1 Open Pit Cut-off Grade Calculation Parameters

Description	Unit	Value
Silver Price	USD/troy oz	18.00
Process Unit Cost	USD/t processed	57.00
Process Recovery	%	85
G&A Unit Cost	USD/t processed	27.00
Selling Cost	USD/troy oz	0.50
Royalty	%	6.5
Mining Dilution	%	30

Note: To calculate the marginal cut-off grade for the open pit it has been assumed that approximately 33% of the G&A costs are associated with mining and therefore the G&A can be reduced to USD27/t ore processed.

Table 15.2 Underground Cut-off Grade Calculation Parameters

Description	Unit	Value
Silver Price	USD/oz	18.00
Processing Unit Cost*	USD/t processed	57.00
Process Recovery	%	85
G&A Unit Cost*	USD/t processed	40.00
Underground Mining Unit Cost	USD/t mined	53.00

Note: Costs at the time of the cut-of grade calculation. The final cost estimates may change.

15.1.1 OPEN PIT MINERAL RESERVE CUT-OFF

The open pit Mineral Reserves were estimated using the in situ rock inventory above the selected cut-off grade by applying 30% mining dilution at zero grade, and 5% mining losses.

The selected dilution and loss factors are as per Russian Mineral Reserve statement standards. In practice, while ore losses and dilution will occur during the mining operation, it is possible that the mining dilution factor may be lower and that part of the diluted material may contain mineralization.

Using the presented parameters in Table 15.4, the marginal cut-off grade for in situ block values is 240 g/t silver. The selected cut-off grade results in continuation of the orebody in some areas, allowing a more practical selective mining during the mining operation.

15.1.2 Underground Mineral Reserve Cut-off

Prior to developing a detailed mine design, analyses of the typical overall mining costs and the anticipated extraction factors such as mine extraction percentage and dilution, is required to develop an understanding of the diluted mineral grades that may contribute to the Project value.

Often these analyses are challenging, and even a 10% variation of the diluted grade may yield sub-economical areas. The mineralisation at Vertikalny typically consists of thin and steeply dipping structures; therefore, it was important to evaluate a number of different underground mining shapes over a range of cut-off grades using industry accepted and proven underground stope optimisation software, Datamine's Mineable Shape Optimizer (MSO).

Tetra Tech used both break-even cut-off grade and the marginal cut-off grade in the determination of the underground Mineral Reserve. The break-even cut-off grade is the lowest grade of material that can be mined and processed economically considering all applicable costs, without incurring a loss and is defined as follows:

Break-even Cut-off Grade = Total Unit Operating Cost/(Unit Price x Recovery)

The break-even cut-off grade was calculated as approximately 305 g/t silver.

The marginal cut-off grade is the lowest grade where the net revenue value of the metal recovered is equal to the cost of mining, crushing, loading, hauling, and processing. By implication it is assumed that in this case the G&A cost is sunk. The marginal cut-off grade calculation is as follows:

Marginal Cut-off Grade = (Total Unit Operating Cost –G&A Costs)/(Unit Price x Recovery)

The marginal cut-off grade was calculated as approximately 223 g/t silver.

In both cases, the cut-off grade corresponds to a diluted grade (i.e., the grade of the material that is available for processing or stockpiling).

Tetra Tech applied the break-even cut-off grade to 95% of the total mineable ore tonnes, and the marginal cut-off grade to 5% of the mineable ore tonnes, to derive an effective Mineral Reserve cut-off grade of 300 g/t silver. The Mineral Reserve cut-off grade was used as a diluted cut-off parameter in Datamine's MSO to develop the economical and practical mining shapes for the Vertikalny underground mine. Mining shapes with 300 g/t silver (diluted) or higher will add value if exploited and processed within the assumed economic environment. The calculation of the two cut-off grades are summarised in Table 15.3 and Table 15.4.

Table 15.3 Break-even Mining Cut-off Grade Calculations

Calculation	Unit	Result
Total Underground Variable Mining Cost	USD	24,394,683
Underground Modified Ore Tonnes Planned	t	457,981
Minimum Mining Width Sub-level Open Stoping	m	1.4
Modification from Report (Dilution)	%	33.33
Processing Cost	USD/t	57.00
G&A Cost	USD/t	40.00
Process Recovery	%	85
Silver Price	USD/tr. oz	18.00
Calculation of Cut-off Grade (Diluted)	g/t Ag	305

Table 15.4 Marginal Mining Cut-off Grade Calculations

Calculation	Unit	Result
Total Underground Variable Mining Cost	USD	24,394,683
Underground Modified Ore Tonnes Planned	t	457,981
Minimum Mining Width Sub-level Open Stoping	m	1.4
Modification from Report (Dilution)	%	33.33
Processing Cost	USD/t	57.00
G&A Cost	USD/t	40.00
Process Recovery	%	85
Silver Price	USD/tr. oz	18.00
Calculation of Cut-off Grade (Diluted)	g/t Ag	223

The key input parameter used for the development of the mining stopes is the 300 g/t silver (diluted) cut-off grade. The Datamine MSO software considers low-grade material as dilution; therefore, dilution does not necessarily occur at 0 g/t silver grades in all areas of the stopes. Where Inferred Mineral Resources or waste is included, the dilution grade was set to 0 g/t silver.

Note that the 300 g/t silver diluted cut-off grade corresponds to approximately 450 g/t in-situ cut-off grade.

15.2 MINERAL RESERVE STATEMENT

The Mineral Reserve was estimated in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (prepared by the CIM Standing Committee on Reserve Definitions, Adopted by CIM Council on November 27, 2010). According to the CIM, a Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a prefeasibility study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined. The Mineral Reserve is inclusive of the reported Mineral Resource.

Proven and Probable Mineral Reserves for the Vertikalny deposit are presented in Table 15.5.

Table 15.5 Vertikalny Deposit Mineral Reserve Statement – April 30th 2017

Category	Cut-off Grade (g/t Ag)	Ore (t)	Ag Grade (g/t)	Ag Metal Content (koz)
Probable – Open Pit	240	400	1,083	13,934
Probable – Underground	450	458	569	8,375
Total Mineral Reserves	-	**858**	**809**	**22,309**

16.0 MINING METHODS

16.1 OPEN PIT MINING

16.1.1 OPEN PIT MINING METHOD SELECTION

A conventional mining method operation consisting of drill and blast, load and haul is planned for the open pit. The mining operation will be carried out as an owner operated whilst the blasting operation will be carried out using a blasting contractor until an owner blasting crew is employed during the mine operation.

The deposit is approximately 1 km in length and ranges in width between 0.5 to 8.0 m. This geometry will require selective mining, accurate grade control and relatively smaller load-and-haul equipment to minimise the mining dilution and losses.

Drilling and blasting is planned in 10-m benches, with a blasthole diameter of 76 mm. Loading will be carried out in four, 2.5-m stacks or three, 3.3-m intervals using backhoe excavators. Twenty-tonne trucks were selected for waste and ore haulage. Following test blasts during pre-production, the option of blasting on 5 m will be investigated. Tetra Tech believes that this will not materially change the operating cost estimate and may have advantages for grade control. Double benching on the final pit wall could also be considered.

The ROM will either be hauled directly to the crusher or stockpiled next to the crusher to be used when required. The waste rock and the low-grade mineralised rock (resource balance as per Russian standards) will be hauled to the WRD and low-grade stockpile area.

16.1.2 PIT SLOPE CHARACTERISATION

SRK developed the open pit and underground geotechnical design criteria that were presented in the 2016 Mangazeisky Project Feasibility Study (Appendix B). This section presents a summary of the work carried out for open pit mining. The result of the kinematic analyses highlight that the maximum overall slope angle is constrained by the bench and berm geometry and inter-ramp angles (IRAs), which are based on the requirement to minimise kinematic instability and trap potential rock fall. Based on the analyses, the overall angles for the pit slope are presented in Table 16.1.

The maximum IRA of 56° is recommended for the hanging wall. Due to planar instability along bedding plans in the footwall, the maximum IRA for the footwall is set to 48°. Steeper slopes will start to undercut the bedding on an inter-ramp scale that could generate major instability.

Table 16.1 Recommended Slope Angles

Zone	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	IRA (°)
Hanging Wall	20	80	10	56
Footwall	20	70	11	48

Both sets of design geometry do not take ramps into account. For trenches, where hard rock is present, the criteria for the hanging wall and footwall shown in Table 16.1 will be used.

16.1.3 PIT OPTIMISATION

APPROACH

Pit optimisation was carried out on the updated orebody model using the GEOVIA Whittle™ software, to verify the economic portion of the orebody to be mined via open pit methods.

The application of the Lerchs-Grossmann algorithm provided a number of cones targeting the higher grade part of the deposit that were not practical for an open pit mine design and operation. The key to unlocking the value of the underlying ore body using an open pit was to use a slot type ramp configuration, instead of the conventional spiral ramp concept, that connected the identified individual cones into a single pit. While part of the slot type ramp adds to the waste rock in the pit design, it allows the pit slope angle to be maximised in the footwall and hanging wall, which results in an optimal strip ratio that still provides an economic open pit boundary. In addition it allows for a smoother integration with the underground mine design. In other words part of the waste stripping in the open pit could actually be considered as the amount of earth moving that could have been required to access the underground mine workings if there were no open pit.

The pit optimisation to define an economic boundary for the South Pit was carried out using shallower pit slope angles where in pit ramping system was necessary to access the mining benches.

OPTIMISATION PARAMETERS

The selected optimisation parameters are provided in Table 16.2.

Table 16.2 Optimisation Parameters

Parameter	Unit	Value	
Average Unit Mining Cost	$/t mined	2.50	
Process Unit Cost	$/t processed	57.00	
G&A Unit Cost	$/t processed	40.00	
Royalty	%	6.5	
Selling Cost	$/tr oz	0.50	
Discount	%	5	
Mining Loss	%	95	
Dilution	%	30	
Process Recovery	%	85	
Silver Price	$/tr oz	18.00	
Overall Slope Angle	**Unit**	**Base Case**	**Shallower Case**
Footwall	degrees	48	42
Hanging Wall	degrees	56	50

The mining cost used in the optimisation is the average mine operating unit cost that was estimated in Tetra Tech (2016).

Optimisation Summary

Initially, two optimisation scenarios were analysed based on the selected sets of pit slope angles: the base case slope angles as provided by SRK (Appendix B) and the alternative slope set approximately 6° shallower than the base case. The selected pit shell in the base case slope scenario was used as a guideline to design the North Pit and the generated pit shell from the shallower slope alternative was used for the South Pit design.

Owing to project time constraints, the optimisation analyses included both the North and South Pit areas together in each optimisation, and therefore the generated pit value in each case represents the overall pit operating cash flow. The North and South pit shells were selected to minimise the strip ratio while maintaining practical access to the open pit and underground operations.

A summary of the optimisation scenario results is presented in Figure 16.1, and Figure 16.2.

In both scenarios, pit shell 33 was selected as the optimal and practical pit boundary for the Project. The selected pit shell shown in Figure 16.1 was used for the design of the North pit and the selected shell shown in Figure 16.2 was used as a guideline for the South Pit design.

If open pit was the only mining method used to exploit the orebody, a larger pit shell could be selected as an optimal open pit boundary; however, the interaction with the underground mine design mitigated this decision.

A plan view of the selected pit shell contours for the pit design is presented in Figure 16.3.

Figure 16.1 Optimisation Summary for the Base Case

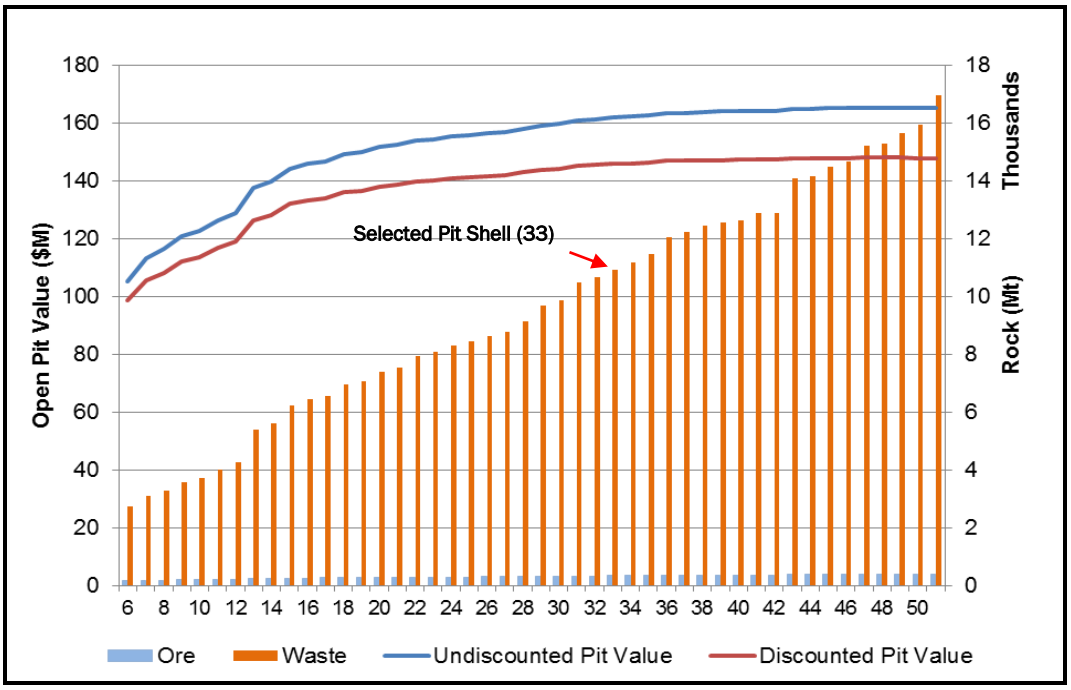


Figure 16.2 Optimisation Summary for the Scenario using Shallower Slope Angle

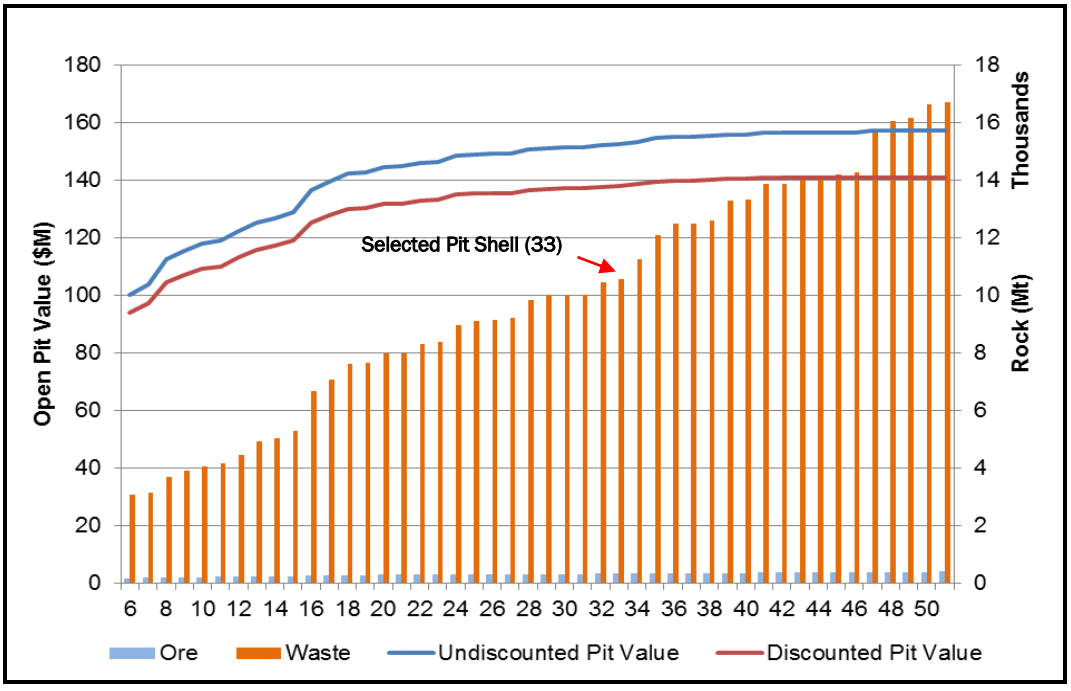
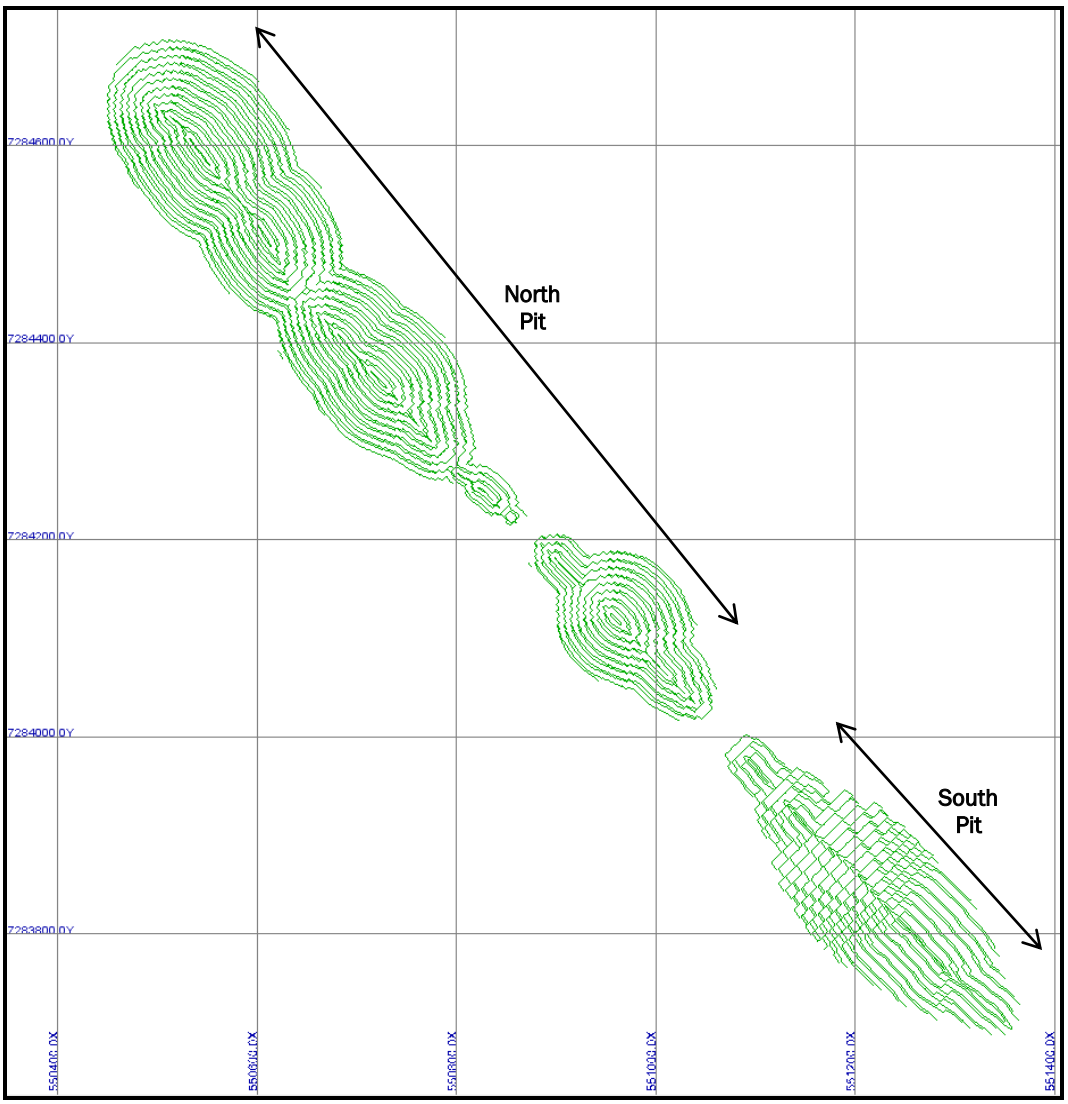


Figure 16.3 Plan View of the Selected Pit Shell Contours



In addition to the initial analyses, and as a final check, optimisation analysis was carried out using an average mining cost of $2.75/t mined on the base case scenario. The result of this analyses showed the effect of the higher mining cost on a pit shell with similar tonnes, as the selected base case, was approximately 1.8% less than the undiscounted pit value in the base case.

Sensitivity Analysis

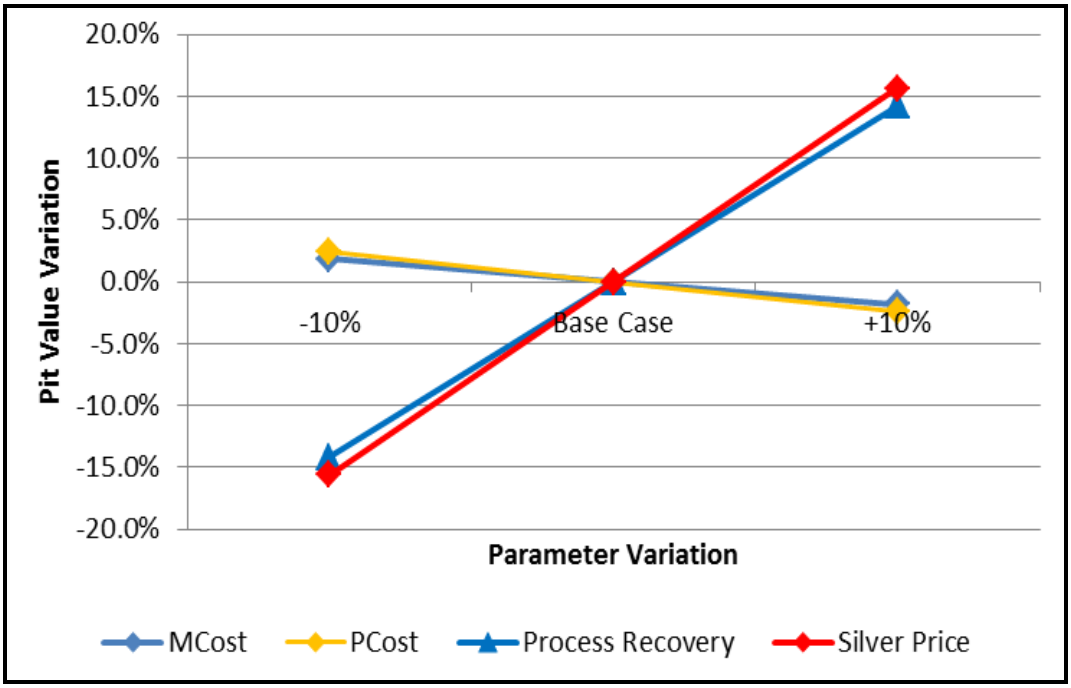
Sensitivity analyses were carried out on the pit operating value in the base case by varying the average mining cost, Pcost (process costs plus G&A costs), process recovery, and the silver price by ±10%. The results are presented in Figure 16.4 as a standard spider graph.

The effect of the mining and processing cost variation on the operating value of the selected pit shell is approximately 2.5%. However, a 10% change in the process recovery or silver price affects the undiscounted pit value by approximately 15%.

Higher process recovery and silver price result in an increase in the generated revenue; hence, the pit value is higher and vice versa.

The higher recoveries and metal prices result in a lower marginal cut-off grade; however, due to the grade distribution in the existing orebody the effect of the cut-off grade on the ore tonnage is negligible.

Figure 16.4 Summary of Sensitivity Analyses on Pit Value



Orebody Sensitivity to Cut-off Grade

As discussed in Section 2.2, based on the selected parameters, the calculated marginal cut-off grade is approximately 240 g/t. A small cut-off grade variation shows a minimal effect on the overall tonnage in the North Pit, while the effect on the South Pit is negligible, if any. The grade-tonnage curves for the North and South pits are presented in Figure 16.5 and Figure 16.6.

The selected cut-off grade results in a better continuation of the orebody, allowing for a more selective mining possibility. This will improve the overall mining loss and dilution during the mining operation.

Figure 16.5 Grade-Tonnage Curve for North Pit

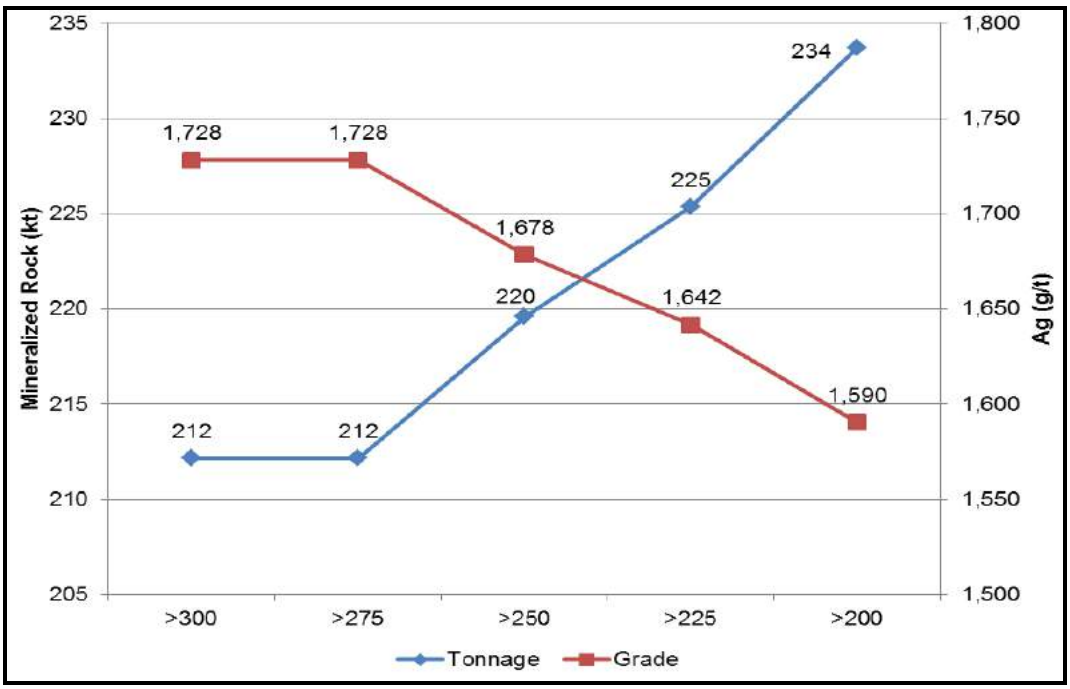
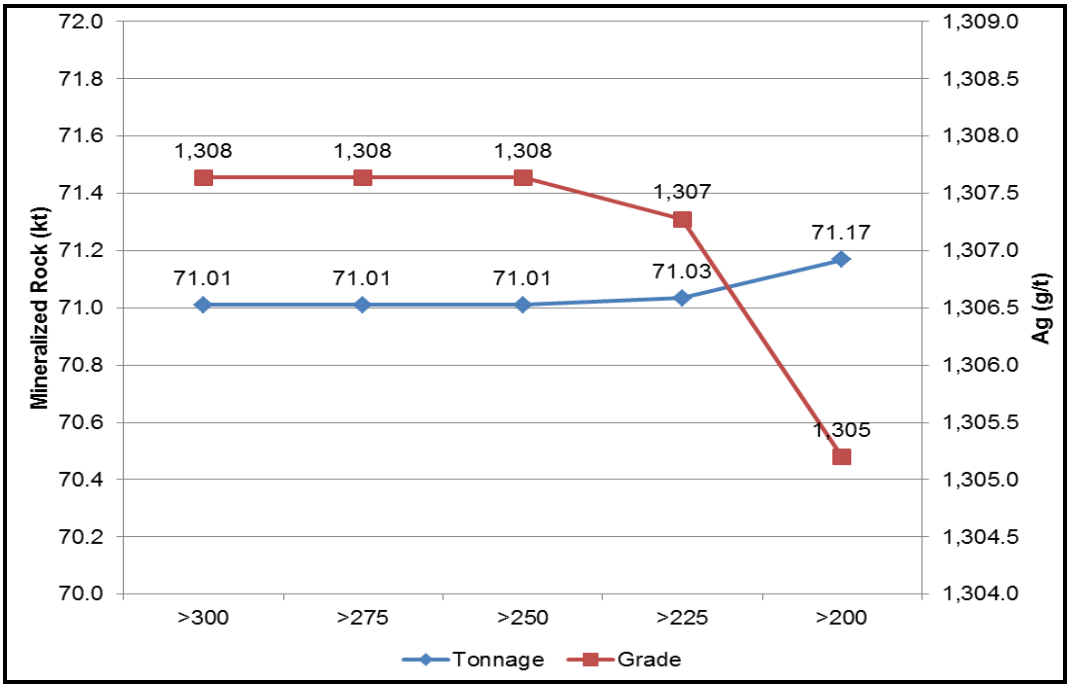


Figure 16.6 Grade-Tonnage Curve for South Pit



16.1.4 Open Pit Mine Design

GEOVIA GEMS™ software was used for the pit and WRD designs and for preparation of the end-of-period pit plans and rock inventory estimation.

The open pit consists of two separate pits; North and South pits, with separate access roads. The North Pit is designed in two phases where the Phase 1 Pit is inside the final North Pit and the South Pit is designed in one phase.

Open Pit Design Parameters

The basic open pit design parameters are determined by the geotechnical design criteria as set out by SRK (Appendix B) and the selected equipment for the mining operation. A summary of these parameters are provided in Table 16.3.

Table 16.3 Pit Design Parameters

Parameter	Unit	Value
Bench Height	m	10
Face Angle	°	80
Berm Width	m	5.0 to 7.3
North Pit Ramp Width	m	10.0 to 16.0
South Pit Ramp Width	m	14.0 to 18.0
Ramp Gradient	%	10
Minimum Working Width for Main Mining Areas	m	15
Minimum Working Width for Bottom/Final Benches	m	<10

North Pit

The North Pit was divided into two phases with the Phase 1 pit designed inside the final North Pit. A slot type ramping system is used as the main access to benches in the North Pit.

The North Pit design took into consideration the integration of the underground mine design by allowing pit benches and the internal ramping system to be used as an access to underground openings. In addition, the design aimed to provide:

- potential in-pit parking or service area for mining equipment
- areas within the pit to construct water collection sumps and pumping stations to prevent pit flooding
- temporary in-pit storage or mineral stockpile in the pit.

The two North Pit phases are shown in Figure 16.7 and Figure 16.8.

The Phase 1 pit is located at the northwest end of the final pit, with a total length of 490 m, maximum width of 105 m, and an overall height of 55 m. The pit exit, on the southeast side of the pit, is on the 1,166 m elevation and the pit bottom is on bench

1,130. A horizontal area of 50 m long and an average width of 25 m was designed on bench 1,145 to allow for a small sump or in-pit multi-purpose parking area.

The haul distance from the pit exit to the crusher is approximately 1,500 m at an average 3.5% gradient. The distance to the WRD and low-grade stockpile located adjacent to the WRD is 250 m. Temporary ramps could be used from various pit locations on the north wall to access both the crusher and the WRD to shorten the haul distances from the upper benches. Such roads should be considered during detail design by the mine planning engineer.

The Phase 2 pit (final pit) is an expansion of the Phase 1 pit, mainly towards the southeast direction. The North Pit is 860 m long and has a maximum width of 150 m. The pit bottom is in the northwest side of the pit on bench 1,085, resulting in an overall slope height of approximately 100 m.

The pit exit is on the 1,154 m elevation contour and in-pit ramp slopes down at 10% gradient to the bottom, only interrupted by a 65 m horizontal area on bench 1,105. The total ramp length excluding the horizontal portion is 500 m. The main ramp is 16 m wide to provide sufficient width for a two-way road traffic and working space during the mining operation. This selected width, in excess of the conventional 3.5 times truck width, is specified in acknowledgement of the unique slot-type ramp system with narrow working conditions when compared to a side hill ramp system where one side of the ramp is typically open allowing for easier excavation. In the final ramp sectors the ramp width is reduced to 10 m resulting in a slower mining operation in the final mining period.

An additional horizontal area is also designed on bench 1,135, next to the ramp, where part of the remaining ore vein could be extracted at the end of the open pit mine life.

Both horizontal areas (wider benches) could be used for water collection or in-pit parking areas, temporary storage, and equipment maintenance areas.

The road distance from the pit exit, located on the southeast side of the pit, to the crusher is approximately 1,900 m with an average 3.5% gradient. The haul distance from the pit exit to the WRD and low-grade stockpile is approximately 450 m. Similar to the Phase 1 pit, temporary ramps could be used to shorten the waste haulage distance from the upper benches. These ramps will be excavated when the pit progresses to the lower benches.

Figure 16.7 Plan View of the Phase 1 Pit (North Pit)

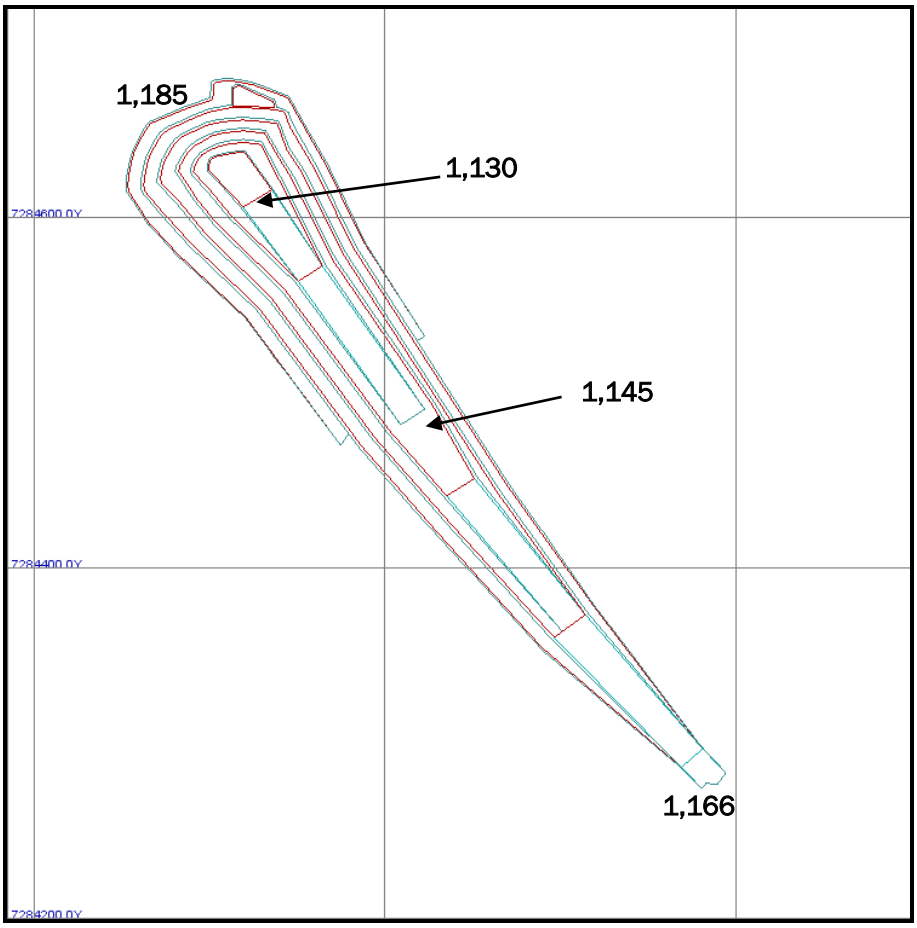
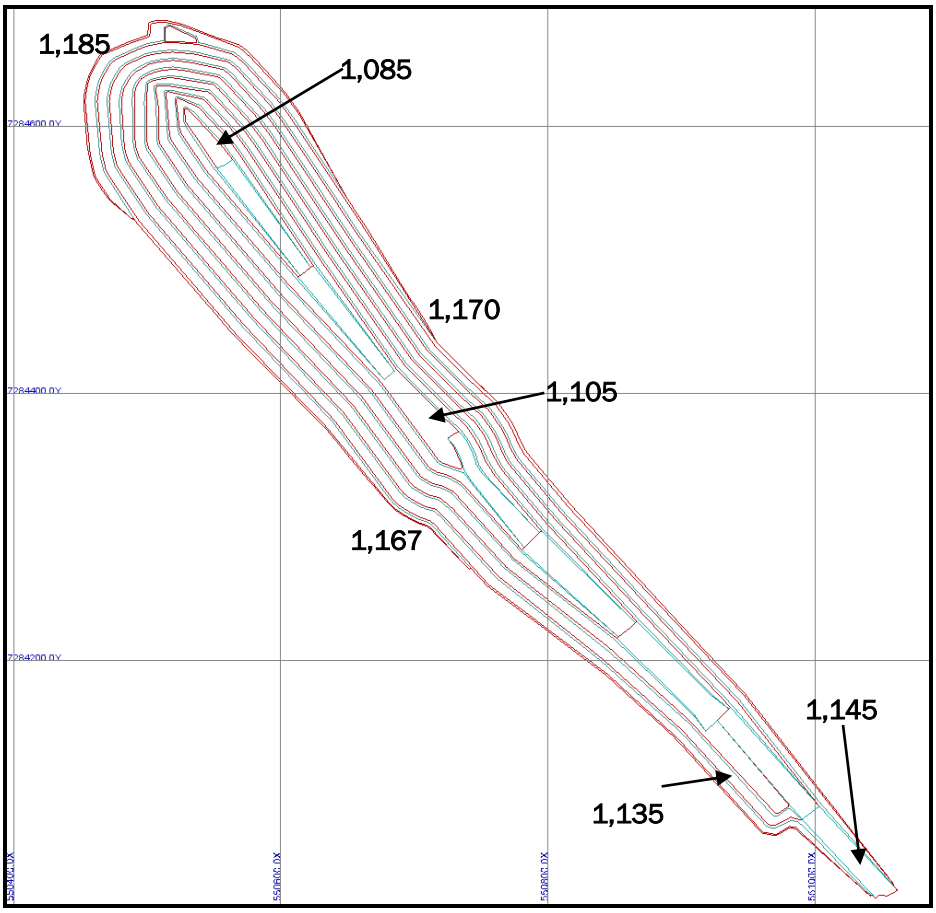


Figure 16.8 Plan View of the Final Pit (North Pit)



South Pit

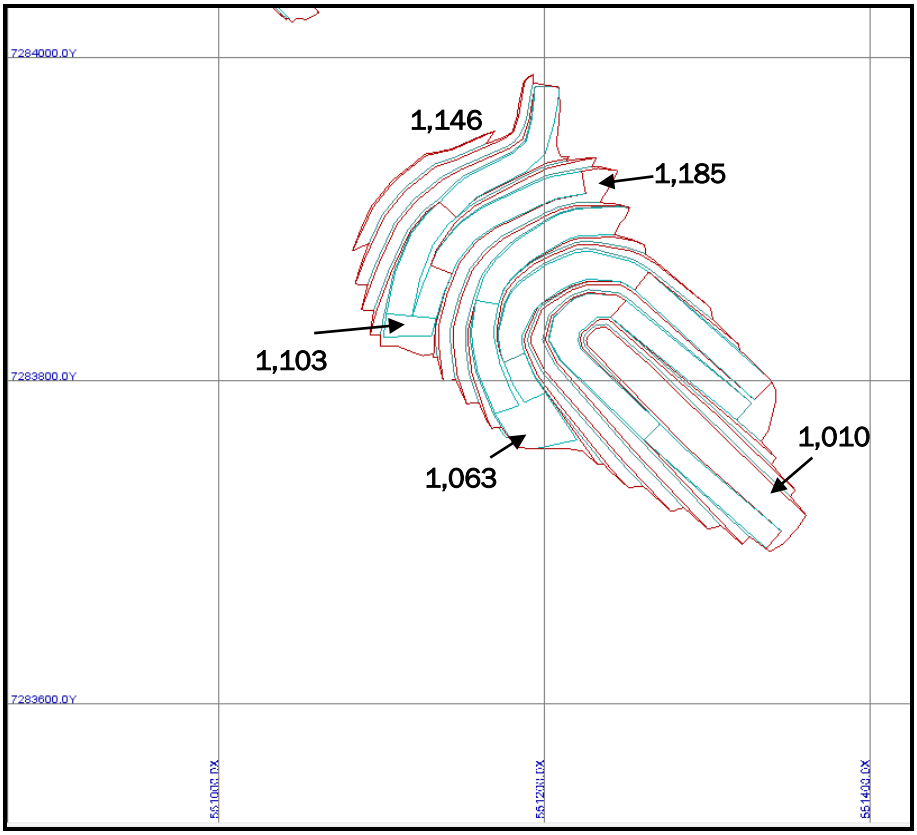
The South Pit, located southeast of the North Pit, is a side-hill pit with separate access road and an in-pit ramping system. The bench by bench access is provided by an in-pit spiral ramping system that, at the end of the open pit life, will be used as a portal to the underground mine (Figure 16.9). Access to the South Pit is from the 1,164 m elevation of the main crusher access road to the South Pit exit on elevation 1,120 m. The haul distance is 590 m at an average 7.5% gradient. Above the pit exit elevation, access could be provided using temporary ramps as an extension of the main road or an in-pit ramp. The same access road will be used to access the WRD and the low-grade stockpile.

The total length of the in-pit ramp is approximately 1,000 m, 300 m of which is above bench 1,090 and 700 m is the ramp from bench 1,080 to the pit bottom. Access for benches 1080 and 1090 is designed using an outside road at an approximate 9.0% gradient. The final ramp sector (1,020 to 1,010 m) is designed at 11% gradient. A backfill area is designed to extend the 1010 m pit bottom elevation to create working space and turning area at the end of the ramp.

The ramp width varies from 14 m on the sides of the pit to 18 m. This is to provide wider access in the areas where the underground portals will be designed and constructed, and to reduce the strip ratio on the sides.

The South Pit is 360 m long, 190 m wide, and the overall height of the northwest pit wall is 136 m.

Figure 16.9 Plan View of the Final Pit (South Pit)



16.1.5 Rock Inventories per Pit Phase/Area

The in situ rock inventory per designed pit phases and areas is shown in Table 16.4. Total Mineral Reserve is calculated after applying mining dilution and losses to the in situ above cut-off grade mineral inventory inside the designed pits.

Table 16.4 In Situ Rock Inventories in the Designed Pit Phases (September 2016)

	Phase 1		Phase 2		South Pit		Total	
	kt	Ag g/t	kt	Ag g/t	kt	Ag g/t	kt	Ag g/t
Ore	123	931	188	1,222	88	1000	400	1083
LG Stockpile	40	97	111	95	114	85	265	91
Waste	1,342	-	5,724	-	2,743	-	9,808	-
Total	**1,504**	**-**	**6,023**	**-**	**2,946**	**-**	**10,473**	**-**

Note: *Totals may not add due to rounding.

16.1.6 Waste Rock Dump and Low-grade Stockpile

The WRD and low-grade stockpile are designed on the hill side, north of the North Pit. In this design, the stockpile is assumed to be on the shallower end of the designed dumping area, on the northwest side.

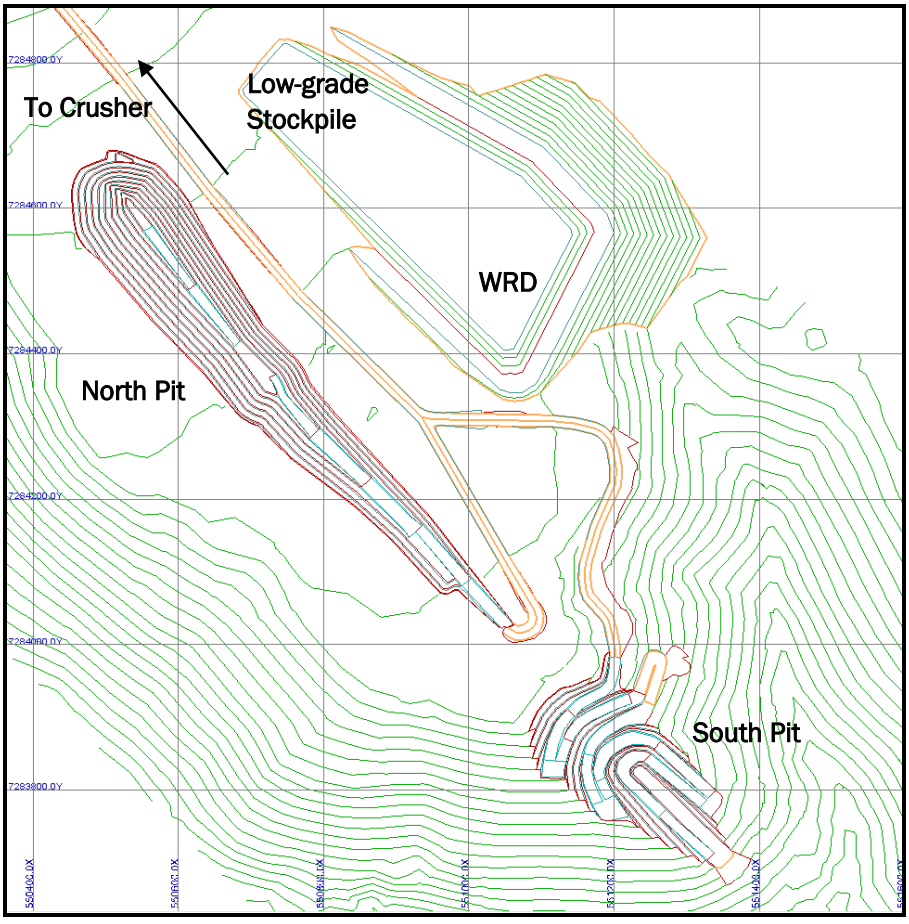
As a result of the natural topography the WRD height is less than 3 m on the northwest side, and gradually increases to approximately 30 m on the south end and 70 to 85 m on the north slope. The WRD is designed at a 25° slope angle with an overall dimension of 650 m in length and approximately 400 m in width.

It must be noted that the provided topography surface was somewhat short of covering the WRD area; therefore, the original surface was expanded for wider coverage. The actual topography may be different compared to the one used as the basis to design the WRD, due to the artificial expansion of the surface digital terrain model, but due to the sufficient space and the possibility to increase the WRD height, no issues are expected in expanding the WRD area.

The total design capacity of the WRD is 3.8 Mm^3.

The general layout of the designed pits, WRD, and low-grade stockpile, as well as the main haul roads are presented in Figure 16.10.

Figure 16.10 General Layout of the Design Pits, Waste Dump and Haul Roads



16.1.7 HAUL ROADS

The haul roads to the North and South pits are designed at a maximum of 10% gradient and 16 m width. The selected road width is more than the required 3.5 times the truck width (20 to 30 t trucks) for a safe two-way road traffic. This allows

for flexibility during winter months for snow clearing. Road-side berms should be constructed with a height of 60% of the largest tire size of the deployed equipment for road safety reasons.

In preparation of the estimate the cut and fill volumes for the access road to the South Pit are estimated to balance the earthmoving requirement. The access road to the crusher is mainly a cut-and-grading operation, with some back filling in a few areas.

The presented road design in the general layout drawing (Figure 16.10) is mainly to show the road centre line and the projected road sides for access to the pits, WRD, and low-grade stockpile. A more detail design will be required for the construction stage.

16.1.8 Open Pit Schedule

The open pit mining schedule was prepared assuming a mining operation start date of May 2017 to allow for the commencement of process plant production in December 2017. The presented schedule was prepared by scheduling in months during 2017 and 2018, Quarters for the remaining life of the open pit.

The rock inventories were calculated using the end-of-period pit plans on a bench-by-bench basis and were used to prepare the monthly schedule after applying the mining dilution and losses to the ore tonnage.

Bench plans were prepared to show the approximate pit face position at the end of each period and to ensure availability of access to each bench and working area when the pit progresses to the next period. Some temporary ramps could be used to minimise the haul distance from the pit benches to WRD/low-grade stockpile and the crusher that will be excavated within the final pit phase. These are not shown in the pit plans.

In preparing the end of period pit plans, the aim was to maintain flexibility in selection of the ore and waste working faces, while exposing sufficient ore for excavation and delivery from the pit benches to the process plan in most periods. Providing access from one period to the following period was also a priority in preparing the plans. The mine planning engineer will be able to select the optimum working faces per period using the grade control results and the detail mining schedule that will be prepared on site.

During the pre-stripping period (May to November 2017), sufficient ore is scheduled to be stockpiled for approximately 3.5 months of process plant production. While the stockpile will be available for use at any time, in this schedule, the use of the initial stockpile is limited to the periods where the grade of the ROM is lower than the stockpiled grade or during periods where, due to stripping requirements, limited ROM is available from the open pit. At the end of the open pit life, when the pit is depleted, the stockpile will be utilised to maintain process plant production and to complement the underground ore tonnage destined for the process plant.

Tetra Tech notes that the use of the crusher stockpile in winter may be hampered by freezing. ZAO Prognoz advised that light blasting is regularly used in Russia to liberate frozen stockpile material. Although the use of stockpile in winter is

minimised in this schedule, 44% of the process plant throughput is scheduled to be supplied from the stockpile in the Q1 2018. The crusher stockpile will be a live stockpile and could be depleted or grown, as required, even if in this schedule there are periods with no stockpile use.

At the start of the project the Resource was declared on the Russian State register at a cut-off grade of 75 g/t silver. Any material with a grade above this cut-off cannot be discarded as waste, which means that the 75 to 240 g/t silver material will be stored in a low-grade stockpile adjacent to the WRD. No dilution or losses are applied to this material. It is likely that during the mining operation some of the low-grade material will be diluted with the ore above the selected cut-off grade. The low-grade stockpile could also be used as a depository for the waste rock containing mineralised material as the mining loss. ARD is not expected from the WRD. Based on the selected economic parameters, this stockpile has no positive economic value and will not be used as a revenue generating material.

The mining schedule, the crusher ROM stockpile movement, marginal cut-off grade stockpile inventory, and the process plant production schedule for the open pit are presented in Table 16.5 and Figure 16.11.

The total scheduled material to be hauled to the crusher is 400 kt at an average silver grade of 1,083g/t. The total waste rock is 10.1 Mt and the low-grade material (75 to 240 g/t category) will be 265 kt at an average 91 g/t silver. The calculated overall strip ratio for this operation is 25.2.

The stockpile remaining at the end of 2020 totals 65,209 t at 316 g/t silver. This low-grade material is drawn down over the next three years, as it is assumed that ore from the underground operation can be fed to the plant from February 2021. Therefore, the high-grade material (open pit or underground) is preferentially fed first and the low-grade ROM stockpile is fed last.

The schedule for the Vertikalny open pit is shown for each bench in Figure 16.12 to Figure 16.46.

Table 16.5 Open Pit Mining Operating, Stockpile Inventory, and Plant Production Schedule

Rock Type	Unit	2017							
		May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ore	t	1,274	-	3,066	9,167	4,463	13,198	6,699	-
	g/t	1,425	-	252	596	1,148	388	994	-
Low-Grade Stockpile	t	-	-	12	3,737	24	3,937	3,273	-
	g/t	-	-	76	87	69	119	77	-
Waste	t	61,178	91,419	93,558	89,358	94,615	71,965	49,418	-
Total	**t**	**62,452**	**91,419**	**96,636**	**102,262**	**99,103**	**89,100**	**59,390**	**-**
Strip Ratio	-	48	-	31	10	21	6	8	-
Stockpile									
Opening Balance	t	-	-	1,274	4,339	13,507	17,970	31,167	37,866
	g/t	-	-	1,425	596-	596	733	587	659
In	t	1,274	-	3,066	9,167	4,463	13,198	6,699	-
	g/t	1,425	-	252	596	1,148	388	994	-
Out	t	-	-	-	-	-	-	-	6,900
	g/t	-	-	-	-	-	-	-	962
Closing Balance	t	1,274	1,274	4,339	13,507	17,970	31,167	37,866	30,966
	g/t	1,425	1,425	596	596	733	587	659	592
Low-Grade Stockpile	t	-	-	12	3,749	3,773	7,710	10,983	10,983
	g/t	-	-	76	87	87	103	95	95
Plant									
Open Pit	t	-	-	-	-	-	-	-	6,900
	kg	-	-	-	-	-	-	-	4,915
	g/t	-	-	-	-	-	-	-	712

table continues...

Rock Type	Unit	2018											
		Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ore	t	7,557	9,273	13,678	10,717	12,801	10,219	9,877	10,311	9,257	9,833	9,628	7,910
	g/t	760	638	658	760	860	761	965	1,169	683	1,131	1,262	1,861
Low-Grade Stockpile	t	-	6,545	3,218	5,515	5,964	1,162	9,635	3,642	3,272	3,921	5,618	4,001
	g/t	-	116	76	98	93	73	92	73	92	81	85	69
Waste	t	175,259	134,553	368,131	340,495	336,424	353,747	347,724	353,372	336,925	346,991	323,789	225,131
Total	**t**	**182,816**	**150,372**	**385,027**	**356,728**	**355,189**	**365,128**	**367,236**	**367,326**	**349,454**	**360,745**	**339,034**	**237,042**
Strip Ratio		23	15	27	32	27	35	36	35	37	36	34	29
Stockpile													
Opening Balance	t	30,966	31,523	32,358	38,369	38,369	41,827	43,006	43,540	44,509	44,725	45,216	45,802
	g/t	592	519	400	294	294	297	270	264	303	249	249	289
In	t	7,557	9,273	13,678	10.717	12,801	10,219	9,977	10,311	9,257	9,933	9,628	7,910
	g/t	760	638	658	760	860	761	965	1,169	683	1,131	1,262	1,861
Out	t	7,000	8,438	9,342	9,041	9,342	9,041	9,342	9,342	9,041	9,342	9,041	9,342
	g/t	1,103	1,105	1,117	927	1056	948	1,035	1.074	963	1,176	1,122	1,151
Closing Balance	t	31,523	32,358	36,693	38,369	41,827	43,006	43,540	44,509	44,725	45,216	45,802	44,369
	g/t	519	400	314	294	297	270	264	303	249	249	289	388
Low-Grade Stockpile	t	10,983	17,529	20,747	26,262	32,226	33,388	43,023	46,665	49,937	53,858	59,476	63,476
	g/t	95	103	99	99	98	97	96	94	94	93	92	91
Plant													
Open Pit	t	7,000	9,041	9,342	9,041	9,342	9,041	9,342	9,342	9,041	9,342	9,041	9,342
	kg	7,240	9,521	9,797	9,478	9,843	9,499	9,810	9,882	9,508	10,446	10,131	10,506
	g/t	1,034	1,053	1,049	1,048	1,054	1,051	1,050	1,058	1,052	1,118	1,121	1,125

table continues...

Rock Type	Unit	2019											
		Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ore	t	9,299	9,299	9,299	9,084	9,084	9,084	9,086	9,086	9,086	9,407	9,407	9,407
	g/t	916	916	916	1,086	1,086	1,086	1,172	1,172	1,172	1,288	1,288	1,288
Low-Grade Stockpile	t	8,406	8,406	8,406	9,976	9,976	9,976	7,907	7,907	7,907	7,887	7,887	7,887
	g/t	78	78	78	89	89	89	91	91	91	92	92	92
Waste	t	264,516	264,516	264,516	353,219	253,391	353,219	348,410	348,410	348,410	263,017	263,017	263,017
Total	**t**	**282,221**	**282,221**	**282,221**	**372,280**	**272,451**	**372,280**	**365,403**	**365,403**	**365,403**	**280,311**	**280,311**	**280,311**
Strip Ratio		29	29	29	40	29	40	39	39	39	29	29	29
Stockpile													
Opening Balance	t	44,369	44,326	45,186	45,142	45,185	44,927	44,970	44,714	44,457	44,502	44,566	44,932
	g/t	388	346	307	249	249	249	249	249	249	249	249	249
In	t	9,299	9,299	9,299	9.084	9,084	9,084	9.086	9,086	9.086	9,407	9,407	9.407
	g/t	916	916	916	1,086	1,086	1,086	1,172	1,172	1,172	1,288	1,288	1,288
Out	t	9,342	8,438	9,342	9,041	9,342	9.041	9,342	9,342	9,041	9,342	9.041	9,342
	g/t	1,112	1,183	1,198	1,090	1,063	1,090	1,147	1,147	1,177	1,295	1,330	1,295
Closing Balance	t	44,326	45,186	45,142	45,185	44,927	44,970	44,714	44,457	44,502	44,566	44,932	44,997
	g/t	346	307	249	249	249	249	249	249	249	249	249	249
Low-Grade Stockpile	t	71,882	80,288	88,694	98,670	108,646	118,622	126,530	134,437	142,344	150,231	158,119	166,006
	g/t	89	88	87	87	88	88	88	88	88	88	89	89
Plant													
Open Pit	t	9,342	8,438	9,342	9,041	9,342	9,041	9,342	9,342	9,041	9,342	9,041	9,342
	kg	10,554	9,462	10,476	10,205	10,906	10,371	10,715	10,715	10,639	12,102	11,932	12,191
	g/t	1,130	1,121	1,121	1,129	1,167	1,147	1,147	1,147	1,177	1,295	1,320	1,305

table continues...

Rock Type	Unit	2020											
		Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ore	t	9,608	9,608	9,608	9,430	9,430	9,430	9,089	9,089	9,089	15,377	15,377	15,377
	g/t	1,370	1,370	1,370	1,343	1,343	1,343	1,442	1,442	1,442	1,185	1,185	1,185
Low-Grade Stockpile	t	6,023	6,023	6,023	6,595	6,595	6,595	8,883	8,883	8,883	11,383	11,383	11,383
	g/t	92	92	92	95	95	95	95	95	95	95	95	95
Waste	t	176,156	176,156	176,156	157,009	56,196	157,009	149,924	149,924	149,924	159,273	159,273	159,273
Total	**t**	**191,787**	**191,787**	**191,787**	**173,034**	**72,221**	**173,034**	**167,896**	**167,896**	**167,896**	**186,034**	**186,034**	**186,034**
Strip Ratio	-	19	19	19	17	7	17	17	17	17	11	11	11
Stockpile													
Opening Balance	t	44,997	45,262	46,131	46,396	46,786	46,874	47,263	47,009	46,756	46,803	52,838	59,175
	g/t	249	249	250	250	251	251	251	251	251	251	277	300
In	t	9,608	9,608	9,608	9,430	9,430	9,430	9,089	9,089	9,089	15,377	15,377	15,377
	g/t	1,370	1,370	1,370	1,343	1,343	1,343	1,442	1,442	1,442	1,185	1,185	1,185
Out	t	9,342	8,740	9,342	9,041	9,342	9.041	9,342	9,342	9,041	9,342	9,041	9,342
	g/t	1,401	1,477	1,401	1,388	1,353	1,388	1,410	1,410	1,448	1,640	1,675	1,640
Closing Balance	t	45,262	46,131	46,396	46,786	46,874	47,263	47,009	46,756	46,803	52,838	59,175	65,209
	g/t	249	250	250	251	251	251	251	251	251	277	300	316
Low-Grade Stockpile	t	172,029	178,052	184,074	190,669	197,264	203,859	212,743	221,626	230,510	241,893	253,276	264,659
	g/t	89	89	89	89	90	90	90	90	90	90	91	91
Plant													
Open Pit	t	9,342	8,740	9,342	9,041	9,342	9,041	9,342	9,342	9,041	9,342	9,041	9,342
	kg	12,983	12,174	12,983	12,643	12,994	13,023	13,186	13,170	13,093	15,317	15,148	15,317
	g/t	1,390	1,393	1,390	1,398	1,391	1,440	1,411	1,410	1,448	1,640	1,675	1,640

table continues...

Rock Type	Unit	2021								Total
		Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	
Ore	t	-	-	-	-	-	-	-	-	400,068
	g/t	-	-	-	-	-	-	-	-	1,083
Low-Grade Stockpile	t	-	-	-	-	-	-	-	-	264,659
	g/t	-	-	-	-	-	-	-	-	91
Waste	t	-	-	-	-	-	-	-	-	9,607,982
Total	**t**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**10,272,710**
Strip Ratio		-	-	-	-	-	-	-	-	25
Stockpile										
Opening Balance	t	65,209	55,867	52,080	52,080	47,559	47,559	47,559	47,559	-
	g/t	316	276	255	255	255	255	255	255	-
In	t	-	-	-	-	-	-	-	-	-
	g/t	-	-	-	-	-	-	-	-	-
Out	t	9,342	3,787	-	4,521	-	-	-	-	-
	g/t	562	562	-	255	-	-	-	-	-
Closing Balance	t	55,867	52,080	52,080	47,559	47,559	47,559	47,559	47,559	-
	g/t	276	255	255	255	255	255	255	255	-
Low-Grade Stockpile	t	264,659	264,659	264,659	264,659	264,659	264,659	264,659	264,659	
	g/t	91	91	91	91	91	91	91	91	
Plant										
Open Pit	t	9,342	8,438	9,342	9,041	9,342	9,041	9,342	716	400,068
	kg	5,247	3,314	2,385	2,308	2,385	2,308	2,385	183	433,389
	g/t	562	393	255	255	255	255	255	255	1,083

Figure 16.11 Open Pit Mining Schedule and Plant Head Grade from the Open Pit

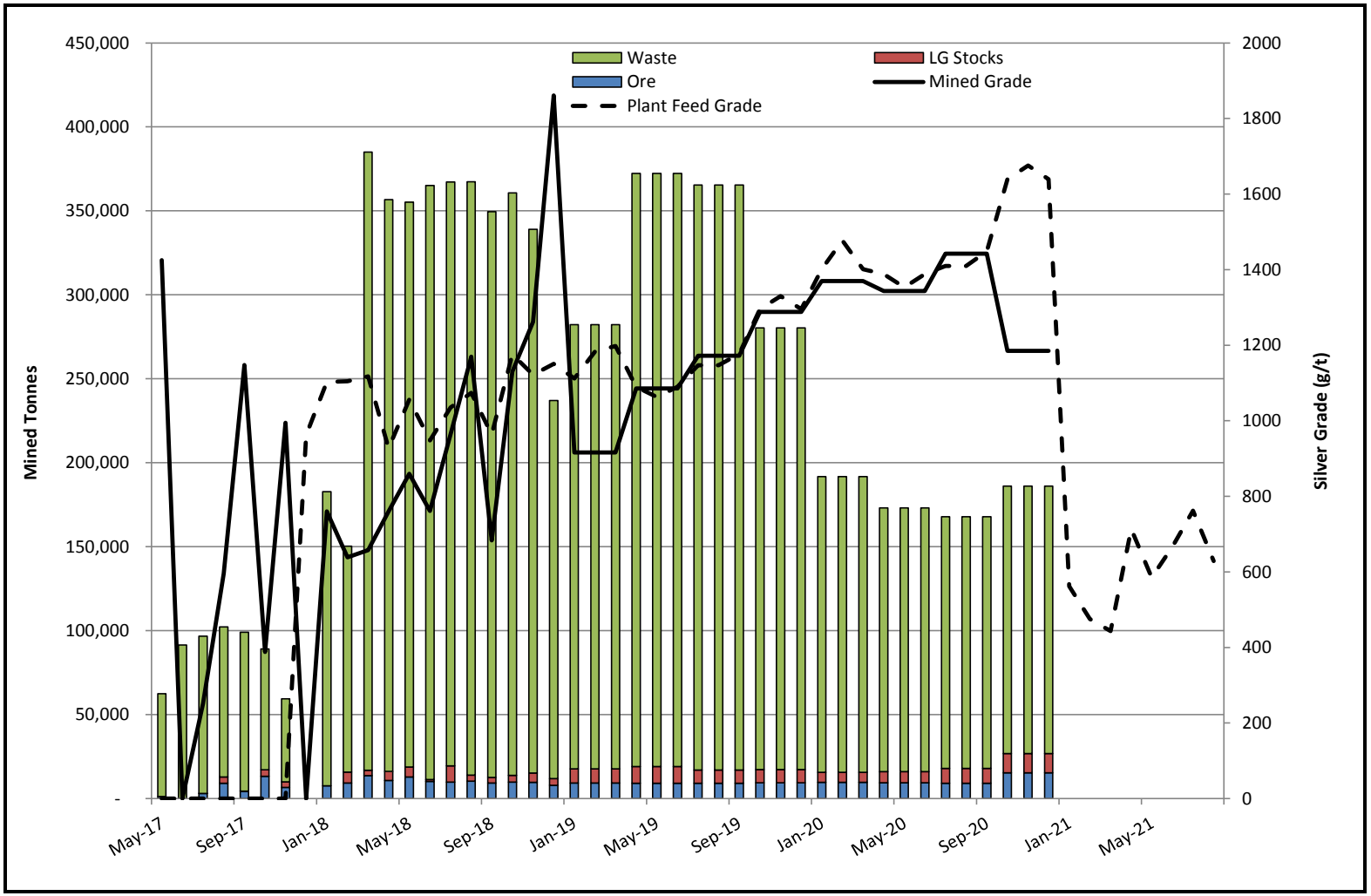


Figure 16.12 Mine Schedule Section Level 1180

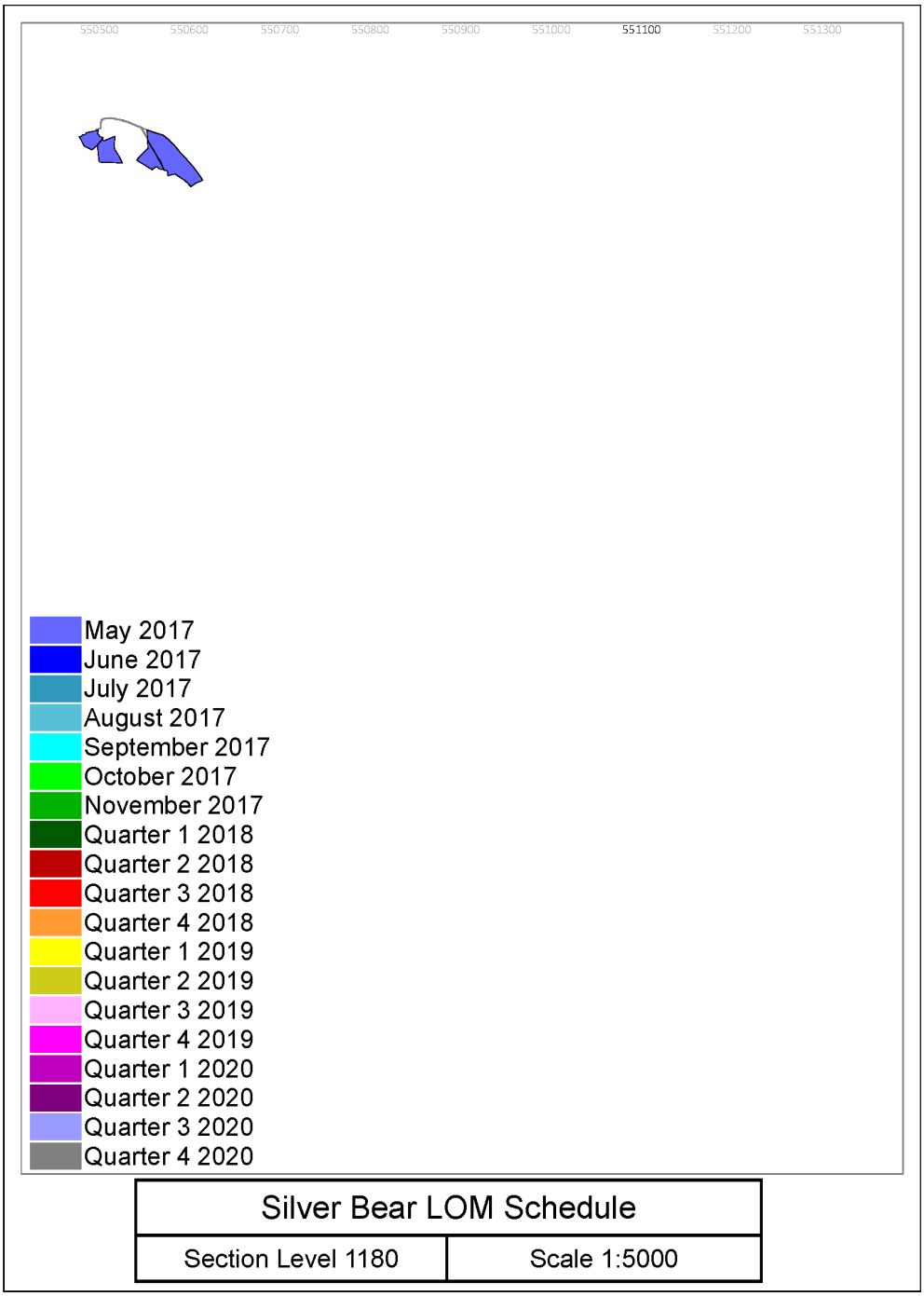


Figure 16.13 Mine Schedule Section Level 1175

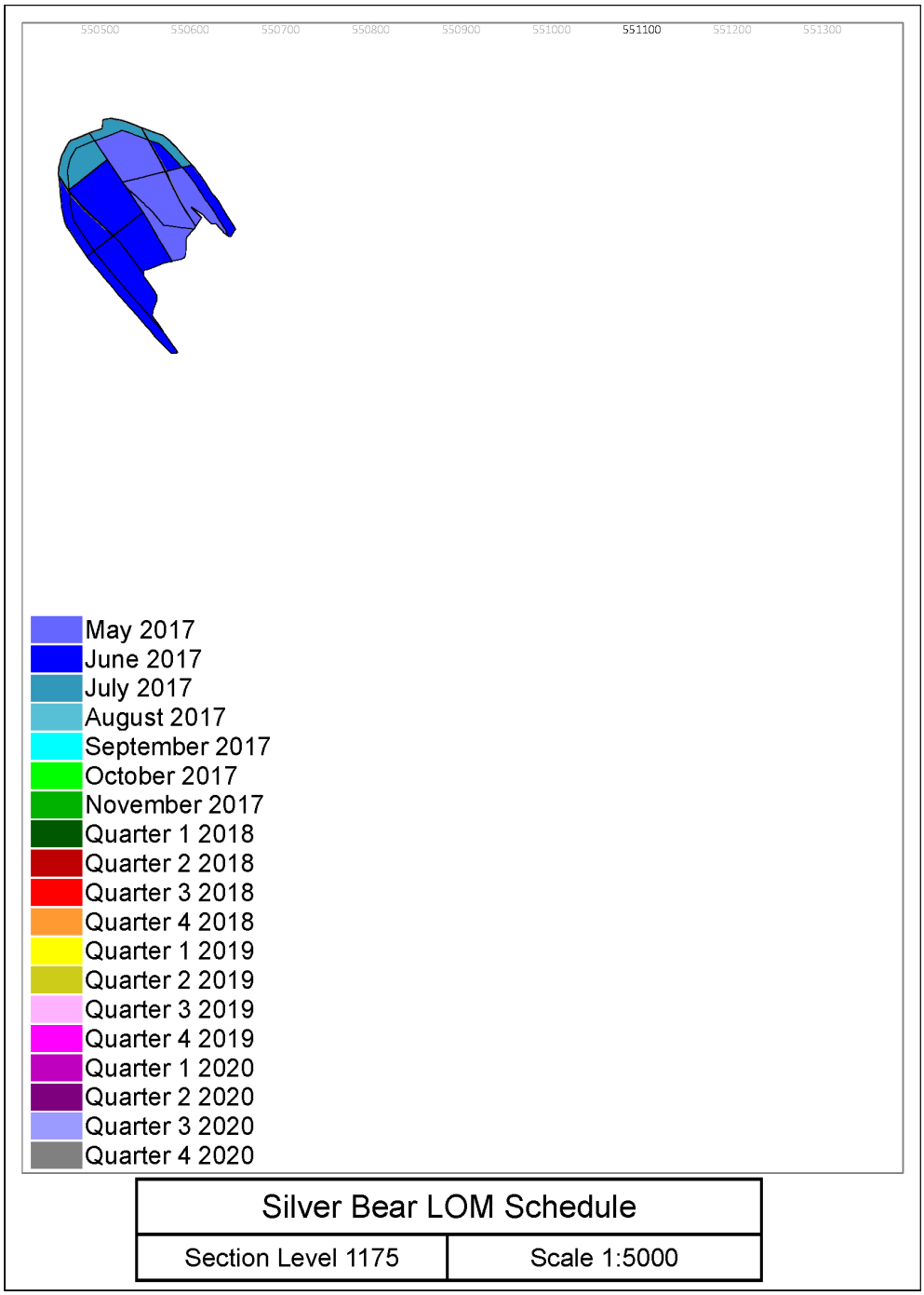


Figure 16.14 Mine Schedule Section Level 1170

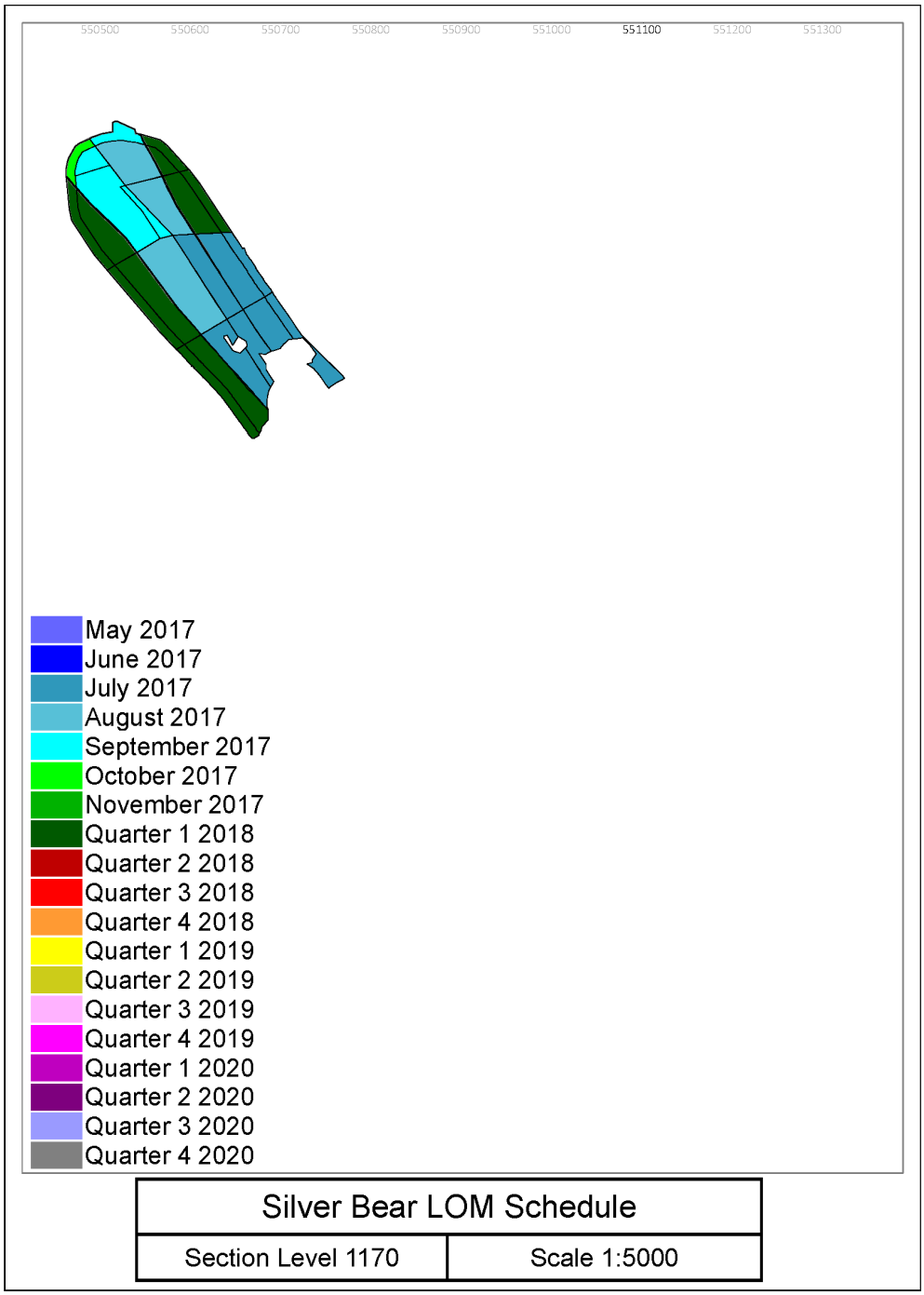


Figure 16.15 Mine Schedule Section Level 1165

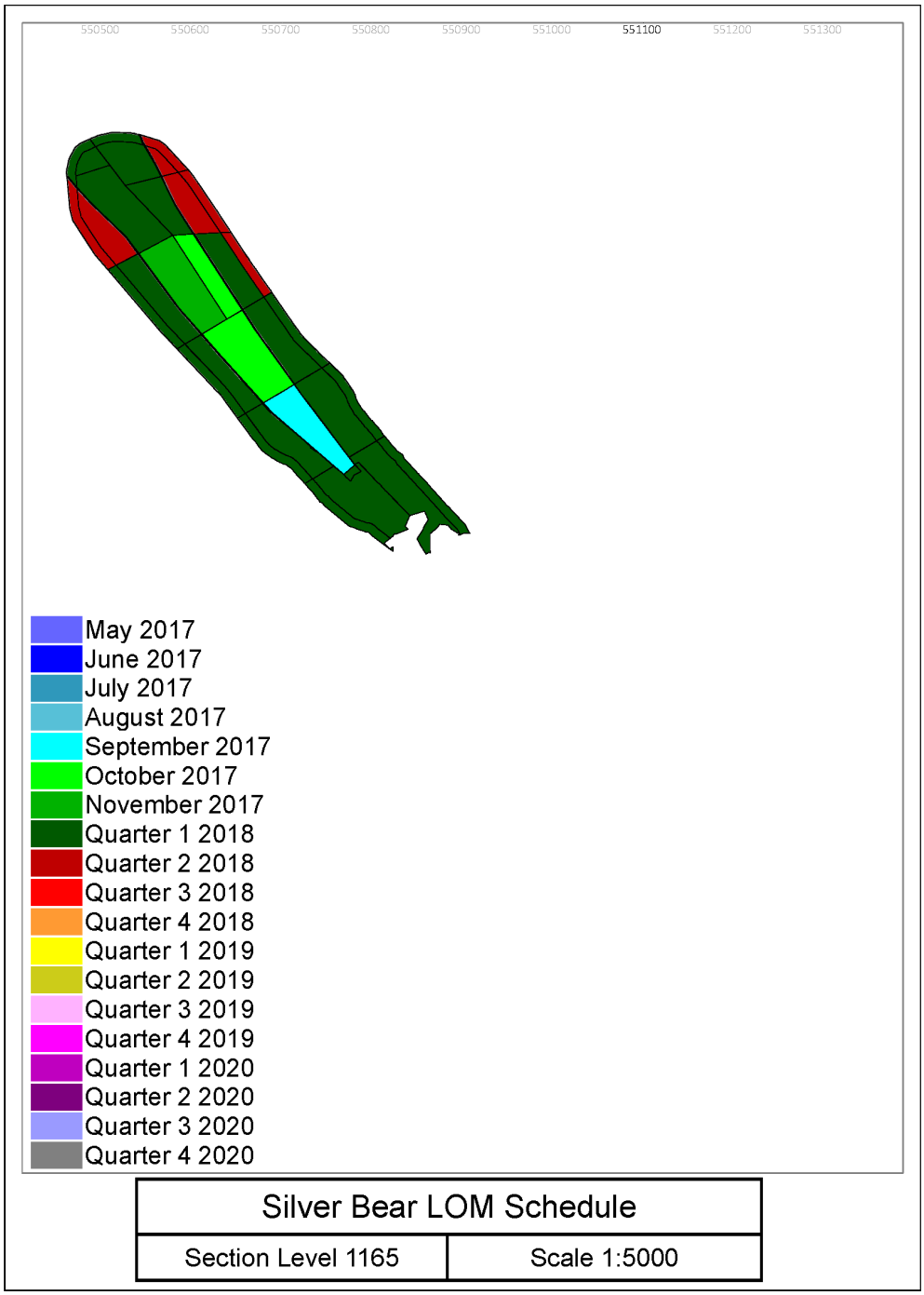


Figure 16.16 Mine Schedule Section Level 1160

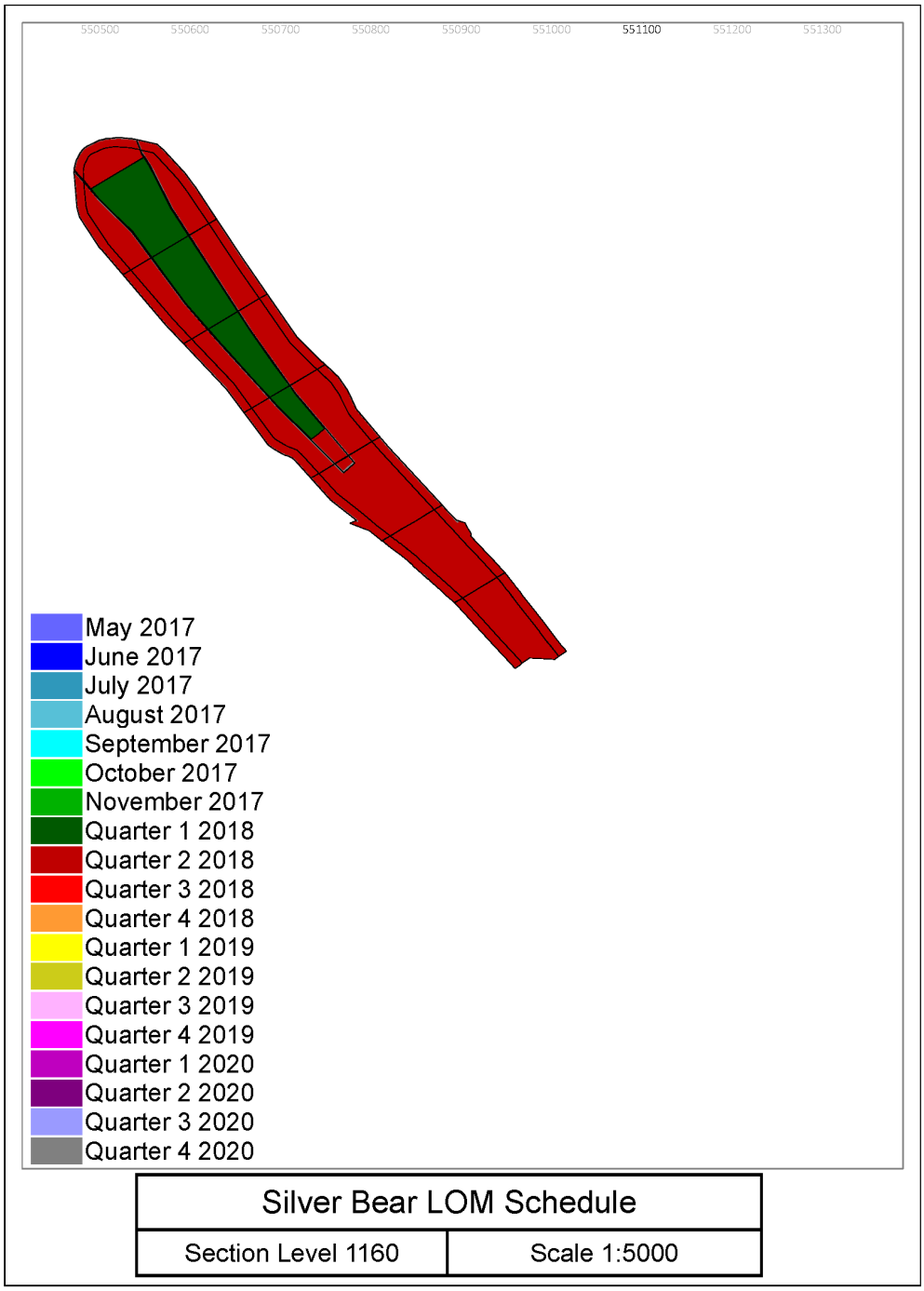


Figure 16.17 Mine Schedule Section Level 1155

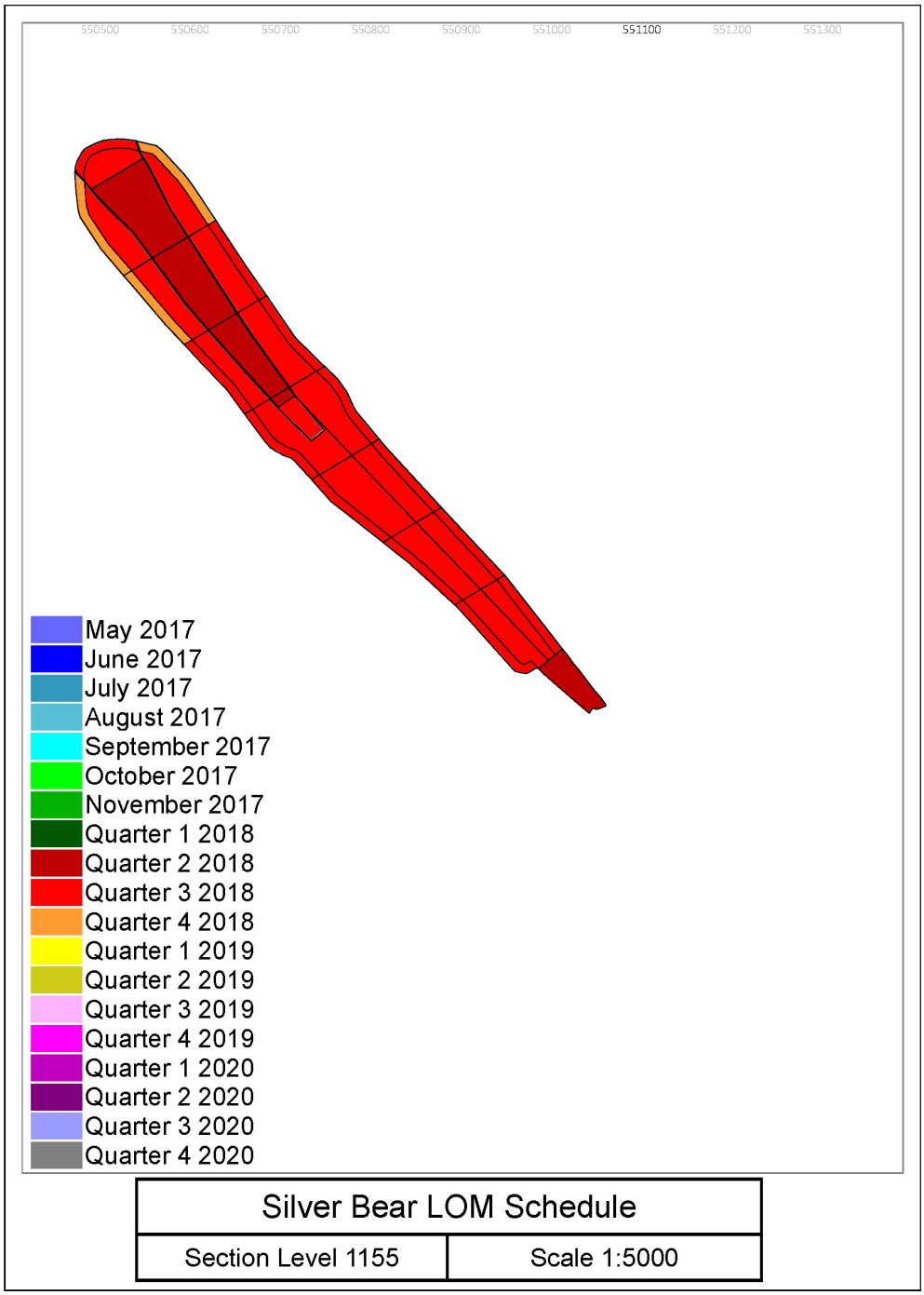


Figure 16.18 Mine Schedule Section Level 1150

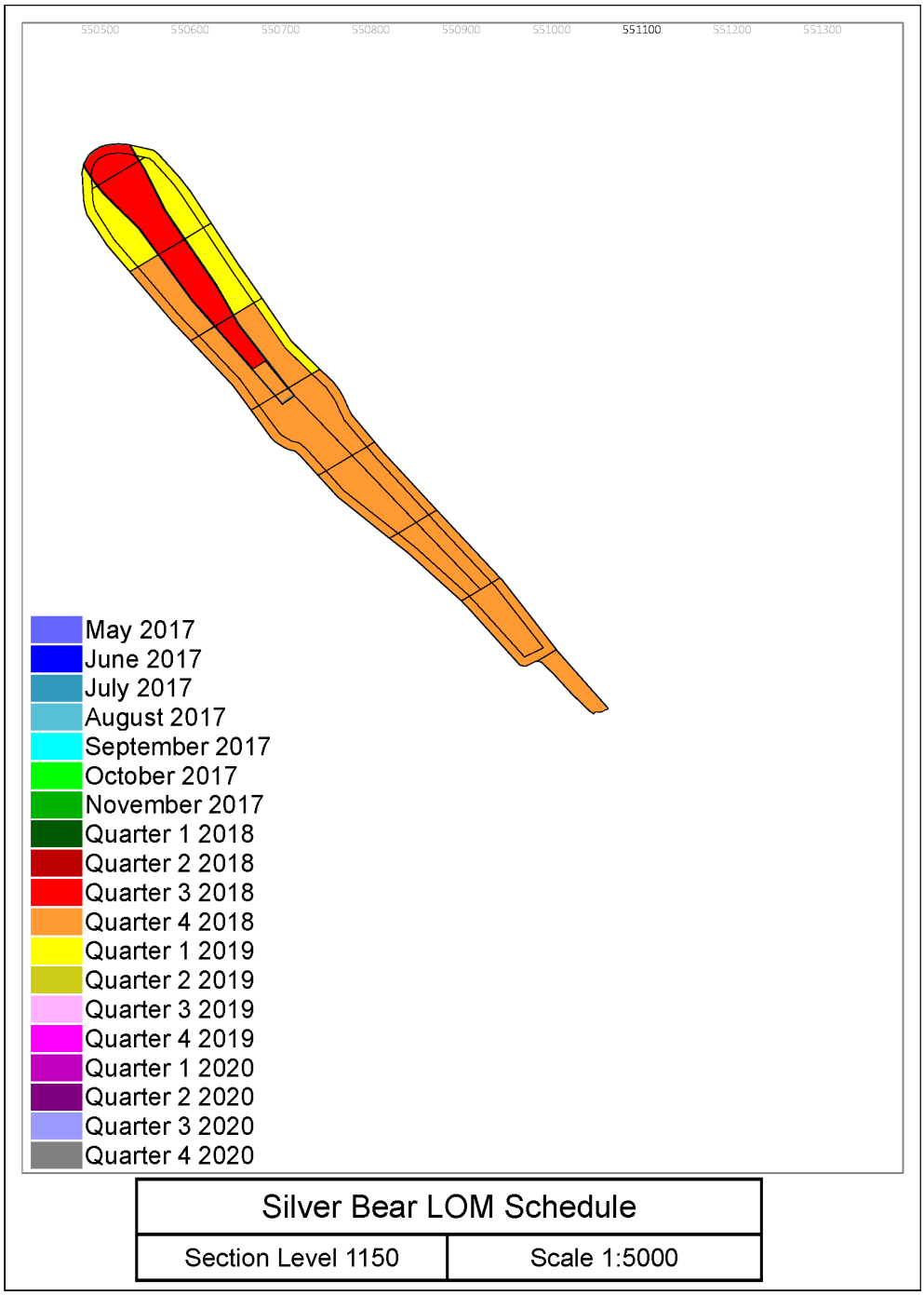


Figure 16.19 Mine Schedule Section Level 1145

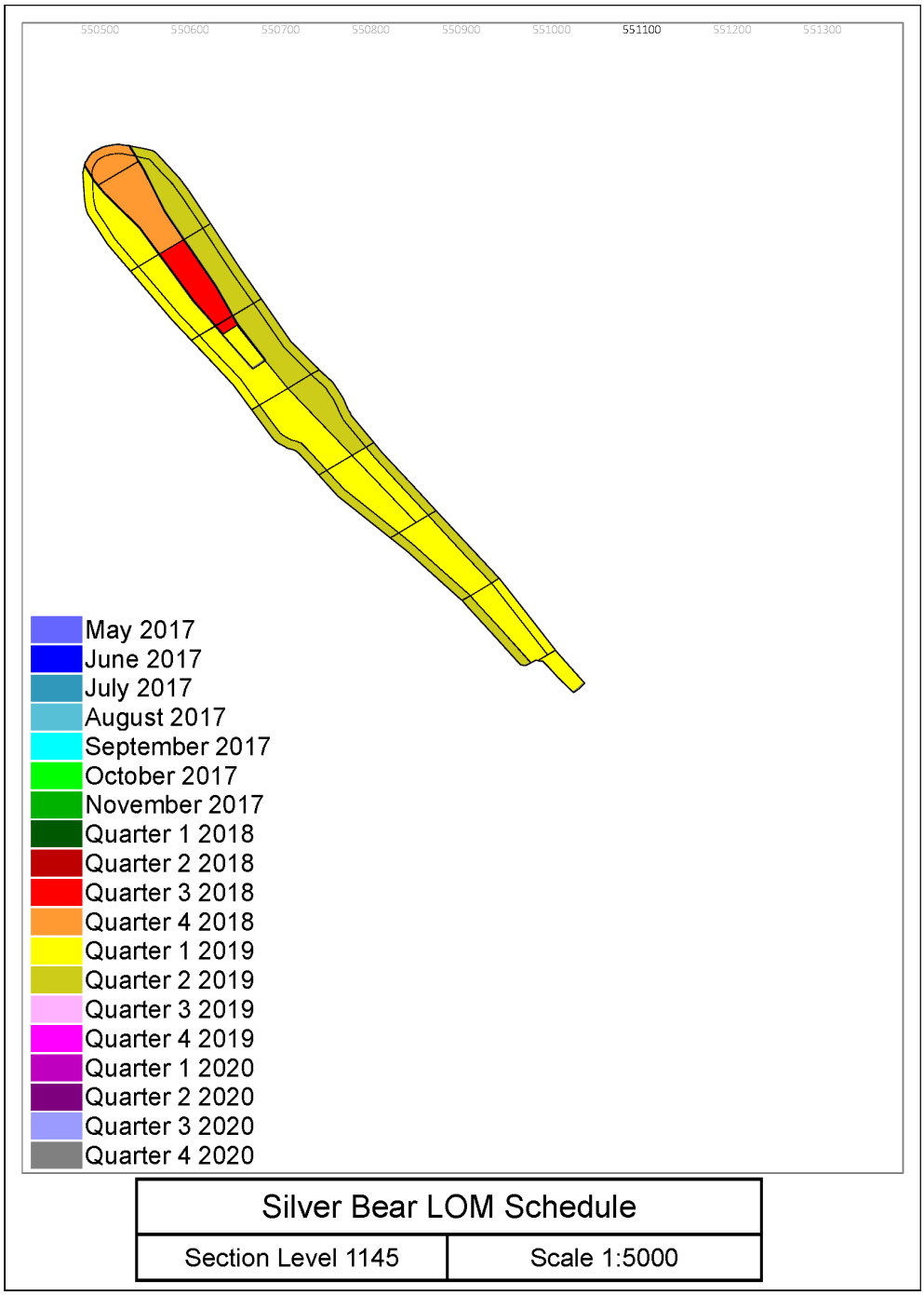


Figure 16.20 Mine Schedule Section Level 1140

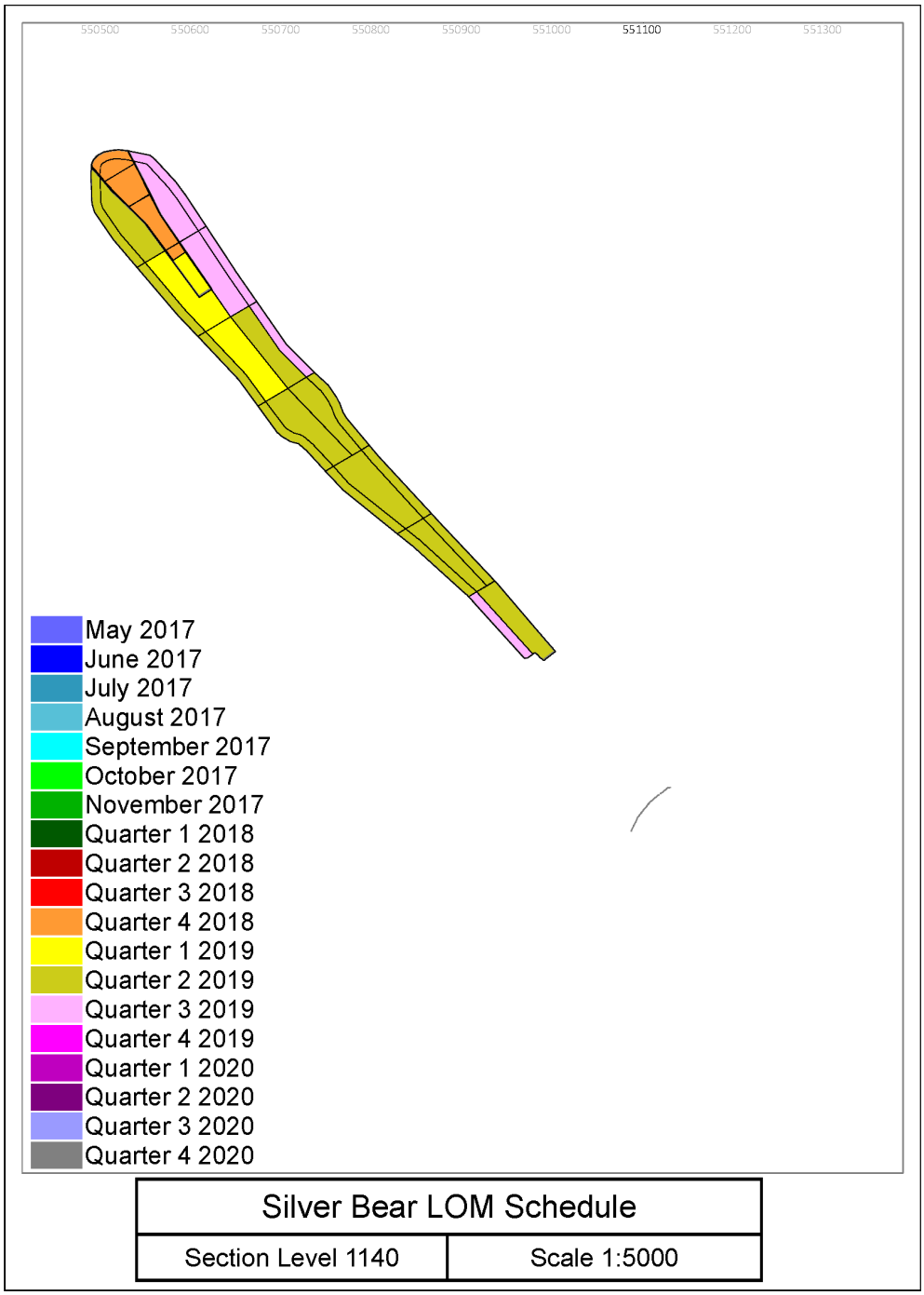


Figure 16.21 Mine Schedule Section Level 1135

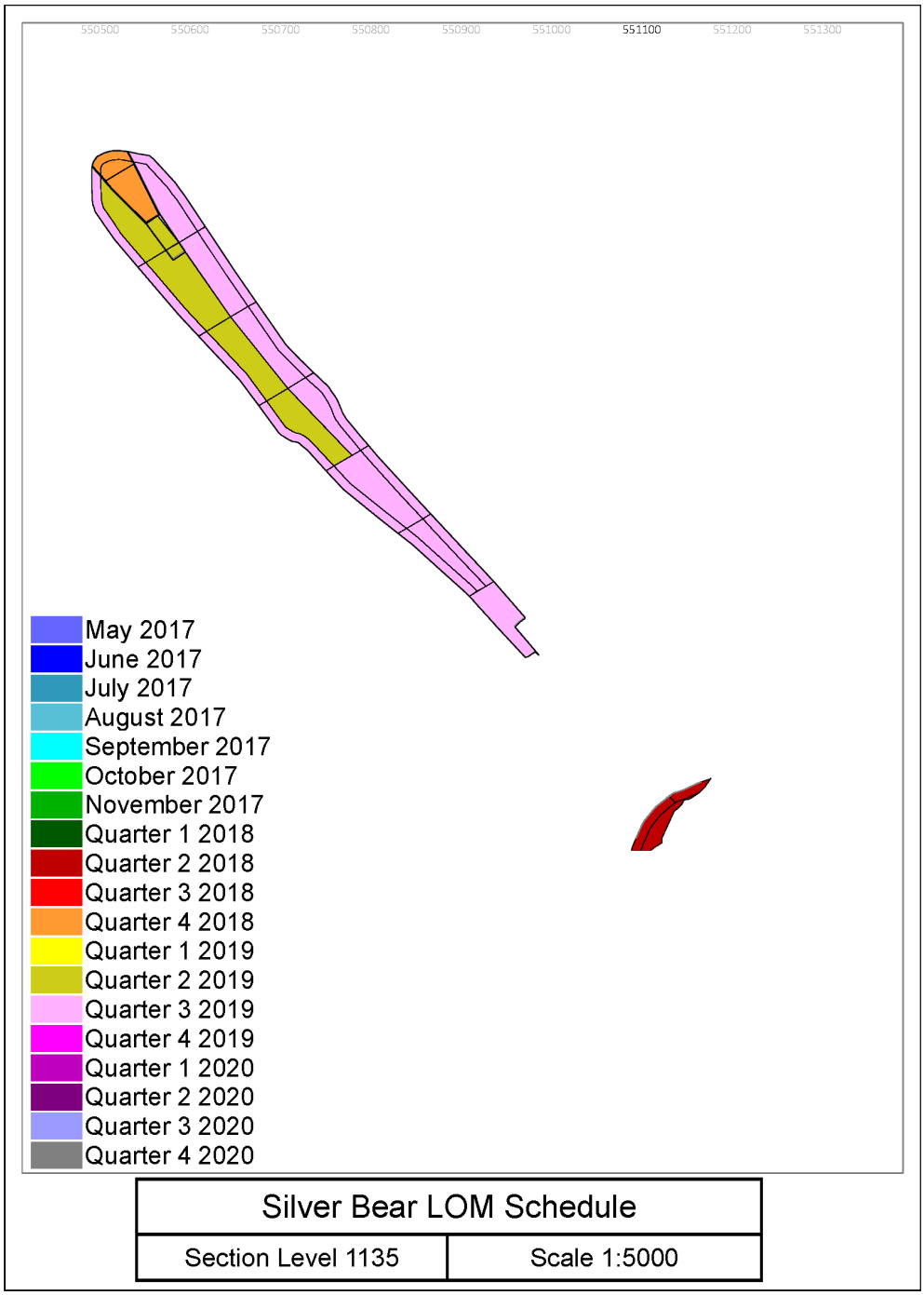


Figure 16.22 Mine Schedule Section Level 1130

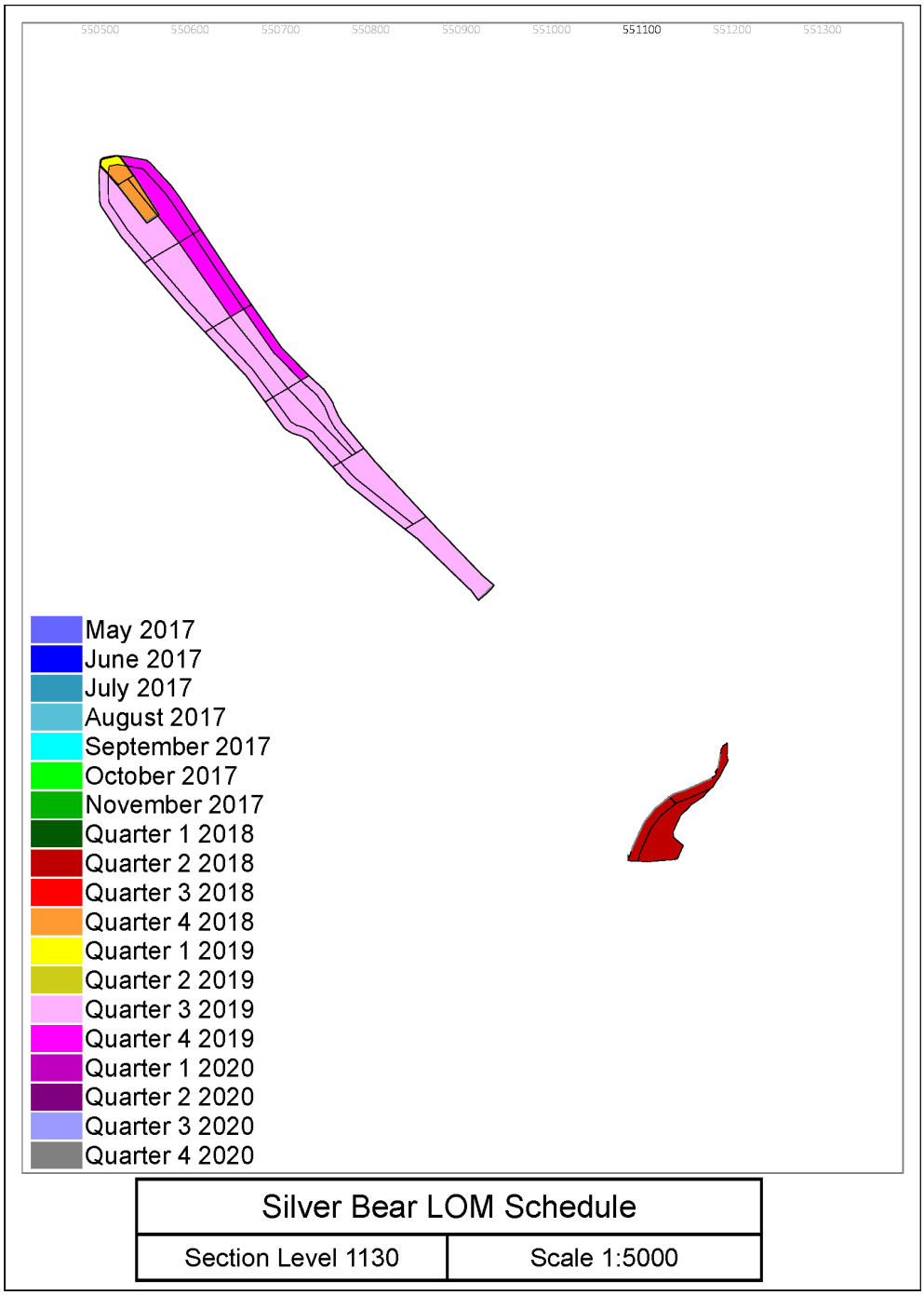


Figure 16.23 Mine Schedule Section Level 1125

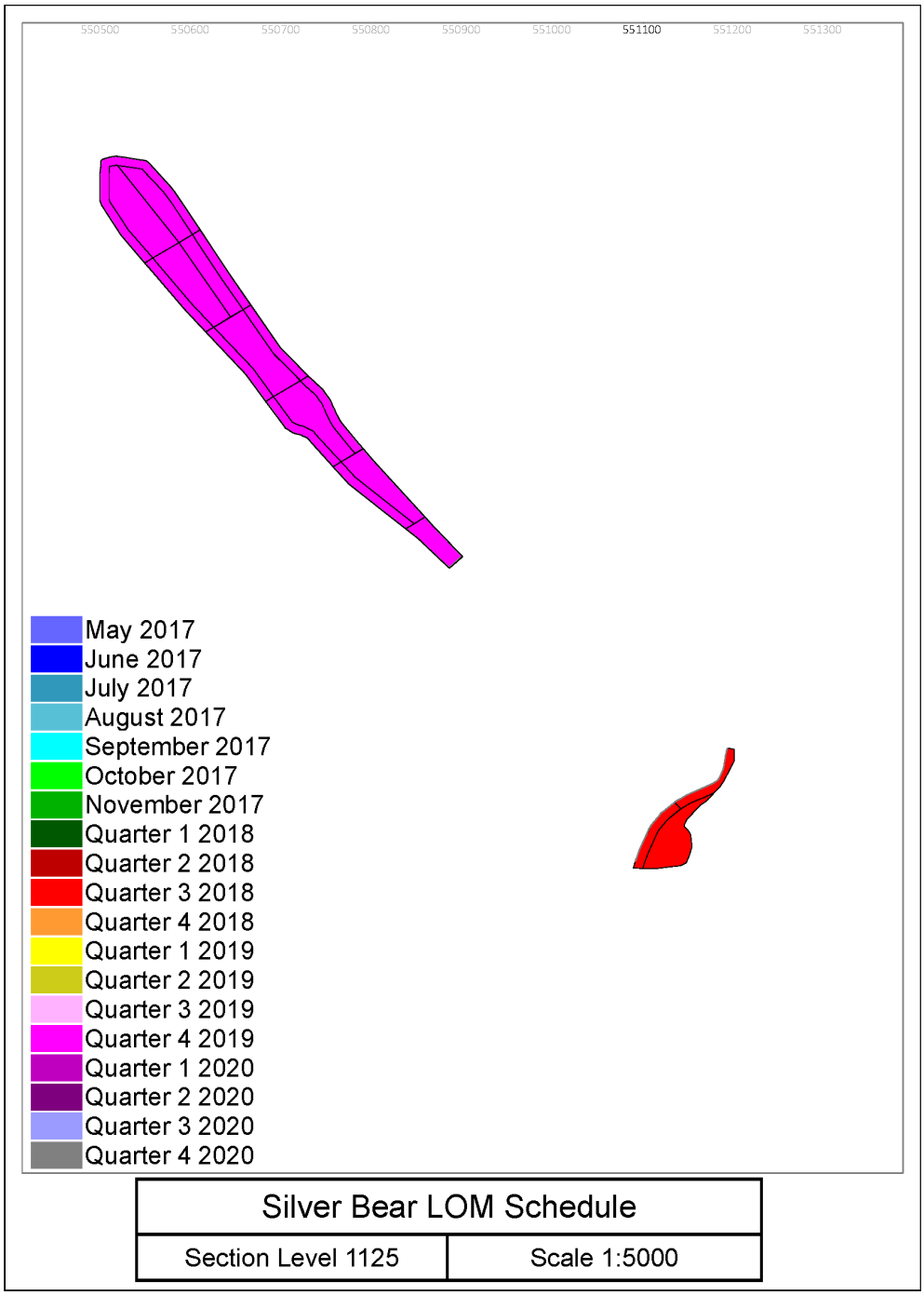


Figure 16.24 Mine Schedule Section Level 1120

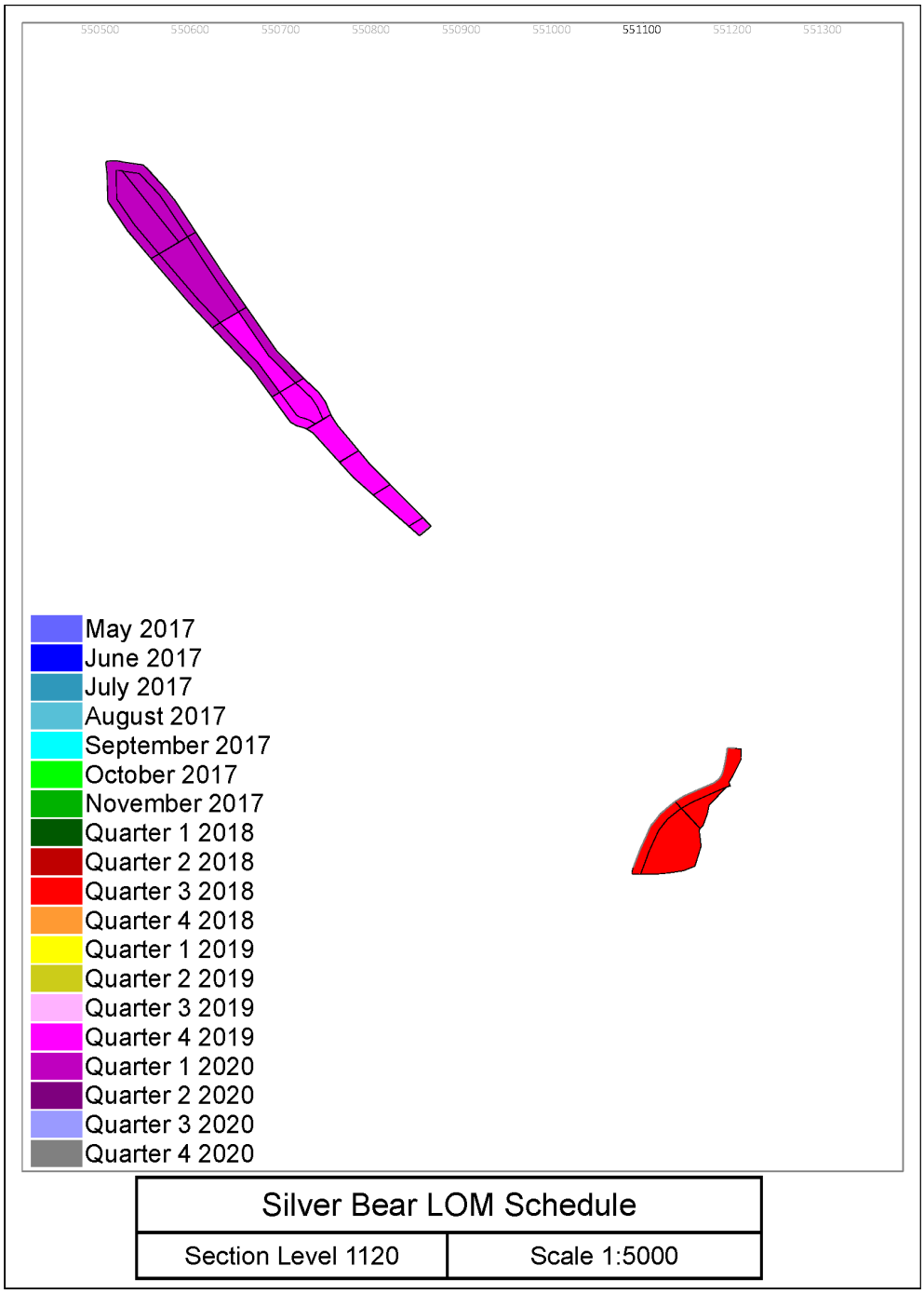


Figure 16.25 Mine Schedule Section Level 1115

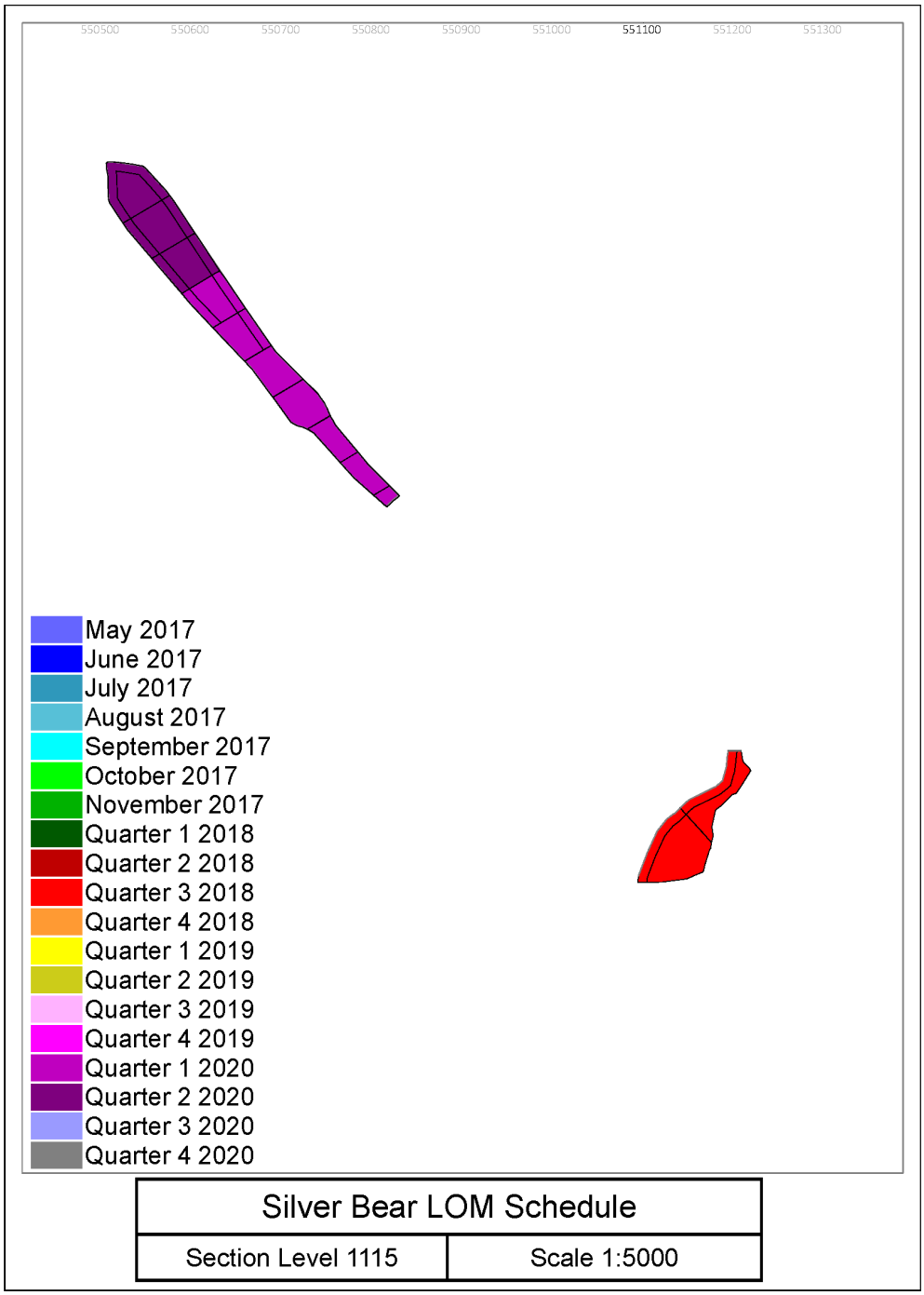


Figure 16.26 Mine Schedule Section Level 1110

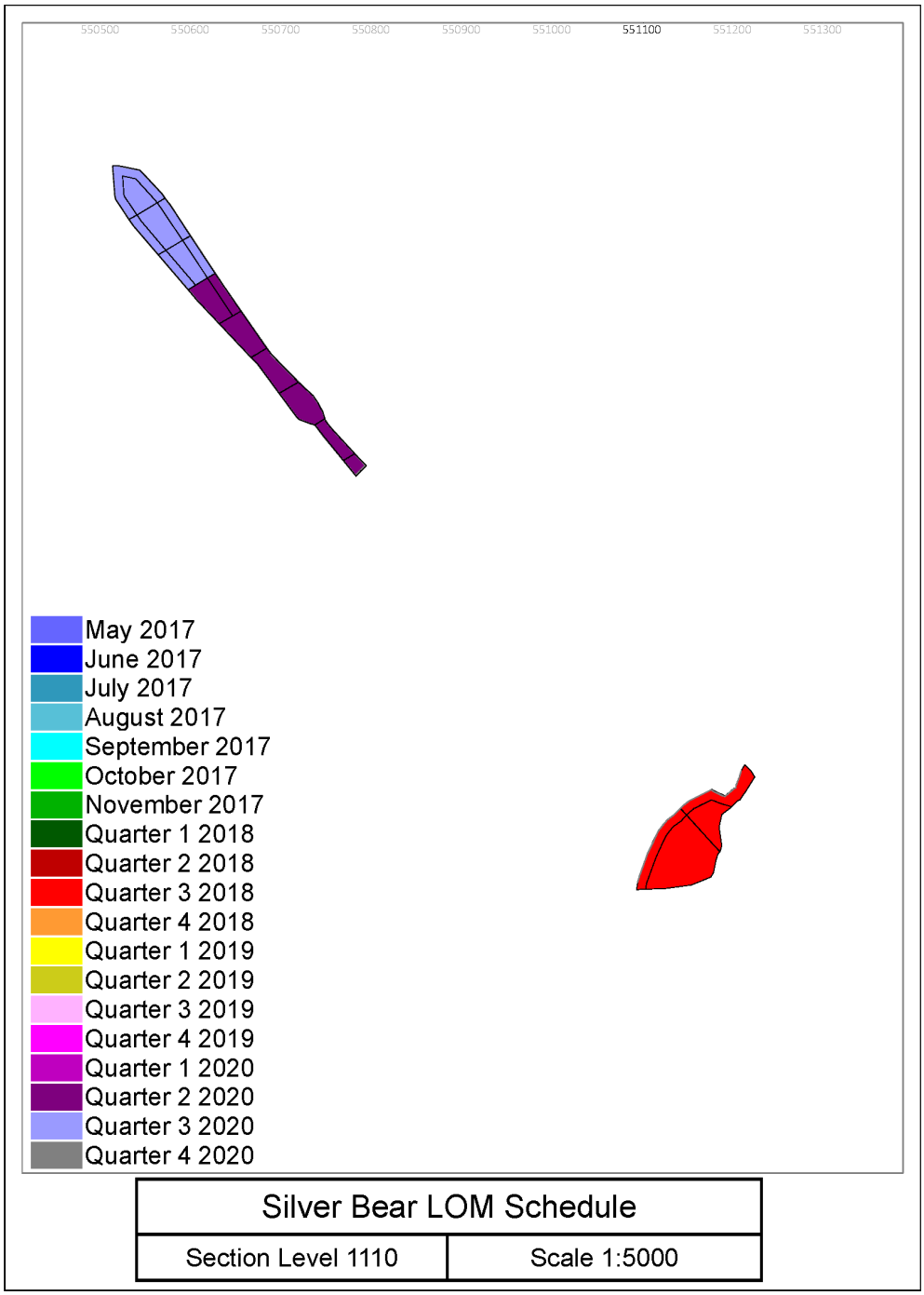


Figure 16.27 Mine Schedule Section Level 1105

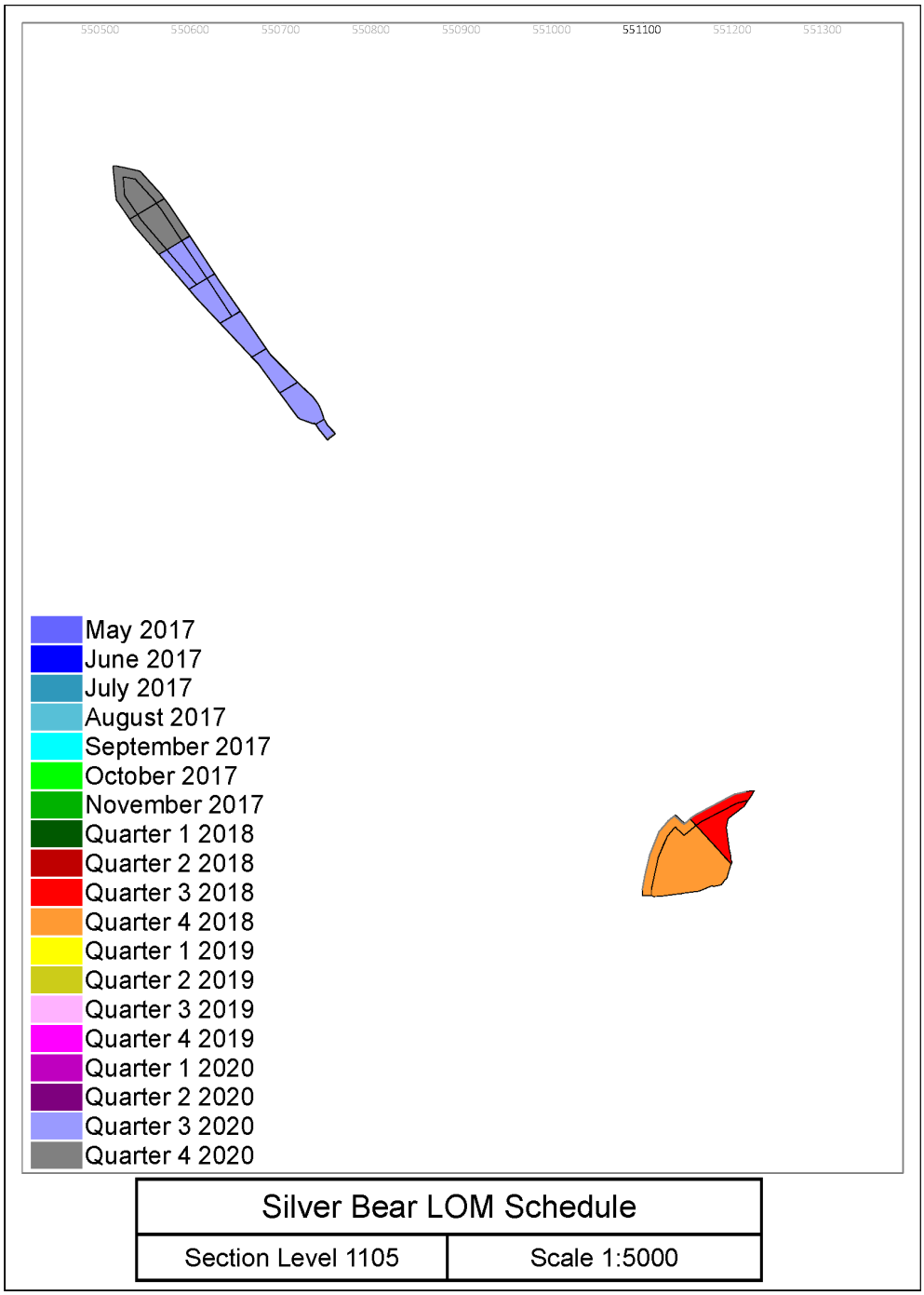


Figure 16.28 Mine Schedule Section Level 1100

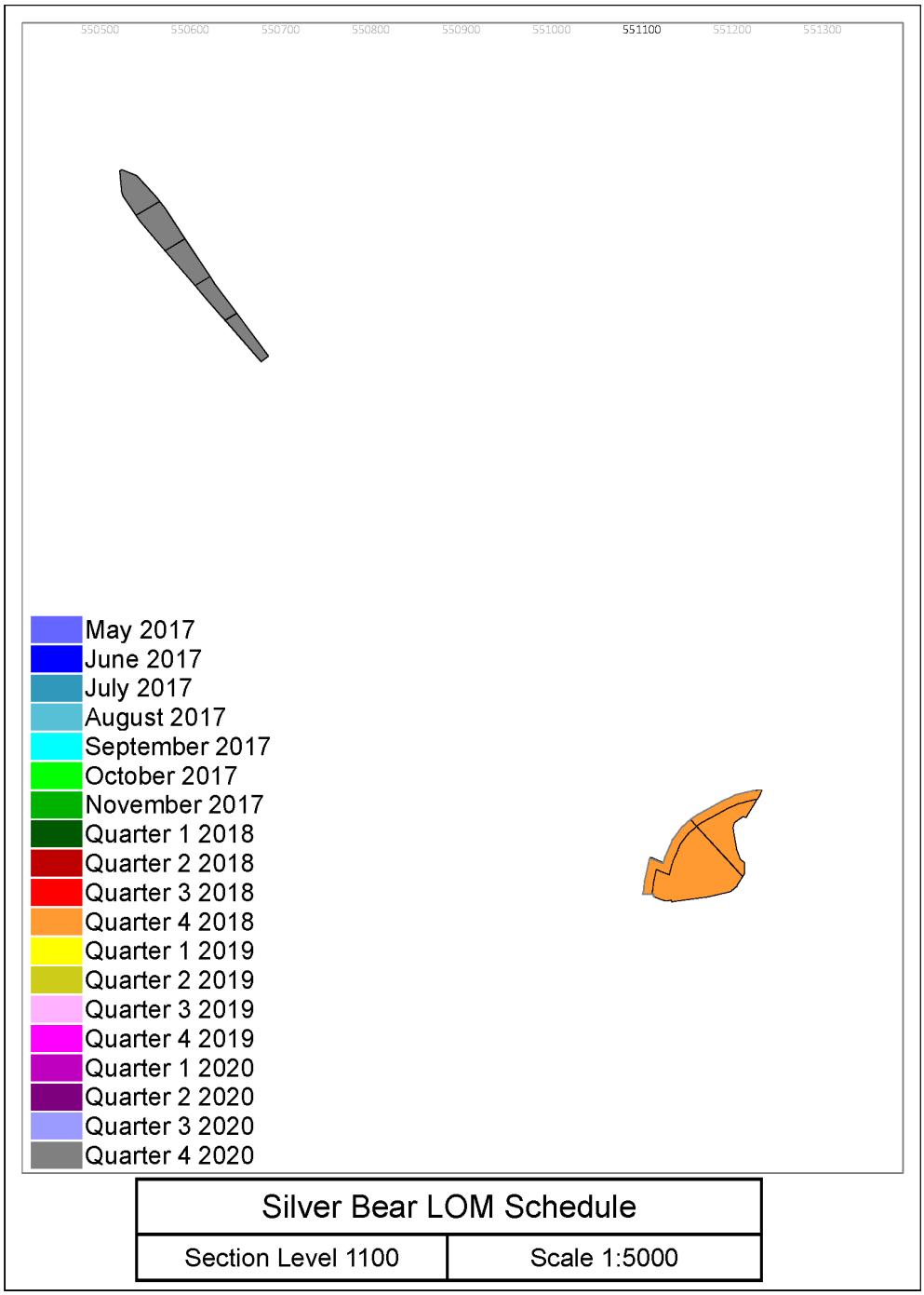


Figure 16.29 Mine Schedule Section Level 1095

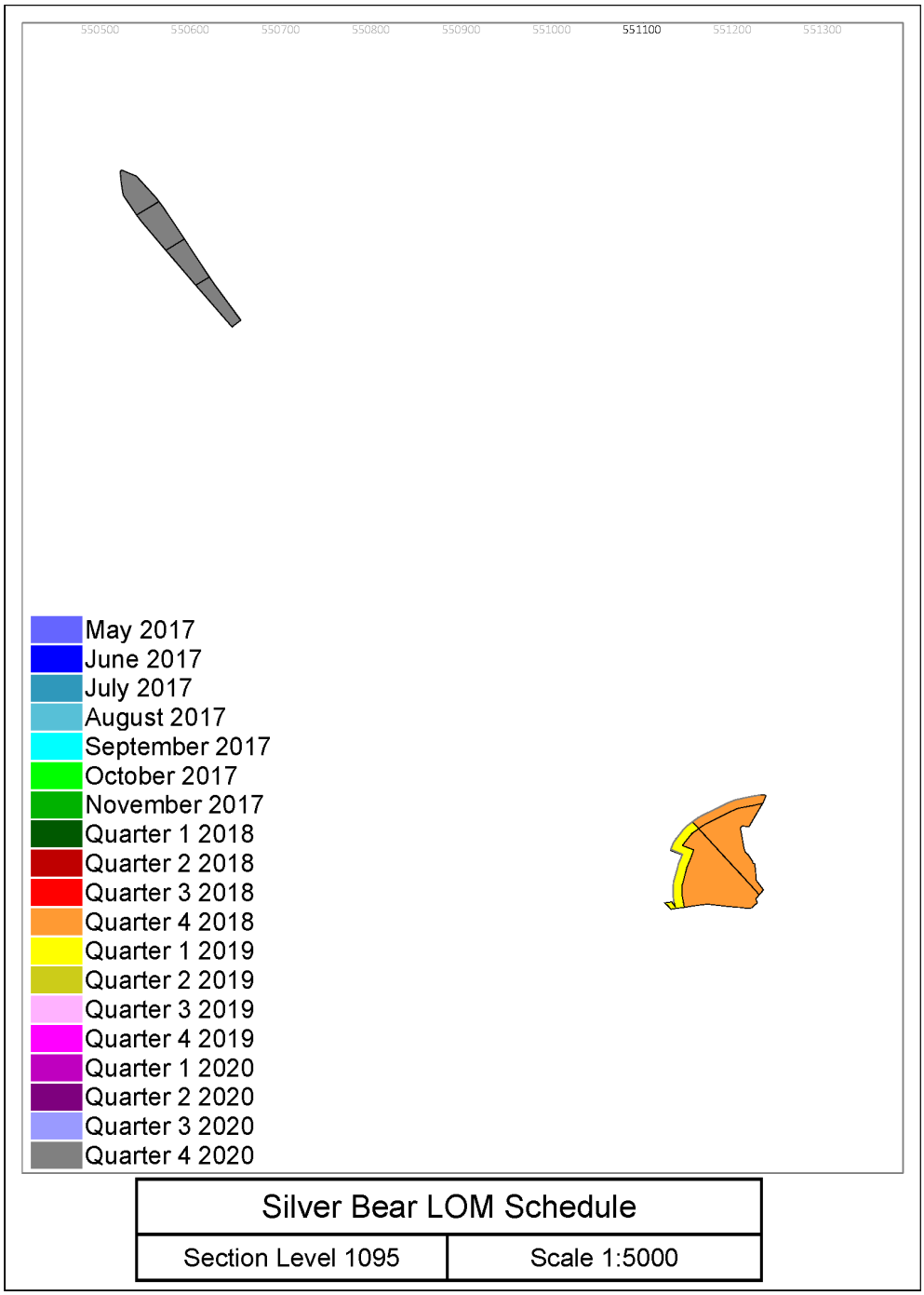


Figure 16.30 Mine Schedule Section Level 1090

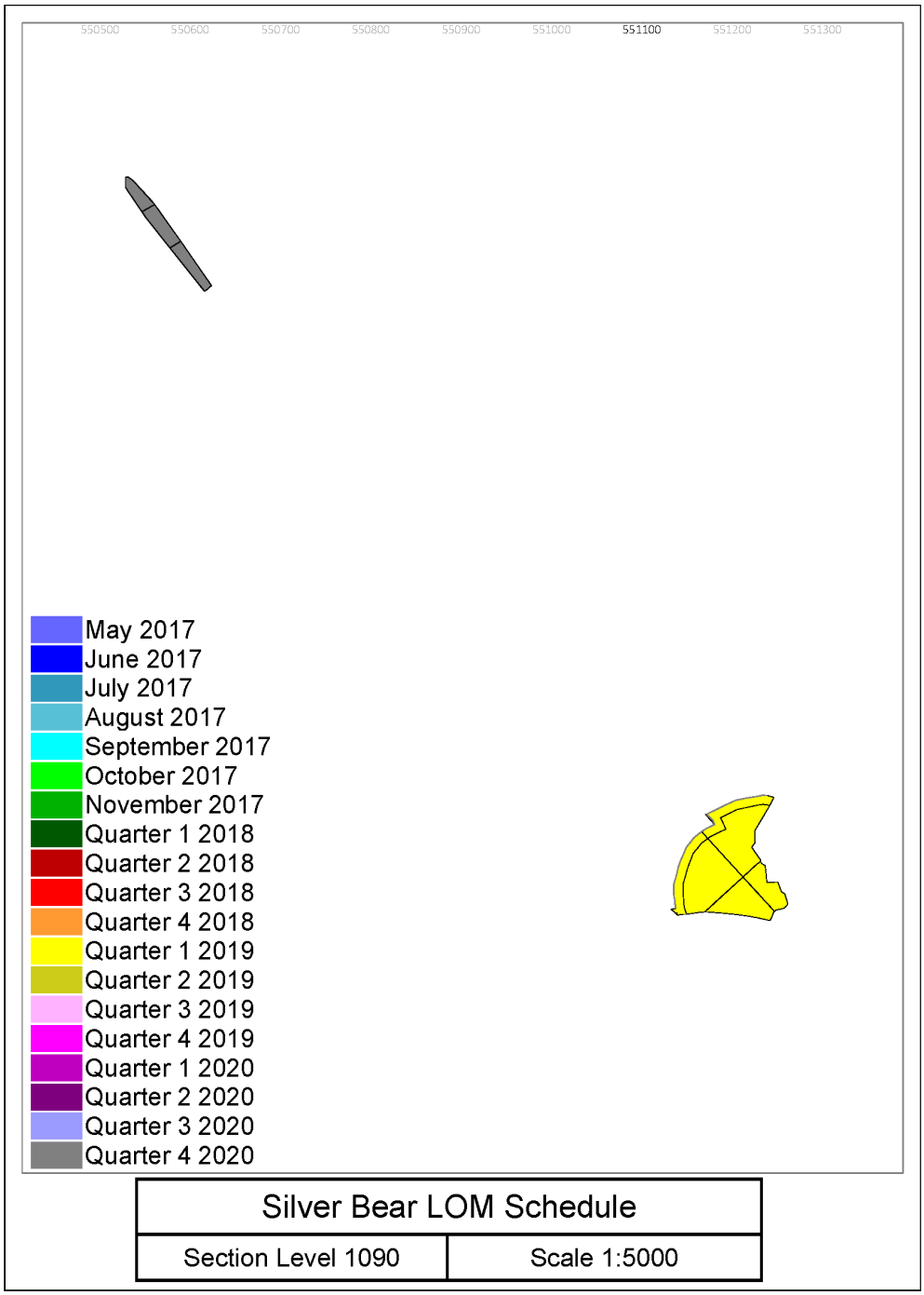


Figure 16.31 Mine Schedule Section Level 1085

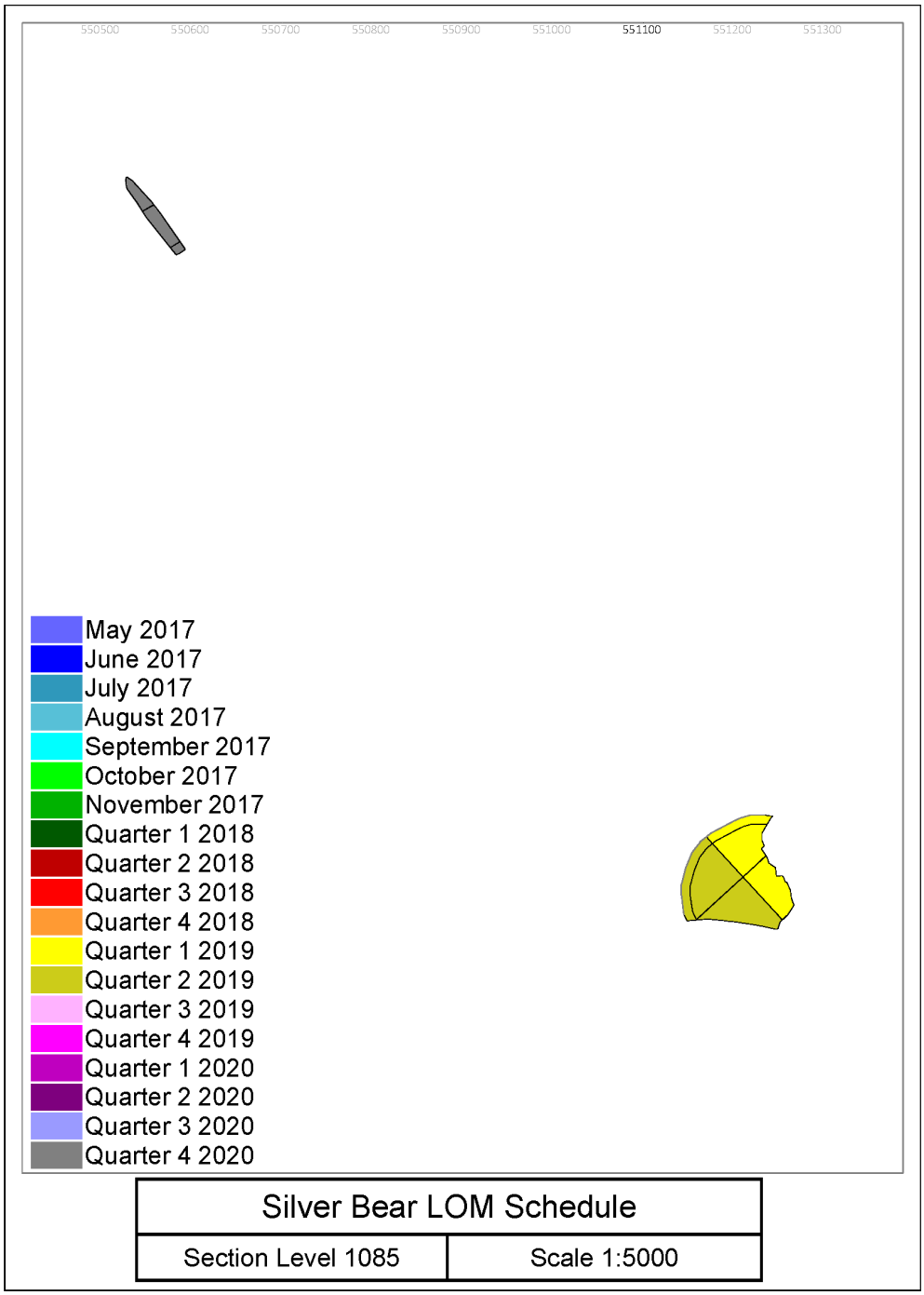
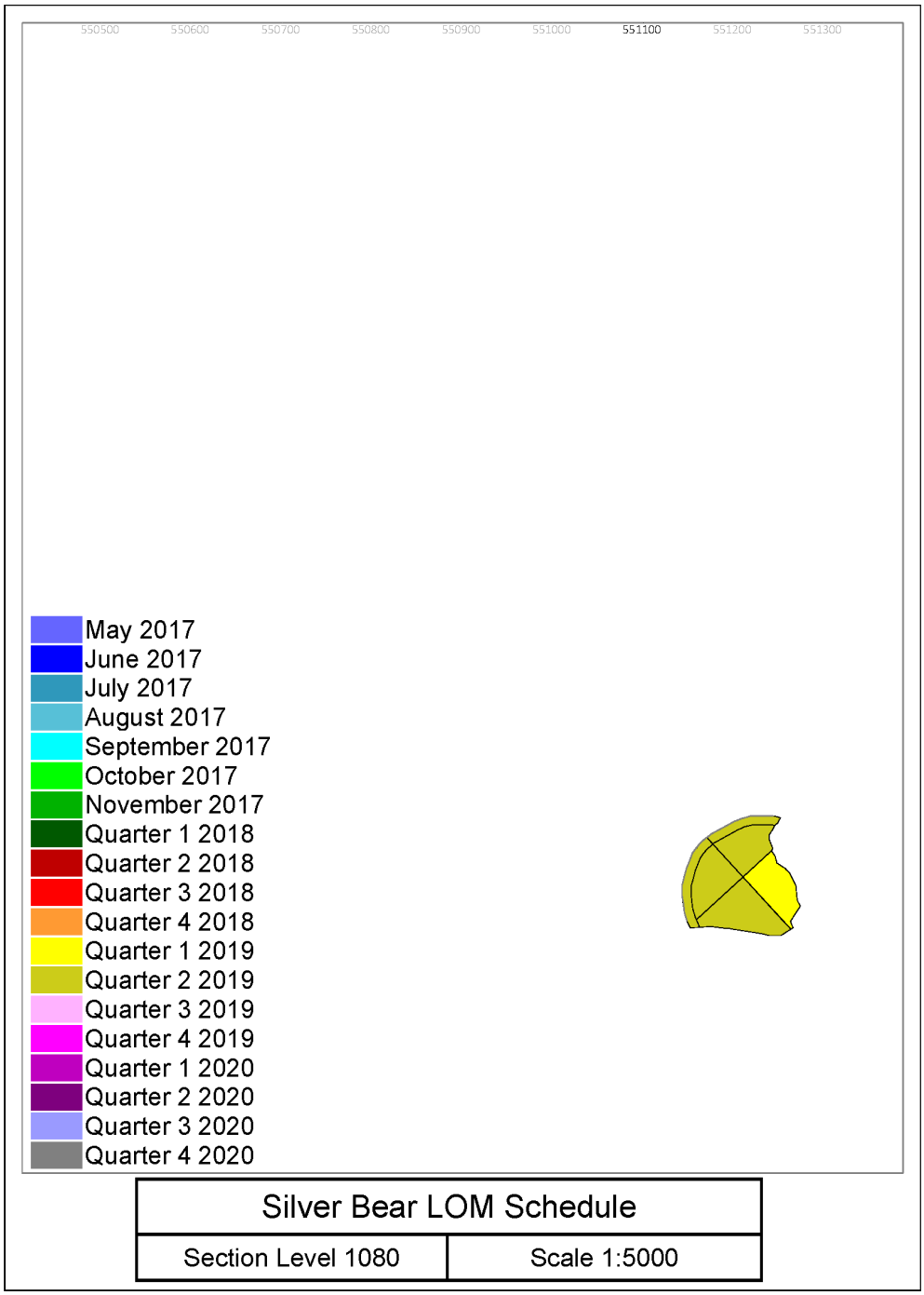


Figure 16.32 Mine Schedule Section Level 1080



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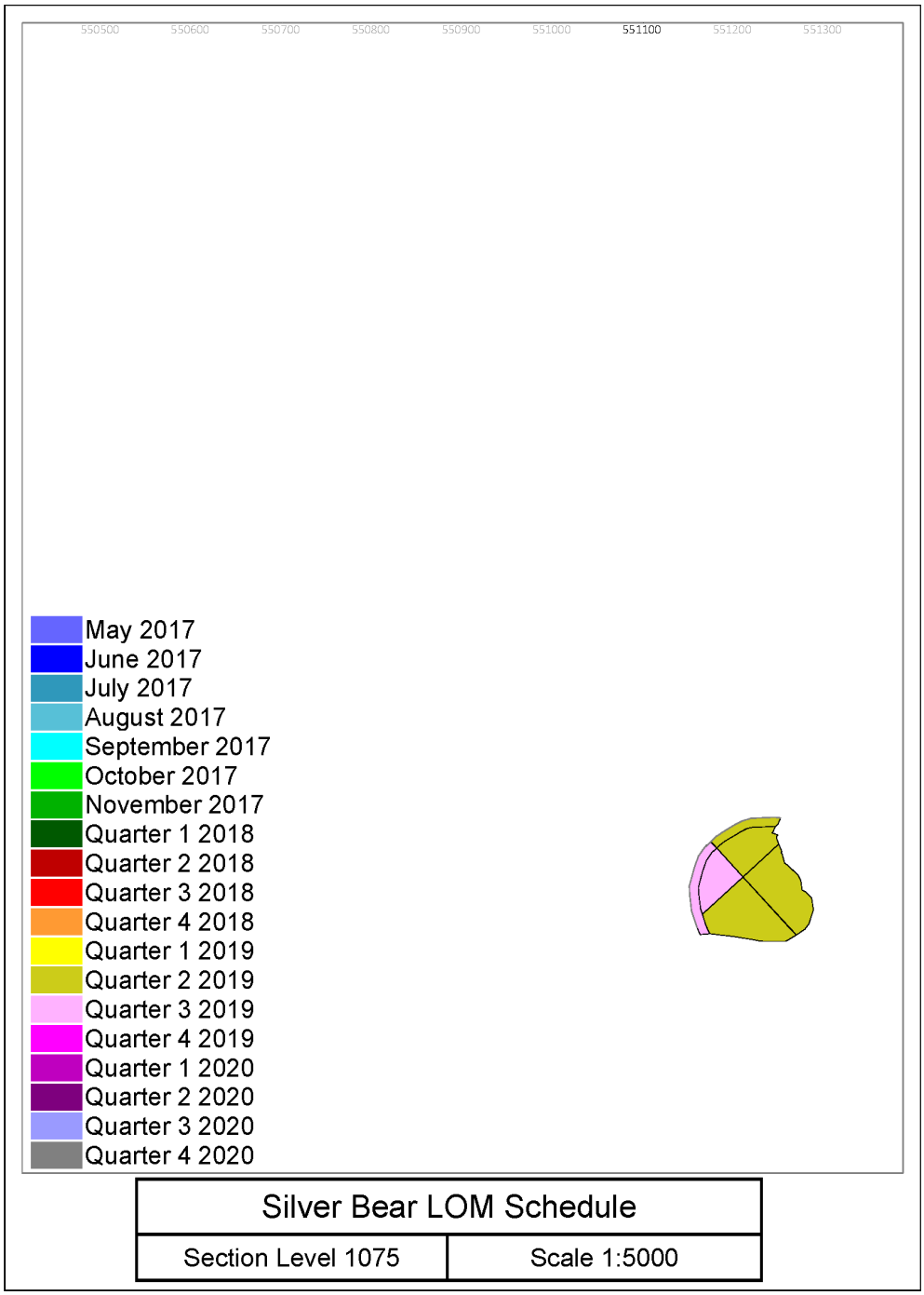

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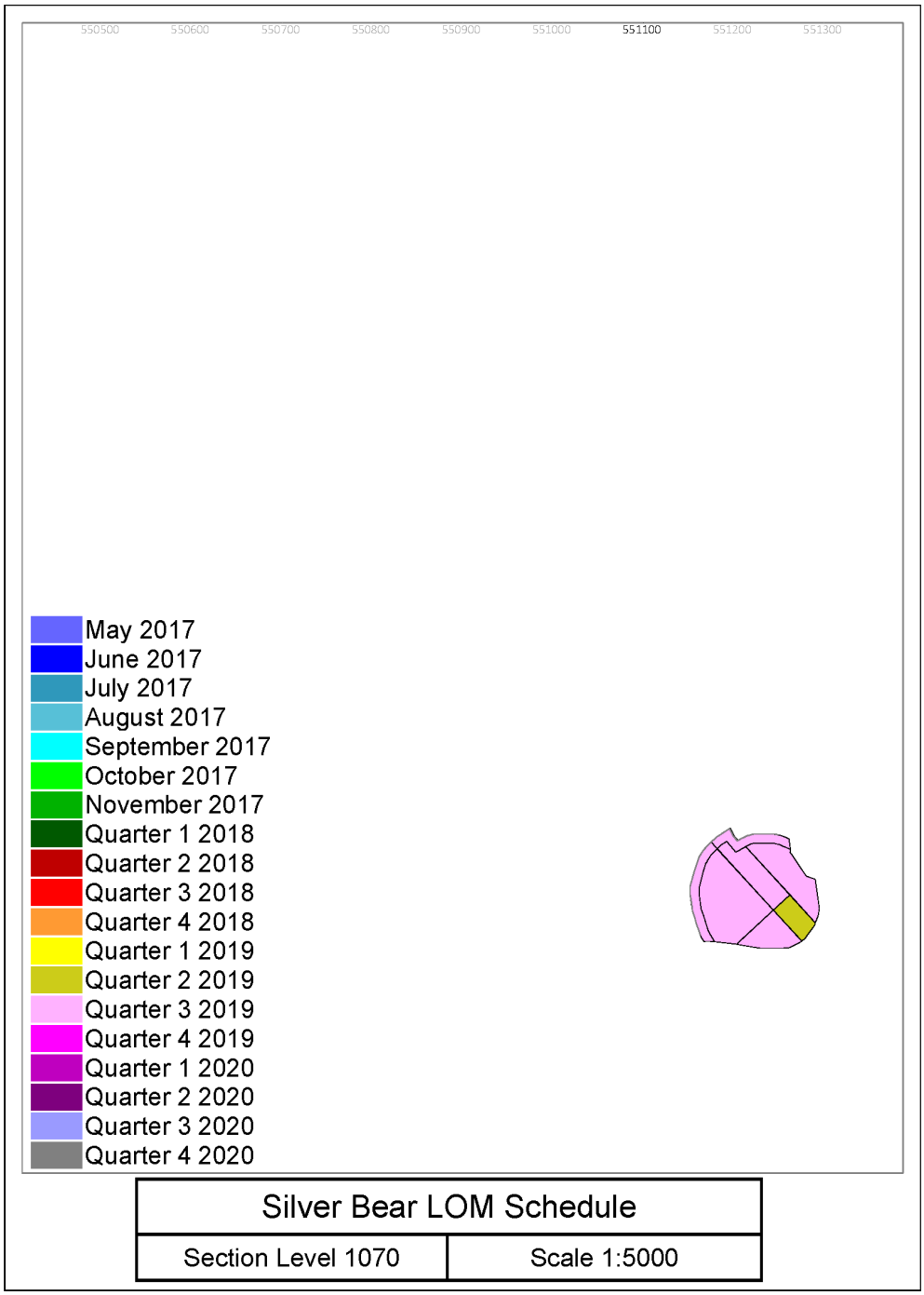

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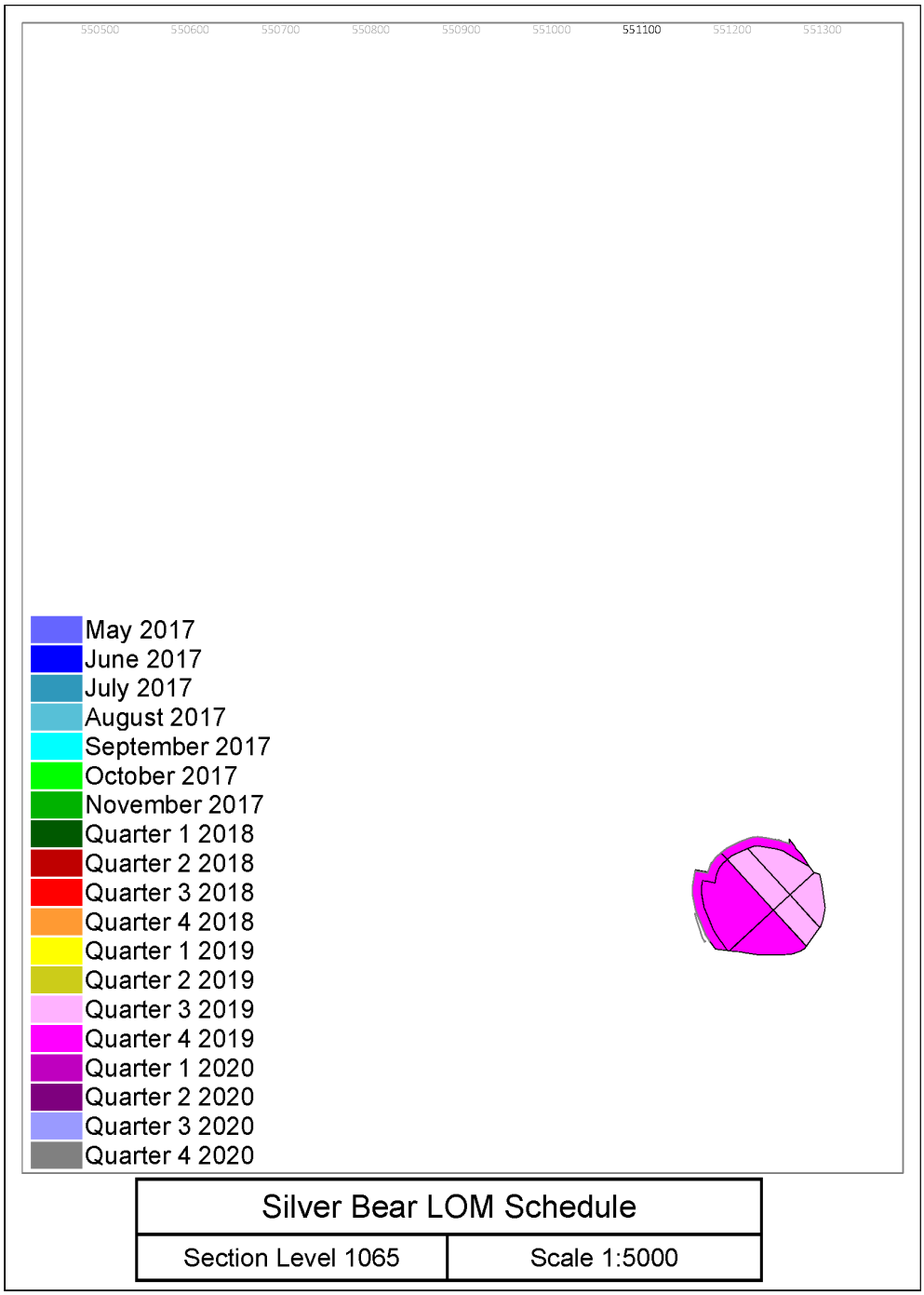

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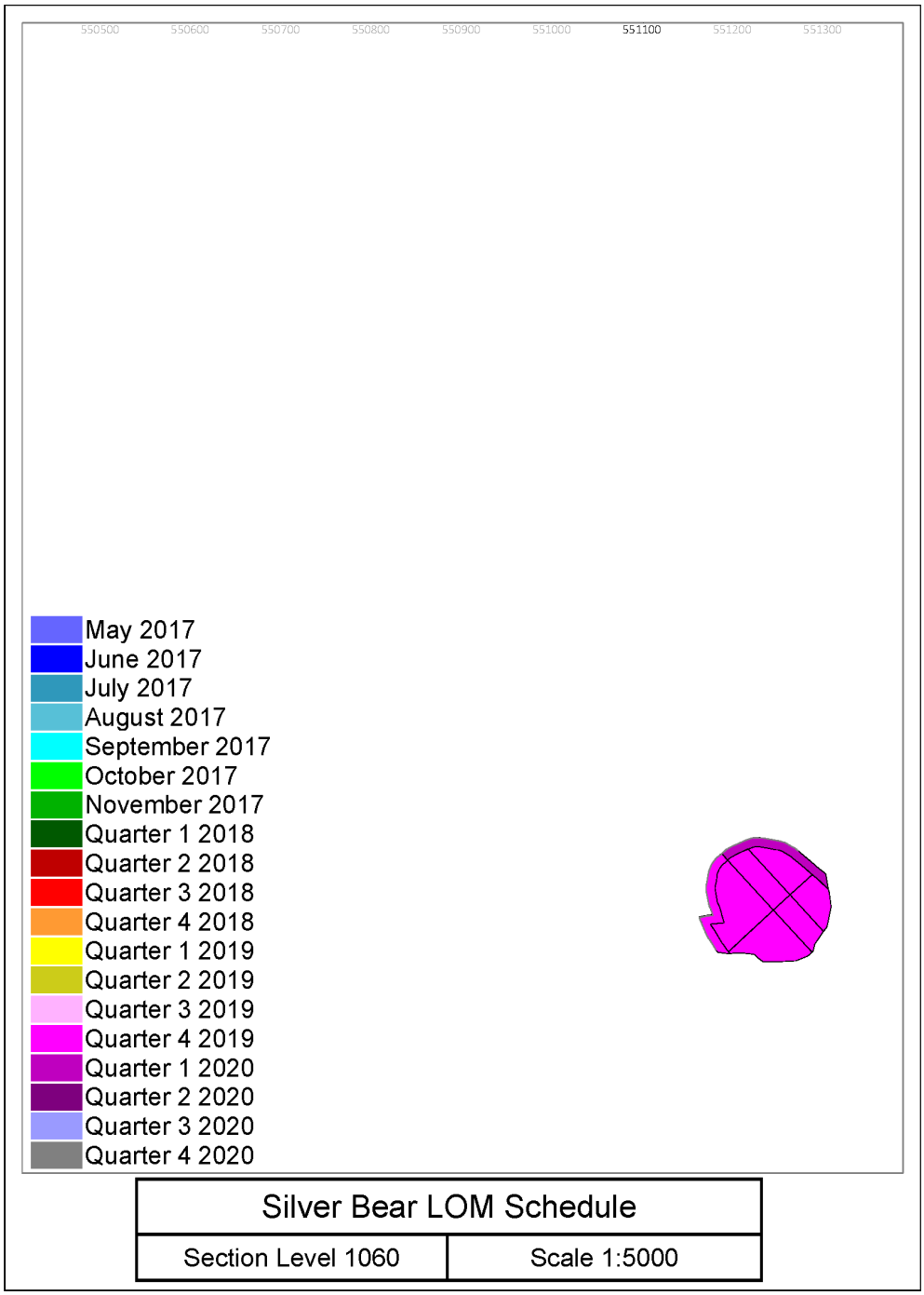

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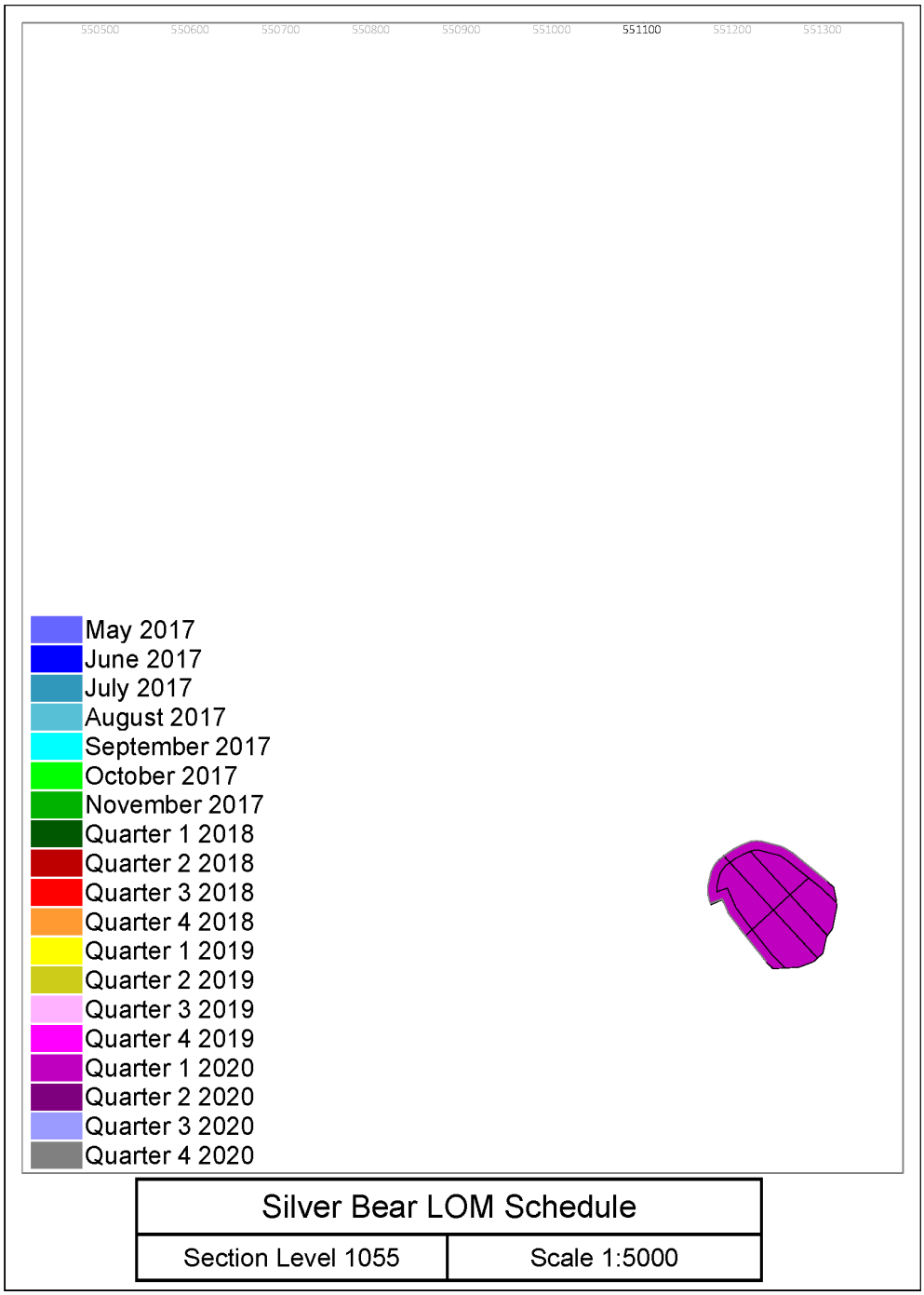

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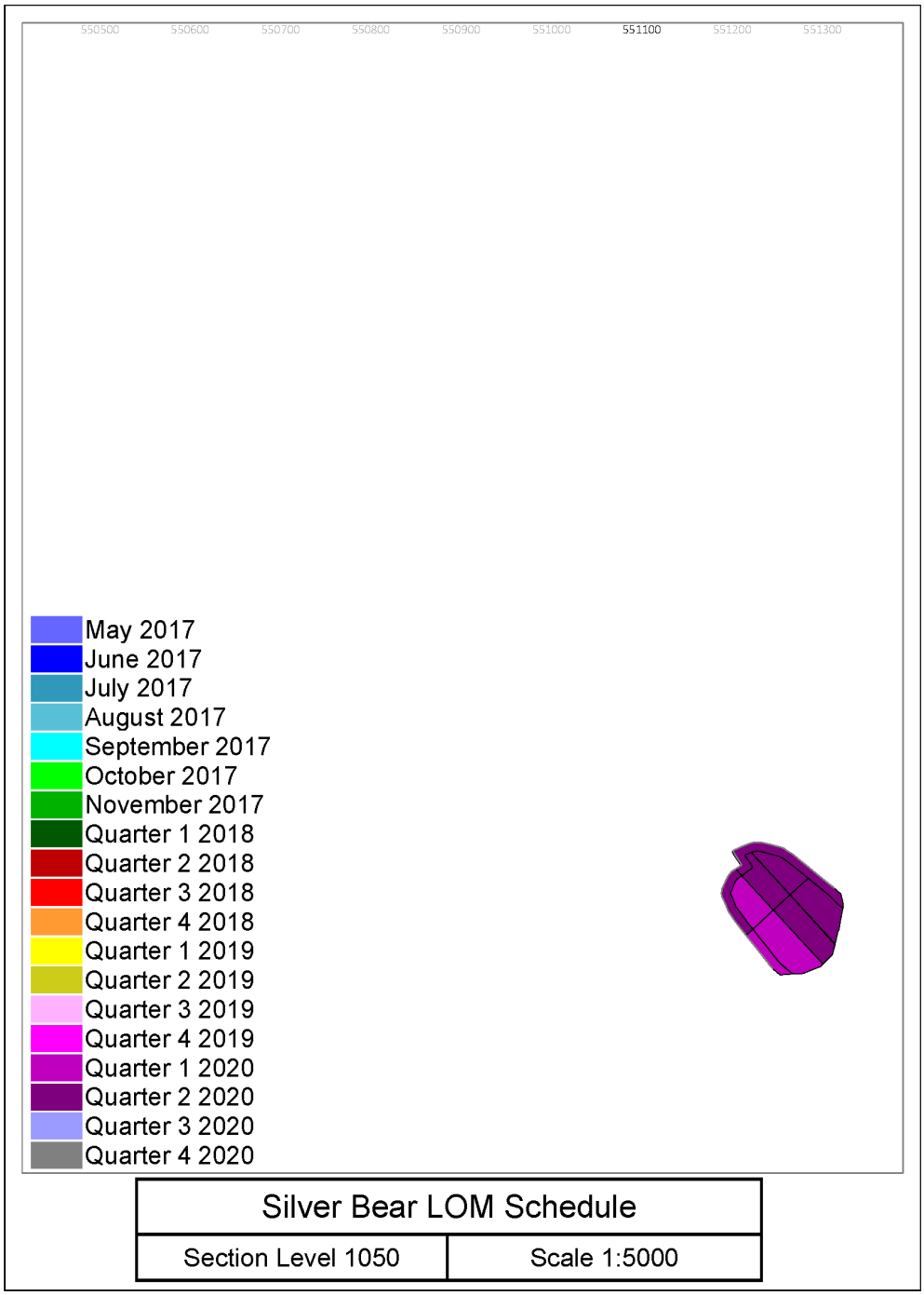

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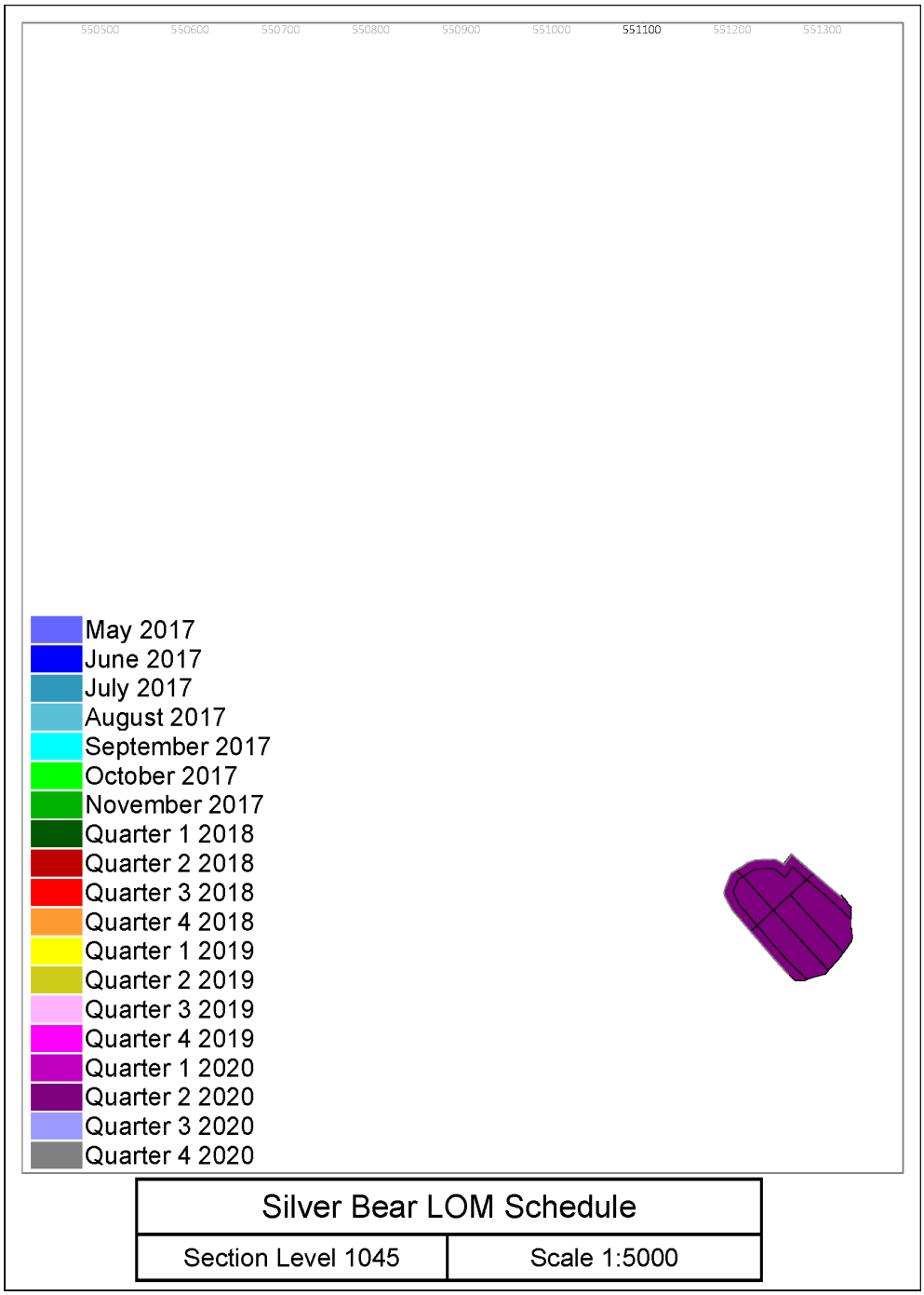

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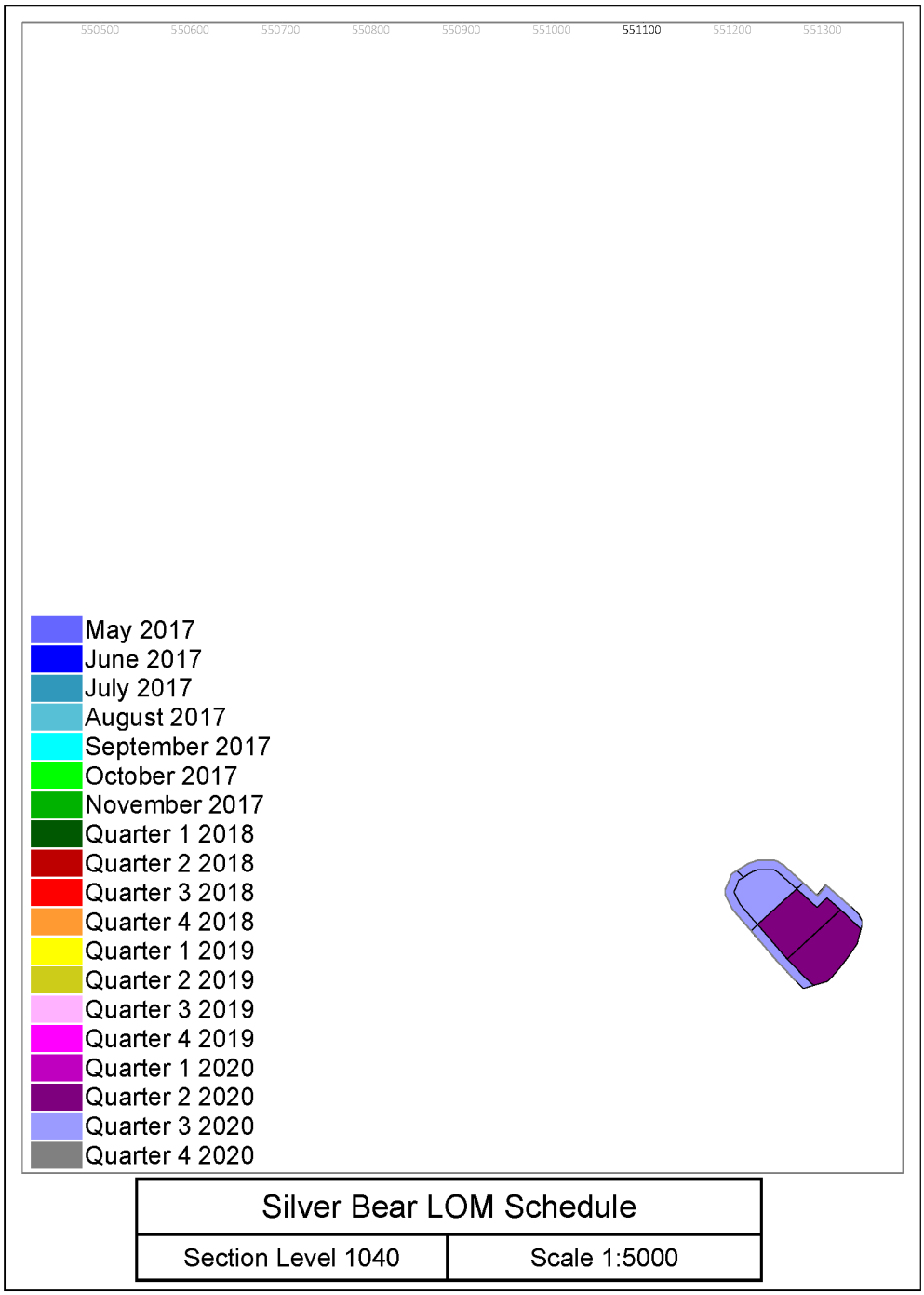

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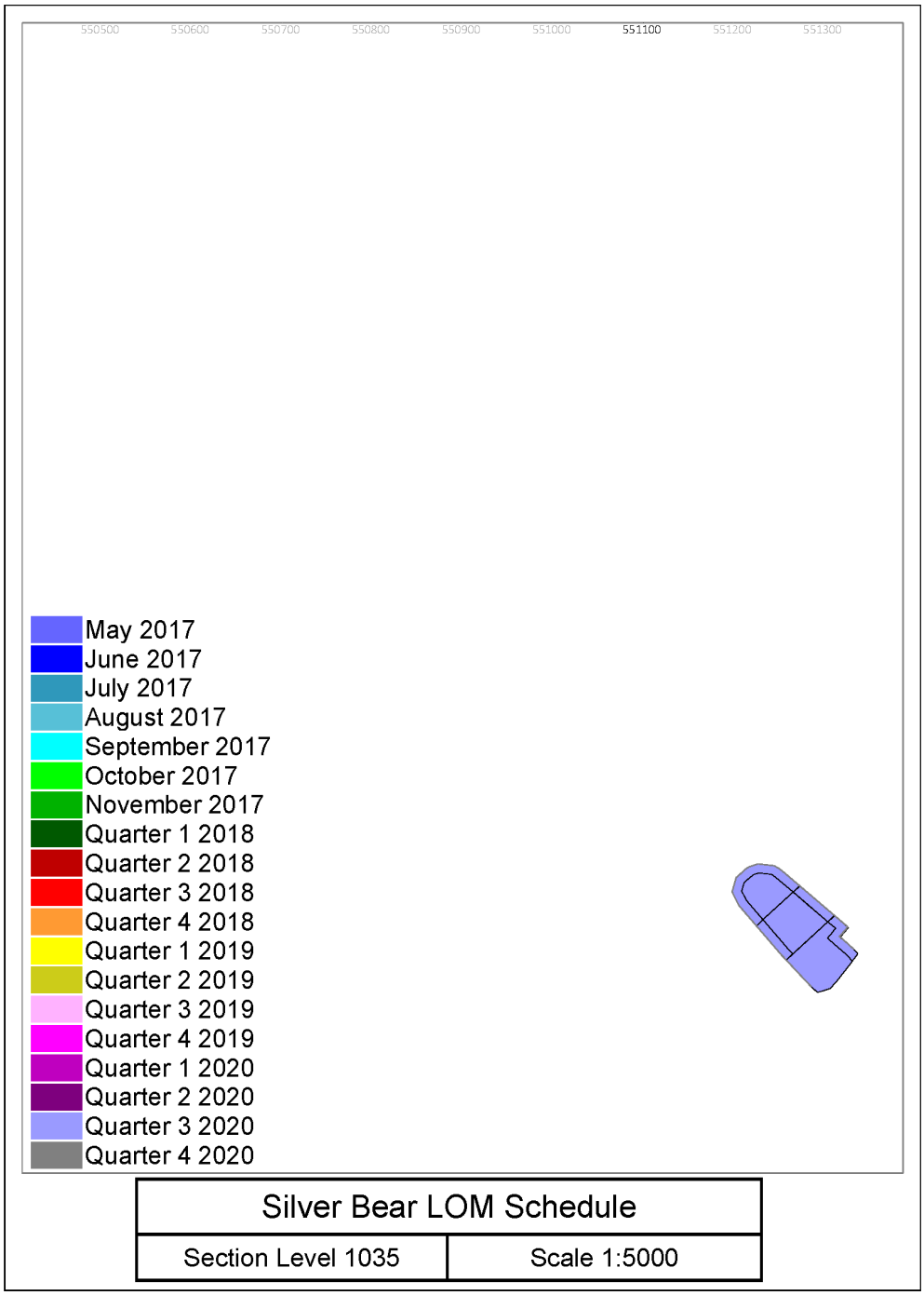

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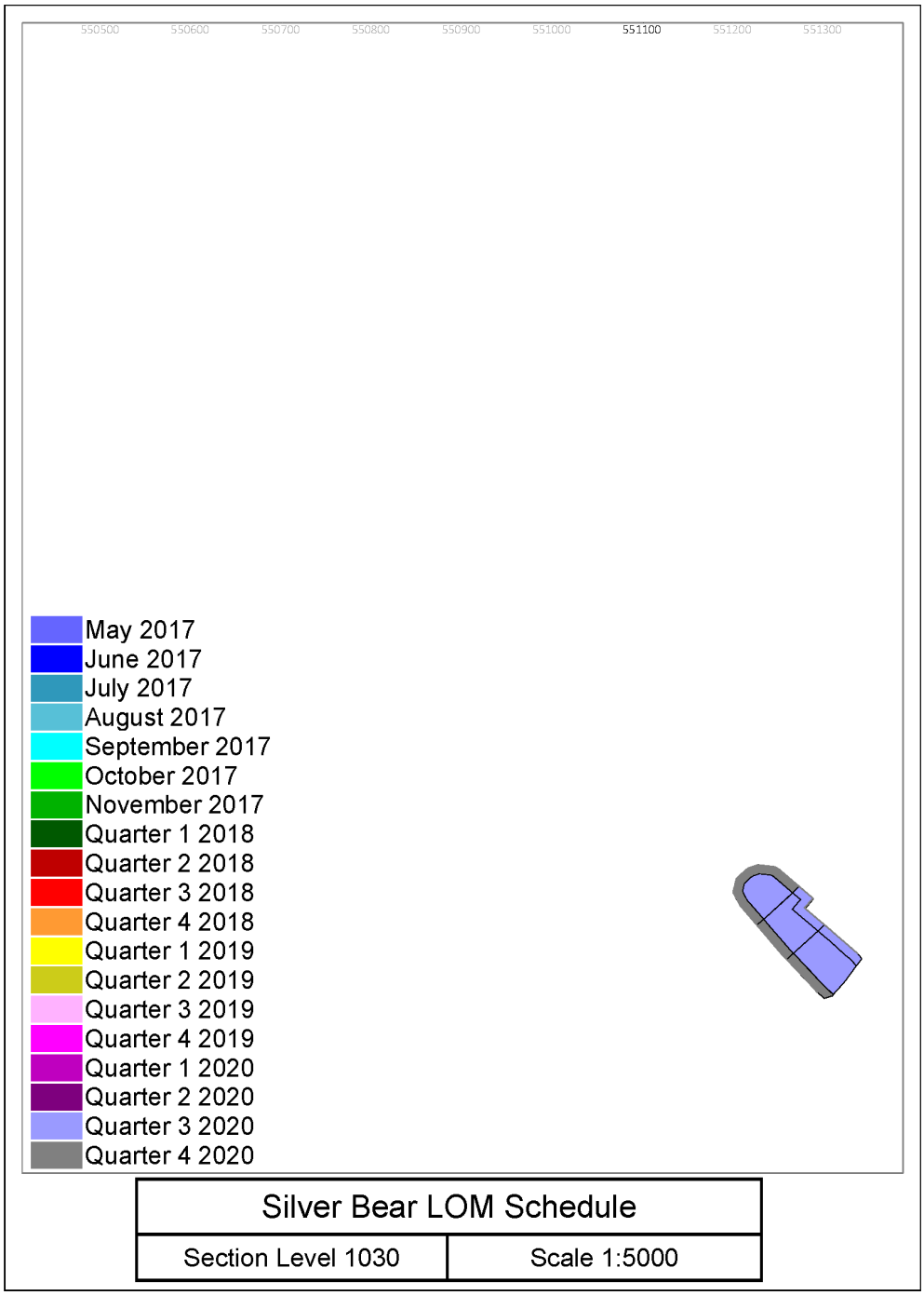

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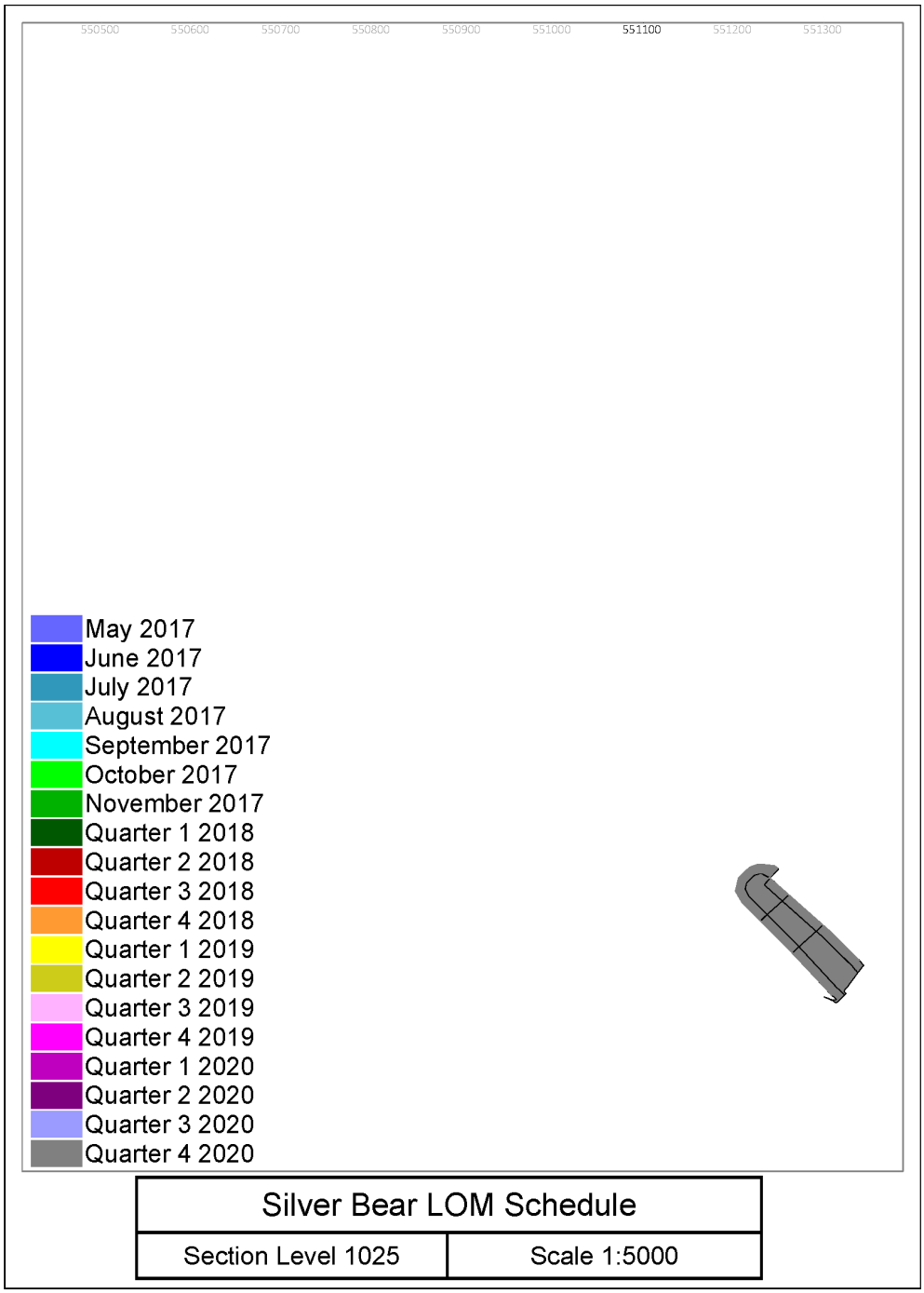

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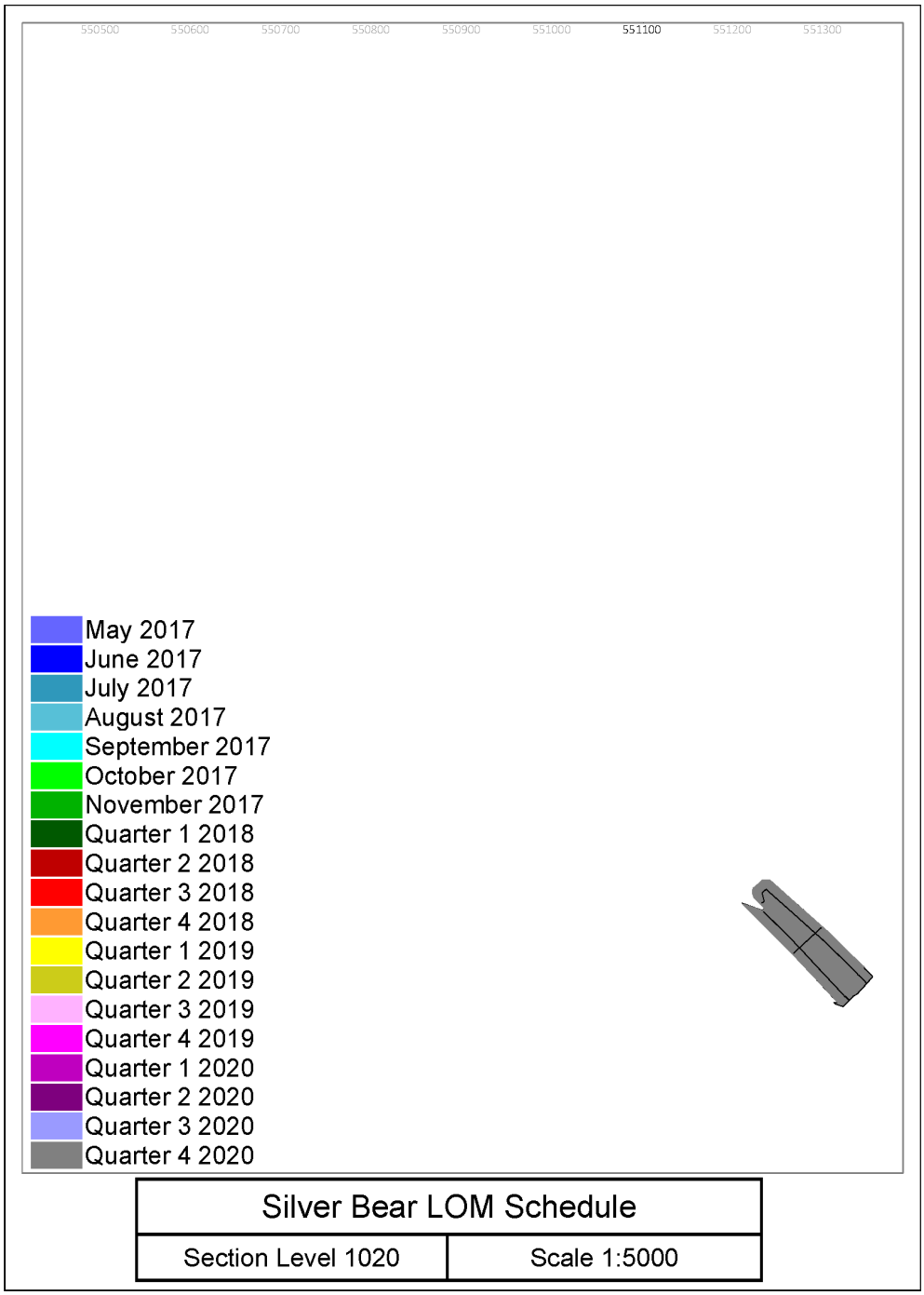

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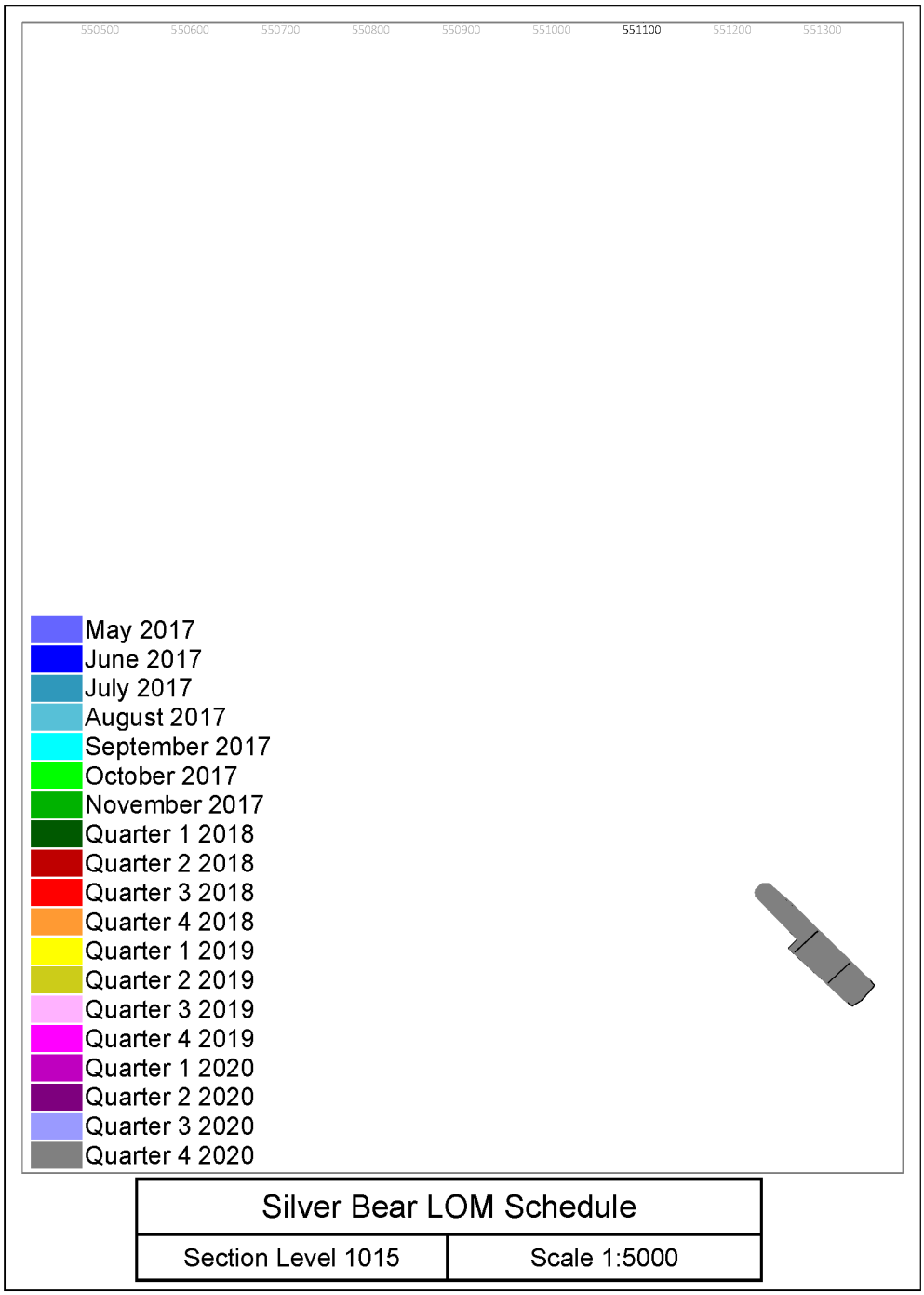
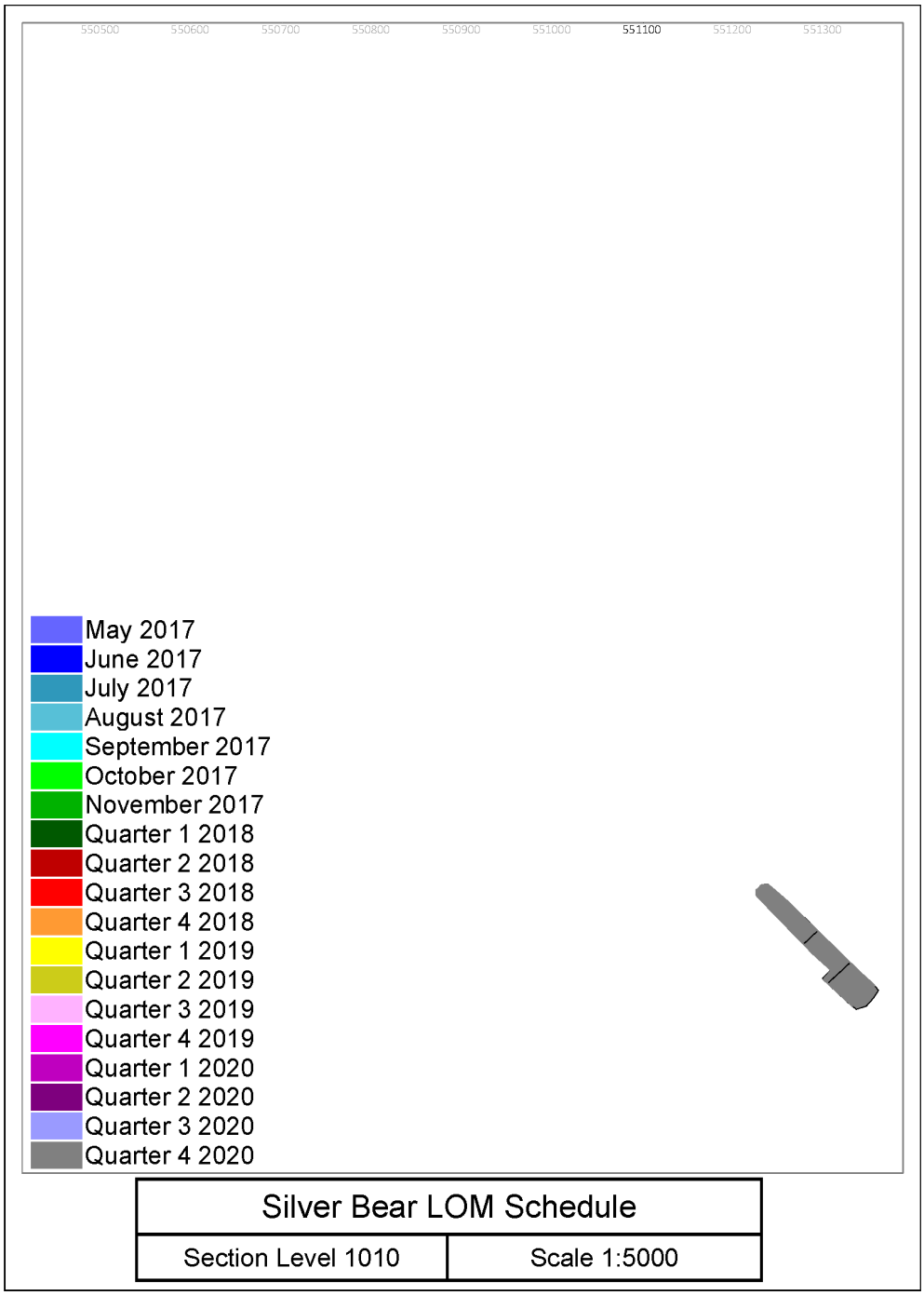


Figure 16.46 Mine Schedule Section Level 1010



16.1.9 Open Pit Mine Operations

Operational Approach

A conventional load-and-haul mining method is planned for the open pit, using relatively smaller equipment to produce 110 kt/a of ore and to carry out the waste

stripping. The mine operation, including maintenance and technical services, will be run by ZAO Prognoz.

Due to the harsh winter weather conditions, personal and equipment fatigue will be a major factor for the safety of the mining operation. Tetra Tech expects relatively higher mechanical fatigue of the mining equipment resulting in higher maintenance costs and lower mechanical availability when compared to an operation in a warmer weather condition.

Strict personal safety rules should be implemented to minimise the risk of the operation during winter.

It must be noted that, owing to the variable vein thickness, there will be areas with a narrower vein width than the excavator resulting in higher than average dilution. This is offset in thicker sections of the orebody where the converse is true.

In order to minimise the dilution and mining losses during the mining operating, the following methods could be deployed:

- approach the orebody from hanging wall front
- limit the ore production to day shift
- light the ore loading face at the night shift
- use smaller buckets for loading ore material
- accurate grade control
- clearly mark the contact between ore and waste in the pit.

A detailed estimation of the dilution was not carried out in this study.

Mine Equipment Requirements

A summary of the estimated fleet requirement to achieve the LOM plan is presented in Table 16.6 and the details are provided in Appendix C. The fleet estimates for drilling, loading, and hauling equipment was carried out using the developed mining schedule, a similar drill and blast pattern as per SRK (Tetra Tech 2016), approximate annual haul distances to the crusher and WRD, number of working faces per period, and the estimated work regime and mechanical availability and utilisation factors.

The ancillary equipment were estimated based on previous experience on similar projects and approximate required hours for the Project.

Table 16.6 Required Equipment Summary

Equipment	Type	2017	2018	2019	2020	2021
Truck	20 t	3	4	4	4	4
Truck	30 t	3	7	8	4	
Backhoe	CAT336	1	1	1	1	1
Backhoe	Cat 349	1	1	1		
Drills	Sandvik DI550	1	2	2	2	-
Track Dozer	CATD9	1	1	1	-	-
	CATD8	2	3	3	2	2
Grader	14M	1	2	2	2	1
Water Cart	-	2	2	2	2	1
Service Truck	-	1	1	1	1	1
Fuel Truck	-	1	1	1	1	1
Lowbed Truck	-	1	1	1	1	1
ANFO Truck	-	1	1	1	1	-
Loader	CAT930	2	2	2	2	1
Pickup Truck	-	10	10	10	6	5
Lighting Units	-	4	4	4	4	2

The equipment currently available on site, is as per Table 16.7. The two Cat 950 Loaders will be used as backup loaders in the pit as required. If the digging conditions prove to be more difficult than planned then a second Cat 336 will be purchased.

Table 16.7 Summary of the Existing Mining Equipment On Site

Equipment	Type	Number On site for Mining
Truck	20 t	4
Truck	30 t	3
Track Dozer	CAT D9R	1
Backhoe	CAT336	1
Backhoe	Cat 349	1
Loader	CAT950	2
Water truck	6 m^3	1
Fuel truck	8 m^3	1

One of the four Kraz 20 t trucks is currently being rebuilt and will only be available in March 2018.

Mining equipment should be utilised with arctic condition packages and the diesel must be winter-grade fuel for the cold months.

Shift Roster

The selected work regime for the mining operation amounts to 330 operating days and two, 12 hour shifts per day. This results in a total of 7,920 hours in a year. The

lower operating days take into account operation stoppage due to extreme weather conditions.

The technical services staff will be on a one shift schedule.

Details of the shift roster, the work regime, and utilised hours per year for the mining equipment are presented in Appendix C.

DRILLING

The selected blasthole diameter for this operation is 110 mm and the drilling pattern in ore will be 2.8 m by 3.2 m and in waste rock will be 3.3 m by 3.8 m. The drill-and-blast will be carried out in 5 m benches. The Powder Factor (PF) for ore and waste will be 0.88 kg/m^3 and 0.56 kg/m^3, respectively. In the estimates, 10% additional drilling is allocated to secondary blasting and control blasting operations.

The selected drill for this operation is a Sandvik DI550 downhole drilling machine, with a drilling diameter range of 90 to 165 mm. This machine is flexible and is equipped with an air conditioned cabin for an all year around operator comfort and is suitable for winter operation.

Two drilling units will be required by May 2018 and they will provide the required drilling capacity until the end of the mine life. The second drill will not be fully utilised and will act as a backup drilling machine in the primary unit breaks down. The estimated equipment hours and the required units are presented in Appendix C.

It is assumed that grade control will be carried out using the collected samples from blasthole drills muck pile. Separate calculations for grade control drilling requirements were not carried out in this study.

BLASTING

Currently, a blasting contractor is carrying out the blasting operation on site. However, with the commencement of the mining operation a dedicated blasting crew employed by the mine will be responsible for the blasting operation. Ammonium nitrate/fuel oil (ANFO) and water resistant explosives (if required) will be used for blasting. It is expected that most of the holes will be relatively dry or only partially wet. The ANFO will be prepared in an on-site plant.

An explosives magazine and an ANFO mixing plant will be available on site to prepare and store the explosives and the blasting accessories.

The annual explosives requirements were determined based on estimated powder factors of 0.88 kg/t and 0.56 kg/t for ore and waste rock, respectively. The powder factors have been calculated for this study and are presented in Appendix C. In addition, a 5% allowance was used in the estimates for a secondary and control blasting operation. The blasting requirements are presented in Appendix C.

Trim and control blasting techniques could be used for the following purposes:

- selective blasting in ore vein
- minimising potential damages to the pit wall due to back breaks

- maximising bench face angles and therefore the overall slope angle
- lowering the risk of fly rock.

The safe blasting distance to the administrative buildings, fuel tank, and other sensitive infrastructure is usually 500 m (local by-laws should be used for this purpose). During the blasting operation, as part of the safety procedures, all administrative and mine operational personnel should be evacuated to a safe distance of not less than 500 m to a covered and secure location. The mining equipment should also be moved to a safe distance from the blast.

LOADING

Backhoe excavators are the most suitable loading equipment that could be used in this open pit mining operation. The flexibility of the backhoe units for selective mining of the blasted orebody, and the manoeuvrability in small working spaces, are a few of the advantages of this loading equipment.

A backhoe excavator with a 1.65 m^3 bucket capacity is already in operation on site. The smaller bucket size will be suitable for selective ore excavation. Due to the low ore production rate, this excavator will also be utilised in the waste stripping operation. The larger bucket size (2.5 m^3) backhoe units will be dedicated to waste stripping. The recommended units are equivalent to CAT 349 and CAT 336 backhoe excavator models.

The existing CAT 950 front-end-loaders on site will be utilised as a secondary loading unit, mainly for depleting the crusher stockpile. If needed, they can be used in the open pit in case of a temporary breakdown of one of the excavators.

Assuming that the CAT 349 is predominately used for waste, and the CAT 336 is mainly used for ore, the loading productivity is estimated at 3.1 Mt/a and 2.2 Mt/a, respectively (Table 16.8). This is sufficient to meet the peak mine productivity of approximately 4.6 Mt/a.

Table 16.8 Loading Productivity

	Units	Material Type		
		Fresh Waste	Fresh Ore	Fresh Waste
Loading				
Loading Unit	-	CAT 349	CAT 336	CAT 336
Bucket Size	m^3	2.5	1.65	1.65
Loading Spot Time	min	0.50	0.50	0.50
Loading Cycle Time	min	0.50	0.50	0.42
First Bucket Dump	min	0.10	0.17	0.10
Haulage				
Truck	-	Scania	KrAZ	KrAZ
Capacity	t	30.0	20.0	20.0
Capacity	m^3	23.0	10.0	10.0
Dump and Spot Time	min	0.50	0.50	0.50
Average Return Distance	m	1,500.00	3,200.00	1,500.00

table continues...

	Units	Material Type		
		Fresh Waste	Fresh Ore	Fresh Waste
Average Speed	km/f	22.00	22.0	22.0
Travel Time	min	4.09	8.73	4.09
Loading Parameters				
Bucket Fill Factor	%	90	90	90
In Situ Density	t/bcm	2/75	3.17	2.75
Swell Factor	lcm/bcm	1.49	1.49	1.49
Loose Density	t/lcm	1.84	2.12	1.84
Moisture Factor	%	5.0	5.0	5.0
Passes	no.	7.2	6.3	6.7
Passes (Rounded)	no.	7	6	7
Passes (Override)	no.	7.0	6.0	7.0
Loaded Quantity	t	20.0	20.0	18.4
Loaded Volume	m3	16.3	9.4	10.0
Loading Productivity				
Total Loading Cycle Time	min	3.60	3.17	3.10
Loader Operator Efficiency	%	100	100	100
Loader Productivity	t/doh	476	361	340
Loader Productivity	lcm/doh	258	170	184
Loading Unit Utilisation	%	75	75	75
Loading Productivity	Mt/a	3.1	2.4	2.2

An allowance has also been made in the schedule for 33 days lost due to weather in the months between November and February.

It must be noted that in addition to the loading capacity of these units, the number of loading faces in the North and South pits are factors used in estimating the required number of loading units in each year.

The details of the excavator requirement estimates are presented in Appendix C.

HAULING

The required dump trucks were estimated by using a mix of 20-t and 30t trucks and the calculated haul distances for ore and waste rock based on the haul profile between the working benches and the respective destinations (ROM pad, low-grade stockpile and waste dump). The average speed for each sector of the road was estimated using the assumed truck speeds on the flat and on gradient to calculate the truck cycle time. Using the calculated cycle time, the truck productivity was then estimated and accordingly the required trucks for each period were determined.

The larger Scania trucks with a 30-t capacity will be a mismatch with the smaller backhoe excavator, resulting in a longer loading cycle time and lower excavator productivity. The Scania trucks are therefore preferentially allocated to the larger CAT 349 and only sent to the CAT 336 or the Cat 950 FEL if there is excess truck capacity.

Allowance has been made for the weather conditions, and the lower mechanical availability of the trucks that are manufactured in Belarus, in determining the optimal

number of trucks per loading equipment. Consequently, the fleet of Scania trucks is progressively increased from three during pre-stripping to eight in 2019. Additionally, it is assumed that the fleet of four KrAZ trucks will be used throughout the life of the open pit. However, one of the KrAZ trucks is being rebuilt and will only be available from March 2018.

The details for the required truck estimation are provided in Appendix C.

Ancillary Equipment

The major ancillary equipment requirements (dozers, graders, water trucks, secondary loader, service truck) were estimated based on the total rock movement, number of loading and dumping stations, rough assumptions regarding earthmoving requirements, regular road maintenance, and snow removal.

A significant amount of mining equipment, such as an excavator, 20-t and 30-t trucks, track dozers, a road grader, and front-end-loaders are already in operation for road building, pre-stripping and other earthmoving operations at site.

Deploying a sufficient number of dozers will result in the following operational improvements:

- increase in dozing capacity on the WRD
- allocation of one dozer per excavator, and improved scheduling of the remaining dozing capacity for road maintenance and WRD
- better maintenance of bench and WRD working surfaces
- clearing and levelling the loading area resulting in an increase in shovel productivity
- improvements on the travelling surface on each bench, and therefore an increase in truck travel speed and lower tire repair and maintenance cost
- improvement in the long-term maintenance cost of the load and haul fleet.

Utilising the smaller size track-dozers will complement the existing larger dozer to achieve the optimum dozing capacity and allow the larger dozer to be utilised in road building activities in 2017 and 2018.

The road maintenance will be carried out by two CAT14M road graders together with two, 20 kl capacity water carts. Utilising larger water carts would reduce the number of filling times; however, it is understood that smaller size models are already in use on site. The use of water carts in above freezing condition is crucial to minimise the road dust; however, during the freezing temperatures there will be minimal or no use for this equipment.

Fuel trucks will mainly be used for re-fuelling the in-pit loading and dozing equipment.

A low-bed truck would be useful for transportation of the dozers and the backhoe excavators for distances greater than 500 m.

A service truck with a fuel and lube compartment could be used for the in-pit maintenance and small repair operation.

The available loading equipment on site could be used as a secondary loading equipment in the pit in case of a breakdown of backhoe excavators and in utilising the stockpile. It could also be used to transport heavier items such as equipment tires, sump pumps, and other items that would fit in the bucket or on the utility handler attachment.

Pickup trucks, lighting units and sump pumps will also be required for the operation.

16.2 UNDERGROUND MINING

16.2.1 UNDERGROUND MINING METHOD SELECTION

An analysis of the mining methods available to exploit a narrow vane, steeply dipping ore body such as the Vertikalny deposit identified two main contenders, namely sub-level open stoping and shrinkage stoping. Tetra Tech selected sub-level open stoping as it offers the most favourable outcome in the three key areas of safety, cost, and dilution. Drawbacks of the labour intensive shrinkage stoping include:

- Shrinkage stoping leaves too much high-value, broken ore in the stopes for extended periods.
- Shrinkage stoping is a low ore productivity operation, which requires numerous stoping areas to be developed simultaneously to meet the production demand and to provide sufficient ore contingencies to smooth out mine production. As a result the development requirements increase significantly when compared to sub-level open stoping.
- Because it requires more people working multiple areas to work at the face, shrinkage stoping exposed miners to rock falls when mining the voids and considered less safe than mechanised sub-level open stoping.
- Shrinkage stoping is a manual, labour intensive mining method, with relatively poor productivity factors and lower operating efficiencies.
- Shrinkage stoping is a very expensive exploitation method, especially for low-production operations, due to the labour requirements and the number of operators required on a two-shift-per-day roster. This effect is exasperated on a remote mine where personnel carry a high G&A cost.
- Planning mechanised loading and hauling with labour intensive stoping methods has always proven problematic. It requires that the stoping mechanisation is coordinated with the loading and hauling strategy to fully realise the benefit of improved mining technology and increase productivity.

It is concluded that the recommended underground mining method for Vertikalny is mechanised sub-level open stoping, with remote stope loading, and the recommended sequence of stope extraction is retreat-top-down mining, with adequate in situ rock pillars left unmined for localised and regional stability.

16.2.2 OPTIMISATION

Numerous underground stope optimisation runs were performed using Datamine MSO software to develop stope shapes for various minimum mining widths and

vertical level spacings. The MSO analyses concluded that a 25-m vertical-level spacing, with a top and bottom ore drive yielding an effective 22-m vertical stope drilling height, is appropriate for the Vertikalny deposit. The high-grade mineralisation envelopes were modelled with surrounding lower grade mineralisation and, when targeting stope shapes within higher grade mineralisation, it is possible to have portions of the low-grade mineralisation as part of the planned mining dilution.

Mining loss factors were applied to simulate the possible blockages that could occur with very narrow planned stopes. Accurate small diameter blasthole drilling, together with an adequate powder factor and accurate blast initiation, should mitigate the risks of stope blockages.

16.2.3 GEOTECHNICAL CHARACTERISATION

SRK prepared an open pit and underground geotechnical study in support of the feasibility study (Appendix B). It was noted that the ground conditions are generally good with no special measures required for orebody extraction.

The updated underground mine design also considered the recommended hydraulic radiuses as documented in (Appendix B).

16.2.4 HYDROLOGICAL AND HYDROGEOLOGICAL CHARACTERISATION

SRK specified and supervised a hydrogeological site investigation to:

- achieve sufficient understanding of the hydrogeology of the proposed mine and assess the dewatering requirements
- quantify and assess the availability of the groundwater storage in the sub-permafrost aquifer to supply the required amount of water to the Project.

The following conclusions were drawn from the site investigations and data analysis (Appendix K):

- Both the open pit and underground mines will be entirely located in the permafrost; therefore, groundwater inflow into the mine workings, if any, will be negligible.
- Surface water runoff during the summer months (May to September) remains the main inflow component that requires active management, especially during exceptional storm events. For average conditions during this period, the required pumping capacity to manage surface water runoff is approximately 100 m^3/h; however, for exceptional storm events (100-year return period) the required pumping capacity is 200 m^3/h. During freezing conditions (October to April) surface water is frozen and precipitation falls in the form of snow; runoff does not occur.

16.2.5 UNDERGROUND MINE DESIGN

MINE LAYOUT

The underground mine will consist of two distinct mining areas: the North Zone and South Zone. As there is no advantage to connect the two mining areas, each zone

will have its own independent access decline system. Ramp accesses were selected for both underground mining zones, taking into account the stopes' proximity to the bottom of the planned open pits and the need to provide for operational flexibility through multiple mining areas and accesses. The South Zone design includes adit access for the upper mining levels from within the South pit, linked to the deeper mining levels with a South Zone ramp system. The South Zone ramp system is necessary to provide adequate ingress and egress opportunities and for ventilation control purposes. Each zone has its own ventilation districts.

Figure 16.47 depicts the underground mine layout designed for Vertikalny.

Sub-level open stoping is typically described as in-ore development with retreat long-hole open stoping extraction. The ore drives, slot raises, and stoping blocks (in-ore excavations) are accessed by a waste decline system located close to the orebody; however, not too close, as there are risks such as blast shock vibrations and excavation interaction stresses that could impact the integrity of the main access development drives if they are located too close to each other.

Figure 16.48 depicts the access option for the South mine. South mine's access consist of a series of in-pit adits with ore drives that ultimately link up to the South mine decline system. The advantage of in-pit access is the shorter distance to the underground mineable resources and to a centralised position from where the decline system can be developed.

Figure 16.47 General Underground Mine Design Layout

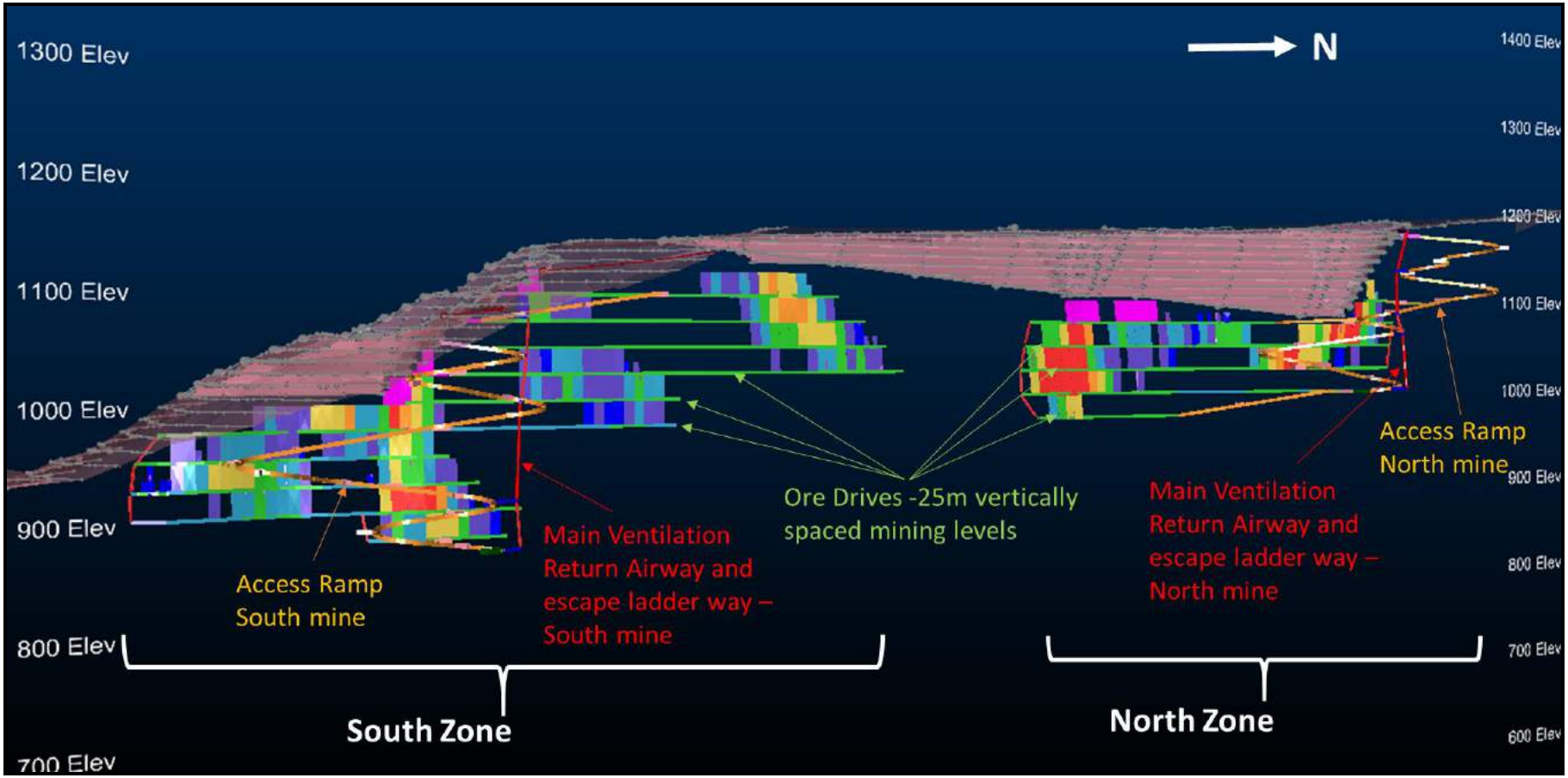
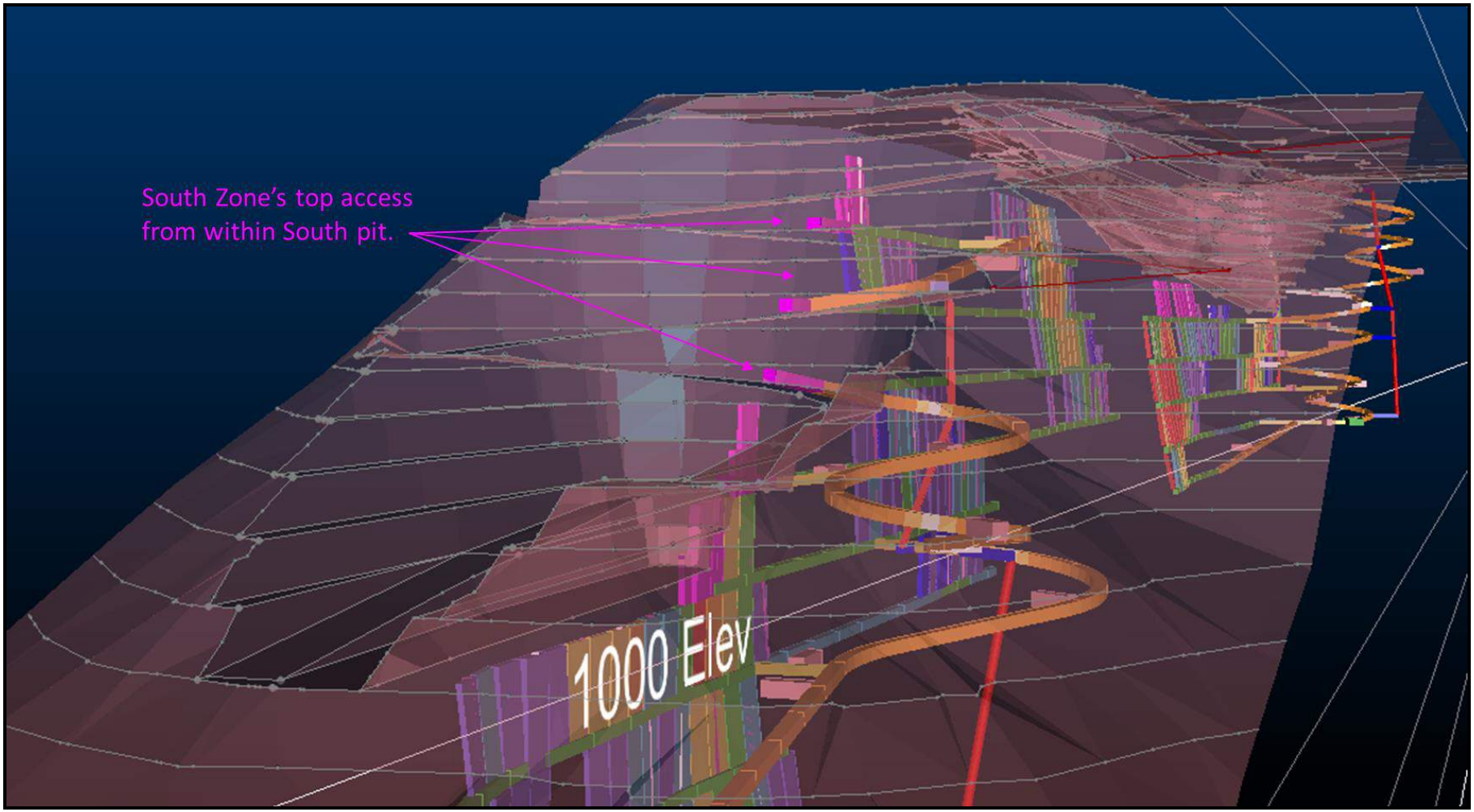


Figure 16.48 South Zone Top-level Access



DEVELOPMENT AND PRODUCTION SCHEDULE

Access Availability

Considering that underground access ramps and adits are tied into the open pit designs, the timing of their development is contingent on the open pit mining schedule. Underground mine development is planned to commence six months prior to the conclusion of the open pit operations to ensure continuous ore feed to the mill, and it will be further five months before the first stope will be ready for production.

The production ramp-up period for Vertikalny is assumed to take around 15 months, at which time 9.2 kt/mo of ore will be consistently produced from the underground operation. The underground ore will be stockpiled due to variations in ore production rates, owing in large part to variations in stoping widths and the linear development advance rate restrictions imposed.

Development Rates

Based on Tetra Tech's experience, a maximum development rate of 145 m/mo (in a single heading) was applied to main development headings. The overall development rate was limited to 295 m/mo, based on the estimated productivity of the development equipment (single boom jumbo and load-haul-dump (LHD)) and the peripheral activities such as grade control in ore drives, support, and service installations. It should take the production crews approximately six months to ramp-up to 250 linear meters per month of overall mine development due to the time it takes to establish sufficient development headings (which is essential for maximum utilisation of the single boom jumbo drill).

Stope Production

Each 25 m high by 10 m strike length stope was designed together with each stope slot raise. The slot raise was used to link activities to the stope activities and each stope rate was calculated based on the time it would take to prepare the stope, develop a stope plan from a grade control block model, to drill the stope long holes, and to blast the stope.

The slot raises will be developed prior to the actual stoping operations (well in advance of stoping commencement) to allow for clearing of slot raises that may be blocked because of poor drilling and blasting achievements and to allow for through-ventilation. Some slot raises may also be equipped with a simple chain ladder should they be considered as temporary escape ways. Ore activities leading up to a typical stope depletion include:

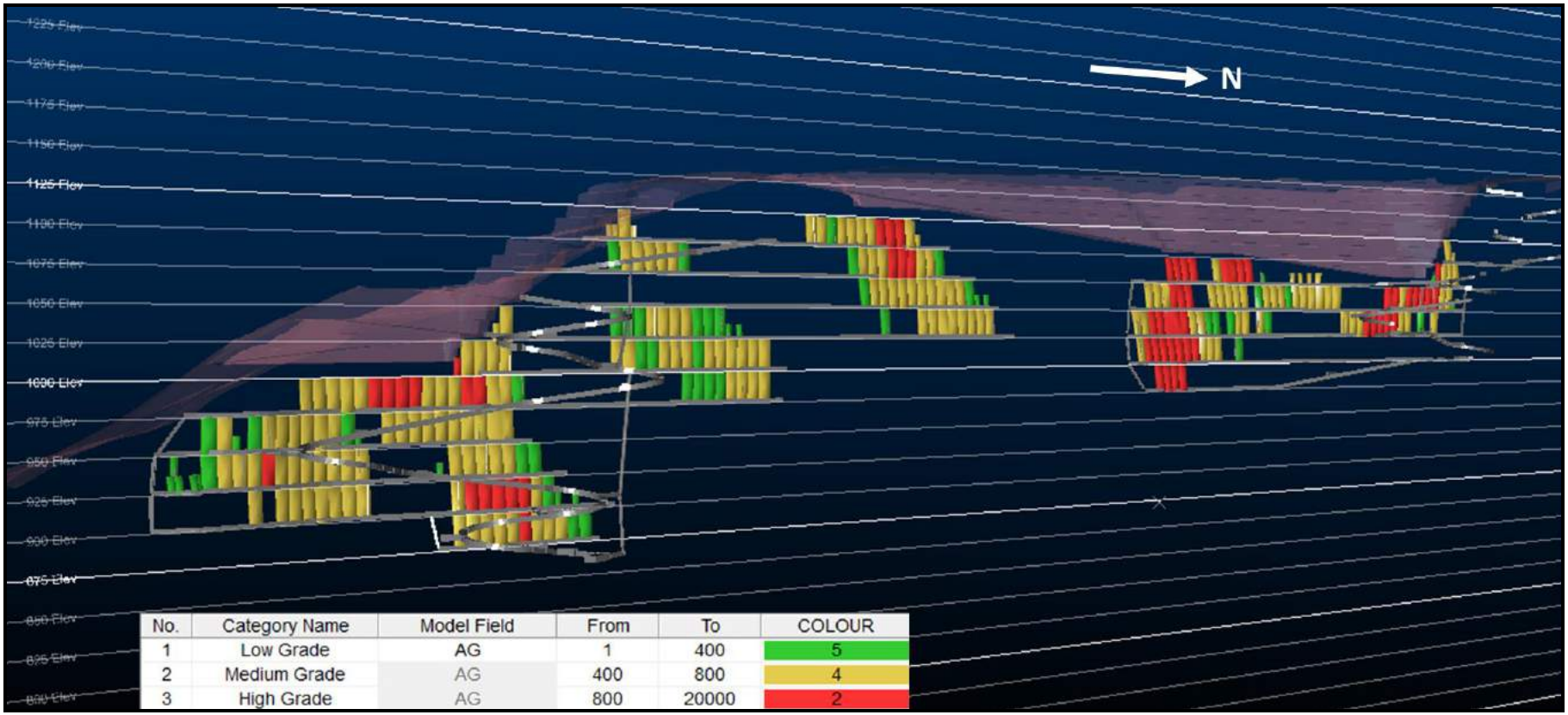
- Ore drives will be developed (3 m wide by 3.2 m high).
- Slot raises will be developed 2 m from the end of each planned mining stope (1.5 m wide by 1.5 m high)
- Drilling and blasting of slots may be done from either the top (downholes) or the bottom (upholes). From experience, clearing of holes when drilling upholes are more efficient and require less re-drilling. There may be the need to drill upholes and downholes in some stopes where stoping widths of less than 1.6 m are planned. Drilling from both sides could potentially

reduce the mining dilution, while allowing improved drilling accuracy, especially where the mineralisation poses undulating geometric challenges.

- Stope production rates of 240 t/d will be scheduled (once the stope is ready and available for extraction).
- The typical stope drilling time required will be three hours or less. The slot raises for each stope contain eight blastholes, and will be drilled in approximately 2.5 hours. The slot raises and the stope blastholes will be drilled sufficiently in advance, prior to a stope being blasted.
- Stope blastholes will only be charged (filled with explosives) a day or two prior to the stope being fired (the explosives standing time in the stopes will therefore be reduced as practically possible).
- Extraction factors will be applied to each stope:
  - Crown pillar stopes (stopes located near the crown pillars below the planned open pits) have a 90% extraction factor applied with a further 5% mining loss.
  - The remainder of the stopes have a 95% extraction factor applied with a 5% mining loss.
- A monthly production target of 9,160 t/mo has been set for scheduling purposes.

Figure 16.49 depicts the designed stopes coloured by grade bin.

Figure 16.49 Stope Grade Profile (Planned Stopes)

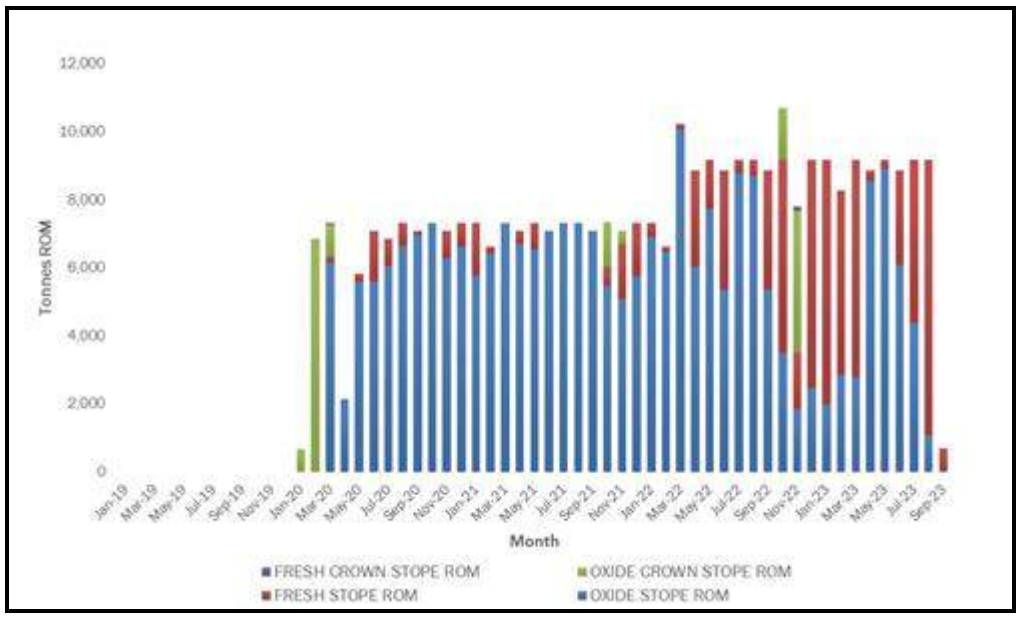


No.	Category Name	Model Field	From	To	COLOUR
1	Low Grade	AG	1	400	5
2	Medium Grade	AG	400	800	4
3	High Grade	AG	800	20000	2

Some smaller stopes are visible; these will be drilled with upholes from the bottom access ore drive.

While stope optimisation was completed in Datamine MSO, the underground designs, sequencing, and production scheduling was developed using Datamine Studio 5D Planner and EPS software.

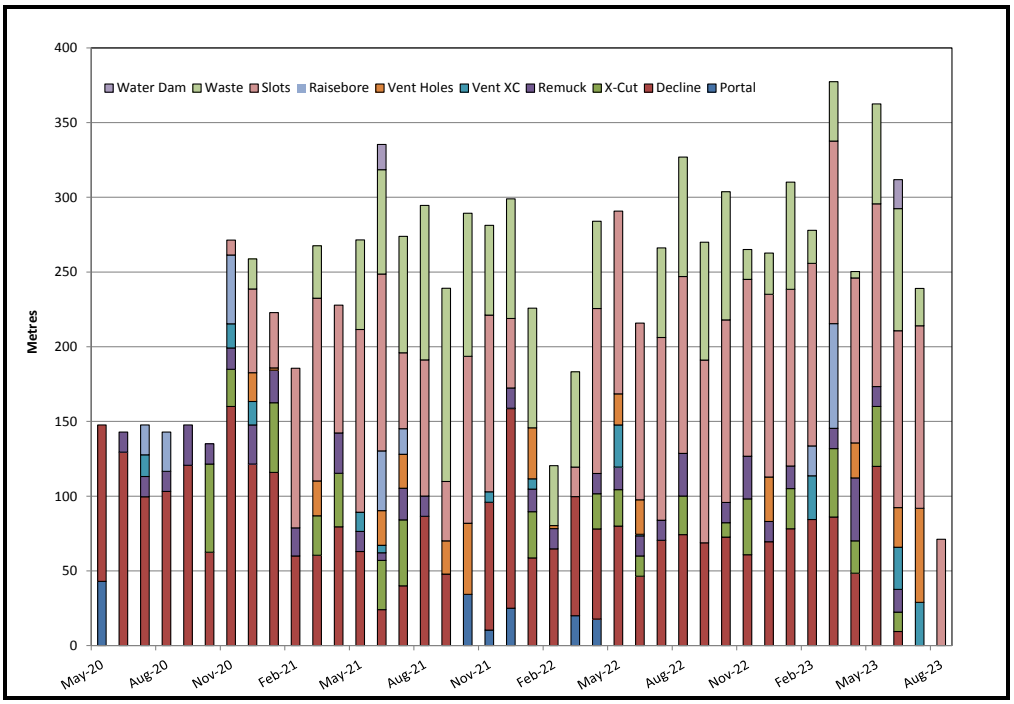
Figure 16.50 Stope ROM Production Profile (Schedule)



Development Schedule

Figure 16.51 depicts the lateral waste development schedule for the proposed underground workings. Waste rock will be sent to the WRD. Development and production schedule rates for the North and South zones take into account material handling and technical management logistics, such as length of ore and waste rock haulage routes, equipment cycle times drill area preparation and ore drive grade control. Due to the overlap between open pit and underground operations at the start of underground development, the blasting schedule for both open pit and underground operations should be adapted to suit shift change. Re-entry periods will be dictated by the blast size and ventilation in the affected areas.

Figure 16.51 Underground Mine Development Schedule



16.2.6 UNDERGROUND MINE SERVICES

ELECTRICAL RETICULATION

The underground mining equipment, and therefore underground service requirements, were calculated based on the production schedule developed for Vertikalny.

Underground power will be generated on surface by a diesel generator located either near the main decline portal of North Zone where a bulk of the initial mining will take place, or strategically located using the minimum electric cabling lengths. A back-up generator will service both mining zones. It is highly unlikely that two generators will be non-functional at the same time; therefore one back-up generator should suffice for the entire Vertikalny underground mine. The North Zone will be fully developed before the South Zone development commences and stoping will only commence in the South Zone once the North Zone is depleted. This allows the strategic planning of services, the potential to reclaim services from the North Zone, and to install some of these services in the South Zone. Underground mine planning includes basic

As the underground mine is fully mechanised, strategically placed electrical sub-stations, which provide power to all headings, are necessary within 500m of any mine heading. An electrical substation will be located on each mining level. Electrical substations can be moved or relocated once a mining level is depleted. This can be scheduled during night shift with switch over scheduled at strategic shift changes.

Power points will be located no further than 80m from active drill headings as the cable reels for the single boom jumbo drills will be limited to 100m. Power points for the single boom jumbo drills will be three phase points, with 60KW rating.

services for every development meter designed; cost calculations assume new services everywhere in the underground mine.

MINE VENTILATION

Ventilation requirements were calculated taking into consideration the simulations concluded in the previous feasibility study (Tetra Tech 2016).Total air volume is calculated as 72 m^3/s, should the following equipment all be running continuously in the same mining section:

- one jumbo
- one development LHD
- one production LHD
- one development truck
- one production truck
- one long hole.

When considering the maximum ventilation flow and the average ventilation requirements, it is deemed acceptable to allow for a 60% continuous equipment utilisation (multi-fleet or equipment scenarios). This also assumes that the equipment will be running full throttle, which is unlikely for the road grader, service vehicles, and light vehicles.

An airflow of 45 m^3/s is the result of using a 0.6 factor on the whole fleet. Primary flow designs will be based upon a 56 to 58 m^3/s delivery system (Table 16.9).

Table 16.9 Ventilation Flow Requirements

Diesel Fleet Equipment	Engine Size (kW)	No. of Equipment Running Simultaneously	Air Requirements Without Multi-unit Factor (m^3/s)	Air Requirement With Multi-unit Factor (m^3/s)
Jumbo	55	1	6	4
Development LHD	90	1	10	6
Production LHD	90	1	10	6
Development Truck	175	1	19	12
Production Truck	175	1	19	12
Long Hole	73	1	8	5
Grader	90	0	0	0
Service Vehicles	70	0	0	0
Light Vehicles	68	0	0	0
Total	-	-	**72**	**45**

The light vehicles, service vehicles, and motor grader were not included in the ventilation calculation as these vehicles should not consistently operate underground during full-production tramming hours. The motor grader may only be used when the haul trucks are being re-fuelled and light vehicles will be parked in a re-muck bay when other equipment runs in the declines.

Ventilation Fans

A total of eight, 55 kW fan units will be in full operation at any one time for total power consumption. There will likely be 10 to 11, 55 kW fans installed, but to allow 45 m^3/s of air through a maximum airway pressure system of 10,000 Pa, 8 fans running at almost full capacity will be required. Therefore, the ventilation power requirements include eight, 55 kW fans running at approximately 80% efficiency.

The cost of ventilation ducting was included as part of the underground mine operating costs and additional secondary 55 kW fans to boost the airflow were also included in the underground mine operating costs under air installed costs.

Water Supply and Pumping

Given the depth of permafrost at the Project, Tetra Tech expects the underground mine to be dry with negligible ground water ingress, however, this will be confirmed with continuous water studies at Vertikalny. For this Feasibility Study, Tetra Tech assumed that water will be reticulated to the mine in a typical 4-inch or 100-mm diameter polyvinyl chloride (PVC) pipe. This will feed a collection sump underground that has a float valve. The collection sump will be situated at the lowest adit of each of the discrete "mines" and will receive water from the levels above via ramps, raises, and drain holes. The stopes will also convey water downwards. The gradients of the underground development will generally facilitate drainage from the mine towards the adits to avoid flooding, but all excess water will be routed to the collection sumps. The underground mine is expected to be a major consumer of water, with limited discharge to the environment. A key point to note—the local laws regarding heating the mine air to 2°C do not apply during permafrost conditions. Therefore, an alternative approach must be taken with the mine water supply to prevent regelation. The incoming water supply pipe should be insulated and salt added upon discharge at the sump. This is a common approach adopted in Russia in these kinds of conditions where underground heating is not used.

Each collection sump is planned to consist of a 10-m long excavation, 10 m^2 in cross section. Dirty water will flow into the sumps for primary settling before cascading over a wall into the sump where the pump will be situated. The supply will be conveyed to the working areas in 100 mm HDPE pipes suspended on the sidewalls of the drives. Water supply for the stopes will utilise 100 mm diameter HDPE pipes reducing to 50 mm rubber pipes for the drills. The key uses of water anticipated for the underground operation are summarised in Table 16.10.

Table 16.10 Water Usage Calculations

Unit	Water Usage (l/min)	Water Usage (l/h)	Drilling (h/d)	Water (l/d)	Water (m^3/d)
Jumbo T1D	45	2,700	14	37,800	37.8
Long Hole Drill Rig	50	3,000	14	42,000	42.0
Watering Ore				25,000	25.0

Water consumption for watering down ore post blasting is estimated at 8% of ROM production. Water content will need to be carefully controlled to reduce the risk of ore freezing during the winter months.

The data in Table 16.10 are at steady state, assuming full production with one jumbo development drill rig and one long-hole drill rig in operation.

Water usage and salt addition will need to be monitored carefully during operations to prevent the build-up of ice in the workings.

During the summer months approximately 60% of the service water could potentially be re-used depending on the extent of the underground workings. The ground conditions and rock temperature will dictate the overall water balance.

16.2.7 Mine Development Schedule

The following underground development excavations were designed to provide main access to the Mineral Resource:

- decline (main access)
- remuck bays (where rock will be re-handled from LHD to haul trucks
- waste access drives (drives from the decline to the ore access drives (ore drives)
- ventilation drives (these are ventilation tunnels connecting the main access drives to the ventilation hole top and bottom holing positions)
- ore drives (these are in ore tunnels/excavations along the strike of the mineralisation zone, giving drilling access for slot raise and stope drilling and loading)
- slot raises are vertical excavations developed from the bottom ore drives (within each stope). These raises are developed to provide a free breaking face during stope blasting and to act as a temporary ventilation raise and escape way (a link between the sub-levels).
- water sump excavations (tunnels developed wherein a large water sump will be established)
- short ventilation holes (drilled by the long hole drill rig) and drop-raised
- longer ventilation holes were scheduled and costed to be developed by raise boring contractors.

All ventilation holes will have a strong polyester escape ladders installed with the steps folded down to reduce air resistance. They can be deployed if these ventilation raises are needed as escape ways.

Figure 16.52 and Figure 16.53 summarise the development end sizes for the Vertikalny underground mine.

Figure 16.52 Development Sizes

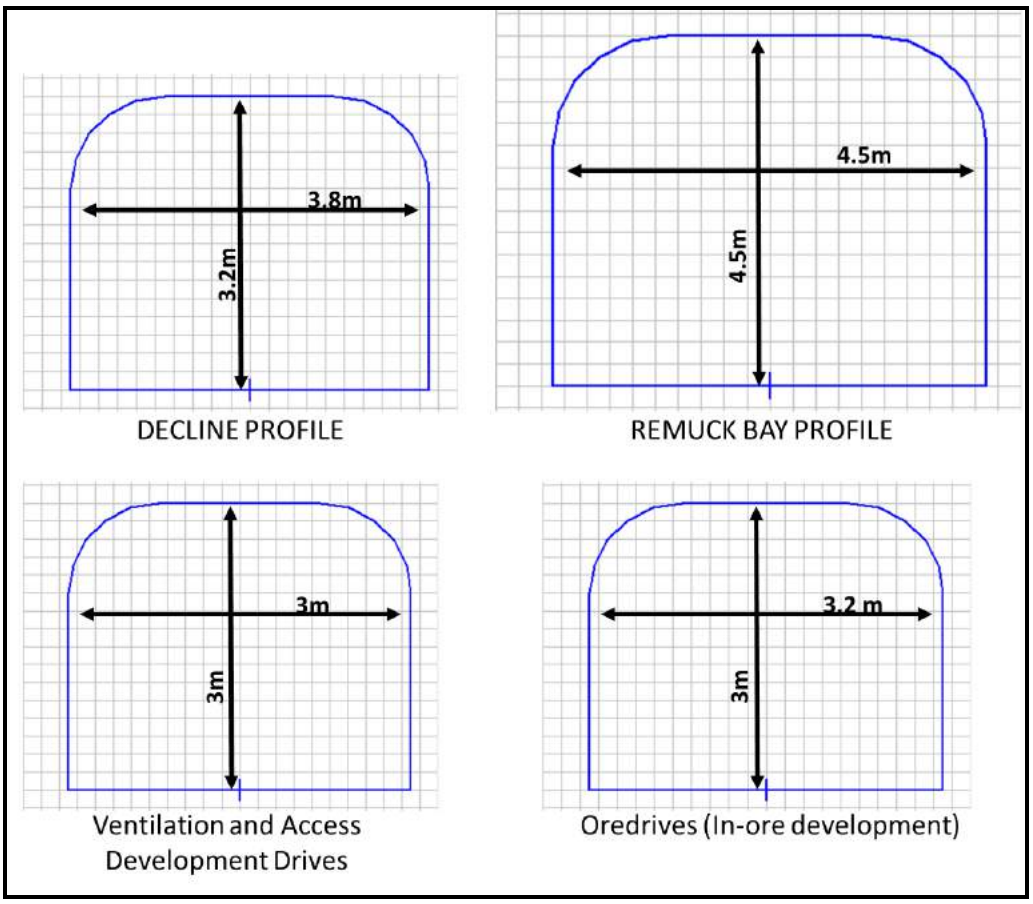
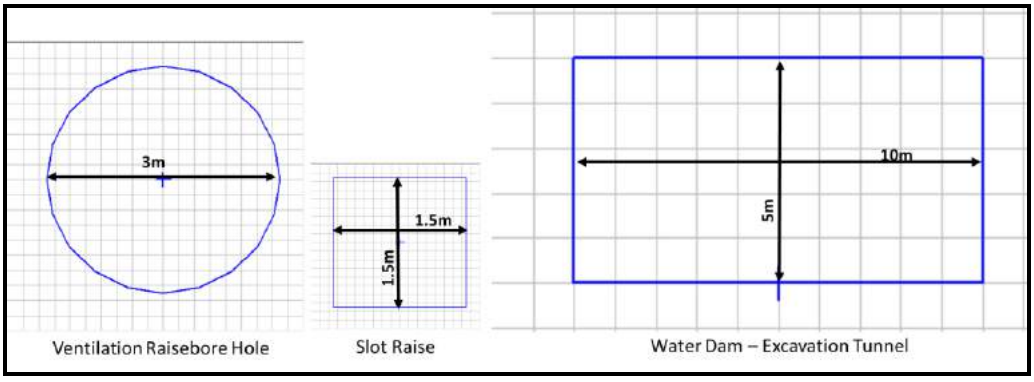


Figure 16.53 Other Excavation Sizes



The development drive end sizes considered the small mining equipment considered for development activities. The following figures illustrate the equipment sizes:

Figure 16.54 Single Boom Jumbo Drill Rig for Development Drilling

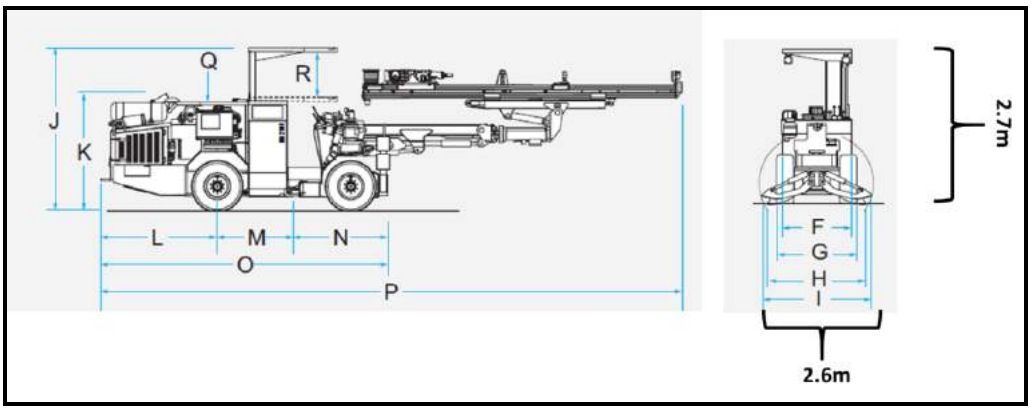
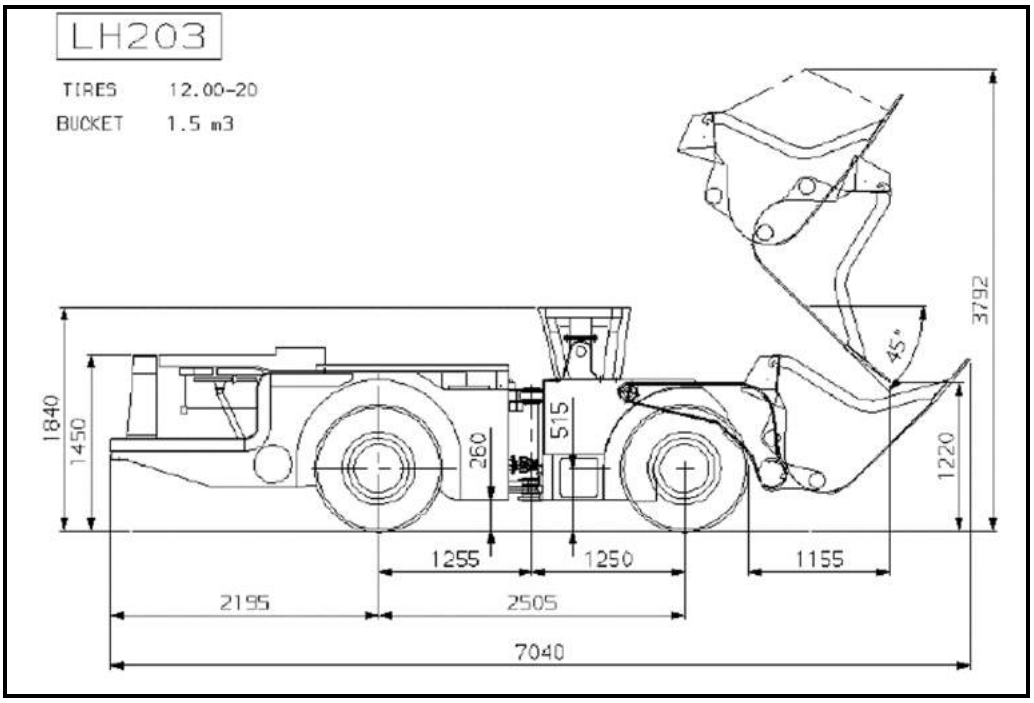


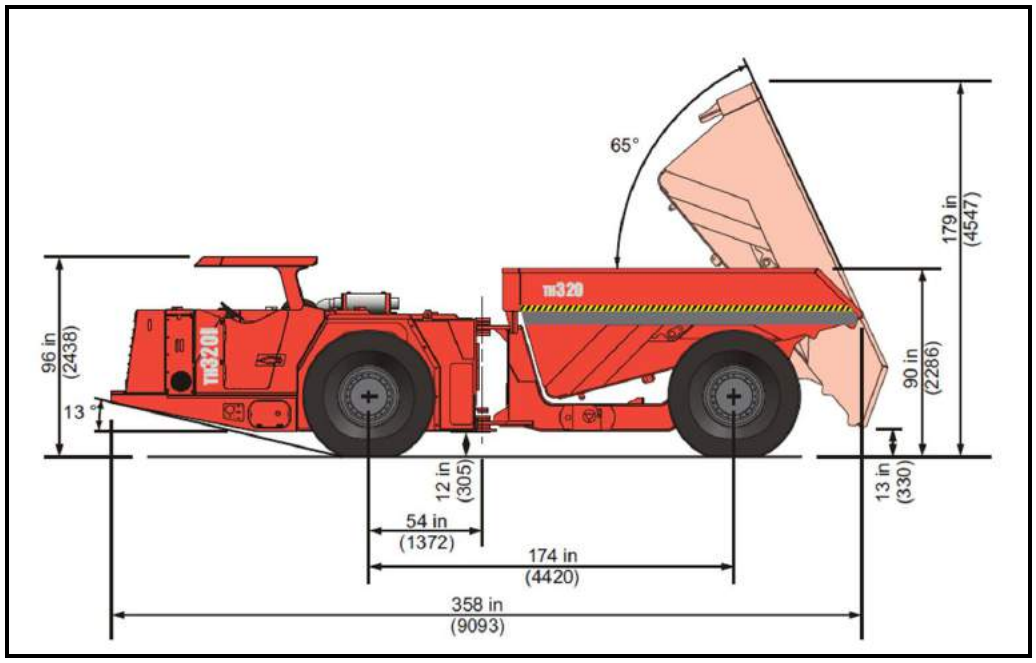
Figure 16.55 Typical LHD Equipment Size Specification Considered



The LHD will only dump rock into haul trucks in the re-muck bays, where the height of excavation is 4.5 m. The width of the LHD is typically 1.5 m.

Vertikalny may likely use 17- or 20-t haul trucks for the underground operations. Tetra Tech considered the development heading sizes that are able to accommodate a 20-t haul truck (Figure 16.56).

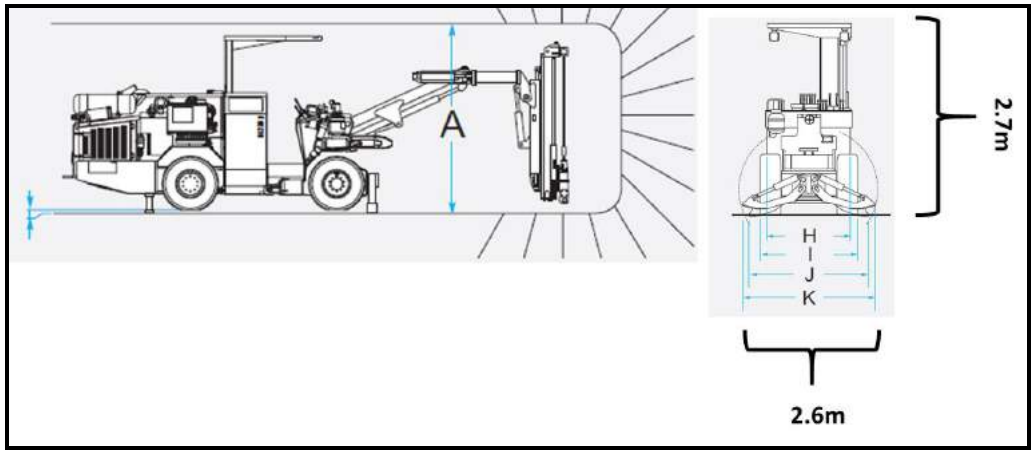
Figure 16.56 20-t Underground Mining Truck Dimensions Considered



The equipment sizes presented does not necessarily reflect Tetra Tech's equipment recommendations. The equipment available in the Tetra tech mining database was used to illustrate typical size specifications.

The ore drive size dimensions were increased (compared to Tetra Tech (2016)) which is necessary to allow for a small long hole stope drill rig (Figure 16.57).

Figure 16.57 Long Hole Drill Rig (Typical Dimensions)



Tetra Tech does not necessarily recommend any specific equipment brand. The equipment figures depict dimensions considered when designing the excavation sizes for the Vertikalny underground mine.

The statistics in Table 16.11 summarise the total development requirements.

Table 16.11 Development Summary

Description	Unit	Summary of Development
Portal	m	151
Portal	m³	2,228
Decline	m	2,931
Decline	m³	39,475
Cross-cut	m	583
Cross-cut	m³	5,246
Waste Drives	m	1,637
Waste Drives	m³	15,719
Remuck	m	523
Remuck	m³	10,598
Water Dam	m	36
Water Dam	m³	1,813
Ventilation Cross-cut	m	194
Ventilation Cross-cut	m³	1,760
Ventilation Holes	m	383
Ventilation Holes	m³	1,531
Raisebore	m	239
Raisebore	m³	1,692
Slots	m	3,119
Slots	m³	7,018
Ore Drives	-	3,599
All Meters - Jumbo Rig	-	9,654

The development schedule is depicted in Figure 16.51.

16.2.8 Mine Production Schedule

The mine production schedule was developed using Datamine Studio 5D Planner and EPS software. Individual mining activities were developed and a practical mining sequence dictated the order of the mining activities. Applicable development and extraction rates were applied to each activity.

The key schedule drivers were:

- rate of individual development end advance
- total utilisation and efficiency of the jumbo development drill rig
- shifts and working hours available
- total advance that can be achieved because of development excavation availability.

The production schedules considered various production rates modelled within Studio 5D Planner and EPS software for each unique mining segment. All the

necessary mining activities were developed within the Gantt-chart type scheduler, which is linked to the underground mine design. Mine depletion sequence rules governed the schedule progression, while the highest ore grade stopes were prioritised for extraction as early as possible. The first half of the silver profile graph (Figure 16.60) depicts the higher average value activities achieved with this particular mine production schedule.

Figure 16.58, Figure 16.59, and Figure 16.60 summarise the underground mine production schedule.

Figure 16.58 Stope ROM Schedule

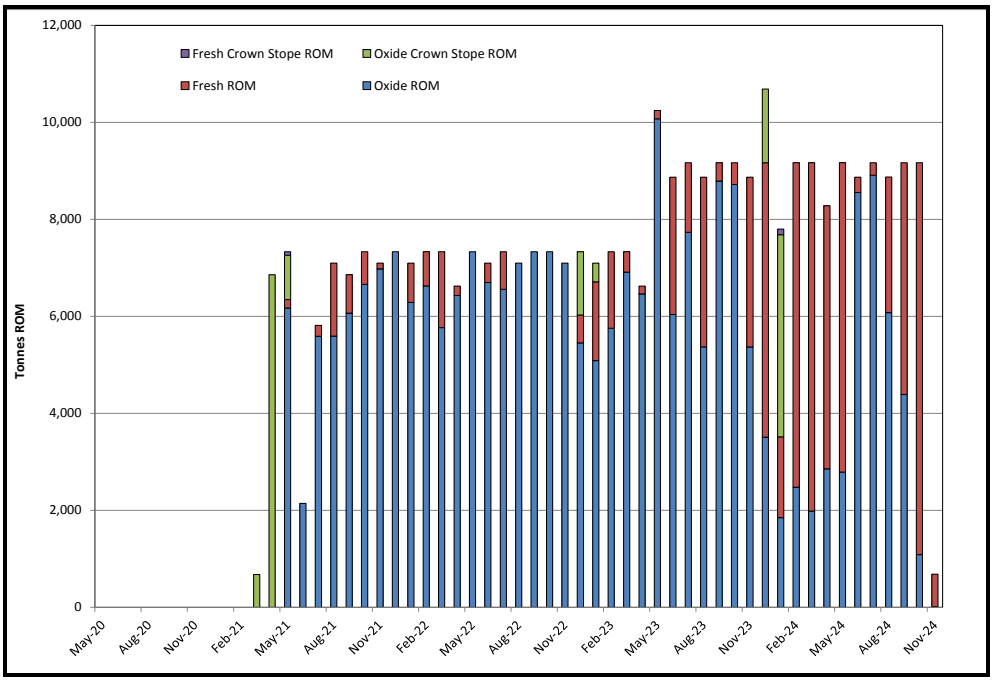


Figure 16.59 Total ROM Production (Monthly)

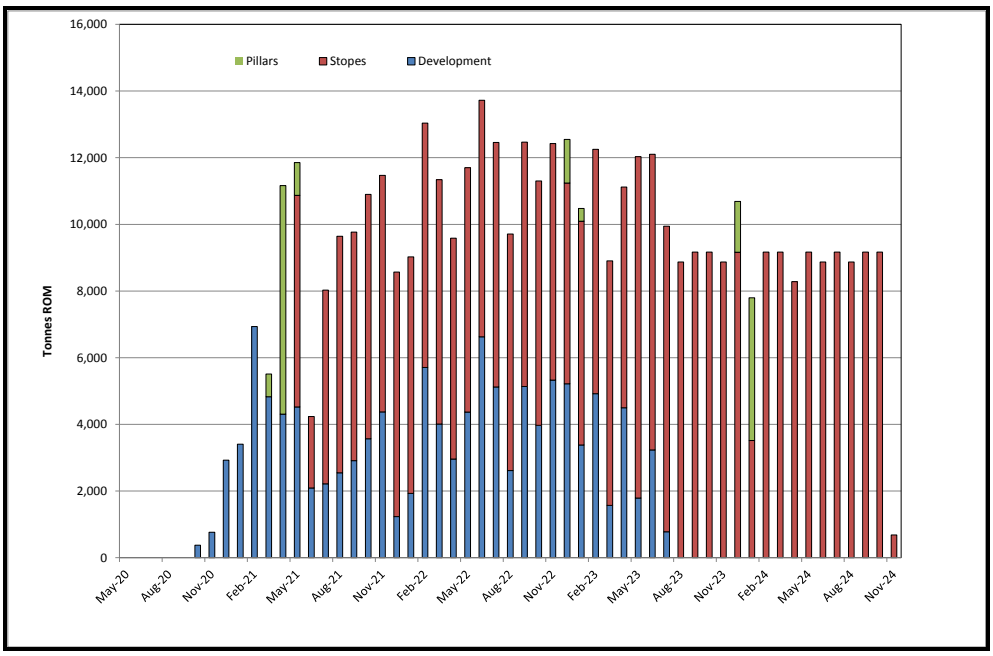


Figure 16.60 Silver Profile (Monthly)

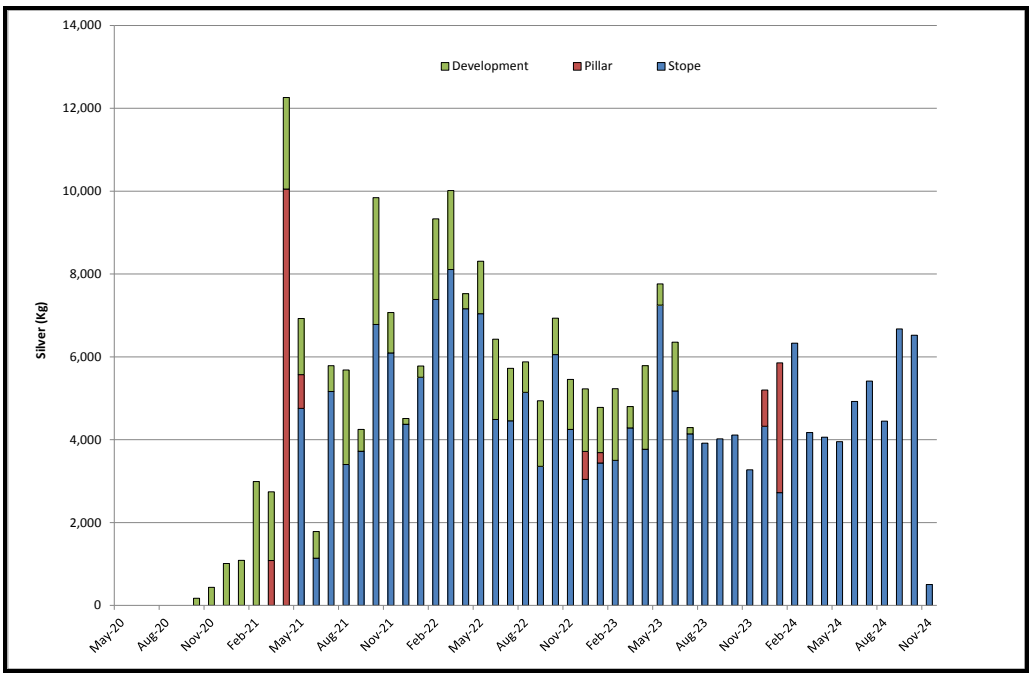
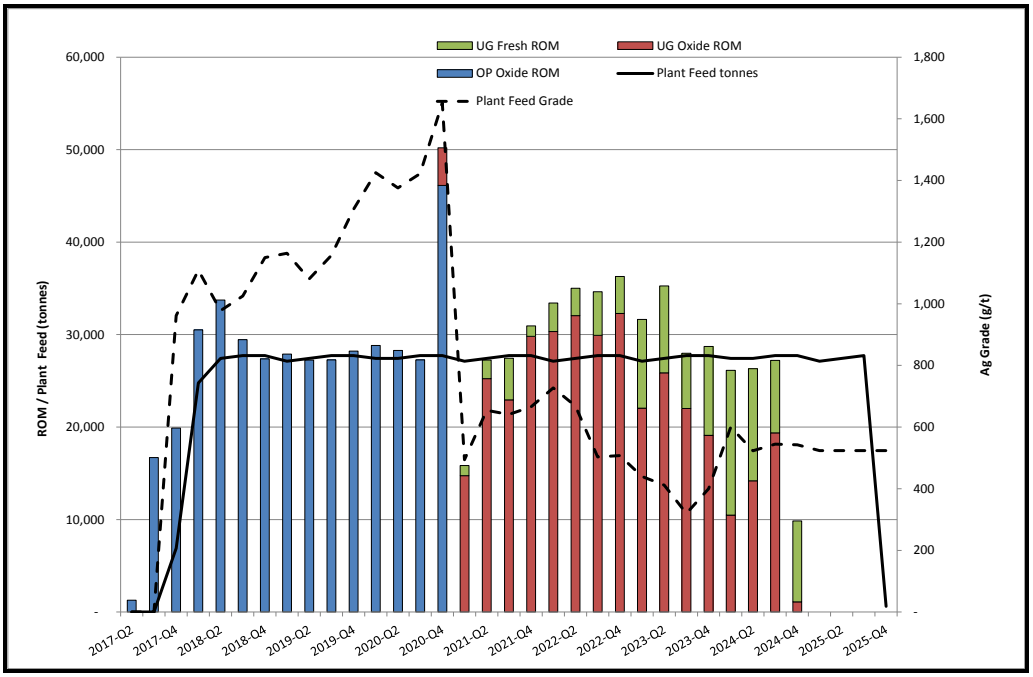


Figure 16.61 shows the combined mine production schedule. Note that the amount of fresh mineralisation increases to the end of the mine life.

Figure 16.61 Combined Open Pit and Underground Schedule (Monthly)



16.2.9 UNDERGROUND MINING EQUIPMENT

The mine design and excavation sizes are dependent upon the equipment selection. The starting point for any mine design is an understanding the production capacity requirements. From the production capacities, the typical loader and truck sizes are calculated. The number of active mining areas then dictate the amount of drilling that would be required to sustain the production capacity requirements, but also to enable the pre-development of areas for contingency measures.

The underground mining equipment for Vertikalny is shown in Table 16.12.

Table 16.12 Equipment Units Simulated

Underground Units Operating	Maximum No. of Units
Jumbo – Single Boom Drill Rig	1
Charger – Small Explosive Charger Cars	2
Develop LHD – 3.7 t	1
Production LHD – 3.7 t	1
Develop Truck –17 t	1
Production Truck – 17 t	1
Shotcrete – Machine With Pump	1
Scissors Lift/Mechanical Lift	1
Long Hole Drill Rig	1
Small Motor Grader -Small Underground Grader	1
Carrier – People – Small Bus/Carrier	1
Fuel Carrier	1
Service Trucks – Utility/Small Trucks With Equipment Mounted	2
Pick-Up/Light Vehicles	4

16.2.10 UNDERGROUND MINE OPERATIONS

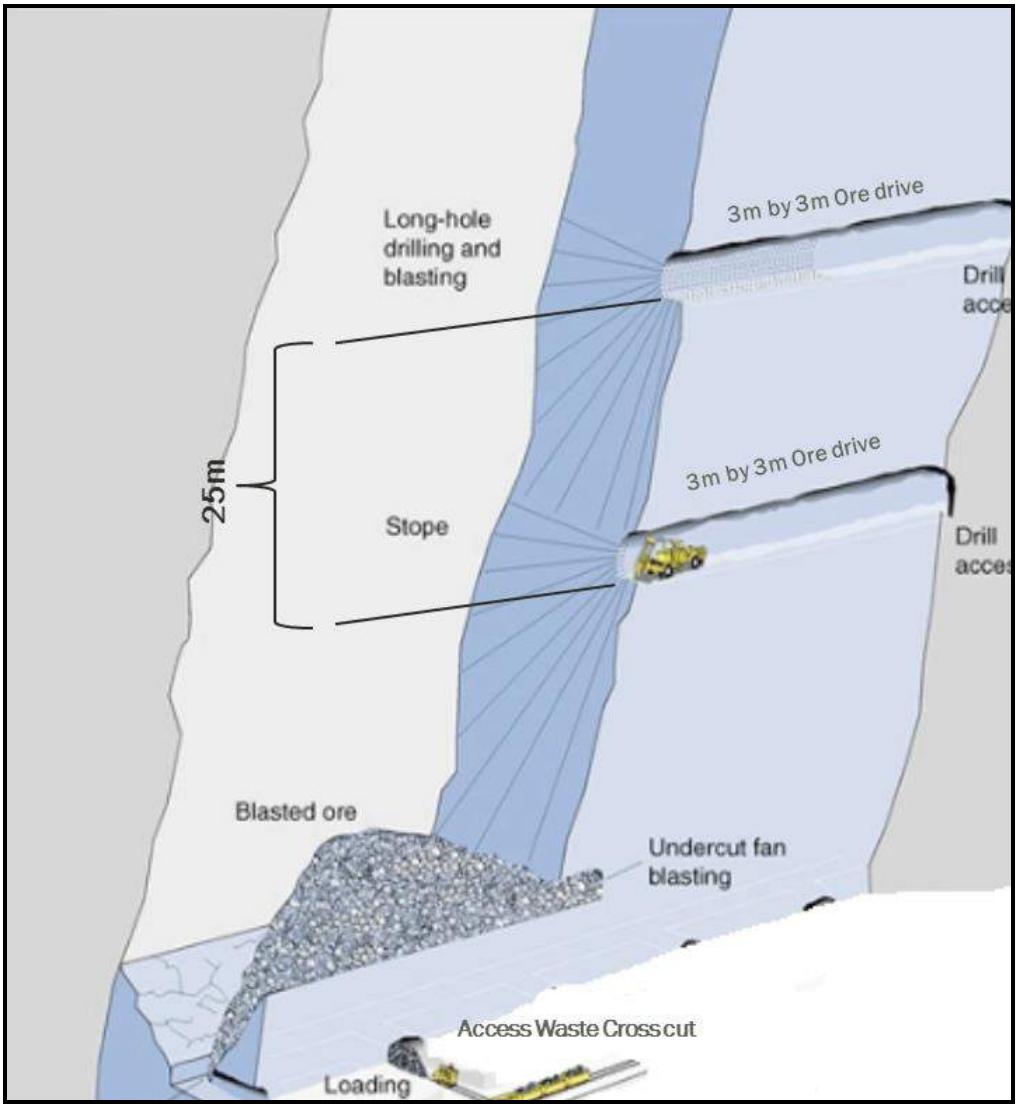
The underground exploitation strategy for Vertikalny is mechanised sub-level open stoping. The entry to the underground mine is gained through a main access decline in the North Zone and a series of in-pit adits in the South Zone. The adits and drives in South Zone is linked with a South Zone decline system.

The stoping method is best illustrated in Figure 16.62. The proposed mining extraction methods are:

- Ore drives minimum 3 m by 3.2 m
- Minimum stope blast width 1.3 m with a single line slightly staggered pattern (single line of 76 mm holes will break out approximately 1.25 m with additional slabbing around 1.3m in very good ground). Stope designed to less than 1.5 m stope widths accounts for 20% of all stope designs. The bottom sections of these stopes still have an ore drive of minimum 3.2 m width.

- 5% mining losses were applied to ore and a further 5% extraction loss to the typical stopes. A 10% extraction losses was applied to stopes near the crown pillar (in situ ore pillar left below the open pit final designs).
- Maximum stope width that can be mined (longitudinally) would be around 10 m before transverse open stoping becomes more practical and cost effective.
- The maximum designed stope width for Vertikalny is 3.9 m.
- The average stope width designed is 1.8 m.
- 17,000 t of Inferred material was included in the stope designs, but the metal from the Inferred material was not included in metal accounting. The Inferred ore tonnes are therefore calculated as part of stope dilution at 0 g/t silver grade.

Figure 16.62 Mechanised Sub-level Open Stoping



16.3 MINE CLOSURE

Spare operating hours for the dozers on site will be used to implement a progressive WRD rehabilitation. This can be achieved shortly after the end of the open phase as the underground mine produces very little waste rock.

The North and South Pits will not be backfilled. Each pit will be surrounded by a 1.5-m-high earth bund. During open pit mining the pits will be kept dry by pumping any water from the pit sumps to the surface collection ponds where it will be treated before discharge or used for processing.

It is expected that after the open pit phase is complete, water will report to the underground workings and discharge to the environment via the adits. The adits will be sealed with steel gates and the water allowed to drain via drain holes and raises to the lowest adit level of the mine. The main collection sump will situated on this level. During operations, water from the mine operations and any inflow from ground water would collect in the sump area. From here water will be topped up with fresh water (if required) and re-circulated to use in the mine as service water. SRK reports that, based on the current understanding of hydrogeology, the operating underground mine is expected to be a marginal consumer of water. Upon closure, the main collection sump would be enlarged to act as a long-term settler that would catch solids before any excess water from the workings enters the environment.

Should the underground mine be depleted and there are no economically feasible reason to continue with underground exploitation of the Vertikalny ore deposit, the following will have to be done during mine closure:

- remove all the ventilation fans (on retreat) from the underground operations
- shut and lock all underground ventilation doors and place no entry signs at each of these doors
- close of all other excavations and remove refuge chamber equipment
- place a permanent fence around the mine openings that prevents people from getting close to the openings and signpost the danger to entry
- clear all steel work and mine equipment and doze and rehabilitate the disturbed areas around the excavations (per the required environmental plan signed-off).

After the pillar recovery there may be some limited surface subsidence above the workings, but this will not be significant due to the narrow orebody geometry.

17.0 RECOVERY METHODS

17.1 SUMMARY

The process design for the Project is based on the test work presented in Section 13.0. The primary objective of the process design was to develop a flowsheet that is as simple to supply, operate, and maintain as possible, whilst maximising silver recovery and minimising power requirements.

The process flowsheet consists of a standard crushing and milling circuit, followed by agitated tank leach of the ground ROM slurry. The leached slurry from the tank leach will go through a simple counter current decantation washing system and the solution will be processed by direct electrowinning to recover silver metal.

The process plant design will have provision for building additional leach tanks in order to process primary sulphide ore in the future.

Tailings from the process plant will comprise filtered solids and a blead stream from the depleted electrowinning solution and will be disposed of in the TMF.

17.2 PROCESS DESIGN CRITERIA

The process design criteria have been derived from the following sources of information:

- test work results, as reported in Section 13.0
- the mill production plan, as described in Section 16.0
- information published in the public domain, industry standard assumptions, and knowledge gained from similar projects and/or unit operations
- equipment manufacturers recommendations
- previous studies completed on the Project.

Table 17.1 lists a summary of the principle process design criteria established for the Project.

Table 17.1 Process Design Criteria

Description	Unit	Value	Source
General			
Type of Deposit:	-	Silver Polymetallic	5
Plant Operating Schedule			
Shifts per Day	shifts	2	1
Hours per Shift	h	12	1
Hours per Day	h	24	6
Days per Year	d	365	6
Weeks per Year	wk	52	6
Operational Days per Year (Milling)	d	336	1
Availability/Utilisation			
Crushing Plant Utilisation	%	75	7
Crushing Plant Availability	%	85	6
Overall Annual Processing Rate	t/a (dry)	110,000	9
Daily Processing Rate	t/d	402	2
Operating Processing Rate	t/h	39	2
Milling Plant Utilisation	%	92	7
Milling Plant Availability	%	91	6
Overall Annual Processing Rate	t/a (dry)	110,000	9
Daily Processing Rate	t/d	327	2
Operating Processing Rate	t/h	15	3
Design Processing Rate	t/h	20	2
Ore Characteristics – Design			
Ore Specific Gravity	SG	2.50 – 2.65	4
Ore Bulk Density	t/m³	1.60	6
Ore Moisture Content	%	5	6
Bond Abrasion Index (A_i) (Average)	g/kWh	0.039	4
Bond Crushability Work Index (CW_i)	kWh/t	7.7	4
Bond Ball Mill Work Index (BW_i)	kWh/t	14.6	4
Head Grade (LOM)	g/t	772	9
Metallurgical Recovery			
Leach Dissolution	%	79.7	4
Overall Silver Dissolution	%	79.7	2
Electrowinning Efficiency	%	99.0	4
Silver Recovery (Oxides)	%	78.1	2
Silver Recovery (Primary)	%	69.6	2
Crushing			
Average Tonnage Rate (Operating Total ROM Feed)	t/h	39	3
Average Crusher Tonnage Rate (Design)	t/h	45	2,3
Average Crusher Volumetric Rate (Design)	m³/h	17	2
ROM Feed F_{80}	mm	200	6
Operating Shifts per Day	shift/d	1	1
Operating Hours per Shift	h/shift	12	1
Operating Days per Week	d	6	6

table continues...

Description	Unit	Value	Source
Primary Crusher Type	-	Jaw (Metso C80 - Equivalent)	2,3,7
Number of Crushers	-	1	2,3,7
Crusher Feed Particle Size F_{95}	mm	300	6
Crusher Feed Particle Size F_{80}	mm	200	6
Crusher Product Particle Size P_{80}	mm	50	6
Reduction Ratio	-	4.00	2
Closed Side Setting	mm	40	6
Secondary Crushing			
Secondary Screen	type	Horizontal Vibrating	6
Area	m^2	1.4	2
Number Needed	-	1	2
Motor Size	kW	5	8
Deck Aperture Size	mm	20	6
Secondary Crusher	type	Fine Cone (Metso GP 100 - Equivalent)	6
Motor Size	kW	90	8
Product Size (P_{80})	mm	13	6
Closed Side Setting	mm	13	8
Stroke	mm	16	8
Fine Ore Bin			
Number of Discharge Conveyors	-	1	2,3,7
Product Size on Discharge Conveyor (Design) P_{80}	mm	13	2
Number of Feeders	-	1	2,3
Type of Feeders	-	Vibrating	2,3,7
Average Tonnage Rate per Stream (Operating)	t/h	15	3
Design Tonnage Rate	t/h	20	2,3,7
Grinding			
Primary Mill Type	-	Wet Overflow Ball Mill	5
Mill Size (diameter x length)	m x m	2.8 x 4.9	2,8
Motor Size	kW	475	2,8
Number of Mills	-	1	2
Bond W_i – Ball Mill (Design)	kWh/t	14.6	2,4
Average Fresh Feed Tonnage Rate	t/h	15	3
Solids/Pulp Density	%	65	6
Top Feed Size	µm	18,000	6,7
Feed Size (F_{80})	µm	13,000	2,6,7
Product Size (P_{80}) – Oxide ore	µm	75	2,4
Product Size (P_{80}) - Primary ore	µm	25	2,4
Ball Mill Recycle Load	%	250	3,6
Mill Speed	% CS	70-75	7
Mill Ball Charge	% VB	30-35	8
Classification Cyclone			
Number of Cyclone (operating)	-	1	2,8
Number of Cyclone (standby)	-	1	6,8

table continues...

Description	Unit	Value	Source
Number of Cyclones (total)	-	2	2,6,8
Cyclone Size Diameter	mm	508	2,8
Inlet Diameter	mm	150	8
Vortex Finder	mm	150	8
Apex	mm	76	8
Product Size P_{80}	µm	75	2,8
Dewatering Cyclones			
Number of Cyclone (Operating)	-	1	2,8
Number of Cyclone (Standby)	-	1	6,8
Number of Cyclones (Total)	-	2	2,6,8
Cyclone Size Diameter	mm	508	2,8
Inlet Diameter	mm	150	8
Vortex Finder	mm	100	8
Apex	mm	38	8
Overflow Pulp Density (nominal)	% solids	0	6
Underflow Pulp Density	% solids	70	6
Leach Circuit			
Solids Feed Rate	t/h	15	3
Number of Parallel Trains	no.	1	6
Feed Rate per Train	t/h	15	2
Pulp Density	% solids	45	2,3
Nominal Flow Rate	m^3/h	24	2,3
Processing Method	-	Agitated Tank leach	5
Leach Tanks			
Type	-	Mechanical Agitation	6
Number of Tanks per Train	-	6	2
Operating Volume (each)	m^3	313	2
Nominal Resident Time (total)	h	72	4
Nominal Resident Time (each)	h	12	2
Tank Dimensions	m x m	7.35 X 7.35	2
Impeller Type	-	UUtecknik Twin Blade PSH55/33	8
Shaft Length	m	6.25	8
Impeller Diameter	m	2.95	8
Motor Power	kW	55	2,8
Leach Air Requirement	l/min	371	2,7
Washing Circuit			
Dewatering Cyclones			
Feed Rate	t/h	15	3
Number of Cyclone (operating)	-	1	2,8
Number of Cyclone (standby)	-	1	2,6,8
Number of Cyclones (total)	-	2	2,8
Cyclone Size Diameter	mm	508	2,8
Inlet Diameter	mm	150	2,8
Vortex Finder	mm	100	2,8

table continues...

Description	Unit	Value	Source
Apex	mm	38	2,8
Overflow Pulp Density (nominal)	% Solids	5	2,8
Underflow Pulp density	% Solids	60	2,8
Clarifiers			
Clarifier Type	-	Lamella	2
Number of Thickeners	-	1	2,8
Clarifier Underflow Solid Density	%	40-50	2,8
Overflow Pulp Density (nominal)	ppm	100	8
Slurry Transportation Method	-	Pumping	2,8
Flocculent Consumption	g/t	25	2,4
Flocculent Solution Strength	g/L	0.5	6
Flocculent Plant Capacity	kg/h	50	6
Flocculent Holding Tank Capacity	h	2.00	6
Filters			
Filter Type	-	Plate & Frame (Recessed Type)	8
Size	m x m	2 X 2	8
Number of Chambers	-	60.00	8
Filter Cake Moisture Content	%	15.00	6
Electrowinning			
Primary Electrowinning	type	EMEW Powder cell	8
Number of Cells	-	140	2,8
Cell Diameter	mm	200	8
Operating Schedule	h/d	24	6
Type of Cathode	-	316SS	7,8
Type of Anode	-	316SS	7,8
Cathode Sludge Removal Method	-	Pressure Back Wash	7,8
Secondary Electrowinning	type	EMEW Polishing Cell	8
Number of Cells	-	80	2,8
Cell Diameter	mm	200	8
Operating Schedule	h/d	24	6
Type of Cathode	-	316SS	7,8
Type of Anode	-	316SS	7,8
Precipitate Filtration			
Filter Type	-	Plate and Frame, Pneumatic/Hydraulic Closure	7,8
Capacity	L	85	2,8
Operating Hours per Day	d	6	6
Filter Cake Moisture	%	20	6
Drying			
Type		Electric Oven	-
Drying Temperature	C	105-120	-
Operating schedule		Daily	-
Final Product Moisture Content	%	2- 4	-

17.3 AVAILABILITY AND UTILISATION

The crushing plant was designed based on 274 operational days per year, to allow 2 days per month of planned shut down for scheduled maintenance and housekeeping, and approximately 2 months of shut down for extreme winter. The crushing plant is expected to have an availability of 75%, with 85% utilisation, in order to give an effective utilisation of 64%.

The milling and leaching plants were designed based on 335 operational days per year, to allow 2.5 days per month of planned shutdown, and to provide 92% availability. The plant utilisation is expected to be 91%, in order to give an 84% effective utilisation.

The ore stockpile was designed with sufficient surge capacity to provide an uninterrupted supply to the mill in order to maintain a high utilisation during the crusher shut down periods.

17.4 LOCATION AND LAYOUT

The proposed process plant will be located on the top of the hill approximately 1,250 masl and northeast of the open pit mining area. The ROM pad, crushers, and conveyers will be located southwest of the process plant to facilitate material handling and crushing from the mining area.

The TMF will be located southeast of the process plant and will handle the dewatered tails from the process plant.

The electrowinning section, office buildings, and metallurgical laboratory will be located within the process plant premises and near the plant access road.

The plant access road will connect the process plant on the top of the hill to the camp facilities located in the valley bottom.

17.5 PRIMARY EQUIPMENT LIST

The primary equipment list (Table 17.2) was prepared based on the process design criteria presented in Table 17.1.

Table 17.2 Primary Equipment List

Plant Major Equipment	No. of Units	Installed Power/Unit (kW)	Total Installed Power (kW)
Vibrating Grizzly	1	12	12
Vibrating Feeder	1	12	12
Primary Crusher	1	148	148
Crusher Screen Feed Conveyor	1	TBD	0
Secondary Crusher Screen	1	12	12
Secondary Crusher Feed Conveyor	1	TBD	0
Secondary Crusher	1	120	120
Secondary Crusher Discharge Conveyor	1	TBD	0
Fine Ore Bin Feed Conveyor	1	18	18
Electric Overhead Travelling Crane	1	18	18
Fine Ore Bin	1	0	0
Fine Ore Bin Vibrating Feeder	1	12	12
Ball Mill Feed Conveyor	1	12	12
Ball Mill Feed Chute	1	0	0
Ball Mill)	1	475	475
Ball Mill Scats Bin	3	0	0
Ball Mill Ball Hopper	1	0	0
Ball Mill Liner Handler	1	1	1
Mill Area Sump Pump	2	15	30
Mill Area Sump Pump	0	15	0
Mill Discharge Pump Box	1	0	0
Mill Classifying Cyclone Feed Pump	2	20	40
Mill Classifying Cyclone Feed Pump	0	20	0
Mill Classifying Cyclone	1	0	0
Mill Classifying Cyclone Overflow Tank	1	0	0
Mill Classifying Cyclone Overflow Tank Agitator	1	12	12
Mill Dewatering Cyclone Feed Pump	2	18	35
Mill Dewatering Cyclone	1	0	0
Leach Tank 1	6	0	0
Leach Tank Agitator 1	6	60	360
Leach Blowers	1	15	15
Leach Air Receiver	1	0	0
Leach Discharge Pump Box	1	0	0
Leach Discharge Pump	2	15	30
Leach Area Sump Pump	4	15	59
Lamella Thickener	1	0	0
Lamella Thickener	1	12	12
Lamella Thickener	1	0	0
Clarifier Under Flow Pump	2	12	24
Leach Dewatering Cyclone	1	0	0
Filter Press Charge Tank	1	0	0
Filter Press Charge Tank Agitator	1	12	12
Filter Press Charge Pump	2	12	24

table continues...

Plant Major Equipment	No. of Units	Installed Power/Unit (kW)	Total Installed Power (kW)
Clarifier Over Flow Pump Box	1	0	0
Clarifier Over Flow Pump	2	12	24
Filter Press	2	12	24
Filter Cake Discharge Conveyor	1	12	12
Electrowinning Package Polishing Cells	1	63	63
Electrowinning Package Powder Cells	1	263	263
Sludge Drying Package	1	12	12
Sump Pump	-	15	0
Lime Package	1	12	12
Caustic Package	1	12	12
Flocculant Package	1	4	4
Cyanide Package	1	12	12
Sump Pump	1	15	15
Process Water Tank	1	0	0
Total Installed Power (kW)	-	-	**1,944**

17.6 PROCESS DESCRIPTION

17.6.1 PROCESS DESCRIPTION OVERVIEW

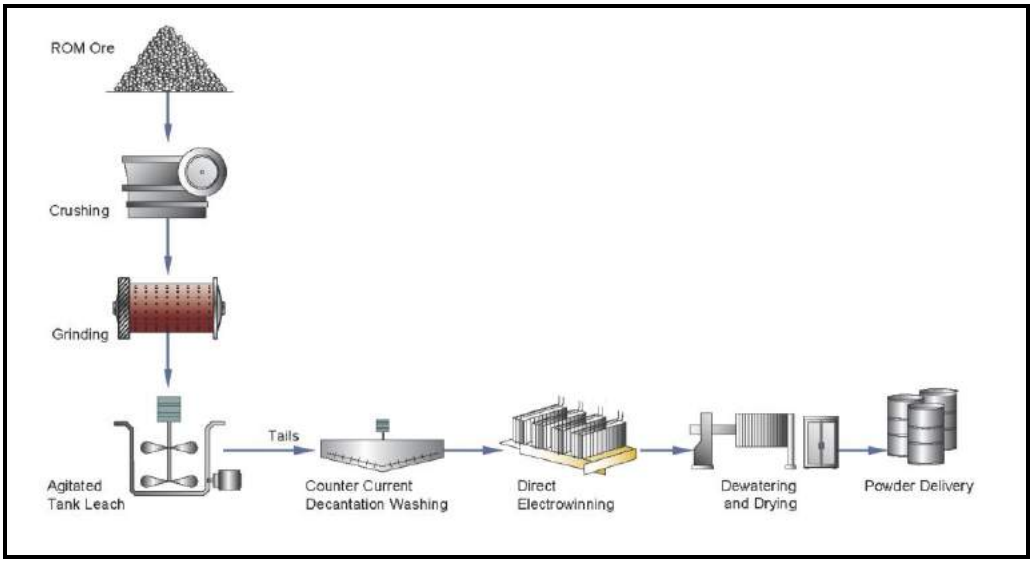
A standard crushing and milling circuit will be employed to produce a suitable feed for cyanide leaching at a size of typically 80% minus 75 µm.

The ground slurry will be cyanide leached to dissolve the silver into a high-grade leach solution. Conventionally, with such high-grade silver ore, a Merrill Crowe process would then be used to recover silver from the solution. The unit processes involved would typically include counter-current decantation (CCD) or filtration for recovery of the silver-bearing solution from the solids, followed by de-aeration, clarification and recovery of the silver using zinc powder, followed by acid digestion, filtration, and smelting. The Merrill Crowe process is both costly and complex to operate and maintain and is not considered ideal for the Project.

Due to exceptionally high silver grades, and the requirement to avoid the Merrill Crowe process, the proprietary direct electrowinning process was selected as the appropriate technology. In this technique, the high-grade, silver-rich solution from the leaching process is passed directly to specially constructed electrowinning cells for direct recovery of the silver onto cathodes that can be washed periodically, dewatered dried and drummed for shipment. This considerably reduces the capital cost, is easier to operate and maintain, and should lend itself to a compact modular design.

The dry solids tailings produced from the selected flowsheet will be dry stacked into the TMF. A block flow diagram of the process plant is shown in Figure 17.1.

Figure 17.1 Block Flow Diagram of the Proposed Process Plant



Source: Tetra Tech

17.6.2 CRUSHING

ROM mineral with a maximum size of approximately 300 mm and 80% passing 200 mm will be trucked to the ROM pad, and the material will feed into the crushing and screening circuit. The crushing and screening unit will be arranged for closed circuit crushing. The crushing circuit will operate over a single 12-hour shift to allow for maximum efficient running of the circuit crushers, which have been sized as per the standard crusher supply.

The ROM mineral will feed to a vibrating grizzly where the fine ore will be removed and passed directly to the secondary crushing circuit screen. The coarse fraction will be crushed in the primary jaw crusher, such as a Metso C80 or equivalent. The product from the jaw crusher will be sent to the secondary crushing circuit screen.

The secondary crushing circuit will contain a secondary cone crusher, such as Metso GP100 or equivalent, operating in closed circuit with a single deck screen to produce a final screen product with a P_{80} of 13 mm, suitable for the ball mill feed. The screen undersize from the secondary crushing circuit will be conveyed to a crushed ore bin where it will be reclaimed and fed to the processing plant as required.

The crushing circuit is designed to operate at 75% availability to allow for operational and mechanical maintenance and winter shut down. Spare sets of jaw and cone crusher liners will be kept on site.

A process flow diagram of the crushing circuit can be seen in the Appendix D.

17.6.3 MILLING

MILLING

The milling and processing plant will operate for 24 h/d at an availability of 91% and process 327 t/d of mineral at an average silver head grade of 772 g/t.

The crushed material will feed directly onto the ball mill via a feed conveyor belt and feed hopper. The conveyor belt will be equipped with a weightometer and a sampler from which the primary accounting information will be obtained (i.e. the throughput, silver head grade, and moisture content).

A single ball mill will be used to achieve the required grind size for effective gravity concentration/flotation and cyanide leaching. The ball mill will be a standard open discharge mill operating in closed circuit with two stages of hydrocyclones in order to produce a ground product suitable for the downstream gravity concentration and leaching process.

The estimated size of the ball mill is 2.8 by 4.9 m installed with a 475 kW motor. Steel ball grinding media will be added to the ball mill feed using a wheeled crane and kibble, with the balls added to the mill feed hopper. Water will be added to the mill feed inlet to achieve an operating mill pulp density of approximately 60 to 65% solids by weight.

The mill product slurry will be pumped to a two-stage cyclone system, where the primary cyclone will function as a classifier with a separation of size of P_{80} = 75 µm for oxide ore and P_{80} = 25 µm for primary ore. The underflow from the primary cyclone will be recirculated back to the ball mill with a design recirculating load of 250%. The overflow from the primary cyclone will be densified at the second cyclone to approximately 45 to 50% solids, and gravitate to the tank leach circuit during the normal operation. The overflow from the densifying cyclone will be pumped to the process water tank. It is anticipated that the solids content of this overflow will nominally be zero.

It is important to note additional milling capacity may be required in the future when the primary ore is processed through the plant,

17.6.4 LEACHING

Leaching of the ROM slurry will be completed in agitated leach tanks, designed to give a total residence time of approximately 72 hours. However, provision will be made to construct additional tanks to increase the residence time to 120 hours in the future, when the primary ore is processed.

The leach tanks will be constructed on progressively lower elevations to allow the flow of slurry from tank to tank using gravity. The leach tanks will be constructed of mild steel, with an active volume per tank of approximately 312 m^3 with an agitator installed. Each leach tank will be equipped with a gas sparger for injection of air into the slurry in order to maintain the required oxygen level in the solution for acceptable process kinetics.

The leach tanks will be constructed with manually operated valves in order to direct the flow through a by-pass line traversing the tank when required.

At the end of the leaching, slurry from the last leach tank will be pumped to the dewatering circuit in order to separate the solids and solution for direct electrowinning.

Lime slurry and cyanide solution will be added to the leach tanks to maintain a pH of approximately 11.5 and free cyanide concentration at least 2, 000 ppm throughout the leaching. The lime slurry and cyanide solution will be prepared in the reagent preparation area and pumped into a ring main, where additions can be made to the leach tanks as required.

The process flow diagram of the leach circuit can be seen in Appendix D.

17.6.5 DEWATERING, FILTRATION AND CLARIFICATION

Solid-liquid separation of the leach product will be achieved using a combination of a dewatering hydrocyclone, a plate and frame filter press, and a high-rate clarifier. The slurry from the last leach tank will be pumped through a dewatering hydrocyclone. The hydrocyclone overflow will be further clarified in a high-rate clarifier in order to provide a suitable overflow solution for the direct electrowinning process.

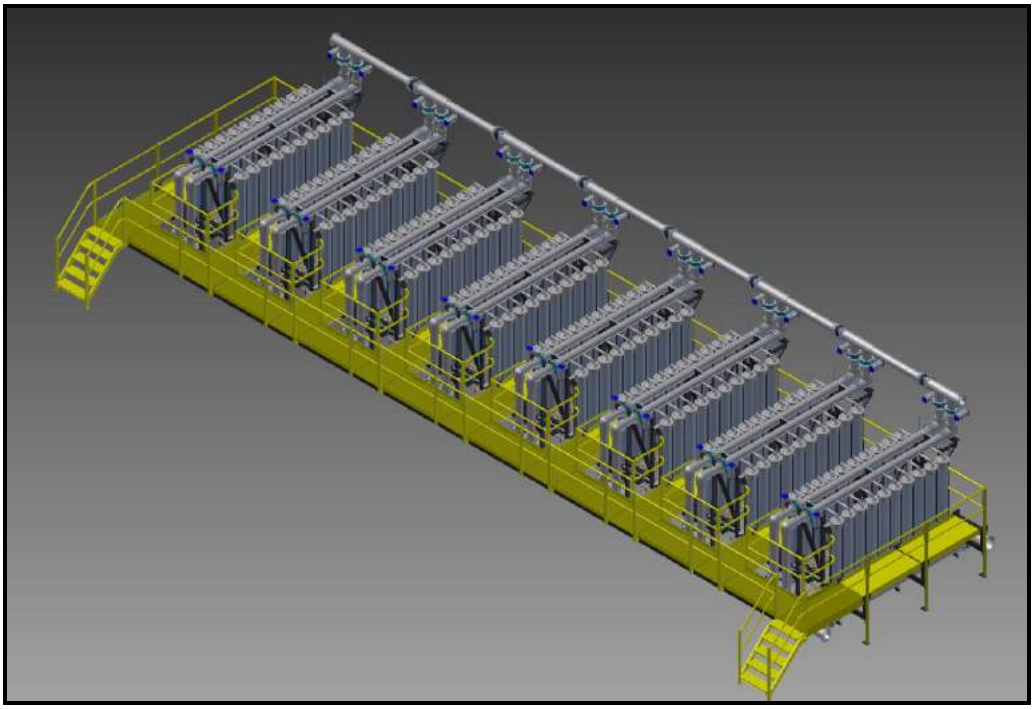
The hydrocyclone underflow will pass into an agitated filter feed tank. From there, the slurry will be pumped to a plate and frame filter press. A buffer tank was installed prior to the filter press to allow for the batch operation of the unit. The filter press cycle will include a wash cycle to remove any silver containing solution from within the leached solid tailings. The wash water will also remove any cyanide that is associated with the filtered solids. The filter press filtrate will feed into the clarifier, with the filtrate produced during the filter cake wash returning to the water recycle circuit for re-use within the plant. The solid product from the filter press will be conveyed to the TMF.

The process flow diagram of the dewatering circuit can be seen in Appendix D.

17.6.6 DIRECT ELECTROWINNING

The direct electrowinning process will employ patented emew® cell technology (Figure 17.2) to recover silver, which can be directly smelted to produce doré silver bars or alternatively the precipitate can be shipped directly to the refinery. The direct electrowinning circuit will be operated in two stages, with both stages consisting of emew® cells.

Figure 17.2 3D Model of an emew® Cell Technology



The primary stage, consisting of 140 emew® powder cells, each 200 mm in diameter will reduce the silver solution from approximately 800 ppm to 50 ppm silver. The secondary stage, consisting of 80 emew® polishing cells, each 200 mm in diameter will reduce the solution to below 10 ppm silver prior to discharge. The entire direct electrowinning plant will be supplied as a modular turnkey package plant by Electrometals.

The individual emew® cells will be constructed from a pair of tubular electrodes and the ends fitted with plastic end caps, allowing high flow and efficient mixing through the closed chamber formed. The capacity of each cell will depend on the silver concentration feeding to the electrowinning circuit; the more diluted the feed, the larger the electrowinning circuit required.

The inside surface of the pipe is the cathode and will be constructed of stainless steel. The anode, which will also be of stainless steel construction, runs through the centre of each cell. During operation, high-grade silver will be deposited on the cathode surface (Figure 17.3) and automatically flushed from the cell by a back wash technique and collected in a tank for filtration and drying before being packed into drums for shipping into a suitable refinery.

Figure 17.3 Silver Powder Deposition on Cathode Surface

Barren solution from the electrowinning process will return to the process water tank and subsequently used as dilution water for mill feed, or used for slurry density control purposes in the leach circuit. It is anticipated that approximately 1% of the barren solution will be discharged following cyanide detoxification in order to avoid a build-up of base metal and deleterious elements within the recycle stream.

The process flow diagram of the direct electrowinning circuit can be seen in Appendix D.

17.7 PLANT SERVICES AND REAGENTS

Typical plant services include an air compressor to supply air to the agitated leach tanks for effective silver dissolution and for instrument air. It is expected that plant instrumentation will be minimal.

17.7.1 WATER

Fresh water will be supplied to a dedicated process water tank and process water pumps will deliver water as required to the plant. Barren solution will be recycled as part of the process to prevent excess fresh water requirements; however, this will be limited to ensure that there is no build-up of deleterious elements within the slurry streams. Approximately 1% of the barren electrowinning solution will be sent to a detoxification circuit for cyanide detoxification and disposal.

17.7.2 CYANIDE AND LIME

The principal reagents are cyanide and lime. Lime will be delivered to site in 1-t bulk bags as hydrated lime powder. The lime will be mixed in a dedicated agitated mixing tank, typically with a capacity for one day's consumption, and transferred to a storage tank where water will be added and the resulting lime slurry pumped into a ring main system delivering the lime to the leaching circuit to maintain the pH at 11.5 as required. The lime bags will be added to the mixing tank using a hoist and bag-

breaker installed above the tank. Lime bags will be stored in a dedicated covered storage area.

Cyanide will be delivered to site as 1-t bulk bags containing sodium cyanide briquettes. The briquettes will be added to an agitated cyanide mixing tank, with the required volume of water, by hoisting the bag onto a bag-breaker situated above the tank and enclosed in a steel chute for dust control. The cyanide solution will be made up to approximately 25% concentration.

A split tank system can be used, whereby two-thirds of the tank would be dedicated to cyanide mixing, and the remaining third used as storage, from which the cyanide solution is pumped via the ring main to the leaching circuit as required to maintain an initial cyanide concentration of approximately 500 ppm in the first leach tank. A bunded concrete spillage area with a dedicated sump pump is required.

Cyanide bags are normally stored in containers with access for a forklift.

A dedicated wheeled crane is required for general plant use, steel ball addition, maintenance, and reagent preparation together with a dedicated fork lift.

17.7.3 Power and Mobile Equipment

Electricity for the process plant will be supplied from the on-site power plant.

One dedicated wheeled crane will be required for general plant use, maintenance, and reagent preparation, together with the additional mobile plant equipment listed in the primary equipment list (Table 17.2).

17.7.4 Laboratory

A fully equipped laboratory will be available on site and will have the following separate areas:

- a sample preparation sector with a space/bench area for sample receiving, drying ovens, size reduction equipment, and adequate bench space for the preparation of mine and geological samples
- an assay laboratory with separate sample preparation and storage areas for samples sourced from different zones of the orebody in order to avoid cross contamination; the laboratory will contain fire assay equipment, scale room, chemical laboratory analysis, and chemical storage
- a metallurgical laboratory including pressure filters, grinding simulation equipment, leaching test equipment, and other miscellaneous metallurgical laboratory equipment as required.

17.8 Control Philosophy

The control philosophy for the process plant was developed to achieve the following objectives.

- metallurgical control/silver accounting across the plant

- control of the process equipment for optimal performance.

17.8.1 METALLURGICAL CONTROL/SILVER ACCOUNTING

Metallurgical control of the process plant will be achieved by sampling of the head, tail, and intermediate streams at appropriate locations. A summary of the proposed sampling locations and methods are provided in the following sub-sections.

MILLING CIRCUIT

The amount of silver input to the milling circuit will be calculated from the mass of ore, as measured by an electromechanical belt weightometer, and the feed head grade will be determined by periodic stop belt sampling of the ball mill feed conveyor.

The amount of silver output from the milling circuit will be calculated from the pulp density, flow rate, and silver grade of the cyclone overflow, determined by periodic sampling and assaying of the cyclone overflow.

LEACH AND DEWATERING CIRCUIT

The amount of silver input and output from the leach and dewatering circuits will be calculated from the pulp density, flow rate, and silver grade (solids and solution assays) measurements at input, output and appropriate intermediate streams. These parameters will be measured in a similar way as described in cyclone overflow sampling.

ELECTROWINNING CIRCUIT

The pregnant solution entering the electrowining circuit and the depleted solution leaving the electrowinning circuit will be sampled periodically, along with the electrowinning powder discharged, in order to determine the silver entering and exiting the circuit. In-line monitors will record the solution flow rates of these streams. From these results the silver recovered from the electrowinning circuit will be determined.

17.8.2 PROCESS EQUIPMENT CONTROL

The process equipment control philosophy has been developed to achieve the following objectives:

- maximize the mill throughput and stabilize the control of product particle size
- provide better control of the grinding circuits by control of the cyclone feed density
- control of mill dewatering cyclone underflow density to maintain leach efficiency
- monitor and control the electrowinning circuit to increase the Electrowinning efficiency and reduce soluble silver losses.

A detailed explanation of process equipment control can be found in Section 18.0.

18.0 PROJECT INFRASTRUCTURE

18.1 OVERVIEW

The Project will require the development of a number of infrastructure components. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and to ensure efficient and convenient operation of the mine haul fleet.

Project infrastructure will include:

- airstrip
- off-site, on-site, and service road access
- mine haul roads
- TMF
- WRD and overburden dump
- ROM pad and crushed ore stockpile
- camp flood protection berms
- borehole water supply and distribution system
- water treatment and distribution
- sewage collection and management
- surface water management
- process and ancillary facilities, including:
  - process plant
  - crushing facility
  - reagent storage and handling
  - facilities for administration and an assay laboratory
  - heavy and light vehicle workshops
  - fuel storage and distribution
  - explosives storage and handling
  - emergency helipad
  - camp accommodation
  - mess and laundry facilities
  - medical centre
- power supply and local distribution network
- communications infrastructure.

18.2 LOGISTICAL INFRASTRUCTURE

The remote location of the mine site and the seasonal availability of the main mine access route prioritises the logistical infrastructure in order to keep the mine supplied with goods and personnel for uninterrupted operation. Supplies and equipment will be consolidated in Yakutsk and the regional ports, before being transported to site during the three to four months that the winter road is open. For nine months of the year, air travel is the only reliable transport to site. This poses a logistical issue, as the size of airplane permitted does not have the capacity to carry bulk supplies. Therefore, a minimum nine-month supply of goods, spares, and consumables needs to be transported to site during the limited three to four month window that the access road is open. Enough warehousing and storage also needs to be provided throughout the supply chain and on site to accommodate a nine-month supply.

18.2.1 PORTS

The Batamai Port on the Lena River is located approximately 370 km south of the Project area and can be accessed via the winter road. The port can accommodate boats and barges of up to 3,300 t, and it has a ferry link to the southern shore of the Lena River, which is navigable from mid-April until late September. The ferry has a 20-t capacity.

As part of the logistics supply line, a Batamai Port upgrade to provide a fuel reloading facility, by means of 500-m^3 plastic tanks, and provision of an open laydown area as a summer storage and holding facility, will enable containerised consumables such as reagents, along with fuel, to be stockpiled during the summer months for haulage along the winter road. This will reduce the current haulage distance of 570 km to 370 km, and reduce turnaround time from approximately six days to three days. The Batamai Port is located on the northern bank of the Lena River; trucks bringing goods to site from the Batamai Port will not have to wait for the Lena River to freeze, which extends the transport window.

18.2.2 RAIL

Yakutsk is served by the Amur-Yakutsk Mainline, a branch line off the Trans-Siberian Railway from Bamovskaya Station near Skovorodino in Amur Oblast. The rail line provides the opportunity for overland transport of equipment and supplies from the deep sea port of Vladivostok to the east, and from rail links to the Black Sea ports to the west.

There are a number of freight handling companies operating Trans-Siberian rail cars transporting containerised freight, as well as loose bulk, machinery, and perishable items, providing an alternative overland transport route for construction and operational equipment and supplies.

18.2.3 ACCESS ROADS

The area is serviced by a number of public winter roads serving villages on route to site. The closest road in the vicinity of the mine site camp is used by reindeer herders during migration, which will temporarily increase traffic. Extra care on the part of drivers will be required during the migration period and some temporary hold up may

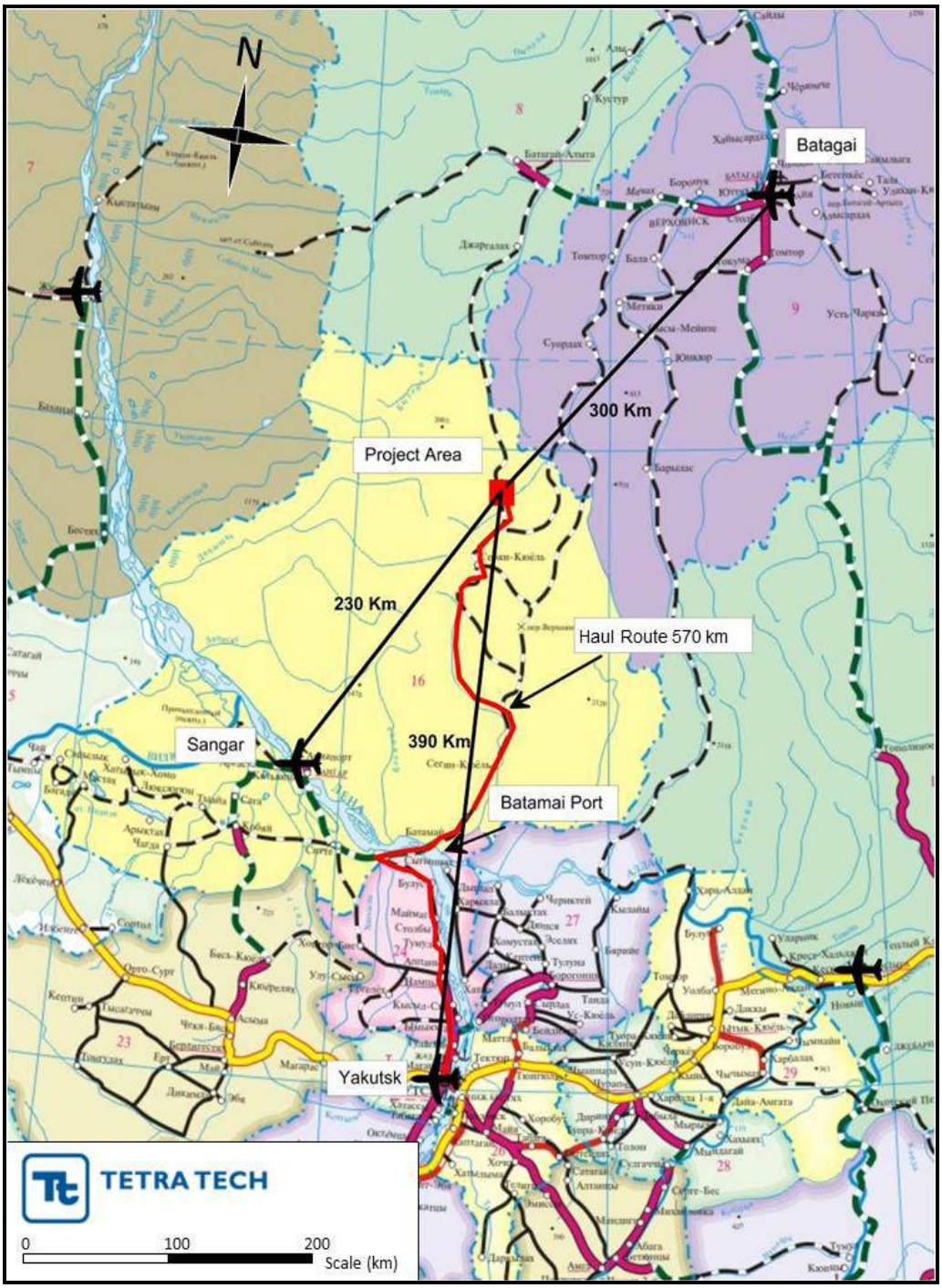
be experienced. The winter roads are usable between mid-December and mid-April (varying from year to year).

The local Sakha authority is responsible for determining when the road is open, and granting permission for the road to be used. Early in the season (20th December) the road opens with a 5-t weight limit and, as the ice thickens, weight limits are raised to 10 t for the official opening on 1st January, and finally 20 t from 15th January to 15th April, although heavier loads can be transported by special arrangement. For general site deliveries, a size limit of 12 m (a 40-ft container) and a load weight limit of 14 t is recommended before special measures are taken.

The current haulage route is shown in Figure 18.1. The total distance by road from Yakutsk to the Property is 570 km. Based on the last five years of moving consumables and heavy equipment to site, a one-way trip takes, on average, two days. The 200-km leg from Yakutsk to Batamai takes eight hours and the remainder of the two days is used to negotiate the more remote section of road between Batamai and the mine site. Generally, the roads are in better condition later in the season and shorter journey times can be expected. Conversely, early season journeys between Yakutsk and the mine site during December and January can take longer than two days. For safety, several trucks will travel in a convoy to ensure assistance is available in the event of a breakdown, accident, or a truck getting stuck.

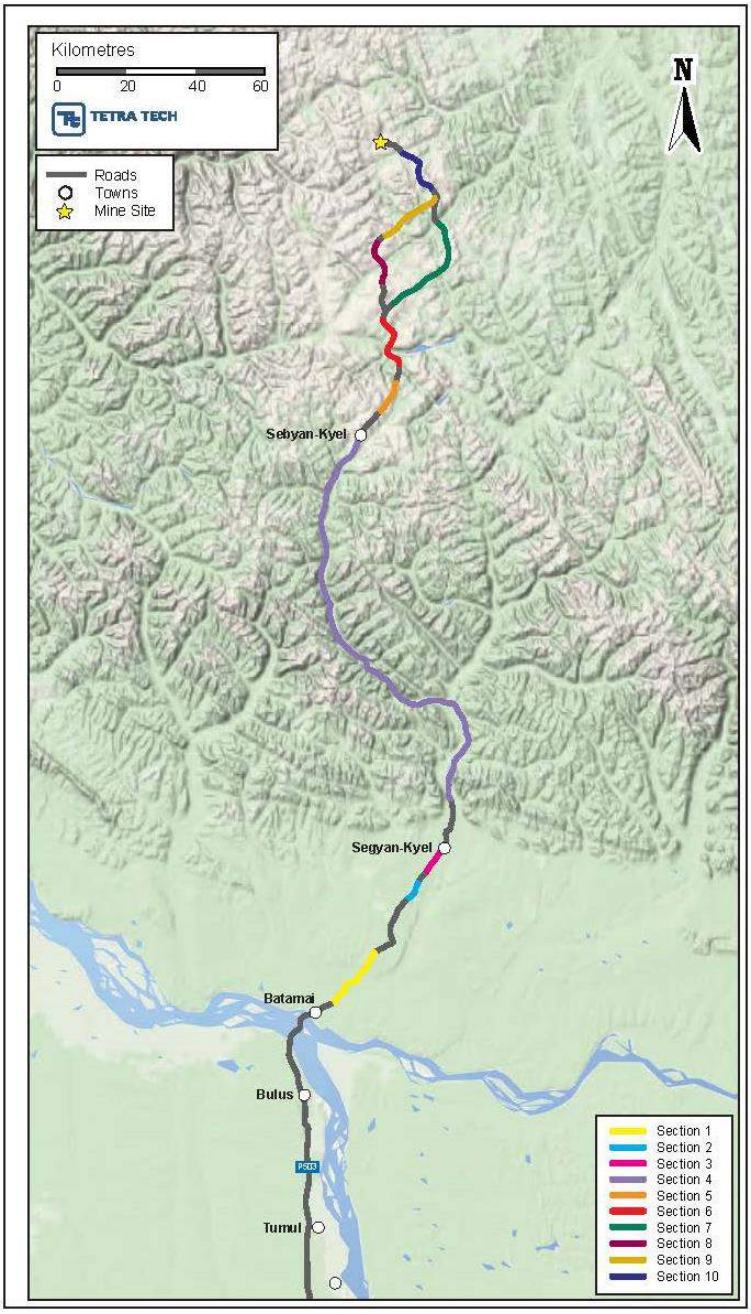
The main obstacles on the route to site are snow, boulders, and fallen trees, which can make driving difficult. Small unfrozen rivers also allow water to flood the ice in some sections between Batamai and site (Figure 18.2).

Figure 18.1 Transport Routes



Note: Red outline = project area; black lines = existing gravel roads

Figure 18.2 Winter Road



Notes:

Road Section	Typical Terrain
1	Narrow forest road with sharp turns
2	Steep mountainous road
3	Swampy road with pigweed
4	Overflow ice, steep river entry and exit.
5	Steep mountainous road
6	Overflow ice, steep river entry and exit.
7	Overflow ice, steep river entry and exit.
8	Overflow ice, steep river entry and exit.
9	Overflow ice, steep river entry and exit.
10	Overflow ice, steep river entry and exit.

Road maintenance is carried out by specialist contractors under the direction of Prognoz, with the maintenance burden increasing in line with road usage. Typically, contractors will maintain the road with a front-end loader, accompanied by a fuel truck, to complete sections of the road without the need to ferry fuel back and forth. Maintenance begins in September to prepare for the official opening of the road on 1st January. Prognoz structure their maintenance around the requirements of truck movements and the burden increases during severe conditions, or after a big flood season. From December to April, a one-way road clearing trip takes approximately two weeks, depending on weather conditions.

Historically, Silver Bear has moved up to 50-t payloads on the winter road for one off special loads, such as a Caterpillar D9 bulldozer. For the 2015/2016 winter road season, the road was upgraded to provide a reliable route for process equipment delivery, in addition to regular bulk deliveries of construction material, spares, and consumables. The tonnages transported to site over the last five years along the winter road are shown in Table 18.1.

Table 18.1 Winter Road Transportation Tonnages

Year	Winter Road Transportation Tonnages (t)
2011	700
2012	2,000
2013	200
2014	200
2015	2,800

The 2015/2016 winter road season transported the normal camp and exploration consumables, equipment, and construction material and consumables. As would be expected during construction, the bulk of the 3,883 t of construction material is made up of structural steel, cement, and rebar, accounting for 3,000 t of material required to be transported. A summary of the winter road transport statistics is provided in Table 18.2, Table 18.3, and Figure 18.3.

Table 18.2 Winter Road Transportation Load Tonnages 2015/2016 by Item

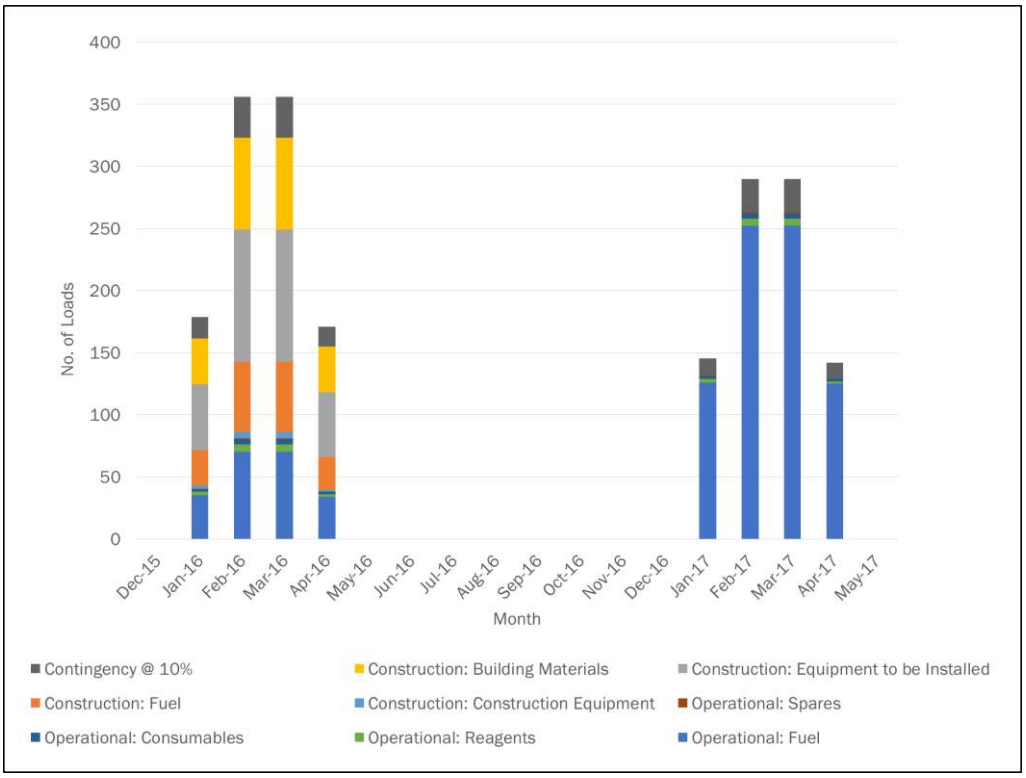
Item	2015/2016 Winter Road Season Loads (t)
Mechanical Equipment	507
Electrical and Control Equipment	310
Piping	63
Valves	3
Structural Steel	1300
Cement and Rebar	1700
Total Construction Materials	**3,883**
Liners	2
Grinding Balls	20
Lime	20
Sodium Cyanide	100
Flocculent	1
Fuel	2,500
Camp and Exploration Consumables	2,800
Total Consumables	**5,443**
Total	**9,326**

During normal operation, fuel deliveries make up the bulk of the loads delivered to site, with typically just over 5,000 t making its way up the winter road to site in a season (Table 18.3).

Table 18.3 Annual Winter Road Transportation Tonnages by Item

Item	Annual Winter Road Tonnages (t)
Mechanical Equipment	515
Electrical and Control Equipment	435
Piping	8
Valves	15
Total Equipment	**973**
Liners	10
Grinding Balls	100
Lime	100
Sodium Cyanide	500
Flocculent	5
Fuel	5,050
Camp and Exploration Consumables	2,800
Total Consumables	**8,565**
Total	**9,538**

Figure 18.3 Estimated Truck Loads to Site 2016/2017



Source: Tetra Tech

Access from Batamai to the north via winter road has been possible historically; however, during the last 20 years the road has had no funding and had run into disrepair. The road could be used in exceptional circumstances.

18.3 SITE LAYOUT

The site layout shown in Figure 18.4, displays the location and orientation of the three main mine site components:

- the Vertikalny open pit area and underground mine
- the process plant facility
- the supporting site infrastructure, which encompasses all roads, buildings, facilities, and structures that support—but do not directly contribute to—the primary purpose of the mining operation.

Site preparation will consist of clearing overburden, and grubbing of areas that will accommodate the surface facilities.

18.3.1 SITE LAYOUT DESIGN BASIS

Silver Bear sought to establish mine site facilities in the immediate vicinity of the main Vertikalny deposit. The layout of the facilities on site was optimized and several concepts were analysed before arriving at the final design, shown in Figure 18.4.

The mine site layout design philosophy was to keep the mine site as compact as is practical (in order to minimise environmental and social impact), keep haul and pumping distances as short as possible, and minimise the amount of surface water that will have to be managed as a result of infrastructure interference.

The area available for mine development around the deposit is heavily constrained; a hilly terrain, with steep slopes, within a large licence area. The mine site layout design has been constrained by:

- the Property licence area
- the Sirelendge River to the north and east of the mine site
- the topography of the mine site.

Local meteorological conditions, as well as the location of equipment, buildings, and storm drainage, were taken into consideration during the site layout design. The process plant is set on the hill top at the closest suitable location to the pit to minimise haul distances, and is oriented to take into account the prevailing wind direction. The prevailing winds recorded on an annual basis show the dominant directions are northeast and southwest.

The plant site layout, shown in Figure 18.5, is based on the concepts of maximised operability, maintainability, and general safety. Special design attention was given to the positioning of process and other equipment in the plant layout to ensure direct and efficient flow of material from the ROM feed point, through to the silver powder production room, minimising conveyor lengths and piping runs.

Figure 18.4 Site Infrastructure Layout

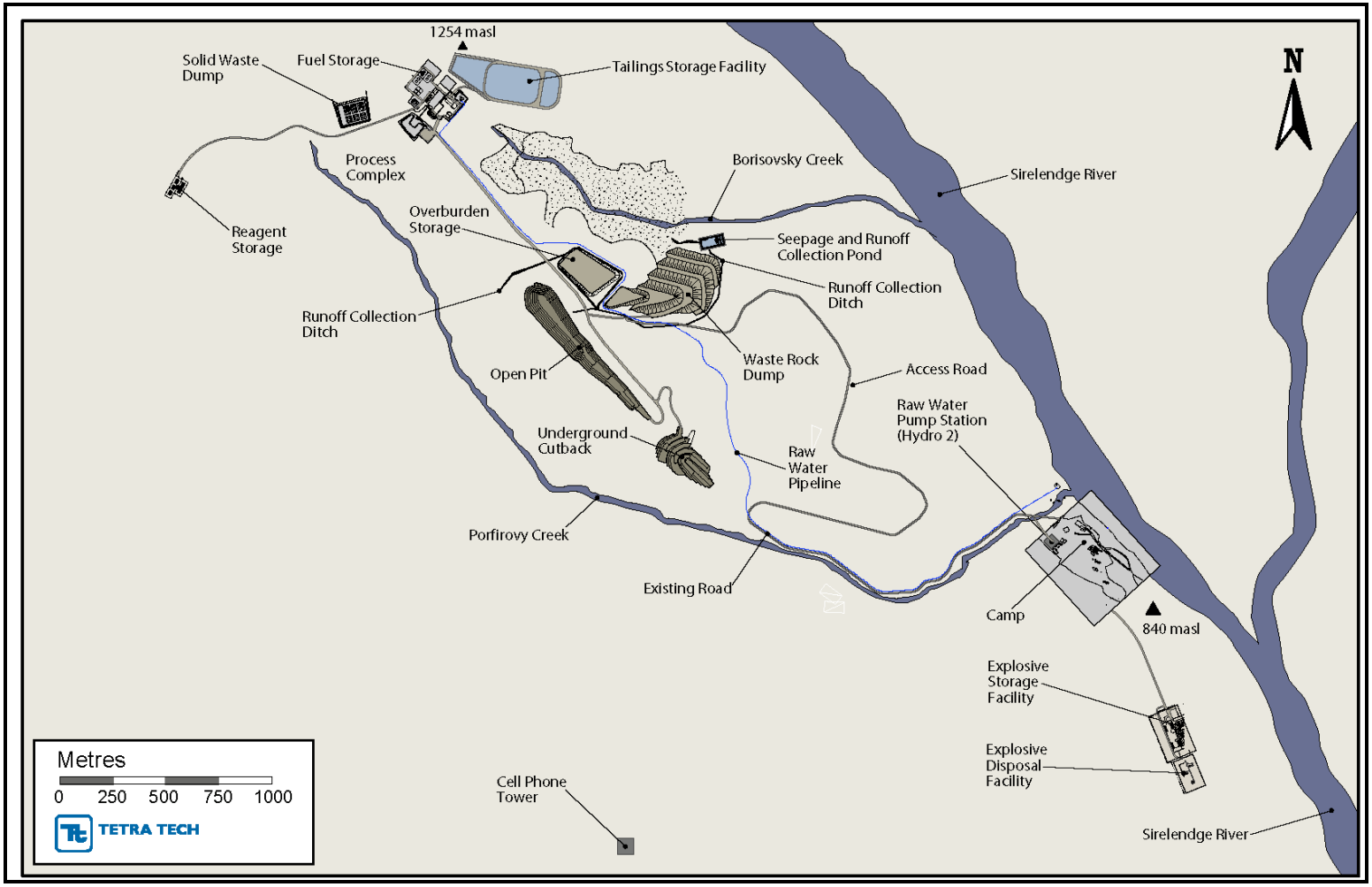
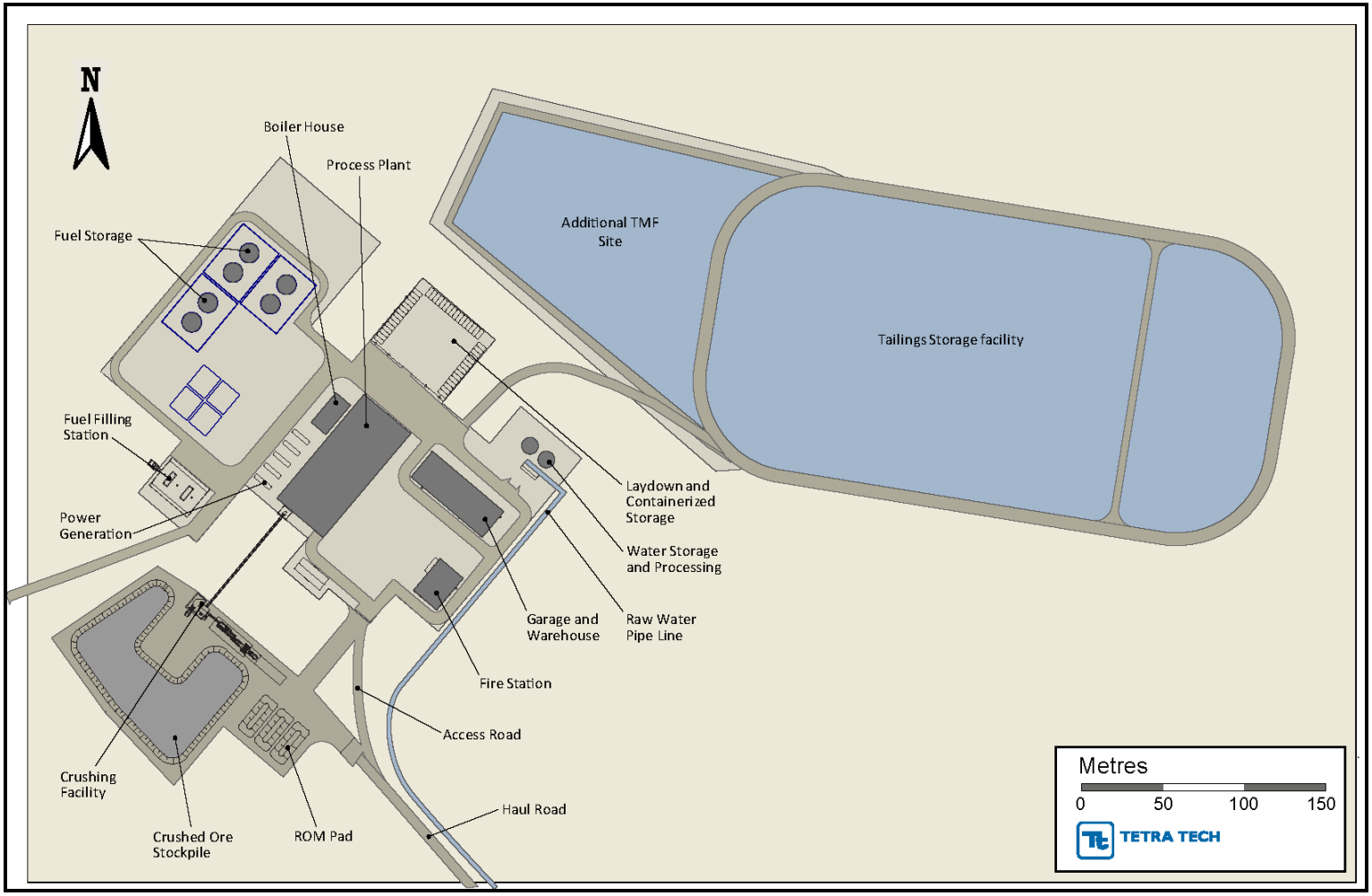


Figure 18.5 Process Plant Facility Layout



18.4 SITE GEOTECHNICAL

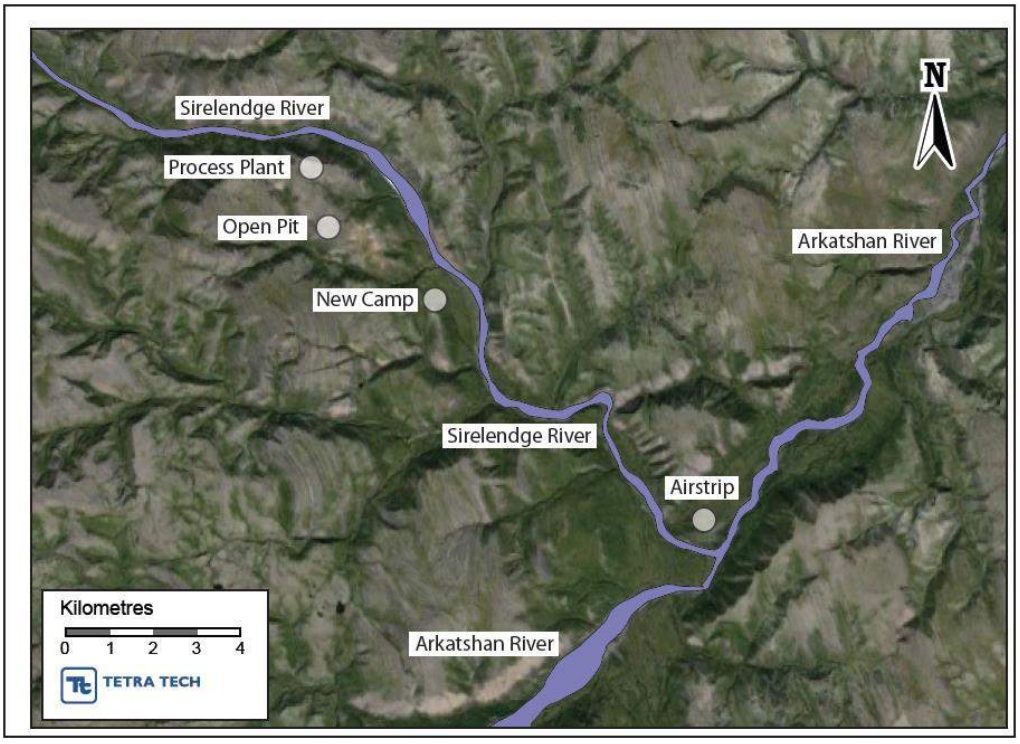
The Project infrastructure will be spread across three distinctly different areas, located along two river valleys, the Sirelendge and the Arkatshan, as shown in Figure 18.6.

The three areas are:

- the old camp site and airstrip
- the new camp area with storage facilities
- the central part of the Project, which will include the process plant, auxiliary buildings, and the dry stack TMF.

The three individual areas are located at three different elevations, in relation to the river levels in the two valleys. Therefore, they present somewhat different geotechnical features and conditions for the design and construction of buildings, roads, berms, pipelines, etc. The following subsections summarize the general soil conditions across the three areas.

Figure 18.6 General Plan Showing the Mine Site Area, Airstrip, and Area River Valleys



18.4.1 OLD CAMP SITE AND AIRSTRIP

The historic camp site and airstrip are located at the junction of the two major rivers in the area, as shown on Figure 18.6. The river valley floors are generally covered with permeable, well-sorted rock fragments, comprising sand to boulder sized soil grains. Smaller particles, such as clay and silt, have also been transported by the rivers and deposited at their junction, forming the floodplain. The subgrade of the

airstrip, located on that floodplain, generally comprises a cobble- and boulder-like rock fragment matrix, with sand, silt, and clay filling the voids of the coarse granular mass (Figure 18.7 and Figure 18.8).

Figure 18.7 Surface Soil Conditions across the Airstrip and along the Road to the Mine Site

Figure 18.8 Bank of the Sirelendge River Next to the Airstrip

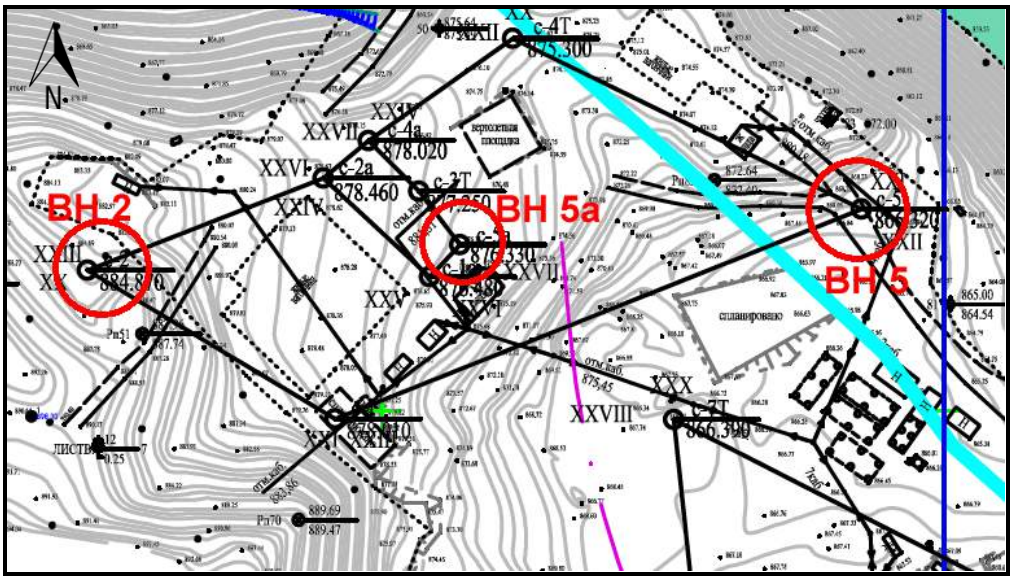
Note: Figure shows gravel, cobbles and boulders along the bank of the Sirelendge River next to the airstrip, with some silt and sand in the voids of the generally coarse-grained granular soil matrix

18.4.2 New Camp Site and Storage Areas

The new camp site is located on an elevated plateau, at the junction of the Sirelendge River and the Porfirovy Valley, as shown in Figure 18.4. The average elevation of the camp site is located approximately 40 m above the adjacent Sirelendge Valley, while the surface of the storage area is located at a somewhat lower elevation, approximately 25 m above the adjacent river level.

As part of an extensive geotechnical investigation, a large number of test holes were drilled/excavated in the new camp and storage areas by a local company (named ООО “НСИ” (NCI)), in order to evaluate the general soil conditions across the site. A detailed geotechnical report (NCI 2016) was completed by NCI, summarizing the thicknesses and properties of the soil overburden across the Property. A copy of that report can be found in Appendix I. A section of the relevant test-hole map for the camp and storage areas is shown in Figure 18.9. The location of three representative test holes (BH 2, BH 5 and BH 5a) are marked with red circles on the figure. A transcript from the NCI (2016) report is shown in Table 18.4, which presents data in connection with the thicknesses and conditions of the soil and rock strata in the area.

Figure 18.9 Representative Geotechnical Test Locations across the New Camp Site and Storage Areas



Note: BH – borehole

Table 18.4 Recent Geotechnical Investigation Field Notes, Presenting the Intercepted Layers at Three Representative Test Locations at the Camp Site

Borehole No.	Thickness (m)	Depth (m)	Description
2	1.0	1.0	Shingle soil with up to 25% grey sand aggregate, loosely frozen, marginally water-saturated when thawed. The detrital material is represented by sedimentary rocks of average durability, of poor degree of roundness.
2	2.1	3.1	Transparent ice, thick, with sand and organic residues of 10 to 15%. From 2.4 m depth ice is porous, mouldy, and free of impurities.
2	1.4	4.5	Gravel soil with up to 37% grey sand aggregate, frozen, of crusted cryogenic texture. Visible ice content ≈0.1 unit fractions, water-saturated when thawed. The detrital material is represented by sedimentary rocks of average durability, of poor and average degree of roundness.
2	1.0	5.5	Transparent ice, thick, with sand and organic residues of 10 to 15%. From 2.4 m depth ice is porous, mouldy, and free of impurities.

table continues...

Borehole No.	Thickness (m)	Depth (m)	Description
2	1.4	6.9	Gravel soil with up to 20% grey sand aggregate, frozen, frozen, of crusted cryogenic texture. Visible ice content ≈0.1 unit fractions, water-saturated when thawed. The detrital material is represented by sedimentary rocks of average durability, of poor and average degree of roundness.
2	8.1	15	Shingle soil with up to 14% grey sandy loam aggregate, frozen, frozen, of crusted cryogenic texture. Visible ice content ≈0.2 unit fractions, water-saturated when thawed. The detrital material is represented by sedimentary rocks, of average durability, of poor to average degree of roundness.
2	-	-	Ground water was not encountered during drilling.
5	5.4	5.4	Shingle soil with up to 35% sandy loam aggregate, frozen, of massive and crusted cryogenic texture. Visible ice content <0.1 unit fractions, the aggregate is plastic when thawed, from 2.0 m depth soil is of crusted cryogenic texture $\Lambda_B \approx 0.1$ to 0.2 unit fractions, the aggregate is fluid when thawed. The detrital material is represented by sandstones and siltstones of average durability and durable, of poor to good degree of roundness.
5	1.6	7.0	Medium sized sand, grey, frozen, of massive cryogenic texture moderately water-saturated when thawed.
5	2.2	9.2	Shingle soil with up to 20 to 25% sand aggregate, frozen, of crusted cryogenic texture. Visible ice content ≈0.1 to 0.2 unit fractions, water-saturated when thawed. The detrital material is represented by sedimentary rocks, of average durability and durable, of poor to good degree of roundness.
5	1.8	11	Gravel soil with up to 20% sand aggregate, frozen, of crusted cryogenic texture. Visible ice content ≈0.1 to 0.2 unit fractions, water-saturated when thawed. The detrital material is represented by sedimentary rocks, of average durability and durable, of poor to good degree of roundness.
5	4.0	15.0	Shingle soil with up to 20% sandy loam aggregate, frozen, of crusted cryogenic texture. Visible ice content <0.1 unit fractions moderately water-saturated when thawed. The detrital material is represented by sedimentary rocks of average durability and durable, of poor to good degree of roundness.
5	-	-	Ground water was not encountered during drilling.
5a	3.1	3.1	Shingle soil with up to 15% brown sand aggregate, frozen, of massive cryogenic texture, from 2.3 m depth of crusted cryogenic texture $\Lambda_B \approx 0.1$ unit fractions, marginally water-saturated when thawed. The detrital material is represented by sandstone of average durability, of average and good degree of roundness.
5a	4.9	8.0	Gravel soil with up to 20 to 25% brown sand aggregate, frozen, of crusted cryogenic texture. Visible ice content ≈0.1 to 0.2 unit fractions, water-saturated when thawed. The detrital material is represented by sandstone of average durability, of good to poor degree of roundness.
5a	-	-	Ground water was not encountered during drilling.

Based on information obtained from Table 18.4, it is evident that the ground conditions on the camp's plateau are represented by an undulating bedrock profile, with weathered rock overburden generally covering the entire area. Pockets of peat and ice were also intercepted in some poorly drained areas within the soil deposits. For example, an ice pocket as large as 2.1 m was intercepted at test hole no. 2, at the extreme west end of the camp site.

Construction of the already erected camp and storage buildings required the removal of peat and ice pockets, and the spreading of cobble- and boulder-like fill, in order to provide appropriate subgrade for the buildings' foundations. It is assumed that all peat and ice was removed from beneath all existing structures, wherever those pockets were intercepted (Figure 18.10).

Pockets of black peat with large tree remains were also noted in some low-lying areas between the storage and new camp area.

Figure 18.10 Soil Conditions across the Storage Areas

18.4.3 Process Plant and TMF Areas

As shown in Figure 18.4, the process plant, the dry stack TMF, and all related structures are proposed for construction across the northern most part of the mine site, on the top of the mountain. The area selected for these structures is generally flat, with gentle slopes leading toward the edges of the plateau, where the surface then drops sharply into the adjacent valleys. The elevation difference between the plant site and the nearby Sirelendge Valley is approximately 400 m.

The soil conditions across the plant and TMF areas are generally represented by weathered rock overburden over shallow bedrock. Based on the reported test results

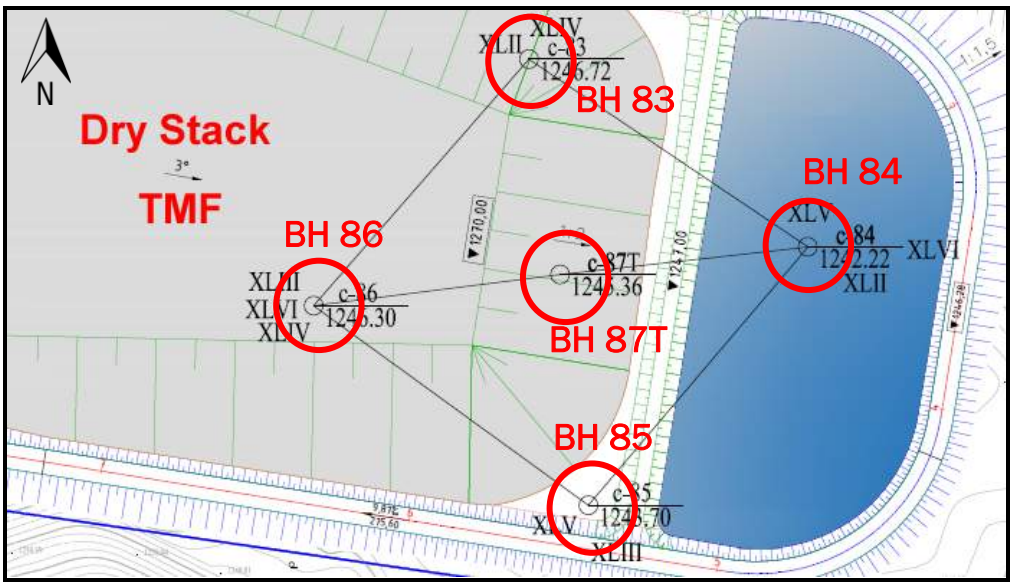
(NCI 2016), the area of the proposed dry stack TMF is covered with shallow weathered rock fragments, down to depths of between 1.2 and 1.5 m, overlying siltstone bedrock.

Beneath the weathered rock, comprising silt-sand-gravel and cobble-sized particles, the underlying rocks may also be fractured/jointed. However, these fractures and joints are the result of the tectonic history of the landform and not the result of the weathering process. Tectonic history is the sequence of earth movements that have brought about the gross arrangement and relative elevation of major topographic elements like mountains. Joints (also termed extensional fractures) are planes of separation within rock masses, on which no shear displacement had taken place. The two walls of the resulting opening in such rock masses typically remain in relatively tight contact. Joints may also result from regional tectonics, faulting, or internal stress release during uplift from lower elevations.

Based on the borehole results from the plant area, the opening of the joints in the permanently frozen sandstone and siltstone rocks varied between 0.5 mm and 3 mm only. During tectonic history of the mountain, water may have percolated down into these cracks and joints within the rock mass, and that water froze when permafrost was formed in the area, without changing the volume of the rock mass. This had been confirmed by the fact that after thawing the ice, found in the joints in the boreholes, the samples were not fully saturated.

The approximate locations of the above mentioned test holes at the process plant and TMF areas are shown on Figure 18.11, and the information obtained during the recent geotechnical investigation, is presented in Table 18.5.

Figure 18.11 Borehole Locations in the Area of the TMF



Note: Figure is not to scale

Table 18.5 Recent Geotechnical Investigation Field Notes, Presenting the Intercepted Layers at Representative Test Locations at the Proposed Dry Stack TMF Site

Borehole No.	Layer Thickness (m)	Depth (m)	Description
83	0	0.1	Topsoil, frozen.
83	1.4	1.5	Gruss sandy loam, brown, frozen, of massive cryogenic texture, is plastic when thawed. The detrital material is represented, primarily, by siltstone of average durability.
83	2.5	4	Grey siltstone, of average durability, fractured very intensely, open-type fissures, 0.1 to 2 mm width, filled with ice and the frozen sandy loam, oriented at an angle 0 to 10° and 80 to 90° to the core axis, tile core recovery.
83	-	-	Ground water was not encountered during drilling.
84	0.1	0.1	Top soil, frozen.
84	1.1	1.2	Shingle soil with sandy loam aggregate up to 20%, frozen, of massive cryogenic texture, marginally water-saturated when thawed, from a depth of 0.8 m of crusted cryogenic texture. Visible ice content <0.1 unit fractions, moderately water-saturated when thawed. The detrital material is represented, primarily, by siltstone of average durability.
84	2.8	4	Grey siltstone, of average durability, fractured very intensely, open-type fissures, 0.1 to 2 mm width, filled with ice and frozen sandy loam, oriented at an angle 0 to 10° and 80 to 90° to the core axis, tile core recovery.
84	-	-	Ground water was not encountered during drilling.
85	0.1	0.1	Topsoil, frozen.
85	1.4	1.5	Shingle soil with sandy loam aggregate up to 20%, frozen, of massive cryogenic texture, soil marginally water-saturated when thawed, from a depth of 1.0 m of crusted cryogenic texture. Visible ice content <0.1 unit fractions, moderately water-saturated when thawed. The detrital material is represented, primarily, by siltstone of average durability.
85	2.5	4	Grey siltstone, of average durability, fractured very intensely, open-type fissures, 0.1 to 2 mm width, filled with ice and frozen sandy loam, oriented at an angle 0 to 10° and 80 to 90° to the core axis, tile core recovery.
85	-	-	Ground water was not encountered during drilling.
86	0.1	0.1	Topsoil, frozen.
86	1.1	1.2	Shingle soil with sandy loam aggregate up to 20%, frozen, of massive cryogenic texture, marginally water-saturated when thawed, from a depth of 0.8 m of crusted cryogenic texture. Visible ice content <0.1 unit fractions, moderately water-saturated when thawed. The detrital material is represented, primarily, by siltstone of average durability.
86	2.8	4	Grey siltstone, of average durability, fractured very intensely, open-type fissures, 0.1 to 2 mm width, filled with ice and the frozen sandy loam, oriented at an angle 0 to 10° and 80 to 90° to the core axis, tile core recovery.
86	-	-	Ground water was not encountered during drilling.
87т	0.1	0.1	Topsoil, frozen.

table continues...

Borehole No.	Layer Thickness (m)	Depth (m)	Description
87т	1.3	1.4	Shingle soil with sandy loam aggregate up to 20%, frozen, of massive cryogenic texture, marginally water-saturated when thawed, from a depth of 1.0 m of crusted cryogenic texture, Visible ice content<0,1 unit fractions, moderately water-saturated when thawed. The detrital material is represented, primarily, by siltstone of average durability.
87т	8.6	10	Grey siltstone, of average durability, fractured very intensely, open-type fissures, 0.1- 2 mm, filled with ice, oriented at an angle 0 - 10° and 80 - 90 °to the core axis, rarer at an angle 20 - 30° to the core axis, tile core recovery, from a depth of 5.5 m fractured very intensely, fissure are orient, primarily, at an angle 80 - 90° to the core axis, rarer at an angle 0 - 10° to the core axis, tile and cylinder core recovery up to 20 cm.
87т	-	-	Ground water was not encountered during drilling.

Representative images of the weathered rock overburden are shown in Figure 18.12 and Figure 18.13. During a recent site visit, silt and clay were also visible within the voids of the larger gravel, cobble, and boulder sized weathered rock matrix. In addition to the silt and clay pockets, attention should also be given to the locations and extents of any ice lenses, intercepted within the soil overburden.

Figure 18.12 Soil (Weathered Rock Fragments) Conditions across the Plant Site

Figure 18.13 General Soil Conditions (Shallow Weathered Rock) Across the Upper Plateau, Approximately 400 m above the Adjacent Sirelendge Valley

18.4.4 CONCLUSIONS

Based on the available data and information in connection with the soil conditions across the Project area, it appears that the predominant soil deposit (overburden) on the site comprises sand- to boulder-sized solid grains (weathered rock fragments), with random silt- and clay-sized particles filling the voids near the surface, mostly at poorly drained localized areas, and in the floor of the Porfirovy and Borisovsky valleys.

The formation of such overburden layers, comprising soil-mixed rock fragments, is a direct result of the extensive physical weathering process within the active zone of permafrost. Physical weathering causes rocks to disintegrate in situ; meaning the rock masses break up near their surfaces. Physical weathering occurs when there are changes in temperature over short periods of time. In permafrost areas, that temperature fluctuation happens within the so-called active layer, near the ground surface only. Two potential geotechnical related risks may be identified on this site:

- Organically contaminated, compressible soils (peat):
  - Peat pockets were intercepted in low-lying, poorly drained areas (in parts of the new camp site, for example), where decomposition of vegetation may have resulted in development of peat pockets. Since most of the buildings at the camp site have already been constructed, it can be assumed that their foundations are not sitting on peat deposits. It is recommended to remove all peat pockets (if encountered) from under all engineered structures (buildings, berms, dams, etc.), proposed to be built on the site.
  - Based on field notes from site personnel and observations during a site visit, it appears that such peaty areas have not been encountered across the upper plateau, where most of the critical infrastructure

components (process plant and dry stack TMF) are planned for construction.

- Large ice pockets within the soil overburden:
  - These pockets represent the highest risk to the stability and integrity of any structure built in permafrost areas. Thawing of potential ice pockets within the soil overburden in the future may result in serious damage to any building, even to rigid structures. It will be imperative during the construction phase that no ice pockets in soils are left in place beneath any engineered structures (buildings and dams). The simplest method for dealing with such situations is to remove all thaw-unstable silty and clayey soils (including potential ice pockets) from under critical structures, and establish their foundations on solid, well-drained rock, or thaw-stable dense, granular soils without any possibility of future frost heave or thaw damage to the substructures or dams.

The word permafrost means permanently frozen ground (soil and rock). It indicates a thermal condition where the temperature of the rock and soil of the upper part of the earth's crust remains below freezing throughout the year. Approximately 50% of the world's permanently frozen ground is located in Russia/Siberia. Permafrost may extend several hundred meters below the so-called active layer, and can be continuous and discontinuous. The approximate thickness of the permanently frozen ground at the mine site is between 300 and 400 m.

Construction of building foundations or other engineered structures in these areas require specific knowledge regarding permafrost and specialized construction techniques. Disturbing thaw-unstable soils in permafrost areas may cause thaw, resulting in uneven foundation settlements and detrimental consequences for structures. Permafrost lies beneath the seasonal frost zone and remains frozen even throughout the summers. Seasonal frost develops over the winter and thaws in the summer. This "seasonal frost zone", located over permafrost, is called the active layer, as shown in Figure 18.14.

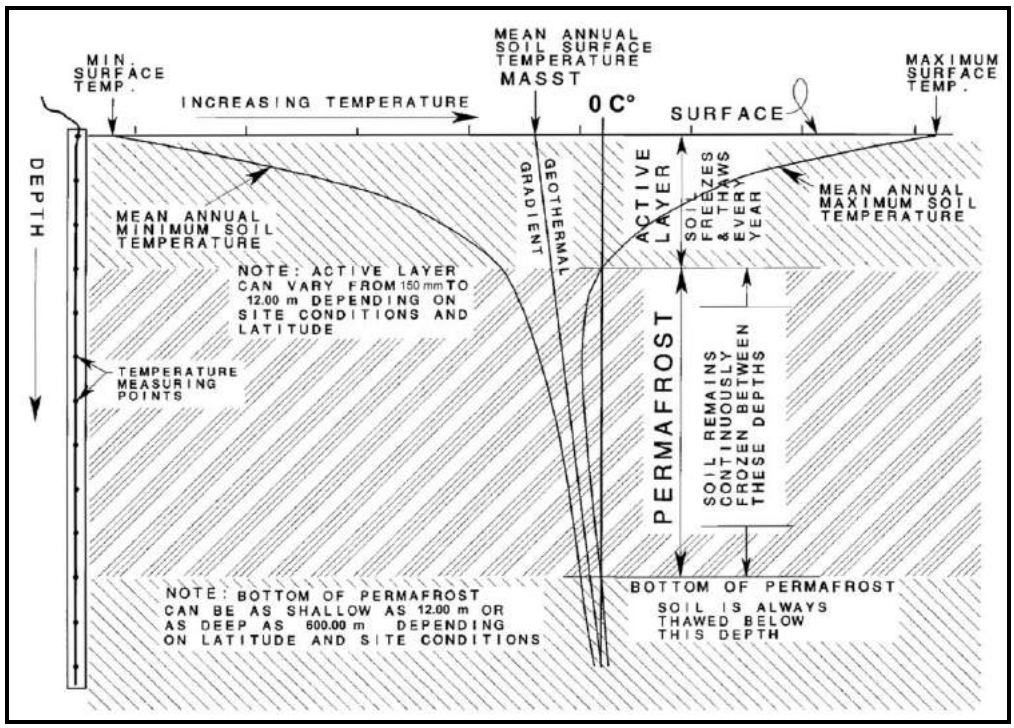
Field ground temperature measurements at the Project site prove that the active layer is less than 2-m thick across the entire Project area.

As pointed out previously, soil explorations were recently carried out across the site (NCI 2016) and a copy of the report can be found in Appendix I. Based on available information, it appears that at the site of the most important structures (process plant and dry stack TMF), sandstone and siltstone bedrock are available as foundation subgrade at relatively shallow depth below grade, and no frost and heave related problems can be anticipated if all thaw-unstable silt and clay-contained soils are removed from beneath all proposed structures.

As a preliminary guide, spread footings for heavily loaded structures will be established on bedrock, with consideration of an estimated bearing capacity value of 500 kPa, assuming that all foundation concrete is established on solid rock, which has been cleaned completely of all loose debris and shatter using an air hose or water jetting procedures. Structures may also be established on well-compacted engineered granular fill subgrade (crushed rock), with consideration of an estimated bearing capacity value of 150 kPa. As pointed out above, all peat, silt, and clay

contained soils and ice pockets must be completely removed from under all engineering structures, prior to construction.

Figure 18.14 Ground Temperature Profile and Extent of Active Layer in Permafrost Areas



18.5 DESIGN BASIS

18.5.1 STRUCTURAL DESIGN BASIS

STRUCTURAL CLIMATIC LOADS

Based on data received for the Project area (latitude 65.65° north and longitude 130.183° east), the climatic data for structural design are as follows.

Wind

The wind pressure at site corresponds to Zone III on the Wind Regions of USSR Map 3. As per SNiP (Russian Construction Codes and Regulations) 2.01.07-85 *Loads and Effects*, this corresponds to a wind pressure of W_o = *0.38 kPa.*

The area type is assumed to be Type B, as per paragraph 6.5 of SNiP 2.01.07-85 *Loads and Effects*.

Snow Load

For the Project site, the snow load corresponds to Zone III on the Snow Regions of USSR Map 1A. As per Table 4 in SNiP 2.01.07-85 *Loads and Effects*, this corresponds to a snow pressure of S_o = *1.0 kPa.*

Seismic

The Project is located in an area with moderate seismic activity representing a medium earthquake hazard. The expected seismic intensity at the proposed mine site does not appear to exceed the Medvedev-Sponheur-Karnik (MSK) scale's VII Category, which has a ground acceleration between 0.8 and 1.6 m/s^2. Typically, MSK scale VII corresponds to a peak ground acceleration (PGA) of 0.1 g (i.e. 10% of gravity acceleration = 0.0981 m/s^2). As per the 2012 International Building Code (IBC), for a PGA of 0.1 g, the following seismic parameters can be obtained:

S_S = *0.2 g*

S_1 = *0.1 g*

Where:

S_S = *0.2 s spectral response acceleration (5% damping), site Class B, and*

S_1 = *0.2 s spectral response acceleration (5% damping), site Class B.*

For further information on the seismic characteristics of the Project see Section 18.5.2

Structural Steel and Concrete

Codes and Standards

All structural designs shall be carried out in accordance with the latest revision of the following standards, regulations, specifications and codes:

- All applicable Russian codes and standards, or equivalent US, Canadian and European codes.
- Concrete, mortar, and grout shall conform to SP 52-101 2003 *Concrete and Reinforced Concrete Structures Without Pre-stressing*, and SP 52-105 2009 *Concrete Structures for Cold Climate and Permafrost Soils.*
- Reinforcement shall conform to GOST 5781-82 *Hot Rolled Steel for Reinforcement of Reinforced Concrete Structures,* welded wire mesh shall conform to GOST 23279-85 *Welded Mesh Reinforcement for Concrete Structures and Products, General Specifications.*
- Cement shall conform to GOST 7473 2010 *Cement Mixtures, Technical Specifications.*
- Masonry shall conform to SN 15.13330.2012 *Masonry and Reinforced Masonry Structures.*
- Structural steel shall conform to GOST 23118-2012 *Structural Steel Elements – General Specifications.*
- Welding shall conform to latest Russian welding standards.
- Galvanization shall conform to European Committee for Standardization codes as per: CEN/TC 262 *Metallic and Other Inorganic Coatings.*
- Painting shall conform to European Committee for Standardization codes as per: CEN/TC 139 *Paints and Varnishes*.

UNITS

All notes, calculations, data instructions, dimensions on drawings, and specifications shall use metric units.

Materials and equipment shall all be specified in metric units. Where certain items are available in imperial sizes only, they shall be specified in metric units after soft conversion, with corresponding imperial sizes specified in brackets.

DESIGN

Preliminary design will be carried out for reinforced concrete structures, foundations, structural steel supports, steel structures, access ways, and platforms. As much as possible, the structural steel shall be modularized so that construction work on site is minimized. Some of the buildings and structures include:

- process plant
- conveyor foundations
- pipe racks
- pre-engineered buildings.

Importance Category

As per SNiP 2.01.07-85, the responsibility level of buildings and structures (classified accordingly to the economic, sociological, and ecological consequences for failure) for the Project shall be Level II, which is the normal responsibility level usually designated for industrial buildings.

Expansion Joints

The maximum length of buildings and structures between expansion joints shall be 120 m.

Deflection

Do not exceed the following deflection (D) to span (L) or height (H) ratios under live loads (Table 18.6 and Table 18.7).

Table 18.6 Vertical D/L

Criteria	Value (D/L)
Crane Runway Beams	1/800
Floor Members	1/360
Roof Members Supporting Plaster	1/360
Roof Framing with Sheet Metal Cladding	1/180

Table 18.7 Lateral D/H

Criteria	Value (D/H)
Walls with Sheet Metal Cladding	1/180
Walls with Un-Insulated Cladding	1/90
Interior Walls with Permanent Partitions Subjected To 0.5 kN/m^2 Normal to the Walls: With Brittle Finishes With Flexible Finishes	 1/240 1/120
Crane Runway Beams* (Subject to Crane Lateral Force)	1/600
Building Column* (Subject to Crane Lateral Force or Wind Force)	1/300
Building Frames (Subject to Wind Forces)	1/300
*Follow manufacturers standards or the following additional crane runway tolerance criteria, whichever produces the most adverse effect:	
Maximum Permissible Tolerance Between Rail Centres	±5 mm
Height Tolerance Between Rails (S = Crane Span)	S/1000
Permissible Height Tolerance Between Two Crane Rail Supports (L = Distance Between Supports)	L/1500

Recommended depth (d) to span (L) ratios for steel beams and trusses are shown in Table 18.8.

Table 18.8 Member Depths (d/L)

Criteria	Value (d/L)
Trusses	1/14
Equipment Support Beams*	1/16
General Platform Beams	1/20
Walkway Beams	1/24

Note: *A deeper section may be required to avoid the beam resonating at the natural frequency of equipment.

Slab on Grade

Slab on grade shall be constructed on engineered fill over suitable existing subgrade, approved by a geotechnical engineer unless noted otherwise.

Minimum thickness of slab on grade is shown in Table 18.9.

Table 18.9 Minimum Slab on Grade

Criteria	Value (mm)
General Areas	150
Light Vehicular Traffic Areas	200
High Wheel Load Areas	250

Live Loads/Imposed Floor Loads

Minimum live/imposed loads are shown in Table 18.10.

Table 18.10 Minimum Live Loads

Criteria	Value (kPa)
General Areas	5.0
Office Areas	5.0 + Partition
MCC/Electrical Rooms	15.0
Mill Operating Floor	15.0
Laydown Areas	12.0

Note: MCC – motor control centre

Roof Loads

In addition to roof live loads, a collateral load of 0.75 kPa shall be allowed for the design of all building roofs unless noted otherwise.

MATERIALS SPECIFICATIONS

Structural Steel

Structural steel will conform to latest edition of Eurocodes shown in Table 18.11.

Table 18.11 Structural Steel Eurocodes

Area	Code(s)
Hot Rolled Structural Steel IPN, HP HD, HE/HL, IPE Shapes, UPN Channels and Steel Angles	Steel Grade S355
Plates and Bars	Steel Grade S275
Hollow Sections	Steel Grade S355
Structural Pipes	Steel Grade S355
Sag Rods and Pins	Steel Grade S275
Bolted Connections	Grade 8.8 bolts
Welded Connections	As per EN ISO 2590 fy = 500 MPa
Anchor Colts	Grade 4.6 bolts
Floor Grating	32 mm x 4.8 mm Welded Standard “Flowforge” type Serrated grating : 38 mm x 4.8 mm
Trench Grating	32 mm x 4.8 mm Welded Standard “Flowforge” type Serrated grating: 38 mm x 4.8 mm
Checkered Plates	8 mm minimum checker plate with raised pattern 6 mm minimum for lightly loaded area
Stair Treads	Grating tread with checker plate nosing 32 mm x 4.8 mm Welded Standard “Flowforge” type Treads with serrated grating: 38 mm x 4.8 mm Stair tread width – 308 mm

table continues...

Area	Code(s)
Railing and Handrails	48.3 outer diameter by 4-mm thick wall round pipe for horizontal handrails, 48.3 outer diameter by 6-mm thick wall round pipe for all vertical handrail posts
Kick Plates	150-mm high

Note: For primary members subject to tensile fatigue stresses, and/or high-impact loads, and/or exposed to cold weather conditions, steel grade used shall be weldable, notch-tough grade.

Concrete

Cement shall conform to EN 197 and use CEM I or CEM type II/A, unless noted otherwise. The Project will use corrosive resistant cement where applicable and sulphate resisting cement where applicable.

The specified compressive strength of concrete is shown in Table 18.12.

Table 18.12 Concrete Specified Compressive Strength

Area	28 Day Strength (MPa)	Maximum Aggregate (mm)	Water/Cement Ratio
Structural Concrete	30	20	0.45 0.40 (corrosive)
Mass Foundation (only if noted as such on the drawings)	15	40	0.45 0.40 (corrosive)
Blinding Concrete (only if noted as such on the drawings)	15	20	0.45 0.40 (corrosive)

Reinforcing Steel

Table 18.13 Reinforcing Steel

Area	Code
Deformed Type Billet Steel Bar	In conformance with GOST 5781-82 Grade A-111(A400) fy = 390MPa
Welded Wire Fabric	In conformance with GOST 23279-85

Grout

Normal grout shall be non-shrink, non-metallic, and non-corrosive cementitious grout. Epoxy-based, non-shrink grout shall be used under sensitive equipment only.

Masonry

The Project will use modular metric size as per Russian codes. The minimum compressive strength of a hollow core concrete block unit shall be 15 MPa, and the compressive strength of masonry grout shall not be less than that of the concrete block used.

18.5.2 PROJECT SEISMIC CHARACTERISTICS

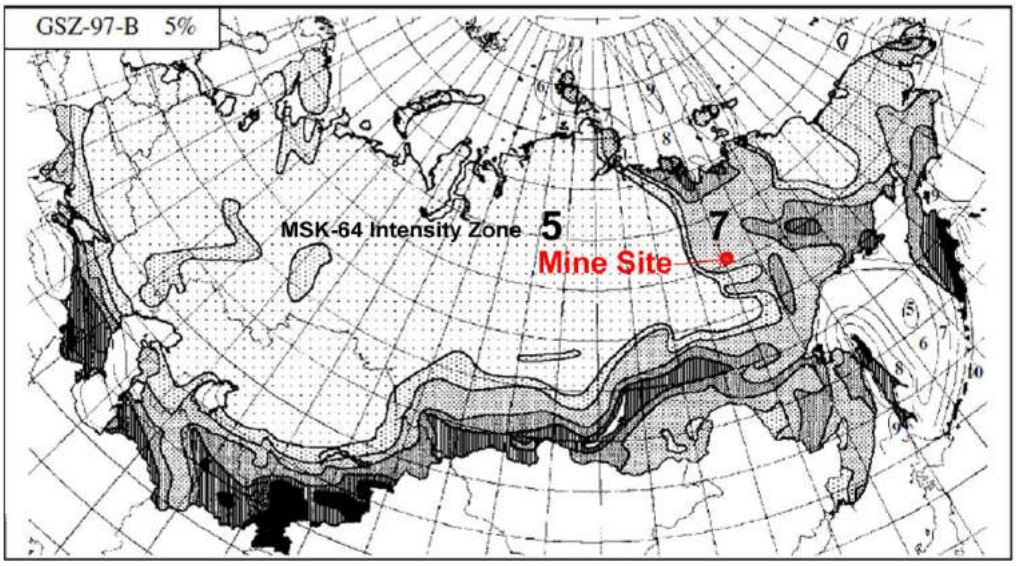
Seismic zoning is among the most complex and important components of modern seismology and seismic design. It is the process of estimating the response of soil layers under earthquake excitations, and thus the variation of earthquake characteristics on the ground surface at a project site. The study of seismicity and an adequate assessment of the seismic hazard are important for all areas where increased seismic hazard may exist and would require the implementation of anti-seismic measures.

A set of new General Seismic Zonation (GSZ) probability maps (GSZ-97-A, GSZ-97-B, and GSZ-97-C) were recently created for the Russian Federation and adjacent seismic regions, on the basis of the unified seismological and geological geophysical database. The GSZ maps of Russia are now being accepted as the basis for, and included in, the new edition of the Russian Federation National Building Code SNiP II–7–81 *Construction in Seismic Regions*.

According to the recently developed GSZ-97 maps, the probability of a possible exceedance of earthquake intensity within 50 years is: 10% (map GSZ-97-A), 5% (GSZ-97-B), and 1% (GSZ-97-C), which correspond to the mean periods of 500, 1,000 and 5,000 years, respectively, for the recurrence of such effect. The seismic effect predicted by these maps is coupled with average ground conditions in keeping with the present construction standards and rules (Russian Federation National Building Code) operating in earthquake-prone districts. The set of GSZ-97 (A, B, C) maps was endorsed by Russia's State Building Committee for the use in construction projects throughout the country. The GSZ-97-A map is recommended for the construction of common residential, public, and production buildings, while the other two maps (GSZ-97-B and GSZ-97-C) are recommended for the erection of structures that should continue in service, even during earthquakes and during work to eliminate their aftereffects (power and water supply facilities, fire stations, communication facilities, and main transportation routes such as highways). This also includes premises housing for a large number of people (hospitals, schools, kindergartens, railway stations, air terminals, theatres, roofed-in markets, stadiums) and for buildings higher than 16 stories.

The zoning on the three GSZ-97 (A, B, C) maps reflect the MSK macro-seismic intensity scale. The MSK scale has 12 intensity degrees and is somewhat similar to the Modified Mercalli Intensity (MMI) scale, used in the United States. The intensity of an earthquake is based on the extent of the damage to buildings and other observed effects on people, near the epicentre of earthquakes. Earthquakes with an intensity degree of up to 6 generally have no impact on buildings of good design and construction. However, in areas with potentially very strong earthquakes, an intensity degree of 7, serious damage may happen to old, substandard, and poorly constructed buildings. Figure 18.15 (GSZ-97B) shows that on the Project site the expected seismic intensity is MSK degree 7; therefore, buildings should be designed to resist potentially strong earthquakes.

Figure 18.15 Peak Ground Acceleration Map of Northern Eurasia – 5% Probability of Exceedance in 50 years



Source: Global Seismic Hazard Assessment Program

As opposed to intensity, the magnitude of the earthquake is a measure of its strength. The magnitude, M, of an earthquake is not based on the extent of the damage to buildings, but determined from the logarithm to base 10 of the amplitude, recorded by a seismograph. The magnitude is typically measured on the Richter Magnitude Scale. The largest known shockwaves on Earth had magnitudes in the 8.7 to 8.9 range.

Figure 18.15 shows the map of seismicity and seismic regionalization of Northern Asia, where zone 4.4 represents the Verkhoyansk region. As shown on the map, although a few earthquakes have been recorded somewhat north of the mine site, the area of increased seismic activity is located well to the east from the site, within the region.

As described earlier in this section, the intensity and magnitude of seismic events are observed and recorded near the epicentre of earthquakes. Based on ongoing field observations by seismic agencies around the world, it is well documented that the intensity of an earthquake may vary greatly within the impacted area of an earthquake. This is related to the fact that regional geology has a large impact on the characteristics of ground motion, generated by seismic activities. This means that a site response to the ground motion may vary in different locations within a seismic region according to the local soil conditions. Country wide seismic zonation maps may, therefore, be somewhat inadequate for detailed seismic hazard assessment of individual sites. Although they provide initial guidance for a general area within a country, that information must be further evaluated with the consideration of the actual site geology (soil and rock conditions) and proposed structural systems for buildings on a particular site.

During Tetra Tech's QP site visit, there was no indication of any earthquake induced slope movements in the area of the mine site. The steep slopes are formed from solid rock masses, and all-weathered rock movements appear to be related to gravity (rolling of dislocated cobbles and boulders off the rock slopes into the adjacent

valleys). It must be pointed out that earthquakes may have detrimental effects in areas where relatively thick saturated soil overburden covers the underlying bedrock, and the soil layers are either sensitive to liquefaction (loose saturated uniform sands) or may lose their shear strength when seismic activity may increase the pore water pressure in them (soft to very soft clays, that are always saturated). Based on field observations, there are no such soil formations at the Mangazeisky site. It should also be pointed out that soil-structure interaction effects that take place during seismic excitations govern the seismic response of foundations for buildings. Except for cases where liquefaction occurred, or sensitive clays lost their shear strength under cyclic loading, foundations failures during earthquakes are extremely rare.

Based on the summarized information in this section, it appears that the Project site is located away from potentially destructive seismic activity areas, and the potential for earthquakes greater than 6.5 on the Richter scale is negligible in the immediate vicinity of the mine site. The relevant seismic hazard map (GSZ-97-B) classifies the area as having intensity degree 7 on the MSK scale, assuming general soil conditions near a potential epicentre, and for structures that should continue in service, even during earthquakes. A review of the actual geology of the site also indicates that the site-specific response to a potential earthquake would be negligible, assuming that all structures are constructed in accordance with relevant design standards, and that all foundations are established either on solid rock or on well drained granular soil (weathered rock) subgrade and are free of any organically contaminated soil or ice pockets.

18.5.3 Architectural Design Basis

The architectural design basis for the site facilities are noted as follows:

- Buildings will be provided for the plant infrastructure. Crusher equipment and equipment not housed within the process plant will be exposed to the environment. Buildings that require large open spaces and overhead cranes will be of portal frame construction. Sandwich panel sheeting will be used to provide insulation, protection against wind and rain, and for security. Large roller doors will be used to provide access and ventilation.
- Where liquids are stored or handled, a sloped bund will be constructed as an integral part of the building, complete with sumps.
- Process plant bunds and sumps will be designed to eliminate cross contamination and short circuiting of process streams.

Buildings and roads will be arranged, where possible, to eliminate the need for vehicles to reverse.

Materials local to the region will be specified for use where possible.

18.6 Site Roads

18.6.1 Access from Airstrip to Camp

The airstrip is located approximately 10 km south of the Property, which includes the mine site and the new camp site (Figure 18.6). The access road to the new camp site

runs along the Sirelendge River, which has seasonally higher water levels in the valley, but remains frozen during most of the year. The condition of the route is generally poor during the summer months and will require upgrading in order to provide acceptable driving conditions from the airstrip to the mine.

The present condition of the valley floor is shown in Figure 18.16. Large quantities of granular rock fragments including boulders, cobbles, and gravel-sized particles are available along the road alignment and may be used to upgrade the existing un-graded road into a gravel-surfaced access road to the mine.

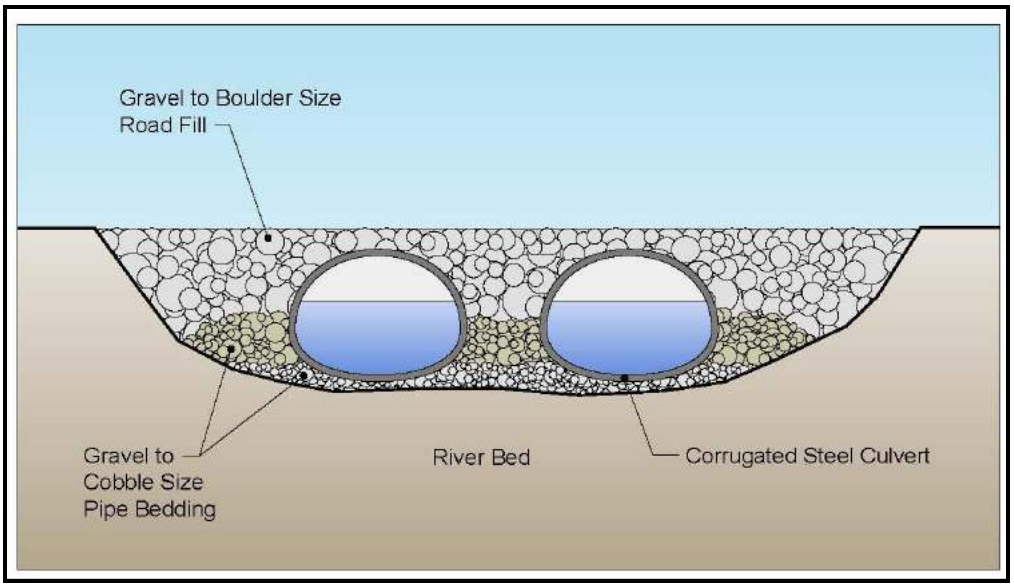
Figure 18.16 Gravel, Cobbles, and Boulders on the Valley Floor of the Sirelendge River

Presently, vehicles drive along the gravelly, cobble bank of the shallow Sirelendge and cross the river approximately 1.5 km downstream from the new camp site. Although the granular soils along the river banks generally provide good riding conditions for ATVs, some upgrade (re-grading) of the surface along the road alignment may become necessary. Since the river valley is generally wide, there is no shortage of clean, inert, granular material on the valley floor for such a road upgrade along the entire alignment.

For sections of road not on the riverbed and above the high-water level, the suggested cross section will be the similar to the internal mine roads specified for the road network across the mine site. The only difference will be in the width of the road, since the traffic between the airstrip and the camp site will require only one lane. The total width of this road will be 5.6 m, as opposed to 9.8 m for all other roads. Upgrading the road should be carried out during the summer months, when excavation and placement of the road fill material will be possible without encountering large quantities of ice and frozen soil. Mine personnel will complete the upgrade and annual road maintenance using equipment on site.

The river crossing will be upgraded with the installation of corrugated metal culverts in order to facilitate the free flow of water in the river during summer months, and easy driving conditions across the river for the ATVs above the culverts (Figure 18.17). The culverts will be sized based on the maximum anticipated river flow and the crossing will be designed to take into account annual fish migration (if any).

Figure 18.17 River Crossing Upgrade with Culverts



It is likely that during many months of the year vehicles will need to drive on ice along sections of the frozen river. To reduce the risk for those working on the ice road, as well as for the drivers, strict attention needs to be paid to testing the ice to ensure that it is capable of supporting the traffic load to which it is subjected. Once the thickness of the ice has been identified, the boundaries of the corridor where ice bearing capacity has been established should be clearly marked with any item that clearly delineates boundaries (e.g., trees, poles, cones), and drivers should ensure that transportation trucks stay within that corridor along the frozen river.

The most frequent and intensive ice-thickness testing will take place early in the cold seasons. As ice thickness increases and drivers become more confident that all thin areas have been identified, testing may be reduced. The ability of ice to support a truck depends on ice thickness, the pressure of the water below the ice as deflection develops, the way the ice initially forms, snow cover, vehicle speed, and the load types placed on the ice cover.

18.6.2 INTERNAL ROADS

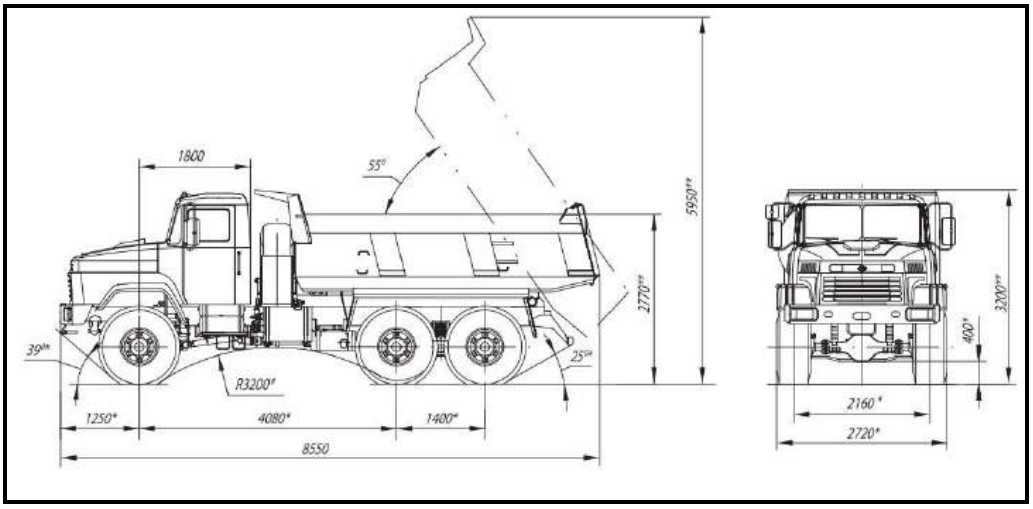
The internal roads across the mine site will connect the new camp site; the process plant; the open pit; and the ore, tailings, waste rock, and other storage facilities. The approximate alignment of the main road is shown in Figure 18.4. The current road alignment between the camp and the process plant area. The first 30% (approximately) of the existing road follows the Porfirovy Creek Valley from the new camp site, before entering a narrow side valley and climbing up toward the mine facilities, located at elevations high above the valley floor of Porfirovy Creek (Figure 18.18). The current road is presently either un-graded or running over the exposed

solid or weathered rock surface, or roughly graded with crushed, angular rock fragment fill.

Figure 18.18 Condition of the Existing Road in the Side Valley, Where Weathered Rock and Granular Fill with Various Thicknesses Cover the Valley Floor

The existing road upgrade, and the construction of any new road sections, will include the placement of acceptable granular fill materials over the prepared subgrade and to the specified width and thicknesses. The width of the road will depend on the specifications of the widest vehicle traveling on the road. At site, this is the KRAZ 65032 dump truck (Figure 18.19).

Figure 18.19 KRAZ 65032 Specifications



Based on the truck's specifications, all mine roads will have a minimum width of 9.8 m (Figure 18.20). This width includes the two shoulders and the minimum safety distance between two trucks driving next to each other in opposite directions. The specified cross section for the road is shown in Figure 18.21).

Figure 18.20 Width of Mine Roads

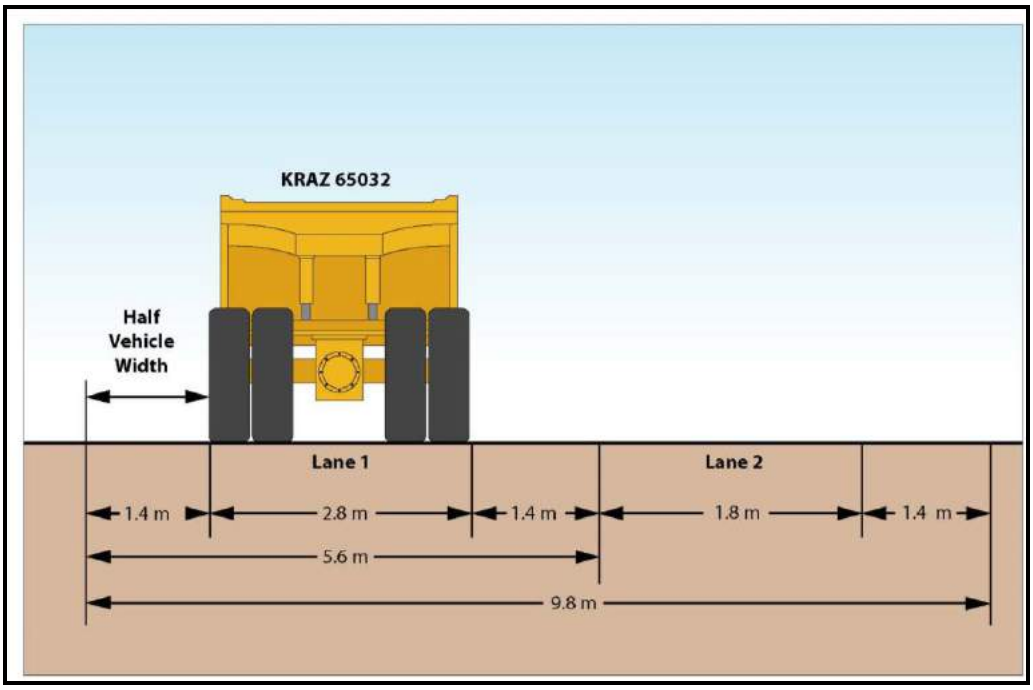
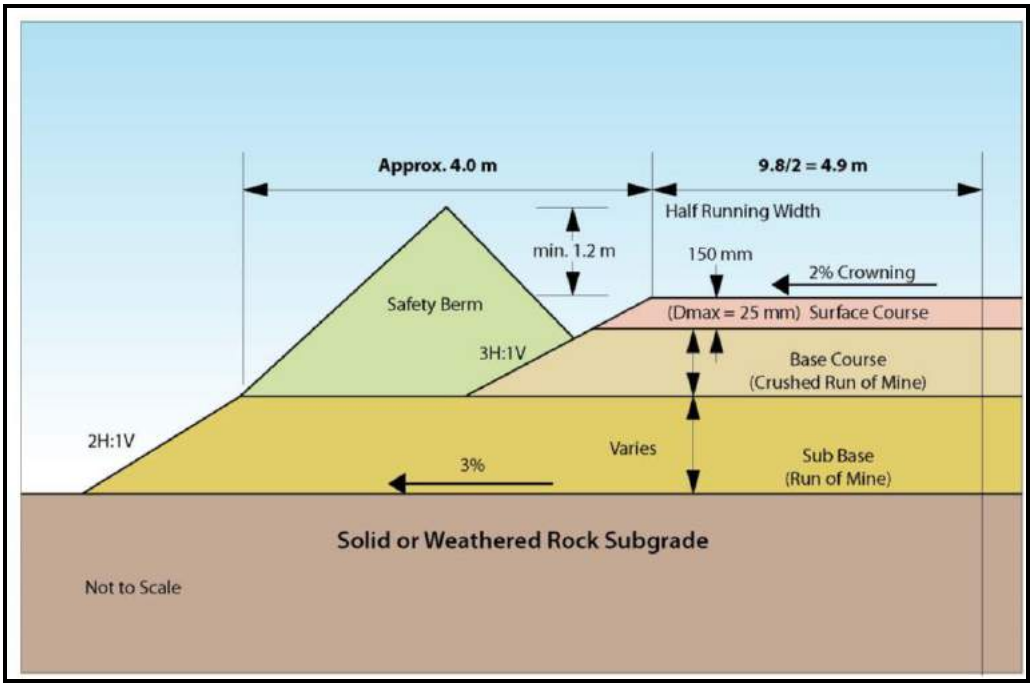


Figure 18.21 Cross Section of Mine Roads, with Side Ditch and Safety Berm



As shown in Figure 18.21, the individual layers within the road structure will be placed, compacted, and shaped to drain all water to the side of the road using cross-

sloping. Cross-sloping involves sloping the subgrade and road slightly to allow surface water to flow across and off the road rather than down the length of the road. Cross-sloping of the subgrade under the subbase layer will be 3%, while the road crowning will slope at least 2% toward the side ditch and safety berm. The surface water from all sources (rain or melted snow) will be conveyed from the roadway immediately in order to prevent road fill erosion, to maintain a stable road surface, and to reduce maintenance and repair costs.

Taking into account the heavy trucks that operate between the camp and mine sites, the minimum thickness of the surface aggregate (angular crushed rock) is specified as 150 mm. The crushed rock layer (D_{max} = 25 mm) will be placed on a compacted base course layer (crushed ROM), which will be placed over the compacted subgrade overburden or on rock. The thickness of the base course is generally dependent on the quality (condition) of the subgrade soil. Based on observations along the existing road, the thickness of the base course at site will be 300 mm and comprise crushed ROM material, with a maximum particle size of 150 mm in diameter.

The load-carrying capacity of the road will depend on continuous, stable support, furnished by the subgrade material, which consists mainly of natural on-site soil overburden or rock; predominantly the weathered rock at the Project site. Subgrade stability requires adequate drainage and proper load distribution by the surface and base courses. Surface and base courses of sufficient thickness and quality are necessary so that the applied stress is less than the unit load capacity of the subgrade. The subgrade conditions along most of the existing road alignment are fairly good, consisting of either solid or weathered rock. It is likely that no additional subbase layer (ROM fill with a particle size of up to 200 mm) will be required between the firm subgrade and the base layer. However, within the valleys at lower elevations (Porfirovy Creek and Borisovsky Creek), it may be necessary to remove the existing soil overburden, comprising layers and pockets of frost susceptible silt and clay. Leaving silt and clay pockets within the road profile may result in softening of the fine-grained soils during thaw periods and in unwanted deterioration of the road structure through excessive differential settlement and even localized embankment failures.

Figure 18.22 Representative Soil Conditions in the Narrow Borisovsky Valley

Note: The cobble/boulder matrix with pockets of finer soils (silt and clay sized particles) in the valley floor

The current roads on site have been constructed by Prognoz personnel using on-site equipment. The proposed road upgrade and maintenance is not dissimilar to their current practice, and it is reasonable that future mine site road construction can be completed by the Prognoz construction team using mine site equipment, waste rock, and the on-site crusher to produce aggregate.

During periods of warm and dry weather, roadbeds will be regularly watered, as required, to keep dust levels to a minimum.

18.6.3 VEHICLE TYPES

Considering the difficult driving conditions—rough terrain, steeper than usual road grades at some sections of the Property, fast flowing water in the river and creek valleys during the short thawing season, ice and snow during the icy, long winter periods—only all-wheel drive, ATVs will operate at site. Based on the latest mining plan, the vehicles shown in Figure 18.23 are considered for the Project.

Figure 18.23 All-terrain Vehicles, Planned for Operation at the Mine Site

Ural 4320

Ural 32552

Ural ATZ Tanker

KRAZ 65032

The mobile fleet shown in Table 18.14 will be used on site as part of the process plant and infrastructure operations.

Table 18.14 Processing and Infrastructure Mobile Fleet

Equipment	Unit	Quantity
Wheel Loader 3.5 m³ Bucket	No	1
Backhoe Loader 1.3 m³ Bucket	No	1
40-t Truck Mounted Crane	No	1
2-t Forklift	No	1
25-t Mobile Crane (already purchased)	No	1
Site Personnel Carriers (Kamaz)	No	2
Lighting Plant	No	10
Mobile Crusher	No	1

18.7 PROCESSING FACILITY

18.7.1 GENERAL APPROACH

The design of the processing facility takes into account the remote location of the mine site and the logistical constraints of the winter road transport window. The general approach is to ensure that machinery and equipment located outside of the building is specified to Arctic grade to allow for operation and maintenance in the extreme cold environment.

The location of the processing facility was chosen to be as close to mining operations as possible to minimise haul distances whilst taking into account the geotechnical conditions and topography of the mine site.

The process building will be winterised and heated to keep the internal temperature within 10 °C and 35 °C depending on the season. The majority of the process equipment, apart from the crushing circuit, will be located within the process plant building.

The main process equipment will be packaged systems supply within defined battery limits as far as possible.

Packaged equipment will be tendered for by the industry standard mining equipment manufacturers, as well as Russian manufacturers. Where possible, provided the standards set out in the equipment specifications and datasheets are met, equipment will be procured from Russian manufacturers to keep capital, lead time, and transport costs to a minimum and to reduce the tax burden. Suppliers will generally be left free to use their expertise to recommend equipment in order to meet the process parameters specified.

Equipment throughout the process plant will be standardised as far as is practical to give commonality of spares and the opportunity to hold fewer unique spares items.

Equipment will be laid out to keep pipe and cable runs as short as possible, whilst maintaining the operability and maintainability of the plant. The processing facility is run on low-voltage power, therefore, the positioning of the motor control centres (MCCs) within the building is crucial to achieve this aim.

The plant and buildings will be laid out to ensure that personnel do not have to unnecessarily venture outside and where necessary to keep distances to a minimum.

18.7.2 PROCESS PLANT BUILDING

The main process plant building will be an 84 m by 24 m portal frame structure with an 84 m by 12 m lean-to structure attached to the southeast side of the main building. The building will be situated in a northeast to southwest direction to take into account the prevailing north-easterly winds. Both the main portal frame and the lean-to are clad with Russian specified and supplied sandwich panels to provide insulation and weather protection.

The main 24-m-wide structure will have an electric overhead travelling crane running the full length of the building, with a 20-t safe working load and a 5-t secondary hoist.

A 150 m^3, steel-constructed, mill feed bin will be attached to the main building at the southwest end of the building to provide buffer storage between the crushing and grinding operations. The bin will be attached to the building and insulated to allow heat from the building to thaw the ore and prevent it from freezing solid within the bin.

The building will house the process equipment downstream of the mill feed bin, reagent mixing, compressor and blower house, tailings handling, small containerised warehousing and workshop, control room, MCCs, and office and change facility.

The office and change facility will be a two-storey enclosed structure located in the lean-to section of the main building, between the reagent mixing and electrowinning process areas. It will be a steel structure with concrete block walls and poured concrete flooring. The concrete blocks will be produced locally on site.

Silver is precipitated from the pregnant solution in an electrowinning process to produce a high-grade silver slurry, which will be dried to produce silver powder. As a result, the area that contains this direct electrowinning equipment, and the adjacent silver room, will be secured and access controlled. The silver room will be located on the ground floor of the office structure and will contain the dewatering, drying and storage facilities, as well as an office for the foreman. The security office will control access to the silver room and will overlook the direct electrowinning area. A dedicated door will allow for secure vehicular access to the silver room for secure loading of the transportation vehicle. The silver powder will be flown to Yakutsk for onward transportation to the refinery once a week; two weeks' worth of storage will be provided to allow for bad weather.

The remainder of the office complex will house process and administration offices on the first floor and change facilities on the ground floor.

On the northwest wall of the process plant, a two-level steel structure will support two MCCs on the first floor, allowing cable access from the ground floor underneath the MCCs, and the control room and laboratory on the second floor, overlooking the grinding and leach sections.

There will be three, 40-foot shipping containers located on the ground floor of the northwest wall of the process plant containing a basic workshop in one, and day to day maintenance stores in the other two.

A roll shutter door in the northern corner of the process plant on the northeast wall will allow vehicular access to the process plant laydown area. The truck bay and laydown area will be serviced by the electric overhead travelling crane. During the colder months, the truck bay and laydown area will be used to park some of the mine fleet in an emergency mine shutdown.

Separate, dedicated vehicular access to the reagent mixing facility will be located in the south corner of the building on the southwest wall. This access will be used to bring the shipping containers used for reagent storage into the reagent mixing and storage area for unpacking as required.

The cyanide mixing and storage room will be an enclosed, separately ventilated room, separated from the reagent mixing area and other plant areas by a concrete block wall. There will be no acid reagents or consumables in the vicinity of this area. Access to the cyanide mixing room will be from outside of the process plant via a dedicated door on the southeast wall to allow the cyanide containers to be brought into the room.

There will be vehicular access to the compressor house from inside the building to allow for maintenance. There is also pedestrian access to the office complex on the southeast wall, and outside to the crusher on the southwest wall, all at ground level.

The building will be heated by a 3.7-MW diesel fuelled boiler, housed in a separate building adjacent to the northwest wall of the process plant building. The boiler will be located next to the process plant power generators to allow for waste heat from the generators to be used, via a heat recovery system, to pre-heat the water prior to entering the boiler.

18.7.3 ROM Pad and Crushing

The ROM pad and crushing facility will be located outside at the southwest end of the process plant. The mine haul trucks will carry ROM ore to the ROM pad, which is comprised of three graded piles. The ore will then be fed into the crusher plant by a front-end loader. During periods of dry weather, as required, ROM pads and stockpiles will be sprayed with water to supress dust emissions.

The crusher facility will comprises two skid mounted modular crusher units and a fine ore bin feed conveyor. The skid mounted units will be bolted to a reinforced concrete slab.

The first skid mounted unit will contain a vibrating grizzly, feed hopper, vibrating feeder, jaw crusher, and discharge conveyor. The jaw crusher will operate in an open circuit and the product will be discharged into the second skid mounted modular crusher unit via the discharge conveyor.

The second unit will contain a feed hopper, double deck vibrating screen, cone crusher feed conveyor, cone crusher, cone crusher discharge conveyor, and screen discharge conveyor. The cone crusher and screen will operate in closed circuit with the screen undersize product reporting to the fine ore bin via the fine ore bin feed conveyor. The crushing plant will be able to crush at a faster rate than the process plant requires. A diverter chute will be installed to divert the crushed ore when the fine ore bin is full. The crushed ore will be stockpiled to give buffer storage and allow continued operation when the crushing plant is down for maintenance.

The crushing facility will be an outside operation. A heated, elevated, containerised operator's hut overlooking the skid mounted units will be located to the east of the crusher units between the crusher and the process plant, with good visibility of the ROM and front-end loader activity.

The stand-alone fine ore bin feed conveyor will be of covered construction to minimise dust.

18.7.4 The Mill

The layout of the plant was designed to keep the building size as small as possible whilst allowing for effective operation and maintenance of the process equipment.

The ore feed will enter the process plant via a vibrating feeder at the base of the fine ore bin located in the western corner of the process plant, discharging onto the ball mill feed conveyor. The ore will discharge from the ball mill feed conveyor into the ball mill feed chute where it will enter the ball mill alongside the mill classifying cyclone underflow product and process water.

The ball mill is a 3 m diameter by 3.9 m, 500 kW ball mill located on a concrete foundation, which brings the mill centreline to 4.1 m above the top of the plant floor concrete slab. Access to the mill will be via steel walkways at the top of the concrete mill foundation.

The ball mill feed chute will run on rails set into the concrete foundation at the feed end to allow it to roll away from the feed head for access inside the mill. Access will be required to pass materials and equipment for liner change and mill maintenance inside the mill. Two manways, 180° apart, will be provided in the ball mill shell to allow for personnel access to the mill. Liners will be handled inside the mill by basic liner handling equipment such as the RME MILLMAST Powershift or equivalent.

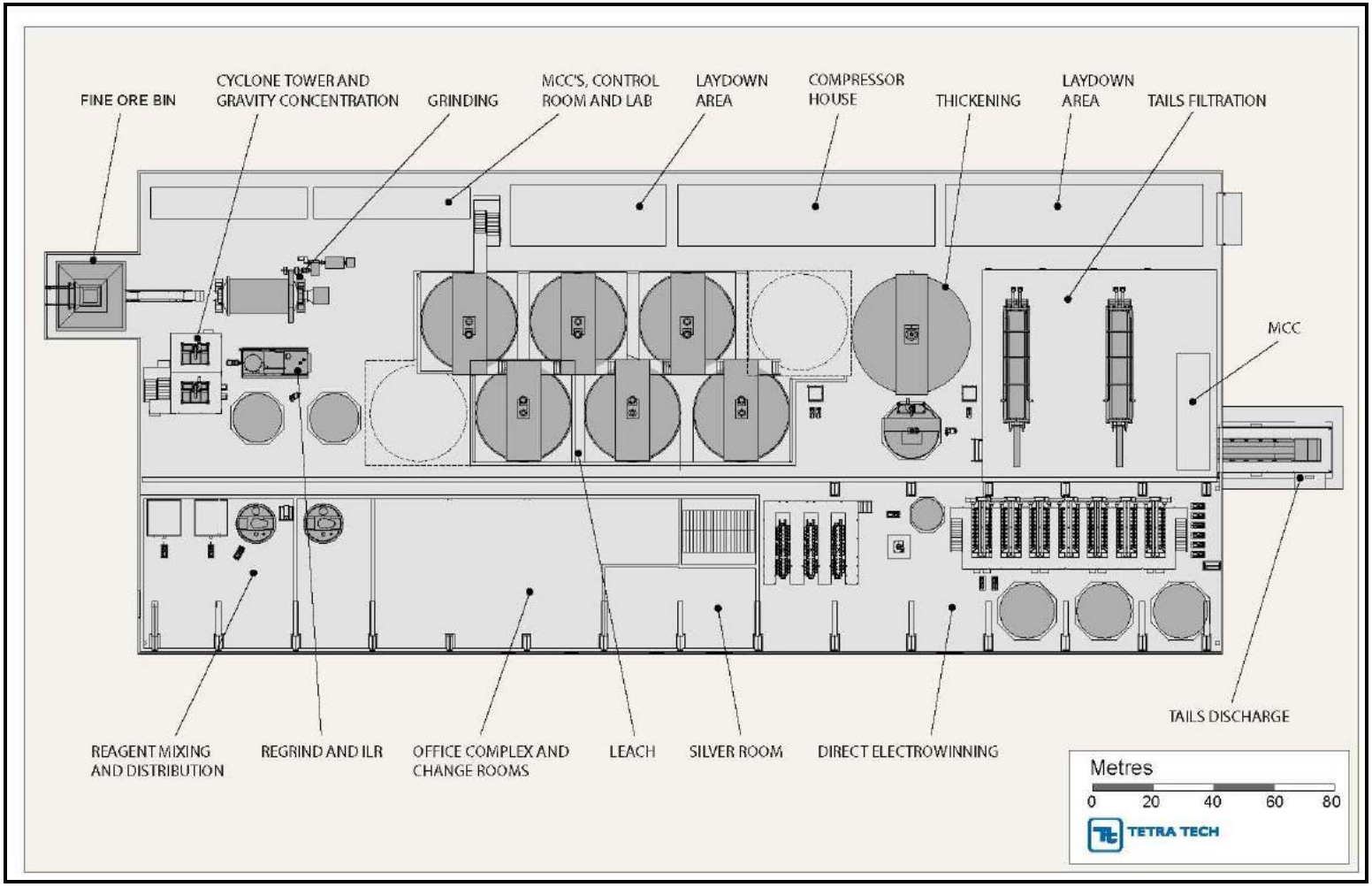
The ball mill will be charged with 90 mm diameter steel balls and must be topped up at a rate of 1 kg of steel balls per tonne of ore processed by the mill, which equates to a feed of five balls per hour. These will be loaded by hand into the feed chute from a storage container adjacent to the feed chute and will not require any ball charging equipment such as kibbles or automatic ball release mechanisms.

The cyclone tower will be located adjacent to the mill foundation, housing the classification and dewatering cyclones, gravity trash screen, and gravity concentrator. On the ground floor level the associated tanks, pump boxes, and pumps will be installed as well as the ILR. The cyclone tower will be a three level steel structure linked to the mill platform by interconnecting walkways and allowing for slurry to flow by gravity through the downstream equipment.

The leach circuit will be made up of six leach tanks. The ball mill will be sized to provide an upside in throughput of 25 to 30% given the current predicted ore characteristics. As each of the six leach tanks represents 17% of the leach capacity, space for two additional leach tanks, one at the start of the leach circuit and one at the end of the circuit, will be required to match the upside in the ball mill. Having eight tanks installed will give a retention time of 120 hours, which will allow both the future fresh sulphide ores and oxide ores to be processed in the same stream. The leach tanks will be 7.7 m in diameter by 7.8 m fabricated steel tanks designed to support the tank top walkways, agitators, gearboxes, and ventilation hoods.

The leach tanks have been arranged to allow the tanks to be of identical design. The design of the tanks includes an adjustable weir on the discharge box to allow for the gravity flow through the circuit to be regulated. Each tank will sit on a concrete ring beam foundation, but the design of the weir will allow each pair of tanks to sit at the same level, as opposed to having each tank foundation at its own individual level. This will simplify the foundations, saving construction costs and improving the operability and maintainability of the plant. Each leach tank will have a glass reinforced plastic fume hood, overhanging the tank by 100 mm and connected to an extraction fan via a common duct. The extraction fan will create a low pressure in the fume hoods to draw air in from inside the process plant and ensure that any hydrogen cyanide release is exhausted via the ducting and through the exhaust stack to the outside environment.

Figure 18.24 Process Plant Layout



The leach solution will report to the thickener via a clarifying cyclone. The thickener will be conventional 9 m, high-rate thickener. The thickener overflow will report to direct electrowinning and the thickener underflow will report to the filter presses for tails dewatering. The liquor from the filter press dewatering will report back to the thickener feed. The thickener will be located at the end of the leach circuit and will sit between the leach circuit, filter press, and electrowinning sections to keep the piping distances to a minimum.

The direct electrowinning package will be modular package containing a set of 140 powder cells and 84 polishing cells. The cells will be mounted on their own steel structure with the headers and manifolds provided by the vendor. The steel structure will be chemically anchored to a reinforced concrete base. The process requires three tanks to enable continuous operation. One tank will be receiving pregnant leach solution, one tank will be cycling through the powder plant and one tank will be cycling through the polishing plant. Each tank will have a capacity of 94 m^3 based on the assumption of changeover of tanks every eight hours and a silver concentration of 1,000 ppm in feed solution. The recommended material of construction for the tanks is fibre reinforced plastic.

The thickener underflow will be dewatered in two, 35 plate filter presses. The filter cake will be sent to the TMF and the liquor will report back to the thickener. Two smaller filter presses were selected over one larger filter press in order to level out the flow of tails downstream, as well as the opportunity for combined spares and staged maintenance. The filter presses were sized to include a cake washing cycle to ensure the highest recovery possible of the silver in solution. The filter presses will sit on a raised structural steel platform to allow for the discharge conveyors to run under the filter presses. Space has been allocated in the design for a third filter press on the platform, as an additional filter press might be required when the fresh sulphide ores are processed due to the finer grind size, or if the throughput of the plant is increased.

Figure 18.25 Electrowinning Concept Drawing

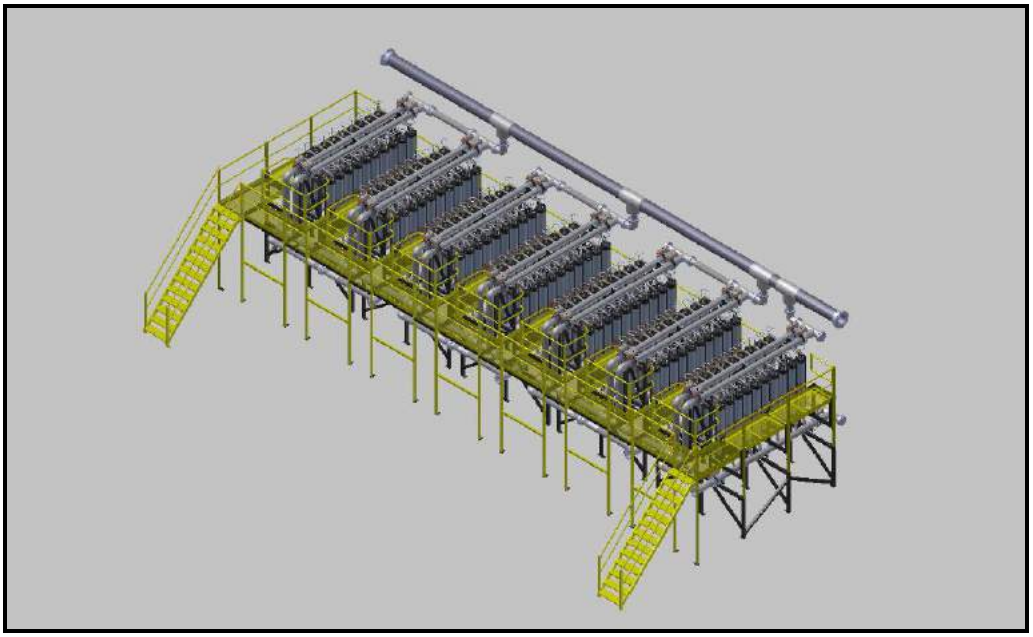


Table 18.15 Equipment Packages

Package	Battery Limits	Scope
Ball Mill	• cyclone underflow into the ball mill feed chute • trommel screen discharge	• ball mill, feed chute, lubrication units, bearings, trommel screen, motor, hydraulic jacks, gearbox, inching drive, PLC
Gravity Concentration and Intensive Leach Reactor	• dewatering cyclone underflow feed via the gravity trash screen • discharge of pregnant solution to electrowinning • discharge of tailings to the leach feed pump box	• gravity concentrator, all interconnecting piping and valve, control instrumentation, field mounted PLC • intensive leach reactor, all interconnecting piping and valve, tanks, pumps, control instrumentation, field mounted PLC
Filter Press	• pressurised slurry feed to filter press feed valve inlet • discharge of filter cake • discharge of process water from filter press	• filter press frame, plates, hydraulics, internal piping and valves, instrumentation, field mounted PLC
Electrowinning	• pregnant solution at the suction of the electrowinning supply pump • discharge of silver slurry from the silver slurry discharge pump	• powder and polishing cells, all supporting structures and access platforms, internal pumps, piping, valves and instrumentation, field mounted PLC and interface with main PLC
Reagents Mixing	• bulk bag reagent feed • discharge flange of reagent distribution tank	• bag breaker, dust hood and ventilation, mixing tanks, distribution tanks, agitators, valves, instrumentation, field mounted PLC

Note: PLC – programmable logic controller

All packages to be supplied top of concrete or top of steel.

The general design philosophy for piping has been to standardise pipe material and pipe sizes where possible to allow for commonality of spares, fittings, valves and pipe equipment. Pipe material has been standardised to ASTM A53 Grade B carbon steel and HDPE PE100 pipe. The largest process pipe diameter on site is 150 mm carbon steel and the most common pipe size is 100 mm diameter HDPE pipe. Piping will be designed to meet with GOST 32388-2013 and plastic piping will be designed to meet with GOST 18599-2001.

Pump boxes, tanks, chutes, bins and other platework, where possible, will be included in equipment package supply. If not available for supply as part of the equipment packages the platework will be fabricated locally on site. Generally, platework will be 6 mm thick carbon steel, with baseplates 10 mm thick, welded with 6 mm continuous fillet or full penetration butt welds. Flanges on chutes will be sealed with silicone sealant. Nozzles and flanges on pump boxes and tanks will be ANSI Class 150. Bolts will be a minimum grade of 8.8.

Conveyors will be specified as complete, turn-key items with top of concrete battery limit. The conveyor package will include all trestles, head chutes, feed chutes, pulleys, idlers, belt, support structure, pull wires, instrumentation, control, walkways and access stairs. Conveyors outside will have wind hoops specified as standard.

Access platforms, walkways and stair towers will be located throughout the process plant to ensure ease of operation and maintenance of the process equipment. Access will take into account emergency procedures and will ensure at least two exit points from access structures with no dead ends. Where stair towers are not feasible for emergency access, cat ladders will provide emergency access and egress from raised platforms.

18.7.5 TAILINGS HANDLING

Dewatered tails will be released from each filter press onto a cake discharge conveyor running under each filter press. The cake discharge conveyor will discharge onto a common conveyor running perpendicular to the filter presses, which will transport the cake through a portal on the northeast side of the building to discharge to a stockpile.

After dumping the ore on the ROM pad, every third ore truck will continue round the process plant building to be loaded with tails from the tails stockpile by a front-end loader. The ore truck will proceed to the TMF, 250 m to the east of the processing facility, and dump the tails in the TMF before proceeding back to the open pit.

18.7.6 PLANT SERVICES

A compressor house will be located in the eastern corner of the building. The compressor house will contain instrument air compressors, an air receiver, and the leach blowers.

Instrument and plant air will be distributed by a 100-mm diameter carbon steel ring main with 65 mm diameter stringers.

For oxide ore, the leach circuit requires 3.18 m^3/min of air at a minimum pressure of 96 kPa(g) at the sparge ring. For future sulphide ore, the leach circuit will require 6.47 m^3/min. The air will be introduced below the agitator blades by a sparge ring to distribute the air throughout the leach solution. Leach air will be distributed from the blower to the sparge rings by a 100-mm diameter carbon steel ring main with 75-mm diameter stringers to the sparge rings.

Process water will be pumped from the process water tank via a 100-mm ring main, supported at a high level around the plant, with 65-mm stringers dropping down to equipment and hose points. The return water will be fed back into the process water tank. Approximately 1 m^3/h of depleted electrolyte will be bled from the process for treatment and filtration to prevent the build-up of weak acid dissociable cyanide, base metals, and total dissolved solids in the recirculated process water.

A containerised treatment plant will be constructed on site and will be housed within the process building. The primary step in the process will be the removal of base metals in the form of metal precipitates. The barren solution discharge, approximately 15 m^3/h, will be conditioned with sulphuric acid and sodium

hydrosulphide in order to precipitate the copper and zinc ions as sulphide metal complexes. The cyanide ions in the solution will be converted to hydrogen cyanide during this chemical reaction which dissolved instantaneously in the process water, as hydrogen cyanide is infinitely soluble. The slurry containing hydrogen cyanide solution and metal sulphides will be pumped into a conical bottomed, static settling tank and dosed with flocculants. The metal sulphide precipitates in the settling tank underflow will be pumped to the filter press feed tank within the process plant tailings disposal area and subsequently discharged to TMF.

The settling tank overflow will be conditioned in another tank with sodium hydroxide in order to regenerate the cyanide. It is anticipated that 90% of the cyanide solution, approximately 13.5 m^3/h, will be recirculated back to the process plant as plant water, reducing water demand and cyanide usage.

A bleed stream of 10% of the process solution, approximately 1.5 m^3/h, will be pumped through a sand filter followed by a series of carbon adsorption columns and cationic and anionic ion exchange (IX) columns in order to remove any remaining contaminants including residual cyanide. The clean water from the bleed stream will be recirculated back to process water tank, used for reagent make-up or in the gravity circuit.

The treatment plant will run year round. During the summer, on average, an additional 3.5 m^3/h of TMF runoff will be processed through the treatment plant. The additional water will be used for reagent make up. Should the tailings runoff marginally exceed the capacity of the water treatment plant, raw water supply to the process plant will reduced and process water used in its place as a temporary solution. Should the tailings runoff increase further, for example, during a 1-in-100-year extreme precipitation event, the runoff will be dosed with calcium hypochlorite in situ, in the TMF runoff collection pond, and mixed using floating agitators and aerators to neutralise the cyanide.

The raw water supply pipe will discharge into an 8-m diameter by 8-m insulated raw water tank, heated with a 15-kW immersion heater. The raw water tank will supply the plant with fresh water, make-up water, top up the fire water tank, and supply the containerised potable water treatment plant with fresh water. The raw water pumping station is detailed in Section 18.16.2.

The potable water will be treated raw water. The potable water treatment plant will be a containerised unit, taking feed from the raw water tank, processing the water and feeding the potable water tank. From the potable water tank, potable water will be pumped round a ring main to the plant facilities, wash rooms and process equipment, namely the ACACIA reactor and gravity concentrator, requiring clean water.

The fire water tank will be located in the water treatment complex alongside the raw water tank and potable water tanks. The fire water tank will be fed from the raw water tank with fresh water. The fire water tank is the feed for the fire water system, which consists of a fire water ring main, fire water pump, jockey pump and backup pump. The fire water ring main will run through the garage warehouse building and process plant building with stringers to provide fire hose points at the required locations.

18.7.7 PROCESS INSTRUMENTATION AND CONTROLS

An uninterruptible power supply (UPS) will supply power to the control and instrumentation system. The control system is designed for semi-automation of the process plant. The control system architecture design consists of a central PLC and a supervisory control and data acquisition (SCADA) operator interface within the process plant, with distributed input/output (I/O) hardware at the various infrastructure sites. The process control includes the following operational modes:

- automatic
- local
- manual.

Every motor is designed to have a field start/stop push-button station to enable local control. Hard wired emergency stops must be provided in locations as chosen by risk assessment.

The instrument design takes into account maximum and minimum operating conditions, desired accuracy, response required, and commonality of spare parts.

It is expected that the control system design will require the involvement of a systems integration specialist.

The infrastructure process control will be designed for a semi-automated control with a minimum amount of operator interaction required. The process plant PLC/SCADA system will control the entire project site, with remote I/O connected back to the central PLC via fibre optic cable.

18.7.8 PROCESSING FACILITY CLOSURE PLAN

Upon closure of the mine, a number of operations will be required to make the plant safe. Structures, buildings, and all equipment will remain in place. All tanks, pipes, and vessels containing potentially hazardous slurries and solutions, including, but not limited to, the leach tanks, electrowinning, reagent mixing and distribution, and pump boxes will be cleaned and flushed thereby neutralising the substances. Oils and coolants will be drained from machinery into sealed barrels. The coolants will be stored in the waste storage facility and oils will be burnt in the process plant boiler.

18.8 SITE FACILITIES

The following section provides an overview of site buildings and supporting facilities. An overall site plan, including arrangement of buildings on the site and layouts for each of the main buildings is presented in Figure 18.5.

18.8.1 GARAGE/WAREHOUSE

Located to the east of, and running perpendicular to the main process plant building is the garage workshop building. The building itself will use an existing 40 m by 20 m by 8.4 m high warehouse building available on site, with a new 20 m by 20 m by 8.4 m high garage extension to make a new 60 m by 20 m building which will be clad

with Russian specified and supplied sandwich panels to provide insulation and weather protection.

The new 20-m garage extension will house three vehicle bays, each with roller shutter door. The bays will be designated a vehicle wash down bay, a welding bay, and a repair bay with an under vehicle access pit. The welding and repair bays will be serviced by a 5-t gantry crane.

The 40-m workshop section of the building will contain a mezzanine floor for storage of lightweight spares. The ground floor of the workshop section will contain general workshop equipment such as hydraulic jacks, drills, sanding machines, pullers, and compressors. In addition to the general equipment there will be specialist sections for tire change and storage, axle maintenance, engine and gearbox maintenance, battery charging and brake maintenance, as well as a foreman's office and washroom facilities.

During periods of extreme weather when mining operations are shut down, the unused vehicle maintenance bays will be used to park some of the mining fleet to prevent them from getting too cold for restart.

18.8.2 Fire Station

The fire station will be located within the process plant complex. The building will be of steel construction and clad with Russian specified and supplied sandwich panels to provide insulation and weather protection.

The building will contain covered parking for the fire fighting vehicles with water tank refill equipment and roller shutter door, shift office, changing room, training room, equipment stores and a mess room.

18.8.3 Operation and Construction Camp Accommodation

The operational camp facilities will be located in the valley bottom next to the Sirelendge River. The operational camp will be an expansion of the original exploration camp. The operational camp will be constructed in two phases. The first phase of camp accommodation will comprise a two storey, 12 m by 49 m, 80 bed accommodation block, mess facilities and food storage for 80 workers, a 128 m^2 bath and laundry complex, and warehousing for housekeeping maintenance spares, food, and consumable storage. The second phase of camp accommodation will comprise a second two storey, 12 m by 49 m, accommodation block configured with 40 beds with the remainder of the building housing rest and recuperation facilities. Water will be supplied via a containerised potable water plant, from the raw water pump station. The camp will have its own 645-kW generator and fuel storage, used to provide power to both the camp and the raw water pump station.

The construction camp will be a 50-person, winter proof, containerised temporary camp with mess and laundry facilities, located next to the process plant.

MEDICAL CENTRE

There will be a medical centre located at the operational camp with a doctor on site. Injured and sick workers will be transported back to the camp in site vehicles for treatment.

18.8.4 AIRSTRIP

The current airstrip allows access to site via helicopter or an Antonov AN2 fixed wing aircraft. The runway is approximately 800 m long and can be extended to 1,200 m. As neither of the aircraft mentioned can fly above 3,500 m in altitude, and the site airstrip does not have telemetry, lost flying days are estimated to be between 50 and 75% due to poor weather conditions. To mitigate for the risk of lost flying days, the shift rosters will be staggered and the secure silver store room will have the capacity to hold several week's production of silver powder.

18.8.5 HELICOPTER PAD

A helicopter landing zone has been designated at the camp area and clinic. Helicopters will be used for medical emergency evacuation.

18.8.6 SITE FACILITIES CLOSURE PLAN

Buildings and equipment, including generators, will remain on site. Any oils, hydraulic fluid, coolant, and other hazardous substances will be drained from equipment and stored in sealed barrels. Oils will be used as fuel in the diesel boilers. All other fluids will be stored in the waste storage facility in sealed barrels. Any equipment with stored energy will be dismantled or made safe.

For a period of at least one year, a small team of personnel, accommodated in the local village of Sebyan-Kyel, will monitor the site on a weekly basis and take samples to ensure that the storage facilities and mine closure measures are effective.

18.9 REMOTE SITE FACILITIES

18.9.1 OFFICES

The Project will be remotely supported by management, administration and engineering teams based in an office located in Yakutsk. These functions do not need to be based at site incurring accommodation, travel, and rotational costs and will only fly out to site as required.

18.9.2 REFINERY

Dried silver powder in sealed and tagged 30 l HDPE drums, flown from site, will be transported directly to a third party refinery from where doré metal will be sold directly. Once the silver metal powder arrives at the airstrip near Yakutsk, no handling (other than off-loading from the plane) or storage of silver metal by Prognoz will be required.

18.10 TAILINGS MANAGEMENT FACILITY

Silver Bear commissioned SRK to undertake the feasibility engineering design for the TMF for the Project.

SRK undertook a TMF site selection study, in which a series of four alternative locations were assessed for development. A waste rock and tailings co-disposal option was also considered as part of this assessment.

The preferred location for TMF development is indicated in the site layout in Figure 18.4.

The TMF will be constructed 0.2 km northeast of the plant site and will cover an area of 7.69 ha. Approximately 0.8 Mt of tailings material will require storage over the LOM.

The full TMF study completed by SRK is located in Appendix J.

18.10.1 TAILINGS PROPERTIES

Tailings will be dewatered at the plant to produce a filter cake, with a minimum solids content of 85% w/w. This dewatering methodology was selected by Tetra Tech and SRK to minimise potential impacts on the environment and simplify materials handling.

As the tailings material was identified as having potential for metal leaching, the TMF has been designed as a fully-lined facility. There will be zero discharge of process affected fluids. To achieve this goal, runoff and any seepage water will be collected in the clarification pond, located at the eastern end of the TMF. The water will then be processed in the treatment plant and used as make-up water in the process circuit.

Limited materials testing has been carried out on representative tailings samples, which indicate that the material consists of predominantly fine grained material.

18.10.2 TMF DESIGN

The TMF consists of two components:

- filtered tailings (or dry cake) storage area
- clarification pond

The waste storage area of the TMF will consist of a high-density polyethylene lined pad surrounded by a rock fill perimeter berm. The perimeter berm will vary in height (maximum height above natural ground of 6 m) and its function will be to separate the clean surface water (runoff water from the surrounding topography) from the process affected water (runoff water from the dry cake storage area).

At the eastern toe of the TMF waste stack, a clarification pond will be excavated below the original ground to the depth of approximately 3 m and the associated perimeter containment dam will be limited to a height of approximately 9 m above

original ground level. The clarification pond will be fully lined with the seepage collection sump at the downstream end of the contained area. Because of the limited height, the perimeter containment dam will be built of rock fill lined with HDPE. Neither internal filters nor a clay core will be constructed within the containment dam section. The rock fill used in its construction should be of sandstone origin (to limit the percentage content of carbonate rock).

18.10.3 TMF Operations Strategy

Tailings will be delivered to the storage site in form of dry cake at 12 to 15 % moisture content using a single 20-t haul truck sourced from the mining fleet. Tailings will be loaded from the stockpile outside the plant onto the truck using a dedicated JCB 4CX ECO backhoe loader (1.3 m^3 capacity). Upon deposition at the TMF site, tailings will be spread in nominally compacted by a D7 dozer (also sourced from the mining fleet). Dozer traffic over previously deposited tailings will aid of a process of compaction and negate the requirement for specific compaction equipment.

The initial and first lift on the top of the HDPE liner in the stacking area should be 1 m thick to protect the liner, following on from this lifts of maximum 0.3 m thickness to aid improved compaction. SRK has taken into account the effect of freezing of tailings during volumetric calculations.

SRK used data obtained from a geotechnical investigation at the site (performed by others) to characterise the foundations beneath the TMF. In addition, limited test work carried out on representative tailings samples has been completed (by others), which has been used by SRK to characterise the shear strength properties of the filtered tailings for design. Based upon this data, SRK has completed slope stability analysis to confirm that the TMF waste slope factor of safety is acceptable under appropriate static and seismic loading conditions.

The TMF construction risks include inability to install the synthetic liner in low temperatures (in winter). If the construction works are not properly scheduled or delayed then the completion of the TMF may be postponed until the spring or summer months.

18.10.4 Closure and Rehabilitation Plan

A closure and progressive rehabilitation plan has been completed for the TMF, which incorporates the following components:

- re-grading of the TMF to ensure all external slopes are less than or equal to 1V:3H
- placement of 1.5-mm HDPE geomembrane across the stored tailings material
- placement of a geotextile protection layer above the HDPE
- installation of a 1.0-m waste rock layer for erosion protection
- installation of a 0.2-m topsoil final cover.

The final tailings surface has been designed to encourage shedding of meteoric runoff to the perimeter of the structure.

Monitoring provisions have been included for both the operational and closure period, which include standpipe piezometers for water quality monitoring.

18.11 WASTE ROCK DUMP, ORE STOCKPILE AND OVERBURDEN STORAGE AREA

The three types of soil and rock streams that will require temporary storage or permanent disposal at selected locations across the mine site are: topsoil (generally weathered rock fragments), ore, and waste rock. The selected locations for the temporary storage or permanent disposal of these materials are shown in Figure 18.4.

The selection of appropriate locations for the storage of these materials is one of the most important aspects of the mine layout design. The selection process is largely influenced by:

- environmental constraints/impacts
- material characteristics
- design volume of the ore/rock streams
- storage capacity
- site availability
- capital, operating, and closure costs
- geotechnical, geological, hydrological and hydrogeological conditions
- ease of the day-to-day operation of mining and processing.

Other site selection considerations include:

- topographic boundaries
- location of ore bodies (open pit/underground shafts)
- surface water management and diversion
- available area and expansion potential
- proximity to existing infrastructure elements
- distance and elevation from the open pit and from the processing plant
- seepage control
- climate
- suitability of seepage and runoff collection pond(s)
- seismic conditions
- sub-grade soil and rock conditions.

18.11.1 ORE STOCKPILE

Ore from the open pit and underground mine will be transported to the process plant by dump trucks. Prior to crushing, the ore will be stockpiled in an area located immediately southwest of the plant, as shown on Figure 18.4. The ore stockpile site is generally flat and has firm ground, and comprises weathered rock fragments over shallow, solid bedrock. No ore stockpile instability is anticipated, as long as the side slopes of the individual ore piles are 1.5H:1V, or shallower.

According to project details, ore will be stockpiled in a heaped formation in four individual areas. Three smaller heaps will be formed to store run of mine ore of different grades. The estimated volume of each small pile will be 460 m^3 and each will individually cover an area of approximately 300 m^2. The use of smaller, individual ROM stockpiles is critical to the mining operation in order to supply product of the correct specification, and smaller ROM stockpiles are usually built depending on the product grades, derived from the open pit faces. Next to the three ROM piles, immediately to the northwest, is the largest of the four stockpiles where the ore will be placed next to the primary crusher in a heaped pile of up to 12,300 m^3 in volume. The approximate area of this primary location will be 5,600 m^2.

The ore stockpiles will be located next to the process plant, on the top of the plateau; therefore, no runoff water is expected to come into contact with the dumps, and only a limited amount of precipitation will enter the voids of the ore, which will eventually end up in the process plant. As a mitigating measure, contact water around the dumps will be collected in ditches and re-used in the process plant via the TMF. Such ditches may be formed around the individual ore stockpile heaps. Based on the available weather data, the anticipated amount of such water will practically be negligible. Any water collected will be transported to the TMF via mobile water bowser.

18.11.2 OVERBURDEN STOCKPILE

Selected areas on site, impacted by mine development, will require clearing of any soil overburden. Such overburden generally comprises layers or mixes of clay, silt, sand, gravel, cobble, and boulder sized rock fragments and soil particles. At the Project site, the overburden generally comprises gravel and cobble sized rock fragments, with limited amount of fine (silt and clay) particles. It is likely that the finer soil fractions have washed down from the top of the mountain and mountain slopes by occasional runoff water during short summer storms. Prior to pouring foundation concrete for the individual buildings, and prior to the start of the open pit, all soil overburden will be removed from those areas and stockpiled for potential re-use during mine closure. Since the largest amount of such soil mass is expected to come from the open pit area, the temporary storage place for the topsoil will be next to the open pit, as shown on Figure 18.4.

The available volume for topsoil storage is 205,790 m^3, while the area of the topsoil heap's footprint is approximately 35,640 m^2. The ground surface at the topsoil storage site slopes somewhat toward south-southeast. A runoff collection trench will be formed around the storage heap in order to facilitate the collection of any storm water and convey it into the seepage/runoff collection and settling pond, which will be constructed at the front of the proposed WRD. The primary purpose of the settling pond will be to capture all solid particles conveyed into the pond by precipitation and

to prevent those solid particles from entering the Borisovsky Valley. It must be pointed out that due to the limited relevant surface area, the anticipated volume of run-off water is practically negligible.

18.11.3 WASTE ROCK DUMP

LOCATION

The waste rock mass generated during the excavation of the open pit and advancement of the underground mine shafts will be placed into a short side valley, located immediately east of the open pit (Figure 18.4). Trees currently cover parts of the side slopes in the valley selected for the WRD; however, these will be removed. Once the dump is closed, new trees will be planted on the top and on the new slopes of the WRD.

At the front of the proposed WRD, next to the junction of the side and the main Borisovsky valleys, a seepage and runoff water collection/settling pond is proposed for construction. The crest elevation of the perimeter berm around the pond is proposed at the 1,039 masl elevation, while the maximum water level in the pond is anticipated at the 1,035.5 masl elevation.

The soil conditions in the Borisovsky Valley can be characterized by relatively shallow overburden over solid rock. The soil overburden generally comprises a matrix of gravel, cobble and boulder sized weathered rock fragments, with some silt and clay filling the voids of the large, angular cobbles and boulders (Figure 18.26). In order to prevent any potential instability and unwanted settlement of the proposed perimeter berms around the settling pond, it is recommended to remove the existing soil overburden from the footprint of the berms and to construct the berms from well-graded, clean rock fill, comprising gravel to cobble sized waste rock fragments. The side slopes of the berms using such material are generally stable, with slopes of 1.5H:1V; however, considering that water flow during storms is potentially faster, the recommended slope inclination should be 3H:1V or shallower. Since fine-grained soil fraction (fine sand, silts, and clay) is available only in limited quantities within the voids of the generally coarse grained matrix, there is no continuous natural filter layer in the area to provide the required turbidity barrier within the settling pond. Therefore, the berms' upstream face and the base of the pond may potentially require the placement of a Type 3 silt barrier in order to prevent any solid particles leaving the pond and entering the Borisovsky Valley. As it was pointed out earlier, the soil overburden across the plateau and side slopes of the mountain comprises clean gravel to boulder sized rock fragments. The anticipated amount of fine soil particles in the settling pond will, therefore, be negligible.

Figure 18.26 General Soil Conditions at the Floor of the Borisovsky Valley (Cobbles and Boulders with Some Silt and Clay Filling in the Voids)

Geochemistry of Waste Rock

Geochemistry tests were completed by the Russian Laboratory of Geoelectrochemistry No. 571, under the supervision of Institute of Oil and Gas Geology and Geophysics on representative waste rock and tailings samples to evaluate the neutralizing potential of these materials using Sobek's method. As part of the testing program, experiments of intense oxidation on 26 selected samples (24 waste rock samples and 2 tailings samples) were completed, together with the evaluation of acid-generating potential (AP) and neutralizing potential (NP) of overburden rocks and tailings. The following paragraphs summarize the test results, provided by the Geoelectrochemistry Laboratory.

Reportedly, the tests resulted in preliminary findings regarding the neutralizing capacity and composition of waste rock, and possible ingredients of drainage water, which may be formed as a result of the interaction of seasonal precipitation with the waste rock within the proposed dump.

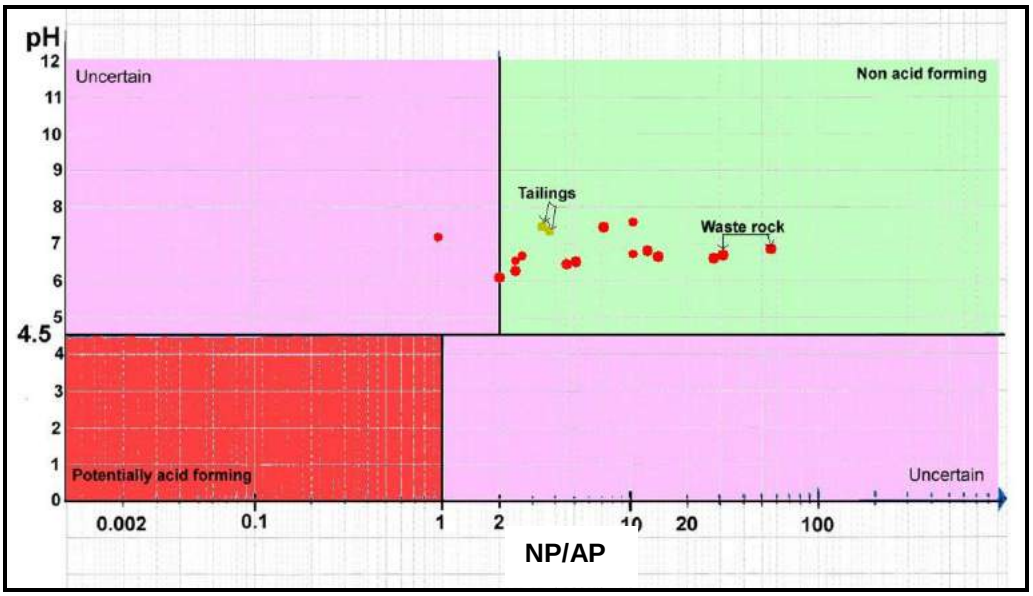
The amount of sulphur in the tested rock samples was relatively low and exceeded 1% in only three rock samples and in both of the two tailings samples. In nine samples the sulphur content was greater than 0.5%. In the remaining samples it did not exceed 0.5%, while in seven samples sulphur content was below the detection limit of 0.01%. The distribution between various forms of sulphur (sulphate and sulphide) in sulphide rocks was sporadic; however, sulphide sulphur prevailed in the tailings samples.

The carbonate content in the waste rock samples varied widely from less than 0.01% to 10% carbon dioxide. In 16 samples the carbonate content exceeded 2%, but at the same time, in 16 other samples, it did not exceed 1%. In general, on the basis of definition of sulphur forms, the preliminary conclusion is that the overburden rocks have relatively high potential for in-dump neutralizing of any acid that may potentially be formed during oxidative dissolution of any sulphides that may be present in some

waste rock streams. The analysed overburden rocks and tailings samples of the Vertikalny deposit that will be placed into the WRD and TMF, are characterized by a high-neutralizing potential, as shown in Figure 18.27. The chart in Figure 18.27 presents the relationship between pH, AP and NP in representative waste rock and tailings samples.

More than 50% of the samples (23 rock samples and both tailings samples) indicated that they are not acid producing, with a high percentage of calcite content in the tested samples. In fact, in three of the samples the calcium carbonate content exceeded 100 kg per 1 ton of rock. There were a small number of test results that indicated uncertainty in connection with neutralizing potential; however, none of the tests indicated specific potential for acid rock drainage (ARD) in the waste rock.

Figure 18.27 Relationship between pH, AP and NP in Representative Waste Rock and Tailings Samples (based on tests results provided by the Geoelectrochemistry Lab)

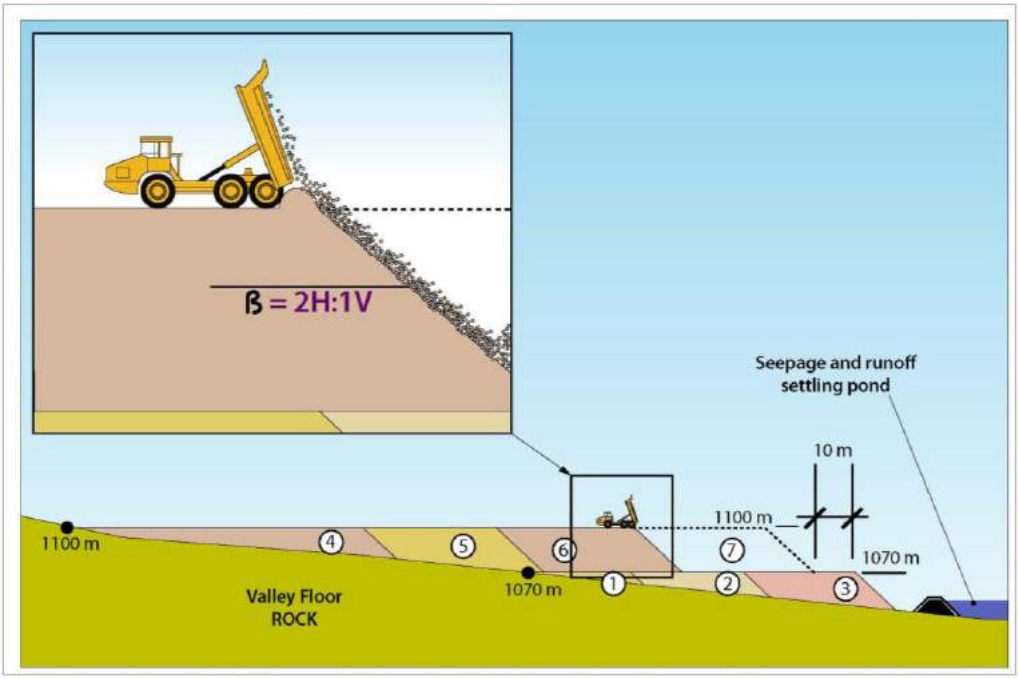


Placement of Waste Rock and Formation of the Waste Rock Dump

The proposed WRD will store up to 2,765,000 m^3 of blast rock material, removed prior to the extraction of ore from the open pit and underground mine shafts. Once the mining operation and waste rock disposal operation is complete, the WRD will attain a maximum height of 135 m, with a footprint of approximately 114,260 m^2 (11.4 ha). The toe line of the WRD will be located at the 1,040 masl elevation, close to the mouth of the side valley, next to the Borisovsky Valley.

The waste rock will be placed into the side valley in stages. The suggested sequence of the stages is shown in Figure 18.28. The placement of the waste rock shall start at the 1070 m elevation (Zone 1) and continue toward the runoff collection pond. The northern and southern sides of the dump will be backed by the existing stable slopes of the adjacent hills, while the front of the WRD (east side) will be formed with stable, benched slopes.

Figure 18.28 Staged Deposition of Waste Rock, Starting at the 1,070 m Elevation Level



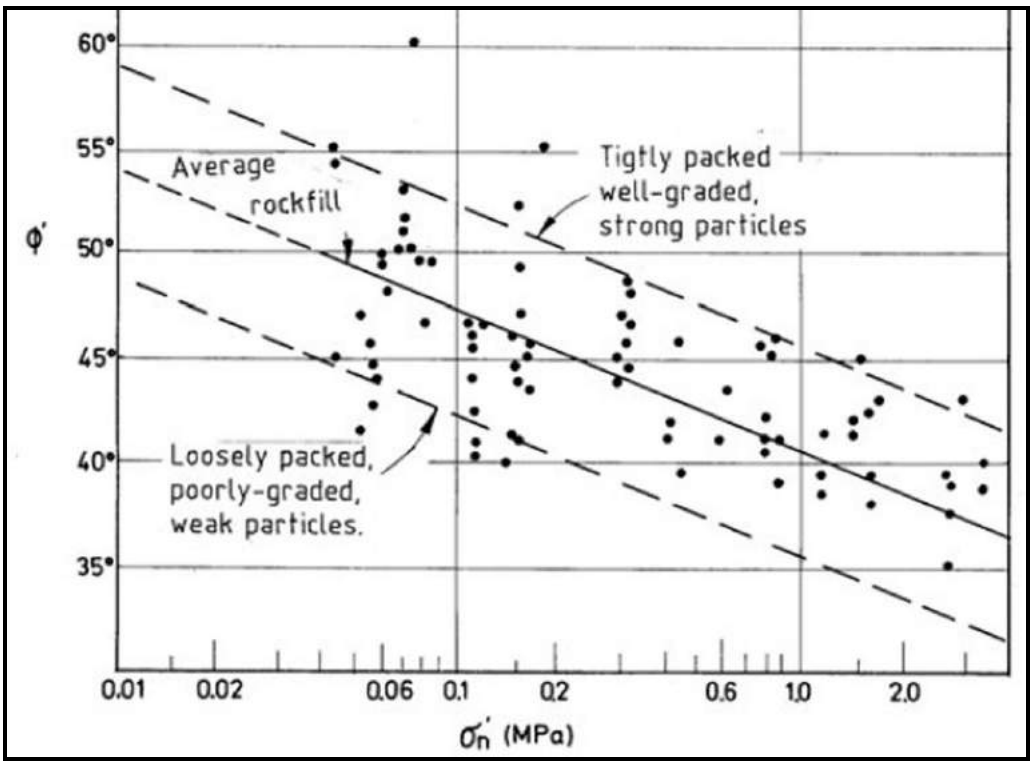
The inclination of the final face of the WRD (zones 3, 7, etc.) will be 2H:1V. That slope will remain stable, even during any potential earthquake that (in the unlikely event) may be encountered in the general area of the mine site. As noted, the potential for any seismic activity in the Project area is negligible; however, an appropriate horizontal seismic co-efficient should be considered when selecting a safe slope inclination for any soil or rock slopes.

The WRD will generally comprise gravel, cobble, and boulder sized blast rock fragments. The internal friction angle of such waste rock material may vary between 35° and 55°, as shown on Figure 18.29. Considering the fill may possibly be end dumped without additional compaction, the estimated friction angle used within the waste mass may conservatively be 42°. Therefore, the factor of safety (FOS) against potential sliding within a 2H:1V slope will be 1.77 at static conditions, well above the minimum value of 1.5.

The maximum height of each lift of the waste rock must be limited to 30 m. At the face of each lift of the WRD, a bench must be formed by stepping back 10 m, as shown on Figure 18.29.

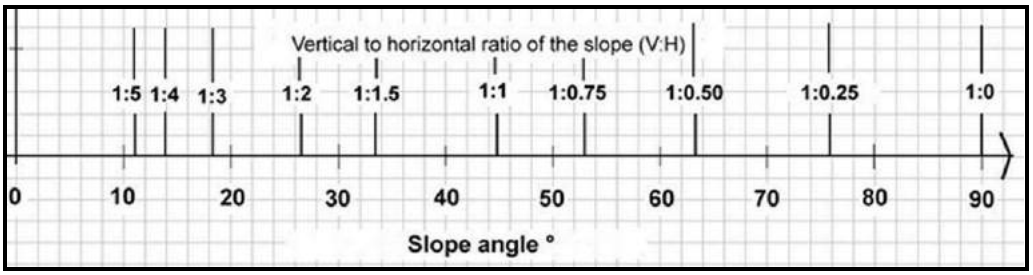
Representative slope inclinations are shown on Figure 18.30.

Figure 18.29 Variation in Internal Friction Angle within a Rockfill Mass



Source: Leps (1970)

Figure 18.30 Representative Slope Inclinations and Slope Ratios (V:H)



As pointed out above, it is anticipated that the waste rock pile will be formed by end dumping the rock fill using rigid frame rear-dump trucks. With such an operation, the larger boulders and cobbles generally roll down on the slope and build up the bottom part of the rock pile. The smaller sizes generally remain within the upper zone. Having smaller particles "blinding" the voids within the larger rock fill cobbles and boulders will help to minimize potential acid generation and metal leaching within the waste rock mass. When the large voids within the rock fill are filled with smaller particles, the potential air movement and seepage of precipitation within the rock fill is reduced, minimizing any metal leaching within the waste rock mass.

Reductions in the volume and rate of flow of drainage, especially through potentially acid generating (PAG) materials (if encountered in the pit and placed into the WRD), will maximize NP effectiveness and reduce metal loadings in the immediate vicinity of the WRD. The physical properties and configuration of PAG and non-potentially acid generating (NPAG) materials within the waste rock can also be used to minimize the leaching of PAG strata. As described previously, more than 50% of the tested

samples indicated that they are non-acid generating. One of the best available techniques to minimize any potential acid generation and metal leaching within the WRD is to place all waste rocks, presently identified as having uncertain NP, within the core of the dump (zones of 1, 2, 4, 5, and 6 in Figure 18.28). All waste rock streams having high degree of NP shall be placed within the stages that will provide the final face of the dump (zones 3 and 7 in Figure 18.28). Diverting any potential runoff water (anticipated to be negligible for all practical purposes) away from the WRD will also limit any potential leaching by incident precipitation. The alignment of the diversion ditch around the WRD is shown on Figure 18.4.

18.11.4 Waste Rock Dump, Ore Stockpile and Overburden Stockpile Closure Plan

The ore stockpile will be depleted during the ramp down of operations with any remaining ore run through the process plant.

During normal operation there will be spare fleet hours for the dozers because during winter mobile equipment will be kept running to stop it from freezing. The dozers will be used for progressive closure of the waste dumps during the ramp down of mining operations. The waste rock will remain in place and vegetation will be planted on the top and slopes of the waste rock dump.

The material contained in the overburden storage area will be used in the remediation of the site on mine closure in areas such as the TMF.

18.12 Site Water Management

A water balance model (WBM) was constructed for the Project using GoldSim, a software package designed to perform Monte Carlo simulations (GoldSim Technology Group, 2010). The WBM predicts the amount of water required to meet the needs of ore processing (milling) operations, underground mining (drilling) operations, and on-site Project staff based on the best available information.

WBM inputs are divided into five types: climate inputs, surface water inputs, human (Project staff) inputs, ore processing and milling-related inputs, and subsurface mining inputs. Each of these inputs is described in the following sections.

18.12.1 Climate

The climate inputs for the WBM consist of specified distributions for the average monthly temperatures at the site. In the absence of any site-specified data, the WBM uses the average long-term monthly temperatures reported by SRK for the Syuryen-Kyuyel meteorological station, which is the closest station to the mine site (Appendix K). Table 2-2 of the SRK report (Appendix K) provides the minimum (T_{min}), average (T_{avg}) and maximum (T_{max}) monthly temperatures in degrees Celsius. GoldSim requires the minimum, mode (most likely; T_{mode}) and maximum as inputs to define the triangular distributions for temperature. The mode of each monthly temperature distribution was calculated using the following formula:

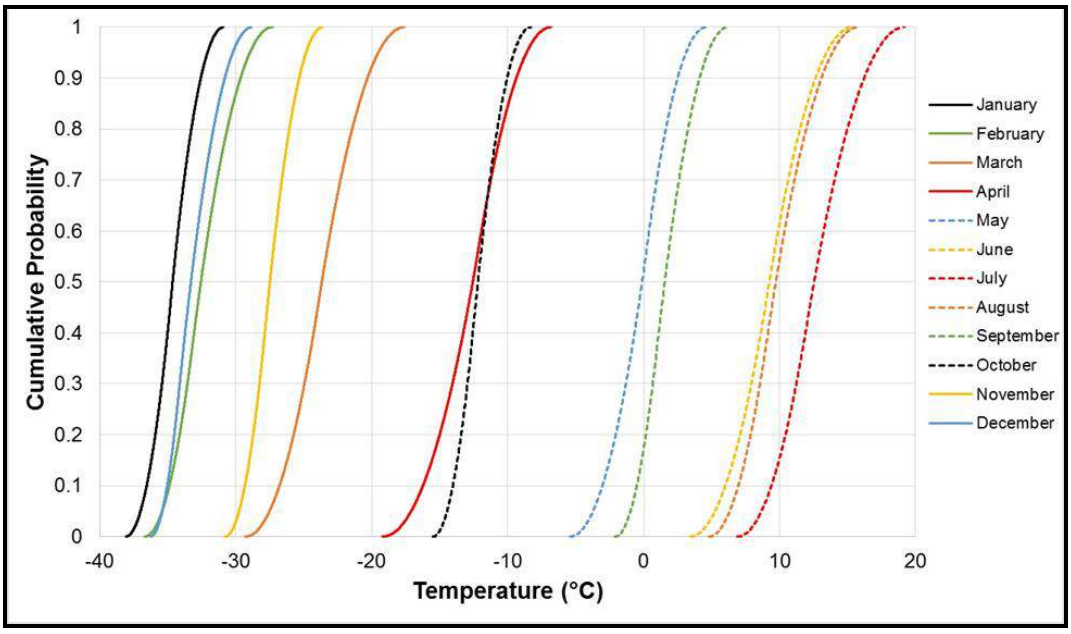
$$T_mode=3*T_avg-(T_min+T_max)$$

The resulting monthly temperature modes, minima, maxima and averages for each triangular distribution are provided in Table 18.16. The resulting temperature distributions are shown in Figure 18.31.

Table 18.16 Average Monthly Temperature Statistics (°C) for the Syuryen-Kyuyel Meteorological Station

Month	Minimum	Mode	Maximum	Average
Jan	-38.1	-34.8	-30.9	-34.6
Feb	-36.7	-33.2	-27.3	-32.4
Mar	-29.3	-23.9	-17.6	-23.6
Apr	-19.2	-12.1	-6.8	-12.7
May	-5.4	0.2	4.6	-0.2
Jun	3.4	9.1	15.4	9.3
Jul	6.9	12.0	19.2	12.7
Aug	4.8	9.1	15.8	9.9
Sep	-2.1	1.0	6.2	1.7
Oct	-15.5	-12.5	-8.3	-12.1
Nov	-30.8	-27.8	-23.6	-27.4
Dec	-36.3	-34.2	-28.8	-33.1

Figure 18.31 Average Monthly Temperature Distributions



Precipitation falling within the pit footprint is assumed to be pumped to the WRD sedimentation pond within 24 hours of precipitation events and will be discharged to the surrounding environment (Appendix C). Therefore, precipitation is not included as an input to the WBM.

18.12.2 SURFACE WATER

Surface water inputs used in the WBM include the Sirelendge River's mean annual discharge and the proportions of this discharge which occur during each month. OOO ISKOM (2015) presented estimates of five percentiles of the mean annual discharge of the Sirelendge River near the Project site. These percentiles were derived from a generalized (i.e., three-parameter) gamma distribution. To generate a full distribution for use in the GoldSim WBM, a generalized gamma distribution was fit to these five percentile values. The recreated distribution (R2 = 0.99966) and the five percentiles from OOO ISKOM (2015) are shown in Figure 18.32. OOO ISKOM (2015) also provides estimates of the monthly distribution of streamflow for an analog river near the Project site (Dulgalakh River at Syuryen-Kyuyel; Table 18.17). These percentages were used in the WBM to divide the Sirelendge's annual discharge between the five months when the river is likely to be flowing.

Figure 18.32 Comparison of the Mean Annual Sirelendge River Discharge Distribution used in the WBM and Select Percentile Values from OOO ISKOM (2015)

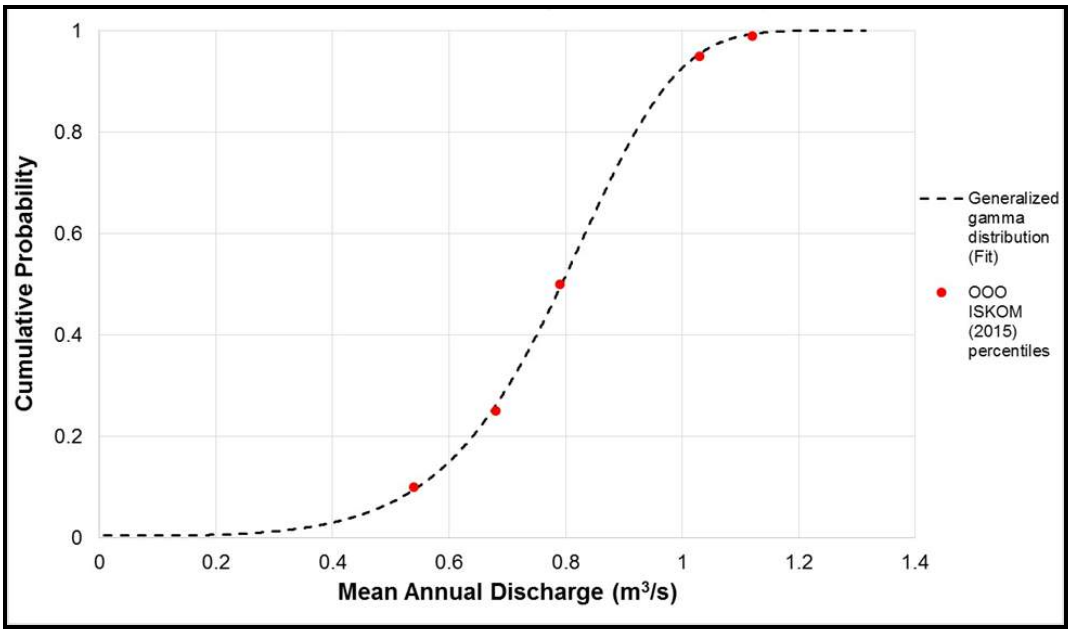


Table 18.17 Percent Mean Annual Flow during each Month, Dulgalakh River at Syuryen-Kyuyel

Month	Percentage of Mean Annual Flow (%)
Jan	0.00
Feb	0.00
Mar	0.00
Apr	0.00
May	0.00
Jun	34.00
Jul	25.80
Aug	28.30

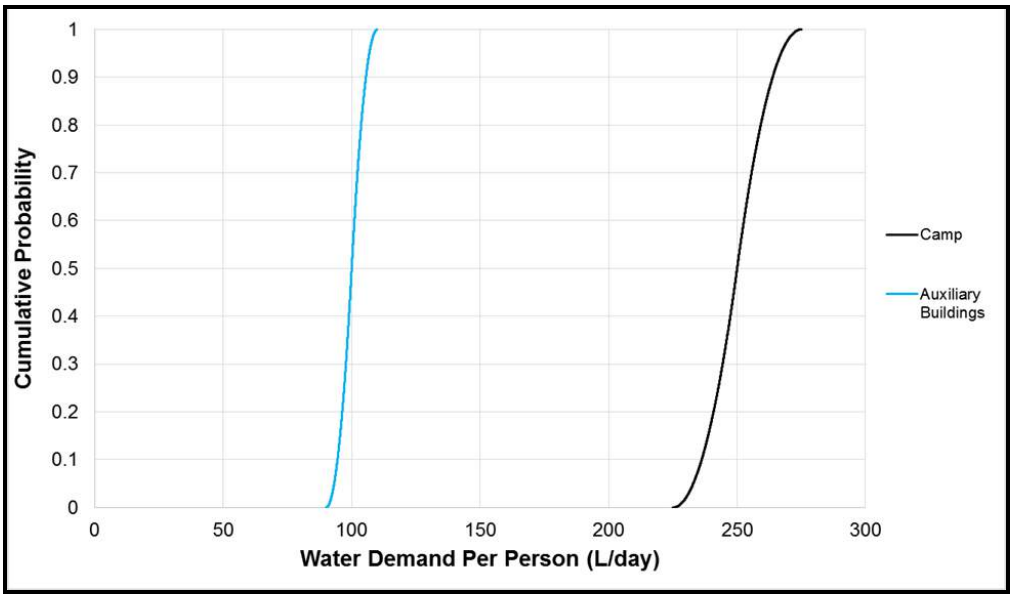
table continues...

Month	Percentage of Mean Annual Flow (%)
Sep	9.53
Oct	2.37
Nov	0.00
Dec	0.00

18.12.3 PROJECT STAFF

The model inputs related to human water use demand include the number of on-site workers present during the LOM, and the estimated per-person water demands at the camp and auxiliary buildings. The total number of workers present throughout the LOM were based on the open pit and underground labour force estimates developed for the Project feasibility study (Appendix C). The per-person water demands for the camp and auxiliary buildings are estimated by Tetra Tech to be 250 l/d/person and 100 l/d/person, respectively. To account for variability and uncertainty in these estimates, the WBM assumes these values are the modes of triangular distributions, and that the minimum and maximum water demands vary from these values by 10%. That is, the camp water demand will vary from 225 to 275 l/d/person and the auxiliary building demand will vary from 90 to 110 l/d/person. These distributions are shown in Figure 18.33.

Figure 18.33 Human-use Water Demand Distributions used in the WBM



18.12.4 ORE PROCESSING

The model inputs related to ore processing (milling) include the rate at which ore will be milled, the water required per ton of ore being milled, and the amount of water that is lost during ore processing. The monthly milling schedule developed during the feasibility study (Appendix C) was used to define the milling rate in the WBM. Tetra Tech staff estimates that the ore processing water demand will be 200 to 260 m^3/d

at the expected milling rate (9,160 tons/mo). The WBM assumes a uniform distribution from 200 to 260 m^3/d for the water requirement at this base milling rate. The unit water requirement (i.e. water volume per ton of ore) is then calculated by the WBM using the value sampled from this distribution and the base milling rate. The two primary sources of water loss during ore processing are expected to be loss to residual tailings water (which cannot be removed during tailings filtration), and other process-related losses. Tetra Tech staff estimates that 12 to 15% of process water will be lost to filtered tailings and 1 to 3% will be lost for other reasons. Uniform distributions were assumed for both of these variables resulting in a total loss of 13 to 18% of process water.

18.12.5 SUBSURFACE MINING

The final set of model inputs are related to subsurface mining operations. The mining report for the feasibility study (Appendix C) specifies that the summer and winter water requirements for drilling and watering down the ore will be 176 m^3/d and 138 m^3/d, respectively. To account for variability and uncertainty in these estimates, the WBM assumes that each of these requirements is the mode of a triangular distribution, and that the minimum and maximum water demand will vary from these values by 10%. As a result, the summer underground water demand varies between 158.4 to 193.6 l/d/person and the winter demand will vary between 124.2 to 151.8 l/d/person. SRK also estimates that during the summer up to 60% of the service water will be recoverable depending on ground conditions and rock temperature. The WBM assumes a uniform distribution from 40 to 60% for the fraction of underground process water that will be reused in summer, which accounts for uncertainty in SRK's upper bound 60% estimate.

The mining report for the feasibility study (Appendix C) indicates that underground mining operations will begin in January 2019 and continue through December 2023. The final input to the WBM is a binary time-series indicating whether or not underground mining is actively occurring during the LOM period.

18.12.6 CALCULATIONS

The model inputs presented in the previous section are combined with a number of necessary simplifying assumptions to determine the Project water balance. During each model simulation (realization) the same sequence of logic and calculations are performed using different combinations of input parameters. The first step in the model calculations is identifying the amount of surface water available for ore processing and/or subsurface mining operations. Because the current mine plan does not involve collecting runoff from the plant area, little or no dewatering of the underground mine is expected, and pit runoff will be pumped to the WRD sedimentation pond and discharged to the surrounding environment, all three of these quantities are defined as zero in the WBM. Therefore, the Sirelendge River is currently the only source of surface water for the Project.

The Sirelendge River is assumed to have no storage capacity, and the river will only be a usable water source throughout the LOM at the concurrent river discharge. Therefore, surface water is assumed to be only available from June through October.

The second step of the model calculations is to calculate the amount of surface water that can be used for ore processing. The demand for new (i.e., not recoverable and reused) ore processing water is calculated using the actual milling rate, the rate of water consumption per ton of ore, and the fraction of process water that can be reused. If the new water demand exceeds the surface water supply (Sirelendge discharge), then the difference is made up by groundwater. Otherwise, all of the ore processing new water demand is met by Sirelendge River water.

The third group of model calculations determines partitioning of the remaining surface water (if any) for underground mining purposes. Underground demand is calculated based on whether or not underground mining is actively occurring, the monthly average temperature, the summer and winter underground water demands, and the fraction of underground process water that will be reusable in summer. If underground mining is not active, the water demand is always zero. However, after underground mining begins in January 2019, if the average monthly temperature exceeds 0 °C, then the underground new (i.e. not reused) water demand is defined by the following equation:

Underground New Water Demand (t) = Summer Demand (1- Summer Underground Reuse Fraction)*

Alternatively, if the monthly average temperature is below 0 °C, then the underground new water demand equals the winter underground water demand.

As with the logic applied to meeting the ore processing water demand, if the underground new water demand exceeds the remaining surface water supply (Sirelendge discharge minus ore processing new water demand), then the difference is made up by groundwater. When the remaining Sirelendge discharge exceeds the underground new water demand, all of this demand is met by the Sirelendge River.

The final set of calculations calculates the total groundwater demand. The total groundwater demand is the sum of the human demands and the portions of the ore processing and underground mining demands that are unmet by the Sirelendge River. As a result, all water for human use is assumed to be reliant on groundwater throughout the entire LOM. This assumption was made to ensure that the direct, personal needs of Project staff are not dependent on the uncertain availability (and quality) of local surface water resources. The total groundwater demand is the main output desired of the WBM.

18.12.7 OUTPUTS

The quantities of greatest interest calculated by the model are the total groundwater demand, the individual components of groundwater demand, the total surface water supply, the rates of surface water use for ore processing, and subsurface mining operations, and the excess available surface water.

Model results are presented and discussed in the following sections.

18.12.8 ASSUMPTIONS

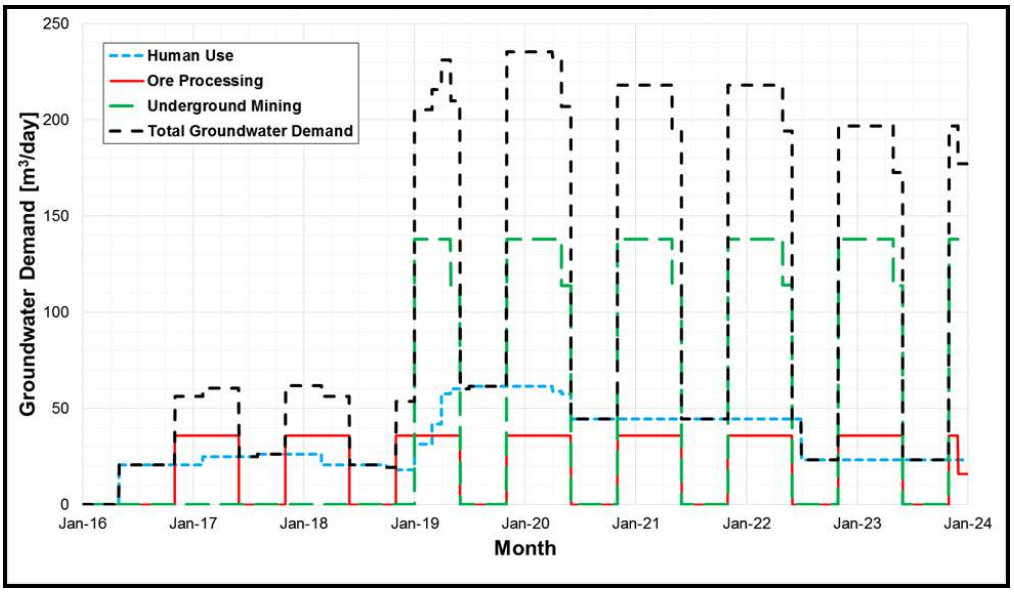
Several assumptions were made in the WBM based on field observations or assumptions made by others. These assumptions are as follows:

- Groundwater flow into the open pit or underground workings is negligible. This assumption is based on the observation that permafrost exists below the proposed mining depth of approximately 300 m (SRK Report Appendix C).
- Drilling in the open pit will not introduce water in the pit. This assumption is based on the proposed method of using compressed air for flushing (SRK Report Appendix C).
- Precipitation falling within the pit footprint is pumped to the WRD sedimentation pond within 24 hours of precipitation events and will be discharged to the surrounding environment (SRK Report Appendix C).
- Runoff from the site will not enter the pit footprint. This assumption is based on constructing diversion structures to reroute surface runoff away from the pit (SRK Report Appendix C).
- Evaporation was assumed to be negligible due to a lack of Project surface water features and freezing conditions that occur approximately eight months of the year.

18.12.9 RESULTS

The WBM simulates 10,000 different realizations of conditions during the approximate eight-year (2016 to 2023) life of the Project. Each of these realizations generates a set of the outputs described above during this period, and these results can be viewed individually or summarized statistically. The average total groundwater demand is shown in Figure 18.34 along with the three components that comprise this demand (human uses, ore processing and underground mining). There is a strong seasonal pattern in all of the non-human related demands because when the Sirelendge River is not frozen (i.e., summer) it typically meets all of the new water demands for both ore processing and underground mining purposes. However, the Project will be utilizing groundwater only to meet its demand.

Figure 18.34 Average Monthly Groundwater Demands Predicted for Human Use, Ore Processing, Underground Mining, and the Cumulative Groundwater Demand from these Sources



During open pit mining in the first three calendar years of the Project (2016 to 2018), the maximum average groundwater demand is approximately 61.9 m^3/d, which occurs in the winter of 2017 to 2018 when all new ore processing water will need to be derived from groundwater and when the number of workers temporarily peaks at 75. The total water demand then rises abruptly in January 2019 when underground mining begins and the number of workers increases from 51 to 90. Since these changes in operations occur during the winter, all of the associated changes in water demands will need to be met by groundwater. The maximum average water demand—235 m^3/d—occurs in the winter of 2019 to 2020 when the labour force is at a maximum (176 people), underground mining operations are active, ore continues to be processed, and the Sirelendge River is frozen.

Table 18.18 presents the annually-averaged total groundwater demand, the median (50th percentile, or "P50"), and other select percentiles. A statistical summary of the monthly total groundwater demand time-series is shown in Figure 18.35.

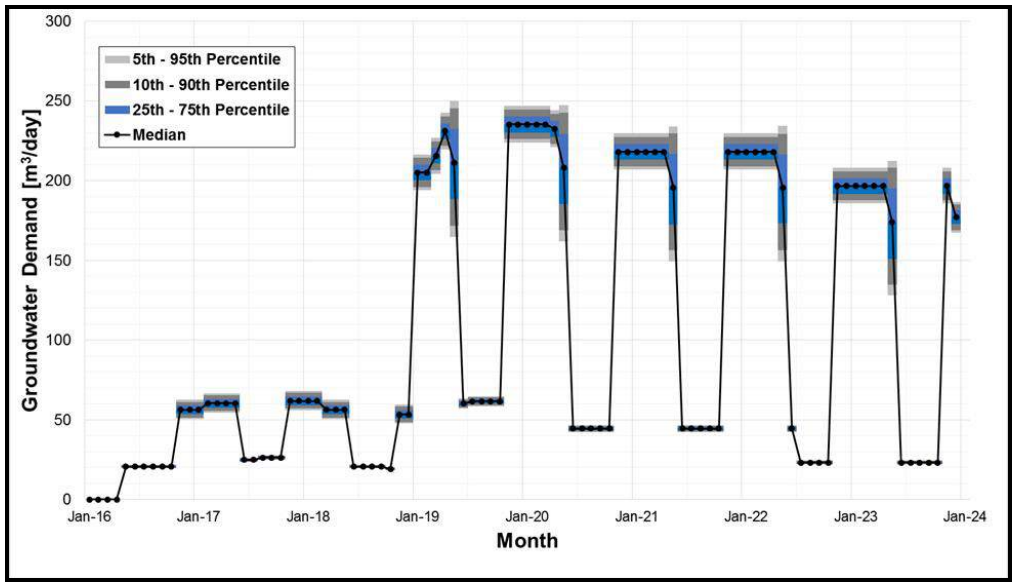
The annual cumulative total groundwater demand statistics in Table 18.18 reveal that the greatest amount of groundwater usage will likely occur in 2019, when approximately 52,000 to 60,000 m^3 of groundwater will be required.

The monthly median values shown in Figure 18.35 fall within 1% of the average values shown in Figure 18.34. The greatest variability in predicted monthly total demand occurs in May of each year, when the average temperature varies from -5.4 to 4.6°C and the amount of underground process water reuse can vary widely from 0% (i.e. temperature below 0°C, and reuse is not possible) to 60%. The highest 95th percentile groundwater demand is 249.9 m^3/d, which occurs in May 2019. Therefore, it can be stated with greater than 95% confidence that, given the WBM's inputs and assumptions, the monthly average groundwater demand will not exceed 250 m^3/d during the life of the Project.

Table 18.18 Average and Select Percentiles of the Cumulative Annual Groundwater Demand

Year	Annual Cumulative Total Groundwater Demand (m³)							
	P5	P10	P25	P50	P75	P90	P95	Mean
2016	6,674	6,778	6,979	7,229	7,484	7,698	7,811	7,234
2017	15,241	15,512	16,037	16,695	17,370	17,940	18,246	16,713
2018	13,788	14,049	14,557	15,195	15,849	16,400	16,694	15,212
2019	52,020	52,735	54,179	56,003	57,772	59,106	59,754	55,959
2020	51,067	51,762	53,163	54,934	56,659	57,966	58,603	54,896
2021	48,507	49,186	50,573	52,340	54,052	55,331	55,948	52,297
2022	44,730	45,386	46,720	48,411	50,050	51,282	51,877	48,369
2023	40,353	40,972	42,261	43,923	45,534	46,725	47,289	43,883

Figure 18.35 Median and Percentile Ranges for the Predicted Monthly Total Groundwater Demand Time-series



18.13 WASTE AND SEWAGE

Regulations state that waste on site is not allowed to be disposed of by burning. Solid waste will be stored in the solid waste storage facility, located to the west of the processing facility, between the reagent storage and ROM pad. The solid waste storage facility will be lined and segregated to handle different waste streams.

A containerised sewage treatment plant will be located at the camp. Sewage from the processing facility washroom facilities will be trucked to the camp sewage treatment plant for processing. As detailed in Section 18.12.3, daily water requirements are in the region of 350 l/d/person, generating approximately 35 m³ of clean waste water from the sewage plant. When the Sirelendge River is flowing, the clean treated waste water will be discharged into the Sirelendge River and the solid, treated cake will be trucked to the TMF and stored with the tails. During periods that

the Sirelendge River is not flowing, the clean, treated, waste water will be taken by bowser to the TMF for storage where the effluent will be treated as TMF runoff.

18.14 SITE STORAGE

18.14.1 REAGENT STORAGE AND HANDLING

Due to the accessibility to site and the constraints of the winter road only accessible for a short season, reagents will be stored in bulk with a minimum capacity of a nine-month supply.

Reagents will be transported to site in shipping containers and stored in a dedicated fenced secure storage area. The containers will sit on a lined pad that will contain drainage channels and sumps to catch surface runoff, which will be sent for treatment.

The containers will remain closed and locked until they are within the appropriate process plant reagent mixing facility for unpacking where appropriate ventilation and personal protection equipment is available for workers to unpack the container. Empty containers will be sealed and sent back to the reagent supply company for refilling. This is particularly relevant for the cyanide containers which cannot be repurposed and would be difficult to dispose of. The main plant reagents and yearly usage are shown in Table 18.19.

Table 18.19 Reagent Usage

Reagent	Quantity (t/a)
Cyanide	500
Hydrated Lime	100
Flocculent	5

The cyanide mixing area within the plant will store a small quantity of cyanide for continued operational supply. The cyanide mixing area will be a ventilated enclosed room with concrete floor, bund wall and sump. The mixing area will include sealed bag breaker, agitation tanks and pump systems.

The lime mixing area will include agitation tanks and pump systems all in an enclosed bunded area.

18.14.2 EXPLOSIVES STORAGE AND HANDLING

An explosives plant and storage facility will need to be constructed on site to meet the annual explosives and initiation product requirements. The explosives plant will be constructed from the specification and supervision of the explosives supplier. The location of the plant is south of the camp, down the Sirelendge River valley, an appropriate distance to maintain the recommended safety zone from the camp and occupied buildings. The plant is expected to have separate bunkered magazines for containerised storage of ammonium nitrate and initiation devices, office, workshop, and change room facilities.

The ammonium nitrate bags will be moved about the facility using a telehandler. The mobile manufacturing unit will be filled by driving onto a concrete apron adjacent to the store and the bags will be lifted over the truck and the ammonium nitrate discharged from the bottom of the bag. All waste will be adequately stored and disposed of in an appropriate manner in the explosives waste facility located further to the south down the valley.

The bunkered magazines will be placed a safe distance from the rest of the plant and from one another to ensure a safe and secure location for the high explosive and the initiation devices. The dimensions of the earth walls around the magazines will be provided by the explosives supplier; these are to contain any uncontrolled initiation and to minimise danger to plant personnel and infrastructure.

The explosives plant will be contained within a high-security fence. The access will via boom gate at the entrance which will be manned by security guards. Only limited personnel will be authorised to enter the facility and the materials made available will be controlled by the permanent security personnel. The components of the plant will be connected by a maintained gravel road.

18.14.3 Fuel Storage and Handling

Diesel and lubricants will be transported from a storage facility in Batamai during the winter road season. For construction and start-up, fuel bladder storage will be used. Each bladder can contain 250 m^3 of diesel. Permanent 12 m diameter by 12 m steel storage tanks will be constructed in the fuel farm at the process facility to store 6,000 m^3 of diesel, with bladder storage available for the remaining 400 m^3 of the nine-month supply volume.

A breakdown of fuel consumption and hence, storage required is shown in Table 18.20. Almost all of the diesel supply will be consumed by the process power station.

A refuelling point next to the fuel storage will be constructed to allow mine fleet and light vehicles to fuel up in a specially equipped area to contain spillage. Spent fuel and lubricants will be drained and stored in drums or approved vessels at a designated area of the fuel storage facility.

Table 18.20 Maximum Yearly Fuel Consumption

Energy User	Power Use (kWh/a)	Fuel Consumption (l/a)
Process Power Station	16,206,000	4,213,560
Camp	205,000	53,300
Raw Water Supply	184,000	47,840
Mine	4,953,231	3,087,244
Plant Vehicles	0	9,600
Camp Vehicles	0	9,600
Total Annual Consumption	**21,548,231**	**7,421,144**
Contingency (15%)	3,232,235	1,113,172
Total Design Consumption	**24,780,466**	**8,534,316**
Nine Months of Storage		6,400,737

18.14.4 Site Storage Closure Plan

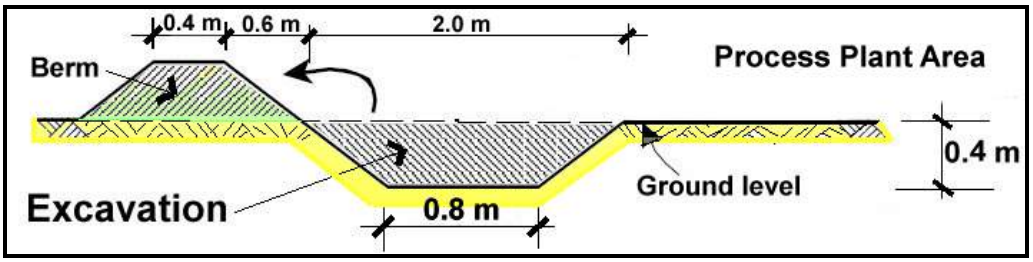
Procurement planning will ensure that the remaining levels of reagents, fuel, and explosives on site at the end of operations are kept to a minimum. A small quantity of fuel and reagents will remain on site for services such as the water treatment plant, which will continue to operate after mine closure. Any small quantities of cyanide will be neutralised and stored within the TMF. Left over ammonium nitrate will be used to fertilize the remediation sites. All other fuel and reagents will be stored at the waste storage facility in sealed barrels. Explosive primers will be properly disposed of in the explosives disposal facility.

18.15 Site Services

18.15.1 Plant Site Surface Water Management

The process plant is proposed for construction on the top of the plateau, as shown in Figure 18.4. The anticipated amount of precipitation is negligible, and no runoff water will be flowing toward the plant area. During the very short thaw period, the negligible amount of precipitation within the small plant site will be collected in a perimeter ditch, excavated around the structures, and the collected water be conveyed to the nearby TMF. The suggested cross section of the perimeter ditch is shown in Figure 18.36.

Figure 18.36 Suggested Cross Section of the Runoff Water Collection Ditch



18.15.2 Raw Water Supply

A raw water pump station, located in the Sirelendge River valley next to the camp, will provide the site raw water requirements from a groundwater borehole. A borehole pump located inside the station will pump well water into a 25 m^3, heated and insulated tank located adjacent to the station structure. A horizontal multiple-stage centrifugal pump, sized for 45 m^3/h at 540 m of head, will pump water from the pump station tank to the raw water tank located at the processing facility.

The pipe line between the pump station and the mill will be routed up the Porfirovy Valley alongside the access road before leaving the road to head north up the valley. The pipe line will meet up with the access road adjacent to the WRD and continue across the top of the hill to the process plant. The overall length of the pipe line will be approximately 4.5 km, and an elevation rise of approximately 400 m. Due the high static pressure, the lower half of the pipe will be constructed of carbon steel. The steel pipe line will be supported on wooden sleepers every 3 m, anchored directly to the bedrock or using pilings as required. The sleepers will be manufactured at the site lumber yard, using locally sourced timber. Expansion loops will be located every

250 m. The upper portion of the pipe line will be constructed of HDPE pipe and will be surface run and stabilized with berms every 30 m. The pipe line will be insulated and heat-traced.

Electricity for the pump station will be provided by a 400-V, three-phase, 60-Hz, 645-kW containerised diesel generator, shared with the camp. Switchgear, motor starters, lighting board, heat trace panels, communication panels, UPS and other electrical equipment will be located in a heated electrical room. The generator fuel tank will be located outside and will be sized to store a week of fuel at full load consumption, approximately 30 m^3.

The preliminary estimation for power requirements for the pump station are shown in Table 18.21.

Table 18.21 Pump Station Power Requirements

Service	Power Requirements (kW)
Borehole Pump	75
Raw Water Supply Pump	250
Tank Heating	15
Lighting and Distribution	1
Heat Tracing	75
Total	**416**

The pump station will be heated using heat recovery off the generator and infrared heaters. Monorails will be located in the pump room to service equipment.

The pump station building will consist of two fire rated rooms: the pump room and electrical room.

18.15.3 PROCESS WATER TANK

The process water tank will be located next to the cyclone tower and ball mill within the process plant. The location of the process water tank was chosen because of the proximity to the processes that require the largest quantity of water addition.

The process water tank will be topped up with fresh feed from the raw water tank and process return feed as barren solution from electrowinning. The process water tank is a 4 m diameter by 4 m fabricated steel tank with sloped tank bottom to allow for periodic cleaning of the settled solids, which will occur small quantities due to the nature of the recycled process water.

18.15.4 POTABLE WATER TREATMENT

Raw water will be processed through a containerised potable water treatment plant to provide potable water. Two treatment plants will be installed on site, one located at the processing facility adjacent to the raw water tank, and one located at the camp to provide potable water for the camp.

The exact specification of the treatment required will be determined by the results of the borehole water tests, but is likely to be a contain filtration, softening, and demineralization processes.

18.15.5 Assay and Metallurgical Laboratories

A metallurgical and a geological laboratory will be available on site.

The metallurgical laboratory will be a containerised laboratory located within the process plant and will handle approximately 10 samples per day. The metallurgical laboratory will include the following equipment:

- laboratory jaw crusher
- laboratory roll crusher
- riffles
- samplers
- sieve shaker
- laboratory ball mill
- bench flotation cell
- bottle roll equipment
- titration equipment
- laboratory pressure filter.

The geological laboratory will be located within the camp area, within close proximity to the core logging building. The geological laboratory will process the wet chemical assays from the mine, process plant, and exploration. Tetra Tech estimates that 12 grade control geological samples will be assayed daily.

18.15.6 Communications

The site wide data and voice telecommunications for the Project will be provided by a Russian cellular network provider. Silver Bear will install a mast on the hill to the south of the main pit and to this, the cell phone company will install their equipment and infrastructure.

The site local area network (LAN) will connect to the cellular network for datalink to the World Wide Web.

The communications systems for the Project will include:

- telephone system
- two separate LAN networks and datalink to the world wide web
- television surveillance system
- loudspeaker and alarm communications systems
- telecommunications cabling to all buildings

The LAN network is to include all cabling, equipment and infrastructure for connection to each building within the process plant area and to remote sites.

Cellular mobile telephones and communication equipment will provide the functions usually covered by site very-high frequency (VHF) radio. The cellular mast and transmission equipment will be installed and maintained by the cellular network provider.

18.15.7 Site Security

Pedestrian and vehicular access to the site will be controlled to provide security for personnel and property and to manage the risk of injury at the mining and processing operations.

The remote location of the site, and the site topography, eliminate the need for full perimeter fencing; however, high risk areas such as the explosives facility, reagent and cyanide storage and the power station will be need to be locally fenced.

Whilst the majority of personnel on site will be operational staff, consideration must be given to the seasonal nomadic reindeer herders that pass through the area.

18.15.8 Operational Personnel Transport

Site personnel will be transported around site, between the camp and process facility, and between the camp and airstrip by two rough terrain personnel carriers, each with a capacity of 10 people excluding the driver.

During each shift change-over, 50 to 55 people will have to be transported to the facilities on the top of the hill which will require 3 to 4 round trips per shift. Mining, processing and engineering services shift change overs will be staggered by 30 minutes to accommodate the 30 minute round trip from the camp to the top of the hill.

18.16 Power Supply and Distribution

18.16.1 Power Supply

Tetra Tech estimated that five, 645 kW diesel generating sets will be required to meet the load requirements of the process plant site.

The estimated operating load and maximum demand are listed in Table 18.22.

The recommended number of generators takes into account the calculated load, a spinning reserve allowance (20% of generator capacity), and an allowance for one cold reserve. The spinning reserve allows for a planned maximum generator loading of 80% following the requirements of Russian standard practice as advised by EMC Consultants.

Containerised generating sets are preferred; however, some assembly may be required on site due to a 14 t site road weight limit.

Table 18.22 Estimated Load Summary

Description	Units	Quantity
Estimated Average Power	kW	1 850
Estimated Maximum Demand	kW	2 200
Generator Capacity	kW	645
Spinning Reserve (per generator)	kW	130
Generators Required for Normal Operation (includes spinning reserve)	-	4
Cold Reserve	kW	645
Total Number of Generators Required	-	5

Heat recovery from the generator exhaust will be used for building heating.

Remote sites such as the camp facilities are by separate power generation. These are listed in Table 18.23. The mine power requirements will change when underground mining commences. Tetra Tech advised that the underground mine is not critically dependent on heaters, pumps or winders and that power outages could be tolerated. The estimated operating load and maximum demand for the underground mine are listed in Section 16.0.

Table 18.23 Estimated Remote Site Loads

Description	Load (kW)
Camp	150
Mine	40
Raw Water Pump Station	416
TMF	40
Explosives Storage Facility	40

18.16.2 POWER DISTRIBUTION

The power station is located adjacent to the process plant building. The generating sets will feed directly on to a 400-V switchgear assembly, which will be located in an air-conditioned, prefabricated building included within the power station package scope of supply. The power station switchgear assembly will contain feeders for distribution of power to the MCCs within the process plant building and for power station services as required.

Power will be reticulated throughout the building on cable ladder attached to building supports or process equipment walkways and support structures.

Three phase low-voltage supply to motors and other services will be 400 VAC.

MOTOR CONTROL CENTRES

The low-voltage MCCs and distribution boards will be fully installed in air-conditioned containerised buildings prior to shipment to site. Each room will be maintained at temperatures in the range of 10°C (winter) to 30°C (summer) with suitable air

conditioning and heating. The buildings for the MCCs and low-voltage distribution boards will be mounted 1 500 mm from the finished ground level to allow bottom entry access for all cabling.

Low-voltage variable speed drives (VSDs) will be supplied as part of the low-voltage MCC package. All VSDs supplied will eliminate low-order harmonics using either an active rectifier or a mains passive filters

EARTHING

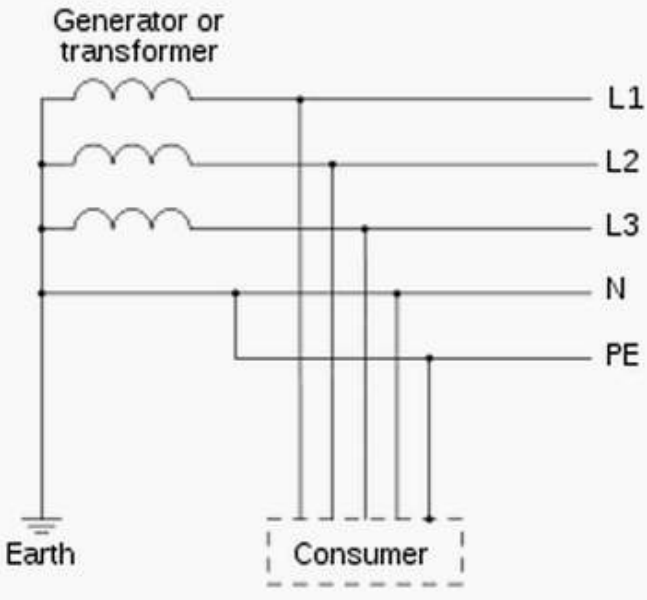
The earthing system will be a "TN-C-S" system as defined in International Electrotechnical Commission (IEC) 60634 (Figure 18.37).

The main earthing electrode system comprises localised earth grids and electrodes around the generators and main switchroom tied by means of poly vinyl chloride (PVC) insulated 120 mm² copper cables to the switch room main earth bar.

The underground earth grid will be installed around and below the generators to prevent dangerous step and touch potentials developing under fault conditions.

Rods will be driven in sufficient numbers and to sufficient depth to achieve less than 1 ohm for substations/MCC rooms and less than 10 ohms at connection points for lightning protection.

Figure 18.37 Earthing System



TN-C-S earthing system: combined PEN conductor from transformer to building distribution point, but separate PE and N conductors in fixed indoor wiring and flexible power cords.

18.16.3 CABLES

Cables will be sized and selected according to IEC 60364. Cables will be installed on heavy duty, galvanised cable ladder mounted on building and support structures. All cables will have steel wire armour protection.

18.16.4 LIGHTING AND SMALL POWER

Roadway and plant lighting will be installed throughout the plant for 24-hour operation.

Small power outlets will be provided throughout the plant. Lighting and small power distribution boards will be located in electrical rooms and plant areas and these will distribute power at three phase and single phase voltage as required.

18.16.5 VENDOR PACKAGES

The plant will have a number of vendor packages to be installed. Vendor packages will in general be supplied with electrical equipment such as motors, and heaters pre-installed. The electrical interface with each package will be designated during detailed design.

19.0 MARKET STUDIES AND CONTRACTS

19.1 PRODUCT

The Project will produce a silver powder of approximately 90% purity. Options exist to either smelt this powder into upgraded quality silver doré bars, or to leave it as powder. Both options will require refining before being available for sale as bullion. The Project plans to produce approximately 2.5 Moz (78 Mt) of silver per annum.

Smelting, should that option be selected, would be done in Yakutsk, and not on site, in order to save on power and staffing requirements at the mine site. The ability and willingness of refineries to accept silver powder was therefore tested. Both Russian and international refineries would accept either the high-grade powder or doré. Hence, the option exists to not purchase the smelter and associated equipment. This would save capital and operating costs, operating and security staff, and physical, secure facilities.

19.2 MARKETS

There are international banks and precious metal traders who will pay spot prices flat, or less a small percentage discount. For example, Metalor in Switzerland would take silver doré at the flat, spot rate (normally London Bullion Market Association (LBMA) fixing). Sberbank and Otkrytiye are two Russan banks that would also buy Silver Bear bullion at a 0.75% and 0.6% discount each, respectively. These institutions have provided some written offers to this effect; however, no contract negotiations have progressed beyond this expression of interest.

The advantage of using Metalor is that they will buy silver powder or doré directly, and refine and buy it with 85% of agreed purchase values paid upon acceptance of the silver at the place of handover, which could be the process plant gates. However, it has become apparent that although not legislated against, it would be extremely difficult to be able to export unrefined product. Indeed, there is a positive drive to keep precious metals within the Russian Federation. In addition to off-take potential by Sberbank, discussions have also been held with Lanta Bank.

Lanta Bank, in Russia, will also contract for the refining and off-take of silver powder or doré, whereas other Russian banks will not. One of the major shareholders of Lanta Bank is Krastsvetmet Refinery—the largest refinery in Russia—and hence Lanta bank can offer the complete collection, transport, refining, and off-take, as well as a series of pricing options. Details of an offer from Lanta Bank are not available at this time, but it is expected to be competitive, as they can provide excellent transport costs due to the volume of precious metals they organise. Lanta Bank is a significant holder and exporter of precious metals.

19.3 REFINERY TERMS

Silver Bear has contacted a number of refineries for terms, based on production of a minimum of 60 t of silver per annum at a grade of 90%, either powder or doré form (Table 19.1).

Table 19.1 Refinery Terms

	Refinery			
	Metalor	Prioksky	Ekaterinburg	Krastsvetmet
Refining Charge ($/oz)	0.19	0.23	1.46	0.59

The wide range of refining charges show that there are some negotiations to follow until a contract can be finalised; however, it does demonstrate that internationally competitive refining charges can be obtained within Russia. It could be expected that a charge of $0.20/oz could be achievable in Russia. A rate of RUB63:00:USD1:00 was used for the Russian refineries noted in Table 19.1. All refineries offer similar overall recoveries.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

Silver Bear retained ERM to review and summarize the work completed regarding the relevant environmental, social, and regulatory factors for the feasibility study for the Project with an effective date June 9, 2016. At that time, ERM considered the work of others as complete and accurate and commented only on the principles upon which Project infrastructure and facilities have been planned in so far as environmental controls. ERM has not undertaken any engineering design and the plans for, and the infrastructure and facilities required to manage water and waste, were designed, described, and reviewed in other sections of this report.

The feasibility study update for the Project is based on a revised mine plan to reflect the results of a more recent drilling program conducted at the Property. ERM has updated its summary of the environmental, social, and regulatory factors on an assumption that no material changes regarding the interaction between the Project and the natural and social environments are introduced with the revised mine plan. As such, the original review of environmental and social factors completed by ERM is considered to remain accurate.

The contents of this section have been updated from the Feasibility Study technical report released in June 2016 based on verbal updates provided by the company only. Regulatory approval documents, where received, were not available for ERM review as part of this update.

20.2 PERMITTING REQUIREMENTS

This section provides a summary of main regulatory requirements that govern the development, operation, and closure of the Project under Russian law, along with a summary of ERM's understanding of the current permitting status of the Project. It is not the intent of this section to provide a detailed list of regulations that may be applicable.

The simplified permitting process for mining Projects in Russia includes the following stages:

1. obtain mining license
2. confirm resources
3. conduct surveys and studies
4. develop and receive approval of project design documentation
5. obtain permits.

20.2.1 Mining License

Mining licenses are granted by the State Authority and outline the resources (quality and quantity of metals to be extracted) and specific conditions for exploration and development. A mining license holder is obliged to follow all the requirements specified in the license.

Silver Bear obtained a license for mining silver, copper, lead, and zinc on the Vertikalnye deposit. The license was issued with a termination date of 1st September 2033.

All the metals noted in the license are required to be extracted under Russian law. The feasibility study includes the mining and processing of ore to extract silver only. Further details regarding mineral tenure are described in Section 4.0. In July 2017, Silver Bear reportedly received approval from the State Authority to recover only the silver and store residual materials for potential future recovery of other minerals when economically feasible. This approval is referred to as the "technical scheme of mining".

20.2.2 Confirmation of Resources

Once the mining license is granted, exploration surveys and preliminary engineering studies must be conducted, as specified by the license. The outcomes of these surveys and studies are used to produce a TEO Conditsy Report (TEO), which is a technical-economical study that specifies conditions for development of the deposit. The TEO needs to include the amount of resources to be extracted, required infrastructure, economic feasibility of the project, and the potential market overview. The TEO is subject to approval by the Russian Central Commission for Reserves, which allows the license holder to initiate the development of the technical design documentation.

Silver Bear developed the TEO Conditsy report and submitted it for approval to Russian Central Commission which has completed its review on April 08th, 2016. A positive conclusion regarding approval was made on April, 25th 2016. After several revisions of the report, Silver Bear received its approval in July 2017, along with the "technical scheme on mining".

20.2.3 Surveys and Studies

Several surveys and studies are required to be conducted for the technical design development and are included as part of the project design documentation.

Necessary surveys include:

- geotechnical surveys
- hydrology, hydrogeology and meteorological engineering surveys
- geological engineering surveys (including surface and ground water surveys for water supply sources)
- environmental engineering surveys.

Silver Bear and its consultants have conducted various studies intended to fulfil these obligations. Survey reports were submitted for State Review process; however, they were not fully accepted by the commission and a number of comments and requirements were provided to Silver Bear by the State. Based on this feedback, Silver Bear conducted additional surveys during summer season of 2016. Silver Bear anticipates submitting the redeveloped reports in November 2016. The completeness of these surveys continues to be the subject of feedback from the regulatory community (see below). Follow-up studies have been required to address significant gaps. Some of these gaps are the result of changes in the Project as the design has progressed.

20.2.4 Development and Approval of the Project Design Documentation

In accordance with Russian legislation, a mining license holder must develop project design documentation. This documentation can be split into different sections and is approved by separate authorities (e.g. mining/waste management, and processing plant/infrastructure).

Silver Bear developed the mining and waste management sections of the project design documentation, which includes an ESIA for these facilities. These sections have been reviewed and approved by authorities. A positive conclusion of the State Environmental Review on these sections was issued in March 2017 (Table 20.1 – Stage 1). Work on the process plant/infrastructure section is currently in progress.

Following approval of the mining/waste management sections of the project design documentation, Silver Bear can submit the completed project design documentation, including the process plant/infrastructure documentation, for further review by the appropriate authorities (Stage 2).

Silver Bear currently accesses the camp and the mine site via land from the existing airstrip. A portion of the route taken traverses alongside the Sirelendge River and crosses it in multiple locations. This is not a permitted road or activity. The Sirelendge River has a status of high fisheries significance.

An all-season access road to the mine from the existing airstrip forms part of the Project as contemplated by the Feasibility Study but does not form part of the project design documentation submitted for permitting purposes. Due to the sensitivity of the adjacent river, all season access is not likely to be permitted by the fisheries authorities. ERM is of the view that all-season access to the mine represents a potentially material risk to the Project that still needs to be addressed. Table 20.1 describes steps (in chronological order) that have already been performed and still must be completed by Silver Bear in order to obtain the relevant permits to start construction and operation activities in compliance with Russian legislative and regulatory requirements. The dates contained in Table 20.1 were provided by Prognoz, and ERM has not completed an evaluation of the likelihood that these dates can be achieved.

ERM notes that Silver Bear continues to progress construction activities associated with the Project, in advance of design development and obtaining all of the necessary permits. In addition to real or perceived risks to safety and/or the environment, conducting these activities without the necessary permits presents procedural risks that could result in regulatory action, including fines, Project re-design and/or

schedule delays to the Project that could be material. This risk was raised with the company during the previous release of the feasibility study in June 2016.

Table 20.1 Mangazeisky Project Design Documentation Development and Approval Process

	Document	Authority	Desired Approval Outcome	Project Status (according to the data provided by Prognoz on 11th November 2016)
Stage 0	Permission for construction of non-hazardous industrial facilities	Non-governmental review (by specialized licensed private company)	Positive conclusion	Conducted on 22nd July 2016.
	Sanitary Protection Zone of groundwater intake	FBHD "Hygiene and Epidemiology Center in Sakha (Yakutia) Republic"	Positive conclusion	Conducted on 11th October 2016.
	Groundwater reserves (C1 category)	Committee for Geology and the Use of Subsurface Resources of Sakha (Yakutia) Republic	Positive conclusion	Expected by 15th November 2016. Protocol not received as of the date of this Report.
	Justification of License on Subsoil Use	Central Commission for Reserves	Positive conclusion	Conducted on 12th April 2016, Protocol received 28th April 2016
Stage 1	Mining and waste management section (including TMF, waste rocks and solid wastes landfill)	Public Hearings arranged by Company and local authorities	Protocol of Public Hearings (with positive conclusion)	Conducted on 18th April 2016, Protocol received 17th May 2016.
		Fishery authority of regional level (Lensk Basin Fishery Administration)	Approval in terms of impacts on surface water bodies of fishery significance and the levels of damage and compensation fees	Received by 16th June 2016. The approval refers to the water intake facilities only. The access road is excluded from the RDD (please see above).
		Federal Supervisory Natural Resources Management Service (Rosprirodnadzor)	Approval of volumes of wastes will be generated by the Project and waste classification with regard to Hazard Classes	Approval has been received by March 2017.
		State Environmental Review	Positive conclusion	Has been issued in March 2017.

table continues...

	Document	Authority	Desired Approval Outcome	Project Status (according to the data provided by Prognoz on 11th November 2016)
Stage 2	Russian Design Documentation including the approved by State Environmental Review mining and waste management part and baseline surveys reports	State Review of the Set of Engineering Baseline Services	Positive conclusion	Expected by September 2, 2017
		State Review of surveys: geological, meteo- and hydrological, geotechnical, environmental for mining and waste management facilities	Positive conclusion	Adverse opinion was received on 12th August 2016. Resubmitted surveys anticipated to be filed by Silver Bear in November 2016. Expected November 2017
		Fishery authority of regional level (Lensk Basin Fishery Administration)	Project Approval of impacts on surface water bodies of fishery significance and the levels of damage and compensations	Received by 16th June 2016. The access road has been excluded from the RDD which ERM considers a gap and a Project risk. All-season use of the access road as contemplated by Silver Bear is unlikely to be permitted by the Fisheries Authority.
		Federal Supervisory Natural Resources Management Service (Rosprirodnadzor)	Project Approval wastewater discharge points	Outstanding
		State Review	Positive conclusion	Adverse opinion on the set of baseline survey reports was received on 12th August 2016. Silver Bear anticipates submitting the mining and waste section of the Russian Design Documentation in November 2016 with an expectation of approval in November 2017.
			Positive conclusion with regard to the industrial and environmental safety of the Project design solutions	Application will be possible when positive conclusion of the State Review will be in place. Application date is expected in January 2018.

	Document	Authority	Desired Approval Outcome	Project Status (according to the data provided by Prognoz on 11th November 2016)
	Explosive and high-toxic substances Transportation authorization	Federal Supervisory on Industrial Safety (RosTechNadzor)	License for transportation and use of explosive and high-toxic substances	Outstanding
	Permission for industrial and domestic waste disposal	Federal Supervisory Natural Resources Management Service (Rosprirodnadzor)	Permission	Expected after the end of the construction and commissioning stages
Stage 3	Russian Design Documentation including and processing plant and infrastructure section (approval of this section by the State Environmental Review is not required). Note: Silver Bear intends to not to include the airstrip and access road from the airstrip from the design documentation which ERM believes is a regulatory gap.	State Review of surveys: geological, meteo- and hydrological, geotechnical, environmental for processing plant and infrastructure	Positive conclusion	Expected application date is not defined yet
		State Service for Public Wellbeing (Glavgosexpertiza)	Approval of the Project in terms maintenance of standards of impacts on workers	Expected application date is not defined yet
		Federal Supervisory Natural Resources Management Service (Rosprirodnadzor)	Project Approval wastewater discharge points	Expected application date is not defined yet
		State Review	Positive conclusion	Expected application date is not defined yet
			Positive conclusion with regard to the industrial and environmental safety of the Project design solutions	Expected application date is not defined yet

20.2.5 PERMITTING PROCESS

Table 20.2 lists the relevant permits required to be obtained by Silver Bear after the approval of the Project by State Review, in order to conduct construction and operation activities in line with legal Russian Federation requirements.

Table 20.2 Required Permits

Permit Title	Construction	Operation	Status
Trigger Activity: General Mining			
License for subsoil use for the purposes of exploration and mining	+	+	Obtained
Conclusion of State Environmental Review (Expertize) on the Project design documentation and results of engineering surveys, Conclusion of the State Review in order to apply on Permit for Construction	+	+	The conclusion of the State Environmental Review on mining and waste section of the design documentation is obtained. The conclusion of the State Review on mining and waste section is expected on January, 2017. The design documentation of the processing plant is in progress and application date is not scheduled yet.
Trigger Activity: Operating Mining Facilities			
Expert Conclusion on Industrial Safety		√	Application has been made
Expert Conclusion of on Industrial Safety of Hazardous Facilities	√	√	Outstanding
Permit for each specific type of technical equipment on hazardous facilities	√	√	Outstanding
Fire safety certificate	√	√	Outstanding
Permission for construction reconstruction or major repair of facilities, in cases where subsoil use is required	√		Outstanding
Certificate of conformity GOST R (Certificate of Quality)		√	Outstanding
Trigger Activity: Use of Explosives			
License for operation of explosion-hazardous production facilities	+	√	Received by Contractor performing blasting operations
License for storage and use of explosive substances for production activities	+	√	Received for Toxic Agents Storage and Explosives Storage operated by contractors.
Permission to conduct blasting operations	+	√	Received by Contractor performing blasting operations.
Trigger Activity: Waste Generation			
Approval of Waste Generation and Disposal Targets (WGDT)	√	√	In progress
Approval of hazardous wastes certificates	√	√	In progress
Expert Conclusion of State Environmental Review of design documentation for waste storage and neutralization facilities	+	+	

table continues...

Permit Title	Construction	Operation	Status
License for the neutralization and disposal of Hazards Class I - IV waste	+	√	In progress Application date is not defined yet
Trigger Activity: Water Abstraction			
License for subsoil use for the purposes of ground water abstraction	+	+	
Decision authorizing use of the water body for wastewater discharge	√	√	Application is with authorities. Decision is expected
Trigger Activity: Environmental Emissions			
Approval of Allowable Discharge Rate (for water emissions)	√	√	Application date is not defined yet
Permit for discharge of contaminants to the environment (water emissions)	√	√	Application date is not defined yet
Approval of Maximal Permissible Emissions (air emissions)	√	√	Application date is not defined yet
Permit for emission of contaminants to the atmosphere	√	√	Application date is not defined yet
Trigger Activity: Private Aviation			
The decision of the State Commission for radio frequencies (SCRF) on allocation of radio frequencies for the use of radio-electronic means		√	Will be operated by Contractor and all required permits will be obtained by Contractor
Conclusion of the expertize on the possibility of allocation of radio frequencies			Will be operated by Contractor and all required permits are obtained by Contractor
Permission to use radio frequencies (RF channels) of Roskomnadzor			Will be operated by Contractor and all required permits will be obtained by Contractor
Registration of radio-electronic means in the bodies Roskomnadzor and receipt of a certificate of registration			Will be operated by Contractor and all required permits will be obtained by Contractor
Certificate of ownership for aircraft			Will be operated by Contractor and all required permits will be obtained by Contractor
Certificate of registration for aircraft			Will be operated by Contractor and all required permits will be obtained by Contractor
Certificate of airworthiness of aircraft			Will be operated by Contractor and all required permits will be obtained by Contractor
Air operator certificate(s)			Will be operated by Contractor and all required permits will be obtained by Contractor

Note: √ - required on the relevant project stage
+ - received by Silver Bear as of August 10, 2017

ERM notes that the Project execution plan and schedule has been de-coupled from permitting activities. Construction activities are being undertaken without the necessary permits in place. Further, the commencement of mining is planned for Q4 2016 and ore processing in Q2 2017, whereas there is a high risk based on the current status that permits will also not be in place to support these operating stage activities by the dates assumed in the FS. In addition to real or perceived risks to

safety and/or the environment, conducting these current and future activities without the necessary permits presents procedural risks that could result in regulatory action, including fines, Project re-design and/or schedule delays to the Project that could be material.

Permitting plans exclude the access road between the airstrip and the mine site. ERM is of the view that this is a gap as upgrades to that existing infrastructure forms part of the Project.

Although Silver Bear has received a positive determination of the State Environmental Review for mining and wastes, changes to the Project based on the updated mine plan presented in the FS need to be presented to the authorities to determine if an amendment to the approval is required.

20.3 ENVIRONMENTAL AND SOCIAL STUDIES

20.3.1 STUDY DESIGN

Environmental and socio-economic studies that have been completed to date and/or ongoing are intended to contribute to the following objectives:

- Characterize the existing physical, biological, and socio-economic conditions to enable an assessment of potential impacts from the interaction with project components and activities, and develop appropriate mitigation and management measures to reduce adverse effects and maximize positive effects. This characterization will also identify and track trends in areas potentially affected by the Project during construction, operation, and closure.
- Characterize the geochemical characteristics of the Mineral Resource extraction process (tailings) and the mine waste that the Project will generate, to develop an understanding of their associated factors that could impact the environment.
- Inform mitigation and monitoring programs that will be implemented during the construction, operation, closure, and post-closure phases of the Project, such that predictions effects can be validated and any unanticipated effects can be iteratively addressed as necessary as the Project proceeds.
- Engage with potentially affected populations through the course of baseline data collection to exchange information on the Project, and to provide local residents, land users, communities, and other stakeholders the opportunity to express their concerns and participate in the ongoing project design and development process.

A list of specific environmental and social studies that have been completed to date are summarized in Table 20.3. These reports are also summarized further in this section.

Table 20.3 Environmental and Social Studies Completed (as of June 2016)

Studies Complete to Date	Date of Surveys	Scope
Air Quality and Noise		
Technical report on environmental baseline conditions in the Vertikalnoye deposit (ERM 2016b)	2015, revised in 2017 in order to incorporate data on the updated site layout	Report includes results of the one-time air sampling and analysis and noise measurement. Report provides results determination of snow as an indirect method of air quality assessment.
Climate		
Technical report on environmental baseline conditions in the Vertikalnoye deposit (ERM 2016b)	2015, revised in 2017 in order to incorporate data on the updated site layout	Report includes description of the general and seasonal climatic characteristic, including main climatic parameters (temperature, moisture, wind, precipitations, etc.).
Report on engineering-hydrometeorological surveys within the Mangazeyskoye mine field in Yakutia region of Russia, produced by OOO ISKOM	2015, revised in 2017 in order to incorporate data on the updated site layout	Report includes description of the general and seasonal climatic characteristic based on archive and relevant literature sources.
Surface Water		
Technical report on environmental baseline conditions in the Vertikalnoye deposit (ERM 2016b)	2015, revised in 2017 in order to incorporate data on the updated site layout	Report includes review of the archive data and field survey of the hydrological characteristics of the Project area, results of water sampling, field and laboratory analysis.
Report on engineering-hydrometeorological surveys within the Mangazeyskoye mine field in Yakutia region of Russia, produced by OOO ISKOM	2015; revised in 2017 in order to incorporate data on the updated site layout	Report includes results of field hydrological and hydro-chemical regime of rivers survey. Description of the hydro-morphological characteristic and hydrological parameters (river flow) of the watercourses.

table continues...

Studies Complete to Date	Date of Surveys	Scope
Geology		
Report on prospecting work within the central part of silver- polymetallic ore “Mangazeyskoye” mine field, produced by “YanGeologiya”	1993-2003	Report includes analysis of mineral resources base of silver within “Mangazeyskoye” mine field based on prospecting work. Report includes description of the geology and permafrost conditions, morphological and geochemical characteristics of the mine field.
Report on assessment of acid potential and geochemical properties of waste rock at the Vertikalnoye deposit of the “Mangazeyskoye” mine field, produced by ERM	2015, revised in 2017 in order to incorporate data on the updated site layout	Report contains the interim findings of investigations of waste rock /tailings properties and includes results of the sampling and analysis of mine waste rock, description of the composition of tailings, etc.
Seismicity survey conducted by Neryungristroiizyskaniya (NSI)	2015	Report contains results of micro shooting and confirmation of seismicity scale of the project area.
Ground Stability and Permafrost		
Report on engineering-geological surveys of baseline conditions of geological environment within the Mangazeyskoye” mine field in Yakutia region of Russia, produced by State State Unitary Enterprise (SUE) “YakutTISIZ”	2005	Geological and geomorphological studies and analysis of hazardous exogenous processes and hydrological phenomena were carried out. This survey allowed gathering detailed information on the geological, geomorphological and cryological features of the Project area, conditions and forms of existence and development of hazardous exogenous processes.
Report on engineering-geological surveys of permafrost-hydrogeological, hydrogeological and engineering-geological conditions of Vertikalnoye license area of “Mangazeyskoye” mine field, produced by LLC “NeryungriStroyIzyskaniya”	2014, revised in 2017 in order to incorporate data on the updated site layout	Hydrometric, morphometric and geothermal studies were carried out. Report also includes information on hazardous exogenous processes.
Technical report on environmental baseline conditions in the Vertikalnoye deposit (ERM 2016b)	2015, revised in 2017 in order to incorporate data on the updated site layout	Report includes desktop description of geomorphological and cryological features of the Project area, geomorphological features and development of the hazardous exogenous processes within the design facilities’ sites.

table continues...

Studies Complete to Date	Date of Surveys	Scope
Groundwater		
Report "Assessment of the capacity for groundwater supply of the mining and processing enterprise in the Vertikalnoye deposit of the "Mangazeyskoye" mine field based on the results of geophysical and hydrogeological studies of drilling exploration wells", produced by LLC "NeryungriStroylzyskaniya"	2015	Report includes the results geophysical and hydrogeological studies of drilling exploration wells within the Project area and results of water sampling and laboratory analysis (please also refer to Section 20.5.1).
Hydrogeological modeling performed by SRK	2015	Report includes the results of hydrogeological modeling performed based on the data of tests on groundwater inflow.
Technical report on environmental baseline conditions in the Vertikalnoye deposit (ERM 2016b)	2015, revised in 2017 in order to incorporate data on the updated site layout	Report includes review of the archive data and recent survey of the hydrogeological characteristics of the Project area, results of field and laboratory analysis.
Soil		
Technical report on environmental baseline conditions in the Vertikalnoye deposit (ERM 2016b)	2015, revised in 2017 in order to incorporate data on the updated site layout	Report based on archive and relevant literature sources and assessment of environment of study area based on the field survey. Report includes description of the general characteristic of the soils and results of the sampling and analysis (geochemical, bacteriological, parasitological).
Radiological Conditions		
Technical report on environmental baseline conditions in the Vertikalnoye deposit, produced by ERM (2016b)	2015; revised in 2017 in order to incorporate data on the updated site layout	Report includes results of the gamma-ray survey, measurement of the maximum GREDR, measurement of the ground surface RFD and natural water sampling to access total specific alpha and beta activity.
Terrestrial Ecosystem		
Technical report on environmental baseline conditions in the Vertikalnoye deposit, produced by ERM (2016b)	2015; revised in 2017 in order to incorporate data on the updated site layout	Report includes review of the archive data and recent floristic survey (flora and vegetation and their habitats) of the Project area, results of vegetation sampling and laboratory analysis.

table continues...

Studies Complete to Date	Date of Surveys	Scope
Terrestrial Wildlife		
Technical report on environmental baseline conditions in the Vertikalnoye deposit, produced by ERM (2016b)	2015; revised in 2017 in order to incorporate data on the updated site layout	Report includes review of the archive data and recent biodiversity survey (invertebrates, vertebrates and birds and their habitats) of the Project area, as well as report includes information on protected species.
Aquatic Life including Fish and Fish Habitat		
Technical report on environmental baseline conditions in the Vertikalnoye deposit, produced by ERM (2016b)	2015; revised in 2017 in order to incorporate data on the updated site layout	Report includes review of the archive data and recent ichthyologic survey (fish and hydrobionts sampling and geochemical analysis) of the Project area.
Specially Protected Natural Areas		
Technical report on environmental baseline conditions in the Vertikalnoye deposit, produced by ERM (2016b)	2015; revised in 2017 in order to incorporate data on the updated site layout	Report includes current data about the local and regional specially protected natural areas situated in the Project area of impact.
Socio-economic and Land Use		
Technical report on environmental baseline conditions in the Vertikalnoye deposit, produced by ERM (2016b)	2015; revised in 2017 in order to incorporate data on the updated site layout	The survey combined quantitative (analysis of socioeconomic statistics) qualitative (expert interviews with representatives of local authorities and community; analysis of the qualitative data on the socioeconomic situation from public sources) methods.
Report on historical and cultural expertise "Assessment of the impact on the ethnological environment in the places of residence of indigenous people and Archeological study for the construction and operation of facilities of the Vertikalnoye deposit of the "Mangazeyskoye" mine field produced by the Institute of Humanitarian Research and Indigenous Studies of the North (Russian Academy of Sciences, Siberian branch)	2015	Report includes survey on identification of objects of archaeological heritage directly on the Vertikalnoye deposit and adjacent areas.

table continues...

Studies Complete to Date	Date of Surveys	Scope
Report on ethnic and cultural conditions in Sebyan-Kuel settlement located in Kobyaysky District of Sakha (Yakutia) produced by the Institute of Humanitarian Research and Indigenous Studies of the North (Russian Academy of Sciences, Siberian branch)	2015	Report includes desktop and field (questionnaire survey and interviews with representatives of locals) ethnological surveys.

20.3.2 EXISTING ENVIRONMENTAL CONDITIONS

Existing environmental conditions are described based on ERM (2016b) a part of the Russian design documentation. The surveys included field works, laboratory, and desktop studies performed by ERM consultants involving local experts and scientific institutes. Report has been prepared in Russian only.

PHYSICAL ENVIRONMENT

The Project is located in the Kobyaysky District (Ulus[1]) of the Sakha Republic in a highland with altitudes of 800 to 2,200 masl (Figure 20.1) A seasonal winter road connects the Project to port facilities on the Lena River and the nearest settlement, Sebyan-Kyuel, is 43 km south-southwest of the Project site. The Project area is located within the Endybal River basin and the largest watercourses include the Arkachan, Endybal, Sirilende, and Fedor-Yurege rivers.

The Project is located in an area characterized by a continental sub-arctic climate, and is significantly influenced by the mountain relief that causes high wind velocity in elevated areas. The average monthly temperatures for January and July are −38.1°C and +13,7°C, respectively. The frost-free period lasts for 52 days, on average; however, some years have had frost persisting throughout the entire summer. Precipitation of approximately 200 mm/a occurs mostly as snow. The approximate number of days with snow cover is 209, and the average snow depth is no greater than 35 cm.

Current air quality baseline conditions are below established criteria for carbon monoxide, nitrogen dioxide, sulphur dioxide, and particulate matter. Baseline air quality in the area is dominated by natural sources and limited to dust from wind erosion, primarily just after snow and ice have melted and under dry conditions. However, available data is limited due to the remoteness of the site and activities limited to ongoing exploration. The air quality in the Project area is within regulatory maximum permissible concentrations (MPC).

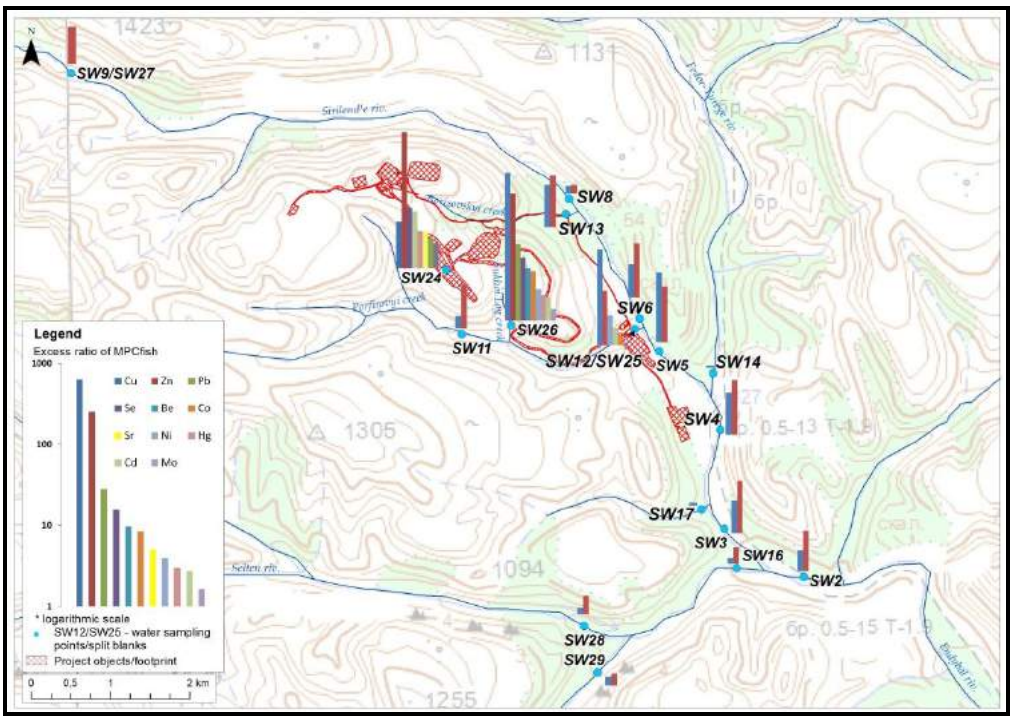
The river system in the Project area consists of the Sirelendge River and its major right-bank tributaries: Borisovsky Creek and Porfirovy Creek, with a tributary, Sukhoi Log Creek. This is the basin of the Yana River which flows into the Arctic Ocean.

Preliminary laboratory tests indicate that concentrations of minerals in some surface water samples and bottom sediments exceed regulatory standards (maximum

permissible concentrations for fishery water bodies (MPC_f[2]). These elevated concentrations (Figure 20.1) are attributed to natural weathering processes across the Project affecting regional watersheds, and to exploration activities in local waterways near the Vertikalny deposit area. The feasibility study does not include an allowance to remediate any historical impacts that may be caused to date by Project activities.

Preliminary analysis suggests elevated concentrations of magnesium, iron, manganese, aluminium, strontium, copper, zinc, cobalt, nickel, cadmium, lead, and selenium in surface water and elevated metal concentrations of cadmium, lead, nickel, zinc, and arsenic in bottom sediments.

Figure 20.1 Surface Water Quality Compared to MPC for Water Courses for Fishery Water Bodies (MPCf)



Terrain of the Verkhoyansk Range is primarily defined by seasonal water erosion during the spring snow melt and storm rainfall that results in eroded low areas separating topographic highs. This seasonal water erosion also defines slope processes in the Project area (Figure 20.3).

Other geological and hydrological processes that affect the Project area terrain include:

- cryogenic, including frost splitting, seasonal heaving, solifluction, deserption, frost sorting, stone-run formation (Figure 20.2), thermokarst
- gravitation: slides, landslides (Figure 20.4), landfalls, mudflows (Figure 20.5), avalanches

[1] 'Ulus' is an administrative division unit of the Sakha Republic equivalent to a district.
[2] Surface water quality standards are determined in Russia for water bodies dedicated to the corresponding water use: MPCs for fishery water bodies (MPC_f) and MPCs for cultural and domestic water use (MPC_{cd})

- fluvial and hydrological: erosion, swamp formation, and ice formation.

Figure 20.2 Alpine-type Terrain of the Area along the Axis of the Verkhoyansk Range

Source: E.V. Garankina

Figure 20.3 Stone-runs of Small Boulders in Near-the-edge Area of the Right Bank of the Borisovsky Creek Valley

Source: E.V. Garankina

Figure 20.4 Micro Landslides on Taluses, Left Bank of the Sirilendie River Valley, Opposite the Workers' Settlement, after a Heavy Rain on 8th August 2015

Source: E.V. Garankina

Figure 20.5 Small Mudflow Channel after a Heavy Rain on 8th August 2015 in the Cleavage of the Northern Slope of Mount Endybal

Source: E.V. Garankina

Permafrost conditions are approximately 230 to 350 m below ground surface, reaching thickness of up to 400 to 500 m on larger plateaus. This is the main factor defining engineering and geological characteristics of the area. Seasonal freezing of the near surface material is relatively fast. On average, it takes two months; freezing speed exceeds 1 cm/d.

Groundwater of the Yana-Indigirka basin consists of two hydrogeological levels:

- perched water, seasonal thaw depth (active zone) water and taliks (under lakes)
- connate water within the cracks and pores of the underlying geologic material below the permafrost zone.

Water within the perched, active zones and talik is insufficient for use at the Project process facility and for drinking due to poor quality and limited/seasonal availability. Additional studies (Table 20.3) confirm the availability and sufficiency of infra-permafrost water to be used for the Project water supply. Based on the work completed by SRK (Section 20.2), and described in Table 20.3, ERM is not aware of any environmental or regulatory constraints preventing the use of this water for this purpose.

BIOLOGICAL ENVIRONMENT

Soil

Soil in the Project area is poorly developed, acidic, and has a high proportion of stones and debris. The soil is also affected by cryogenic deformation and thixotropic and suprapermafrost gleying resulting in limited soil regeneration.

Common soil types of the area and their associated location and vegetation community are summarized in Table 20.4.

Table 20.4 Summary of Terrestrial Ecosystem Communities

Soil type	Location/Community
Typical brown soils	Cowberry-lichen tundra and upper taiga larch open woods at narrow watersheds and steep slopes of the middle taiga zone (10%)
Podzolized cryosolic soils	Larch moss-cowberry forests in river valleys
Mountain tundra typical and gleyic soils	mountain tundra belt on well drained and poorly drained flat areas
Dry-peaty brown and high moor peaty brown soils	Middle and lower parts of slopes under ledum-sphagnous larch woods in river valleys and intermountain nicks
Chernozem-like soils	Rare
Alluvial soils	Valleys of rivers and creeks
Al-Fe-humus abrasems (disturbed brown soils)	Anthropogenically modified areas as a result of mineral exploration activities
Meadowed brown soils	Anthropogenically modified areas as a result of mineral exploration activities

Sampling and testing of soils suggest that in accordance with the Russian standard, the vast majority of soils within the Project area can be used, based on their environmental characteristics, without limitation. There have been some contaminated soils identified through field surveys, notably in the area along the access road (near the mouth of the Sukhoi Log Creek). These contaminated soils are likely the result of geological and exploration works that have been conducted to date. Remediation plans for these contaminated soils that are currently known, and

any found during the Project life, will need to be developed to outline measures such as testing, excavation, and either disposal to landfill locations, or blended/layers as part of site pad construction. The feasibility study does not include an allowance to remediate any historical impacts that may be caused to date by Project activities.

Plant Life

The Project area is located in the mountains of the Verkhoyansk district of the Bering northern taiga sub-province of Cajander larch (Larix cajanderi) forests. Dominant types of plant associations are larch forests and woodlands, and mountain tundra. In valley complexes, chosenia woods and willow shrubs also occur.

Protected species identified within the Project area include five species of vascular plants and three fungi species listed in the Red Data Book of the Russian Federation and the Red Data Book of the Republic of Sakha. Four species were identified within 1,000 m of Project infrastructure that will require protection. There are also unique ecosystems to these areas that will also require protection. Several small steppe areas that are the relicts of the Late Plestocene occur only sporadically and are rare in the north-eastern part of Yakutia.

Valuable medicinal and food plants occurring within the Project area are cowberry, ledum, blueberry (*Vacinum uliginosum*), cranberry, dwarf pine (*Pinus pumila*). These species are a wide-spread plant resource across the entire Yakutia taiga, and the due to the remote location, the potential for harvesting volumes in the Project area is low.

Animals

The diversity of terrestrial vertebrates (wildlife) in the Project area is low due to climatic conditions and specific features of the mountain terrain. In addition, the Verkhoyansk Range, which extends from the Arctic Ocean to the Sea of Okhotsk coast, serves as a natural barrier for the migration of animals.

Important wildlife species in the Project area include Snow sheep, elk, Musk deer, and Reindeer that support economic livelihood for local residents and communities. Other wildlife in the area includes brown bear, gray wolf, wolverine, lynx, sable, stoat and the black-capped marmot. Primary game/fur animals hunted by the local population (i.e. reindeer herders) are squirrel and ermine and, in recent years, sable has been steadily becoming more popular as a game/fur animal. Birds noted in the area include willow ptarmigan, Eurasian sparrowhawk, merlin, common kestrel, golden eagle and Eurasian eagle-owl. The maximum diversity and specificity of wildlife species is observed in river valleys in the mountain forest belt. The lowest diversity is in the forest belt of mountain slopes, specifically in mountain tundra complex.

No amphibians or reptiles occur in the Project area.

The majority (50%) of the entomofauna species occur in meadow-type habitats with forests and meadow-forests providing habitats for 24% and 20% of insect species, respectively. The remaining 6% of species occur in mountain-tundra, tundra, bog, steppe, and meadow-steppe habitats.

Within the Project area and area of influence there are no species of mammals listed in the Red Data Book. However, the black-capped marmot (*Marmota camtschatica*, Pallas) are registered in adjacent area (N65°42,119 E129º59,219), (N65°43,465 E130º9,588) to the Project.

Two species of birds (golden eagle and eagle-owl) and three Lepidoptera species (*Parnassius tenedius* (Ev.), *Parnassius phoebus* (F.) and *Colias hecla viluensis* (Mén.) that are listed in the Red Data Book of the Russian Federation and the Red Data Book of the Republic of Sakha were identified within Project area. The closest nests are located within 1,000 m from Project facilities. Monitoring of these nests is recommended during Project construction and operation.

Key fish species occurring in watercourses of the Project area are grayling and burbot, both of which are not rare or protected and have no commercial fishing value. Local residents undertake recreational fishing for household consumption.

In accordance with the regulations of the Federal Agency of Fisheries, the rivers within the Project area fall under the following fishery categories:

- the Arkachan River can be classified as a water body of the highest fishery category due to the occurrence of *Coregonus lavaretus pidschian* and *Hucho taimen*
- the Endybal, Sirelendge and Fedor-Urege Rivers fall under the first fishery category as watercourses used for reproduction, wintering, and feeding of *Thymallus arcticus.*

According to Russian Federation regulatory requirements, the water quality of watercourses with fishery significance should be maintained at the same level, with no contaminants discharged without treatment, assuring the meeting of regulatory standards. As illustrated in Figure 20.1, these waterways already exceed the MPC_f; in particular there are elevated copper and zinc concentrations.

There are no Special Protected Natural Areas (SPNAs) in the Project area; however, more than 20 SPNAs of local and regional value have been established in the Kobyaysky Ulus and other surrounding uluses of the Republic of Sakha (Yakutia) (within a 200 km distance from the license area), including one national park, 16 resource reserves, 6 no-take zones, and 5 unique lakes. The nearest SPNAs to the Project are shown in Figure 20.6 and include:

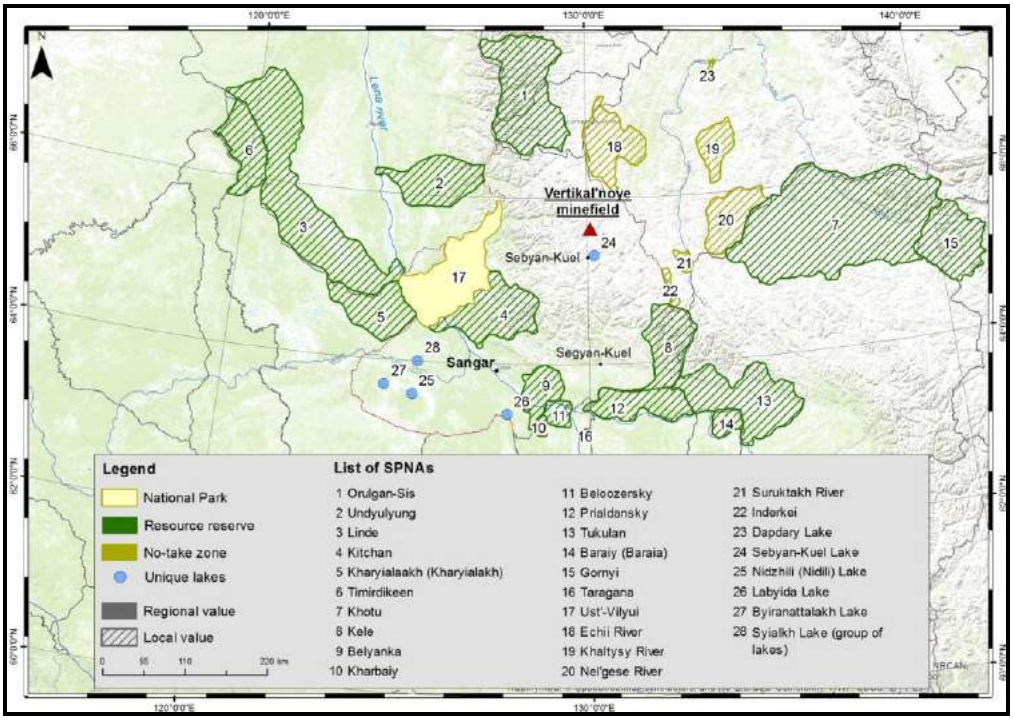
- Echii River no-take zone with an area of 229,800 ha - of the local significance;
- Unique Lake Sebyan-Kyuyol with an area of 2 071 ha – of the regional significance.

No Ramsar territories (the Convention on Wetlands of International Importance, especially as Waterfowl Habitat) are located in the Project area of influence.

Key bird area “Forty islands” (RU3064) is included in the list of Important Bird and Biodiversity Areas (IBAs) developed by BirdLife International and located within the territory of SPNA natural park “Ust’-Vilyuysky”. It is included in “indirect list” of

Ramsar wetlands as it provides habitat and nesting areas for approximately 20 rare bird species, including water fowls.

Figure 20.6 SPNAs in the Surroundings of the Project Area



20.3.3 EXISTING SOCIO-ECONOMIC CONDITIONS

Existing socio-economic conditions are provided based on the report on the results of socio-economic survey performed by ERM in 2015 (ERM 2015).

The Project area (Figure 20.7, license area) is located a considerable distance from both the Kobyaysky district (Ulus) center Sangar (approximately 330 km by roads or 200 km by air) and from the republic administrative center Yakutsk (approximately 400 km by roads).

The Project Area is located in the Lamynkhinsky Nasleg ("Nasleg" is the lower level of the administrative territorial division of the Sakha Republic).

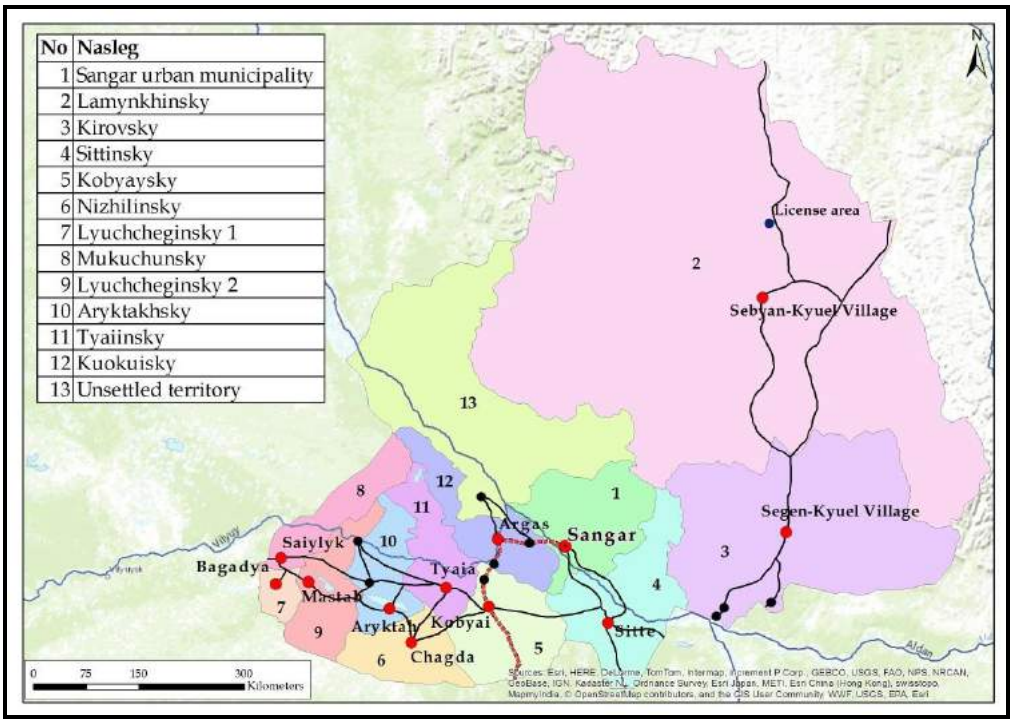
The Project area is a zone of residential area of indigenous peoples of the North of the Evens (the Evens) (Lamuts): 714 of 773 people of the Nasleg (92%).

The Evens are one of the indigenous peoples of the north living in vast north eastern territories of the Russian Federation. The Evens population of the Lamynkhinsky Nasleg is steadily increasing. Remoteness and restricted accessibility have preserved the traditional Evens culture within the Lamynkhinsky Nasleg.

Traditional Evens economy is reindeer herding, supplemented with fishing and hunting game for meat and fur. The Evens families are private owners of reindeer herded together with reindeer of State Unitary Enterprise (SUE) Sebyan. The Project footprint and the seasonal access road will intercept grazing areas traditionally used

for reindeer herding. Silver Bear has initiated a process to assess and valuate compensation that will be provided to the affected herders. Negotiations are not complete.

Figure 20.7 Location of License (Project) area in Kobyaysky District



Reindeer husbandry in the Lamynkhinsky Nasleg also includes reindeer herders of SUE Sebyan. SUE Sebyan consists of several teams each using its own grazing/herding areas and camping sites; two teams (No. 5 and No. 9) have been using the Project area:

- Team No. 5 traditionally migrates in the area of the abandoned Village of Endybal. The team is comprised of six reindeer herders and three chum-keepers and grazes more than 1,000 reindeer, including 583 heads of SUE Sebyan and privately owned animals. Routes of late-autumn and early spring movements are located near the Endybal village and partly near the Project area. A fawning ground used by the team in April/May is located near the former settlement not far from the license area.
- Team No. 9 of SUE Sebyan migrate near the winter road of regional significance, Batamai to Sebyan-Kyuel. As of March 2015, the herd numbered 1,000 reindeer, including 500 privately owned animals. The team is comprised of six reindeer herders and three chum-keepers.

The Evens families have no legal title to land; however, grazing and herding areas of the SUE Sebyan are historic areas determined during the time of the former Soviet Union.

The Evens families of the Lamynkhinsky Nasleg preserve their traditional work as reindeer herders. The local school and community clubs make efforts to preserve the Evens language, national culture, folk songs and dances; however, the economic and

social life of the Evens is currently undergoing a transformation as young villagers now rarely become reindeer herders, refusing to migrate with reindeer all-year round. Reindeer herding practices are also changing, which results in shorter migration routes, fewer campsites, and camp moving less often. The Evens are simultaneously integrated into the Russian cultural space. All these external influences threaten the traditional Evens lifestyle.

The Association of the Indigenous Peoples of the Sakha Republic (under the presidency of Andrey V. Krivoshapkin, Sebyan-Kyuel Village) is the main public organization uniting indigenous communities in the region.

The Project area can be regarded as having a high potential in terms of the use of its ecosystem services by the local communities. The most extensively used ecosystem services are in areas around Sebyan-Kyuel, along the winter road, and reindeer herders' migration routes.

The most important ecosystem services are graze land resources extensively used by the local reindeer herders. Other important land use in the area are hunting and fishing resources as one of the main sources of household incomes, and an important food source for the local communities. Hunting is predominantly commercial, the sable being the main game animal. OJSC FPK Sakhabult (private commercial enterprise of agricultural and industrial specialization, specializes on hunting and processing of fur) holds a commercial game hunting license with a wildlife management area (WMA) of 1,835 ha within the Project area (17% of the total area of Ulus).

Fish caught in the local lakes and rivers are the second most important local source of food after reindeer meat. Residents of Sebyan-Kyuel mostly access fish on the lakes located east of the settlement. The locals also fish at the rivers, including the area near the former Endybal Village (the Arkachan River).

Timber resources are significant for the local communities as the majority of houses have stove heating. However, timber reserves are small in mountainous areas. Mature forests around settlements have essentially been cleared.

Wild berries (bog whortleberry, cranberry, cowberry, currants, etc.), mushrooms (orange-cap boletus, orange agaric, yellow boletus, etc.) and medicinal herbs are gathered around the village primarily for personal consumption.

According to official information, no cultural heritage sites (of federal, republic, or local (municipal) significance) listed in the Russian National Uniform State Register of Cultural Heritage Sites or identified cultural heritage sites are located within the Project area. An archaeological survey was completed as part of the ethnological assessment during the summer of 2015. The survey confirmed an absence of any objects of cultural heritage within the Project site.

Some sites of spiritual and religious value for the local community are located near Sebyan-Kyuel and the Project site. A shaman's grave (northwest of the Project) and ancestral burial sites (in the former Endybal Village) are located 10 to 15 km from the Project. Some sites of worship (shaman's grave, ancestral graves, sacred trees) are located near the winter road and include these ancestral burial sites worshipped by

individual families and the Evens worship graves of shamans and sacred natural sites.

20.3.4 STAKEHOLDER AGREEMENT

Major stakeholder groups connected with the Project were identified during the scoping stage of the Project. Table 20.5 includes the key groups of external and internal stakeholders.

Activities undertaken as of June 2016 by Silver Bear and its consultants have included public hearings in line with requirements of the RF regulations and informal consultations with local communities and authorities during the data collection in the course of the socio-economic study.

Key interests and concerns identified by ERM that have been raised as of June 2016 through the stakeholder engagement include:

- employment opportunities for young people from the village of Sebyan-Kuel that would reduce the outflow of young people in other settlements of Yakutia
- negative impact on traditional activities of the community of Sebyan-Kuel, mainly related to reindeer herding
- overall effect to the health of reindeer herders, and the residents of Sebyan-Kuel.

ERM helped Silver Bear establish formal stakeholder engagement procedures, including the development of internal and external grievance mechanisms.

Prognoz signed an agreement for social and economic cooperation with administration of the Nasleg (municipality) and the Ulus (district) in March 2013. The agreements are aimed to cover the relationship between Silver Bear and local administrations in the course of the exploration activities. In May 2016 these agreements were amended and extended to the term of construction stage.

The aim of such agreements is to define the size and/or form of financial and other assistances to local communities, the communication channels, tools of support distribution etc. Current agreement includes support local community to construct Cultural Center with provision construction materials and logistics, fuel delivery etc. As reported by the Company at the date of reporting all current obligations are performed by ZAO Prognoz in due time. No potential complaints from authorities or local community are expected by the Company with regard to implementation of these activities.

Table 20.5 Key Project Stakeholder Groups

Group	Stakeholders
Indigenous people (Evens)	• Indigenous people whose reindeer herding camps and routes are located close to the Project site
Residents of the settlements located near the Site and associated infrastructure facilities	• Residents of Sebyan-Koel, Segen-Koel and Batamay settlements
Organizations and individuals whose property rights will be affected by temporary or permanent withdrawal of land plots during the Project implementation	• Enterprises of the Lamynkhinsky nasleg, that are located close to the Project site (mainly commercial hunting and herding groups, Sakha-bult and SUE Sebyan)
Government authorities and regulators	• Federal authorities • Sakha Republic regional authorities; • Local authorities (administrations of the Kobaysky ulus, Lamynkhisky and Kirovsky naslegs)
Non-government organizations, independent experts	• Specialized environmental, public and research organizations, experts (Association of the Indigenous people of Sakha republic, etc.) • Public organizations promoting social programs • Other public organizations in the Project area
Beneficiaries of Silver Bear's social programs	• Communities of Sebyan-Koel, Segen-Koel and Batamay settlements
Mass media	• Local and regional internet recourses: • Newspaper "Yakutiya" • Local and regional internet recourses: – www.ysia.ru – www.sakhalife.ru – www.gold.prime-tass.ru – www.vedomosti.ru
Personnel of Silver Bear, Organizations involved into the Project implementation and subcontractors	• Construction and project organizations involved into the Project implementation • Company's personnel and subcontractors' employees • Consultants involved into the Project implementation
Organizations involved into personnel training and selection	• Educational institutions on basis whereof training of personnel for the enterprise is realized • Recruitment centers

20.4 Environmental and Social Impact Assessment, Mitigation and Management Plans

20.4.1 Environmental and Social Impact Assessment

Consideration of environmental and social factors throughout the entire lifecycle of the Project (preparatory works, construction operations, production operations, and decommissioning) is an essential prerequisite to successful projects.

In order to meet both Russian and international requirements, the following must be undertaken as part of the impact assessment process (as described in Section 20.1.2):

- As required by Russian regulatory requirements, and based on a range of studies and surveys as listed in Table 20.1, Silver Bear developed Russian design documentation including materials for an ESIA and an environmental protection plan (EPP). At the time of preparation of this section Silver Bear has completed development and received approval of that part of the Russian Design Documentation including materials of Environmental Impact Assessment and Environmental Protection Plan covering the mining and waste management sections. Russian Design Documentation for processing plant remains under review by the regulatory authorities.
- In accordance with international requirements Silver Bear completed the preliminary international Environmental and Social Impact Assessment (ESIA) based on Feasibility Study in 2016. At the time of this report, the Company has decided to complete the ESIA report as soon as Russian Design documentation will be approved and no further design changes will be required.
- ERM has developed international Environmental and Social Impact Assessment (ESIA) based on Feasibility Study as originally released. The ESIA was completed by ERM on behalf of Silver Bear in July 2016. The Environmental and Social Management Plan (including specific MPs) to address the impact and issues identified by the ESIA were developed by ERM in August 2016. However it is recommended to perform the Gap Analysis of Silver Bear's performance against the IFC Performance standards based on recent Project's updates and to develop an Environmental and Social Action Plan which will include steps to cover the gaps identified.

Developing two sets of documents in accordance with Russian and international requirements is a common practice for Russian projects due to the different format and level of detail required by the Russian regulations, as compared to international community.

As the ESIA has not yet been completed as of the date of this report, the assessment of potential effects of the Project, the development of mitigating and management measures, and the establishment of monitoring programs embedded in the feasibility study are preliminary and expected to evolve over time. The results of ongoing work, including ongoing environmental studies, stakeholder engagement, and permitting have the potential to influence the design, operations, and economic results of the Project. Having not completed the ESIA, there is a risk that there the Project may cause environmental and social impacts that could otherwise have been avoided. This risk is amplified given that construction activities are already underway.

The State Environmental Review was completed and approved after the effective date of the FS. As such, the FS does not consider the impact any terms and conditions associated with this approval may have on the Project and its design. ERM has not completed a gap assessment of the FS Project and the requirements associated with the State Environmental Review.

20.4.2 Environmental and Social Management Plan

Table 20.6 presents a summary of key identified environmental and socio-economic aspects of the local area and recommended mitigation measures and management for Project design and planning. Given that the project design development and construction activities are ongoing, there may also be additional mitigation and management measures under consideration to further reduce the negative effects during construction and operation, and to meet closure objectives.

Table 20.6 Environmental and Social Potential Impacts and Proposed Embedded Design Mitigation

Aspect	Risk/Potential Impact	Mitigation & Control Measures
Geology	• ARD/ML from newly exposed rock surfaces, rock used for infrastructure construction, surface storage of waste rock and/or processed ore tailings, resulting in impacts to surface water sources; and • Mobilization of fine particles by wind or water, potentially leading to increased amounts of dust in the air or deposition to area land and water.	• Preliminary studies indicate that the waste rock and tailings generated by the Project will be NAG but will likely leach elevated concentrations of metals including zinc, lead, cadmium, silver, and arsenic and antimony. These parameters are also elevated in the existing surface water environment. • Roads and pads used for infrastructure development will use surface and/or quarry rock designated as low risk for ARD/ML. • Waste rock will be stored on surface in a dedicated facility. Seepage from the waste rock will be combined with contact water from the site and directed to an unlined exfiltration pond to remove suspended solids. Water quality will be monitored to confirm compliance with regulatory requirements for release to the environment. • Tailings will be stored on surface as a filtered dry stack. The facility will be lined and during operations any excess water will be pumped from a clarification pond and directed to the process plant for recycling. The TMF will operate as a zero discharge facility during normal operations. • Further testing of materials will be required to validate initial conclusions regarding the potential for ARD/ML. The feasibility study assumes that beyond the use of an exfiltration pond, downgradient of the WRD water treatment will not be required to protect the environment and meet regulatory requirements. Further work is required to validate this assumption.

table continues...

Aspect	Risk/Potential Impact	Mitigation & Control Measures
Ground Stability and Permafrost	• Siting and/or design of facilities results in permafrost degradation, ground stability, or other alterations of the active or inactive soil layers resulting in facility failures and impacts to the receiving environment.	• A siting trade-off study was completed by SRK (Appendix K) and geotechnical conditions were considered as part of this study. The TMF and WRD are located in areas covered with shallow, generally-granular overburden, over sedimentary bedrock. The feasibility study concluded that the risk of soil subgrade instability is low.
Groundwater	• Impacts to groundwater regime due to the use of water for mining and milling purposes.	• A water sourcing study was completed by SRK (Appendix K others and concluded that sufficient groundwater is available to support Project demands. Drawdown of the deep aquifer is not expected to result in unacceptable impacts to the natural environment and there are no other users of this water. Mining occurs within the zone of permafrost and is not expected to have a material impact the groundwater regime.
Surface Water Quality	• Impacts to the water quality in the Sirelendge River and/or its tributaries due to the physical disturbance to river sediments, discharge of treated sewage, seepage of water with elevated TSS and/or metals from the exfiltration pond downgradient of the WRD and runoff from project infrastructure. Deposition of dust to water bodies from use of roads and from the dry stack TMF. • Spills to surface water bodies from hydrocarbon storage and transport.	• The general site arrangement for the Project site was designed to protect surface water quality by directing contact water through a series of runoff collection ditches to a seepage and runoff collection pond adjacent to the northeast boundary of the WRD. This pond will act as a sediment control pond allowing sediment to settle out prior to release of excess water to the receiving environment. Sewage from the camp site will be treated and trucked to the TMF. Monitoring of site discharges and of the receiving environment will be undertaken to confirm the effectiveness of controls. • The feasibility study assumes that water content and weather conditions will suppress dust from the TMF. This assumption will be validated through site monitoring and further mitigation, including the use of dust suppressants, if necessary. • An all-season access road to the mine from the existing airstrip forms part of the Project as contemplated by the Feasibility Study. This road generally follows the shoreline of the Sirelendge River and must be crossed as the airstrip and camp are on opposite sides of the river. The Sirelendge River is sensitive fish habitat. Due to the sensitivity of the adjacent river, environmental impacts are possible and all-season access is not likely to be permitted by the fisheries authorities. ERM is of the view that all-season access to the mine represents a potentially

Aspect	Risk/Potential Impact	Mitigation & Control Measures
		material risk to the Project that still needs to be addressed. Consideration should be given to the relocation of the airstrip to the other side of the river in order to reduce the potential impacts to the river. • Standard operating procedures, such as restricting travel to authorized areas, speed control, and road watering to control dust will reduce the risk of environmental impacts. Hydrocarbon storage facilities designed with secondary lining and the establishment of spill response plans will also reduce the risk.
Hydrology	• Disruption of the natural flow patterns of surface runoff to Sirelendge River and its tributaries due to the Project footprint resulting in a loss of drainage and potential impacts to the aquatic environment. • Impacts to the surface water environment due to loss of water to the open pit or underground mine workings.	• Compact project footprint located at the headwaters of the drainage basins for the Porfirovy Creek and the Borisovsky Creek. The main project infrastructure intersects approximately 13% and 4%, respectively, of the total catchment area for these tributaries and 0.51% for the Sirelendge River. • Potential reduction in flows considered minor. Hydrological monitoring should be conducted during operations to confirm the impacts to the aquatic environment are not occurring. • Mining is contained entirely within the zone of continuous permafrost. All zones of mining are above the invert elevations of the local creeks and the Sirelendge River. Excess contact water that collects in the pit and underground workings (precipitation and drill water) will be collected in sumps and pumped to the exfiltration pond adjacent to the WRD.
Aquatic Life (including fish and fish habitat)	• Project activities directly removing or altering fish habitat (e.g. culvert installations and reduction in stream flows below normal low levels). • Project activities potentially affecting water quality or sediment quality.	• No direct impacts to fish and fish habitat are expected as a result of Project infrastructure. Any discharges to the receiving environment will meet regulatory discharge criteria and the receiving environment will be monitored. • An all-season access road to the mine from the existing airstrip forms part of the Project as contemplated by the Feasibility Study. This road generally follows the shoreline of the Sirelendge River and must be crossed as the airstrip and camp are on opposite sides of the river. The Sirelendge River is sensitive fish habitat. Due to the sensitivity of the adjacent river, environmental impacts are possible and all-season access is not likely to be permitted by the fisheries authorities. ERM is of the view that all-season access to the mine represents a potentially material risk to the Project that still needs

Aspect	Risk/Potential Impact	Mitigation & Control Measures
		to be addressed. Consideration should be given to the relocation of the airstrip to the other side of the river in order to reduce the potential impacts to the river. • Standard operating procedures, such as restricting travel to authorized areas, the implementation of spill response plans, and the use of dust control will reduce the risk of direct and indirect impacts to fish and fish habitat.
Terrestrial Ecosystem	• Loss of ecosystems and vegetation to footprint of Project and supporting infrastructure. • Degradation of ecosystems and vegetation through increased dust deposition, potential introduction of invasive plants, alteration of local hydrology, and effects caused by chemical spills.	• Compact project footprint that directly impacts approximately 637 ha of terrestrial habitat. Use of a seasonal road for resupply purposes to reduce impacts to the terrestrial environment, protected plant species identified in the Project area will require special measures that may include protection zones, replanting programs and education and awareness programs. There are also unique ecosystems to the area that will need similar special measures. • Standard operating procedures, such as restricting travel to authorized areas, dust control, spill response planning, and vehicle washing will reduce operational level impacts to the terrestrial environment.
Terrestrial Wildlife	• Habitat loss (direct and indirect). • Changes in movements and/or behaviors. • Mortality (direct and indirect). • Attraction of animals to human use sites.	• Compact project footprint that directly impacts approximately 637 ha of terrestrial habitat. Use of a seasonal road for resupply purposes to reduce impacts to the terrestrial environment. • Loss of grazing and migration areas for reindeer will be offset through a compensation agreement with affected herders. • Standard operating procedures, such as restricting travel to authorized areas, restricted speed limits on travel, avoiding nesting areas, and effective domestic waste management will reduce operational level impacts to local wildlife.
Air Quality and Noise	• Airborne particulates • Sulphur dioxide; • Nitrogen oxide; • Fugitive dust • VOCs • GHGs • Diesel engine exhaust from vehicles.	• Increased dust emissions (with elevated lead and arsenic) due to drilling, mining, processing activities (loading, crushing, grinding) and road traffic are the primary sources of air emissions. Installation of dust control and collection systems (e.g. reagent mixing, grinding) in the processing plant and standard operating procedures, such as restricting travel to authorized areas, speed control, and road watering to control dust along roadways. Use of modern equipment and standard operating procedures for the efficient use

Aspect	Risk/Potential Impact	Mitigation & Control Measures
		of fuel will minimize emissions such as mono-nitrogen oxides. Equipment designed to Russian design standards to reduce noise to acceptable levels.
Climate	• Release of GHGs.	• Use of modern equipment and standard operating procedures for the efficient use of fuel.
Socio-Economic and Land Use of Indigenous People Communities	• Employment and income opportunities, as well as other issues. • Education, training, and skills development opportunities. • Business opportunities and economic development. • Indigenous community stability and well-being impacts. • Loss of livelihoods and income of indigenous people due to loss of reindeer grazing areas.	• Establishment of socio-economic management plan that includes the following measures: – project recruitment, employment and training plan to maximize local employment – cooperation programs with central and local educational institutions; on-the-job training programs – local content and procurement plan to maximize business and economic opportunities – livelihoods restoration plan including establishment and distribution of a compensation fund to offset the Project impact and regular communications with the local herders and indigenous communities – community development program.

Note: TSS – total suspended solids; VOG – volatile organic compound; GHG – greenhouse gas

20.5 WATER AND WASTE MANAGEMENT

This section briefly describes the interaction of the Project with the environment during the operations phase, as it specifically relates to the management of water and wastes based on the current design of the facilities described in other sections of the feasibility study.

20.5.1 WATER MANAGEMENT

WATER SUPPLY

Water supply (Section 18.12) will be provided from a groundwater well located in the Sirelendge River valley next to the camp. From a storage tank at the pump station, the raw water supply will be pumped via a 4.5-km pipeline to the raw water tank located at the processing facility. The pipe line between the pump station and the mill will be routed up the Borisovsky Valley and along the ridge to the west of the road. There are no any other users of ground water in the area and work completed by NSI (2016) has concluded that no material environmental impacts are expected from the use of this water.

During construction, water supply will be extracted from the Sirelendge River and trucked to the camp and process plant until the pump station and pipeline are available to provide the water supply.

Site Contact Water

The general site arrangement for the Project has been designed to protect surface water quality by directing contact water through a series of runoff collection ditches to a seepage and runoff collection pond adjacent to the northeast boundary of the WRD. This pond will act as a sediment control pond allowing sediment to settle out prior to release of excess water to the receiving environment (Borisovsky Creek).

Contact water in the open pit and underground mine workings is anticipated to be limited to drill waste water and precipitation, as these operations are within continuous permafrost conditions and no groundwater inflow is expected. This contact water will collect in sumps for reuse in drilling if feasible, or pumped to the seepage and runoff collection pond adjacent to the WRD.

The current water balance developed for the Project should be expanded to include site-wide surface water sources, including contact water that collects in the seepage pond, water from the TMF, and open pit and underground mine water. This water balance should be developed to characterize both the quality and quantity of water that will be managed site wide. The current model was not used to predict water quality. The water balance should be used in conjunction with a site-wide water monitoring program to validate model predictions and confirm that environmental mitigation measures are effective and discharge can occur in accordance with anticipated permit conditions without further treatment.

20.5.2 Tailings and Waste Management

Solid Waste and Sewage

As outlined in Section 18.13, solid waste will be stored in the solid waste storage facility, located to the west of the processing plant. The solid waste storage facility will be lined and segregated to handle different waste streams.

A sewage treatment plant will be located at the camp to accommodate the camp and sewage trucked from the processing plant washroom facilities. Treated waste water will be trucked to the TMF and the solid sludge from the sewage treatment plant will be trucked to the TMF and stored with the tails.

Geochemistry of Waste Rock and Tailings

To date, the preliminary characterization of the waste bedrock and saprolite has been undertaken by ERM on a limited number of samples and incorporates testing to classify solid material based on parameters that leach from the material (ERM 2016b).

Static acid-base accounting methods indicate that the NP is in excess of the acid AGP due to the elevated carbonate and low-sulphur content for most of the waste rock and tailings. In accordance with the assessment of AGP and geochemical characteristics of waste rocks, the elements with high concentrations in waste rock are zinc, lead, cadmium, silver, arsenic, and antimony. A wide range of elements characterized by high concentrations were found in aqueous extractions that simulate expected waste rock drainage. Concentrations of the following elements may potentially exceed the MPC established under Russian law: sulphate ion, magnesium, iron, aluminium, zinc, selenium, thallium (1.1 to 13.0 MPC), manganese, cadmium

(1.1 to 140.0 MPC). However, the levels appear to be generally within the hydro-chemical background levels found in the Project area. Acceptable water quality for discharge will be further confirmed with the authorities when Silver Bear applies for the discharge permit. The feasibility study does not include an allowance for active treatment of seepage waters from the waste rock or tailings waste.

It is recommended to conduct further ARD/ML studies in order to align sampling with final mine layout and to get statistically reliable results.

Consideration of this potential for ARD/ML has been considered in Sections 18.10 and 18.11.

Waste Rock Dump and Overburden Stockpiles

The open pit will have a dedicated WRD and overburden stockpile during construction and operation. The waste material will consist of waste bedrock and waste saprolite. Details on the design of these stockpiles, including the associated peripheral and internal drainage systems are provided in Section 18.11.

Tailings

The whole tailings generated in the process plant will be discharged into the TMF. The TMF will be constructed 0.2 km northeast of the process complex and will cover an area of 7.69 ha, storing approximately 0.8 Mt of dry (85% w/w) tailings material over the 7.4-year LOM. The dry waste storage area will consist of a HDPE lined pad surrounded by a rock fill perimeter berm. There will be a clarification pond at the eastern toe of the TMF waste stack with associated perimeter containment dam.

During construction and operation phases, and under normal conditions, the TMF will be a zero discharge facility. Seepage will be diverted to the process plant after the clarification pond. Details on the TMF design, including the schedule for dam construction are provided in Section 18.10. This design results in a low risk to the receiving surface water environment during normal operations.

20.6 ERM Comments

The feasibility study update for the Project is based on a revised mine plan to reflect the results of a more recent drilling program conducted at the property. ERM has updated its summary of the environmental, social, and regulatory factors on an assumption that no material changes regarding the interaction between the Project and the natural and social environments are introduced with the revised mine plan. As such, the original review of environmental and social factors completed by ERM is considered to remain accurate.

The contents of this section have been updated from the Feasibility Study technical report released in June 2016 based on verbal updates provided by the company only. Regulatory approval documents, where received, were not available for ERM review as part of this update.

ERM provides the following overarching comments and recommendations.

- Project documentation required for the Russian regulatory review and approval process has been developed in parallel with completion of the FS and an ESIA in accordance with international standards has been initiated but is incomplete. The assessment of potential effects of the Project, the development of mitigating and management measures, and the establishment of monitoring programs embedded in the feasibility study are considered to be preliminary and are expected to evolve over time. The results of ongoing work, including ongoing environmental studies, stakeholder engagement, and permitting have the potential to influence the design, operations, and economic results of the Project.
- The Project is currently undergoing the Russian review and approval process. As part of this process, the environmental and social aspects of Project have and will continue to be evaluated by the regulatory authorities. There can be no guarantees that the Project will be approved under Russian law, the timeline for any decision, and if the Project is approved, that there won't be terms and conditions that have a material impact on the feasibility of the Project. ERM recommends that Silver Bear continue to actively engage with government authorities and community stakeholders as part of this process.
- Silver Bear continues to progress construction activities associated with the Project, in advance of the design development and obtaining all of the necessary permits, ERM notes that the Project execution plan and schedule has been de-coupled from permitting activities. Construction activities are being undertaken without the necessary permits in place. Further, the commencement of mining is planned for Q4 2016 in addition to real or perceived risks to safety and/or the environment, conducting these current and future activities without the necessary permits presents procedural risks that could result in regulatory action, including fines, Project re-design and/or schedule delays to the Project that could be material. ERM recommends that a detailed compliance assessment be undertaken and that Silver Bear work with regulatory authorities to address current and potential permitting/compliance gaps. This risk was raised with the company during the previous release of the feasibility study in 2016.Silver Bear has expressed that they expect to have received regulatory approval of the Project and the needed construction permits by December 2017. Permitting delays are not uncommon for mining projects and have already been experienced on multiple occasions by the Project proposed by Silver Bear. ERM again emphasizes the risks being knowingly taken by the Company by continuing construction activities without the necessary permits in place.
- The State Environmental Review was completed and approved after the effective date of the updated FS. As such, the FS does not consider the impact any terms and conditions associated with this approval may have on the Project and its design. ERM has not completed a gap assessment of the FS Project and the requirements associated with the State Environmental Review.
- The State Environmental Review was completed prior to the updated FS based on an optimized mine plan. ERM recommends that Silver Bear confirm with the regulatory authorities if an amendment to the approval is required.

- Project level socio-economic framework agreements have been negotiated with local stakeholders to mitigate potential socio-economic impacts of the Project. Cost estimation of social investments will be defined on the basis of annual consultations with authorities... ERM has not reviewed any cost allowances associated with these agreements that may have been included in the financial analysis of the Project presented in Section 21. Environmental and social cost allowances were provided by Silver Bear to Tetra Tech.
- Silver Bear has expressed its intention to comply with the Equator Principles and the IFC Performance Standards. A gap assessment against these international frameworks was completed for Silver Bear in November 2014 by ERM and an ESIA was initiated but remains incomplete. ERM recommends that Silver Bear continue efforts to align the Project with these international frameworks as a means to inform project design and the development and implementation of project specific environmental and social management and monitoring plans. An updated gap assessment is recommended to assess the current level of alignment of the Project as defined by the feasibility study. The more advanced the Project the more difficult it may become to fill gaps remaining.
- Further engagement with local indigenous communities with regard to land and resources restoration is recommended to ensure that social requirements of the IFC Performance Standards are met.
- Further environmental studies are recommended to enhance the dataset characterizing the aquatic environment and confirm the source of elevated metals found in the local and regional waterways, including the potential impact of exploration and development activities that have already taken place. These studies should be used to inform the permitting process and to validate the effectiveness of environmental controls planned to protect the receiving environment.
- An all-season access road to the mine from the existing airstrip forms part of the Project as contemplated by the Feasibility Study but does not form part of the Project Design Documentation submitted for permitting purposes. The road route is adjacent to, and within the bed of the Sirelendge River (rated as a river of high significance) and its tributaries which presents a risk to water quality and fish and fish habitat and is not in accordance with good international practice for environmental protection. Due to the sensitivity of the adjacent river, current and future use of this all-season access route is not likely to be permitted by the fisheries authorities. ERM is of the view that all-season access to the mine represents a potentially material risk to the Project that still needs to be addressed.
- Vehicular travel adjacent to, and within the bed of, the Sirelendge River and its tributaries can present a risk to water quality and fish and fish habitat and is not in accordance with good international practice for environmental protection. The current plans also present a permitting risk with Russian authorities. ERM recommends Silver Bear undertake a trade-off study and evaluate the potential for relocation of the airstrip and re-alignment of the connecting road network. The project design documentation should be updated to include upgrades to this road and the airstrip as per Russian legislative requirements.

- The current water balance developed for the Project should be expanded to include site-wide surface water sources, including contact water that collects in the seepage pond, water from the TMF, and open pit and underground mine water. This water balance should be developed to characterize both the quality and quantity of water that will be managed site wide. The current model was not used to predict water quality. The water balance should be used in conjunction with a site-wide water monitoring program to validate model predictions and confirm that environmental mitigation measures are effective and discharge can occur in accordance with anticipated permit conditions without further treatment.
- During normal operations, the TMF will operate as a zero discharge facility; however, further study should be undertaken to define the conditions under which discharge to the environment is possible to assess the acceptability of this discharge. Further characterization of the tailings material and the potential for ARD/ML is required. The ARD/ML work completed to date is preliminary.
- A seepage and surface runoff collection pond will be constructed down gradient of the WRD that will act as a sediment control pond allowing settling of suspended solids prior to releasing water to be released to the environment. Further study is recommended to characterize the predicted quality and quantity of this waste water to confirm the assumed acceptability of release without additional management or treatment. This study should include further characterization of the waste rock that will be generated by the project and confirm that no further mitigation is required to protect the environment during operations and closure phases. The feasibility study does not currently include an allowance for further treatment of seepage that is released from this pond.
- The establishment of more detailed management and monitoring programs will need to be designed and implemented to support construction, operations, and closure phases of the project. Current plans are conceptual in nature. Potential costs associated with the implementation of these programs have been included as an allowance as part of the G&A costs presented in Section 21. ERM has not evaluated these costs and has not commented on their adequacy.
- The existing planning for closure and reclamation (as presented by others) is currently at a conceptual stage. Further planning and costing is required to support future stages of the Project and validate the assumptions made in the study. Emphasis should be placed on defining the required technical studies needed to inform the evolution of the closure and reclamation plan over the life of the Project such that the end goals are met. Particular attention should be placed on infrastructure decommissioning, the closure of waste facilities and the management of water post-closure, including the potential need for water treatment.

20.7 CLOSURE AND RECLAMATION

Plans for closure and reclamation are addressed by discipline, in other sections of the report.

21.0 CAPITAL AND OPERATING COST ESTIMATES

21.1 CAPITAL COSTS

21.1.1 SUMMARY

The total estimated initial capital cost for the design, construction, installation, and commissioning of all facilities and equipment is $65.0 million. A summary breakdown of the capital cost is provided in Table 21.1 and further capital cost details are provided in Appendix I.

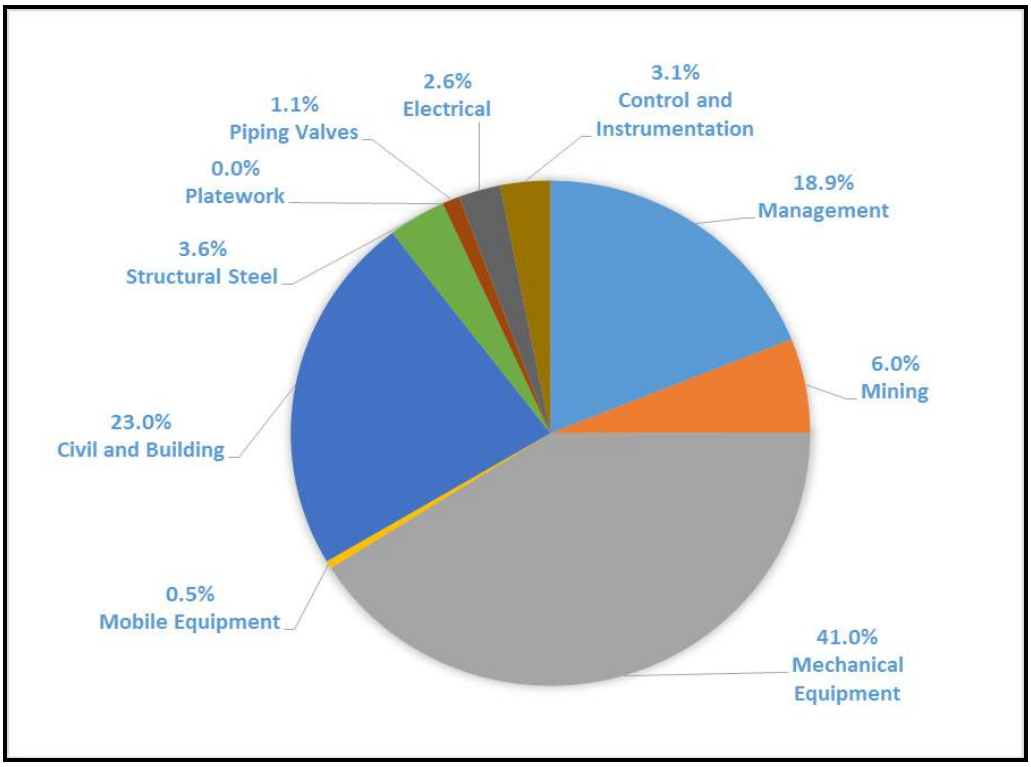
Table 21.1 Capital Cost Summary

Area	Initial ($)	Sustaining ($)	Total ($)
Mining	4,151,164	13102,368	17,253,532
Processing	15,383,403	700,000	16,083,403
Infrastructure	4,584,021	-	4,584,021
Utilities	2,708,651	-	2,708,651
TMF	1,350,566	1,325,776	2,676,342
Site Facilities	7,221,388	-	7,221,388
Off-site Facilities	43,327	-	43,327
Project Indirects	16,388,012	1,016,409	17,404,421
EPCM	5,439,947	20,248	5,460,195
Owner's Cost	7,241,830	100,000	7,341,830
Allowances	462,031	831,232	1,293,263
Total	**64,974 340**	**17,096,033**	**82,070,373**

This estimate includes direct field costs required to execute the Project, plus indirect costs associated with design, construction and commissioning. This estimate is based on pricing as of Q3 2015, with no allowances for inflation or escalation. Elements of actual expenditures until June 2017 have been included, which further reduce the exposure to inflation or escalation. All currencies in this capital cost estimate is expressed in US dollars, unless otherwise noted.

Figure 21.1 illustrated an alternative view of costs per discipline showing the relational costs for the main packages required to establish the mine.

Figure 21.1 Cost Percentages per Discipline



21.1.2 BASIS OF ESTIMATE

CLASS OF ESTIMATE

This estimate is a Class 3 Estimate prepared in accordance with the AACE International (AACE) Cost Estimate Classification System. The estimate accuracy interval is -10% to +15%.

ESTIMATE BASE DATE AND VALIDITY PERIOD

This estimate was initially prepared with a base date of Q3 2015. The quotes used in this feasibility study estimate were originally obtained in Q3 2015 and were updated in accordance with actual procurement information.

Approximately 90% of the initial project capital estimate was committed by the time the estimate was updated, and thus the combined effect of highly competitive pricing by suppliers due to the current economic challenges and level of orders already placed for fixed amounts makes the further potential impact of cost escalation low.

RESPONSIBILITY MATRIX

The Project estimate was produced by Tetra Tech as the lead consultant. The capital cost estimate was developed by a team of engineers, procurement specialists, and cost estimators with inputs from the client, which are presented by company and area of responsibility in Table 21.2.

Table 21.2 Estimate Responsibility Matrix

Area	Company
Mining (open pit)	Tetra Tech
Mining (underground)	Tetra Tech
Processing	Tetra Tech
Infrastructure	Tetra Tech
Site Facilities	Tetra Tech (with input from Silver Bear)
Indirect Costs	Tetra Tech (with input from Silver Bear)
Owner's Costs	Silver Bear
Allowances	Tetra Tech

Estimate Approach

The main allocation of the estimate is the three-tier work breakdown structure, named Area, System and Estimate. Other allocations such as discipline are used as well to assist understanding, analysis and abstraction of information from the estimate. Reporting is done primarily to the work breakdown structure (WBS).

Prices and costs were determined based on international construction knowledge, combined with input from Prognoz obtained from contractors with local knowledge of climate conditions and government restrictions.

The estimate is primarily based on package costs, and client owned and sub-contractor team costs. Quantities not included in package supply, specifically civil and structural elements were estimated separately.

Equipment

Generally, for major equipment, three or more suppliers were approached to provide a budget quotation. The vendors, selected from a Tetra Tech-approved vendor's list, were asked to provide equipment price, spares costs, delivery lead times, and freight costs to a designated marshalling yard for each major equipment package. Long lead item procurement overlapped with the feasibility study activities and where available, the firm equipment prices were used; significantly improving the estimate accuracy in these areas. A few major items were excluded from the standard procurement approach:

- Mining fleet: Silver Bear expressed a preference for Caterpillar as the supplier of the main mining fleet and only their prices were considered, except the trucks, which are Scania (30 t) and Kraz (20 t).
- Electrowinning process: this is a crucial part of the plant and was negotiated with Electrometals Canada Inc. who are specialists in this type of plant, and assisted the planning and design process.
- General allowances: no design or other indicative detail is available.

Other items in the capital cost estimate are based on the following information:

- Actual procurement amounts.
- all equipment and material costs are included as Free Carrier (FCA) manufacturer plant Incoterms 2010; other costs, such as spares, taxes, duties, freight and packaging, are covered in the indirect costs section of the estimate
- actual amounts of orders already placed, where available.

The small scale of the operation combined with a need to keep capital cost low and the site activities compact has meant that Silver Bear can use an owner-managed team to complete general earthworks and road construction. As a result the earthmoving costs were calculated taking into account the use of available materials on site and a team of directly employed personnel forming a fixed monthly cost during the construction period.

It is assumed that ramps and haul roads will be constructed from materials generated during mining activities and that the cost outside mining and discard, which is included in the mining costs, will be limited to some shaping, levelling, and compacting activities only. Unit rates were calculated from database rates, but with the aforementioned low effort for a fit-for-purpose approach taken into account.

CONCRETE

The cost for concrete works is mainly contained in materials purchased, labour expended, and projections to completion. Where applicable for work planned forward, the rate of RUB15 600 ($236,36)/$m^3$ was applied for the material (cement, reinforcement steel and formwork) portion with the labour produced under the team allowance. Aggregates will be supplied on site and the limited costs for preparation thereof included in general site costs.

STRUCTURAL STEEL

The structural steel elements are mostly in design/build packages for which the best available package costs were included. For secondary support and access, steel quantities were assessed based on experience of similar plants.

The steel costs are based on actual purchases.

ELECTRICAL, AND CONTROL AND INSTRUMENTATION INSTALLATION

The cost for electrical infrastructure and distribution consists of the following main elements:

- power generation, for which quotations for containerised generation units were considered
- switchrooms (facilities and switchgear), for which preliminary quotations were compared with experience of costs for similar installation and provisions allowed accordingly
- power distribution, which is localised as power generation units will be placed local to different main areas,

ARCHITECTURAL

The 160-person accommodation camp capital estimate is based on the actual cost of constructed the first half (80-person) of the facility in 2015. Allowances for furniture and equipment are included in the estimate though not indicated separately.

In addition, a specification and schedule were provided outlining the list of structures, their respective sizes, and methods of construction and function for estimating purposes.

The main workshop complex includes all the general facility requirements and was estimated based on expected footprint size, main elements, and considering local package-type costs for similar structures.

CONSTRUCTION AND INSTALLATION

Construction and installation costs were estimated in accordance with the project execution plan, where materials and goods are procured separately from the construction and installation. The construction and installation costs are thus based on the cost of teams for the estimated durations required to execute the works (Table 21.3).

Table 21.3 Construction and Installation Teams

Team	Months on Site
General Construction (medium-sized team focused on earthworks, concrete and building)	32
Main Civils	12
Main Structural, Mechanical, Platework and Piping	12
General Structural, Mechanical, Platework and Piping (smaller works leading commissioning)	8
Electrical and Instrumentation	7
Commissioning Support	2

The cost of accommodation, fuel, electrical power and general site requirements such as medical services are costed in the general provision of such services and not as part of the construction and installation cost.

PIPING

Piping with major supply packages are included in the package costs. The piping excluded from equipment package supply constitutes a minor portion of the estimate. Tetra Tech compiled material take-offs from the process and instrumentation diagrams and process plant model and used historical supply and install rates adjusted for local conditions. Where possible, the original Tetra Tech design rationalised pipe size and material specification to minimise pipe inventory. HDPE is preferred to lined pipe. Heat tracing and lagging are allowed for on long, exposed pipelines. The current cost estimating is mainly based on actual procurement costs.

Platework, Tanks, and Liners

Quantities for all platework and liners for tanks, launders, pump boxes, and chutes, which fall outside lump sum packages, are minor and allowances were made for items outside the packages on estimated quantities multiplied with rates. The minor platework items are estimated to be constructed from site off-cut materials, with larger portions made up on site from raw materials procured directly by the owner.

Mobile Process Equipment

New mobile equipment cost estimates were based on indications from the Prognoz accounting as well as database information for sustaining capital procurement.

Communications

The communications estimate is based on a satellite Internet connection and mobile phones for general site use.

Transport Costs

Transport costs were considered individually for significant items, and further estimated with allowances by items as well as generalised allowances for bulk material transport. Major equipment transport costs were taken from quotations and a further overall lump sum allowance was included to further take into account consideration for the winter road constrictions. The costs include a second winter road period, as well as efforts for summer transport reflected in the actual costs.

Project Currency and Foreign Exchange

The capital cost estimate is expressed in US dollars. Costs submitted in other currencies were converted to US dollars based on the foreign exchange rates listed in Table 21.4.

Table 21.4 Foreign Exchange Rates

Currency	Currency Conversion	Inverse Rate
AUD	0.7255	1.378322
CAD	0.7645	1.307993
EUR 1	1.1150	0.896837
EUR 2	1.1146	0.897183
EUR 3	1.0900	0.917431
EUR 4	1.0900	0.917431
GBP	1.4452	0.691951
GBP 1	0.7092	1.410000
USD	1.0000	1.000000
ZAR	0.0638	15.678896

An allowance was included for the Euro and Ruble at various rates related to respective alternate exchange rates, allowing for instances where a specific contract, agreement or payment is already fixed and the base rate thus not subject to current day fluctuations. The items going forward in Ruble is set at RUB66 to the American

Dollar. The Ruble rates of completed transactions ranged from RUB55 to RUB77.93 to the dollar.

Duties and Taxes

Duties and taxes are included in this estimate as part of the project indirect costs.

Measurement System

The International System of Units (SI) measurement system is the standard measurements system used in this estimate.

Closure Costs

The activities to perform the majority of the mine closure will use personnel and machinery already accounted for in the operating and capital costs. Earth moving and fabrication of physical barriers, making machinery safe, emptying vessels and moving waste materials to the waste dump will be systematically performed as final production ramps down at the end of mine life.

21.1.3 Direct Costs

Open Pit Mining Capital Costs

The initial capital costs are shown in Table 21.5.

Table 21.5 Initial Open Pit Capital Costs

Area	Cost (US$)
Pre-production	
Labour	616,920
Blasting	569,660
Maintenance and Fuel	255,388
Leasing	1,077,401
Pre-production Subtotal	**2,519,369**
Mining Equipment	
Open Pit Engineering Equipment	1,339,329
Total Open Pit Capital Cost	**4,151,164**

Underground Mining Capital Costs

The underground mining capital costs are shown in Table 21.6.

Table 21.6 Underground Mining Capital Costs

Description	Year 3 (US$)	Year 4 (US$	Year 5 (US$)	Year 6 (US$)	Total (US$)
Lateral	1,494,974	826,665	888,122	399,362	3,609,123
Raisebore	399,387	247,419	303,190	1,036,376	1,036,376
Surface Infrastructure	-	-	-	-	-
Underground Infrastructure	2,693,318	-	-	2,693,318	2,693,318
Underground Mobile Equipment	1,787,061	2,965,710	-	4,752,71	4,752,771
Ventilation System	-	-	-	-	-
Total	**6,374,740**	**4,039,794**	**974,312**	**702,742**	**12,091,588**

PROCESS PLANT

Tetra Tech provided a detailed equipment list with the description, size, and unit cost for each piece in accordance with the process flow sheets and equipment lists. Budget quotations were obtained for all the following major items based on preliminary specifications. Table 21.8 outlines the major procurement packages. Purchasing contracts have been negotiated and awarded for the regrind mill, filter press, ball mill and the electrowinning packages and other major equipment.

To provide an indication as to the scale of the Project, the key quantities that were derived as part of the feasibility study design are shown in Table 21.7. The actual values may be slightly different following detail design and procurement in the months following completion of the study. The effect of these changes have been discounted in the capital estimate following a review of the project records.

Table 21.7 Key Quantities

Description	Unit	Quantity
Concrete	m^3	3,700
Structural Steel	t	1,200
Platework (excluding leach tanks)	t	95
Piping	m	6,500
Electrical and Earthing Cables	m	47,100

Quotations were obtained and analysed from the vendors shown in Table 21.8. Costs from the selected vendor (highlighted in grey) are included in the estimate.

Table 21.8 Mechanical Equipment Procurement Packages

Item	Procurement Package Name	Vendor
1	SB15005 Regrind Mill	Metso Minerals
		Glencore
		Solo Resources
		Outotec
		FLSmidth
		ThyssenKrupp
2	SB15020 Filter Press	Tenova Delkor
		Bilfinger
		Outotec
		Solo Resources
3	SB15007 Intense Leach Reactor and Gravity Concentrator	FLSmidth Knelson Technologies
		Outotec CIS
		Solo Resources
		Gekko Systems RSA
4	SB15002 Electrowinning	Electrometals
5	SB15012 Crusher Plant	Metso Minerals
		PT MIning
		Citic Heavy Industries
		Austraco HK
		FLSmidth
		Osborn Engineered
		ThyssenKrupp
		Souyz Gornykh Mashin (actual supplier)
6	SB15021 Thickener	Tenova Delkor SA
		Metso Minerals
		FLSmidth
		Outotec
7	SB15013 MCC Switchrooms	ABB Ltd (Prognoz)
		Siemens LLC (Prognoz)
		Vostochnaia Technica (Prognoz)
		I-control Consulting
		Denwa Engineering
8	SB15001 Ball Mill	FL Smidth
		Outotec
		Thyssenkrupp
		Metso
		Citic
9	SB15066 Ball Mill Logistics	ICTL Belgium
		SDV South Africa
		STS Logistics Russia
		UTI

Table 21.9 provides a two-tier summary of the initial capital cost estimate for the processing area by system. Included in each of the elements are the civil, structural

and electrical work required to supply and install. Transport of equipment is captured in project indirect costs.

Table 21.9 Processing Capital Cost Estimate

Area	Cost ($)
Crushing and Stockpiling	
Primary and Secondary Crusher Package	451,552
Grinding	
Ball Mill	1,002,599
Regrind Mill	96,140
Conveyors	688,907
Ancillary Equipment	367,341
General	64,790
Subtotal	2,219,777
Gravity Concentration and ILR	
Gravity Concentration	364,357
ILR	421,359
Subtotal	785,716
Leaching	
Tanks	499,600
Leach Tank Agitator	283,006
General	743,038
Subtotal	1,525,644
Decantation Washing	
Thickening	433,209
Filter Press	836,463
General	325,103
Subtotal	1,594,775
Electrowinning and Smelt House	
Electrowinning	2,711,046
Smelt House	0
Subtotal	2,711,046
Reagents	
General	301,507
Plant Infrastructure and Utilities	
Weather Shield	5,446,730
Subtotal	5,446,730
Mobile Equipment	
Materials Handling Equipment	346,656
Total Processing Capital Costs	**15,383,403**

The short mine life and simplicity of the processing plant does not warrant substantial sustaining capital investment as most components will last the full LOM. An amount of $280,000, approximately 2.0% of the equipment cost, was allowed for in each of the years three and five. An amount of $140,000 was allowed for in the first year of production as a provision for plant optimisation.

General Infrastructure and Facilities

The general infrastructure includes bulk earthworks, roads, power generation, communications, site water management, heating, fire management and ancillary buildings (Table 21.10). Site facilities include buildings and fuel storage (Table 21.11). An allowance of $101,464 is included for upgrade of the off-site port facility at Batamai.

Table 21.10 Infrastructure Capital Cost Estimate

Area	Cost ($)
Site Preparations	
Bulk Earthworks	Included
Roads and Access	
Site Roads	Included
Power Generation	
Establishment	1,569,173
Fuel Storage	45,247
Subtotal	1,614,420
Overall Site Controls and Communications	
Communications	586,723
Process Control	200,906
Subtotal	787,629
Site Water Management	
Site Drainage	Included
Cleaning Ponds	12,053
Water Supply	488,590
Subtotal	500,643
Heating	
Boiler	617,398
Fire Management	
Water Supply	68,636
Equipment	995,295
Subtotal	1,063,931
Total Infrastructure Capital Costs	**4,584,021**

Table 21.11 Site Facilities Capital Cost Estimate

Area	Cost ($)
Ancillary Buildings	
Accommodation Complex	3,308,378
Cyanide Store	715,773
Explosives Storage and Mixing	354,215
Heavy Vehicle and Maintenance Workshop	1,094,402
Containerised Store	116,239
Subtotal	5,589,007
Fuel Farm	
Fuel Storage	1,632,381
Total Site Facilities Capital Costs	**7,221,388**

Table 21.12 Utilities Capital Cost Estimate

Area	Cost ($)
Overall Electrical	
Distribution	1,314,864
Sewerage	
Sewage Collection and Treatment	213,705
Water Supply	
Water Supply	933,588
Raw Water	223,428
Potable Water	12,839
Subtotal	**1,169,855**
Domestic Waste	
Waste Collection and Storage	10,227
Total Utilities Capital Costs	**2,708,651**

Waste Rock Dumps, Ore Stockpiles, and Flood Protection Berms

A source of benign waste rock, no permanent ore stockpiles and the year-round low to very low ambient temperature means that these material heaps will not require lining. Any minor earthworks, protections berms, and diversion trenches will be created as required by the Silver Bear site construction team. The cost of this has been incorporated in construction team costs and an extra over allowance for the waste rock dump settlement pond construction.

Tailings Management Facility

SRK has prepared MTOs to allow for the development of a cost estimate for all capital, sustaining capital and operational expenditure items associated with construction of the TMF and clarification pond. The capital and sustaining capital costs have been sub-divided in accordance with Table 21.13.

Table 21.13 TMF Capital and Sustaining Capital Expenditure Summary

Description	Comments
Capital Costs (Year -1)	• perimeter bund construction • liner installation • pond excavation • pond lining • front end loader (not costed: will use plant equipment) • pump purchase
Sustaining Capita Costs (Year 9)	• closure costs • liner installation • slope regrading • capping installation • topsoiling

The total initial and sustaining direct capital cost estimates are $1,350,566 and $1,325,776 respectively. In additional provision is made in the estimate for indirect capital costs. As a proportion of the direct cost, an allowance was made in each case for an additional approximately 2% for design engineering, 5% for contractor mobilisation and demobilisation $151,856 contingency on the sustaining capital cost estimates. The contingency for initial capital is calculated within the overall contingency.

Mine Closure

Once the mining operation has stopped and the remaining ore in the stockpiles is processed, machinery and personnel will be reassigned to complete the earthworks required for mine closure. Operating costs of the machinery have taken into account the requirement to keep machinery running when not in active duty to prevent freezing. This time will be utilised for the mine closure activities.

Under Owner's Cost, a cost of $100,000 has been allocated for crew, fuel and equipment to monitor site for 1 year post closure. The personnel will live in the local village and travel to site on a weekly basis to monitor the site and take environmental samples.

Table 21.14 Mine Closure Cost Estimate

Area	Cost ($)	Notes
Mining Underground	-	Included in operating costs
Mining Open Pit	-	Included in operating costs
Process Complex	-	Included in operating costs
TMF	1,325,776	As detailed above
Infrastructure	100,000	Crew, fuel and equipment for 1 year monitoring and sampling at site
Total Closure Costs	**1,425,776**	-

21.1.4 INDIRECT COSTS

The indirect capital costs include project indirect costs, EPCM, Owner's cost, and allowances. Table 21.15 shows the summary of the project indirect costs.

Table 21.15 Indirect Capital Cost Estimate

Area	Cost ($)
Construction Indirects	
General	2,252,703
Spares	
Strategic Spares	536,029
Operational Spares	222,735
Subtotal	758,764
Initial Fills	
General	99,933
Lubricants	33,929
Subtotal	133,862
Freight and Logistics	
Transport	7,159,084
Taxes and Duties	145,664
Subtotal	7,304,748
Vendors	
General	50,000
Financials	
Insurance	118,140
Construction and Installation	
General	5,769,795
Total Indirect Capital Costs	**16,388,012**

In addition, SRK estimated that the TMF construction indirect costs for the initial and sustaining capital items will be $93,863 and $70,867 respectively, which is incorporated with the overall project indirects.

Even though the project will be executed in a bespoke owner managed execution strategy, Tetra Tech has categorised the cost associated with engineering, procurement and construction management in a conventional way for ease of comparison. EPCM costs constitute approximately 9.0% of direct capital cost (excluding the mining) and is shown in Table 21.16.

Table 21.16 Initial Capital EPCM Cost Estimate

Area	Cost ($)
Engineering and Design	3,658,858
Procurement and Expediting	416,744
Construction Management	1,364,345
Total EPCM Capital Costs	**5,439,947**

SRK estimates that the engineering cost associated with the initial and sustaining capital work will be $26,818 and $20,248, respectively. The $26,818 is included in the initial capital EPCM cost shown in Table 21.16.

Owner's Cost

The Owner's Cost makes allowance for Owner activities associated with the Project, and includes allocations of head office costs for Prognoz in accordance with the significant attention to Project matters by Prognoz.

A general allowance was made for managing and executing the Project. Dedicated allowances were made for insurances and operational readiness.

Table 21.17 shows the summary of the Owner's costs.

Table 21.17 Owner's Cost Capital Cost Estimate

Area	Cost ($)
Construction Management	54,214
Operational Readiness	38,138
Owner's Cost	6,319,562
Licensing and Permits	829,916
Total Owner's Cost Capital Costs	**7,241,830**

Allowances

The cost provisions for cost to completion were generally allocated to the appropriate areas. The remaining allowance of $462 031 is for the additional scope for an Assay laboratory and a tacheometer.

The sustaining capital estimate includes the following contingency allowances:

- Underground Mining Equipment (Year 4) $599,746
- Underground Mining Equipment (Year 5) $45,331
- Underground Mining Equipment (Year 6) $34,299
- TMF Closure (Year 7) $151,856.

21.1.5 QUALIFICATIONS – ASSUMPTIONS AND EXCLUSIONS

QUALIFICATIONS

In developing the capital cost estimate, Tetra Tech made the following assumptions:

- concrete aggregate and suitable backfill material will be sourced locally
- construction activities will be continuous and executed primarily with locally sourced construction and installation teams
- bulk materials such as cement, reinforcing steel, structural steel and plate, cable, cable tray, and piping will be available on site when required.

EXCLUSIONS

The following items were excluded from the capital cost estimate:

- cost escalation during construction
- piling foundations other than any costs included in the camp construction actual expense lump sum amount
- airstrip and helipad other than any costs included in the camp construction actual expense lump sum amount
- winter road and maintenance
- new roads outside plant area
- weighbridge
- interest on loans and other financing costs during construction
- schedule delays and associated costs beyond December 2017 for the initial capital costs, such as those caused by the following:
  - large scale unexpected ground conditions
  - extraordinary climate events
  - labour disputes
  - receipt of information beyond the control of EPCM team
  - schedule recovery or acceleration
  - financing costs
  - taxes (except as supplied by Owner)
  - cost outside Tetra Tech's battery limits
  - sunk costs
  - research and exploration drilling costs
  - force majeure
  - currency fluctuations.

The following items have been excluded from the capital cost estimate but are included in the financial model:

- sustaining capital
- working capital
- closure costs
- salvage values
- operating costs.

21.2 OPERATING COST ESTIMATE

21.2.1 SUMMARY

The operating cost estimate for the Project consists of mining, processing, G&A (which includes TMF and site water management costs).

The total LOM average operating cost is estimated to be $153.78/t ore processed. The operating cost estimate is expressed in US dollars, unless specified. Table 21.18 summarises the unit operating costs of each area.

Table 21.18 LOM Averaged Operating Cost Summary

Area	Unit Cost ($/t processed)
Mining	59.61
Processing	52.02
G&A	42.16
Total	**153.79**

Note: Total may not add due to rounding.

21.2.2 Basis of Operating Cost Estimate

Labour

The permanent work force for the Project will be as local to Yakutsk as possible, and certainly from within Russia. From time to time, international specialist consultants will be engaged for short periods to resolve specific issues; however, their cost has been accounted for separately.

Personnel on the mine site will work on a four-week-on/four-week-off roster. The on- and off-cycles start when the workers report for duty at and when they are returned to the office in Yakutsk. In general, personnel are only paid during the on-cycle.

In accordance with Russian Federation employment practise, working hours and subsequent remuneration is controlled on a daily basis. The first eight hours are considered normal working hours and are included in the basic wage. The next two hours are paid at one and half the basic rate, and any subsequent hours are paid at double the basic rate. No one is allowed to work a shift of longer than 12 hours for more than two days in a row. The basic rate is based on the annual basic salary earned over 1,968 h/a (the so called norm hours).

Each employee is entitled to 52 days of annual leave of which up to 28 days can be taken during the on-cycle of the work rotation. The remainder is paid leave during the off-cycle. Public holidays falling on the on-cycle will be remunerated at double the basic rate, while those that fall on the off-cycle are unpaid.

Personnel working on the mine site generally fall into three categories: shift workers, day shift workers, and senior management. Personnel from the senior management group are not replaced when they are on the off-cycle, but rely on a senior subordinates or fellow managers to act in their stead. As a result, this group's roster is staggered to ensure maximum cover by senior personnel is maintained. In general, rosters will be staggered to provide operational continuity and mitigate the risk of severe disruption during flight delays.

Technical services will be common to mining, processing, and infrastructure to minimise redundancy. This is particularly important when providing cover for annual leave and illness.

The cost of each role on the mine site has been calculated to include basic and overtime pay (including income tax and social insurance) and takes into account annual leave, leave replacement cost, public holidays, and the number of people that would have to be employed to cover the shift and roster requirements. For example, the basic annual salary (related to the norm hours) for a plant operator is RUB840,000, whereas the cost of one plant operator shift position is RUB5,375,233 and requires the employment of four people. Table 21.20 shows the list of roles used to estimate the operating cost for the Project.

Labour, tax and social insurance rates were provided by Prognoz.

Fuel and Power

Winterised diesel oil will be used for all internal combustion engines and boilers on site. Fuel will be source from any number of local supplier and transported by truck to site. In Q4 2015 Prognoz paid RUB61,700/t for fuel on site.

The power cost was then estimated using an assumed power cost of $0.21/kWh, which assumes on-site power generation as shown in Table 21.19.

Table 21.19 Power Generation Cost Estimate

Cost Component	Unit	Value
Generator Fuel Consumption	l/kWh	0.26
Fuel Price on Site	$/l	0.77
Power Generation Fuel Cost	$/kWh	0.20
Power Generation Non-fuel Cost	$/kWh	0.01
Total Power Generation Cost	$/kWh	0.21

Table 21.20 Personnel Roles

Band	Sub-grade	Role	Labour Type	Title	Typical Roles	2016 Salary incl. Tax and Insurance (RUB/a)	Overtime Rate (RUB/a)	Total Payroll (RUB/a)	Leave Replacement Cost (RUB/a)	Total Cost to Company (RUB/a)	Cost/Shift Position (RUB/a)	Cost/Day Shift Position (RUB/a)	Cost/ Management Position (RUB/a)
F	2	Policy Making	Top Management	C-Level Management	CEO	-	-	-	-	-	-	-	-
	1				CFO	-	-	-	-	-	-	-	-
E	3	Programming	Senior Management	General Manager	General Director	4,200,000	1,539,181	5,739,181	-	5,739,181	22,956,725	11,478,363	5,739,181
	2				Mine Manager/Operations Director	3,840,000	1,407,251	5,247,251	-	5,247,251	20,989,006	10,494,503	5,247,251
	1				Plant Manager	3,480,000	1,275,322	4,755,322	-	4,755,322	19,021,287	9,510,643	4,755,322
D	3	Interpretive	Middle Management	Department Manager, Superintendent	Senior Engineer	3,000,000	1,099,415	4,099,415	-	4,099,415	16,397,661	8,198,830	4,099,415
	2				Engineer	2,700,000	989,474	3,689,474	-	3,689,474	14,757,895	7,378,947	3,689,474
	1				Engineering supervisors/Junior Engineer	2,400,000	879,532	3,279,532	-	3,279,532	13,118,129	6,559,064	3,279,532
C	3	Routine	Skilled	Supervisor, Technician	Foreman/Shift Boss	1,800,000	659,649	2,459,649	419,940	2,879,589	11,518,357	5,759,178	2,879,589
	2				Senior Technician	1,500,000	549,708	2,049,708	349,950	2,399,658	9,598,631	4,799,315	2,399,658
	1				Technician	1,200,000	439,766	1,639,766	279,960	1,919,726	7,678,905	3,839,452	1,919,726
B	2	Automatic	Semi-skilled	Charge hand, Machinist	Operator, Cook, Security, Truck Driver	840,000	307,836	1,147,836	195,972	1,343,808	5,375,233	2,687,617	1,343,808
	1				Cook Assistant	780,000	285,848	1,065,848	181,974	1,247,822	4,991,288	2,495,644	1,247,822
A	2	Defined	Unskilled	Labourer	Cleaner, Laundry	660,000	241,871	901,871	153,978	1,055,849	4,223,398	2,111,699	1,055,849
	1					600,000	219,883	819,883	139,980	959,863	3,839,452	1,919,726	959,863

21.2.3 Open Pit Mining

The open pit operating costs have been estimated based on project-specific information provided by Silver Bear, and are described in the following subsections.

Labour Costs

Silver Bear provided the labour rates in a Patterson scale type format. Positions were allocated a grade and shift roster to generate the monthly cost of position, including burden and all costs Silver Bear. Table 21.21 shows the number of positions for each year, and the cost per month to operate the open pit. Some positions were shared with other cost centres, which accounts for the non-integer values.

Table 21.21 Open Pit Labour

Position	Grade	Shift	2016	2017	2018	2019	2020	Cost RUB/mo	Cost USD/mo
Mine Personnel									
Drill Operator	B2	shift	3.0	3.0	3.0	2.0	1.00	447,936	6,787
Drilling Assistance	B1	shift	3.0	3.0	3.0	2.0	1.00	415,941	6,302
Backhoe Operator	B2	shift	2.0	3.0	3.0	2.0	1.00	447,936	6,787
Truck Driver	B1	shift	6.0	9.0	9.0	8.0	3.00	415,941	6,302
Truck Driver (Road Building)	B1	day	1.0	1.0	1.0	1.0	1.00	207,970	3,151
Dozer Operator	B2	shift	3.0	4.0	4.0	3.0	2.00	447,936	6,787
Grader Operator	B2	day	0.5	1.0	1.0	1.0	0.50	223,968	3,393
Loader Operator	B2	shift	0.5	0.5	0.5	0.5	0.50	447,936	6,787
Watercart driver	B1	day	1.0	1.0	1.0	1.0	1.00	207,970	3,151
Blaster	C3	day	1.0	1.0	1.0	1.0	1.00	479,932	7,272
Blasting Crew	C1	day	2.0	2.0	2.0	2.0	1.00	319,954	4,848
Helper	A2	shift	2.0	2.0	2.0	2.0	1.00	351,950	5,333
Mine Personnel Total	-	-	**25.0**	**30.5**	**30.5**	**25.5**	**14.0**		
Mine Technical Office									
Mining Engineer	D3	day	2.0	2.0	2.0	2.0	2.00	683,236	10,352
Geologist	D1	day	2.0	2.0	2.0	2.0	0.25	546,589	8,282
Geology Technician	C1	day	2.0	2.0	2.0	2.0	1.00	319,954	4,848
Surveyor	C2	day	1.0	1.0	1.0	1.0	0.25	399,943	6,060
Survey Crew	B2	day	1.0	1.0	1.0	1.0	0.50	223,968	3,393
Labourer	A2	day	1.0	1.0	1.0	1.0	0.50	175,975	2,666
Mine Technical Office Total	-	-	**9.0**	**9.0**	**9.0**	**9.0**	**4.50**		
Workshop									
Mechanic	C1	shift	1.0	1.0	1.0	1.0	0.50	639,909	9,696
Turner and Fitter	C1	day	1.0	1.0	1.0	1.0	0.50	319,954	4,848
Field Mechanic	C1	day	1.0	1.0	1.0	1.0	0.50	319,954	4,848
Workshop Total	–	-	**3.0**	**3.0**	**3.0**	**3.0**	**1.50**		
Total Open Pit Labour	-	-	**37**	**42.5**	**42.5**	**37.5**	**20.00**		

Drilling and Blasting Costs

Open pit blasting parameters were estimated using the data shown in Table 21.22. The powder factors take into account that ore must be more finely fragmented than the waste material.

Table 21.22 Open Pit Blasting Parameters

Parameter	Unit	Waste	Ore
Hole Diameter	m	0.076	0.076
Burden	m	1.9	2.2
Spacing	m	2.1	2.5
Bench Height	m	10	10
Sub-drill depth	m	1.216	1.216
Volume of Rock to be Broken	m^3	39.9	55
Stemming Length	m	2.28	2.28
Powder Factor	kg/m^3	0.64	0.88
Explosive Weight per hole	kg	35.00	35.00
Tonnes Broken per Blasthole	t	151.25	126.483
Metres Drilled per Blasthole	m	12.28	12.28
Re-drilling	%	5	5
Total Metres Drilled per Tonne of Rock	m	0.09	0.10
Total Holes Drilled per Tonne of Rock	-	0.0069	0.0083

The powder factors in Table 21.22, along with local rates provided by Silver Bear, were used to determine the drilling and blasting costs per tonne as shown in Table 21.23.

Table 21.23 Drilling and Blasting Costs

Description	Unit	Waste	Ore
Exchange Rate	RUB/USD	66	66
ANFO	RUB/kg	50	50
	USD/kg	0.76	0.76
Detonator and Fuses	$/hole	1.52	1.52
Booster	$/hole	0.73	0.73
Detonating Cord	$/hole	1.83	1.83
Contingency (5% on damaged ignition items)	$/hole	0.41	0.41
Drill Bit	$/h	7.9	7.9
Shank Adapter	$/h	3.5	3.5
Drill Rods	$/h	5.1	5.1
Couplings	$/h	2.125	2.125
Hammer	$/h	3.5	3.5
Drill Consumable Cost	**$/h**	**22.13**	**22.13**
	$/t	**0.10**	**0.12**
Blasting Cost (excluding labour, maintenance, and fuel)	**$/t**	**0.22**	**0.26**

Equipment Maintenance and Fuel Costs

Equipment maintenance and fuel costs are based on the estimated working hours for each unit in the mining fleet. Truck operating hours are based on planned haul distances, cycle times, and number of cycles required to match scheduled production. Excavator operating hours are matched to the number of trucks loading cycle times. Drill rig operating hours were estimated based on the drill and blast calculations shown in Section 5.2.2. During winter months most equipment is assumed to be continually operating to avoid freezing due to the extreme temperatures.

The truck cycle times and fuel usage were estimated using the parameters shown in Table 21.24 combined with haul distance determined by bench location and type of load.

Table 21.24 Truck Cycle Times and Fuel Usage Parameters

Parameter	Unit	Value
Speed on Bench	km/h	20.00
Speed on Ramp	km/h	15.00
Speed on Road (flat)	km/h	30.00
Speed on WRD	km/h	30.00
Speed on OSP	km/h	30.00
Speed on ROM Pad	km/h	15.00
Heavy Fuel Consumption	l/hr	28.00
Normal Fuel Consumption	l/hr	22.00
Idle Fuel Consumption	l/hr	12.00
Ore Truck Loading Cycle Time	min	6.00
Ore Truck Dump Cycle Time	min	0.75
Waste Truck Loading Cycle Time	min	3.00
Waste Truck Dump Cycle Time	min	0.75

Fuel and maintenance costs were estimated on cost per hour using a delivered-to-site fuel cost from Silver Bear of $0.77/l. The average fuel consumption and maintenance costs are shown in Table 21.25.

Table 21.25 Fuel Consumption and Maintenance Costs

Equipment	Type	Fuel Consumption	Maintenance Spares (USD/h)
Truck	Kraz 65032	22 l/h	8
Backhoe	CAT 336	35 l/h	19
Drill	AtlasCopco T40	26 l/h	31
Trackdozer 1	CAT D9	60 l/h	48
Trackdozer 2	CAT D8	42 l/h	26
Grader	CAT 14M	23 l/h	26
Water Cart	Ural 4320	20 l/h	10
Service truck	Ural	12 l/h	3
Fuel Truck	Ural	20 l/h	3
Lowbed truck	Ural	20 l/h	3
ANFO Truck	UAZ	20 l/h	7
Loader	CAT 950	15 l/h	10
Pickup Truck	UAZ	15 l/d	1
Lighting Units	-	25 l/d	2

Leasing Costs

Leasing costs are based on current monthly leasing costs for existing leased equipment, with the drill rigs assumed to have the same lease costs as the CAT 336 shovel. The CAT D8R lease cost has been pro-rated against the D9R lease cost, based on capital cost. Monthly leasing cost are shown in Table 21.26

Table 21.26 Monthly Leasing Costs

Equipment	Type	Leasing Cost (USD/mo)
Backhoe	CAT 336	2,665
Drill	AtlasCopco T40	2,665
Trackdozer 1	CAT D9	6,564
Trackdozer 2	CAT D8	4,400
Grader	CAT 14M	3,100

Open Pit Costs

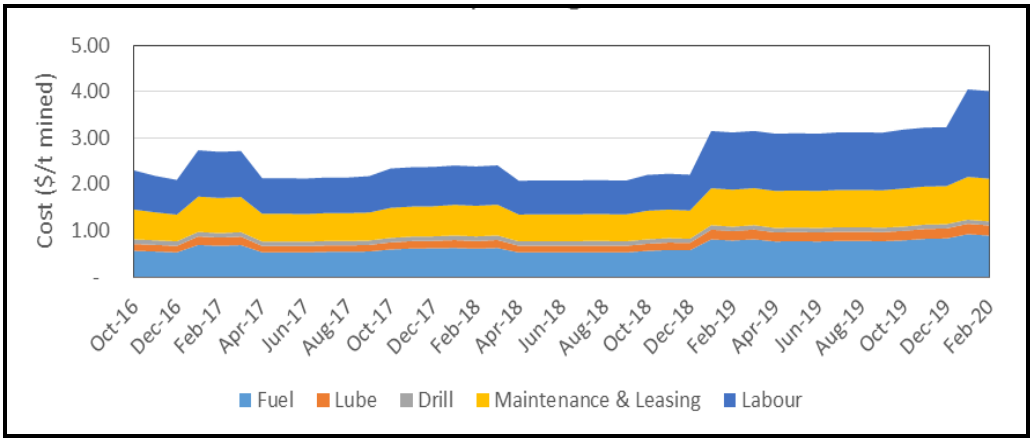
The average mining operating unit costs per tonne of material moved, which includes material moved from the ROM pad to the ROM bin are shown in Table 21.27.

Table 21.27 Average Open Pit Mining Unit Operating Costs

Item	Cost (USD/t mined)
Fuel	0.64
Lube	0.17
Drill	0.09
Blast	0.20
Maintenance	0.67
Labour	0.97
Total Cost per Tonne	**2.74**

Figure 21.2 shows how the cost per tonne of material moved changes over the LOM with varying production rates and haulage distances.

Figure 21.2 Average Open Pit Mining Unit Operating Costs



21.2.4 UNDERGROUND MINING

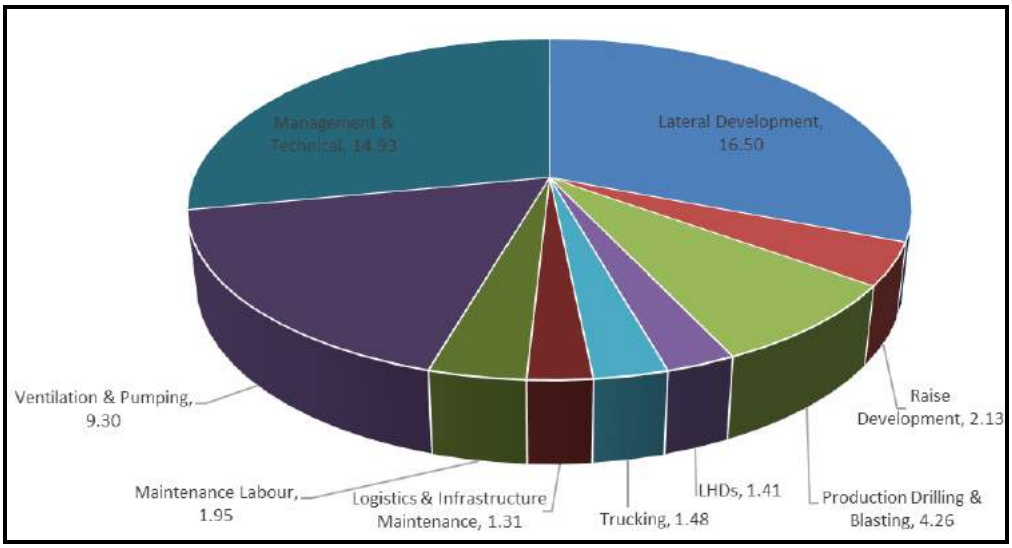
Table 21.28 summarises the underground mine operating costs.

Table 21.28 Underground Mining Operating Costs

Area	Cost (USD/t)
Lateral Development	16.50
Raise Development	2.13
Production Drilling & Blasting	4.26
LHDs	1.41
Trucking	1.48
Backfill	0.00
Crushing & Conveying	0.00
Logistics & Infrastructure Maintenance	1.31
Maintenance Labour	1.95
Ventilation & Pumping	9.30
Management & Technical	14.93
Total Underground Costs	53.27

The underground mining operating costs are also presented in Figure 21.3.

Figure 21.3 Underground Mining Operating Costs



The lateral development costs accounts for approximately 31% of the total underground mining cost, with management and technical staff costs (including the management consumable costs) accounting for another 28% of the total underground mining costs.

The development activities and the management and technical staff should therefore be the focus of cost improvement initiatives pertaining to underground mining at Vertikalny.

21.2.5 PROCESS

The process operating cost includes the following estimated components:

- labour (operations and maintenance)
- reagents and consumables (plant)
- maintenance spares
- power.

The total plant operating cost for treating the oxide ore using the proposed flowsheet (Section 17.0) is estimated at $52.02/t. The process operating cast for the primary ore is estimated at $123.71/t.

The process plant operating cost summary for the oxide and primary ores is presented in Table 21.29.

Table 21.29 Process Plant Unit Operating Cost Summary

Area	Unit Cost Oxide Ore ($/t processed)	Unit Cost Primary Ore ($/t processed)
Labour	8.28	8.28
Reagents and Consumables	19.13	87.31
Maintenance and Spares	2.17	2.17
Power	23.44	25.95
Total	**52.02**	**121.78**

LABOUR

A summary of the process plant labour costs for both the oxide and primary ore is shown in Table 21.30.

Table 21.30 Plant Labour Cost Estimate

Role	Rate Description	Type of Shift	No. of Positions Required	No. of Personnel in Camp	No. of Payroll Personnel	Total Unit Cost (RUB/a)	Total Unit Cost ($/a)	Total Cost ($/a)	Cost ($/t)
Plant Manager	Chief Plant Engineer	Manager	1	1	1	4,755,322	72,050	72,050	0.66
Metallurgist	Engineer	Day Shift	1	1	2	7,378,947	111,802	111,802	1.02
Plant Superintendent	Technical Foreman	Day Shift	1	1	2	5,759,178	87,260	87,260	0.79
Plant Operators	Operations Clerk	Shift	5	10	20	5,375,233	81,443	407,215	3.70
Electrician	Technician	Day Shift	1	1	2	3,839,452	58,174	58,174	0.53
Fitter	Technician	Say Shift	2	2	4	3,839,452	58,174	116,347	1.06
Boiler Maker	Technician	Day Shift	1	1	2	3,839,452	58,174	58,174	0.53
Total	-	-	**12**	**17**	**33**	34,787,038	527,076	911,022	8.28

Reagents and Consumables

Reagent consumptions were estimated based on the feasibility study test work results, from experience in similar projects, or using industry standard assumptions. The annual reagent consumptions were calculated by multiplying the hourly consumptions by the plant operational hours. The unit cost of some of the consumables and reagents were provided by client. Cost data from the Tetra Tech database were used for the remaining consumables, assuming a free on board (FOB) price for delivery in Yakutsk. The shipping cost from Yakutsk to the Project site was applied at a rate of $310/t of material. The summary of the reagent and consumable cost estimation is shown in Table 21.31.

Table 21.31 Reagent and Consumables Cost Estimate

Description	Units	Cons.	Cost ($/unit)	Cons (tons/a)	Cost ($/a)	Shipping Cost ($/a)	Annual Cost ($/a)	Annual Cost ($/t)
Primary Crusher Liners	set/a	1	25,000	3	25,000	775	25,775	0.23
Cone Crusher Liners	set/a	1	40,000	3	40,000	775	40,775	0.37
Mill Liners	kg/t	0.06	5.5	6	34,130	1,603	35,733	0.32
Grinding Balls	kg/t	0.7	0.64	72	46,329	18,613	64,942	0.59
Lime	t	1.2	0.1	132	17,160	34,100	51,260	0.47
Cyanide	t	6.8	2.23	753	1,679,109	194,516	1,873,625	17.03
Flocculant	t	0.03	4	3	11,275	710	11,985	0.11
Total	**-**	**-**	**-**	**-**	**-**	**-**	**2,104,095**	**19.13**

Note: Cons. = consumption

Maintenance and Spares

Maintenance and spares were calculated assuming 3% of the cost of the mechanical equipment. The maintenance and spares cost estimation is shown in Table 21.32.

Table 21.32 Maintenance and Spares Cost Estimate

Description	Unit	Cost
Mechanical Equipment Cost	$	7,963,779
Three percent of Equipment Cost	$	238,913
Unit Operating Cost	$/t	2.17

Power

Annual power consumption in was calculated in kilowatt hours by multiplying the absorbed power requirements for each area by the annual hours of operation. The summary of the power cost estimate is shown in Table 21.33.

Table 21.33 Power Cost Estimate

Description	Absorbed Power (kW)	Annual Power Consumption (kWh)	Cost ($/a)	Cost ($/t)
Crushing	148	972,360	203,223.2	1.85
Milling	786	6,048,955	1,264,231.6	11.49
Leaching	404	2,770,403	579,014.1	5.26
Washing and Clarification	114	915,525	191,344.8	1.74
Electrowinning	72	580,262	121,274.8	1.10
Sludge Drying	27	135,605	28,341.4	0.26
Utilities and Reagents	48	386,842	80,849.9	0.73
Total	**1,664**	**12,338,434**	2,578,733	22.44

The detailed breakdown of the process operating cost for the primary ore can be seen in Appendix J.

21.2.6 TAILINGS MANAGEMENT FACILITY

The operating expenditure components for the TMF have been sub-divided as per Table 21.34.

Table 21.34 TMF Operating Expenditure Summary

Description	Comments
Operating Costs (Year 1-8)	• haulage of tailings from plant stockpile, placement, dozer rework to form dry stack facility • clarification pond water treatment • pump costs

The annual operating expenditure for the TMF is $39,175. This includes costs associated with clarification pond, treatment and pumping of excess fluids from the Clarification Pond.

The costs associated with haulage of tailings from the plant stockpile to the TMF have been captured in the mining operating costs, as the vehicles from the same fleet will be utilized for this task.

The operating costs do not include allowance for environmental waste disposal charges as these are being investigated by the Client. The Mangazieisky tailings have been preliminarily classified as Class III waste which could incur a levy of $7.88/t of tailings.

21.2.7 GENERAL AND ADMINISTRATIVE

SUMMARY

The average life of mine G&A costs are estimated at $42.16/t processed (Table 21.35) and consist of two components:

- labour – general manager and staffing in accounting, purchasing, and environmental departments, and other G&A departments
- G&A expenses, including insurance, administrative supplies, medical services, legal services, human resources related expenses, travelling, accommodation/camp costs, air crew transportation, and external assay/testing.

Table 21.35 Average LOM G&A Operating Costs

Description	Cost ($/a)	Cost ($/t processed)
Labour	1,757,834	16.11
Expenses	2,843,718	26.06
Total	**4,601,552**	**42.16**

G&A varies over the mine life and the numbers shown in Table 21.52 are an average.

EXPENSES

G&A expenses vary between $2.7m and $2.9m per year owing to the fact that approximately 70% of the G&A expenses are considered fixed over the LOM. Personnel transport, camp and laboratory costs contribute almost 70% or an average of $1.99 million to the annual G&A cost. The estimate shown in Table 21.36 is based on a nominal number of a 100 people on site.

Tetra Tech estimates that 40 grade control and 10 metallurgical assays will be processed per day at an estimated cost of RUB1,980 per assay, resulting in a total annual cost of RUB36,135,000 (USD240,900).

As a result of runway restrictions the only aircraft available locally that can be used is an Antonov AN-2, single engine biplane with a carrying capacity of 1 t or 10 passengers. The current cost of a charter flight is RUB380,000 per return journey. It is expected that this cost will increase to RUB400,000 to complete the flight all the way to Yakutsk, instead of the smaller territorial runway currently used.

Every four weeks, approximately a 110 people will be flown to and from site at a cost of RUB52,800,000/a. Separate charter flights (using the same AN2) will transport the silver powder product to the refinery and as monthly production will be approximately 6.5 Mt, this will mean between six and seven trips per month for the silver alone. Metal transport costs are discussed in Section 22.7.

Table 21.36 Average G&A Expense Breakdown

Description	Cost ($/a)
Clinic	7,576
External Assay Checks	45,455
Laboratory	547,500
Personnel Transport	800,000
Logistics	218,182
Light Vehicle Fuel Use	14,718
IT and Communications	70,000
Building Maintenance	9,091
Training	7,576
Raw Water Supply	40,129
EMP	60,606
TMF Operating Cost	39,175
Corporate	336,089
Camp	643,922
Total	2,840,018

Logistics expenses comprise warehouse rental and winter road maintenance. Provision is made for RUB225,000 per month for each of the warehouses at Batamai and in Yakutsk and RUB9,000,000 per year is set aside for winter road upkeep, resulting in an estimated annual logistics cost of RUB14,400,000.

Silver Bear will set aside RUB500,000 per year for training.

The estimated annual raw water supply cost of US$40,129 consists mostly of fuel consumption of the power supply (USD36,629) with the balance an allowance for maintenance and sustaining capital.

In the absence of an environmental management plan, Tetra Tech has made an allowance of RUB2,500,000 per year for environmental management measures and, based on advice from Silver Bear obtained from local operations, RUB1,500,000 per year for social compensation. These numbers will be reviewed once the ESIA and resettlement action plan has been completed by ERM.

The corporate expense as been provided by Silver Bear as shown in Table 21.37. Silver Bear estimate that half of this cost is attributable to the Project.

Table 21.37 Corporate Expenses

Component	Cost (RUB/a)
Head Office Overhead	39,046,019
Legal Fees	included in Head Office Overhead
Insurance	931,984
Consultants	included in Head Office Overhead
Permitting and Licensing	4,385,800
Total	**44,363,803**
Project	**22,181,902**

As the mine site is remote, the vast majority of personnel will be housed in a camp on the mine site. The number of people on the mine site will fluctuate between 51 and 160 over the mine life. Table 21.38 shows the breakdown of the costs for a nominal 100 person camp.

Table 21.38 Camp Expenses

Cost Item	Cost (RUB/a)
Subsistence	36,135,000
Power	2,863,853
PPE	500,000
Housewares and Living Supplies	2,500,000
Camp Maintenance Parts	500,000
Total	42,498,853

The cost of housewares and living supplies was provided by Prognoz who extrapolated from the historical camp costs.

LABOUR

Labour associated with camp operation and maintenance, general maintenance, security and health and safety is included in the G&A costs. A breakdown of the personnel requirements and associated estimated costs are shown in Table 21.39.

Combining the maintenance of vehicles across the mine site will reduce the total number of mechanics employed and eliminate redundancy.

Table 21.39 G&A Labour Costs

Position	Shift	Number of Positions	Job Grade	Unit Cost (RUB/a)	Unit Cost ($/a)	Total Cost ($/a)	Number of Payroll Personnel	Number of Personnel on Site
Boiler maker	Day Shift	2	C1	3,839,452	58,174	116,347	4	2
Camp Manager	Manager	1	D1	3,279,532	49,690	49,690	1	1
Cleaning Staff	Day Shift	2	A1	1,919,726	29,087	58,174	4	2
Diesel Mechanic	Day Shift	2	C1	3,839,452	58,174	116,347	4	2
General Manager	Manager	1	E2	5,247,251	79,504	79,504	1	1
Electrician	Day Shift	2	C1	3,839,452	58,174	116,347	4	2
Services Engineer	Manager	1	D3	4,099,415	62,112	62,112	1	1
Fitter	Day Shift	2	C1	3,839,452	58,174	116,347	4	2
HSE Officer	Day Shift	1	C1	3,839,452	58,174	58,174	2	1
Cooke	Day Shift	1	B2	2,687,617	40,721	40,721	2	1
Cook's Assistant	Shift	2	A2	4,223,398	63,991	127,982	8	4
Lab Manager	Day Shift	1	C3	5,759,178	87,260	87,260	2	1
Lab Technician	Day Shift	3	C1	3,839,452	58,174	174,521	6	3
Laundry Staff	Day Shift	2	A2	2,111,699	31,995	63,991	4	2
Machine Operator[1]	Day Shift	2.5	B2	2,687,617	40,721	40,721	5	2.5
Doctor	Day Shift	1	D1	6,559,064	99,380	99,380	2	1
Maintenance Planner	Day Shift	1	B2	2,687,617	40,721	40,721	2	1
Security Staff	Shift	3	B2	5,375,233	81,443	244,329	12	6
Warehouse Foreman	Day Shift	1	C1	3,839,452	58,174	58,174	2	1
Total	-	**31.5**	-	-	-	1,706,241	70	36.5

Note: [1]Position shared with mining

22.0 ECONOMIC ANALYSIS

An economic evaluation was prepared for the Project based on a pre-tax financial model. The base case uses a variable silver price of $19.65/oz, $18.57/oz, $19.62/oz, $19.79/oz, $19.72/oz, and $19.92/oz during Q1 2018, Q2 2018, Q3/4 2018, 2019, 2020 and 2021, as well as the remaining project life, respectively, with a LOM weighted average silver price of $19.76/oz. The exchange rate applied in the base case is RUB66.00/USD1.00. All currency units are US dollars unless otherwise specified. For the 8-year LOM and 858,000 t Mineral Reserve, the following pre-tax financial parameters were calculated:

- 48.2% IRR
- 1.9-year payback on the $65 million initial capital
- $88 million NPV at a 5% discount rate.

Silver Bear commissioned PwC in Toronto, Canada, to prepare tax calculations for use in the post-tax economic evaluation of the Project with the inclusion of applicable Russian income and mining taxes. PwC estimated the applicable taxes in two cases. The first case does not account for the Far East Tax Incentives and the second case takes into account the Far East Tax Incentives (refer to Section 22.4 for further details). The following post-tax financial results were calculated for the base case without considering the Far East Tax Incentives:

- 28.4% IRR
- 2.4-year payback on the $65 million initial capital
- $46 million NPV at a 5% discount rate.

The following post-tax financial results were calculated for the base case when considering the Far East Tax Incentives:

- 44.3% IRR
- 2-year payback on the $65 million initial capital
- $77 million NPV at a 5% discount rate.

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

22.1 PRE-TAX FINANCIAL ANALYSIS

22.1.1 MINE/METAL PRODUCTION IN FINANCIAL MODEL

The life-of-project average material tonnages, grade, and silver production are shown in Table 22.1.

Table 22.1 Mine/Metal Production from the Mangazeisky Silver Project

Description	Value
Total Tonnes to Mill	858,049
Average Annual Tonnes to Mill	109,466
Total Waste Tonnes Mined	10,233,111
LOM (years)	7.8
Average Head Grade	
Silver (g/t)	809
Total Mill Production	
Silver ('000 oz)	17,090
Average Annual Production	
Silver ('000 oz)	2,180

22.1.2 Basis of Financial Evaluation

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop the annual recovered silver production from the relationships between tonnage processed, head grades, and recoveries.

Payable silver values were calculated using the base case silver prices. The net invoice value was calculated for each year by subtracting the applicable refining charges from the payable metal value. The at-mine revenues were then estimated by subtracting the transportation and insurance costs. Operating costs for mining, processing, site services, G&A, and others were deducted from the revenues to derive the operating cash flows.

Initial and sustaining capital costs, as well as working capital, have been incorporated over the LOM. Capital expenditures were then deducted from the operating cash flow to determine the net cash flow before taxes.

Initial capital expenditures include costs accumulated prior to the first production of silver; sustaining capital includes expenditures for mining and processing additions, replacement of equipment, and tailings embankment construction.

The pre-production construction period is assumed to be three months. The NPV of the Project was calculated at the beginning of this three-month period.

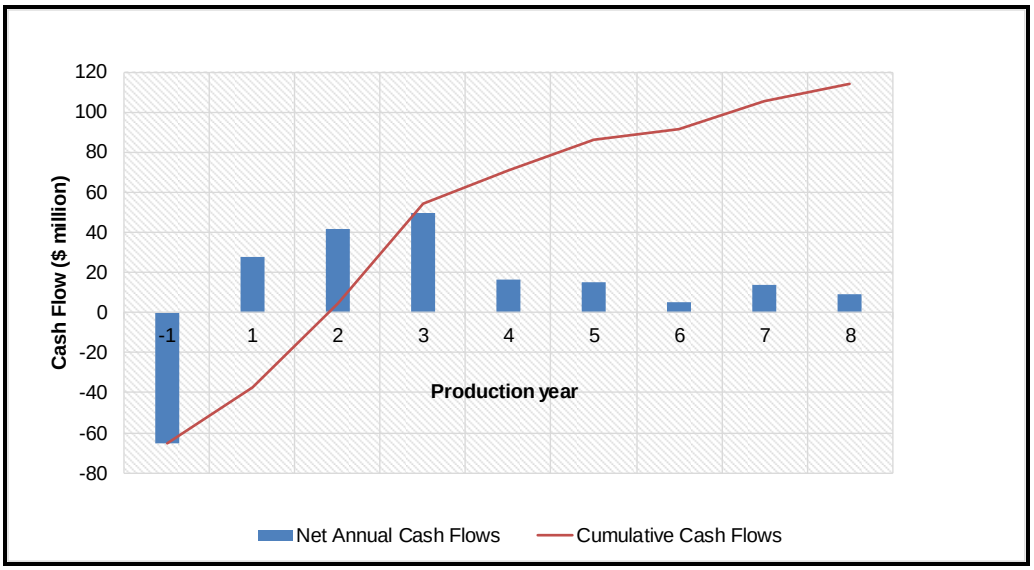
Working capital is assumed to be three months of the annual on-site operating cost and fluctuates from year to year based on the annual operating cost. The working capital would be recovered at the end of the LOM.

Initial and sustaining capital costs during LOM, were estimated at $65 million and $17 million, respectively.

Mine closure and reclamation costs were not included in the economic analysis.

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

Figure 22.1 Pre-tax Undiscounted Annual and Cumulative Net Cash Flows



22.2 SUMMARY OF FINANCIAL RESULTS

The pre-tax financial model was established on a 100% equity basis, excluding debt financing and loan interest charges. The pre-tax financial results for the base case are presented in Table 22.2.

Table 22.2 Summary of Pre-tax Financial Results

Item	Units	Value
Metal Price & Exchange Rate		
Silver Price (LOM Weighted Average)	USD/oz	19.76
Exchange Rate	RUB/USD	66.00
Net Revenue		
Recovered Silver	koz	17,256
Recovered Silver Value	$000	340,965
Losses	$000	0
Smelter Deductions	koz	17.3
Smelter Deductions Value	$000	341
Payable Silver	koz	17,239
Payable Silver Value	$000	340,624
Refining	$000	3,275
Net Invoice Value	$000	337,348
Transportation Costs	$000	6,699
Insurance	$000	240
At-Mine Revenue	$000	330,409
Royalty	$000	0
Net Revenue	$000	330,409
On-site Operating Costs		
Mining	$000	46,084
Processing	$000	51,982

table continues...

Item	Units	Value
G&A	$000	35,974
Total on-site operating costs	$000	134,040
Operating Cash Flow	$000	196,370
Capital Costs		
Initial	$000	64,974
Sustaining	$000	17,096
Reclamation	$000	0
Total Capital Costs	$000	82,070
Financial Results		
Pre-tax NCF	$000	114,299
Pre-tax NPV at 5% discount rate	$000	87,680
Pre-tax IRR	%	48.2
Pre-Tax Payback	years	1.9
Cash Cost	$/oz Ag recovered	8.37
Capital Cost	$/oz Ag recovered	4.80
Total Cost	$/oz Ag recovered	13.17

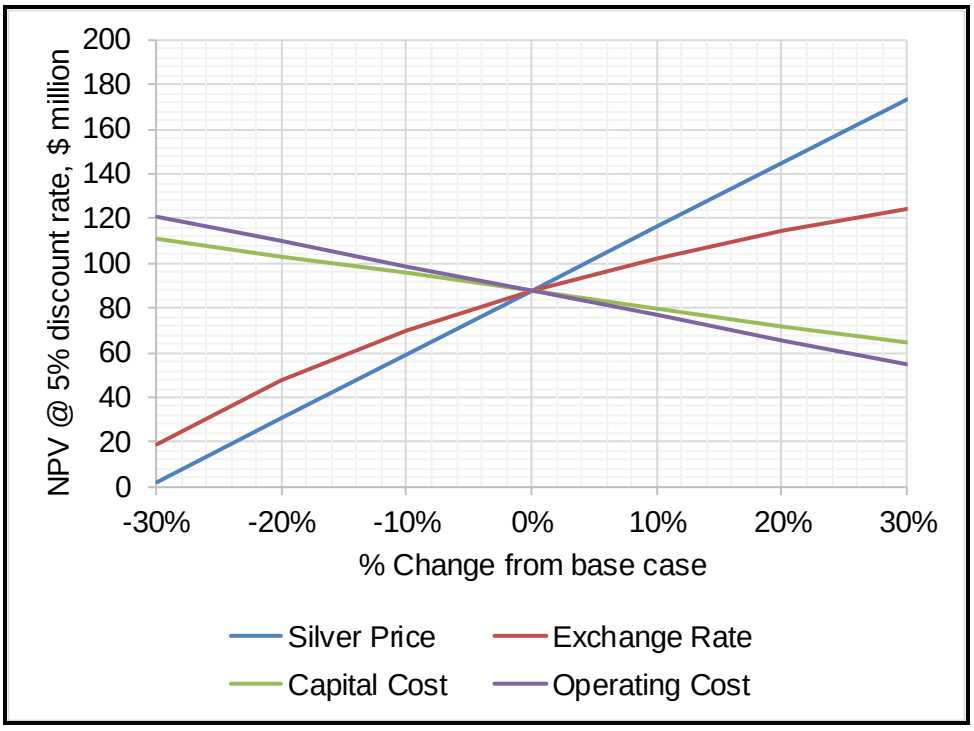
22.3 SENSITIVITY ANALYSES

Tetra Tech investigated the sensitivity of NPV, IRR, and payback period to the key project variables. Using the base case as a reference, each of the key variables was changed between -30% and +30% at a 10% interval, while maintaining the other variables constant. The following key variables were investigated:

- silver price
- USD/RUB exchange rate
- capital costs
- on-site operating costs.

The Project's pre-tax NPV, calculated at a 5% discount rate, is most sensitive to silver price followed by exchange rate, on-site operating costs and capital costs, as shown in Figure 22.2.

Figure 22.2 Pre-tax NPV Sensitivity Analysis



As shown in Figure 22.3, the Project's pre-tax IRR is most sensitive to silver price followed by exchange rate, capital costs and on-site operating costs.

Figure 22.3 Pre-tax IRR Sensitivity Analysis

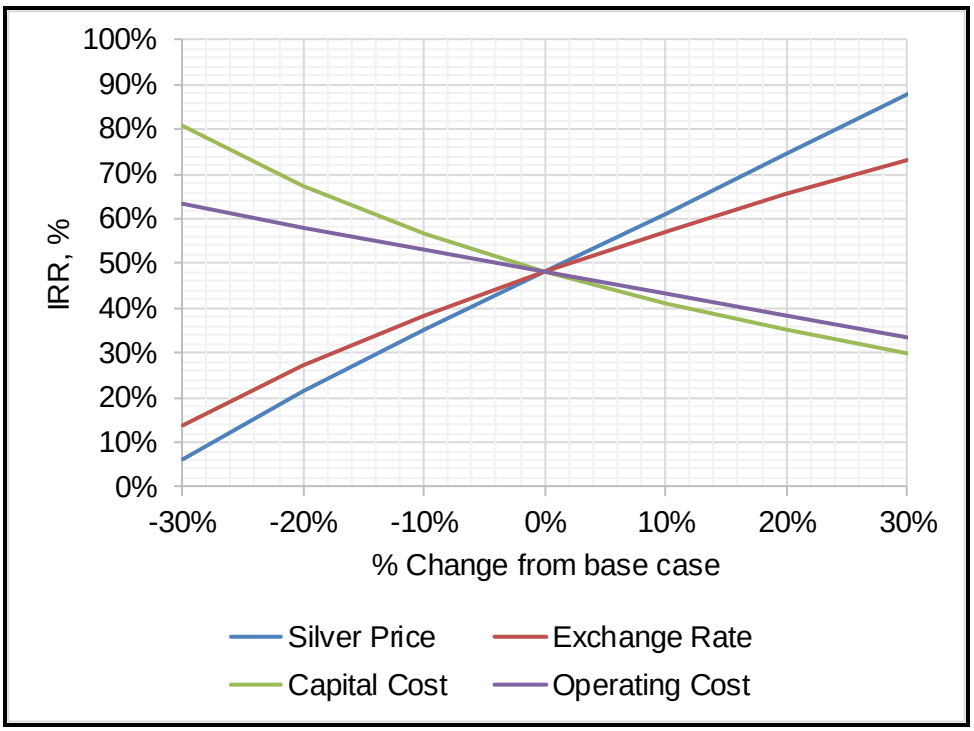
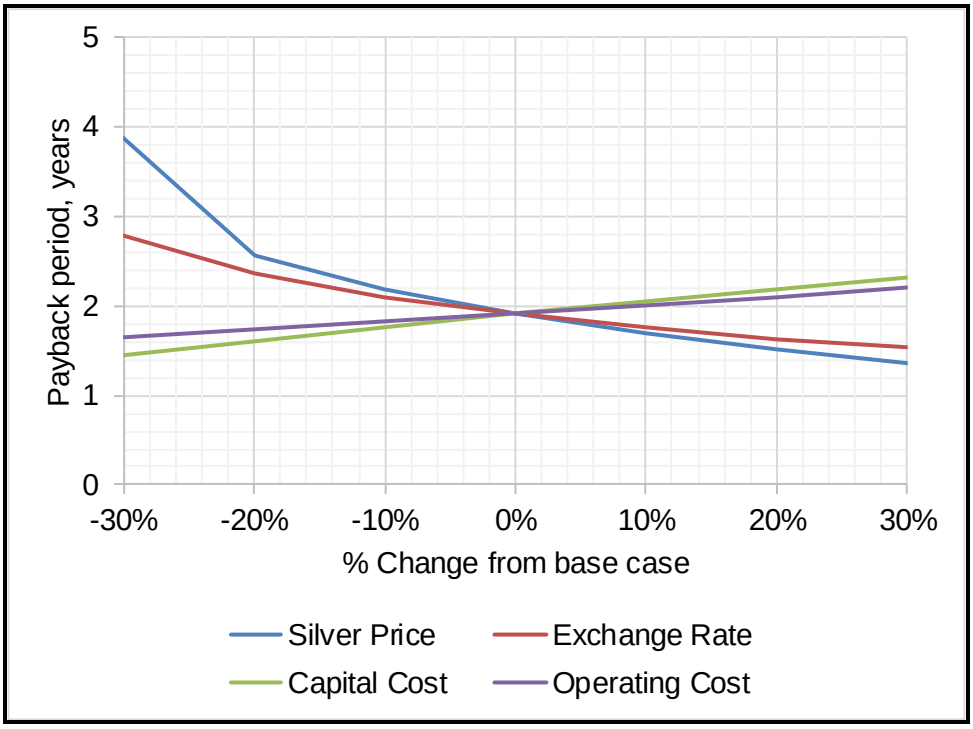


Figure 22.4 shows that the payback period is most sensitive to silver price and exchange rate followed by capital costs and on-site operating costs.

Figure 22.4 Pre-tax Payback Period Sensitivity Analysis



22.4 POST-TAX FINANCIAL ANALYSIS

Silver Bear engaged PwC in Toronto, Canada to estimate the applicable Russian income and mining taxes for inclusion in the post-tax economic evaluation of the Project. PwC has estimated the applicable taxes in two cases. The first case does not account for the Far East Tax Incentives and the second case takes into account the Far East Tax Incentives.

22.4.1 TAX ASSUMPTIONS

PwC prepared the tax estimate for inclusion in the financial model based on the following facts and assumptions, as applicable on 03 October 2016, being the date on which the final draft of the model was prepared for inclusion in the financial analysis.

PROFIT TAX

- For the purposes of the profit tax calculation, net revenue (already net of off-site operating costs) less on-site operating costs, which are included in the financial model, were used as the Silver Bear's taxable income before various deductions.
- On-site and off-site operating costs were treated as tax deductible in the period in which they were incurred.

- Capital expenditures may be deducted for profit tax purposed through the depreciation of fixed assets based on their useful lives, which is determined based on the regulations issued by the Russian government. PwC has assumed that the fixed assets will have useful lives that span over the same period as the Project's life, which is eight years as given in the financial model. Actual useful lives of Silver Bear's fixed assets, however, may vary from PwC's assumptions.
- Certain capital expenditures may be depreciated for profit tax purposes in the period the fixed assets are put into operation (called the "depreciation premium"). Depreciation premium may vary from 10 to 30% depending on the nature of the particular fixed asset and its useful life. For the purposes of computing depreciation premium, PwC has assumed that Silver Bear's fixed assets generally have useful lives of 3 to 20 years.
- Sustaining capital costs provided in the financial model were treated as part of the capital expenses that are subject to depreciation for the profit tax purposes, with the assumption that these capital costs are generally related to the renovation of Silver Bear's fixed assets over their useful lives.
- PwC assumed that the working capital included as part of capital expense in the financial mode would not be tax deductible for profit tax purposes. Also, royalties or other concessions that may not have been granted in respect of the Project have been ignored.
- Tax losses accumulated during a particular year may be carried forward and utilized against the taxable profits of Silver Bear. According to the current Russian tax legislation, tax losses can be carried forward for up to 10 years.
- Working capital return in year eight of the Project was assumed to have no Russian tax implications, and hence was ignored.
- PwC assumed the best-case scenario for the application of the Far East Tax Incentives in the tax calculation. The law allows investors to use a reduced rate for profit tax purposes—zero rate for federal tax and a reduced rate for regional tax based on the following time frame:
  - 0 to 10% over the first five years from the year when the initial income from an investment project is received
  - 10 to 18% over the subsequent five years.

The specific regional rates are determined by the laws of the relevant Russian regions. There is a risk that the Yakutia Republic, where the Project is located, may not apply the lowest regional tax rate. PwC applied a reduced regional tax rate to the first year in which Silver Bear generates a taxable profit base for purposes of the tax calculation, but can provide no assurance that this rate will be applicable when the Project enters commercial production.

VALUE ADDED TAX)

- Under the Russian tax legislation, sales of precious metals mined by a taxpayer are subject to 0% output Value Added Tax (VAT) rate.
- Russian input VAT should be paid to the suppliers as part of the price of the goods or services. This amount can then be recovered from the Russian

government. Accordingly, PwC did not include input VAT estimation in the tax calculation as it does not represent at cost to Silver Bear. Cash outflow related to payment and further recovery of input VAT does not usually last for more than six months, but such recovery may not apply to the future. A failure to recover VAT credits on a timely basis may have a material impact on the cash flow of the Project.

Mineral Resources Extraction Tax

- The tax rate of the Mineral Resources Extraction Tax (MRET) established for precious metals (other than gold) is 6.5%. This tax rate applies to the sales value of precious metals mined by the taxpayer during the tax period.
- A zero tax rate would apply to the losses of minerals that are incurred during the mining process within normal limits (the "normative losses"). The limits of losses are established annually by the public authorities to a particular field. Hence, these limits may change by the time the Project enters commercial production.
- For the purposes of the tax calculation in the financial model, PwC assumed the following scenario for the MRT regional tax incentive:
  - 0% of the standard rate is applicable for the first two years of the Project
  - 20% of the standard rate for years 3 and 4 of the Project
  - 40% of the standard rate for years 5 and 6 of the Project
  - 60% of the standard rate for the remaining life of the Project up to year 8.

  PwC cannot provide assurance that these are the rates that will apply to the Project,

Property Tax

- Russian property tax is applicable to the net book value of immovable property at a tax rate of 2.2%.
- All capital expenditures that were considered to be fixed assets and depreciated for profit tax purposes were assumed to be subject to property tax.

Taxation of Employees

- PwC assumed that both personal income tax and social insurance contributions relating to the employees have already been included in the Model as operating expenses. Therefore, PwC has not included these separately in the tax calculation.

Application of Far East Incentives

- PwC assumed that Silver Bear continues to be eligible for the Far East Incentives based on the letter from the Ministry of Economics of Yakutia Republic that was previously provided to PwC on file when performed the previous work in 2014 on the PEA for the same Project. PwC has also assumed that Silver Bear has or will make the corresponding application or

submission of supporting documentation, as required, for this program. PwC was not provided with further information or subsequent determination regarding these incentives for purposes of the profit tax calculation. The receipt of any incentives will depend on future government approval in Russia.

22.4.2 TAXES AND POST-TAX RESULTS

At the base case silver price and exchange rate used for this study, the total estimated taxes payable on the Mangazeisky Project profits over the 8-year LOM are $48.66 million, without the Far East Tax Incentives, and $12.75 million with the Far East Tax Incentives. The components of the various taxes that will be payable and the post-tax results for the base case are shown in Table 22.3.

Table 22.3 Taxes and Post-tax Results

Tax Component	Option 1: Without Far East Tax Incentives	Option 2: With Far East Tax Incentives
LOM Weighted Average Silver Price ($/oz)	19.76	19.76
Exchange rate (USD/RUB)	66.00	66.00
Property Tax ($ million)	7.79	7.79
MERT Tax ($ million)	21.46	4.79
Profits Tax ($ million)	19.41	0.17
Total Tax ($ million)	48.66	12.75
NCF ($ million)	65.64	101.55
NPV @ 5% Discount Rate ($ million)	46.38	77.45
IRR (%)	28.4	44.3
Payback Period (Year)	2.4	2.0

22.5 ROYALTIES

Tetra Tech relied on Silver Bear to provide the royalties for the economic analysis of the Project. Silver Bear confirmed that there is no any private royalties applicable to the Project.

22.6 SMELTER TERMS

Tetra Tech relied on Silver Bear to provide the smelting and refining terms applicable. The following refining terms were applied in the financial analysis:

- percent payment: 99.9% of delivered silver
- refining charge: $0.19/oz of payable silver.

22.7 TRANSPORTATION AND LOGISTICS

Tetra Tech relied on Silver Bear to provide the following transportation costs:

- cost per flight from mine site to Yakutsk: RUB400,000/trip, weekly
- cost of flight from Yakutsk to refinery: RUB200,000/t of silver
- cost of transport to off-take entity: RUB320,000/t of silver.

22.8 INSURANCE

Tetra Tech relied on Silver Bear to provide the insurance cost of $30,000/year.

23.0 ADJACENT PROPERTIES

There are no adjacent properties.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION PLAN

The project execution plan (PEP) is a project document that describes the relationship between engineering, procurement, and construction management activities. The PEP establishes a plan that is integral to the organization strategy and requirements, to manage and control the delivery of a project in a safe and cost-effective manner, and is intended to:

- provide a brief description of the proposed approach to project execution
- outline effective safety and quality methods
- anticipate and reduce or offset scheduling issues
- provide an approach and a description of the requirements for design and EPCM
- limit or avoid capital cost overruns
- anticipate issues related to construction during the design process
- provide a plan for overall project risk management
- outline how vendors and contractors will deliver quality products and services
- address delivery times for long-lead items on the critical path
- identify effective methods for forecasting, reporting, and controlling costs
- develop a detailed plan for project pre-commissioning, commissioning, and handover
- outline project safety management
- address QA/QC requirements and provide a brief project QA/QC plan
- develop the following PEP-related documents:
  - program schedule
  - Prognoz operating manual.

24.1.1 SCOPE OF WORK

Silver Bear intends to create a mine, and associated processing facility, to exploit the vertical vane of silver mineralisation in the Vertikalny deposit within their property license area. The scope of the Project extends to the development of all on-site infrastructure and facilities, as well as provision for off-site facilities such as

administration offices in Yakutsk, warehouse facilities along the winter road from Yakutsk to the mine site, and silver metal storage and handling facilities.

The Project scope of work includes the procurement of engineering, fabrication, supply, and construction services, to design and specify all equipment, civil and structural features, electrical supply, and reticulation features and control systems. The major areas include:

- on site:
  - open pit and underground mine
  - metallurgical processing facility
  - associated on-site infrastructure
  - TMF
  - waste rock dumps and ore stockpiles
  - site roads
  - power supply
  - camp
- off site:
  - Yakutsk administrative office
  - Yakutsk warehouse and storage facility
  - winter road
  - Batamai port facilities.

EXCLUSIONS

Items specifically excluded from the Project scope of work include:

- exploration
- work associated with the development of adjacent geological targets or resources
- head office activities not associated with the Project
- Silver Bear corporate activities not associated with the Project
- new airstrip.

24.1.2 OBJECTIVES AND EXPECTATIONS

The immediate objective for Silver Bear is to become a world-class producer of silver from the wholly-owned Prognoz Vertikalny Resource in the shortest period of time. This must be achieved whilst continuing strategic exploration works, and evaluating and developing additional resources for Silver Bear's continuous growth at optimised capital and operating costs.

GOALS AND EXPECTATIONS

The Project will be planned and executed to meet the following goals and expectations:

- Provision for future expansion:
  - The Project has been designed to exploit the Vertikalny deposit and work has already begun on finding opportunities to introduce ore from satellite deposits. A provision will be made in all areas of the design and construction to allow for future modifications and additions; specifically, space will be allocated adjacent to the processing facility to allow for a concentrator building.
- Financial performance:
  - The Project specific initial capital will be less than $60 million, provide a payback of less than two years, and provide an IRR in excess of 35%.
- Fast track:
  - The final study phases and project execution will happen concurrently. The execution of traditionally sequential tasks need to be overlapped in order to shorten delivery time.
- Sustainable development:
  - The Mangazeisky Silver Program seeks to exploit the Mineral Resources under license to the mutual benefit of the community and Silver Bear investors. The Project forms part of this strategy, and seeks to develop the mine in a responsible and sustainable manner.

KEY PERFORMANCE INDICATORS

Key performance indicators (KPI) outline the factors that are crucial to the success of a project. The Project KPI are shown in Table 24.1.

Table 24.1 Key Performance Indicators

KPI	Unit	Value
Mining Production Rate	kt/a	110
Plant Availability	-	>0.80
Metallurgical Recovery	-	>0.75
Total Operating Cost	$/t processed	<55.00
Environment	-	minimise harm and impact

24.1.3 PROJECT CONSTRAINTS

INTENT

The Project forms part of the Mangazeisky Silver Program, which aims to grow the operating asset base of Silver Bear, and provide a return on investment for all shareholders through the extraction of minerals under the license area held by Prognoz. A key outcome for the Project is to deliver a net positive cash flow that will enable Silver Bear to invest in the expansion of operations in the region.

STATUTORY AND COMMUNITY REQUIREMENTS

The execution of the Project, and the associated permit and license applications required, are subject to Russian federal and regional regulatory frameworks. These

frameworks include Gosudarstvennyy Standart (GOST), ROSNEDRA, and the Russian Construction Codes and Regulations (SNiP). Part of the application and approval process includes the submission of formal studies including TEO (feasibility study), Proekt (design documentation), and OVOS (environmental impact assessment).

In addition, Silver Bear, as a TSX-listed public company, subscribes to the international standards of Equator Principles, environmental practice in line with the World Bank and International Finance Corporation (IFC) standards, and the International Cyanide Code.

International Trade Embargoes

The Project is executed in Russia and will be subject to limitations on trade from and with Russia. This includes US embargoes on trade with Russia, and Russian embargoes on trade with Turkey.

Project and Site-specific Constraints

The following constraints apply to the Project:

- Initial capital costs must be less than $60 million.
- All power will be self-generated.
- All fuel will be transported by truck on the winter road and stored on site.
- Extreme winter conditions prevail for approximately nine months of the year.
- Site access for large and heavy loads is limited to the winter road.
- Personnel and small, incidental loads can be flown in by chartered plane or helicopter.
- Water is scarce, available seasonally and mostly locked up in permafrost.

The Project needs to be self-reliant as there is no logistic or infrastructure support in the region.

Technology

The specification and operation of equipment is governed by the extreme weather and the challenging transport logistics. This leads to the following technology constraints:

- Operating mechanical equipment outside requires special provision for the heating of lubricants, fuel, and hydraulic oil. It may not be possible to start and stop machinery in a conventional way.
- The remote location and the impact of adverse weather on flight schedules requires a self-reliant operation with the following features:
  - simplicity
  - conventional, low-tech equipment
  - standardization
  - redundancy

- specification of lower availability.

Where possible, the selection of technology should consider Silver Bear's aim to employ personnel as local to the site as possible.

Language and Culture

Russian and English are the languages used to execute the Project. Table 24.2 provides a summary of the language used in each phase by each stakeholder.

Table 24.2 Language Utilisation

Description	Silver Bear	Prognoz	EMC	Tetra Tech	ERM	Vendor
Internal	English	Russian	Russian	English	English/Russian	Various
Reporting Execution	-	-	Russian	English	English	English/Russian
Operation	English	Russian	-	English	-	Russian

It is acknowledged that there is a difference in culture and professional practice between the stakeholders, and this needs to be taken into account at every phase of the Project. Clear communication of intent and required outcome, without assuming that the other party automatically knows what is expected of them, is essential.

24.1.4 Organization and Responsibilities

Overview

Silver Bear intends to use Russian nationals, specifically people from the Sakha province, to build a team to construct and operate the Project. Specialist skills will be imported from further away where required, and limited to temporary appointments where possible.

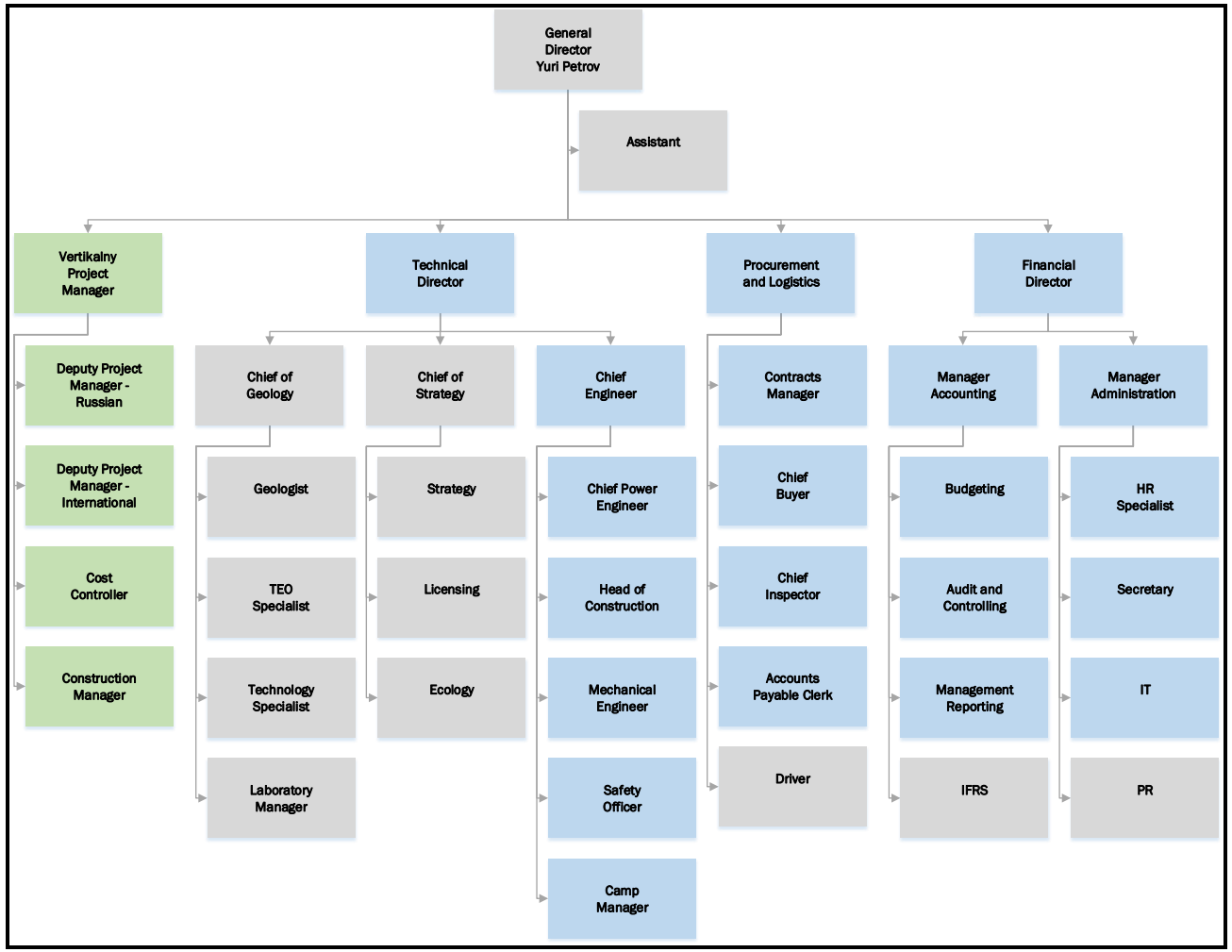
The Project will be managed from the Yakutsk administration office; headquarters of Prognoz.

Silver Bear has grown the Project team gradually, starting with a core team during the exploration phase, moving through feasibility with the addition of project management and senior engineering functions, and into execution with further engineering, logistics, procurement, contract management, and construction management roles. Silver Bear will complete the recruitment of the operational staff during Q3 2017 to allow for a period of induction and training. Hot commissioning will be completed by the operational staff under the guidance of the on-site consulting engineering team.

Project Organization Structure

The Silver Bear team will consist of personnel specifically employed to execute the Project, and head office personnel located in Yakutsk allocated to support the Project. Figure 24.1 shows the planned organizational structure for Prognoz. Blue indicates permanent staff allocated to the Project, and the green denotes new personnel specifically employed for the Project.

Figure 24.1 Project Organizational Structure



ROLES AND RESPONSIBILITIES

The Project is nearing the end of the construction phase. Contractors on site are managed by Silver Bear's Project Manager. Engineering companies such as Tetra Tech are employed on an ad hoc basis to resolve issues arising or offer quality control services of installations.

24.1.5 PROGRAMME

Silver Bear reports that the Project is 90% constructed with planned cold commissioning in November 2017 and hot commissioning in December 2017. Pre-production mining will continue until November 2017 to produce a stockpile of approximately 200,000 m^3 of ore. This quantity is limited by the amount of fuel available during 2017.

24.1.6 PROJECT COMPLETION

MECHANICAL COMPLETION

Mechanical completion is a term used to designate that point in time when the contractor is considered to have completed his work such that the facility can be operated in a safe manner. It may not be completely finished at that time, and it could be just systems within the overall project (a building or fresh water system, etc.). Mechanical completion is often descriptive of substantial completion at which time a complete list of deficiencies remaining is developed by the contractor, the construction management team, and the Owner and used to measure the progress to final completion.

Silver Bear will act as the project manager and Owner on the Project. There will be a clear delineation of the roles within the organisation such that the construction team hands over to Silver Bear operational team. This clear differentiation between construction and operation is an essential quality control measure and instils the discipline associated with a client contract relationship. Moreover, this approach is essential to distinguish between capital and operating cost in the Silver Bear accounting system.

Each process system or ancillary facility will be checked for compliance with drawings and specifications, vendor data, and lubrication requirements. Mechanical and electrical capital equipment will be checked for proper installation, alignment, and rotation. Conveyors will be run without load to verify belt alignment. Tanks and piping will be water/air tested. Electrical equipment and circuits will be checked for proper installation. Instrumentation circuits will be checked and instruments will be zero-calibrated. When all installations have been verified, each system will be run under no-load conditions. Permanent records will be maintained for each piece of equipment.

Mechanical completion of systems and facilities is a prelude to the commissioning of the overall plant. By this time the Owner's operating personnel have punch listed the system, and the construction management team has worked with the contractors to finish things up to the state needed. Critical utility features will have been completed before mechanical completion such that water is available for hydro testing of piping

and tanks, air is available to test the pneumatics, and permanent power is available to ”bump” the motors.

COMMISSIONING

Existing operations and maintenance personnel will be used at the time of commissioning, testing, and start-up. Subsequent to the commissioning period, scheduled training for supervisors and operations personnel will be scheduled for a two-week period during which a handover to operations will occur. An exact schedule will be developed in collaboration with the selected contractor during the future work plan phase preceding project execution. As well, a detailed and itemised list of KPIs will be developed to ensure satisfactory performance of the equipment prior to hand off to operations.

Commissioning will be managed by Silver Bear’s Owner’s team based on the timing as follows:

- Cold commissioning or subsystem level commissioning will be completed by the Silver Bear’s project team under the control of the Silver Bear project manager to validate the correct and safe operation of subsystems. No material is used during the commissioning, but for liquid transfer and holding systems, water is often used in lieu of product. Cold commissioning of a subsystem can only commence if all components in the system have been issued with a mechanical completion certificate.
- Hot commissioning follows the completion of all subsystem commissioning and is completed by the Silver Bear permanent operational team of the mine with assistance from the construction team under the leadership of the relevant operational manager. Hot commissioning can only commence if all subsystems have been issued with a cold commissioning certificate.

The sequence of system commissioning is vital to shifting the construction schedule from general area completion to more specific system completion to suit the commissioning and start-up of the entire facility. The commissioning sequence will be developed at the start of the Project. System identification and prioritisation must be expedited to allow for the construction schedule adjustments and the completion of the work, in order to satisfy the established commissioning sequence. During the latter part of construction, the project team will execute the plan based on work priority. Packages will be assembled to include all sign off and test documentation, drawings and vendor information for each system that will be commissioned.

The various systems will be transferred to the Owner’s operations team once they are completed and determined by the construction management team to be suitable for safe operation. The Owner’s team will consist of plant operators and maintenance staff who will enlist the help of vendors, contractors and construction management personnel as needed to trial the systems until they are finally accepted by the Owner’s operations management. The transfer of systems will be formally documented and include all mechanical/electrical testing documents and vendor’s information.

HANDOVER

Handover from the Silver Bear project team to the Silver Bear operational team will happen formally at the end of hot commissioning following a performance test. Although an internal handover, the formalization of this step marks the handover of responsibilities and a change in reporting and accounting practice as well as the conversion from a capital to an operational budget.

CLOSE OUT

During close out of the Project, the Silver Bear project manager will audit and update all project information including manuals and drawings. The project team will be disbanded and outstanding obligations closed out.

24.2 PROJECT RISKS AND OPPORTUNITIES

24.2.1 RISKS

A risk matrix was prepared during the feasibility study phase and will be used as the basis to track the risks on the Project and tracked and reported on a monthly basis. The risk matrix will be proactively monitored and mitigation measures implemented to minimise risks during the execution of the Project. The most significant risks (identified as critical risks, Table 24.3) related to project execution as identified in the risk matrix are show below with their mitigation strategies.

The main environmental risks associated with the Project are separately assessed and summarised in Section 20.5. In order to minimise the level of risk related to the environmental aspects, health and safety and security measures have been identified. In addition, an ERP will be implemented at the earliest stages of the operation phase to manage those risks that cannot be completely eliminated by the protection measures already in place and to plan emergency interventions if an accident were to occur.

24.2.2 OPPORTUNITIES

Silver Bear will look to expand their operation by incorporating satellite deposits in the mine license area. The sunk capital cost associated with the infrastructure development for the Project may lower the cut-off grade for subsequent deposits and subsequently the hurdle to entry. The most notable current target is the Mangazeisky North silver deposit.

Table 24.3 Critical Risks

Description	Impact	Mitigation
Mishandling of sodium cyanide during processing.	Contamination of environment (land and rivers) subject to exposure during a spill.	Only let trained personnel handle sodium cyanide. Train medical personnel on site in sodium cyanide emergency care. Medical evacuation plan.
Insufficient water supply due to inadequate season data collection.	Stop operation until water supply is secured.	Conduct full hydrogeological program in the spring. Find alternate winter source of water.
Data from metallurgical test work done on the samples provided by Silver Bear in 2014 does not reflect mine life ore behaviour.	Predicted recoveries may not be accurate. Lower revenue possible.	Complete metallurgical test work representative of life of mine (see Section 13.0)
There is a possible levy on the deposition of tailings in the TMF by the local authorities that relates to the classification of that material. Discussions with the local authorities are ongoing, however there is a possibility that a charge of $7.88/t of tailings may be payable.	An increase in the operating cost of the mine.	According to Silver Bear, the payment of this levy is negotiable, subject to ongoing test work. Silver Bear has committed to ongoing test work and ongoing discussions with the local authorities.
Constructing the mine prior to final approvals and permits to construct means that the terms of the approvals may differ from the current design.	Parts of the design and/or constructed facilities may have to be changed leading to extra cost and possible delays.	Employ experienced Russian design agency to design in strict accordance with regulations. Keep the authorities informed and appoint a dedicated team in Yakutsk to manage the design documentation and permit approval process.
The approvals and permits for operation may be later than expected.	Without permits to operate the hot commissioning of the plant will be delayed accordingly.	Employ experienced Russian design agency to design in strict accordance with regulations. Keep the authorities informed and appoint a dedicated team in Yakutsk to manage the design documentation and permit approval process.
Inability to install the synthetic liner for TMF in low temperatures.	Delay of TMF commissioning.	Proper work scheduling in warmer weather periods.

24.3 HYDROGEOLOGY AND WATER SUPPLY

24.3.1 SITE CHARACTERISTICS

CLIMATE

The Project area climate is sharply continental, characterised by large fluctuations in temperature both within the year and during the day. Over the year, the temperature difference between the coldest and the warmest months can reach 58 to 62°C. In winter, the territory is under the predominant influence of the Siberian anticyclone that contributes to overall stable cold weather. In summer, the territory is mainly in a

low pressure area. Summer is hot but short, and nights are usually cool; overnight freezing is possible during the summer months.

There is no weather station at the Project site. Descriptions of some of the weather parameters in this section are based on the weather data from the two nearest meteorological stations—Sebyan-Kuel and Syuryen-Kyuyel—which are located south of the Project site, approximately 45 km and 80 km away, respectively. Sebyan-Kuel station has been operating since 2010, and Syuryen-Kyuyel station operated from 1941 to 1988.

Air Temperature

The average annual temperature according to Sebyan-Kuel and Syuryen-Kyuyel weather stations is -12.5 °C and -11.9 °C, respectively. The coldest month is January, with average monthly air temperatures of -38.1 °C at Sebyan-Kuel and -34.6 °C at Syuryen-Kyuyel. The average monthly temperature in July, the warmest month, is +13.7 °C at Sebyan-Kuel and +12.7 °C at Syuryen-Kyuyel.

Precipitation

The average annual precipitation at the Sebyan-Kuel and Syuryen-Kyuyel weather stations is 203 mm and 337 mm, respectively. More than 60% of the precipitation occurs during the warm season (May to September). The minimum rainfall over the year is registered in March. The highest average amount of precipitation occurs in June and July (59 mm and 76 mm, respectively) and the lowest in the winter months. Table 24.4 presents data of average monthly and annual precipitation.

Table 24.4 Average Monthly and Annual Precipitation (mm)

Station	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Sebyan-Kuel	3	3	3	8	12	59	39	31	25	13	3	4	203
Syuryen-Kyuyel	7	6	5	11	30	67	76	68	38	13	9	7	337

The first appearance of snow cover occurs typically in September. A stable snow cover forms on average between 28 September and 11 October.

Topography and Hydrography

Figure 24.2 illustrates a 3D rendering of the topography and hydrographical network in the Project area. The Property lies in a mountainous region, with elevations ranging from 800 to 1,400 masl. The main ridges have steep slopes (25 to 30°) and rounded crests that are 200 to 500 m above the valley floors.

The main water course adjacent to the Verticalny deposit is the Sirelendge River, which flows in a northwest to southeast direction into the Arkachan River, approximately 10 km downstream of the Project site. Two local water courses within the Verticalny deposit area flow into Sirelendge River: the Porfirovy and Borisovsky streams. The entire hydrographical network connects to the basin of the Yana River, which flows north into the Laptev Sea. Surface water flow in the area ceases during the winter season due to the freezing temperature, which causes the formation of ice along the river and stream beds.

24.3.2 SCOPE OF SITE INVESTIGATIONS

HYDROLOGICAL FIELD WORK

Surface water monitoring was carried out in three locations:

- the Porfirovy Stream, 5.7 km upstream of its junction with the Sirelendge River
- the Borisovsky Stream, 1.8 km upstream of its junction with the Sirelendge River
- the Sirelendge River, 8 km from the junction with the Arkachan River.

HYDROGEOLOGICAL FIELD INVESTIGATIONS

Thermistors were installed in shallow boreholes that were drilled as part of geotechnical investigations. These boreholes, which have an average depth of 30 m, were used to monitor the temperature of the shallow geological formations.

Hydrogeological Boreholes

Deep hydrogeological boreholes were drilled in the Project area for two purposes:

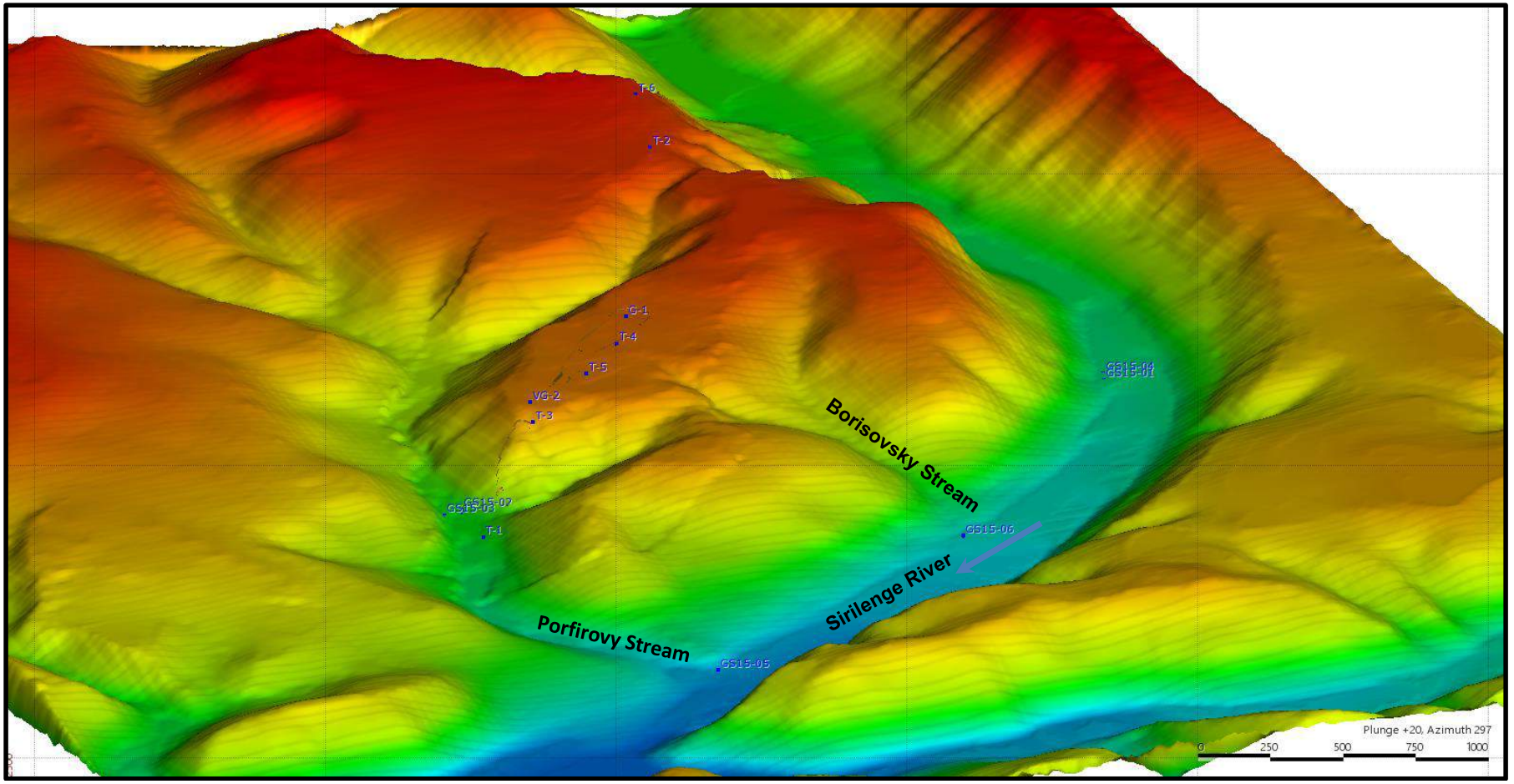
- Due to freezing of surface waters over approximately eight months of the year, an alternative reliable source of water is needed. The sub-permafrost aquifer was identified as the main target for water supply investigation.
- A deep borehole was drilled into the ore deposit to investigate if parts of the future underground mine will intersect the sub-permafrost aquifer.

Water Supply Investigations

Subsequent to the surface geophysical survey, which targeted three areas, locations for water supply investigations were defined for intrusive investigation. The following boreholes were successfully drilled and tested for the water supply investigation: GS15-01, GS15-04, GS15-05, and GS15-06 (Figure 24.2 and Figure 24.8). The four boreholes are located along the Sirelendge River valley, and have a vertical depth of approximately 250 m below ground surface.

Two boreholes, GS15-01 and GS15-04, were drilled and pumping tests completed in June 2015 (Figure 24.2). The third borehole (GS15-05) was drilled and tested in November, at the junction of the Porfirovy Stream and Sirelendge River. The observed average free flow (without pumping) from GS15-05 was approximately 1.65 L/s. Since the artesian outflow from this borehole exceeded the capacity of the pump that was available on site (1.39 L/s; 5 m^3/h), a shut-in test was carried out to monitor water pressure recovery instead of conducting a pumping test.

Figure 24.2 3D View of the Project Area Topography and Hydrography, Showing Borehole Locations



Full recovery of water pressure seems to have been achieved during the 77 minute test. After the test, the flow rate was manually measured regularly until the hole froze on 13 November 2015. The artesian flow measurements for borehole GS15-05 are shown in Table 24.5. Before freezing, the borehole was equipped with a string of thermal sensors for geothermal monitoring. The thermal sensors were installed at the following depths: 1, 10, 25, 50, 100, 150, 200 and 230 m.

Table 24.5 Measured Artesian Outflow Rates in Borehole GS15-05

Date	Time	Flow (L/s)	Daily Average (L/s)
08/11/2015	15:25	1.64	1.64
10/11/2015	08:42	1.53	1.67
10/11/2015	16:05	1.80	
11/11/2015	09:55	1.66	1.66
11/11/2015	14:00	1.64	
11/11/2015	18:00	1.67	
12/11/2015	10:00	1.57	1.51
12/11/2015	14:00	1.55	
12/11/2015	17:00	1.40	
13/11/2015	FROZEN		

Borehole GS15-06 was drilled at the junction with Borisovsky Stream, and a pumping test was completed in November 2015. Subsequent to testing, water samples were taken from all four boreholes and sent to laboratory in Yakutsk for analysis.

Mine Hydrogeological Investigations

Two boreholes (VG-2 and G-1) were drilled and equipped with a string of thermistors in the mine area. The thermistors were allowed enough time to equilibrate before reliable temperature measurements were taken a number of times. Neither of these two holes traversed the full thickness of the permafrost; nevertheless, it was possible to quantify the geothermal gradient using the temperature data collected from the 300 m-deep borehole VG-2, and estimate the permafrost thickness in that location.

24.3.3 Field Data Analysis and Modelling

Surface Water Hydrology

Hydrological Catchments and River Flow Rate

Locations of the stream/river flow monitoring points are shown in Figure 24.3.

Figure 24.3 Local Hydrological Catchments and Location of Flow Monitoring Points

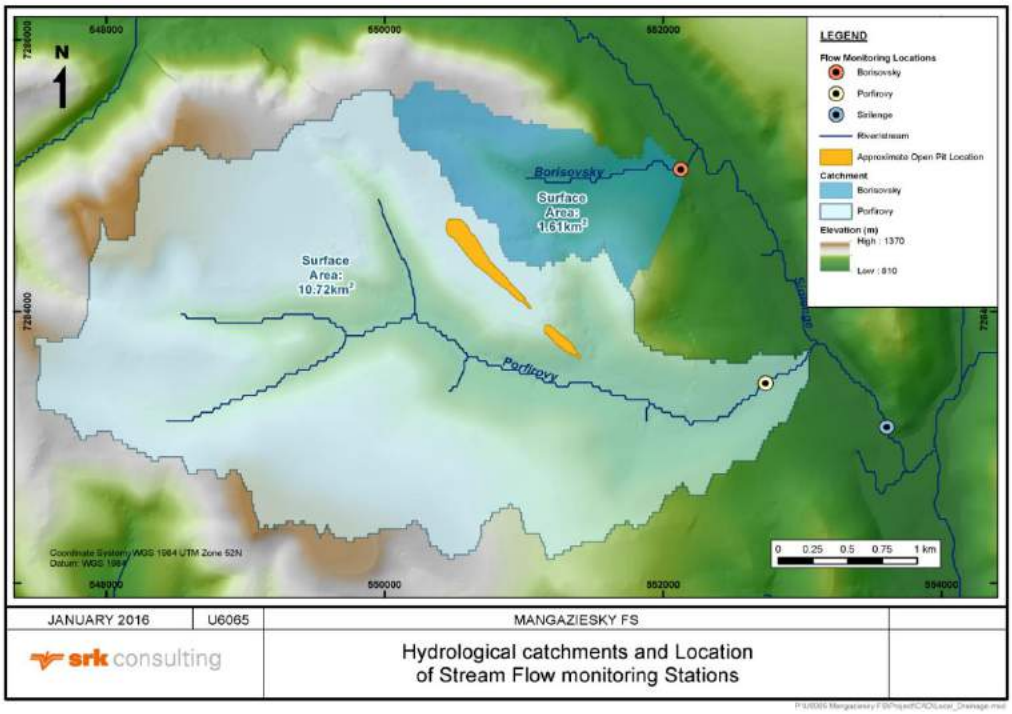
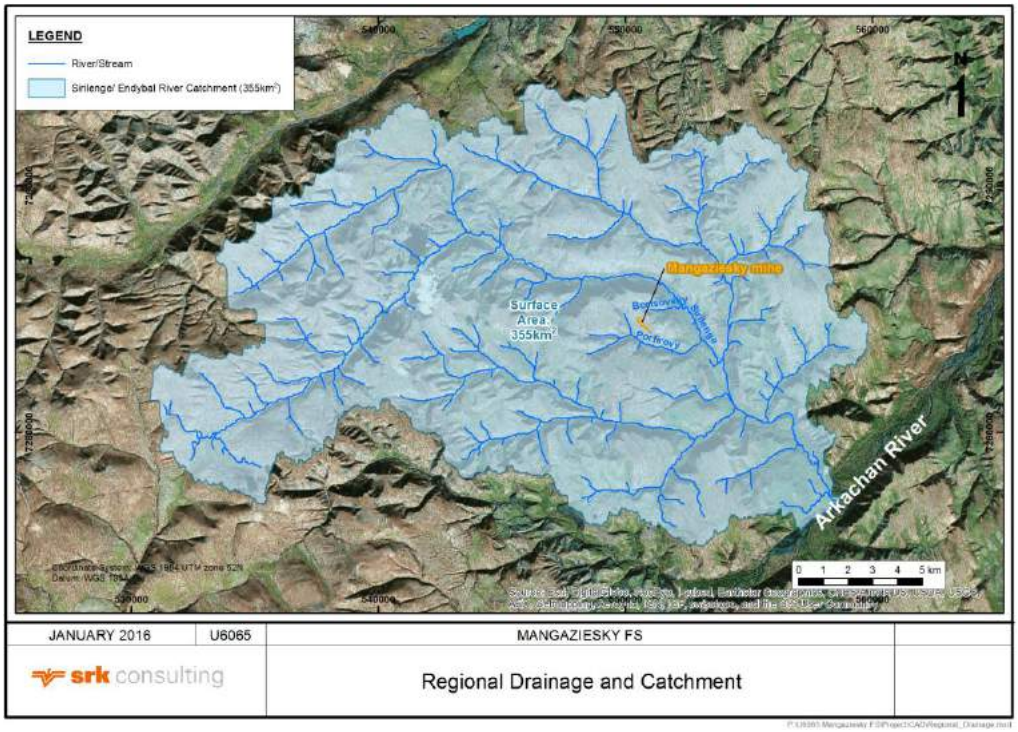


Figure 24.4 Regional Drainage Catchment



All four stages of the open pit fall within one sub-catchment, and will drain into the Porfirovy Stream, which is a tributary of the Sirelendge River and has a drainage surface area of approximately 10.7 km^2 (Figure 24.3). Draining to the east of the ridge, upon which the pit is located, is the Borisovsky Stream, which is a small

tributary of the Sirelendge River and drains a smaller area of just about 1.6 km^2. The Sirelendge River flows 10 km downstream into the Arkachan River (Figure 24.4). The total surface area of the river catchment at the Arkachan River confluence is 355 km^2. The values of the average annual discharge within the three main water courses in the Project area are estimated and summarised in Table 24.6.

Table 24.6 Average Annual Discharges or Surveyed River Sections

Monitoring Point	Watercourse Name	Average Perennial Water Discharge (m^3/s)
1	Porfirovy Stream 5.7 km from the mouth	0.050
2	Borisovsky Stream 1.8 km from the mouth	0.013
3	Sirelendge River, 8 km upstream of the junction with Arkachan River	0.79

Estimation of Potential Storm Magnitude

The storm magnitudes for the Project were estimated by SRK using the precipitation data from the Syuryen-Kyuyel station, with a 10% increase applied to these data because of the lengthy distance from the site. For a range of return periods for the station, storm depths were calculated using the annual maxima method which utilises the Gumbel extreme value distribution and a variety of plotting position formulae, including Weibull, Gringorton, Cunnane and Hazen. The results obtained using the Cunnane plotting position formula were adopted for storm water management design for the Project because it is more reliable at higher return periods. The estimated storm depths for the Project area are shown in Table 24.7.

Table 24.7 Estimated Storm Depths for Mangaziesky Site

Return Period (year)	Estimated 24 hour Extreme Rainfall Depths (mm)			
	Weibull	Cunnane	Gringorton	Hazen
2	28.46	28.40	28.39	28.38
5	40.66	39.87	39.77	39.63
10	48.74	47.46	47.31	47.07
20	56.48	54.75	54.54	54.21
50	66.51	64.17	63.90	63.46
100	74.02	71.24	70.91	70.39
200	81.51	78.28	77.90	77.29
500	91.38	87.57	87.11	86.39
1000	98.85	94.59	94.08	93.28

Estimation of Surface Runoff into the Open Pits

Because the proposed pit will be located fully in the permafrost, surface water will be the main source of inflows into the pit. SRK calculated estimates of surface water runoff for average rainfall conditions and exceptional storm events.

Peak Runoff Estimation

Storm water runoff into the pits was estimated as: storm depths (Table 24.7) multiplied by the pit area, times a runoff coefficient, which is 0.8 for all pit phases, except Phase 3. A higher runoff coefficient of 0.9 was used for Phase 3 because of pit steepening, rather than lateral extension, during this phase. The estimated daily runoff volumes are shown in Table 24.8.

Table 24.8 Estimated Peak Flow from Storm Events for Five Defined Return Periods

Daily Peak Volumes Estimates (m^3/day)					
Return Period T =	5 years	10 years	20 Years	50 Years	100 Years
Rainfall (mm/day)	39.9	47.5	54.8	64.2	71.2
Pit Stage 1 (area: 49,453 m^2)	1,577	1,878	2,166	2,539	2,818
Pit Stage 2 (area: 74,514 m^2)	2,377	2,829	3,264	3,825	4,247
Pit Stage 3 (area: 74,514 m^2)	2,674	3,183	3,672	4,303	4,778
Pit Stage 4 (area: 19,763 m^2)	630	750	866	1,015	1,126

HYDROLOGY AND GROUNDWATER ANALYSIS

Analysis of Permafrost

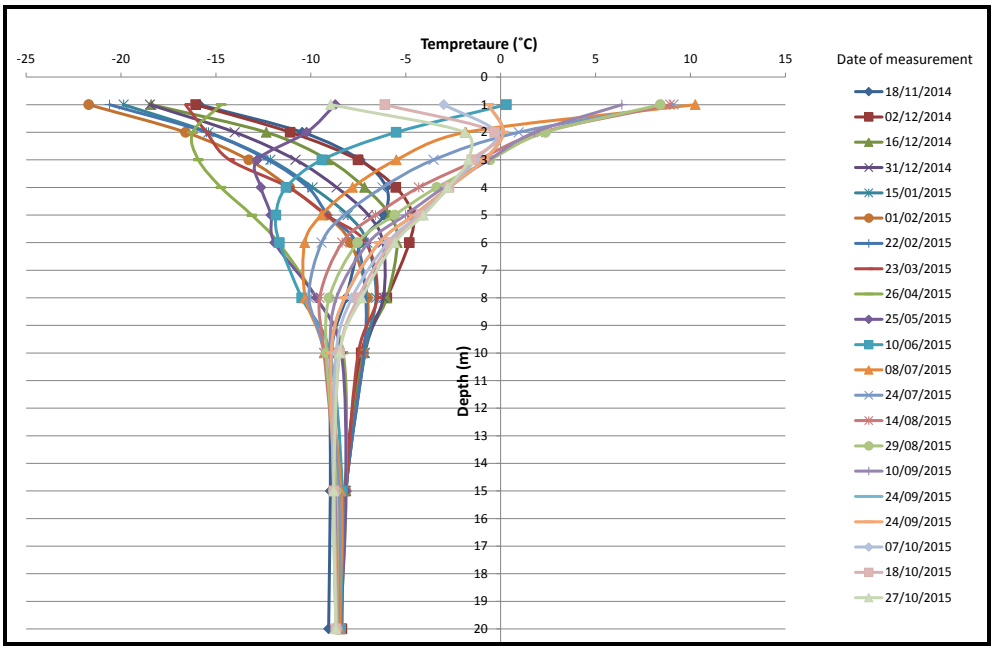
As mentioned in the previous sections, thermistors were permanently installed in two shallow boreholes (T-005 and T-006, both 30 m deep), and three deep boreholes (G-1, VG-2, and GS15-05,) in order to monitor temperature of the geological formations at various depths.

Estimation of the Active Layer Thickness

The active layer is the top horizon of the geological formations where the temperature level during the warm season reaches the thawing point (0°C). Year-around measurements of temperature profiles were only collected from two boreholes (T-005 and T-006). The temperature profiles obtained from thermistors in borehole T-006 are shown in Figure 24.5, as an example. Data from the two boreholes were used to estimate the thickness of the active layer in the Project area.

The temperature profiles from both boreholes indicate that the maximum thawing thickness of 2.75 m is reached in late August in borehole T-006, whilst the maximum thawing thickness of 1.75 m was reached in borehole T-005.

Figure 24.5 Temperature Profiles in Borehole T006



Permafrost Thickness in Deep Borehole Locations

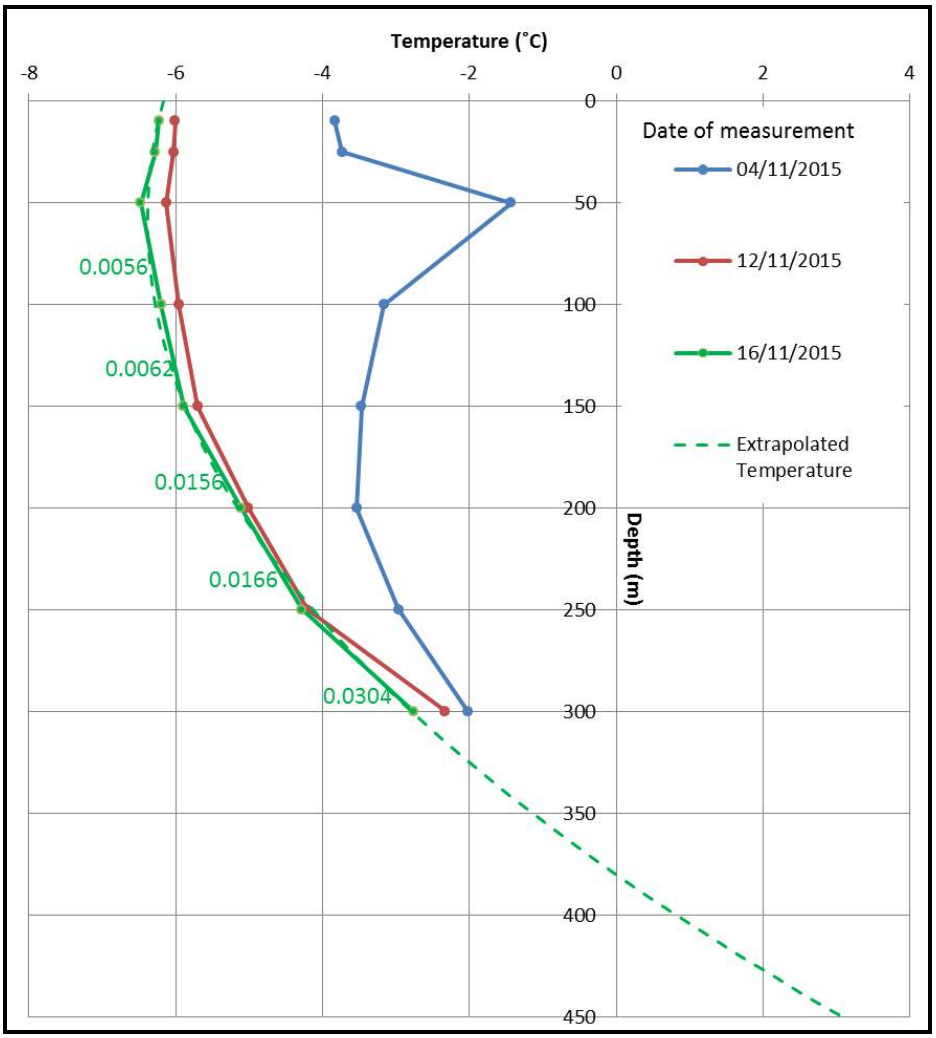
The deepest (300 m) hydrogeological borehole in the Verticalny deposit (VG-2) was intended to investigate if the underground mine will be located fully in the permafrost or if some parts of it will intersect the underlying aquifer. Thermistors were installed in borehole G-1 down to a vertical depth of 86 m. The depths of the permafrost bottom in boreholes VG-2 and G-1, which did not traverse the full thickness of the permafrost, have been estimated from the geothermal gradient obtained from the temperature measurements. An example of geothermal gradient estimation from the measured geothermal profiles and interpolation of the permafrost bottom depth is shown for borehole VG-2 in Figure 24.6.

The depth of the bottom of the permafrost layer in the four boreholes in the river valley, which reached the underlying aquifer, was estimated from drilling observations, as reported by the contractor, and geophysical survey of the boreholes. Table 24.9 shows the estimated depth of the bottom of the permafrost in all six hydrogeological boreholes. As expected the permafrost is thinner along the river valley and gets thicker toward the highland around the orebody.

Table 24.9 Borehole Coordinates and Depth of Permafrost

Borehole ID	Easting	Northing	Collar Elevation RL	Depth to Bottom of Permafrost (m)
G-1	550513.071	7284597.004	1,181.757	400
VG-2	551069.466	7283952.311	1,153.468	380
GS15-01	551087.107	7286148.452	884.061	191
GS15-04	551029.585	7286174.666	885.598	157
GS15-05	553008.593	7283704.628	847.476	189
GS15-06	552243.048	7285028.786	873.085	220

Figure 24.6 Geothermal Gradient and Interpolation of the Permafrost Bottom Depth in Borehole VG-2



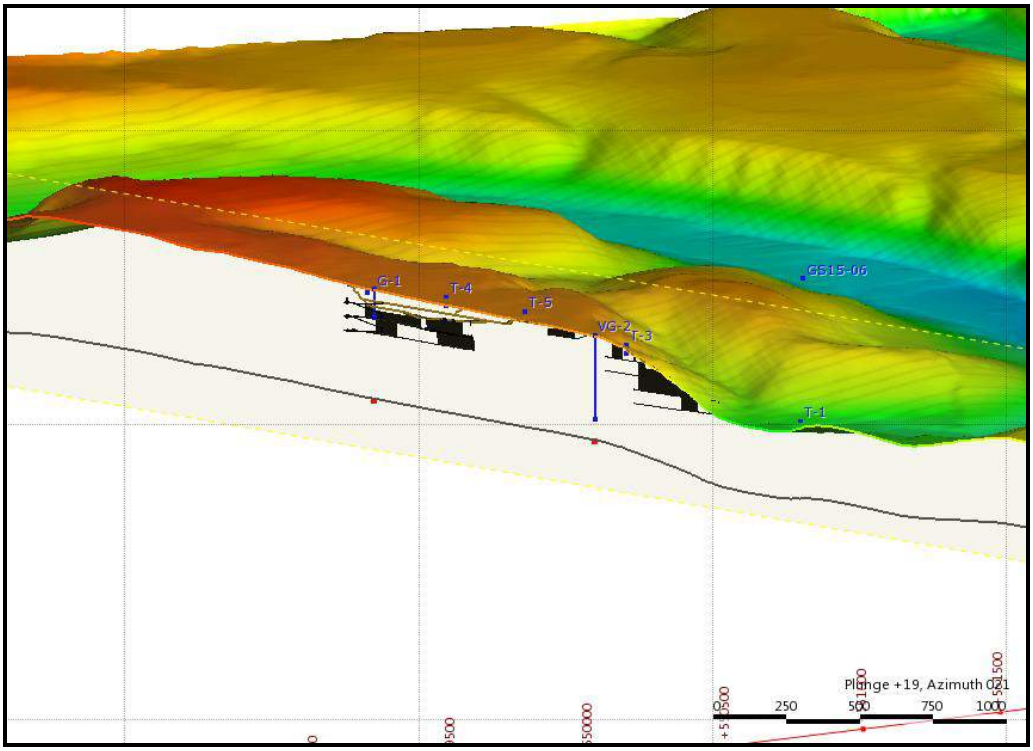
Interpolation of the Permafrost Bottom Surface

The permafrost data analysis revealed the existence of a significant correlation between the topographic elevation and the elevation of the bottom of the permafrost layer. Therefore, the spatial variation of the bottom of the permafrost (3D surface) was constructed using linear regression interpolation, based on such correlation in boreholes GS15-01, GS15-05, GS15-06, VG-2 and G-1, and topographic data. The interpolated surface of the bottom of the permafrost shows that the thickness of the permafrost ranges between 160 m in Sirelendge River valleys and 510 m in the mountain range.

The cross section in Figure 24.7 illustrates the position of the proposed underground mine workings with respect to the bottom of the permafrost. The red dots along the vertical borehole positions represent the estimated/measured bottom of the permafrost at the borehole position, whilst the black line joining such dots shows the inferred bottom surface of the permafrost (interface between the permafrost and sub-permafrost aquifer). The proposed underground mine is shown in dark black polygons and polylines. The cross section shows that the proposed underground

mine will be located fully within the permafrost, well above the sub-permafrost aquifer.

Figure 24.7 Cross-section along Boreholes G-1 and VG-2 Showing Position of the Permafrost Bottom in Respect to the Underground Mine (Black)



Conceptual Hydrogeological Model of the Sub-Permafrost Aquifer

Considering the significant thickness of the permafrost in the Verticalny deposit, and according to borehole cores, the sub-permafrost aquifer is located far beneath the weathered rock horizon, and groundwater flow seem driven by geological structures, mainly tectonic faults and fracture networks. This is further confirmed by the difference in the achieved borehole yield observed in the tested boreholes.

The artesian condition observed in all boreholes in the Sirelendge River valley confirms the confining property of the permafrost layer, which isolates the aquifer from surface waters. Recharge of the sub-permafrost is expected to occur during the warm season along post-permafrost faults. The aquifer recharge zones have not been identified during the site investigation. The artesian hydraulic head varies between 14 m in the downstream hole (GS15-05), and 1.1 m in the most upstream hole (GS15-04). The hydraulic head in the intermediate hole GS15-06 is 7 m above ground level. Such distribution of the artesian groundwater level may be a sign that a source of the aquifer recharge is located at a short distance upstream of borehole GS15-04.

Groundwater Level Map and Flow Direction

Groundwater level (hydraulic head) data are available from the four boreholes along the Sirelendge River valley, which reached the sub-permafrost aquifer. The measured and inferred water levels (hydraulic heads) in the four boreholes are shown in Table

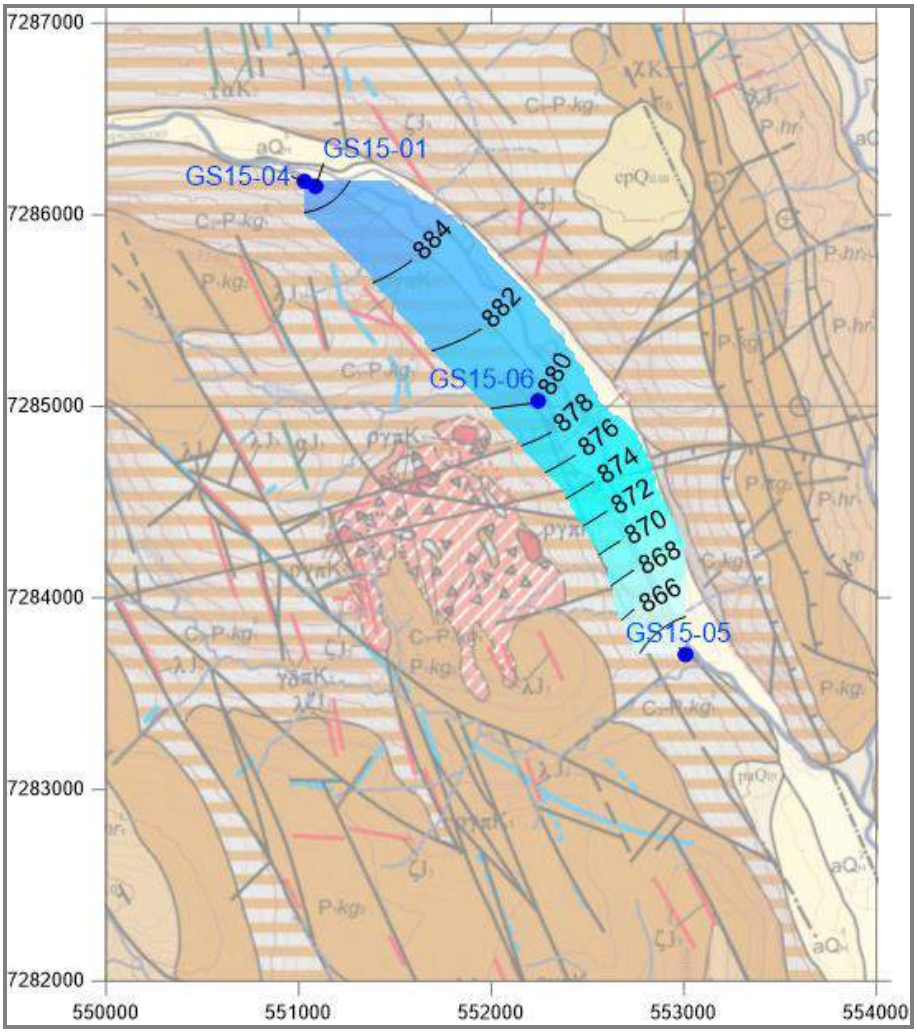
24.10. These data are used to create groundwater level contours along the Sirelendge river valley to illustrate the natural groundwater flow direction (Figure 24.8).

It is important to notice, however, that the magnitude of hydraulic gradient can also be affected by seasonal variation of water levels: measurements in boreholes GS15-01 and GS-04 were taken in June 2015 (under warm weather conditions) whilst the measurements in boreholes GS15-05 and GS15-06 were taken in November 2015 (freezing conditions).

Table 24.10 Measured and Inferred Groundwater Levels in the Boreholes

Borehole ID	X	Y	Collar Elevation (mamsl)	Measured Head (mamsll)	Head Above Ground (m)
GS15-01	551087.1	7286148	884.06	886.46	2.4
GS15-04	551029.6	7286175	885.5	886.6	1.1
GS15-05	553008.6	7283705	847.84	861.84	14
GS15-06	552243	7285029	873.08	880.08	7

Figure 24.8 Groundwater Level Spatial Distribution (Contour Levels in mamsl)



Analysis of Pumping Tests and Estimation of Aquifer Parameters

Data collected from the hydraulic tests in the four boreholes along the Sirelendge River valley were analysed using the professional software AQTESOLV. The results of the data analysis are summarised in Table 24.12.

The estimated hydraulic conductivity of the sub-permafrost aquifer at borehole GS15-05 is one order of magnitude higher than that at hole GS15-06. The conductivity at the most up-gradient boreholes GS15-01 and GS15-04 is approximately four times that of borehole GS15-06. This is likely due to the fact that borehole GS15-06 is located in an intrusive formation with the absence of fault, as opposed to borehole GS15-05, which seems to be located near a fault zone, according to the background geological map in Figure 24.8. The analysis of the data provides some estimate of storage coefficient of the aquifer; however, it is known that data from the pumping well alone can result in erroneous storage coefficient estimates and therefore the values obtained from these tests should be used carefully.

Borehole Water Quality

Water chemistry was obtained from laboratory analysis of water samples collected from four boreholes in the Project area (GS15-01, GS15-04, GS15-05 and GS15-06). The water quality data are compared with the international guidelines set out by the World Health Organisation (WHO 2011) for drinking water quality (Table 24.11). The comparison is for indicative purposes only and does not infer any judgement on the suitability of water for long-term usage, human or animal consumption, or for discharge to the environment.

Table 24.11 Borehole Groundwater Quality Data

Parameter	Unit	Russian MPC	WHO MPC (2011)	GS15-01	GS15-04	GS15-05	GS15-06
Odour	points	2		3	3	2	2
Taste	points	2		0	0	0	0
Colour	degrees	20		15	5	<5	27
Turbidity	mg/L	2.6		4.9	2.4	0.4	4.7
pH	Unit, pH	6-9	6.5 - 8.5*	7.5	7.5	7.3	7.5
Oxidability	mg/L	5		1.1	1.3	1.7-0.3	2.7
Nitrates	mg/L	50	50	<0.1	<0.1	<0.1	<0.1
Total hardness	mg-eq/L	7		14	11	4.8	4.3
Solid residue	mg/L	1,000		1,429	1,200	1,181	2,277
Chlorides	mg/L	250*	250*	714†	8.5	278†	203
Sulphates	mg/L	500	250*	12	10	<10	<10
Total Iron	mg/L	0.3	0.3*	0.8†	0.4†	1.2†	0.71†
Copper	mg/L	1		<0.002	<0.002	<0.002	<0.002
Zinc	mg/L	5		<0.005	<0.005	<0.005	<0.005
Arsenic	mg/L	0.01	0.01	<0.01	<0.01	<0.01	<0.01
Lead	mg/L	0.01	0.01	<0.006	<0.006	<0.006	<0.006
Fluorine	mg/L	1.5		<0.04	<0.04	<0.04	<0.04
Cadmium	mg/L	0.003	0.003	<0.0005	<0.0005	<0.0005	<0.0005
Beryllium	mg/L	0.0002		<0.0001	<0.0001	<0.0001	<0.0001

table continues...

Parameter	Unit	Russian MPC	WHO MPC (2011)	GS15-01	GS15-04	GS15-05	GS15-06
Boron	mg/L	2.4	2.4	<0.05	<0.05	<0.05	<0.05
Mercury	mg/L	0.006	0.006	<0.0001	<0.0001	<0.0001	<0.0001
Molybdenum	mg/L	0.25	-	<0.025	<0.025	<0.025	<0.025
Manganese	mg/L	0.1	0.4*	<0.005	<0.005	<0.005	<0.005
Hydrocarbons	mg/L	0.3	-	0.6	0.9	-	-
Benzopyrene	µg/L	0.005	-	<0.002	<0.002	<0.002	<0.002
Gross α-activity	Bq/L	0.2	-	0.053	0.046	-	-
Gross β-activity	Bq/L	1	-	0.11	0.11	-	-

Note: * Denotes an aesthetic/taste/odour based criterion i.e. not a health based value.
† Denotes exceedance of maximum limit allowed by WHO (2011) for drinking

It remains relevant to notice, however, that seasonal water quality variation may exist if aquifer recharge areas are located close to the borehole locations. The duration of pumping or free flow after completion of borehole drilling may also have some impact on the water chemistry. Boreholes GS15-01 and GS15-04 were sampled in June 2015, whilst the other two holes (GS15-05 and GS15-06) were sampled in November.

As shown in Table 24.11, no WHO health-based guidelines were exceeded in all four boreholes. However, iron and chloride were reported at concentrations exceeding aesthetic acceptability guidelines (criteria for taste, odour, and appearance) for drinking water. Water quality of the sampled boreholes was also compared to Russian standard (Table 24.11) MPC, since the water supply quality should comply with such regulation. Accordingly, maximum limits apply also to colour, odour, turbidity, and total hardness of water. Water from borehole GS15-01 shows exceedance of maximum limit for odour, turbidity, and total hardness, whilst GS15-04 borehole water shows exceedance for odour and total hardness, and water from borehole GS15-06 shows exceedance for colour and turbidity. The exception is borehole GS15-05 where no exceedance was observed for the additional parameters defined by Russian regulation, and therefore, only two exceedances remain valid for both WHO and Russian standards.

Comparison of the Investigated Water Supply Boreholes

The water supply investigation boreholes were compared in terms of the achieved pumping yield and drawdown, the estimated aquifer parameters, and water quality. Taking into account all of the above criteria, borehole GS15-05 seems to be the most suitable site for the Project water supply, as it generally fulfils both the required water yield and quality.

Table 24.12 Comparison of Water Supply Potential of the Boreholes

BH ID	Achieved Debit (L/s)	Induced Drawdown (m)	Estimated Hydraulic Conductivity* (m/d)	Estimated Storativity	Remarks on Water Quality and Exceedance of Maximum Allowed Levels	Ranking of Suitability
GS15-01	1.38	14.46 m (12.06m + artesian head 2.4m)	0.26	0.00012	Iron and chloride concentrations exceed the WHO aesthetic guidelines limits for drinking water. It shows also exceedance of the Russian maximum limits for odour, turbidity and total hardness.	3
GS15-04	1.35	8.35 m (7.25m + artesian head 1.1m)	0.28	-	Iron concentration exceeds the WHO aesthetic guidelines limit for drinking water. It shows also exceedance of the Russian maximum limits for odour and total hardness. It has the second lowest salinity (close to that of GS15-05)	2
GS15-05	1.65	14 m (all is only Artesian head)	0.78	0.00016	Total iron and chloride concentrations exceed the WHO aesthetic guidelines limits. No exceedance is found for the additional parameters defined by the Russian regulation. It has the lowest salinity level despite the limited free outflow duration before freezing.	1
GS15-06	0.53	29.37 m, including artesian level (7m)	0.07	1.4E-05	Iron and chloride concentrations exceed the WHO aesthetic guidelines limits for drinking water. It shows also exceedance of the Russian maximum limits for colour and turbidity.	Not suitable

Note: *assuming aquifer thickness of 50 m

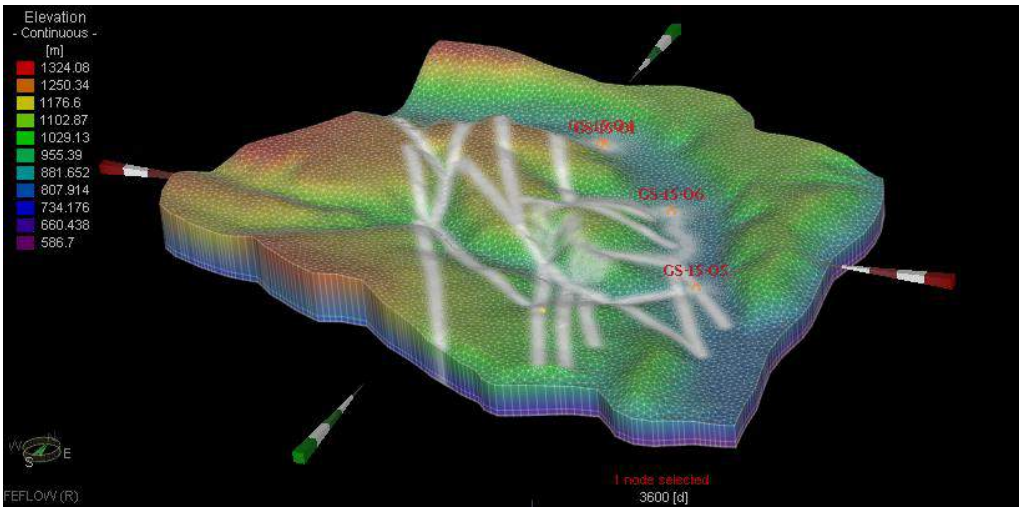
GROUNDWATER MODEL TO SIMULATE PUMPING FROM BOREHOLE GS15-05

A numerical groundwater flow model was constructed using the finite element groundwater modelling software "Feflow" to simulate pumping from a potential water supply well at the location of borehole GS15-05. The model objective is to assess water availability and simulate the consequence of long-term pumping of groundwater from the sub-permafrost aquifer. It also compensates for the lack of long lasting pumping test that was not possible in borehole GS15-05 during the site investigation in November 2015 due to the rapid freezing of the borehole.

Model Extent and Structure

The numerical groundwater model is made up of two layers as shown in Figure 24.9. The top layer is set to be non-active to represent the impervious permafrost layer. The second layer represents the confined sub-permafrost aquifer, which is subject to the pumping simulations.

Figure 24.9 Extent and Structure of the Groundwater Model (white lines are faults)



The discretization of the model was made such that the elements are refined around the faults, the intrusive dykes, and the borehole locations where steep hydraulic gradient can take place during pumping. Accordingly, the size of the finite elements around the investigated boreholes was set to 0.1 m, and around the geological contact set to 10 m.

The thickness of the second layer of the model was set to 50 m, based on the conceptual understanding of the site hydrogeology. The topology of the model layer surfaces was defined according to the surface of the bottom of the permafrost that was estimated from topography and presented earlier in this report. The bottom of the aquifer follows the bottom surface of the permafrost, allowing for 50 m thickness.

Model Hydraulic Boundaries and Parameters

All model boundaries were assumed no flow boundaries and the groundwater model domain was split into three main subdomains:

- fault zones
- intrusive rock formations
- sedimentary rock formations surrounding the fault zones and intrusive rock formations.

The highest hydraulic values for the aquifer were obtained from the test at borehole GS15-05, and assigned to the fault zone. The lowest hydraulic values obtained from the test at borehole GS15-06, was assigned to the intrusive rock mass in the model. Boreholes GS15-04 and GS15-01 were drilled within a sedimentary rock formation, outside the fault zones and intrusive rock mass. The hydraulic properties obtained from the tests at these boreholes were assigned to the larger model area. Similar to the hydraulic conductivity, the specific storage properties were adopted from the results of the field tests, as shown in Table 24.13.

Table 24.13 Hydraulic Properties used for the Numerical Model Layer 2 (Sub-permafrost Aquifer)

Model Zone	Hydraulic Conductivity (m/d)			Specific Storage (S_s)*
	K_x	K_y	K_z	
Intrusive Rock Mass	0.07	0.07	0.07	$2.80E^{-07}$
Fault Zones	0.8	0.8	0.8	$3.20E^{-06}$
Tectonically Unaffected Sedimentary Rock Formations	0.29	0.29	0.29	$2.40E^{-06}$

Note: *Storage coefficient divided by the aquifer thickness of 50m

No groundwater recharge was considered to replenish the aquifer. Such an approach is explicitly conservative to simulate a worse-case scenario, whereby all water pumped for the project water supply would be taken from the current groundwater storage in the sub-permafrost aquifer, without allowing for the renewal of such storage.

Model Calibration

The numerical model was calibrated using the groundwater level observations during the shut-in test in borehole GS15-05. The model was iteratively run until it reproduced satisfactorily the hydraulic head that was observed during the test.

Model Simulation Scenarios

Various water supply scenarios were simulated in the model to assess the possibility of using the sub-permafrost aquifer as a potential source of water for the Project:

- Scenario 1: Water supply pumping at a rate of 200 m^3/d (2.3 L/s), 8 mo/a over a 10-year period. The assumed pumping pattern of 8 mo/a is meant to reflect the annual change in climatic conditions that regulate the availability of surface water as an alternative source of water supply during four months

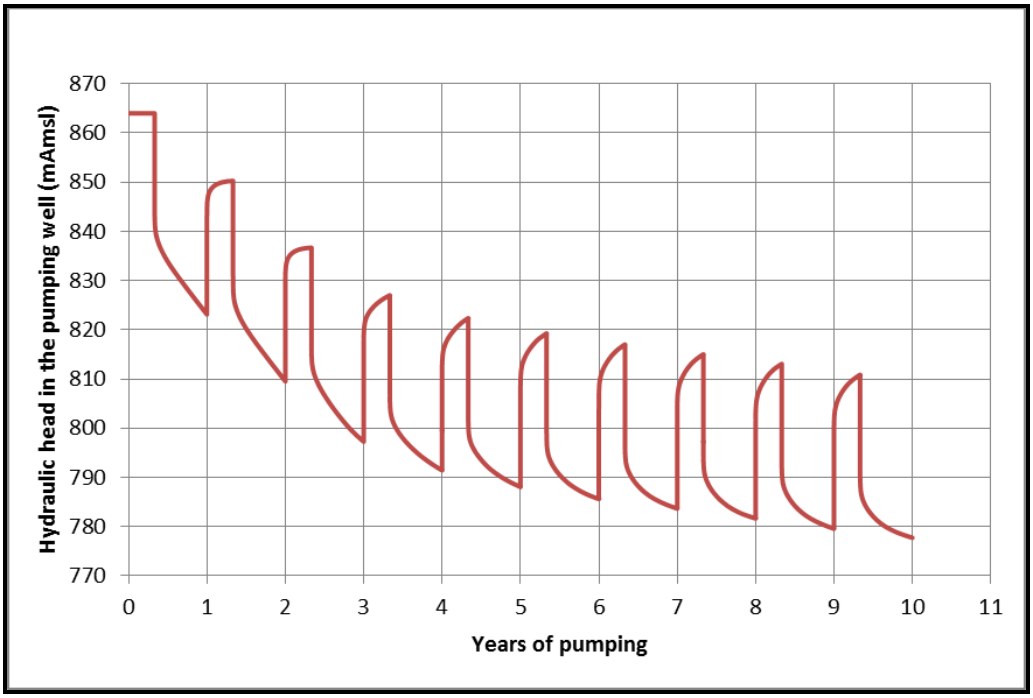
of the year (June to September) when water in the adjacent Sirelendge River is not frozen.

- Scenario 2: Water supply pumping non-stop 12 mo/a at a rate of 200 m³/d (2.3 L/s) over a 10-year period. This conservative scenario is based on the assumption that no surface water will be pumped from the river. The required amount of water supply is pumped from the borehole over the entire year.
- Scenario 3: Water supply pumping at a rate of 260 m³/d (3 L/s), 8 mo/a over a 10-year period.

Results of Scenario 1 (Pumping 8 mo/a at a Rate of 200 m³/d)

The simulated variation in the groundwater level in the water supply well for Scenario 1 is presented in Figure 24.10. The simulated head in the pumping well at the end of the 10 year simulation period drops to 778 mamsl, which is the equivalent of a maximum drawdown of 86 m (from the initial 864 m head) in the pumping well. Comparing this drawdown to the elevation of the bottom of permafrost indicates that after 10 years of pumping from the supply well, the aquifer still remains in a confined state. The results suggest there is sufficient groundwater in the confined storage to supply the processing plant for 10 years of operations.

Figure 24.10 Predicted Water Level in the Water Supply Well – Scenario 1



Results of Scenario 2 (Pumping 12 mo/a at a Rate of 200 m³/d)

The groundwater model simulation results are shown in Figure 24.11 and suggest that the aquifer storage won't be depleted by pumping; however, there will be locations across the model area (outside the Project site) where the sub-permafrost aquifer may change from a confined to unconfined state. The simulated cone of depression in the pumped borehole resulting from pumping according to Scenario 2

is shown in Figure 24.12. The predicted maximum induced drawdown in the pumping well is 93 m.

Figure 24.11 Predicted Water Level in the Water Supply Well – Scenario 2

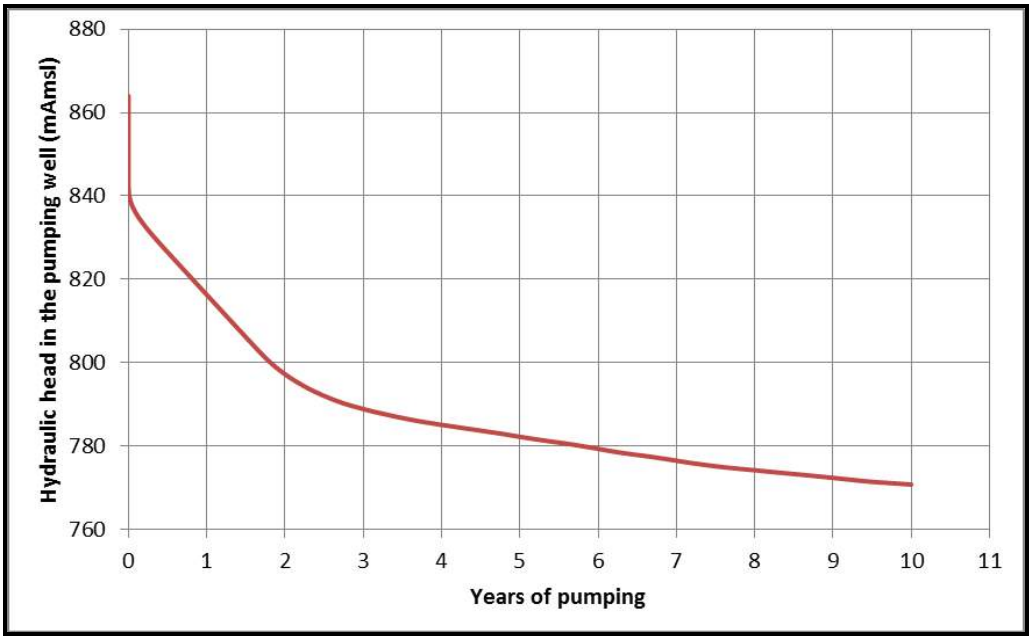
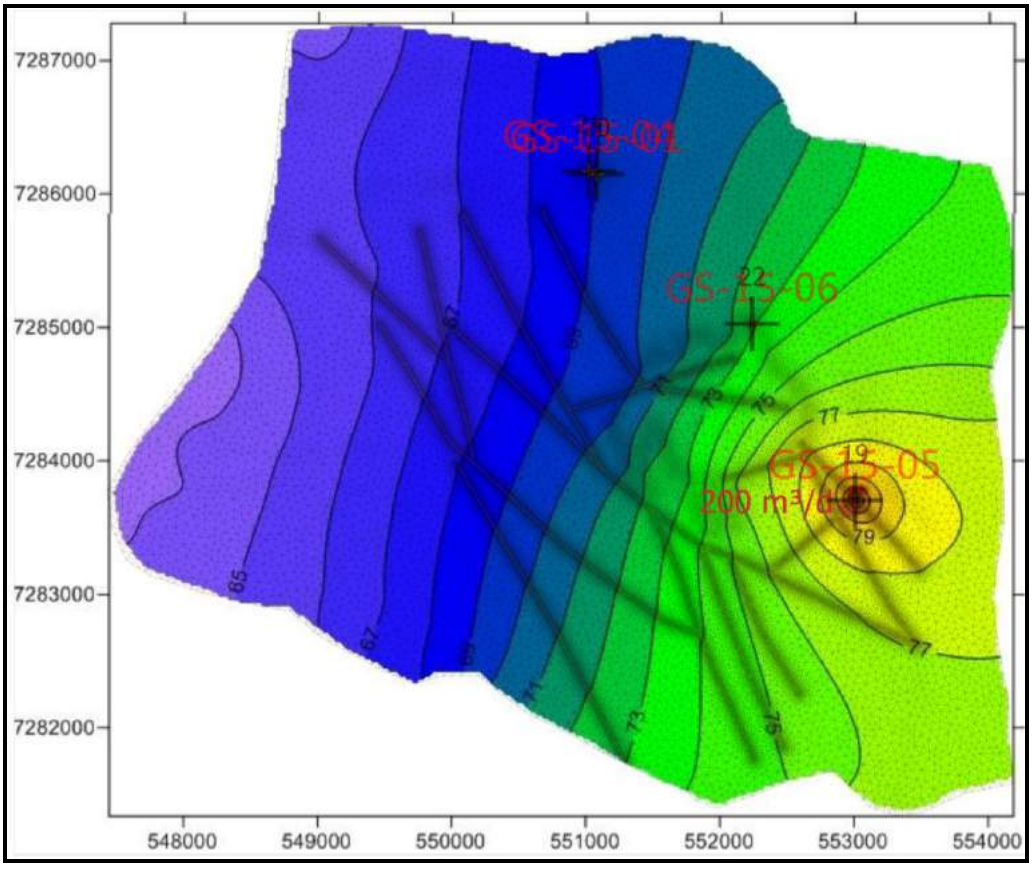


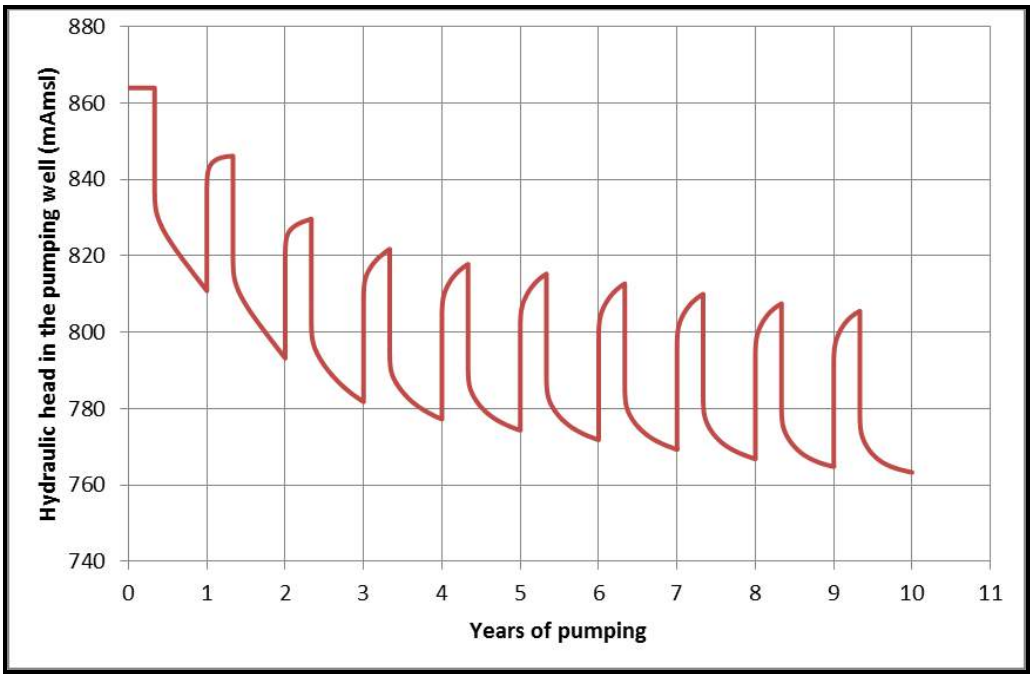
Figure 24.12 Predicted Cone of Depression Due to Pumping – Scenario 2 (m)



Results of Scenario 3 (Pumping 8 mo/a at a Rate of 260 m³/d)

The predicted hydraulic head in the pumping well over time, according to Scenario 3, is shown in Figure 24.13. The model results suggest that the sub-permafrost aquifer storage will not be depleted by the end of the assumed pumping period of 10 years. The predicted maximum induced drawdown in the pumping well, according to Scenario 3 is approximately 101 m.

Figure 24.13 Predicted Water Level in the Water Supply Well – Scenario 3



Conclusions of the Groundwater Model Simulation

The results of the two model scenarios of pumping of 200 m³/d over a 10-year period suggest the availability of enough water in the sub-permafrost aquifer to supply the Project. Increasing the required amount of water supply to 260 m³/d in the third scenario over 10 years did not result in much difference, thus indicating the availability of enough storage in the aquifer. Furthermore, a conservative approach was adopted in the model by not including any aquifer recharge, in order to simulate worst case scenario.

24.4 MANGAZEISKY NORTH OPEN PIT OPPORTUNITY

24.4.1 INTRODUCTION

Silver Bear's stated objective is to become a world-class producer of silver from the wholly-owned Prognoz Vertikalny Resource, whilst continuing strategic exploration works and evaluating and developing additional resources for Silver Bear's continuous growth at optimised capital and operating costs. Upon inspection, it was clear from the outset that the Mangazeisky North deposit contains a large enough Indicated Resource at a sufficient average silver grade to warrant a PEA.

Although the scale of the deposit did not allow it to carry significant capital cost, its proximity to the Vertikalny mine site made it ideal as a satellite mine that could be mined by the same equipment and processed using the same plant as the Vertikalny operation, matching the required production rate. Mangazeisky North offers an opportunity to increase the overall mine life and extend the useful life of the capital equipment on the Mangazeisky property.

The following paragraphs contain a summary of the PEA.

24.4.2 Mining Methods

The Mangazeisky North deposit extends approximately 1,600 m along strike. The vein strikes north-northwest/south-southeast and dips to the northwest at an approximate 50° dip. The deposit is open at depth to the southeast, but closed to the northwest. The mine plan outlined in this section covers only the area of the vein in the geological resource that extends from the surface outcrop on the hilltop to a depth of approximately 500 m.

Geotechnical Considerations

Other than the drilling results available from 79 drillholes (totaling 12,373 m) drilled in the Central Zone of the Mangazeisky North vein during 2008, limited geotechnical information was available during preparation of the mine design.

No geotechnical studies have been completed for the Mangazeisky North deposit in order to establish in situ rock mass strength. Therefore, the open pit geotechnical parameters for the Mangazeisky North deposit have been adapted from those proposed for the Vertikalny deposit (Table 24.14) on the assumption that the rock properties are very similar.

Table 24.14 Recommended Slope Angles for Vertikalny

Zone	Stack Height (m)	Bench Face Angle (°)	Berm Width (m)	IRA (°)
Hanging Wall	20	80	10	56
Footwall	20	70	11	48

Note: The maximum stack height comprises four benches of 5 m each.
IRA – inter-ramp angle
Source: Tetra Tech (2016)

For the Mangazeisky North deposit, the moderate dip of the orebody means that, if the ramps are preferentially located on the hanging wall side the pit, the slope angle for the footwall (west) side of the pit can be increased to 50°. This means that the toe of each 20 m stack will need to be offset from the mineralised contact by at least 11 to 12 m so the pit wall does not intersect the orebody. This assumes there are no adverse structures that run parallel to the footwall that could cause localised failure of the pit slope.

In practice, the overall slope angle (OSA) for the western-side pit wall will be considerably less than 50° due to the fluctuations in dip and the parallel mineralised structures that cause the pit wall to step out and leave extra wide berms.

The OSA design for the hanging wall pit slopes is set at 45° to allow for the ramps and to maintain a maximum IRA of 50°. A 9.5 m berm will be inserted for each 20 m stack to achieve an IRA of 50°. The width of this berm means that it can be used for haulage, as well as providing ample space for clean-up of any spillage.

It is anticipated that the ground conditions will be suitable for the proposed mining method; however, Tetra Tech recommends that detailed geotechnical investigations and analyses are undertaken for future studies.

Permafrost

The mine will be developed in permafrost conditions which are likely to improve ground conditions and stability.

Mining Method Selection

Open Pit

The Mangazeisky North deposit vein dips to the northeast at approximately 50°, with a width between 1.0 and 3.0 m, and an average width of 1.5 m. Its general dimensions, geometry, and the location of the outcrops make it amenable to conventional open pit mining.

The open pit operation will follow the same pattern as that used for the Vertikalny deposit, with relatively small excavators (1.6 to 2.5 m^3) matched with 30 to 40 t haul trucks. Once the open pit operation at Vertikalny is close to completion, the mining equipment used at Vertikalny will be progressively moved across to Mangazeisky North.

As with the Vertikalny deposit, selective mining of the veins will be required to minimise loss of mineralised material, along with the addition of waste dilution. The bench height has therefore been limited to 5 m and each bench will be mined with two, 2.5 m flitches.

The Mangazeisky North deposit will not have its own processing plant; material mined from the pit that is above the cut-off grade will be transported to the run-of-mine (ROM) pad at Vertikalny for processing. Mineralised material above 75 g/t silver and below the cut-off grade will be stockpiled at Mangazeisky North as a potential source of mineralised material in the future.

It is assumed that waste material will be dumped outside the perimeter of the open pit on the flanks of the Mangazeisky North deposit. There is potential for in-pit dumping of waste, but this will sterilise the Mineral Resource below the pit unless underground mining is considered.

Mineral Resource Extraction

Mining Strategy

The long strike length of the Mangazeisky North vein allows it to be mined in different phases if required; initially targeting the high-grade zones that exist throughout the deposit to maximise revenues and minimise the payback period. The most

accessible high-grade outcrops along the hilltop will be mined first as part of the starter pit.

The pit optimisation identified two distinct pits which can be treated as phases. The southern portion is by far the largest and can be further subdivided if required.

In practice, the phases will generally be mined simultaneously, as the limitations on vertical advance rate and total production rate mean that there must be sufficient exposure of mineralised material at all times. This means that the deposit will effectively be mined in a top-down approach following the predominant dip direction.

Cut-Off Grade

The tonnes confined within the mine plan were derived by applying an internal (mill) cut-off grade to the diluted model grade as derived in Table 24.15.

Table 24.15 Cut-Off Grade Calculation

Parameter	Unit	Value
Sale Price	$/ troy oz Ag	20.0
Refining Cost	$/troy oz Ag	0.5
Process Recovery	%	85.0
Mining Recovery	%	95.0
Waste Dilution	-	as modelled
Royalty Factor	%	6.5
Mining Cost	$/t processed	2.53
Processing Cost	$/t processed	53.4
G&A Cost	$/t processed	41.0
Calculated Cut-off Grade	g/t	190

Note: Process cost includes a transport cost of $1.40/t

It should be noted that the dilution factor is included in the planning model and that the cut-off grade applies to this diluted model. The methodology behind the modelling of dilution is discussed further in this section.

The internal cut-off was rounded up to 200 g/t to account for the uncertainty in the estimation of the modifying factors. This is regarded as an acceptable level of accuracy for a PEA.

Mine Production Rate and Mine Life

The production for the Mangazeisky North open pit was calculated using typical industry estimation practices and equate to the rates used in the Vertikalny Feasibility Study (Tetra Tech 2016c). Maximum open pit production was constrained by:

- a vertical advance rate per pushback of up to 60 m/a
- a total mine production rate of up to 10,000 t/d
- an average feed rate to the Vertikalny ROM pad of 330 t/d
- 33 d/a lost due to severe weather conditions.

These rates are dependent on the strike length of a pit developed at any point in time, and the vertical open pit depth progression realised per annum.

The mine life for the Mangazeisky North open pit has been estimated at 2.0 years based on a mineral inventory of 194 kt and an average production of 110 kt/a of mineralised material**. Note that Inferred material is treated as waste and is given zero value in the pit optimisation, mine schedule and financial modelling.**

Pre-production at Mangazeisky North is expected to overlap with mining the Vertikalny open pits by up to six months. Upon completion of Mangazeisky North production, mining will switch back to the underground operation at Vertikalny.

Mining Extraction Plan

The mining extraction plan starts with initial open pit trench mining, targeting the high-grade silver mineralisation zones with the progression of the trenches, to fully developed pushbacks using a bench type extraction.

A breakdown of the total Mineral Resource and the extraction is detailed in Table 24.16.

Table 24.16 Mangazeisky North Mineral Resource Extraction Summary

Parameters	Tonnes (t)	Contained Ag (koz)	Ag Grade (g/t)
Total Mineral Resources @ $20.00/troy oz	593,231	9,280	487
Planned Extraction – Open Pit ROM	193,202	4,193	675

Note: The quantities shown include appropriate mineralisation losses and planned mining dilutions at a cut-off grade of 200 g/t silver. The Mineral Resources include Measured, Indicated, and Inferred classifications whilst the planned extraction only includes Measured and Indicated Resources

Dilution

When mining narrow veins, such as those found at Vertikalny or Mangazeisky North, a higher degree of mining dilution is expected. At Vertikalny, the planned dilution was calculated using industry standard over break assumptions for a true mineralisation width of 1 to 2 m, resulting in a dilution factor of 30%. In addition, a mining recovery of 95% was applied to reflect the style of selective mining.

The situation at Mangazeisky North is slightly different to that of Vertikalny because the true vein width is generally less than that at Vertikalny, and a significant proportion of the model has isolated blocks of mineralisation with a minimum cell width of 1 m. These blocks will have a higher degree of dilution and loss than those where the zones are more contiguous and the vein width exceeds 2 m.

To test this theory, a Datamine modelling procedure was used called DILUTMOD. The procedure allows the user to specify a dilution skin width in the X, Y, and Z directions for all blocks that are flagged as a contact between mineralisation and waste. The details of the methodology are described in a case study by Wheeler and Randall (2016).

Using a skin dilution width of 0.25 m in the X and Y directions, and 0.1 m in the Z direction, the grade dilution factor was estimated to be 36% at a zero cut-off grade, which means that any dilution is at a zero silver grade. This is appropriate in the context of a vein type orebody such as Mangazeisky North, where there is a sharp contact between mineralised and non-mineralised material.

Figure 24.14 shows that the width of the orebody has a significant impact on the dilution factors. With a minimum block size of 1 m x 1 m in the horizontal direction, the in situ grades for the vein (coloured purple) are reduced by approximately 10% due to the skin dilution. Conversely, the grade of the un-mineralised blocks of the footwall and hanging wall are raised to above the cut-off. In this instance, there is an increase in the tonnage of material above cut-off, but a significant reduction in average grade.

In areas of the mine where the orebody has isolated blocks (Figure 24.15) that form stringers, the dilution can exceed 40% and, in some circumstances, there will be loss where the diluted grade falls below the cut-off. It should be noted that the orientation of the blocks to the strike of the vein is an important factor and can result in loss and dilution in its own right. It is difficult to measure this, and difficult to avoid if the strike direction varies.

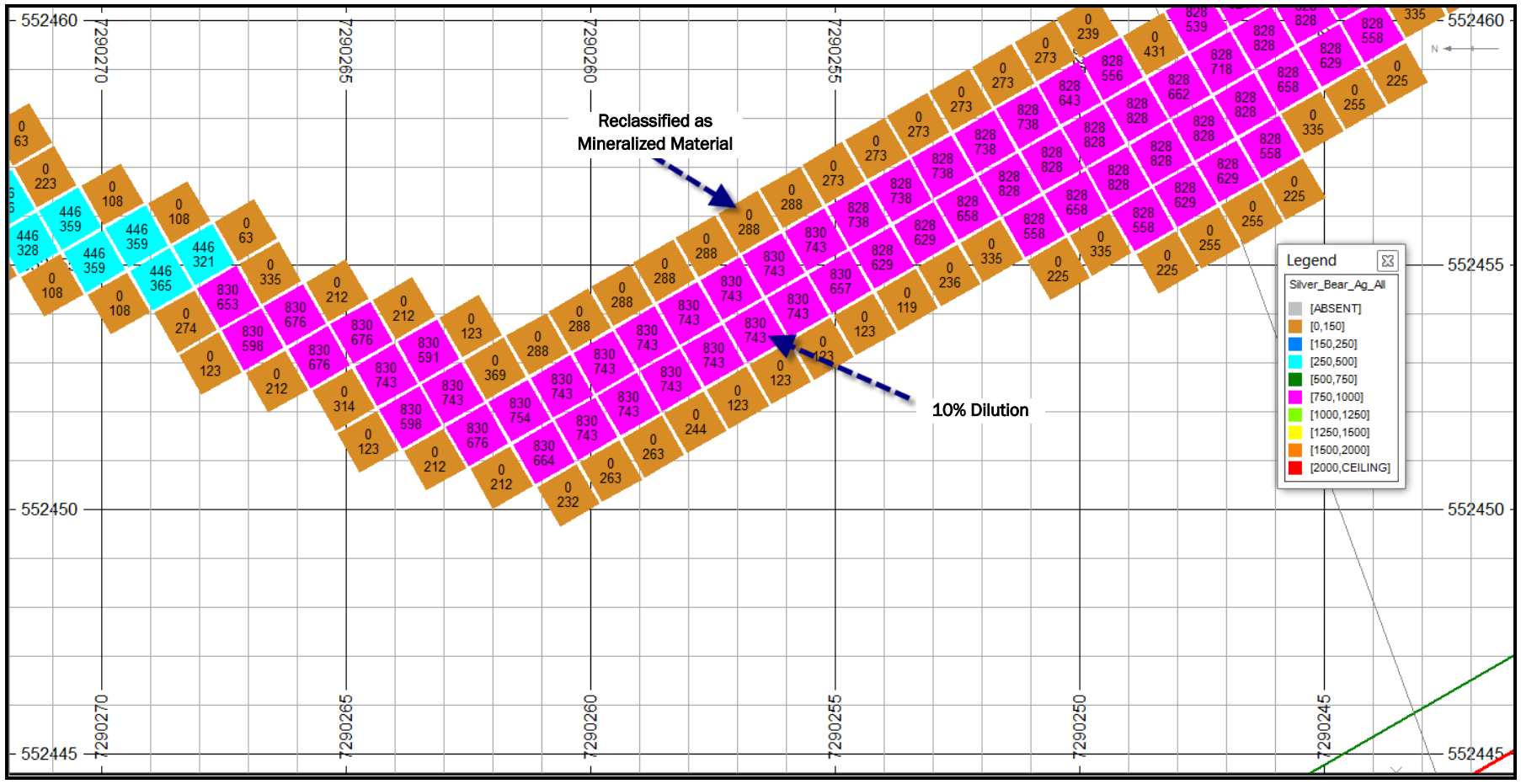
Dilution and loss are caused by a number of factors:

- the grade estimation method
- the physical aspects of blast movement
- interpretation of dig lines
- the ability to excavate to the dig lines.

The modifying factors for mining recovery and dilution are, in this instance, proxies that equate the estimated tonnage and grade in the model with the actual feed quantities to the process plant.

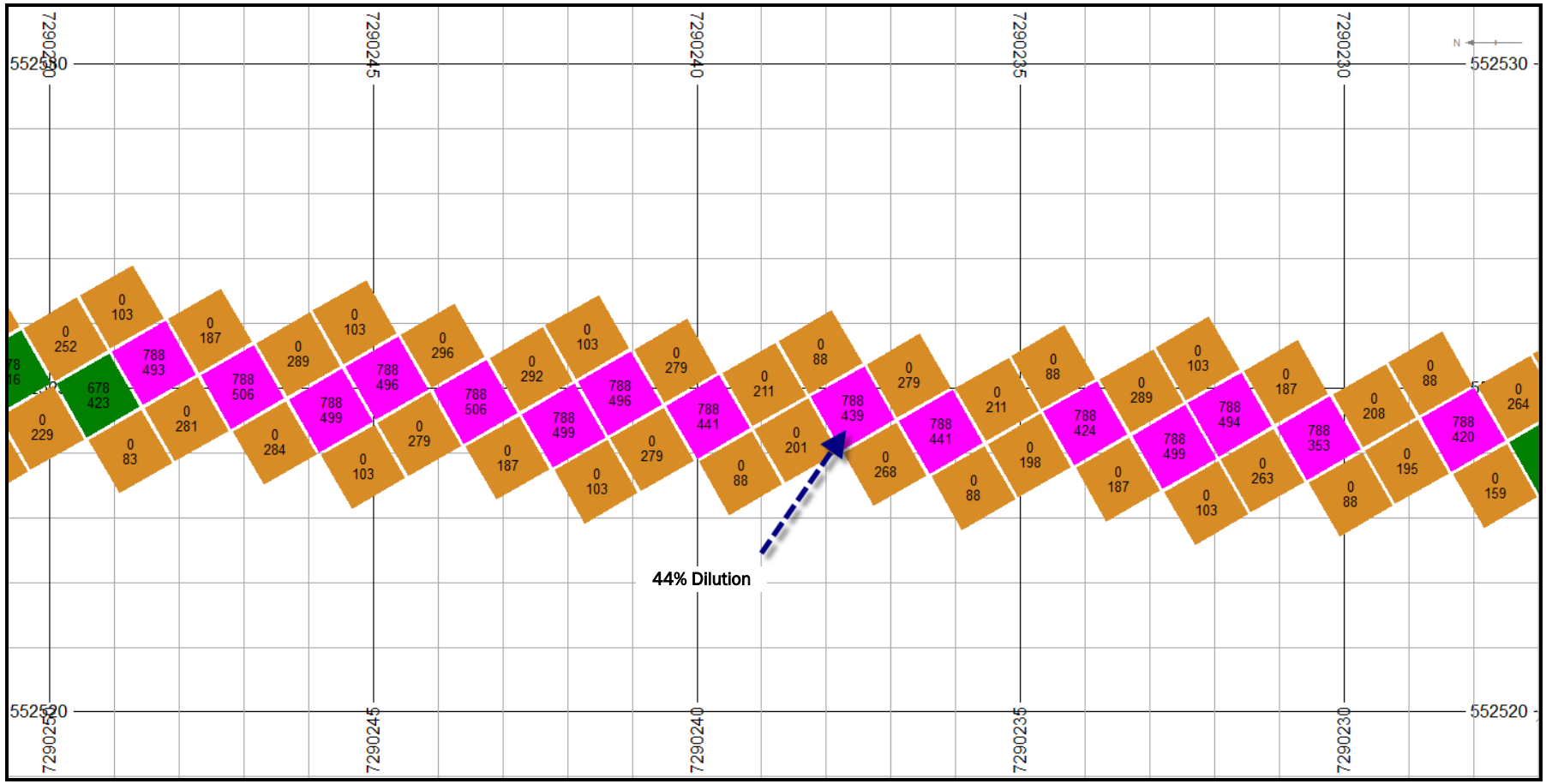
Once in production, reconciliation studies will be required at each stage of the process (grade estimation, grade control, and mine operations) to break down these factors into their constituent parts. However, at this level of study, the methodology presented here is appropriate, and has the advantage that the dilution is driven by the orebody geometry. This was a significant factor in determining the economic pit limit and would have been misleading to just use global factors.

Figure 24.14 Plan View of the Resource Model Showing In Situ and Diluted Grades (Main Orebody)



Source: Tetra Tech

Figure 24.15 Plan View of the Resource Model Showing In Situ and Diluted Grades (Stringers)



Source: Tetra Tech

Grade Control

In order to minimise loss and waste dilution, the mining method and grade control procedures are critical to the efficient running of the mine. This is particularly true at Mangazeisky North as the moderate dip of the orebody poses particular issues with regard to mining recovery and waste dilution at the contacts. This was minimised by selecting 5 m a bench height and mining the benches in two, 2.5 m flitches (3 m when allowing for swell).

The narrow width of the veins is another critical factor; even with a relatively small excavator (1.6 m^3), it will be difficult to accurately mine to the mineralised contacts if the vein width is less than 2 m. The approach taken was to model the skin dilution and include a substantial amount of footwall and hanging wall waste. The waste dilution results in the grade being reduced by at least 30% and substantial amounts of the Mineral Resource excluded as being uneconomic.

An essential part of grade control will be to complete infill drilling to accurately identify the mineralised contacts, and make a decision on where to place the dig lines that define the material types (e.g. mineral and waste).

Blasting on 5 m benches should ensure adequate control in the form of sufficient samples (controlled by burden, spacing, and downhole sample spacing) and visual identification of contacts. However, it is not ideal in terms of the configuration of the blast with the selected drills. With a hole size of 105 mm in diameter, the ratio of charge length to stemming means there will be considerable movement during blasting. The blast displacement needs to be limited by choke blasting and it is possible that blast mates may help to limit vertical movement.

Good operating practice will include mining perpendicular to the dip direction to expose the toe of the hanging wall on each flitch, as well as mining along strike to mine to the toe of the footwall. Waste contamination is inevitable due to the rill angle of the muck pile and visual identification by the geologist will be required to assist the operator. This means that mineral loading should normally only take place during daylight hours, although provision for lighting plants means that it may be possible to continue where the orebody is wider and more continuous.

MINE OPTIMISATION

Open Pit Optimisation

During initial mining assessments, Tetra Tech determined that the Mangazeisky North deposit is best suited to open pit mining. The extension of the orebody in depth, below the open pit, is unlikely to support underground mining.

The open pit optimisation parameters and the geotechnical parameters used in Datamine NPV Scheduler software (NPV Scheduler) are summarised in Table 24.17 and Table 24.18, respectively.

Table 24.17 Optimisation Parameters (Base Case)

Parameter	Unit	Result
Silver Price for Optimisation	$/troy oz	20.0
Metallurgical Recovery of Silver	%	85.0
Royalty	%	6.5
Refining Cost	$/troy oz	0.5
Mining Cost	$/t mined	2.5
Mining Recovery	%	95.0
Transport to Vertikalny	$/t hauled	1.4
Processing Cost	$/t processed	50.0
G&A	$/t processed	42.0

Table 24.18 Geotechnical Parameters

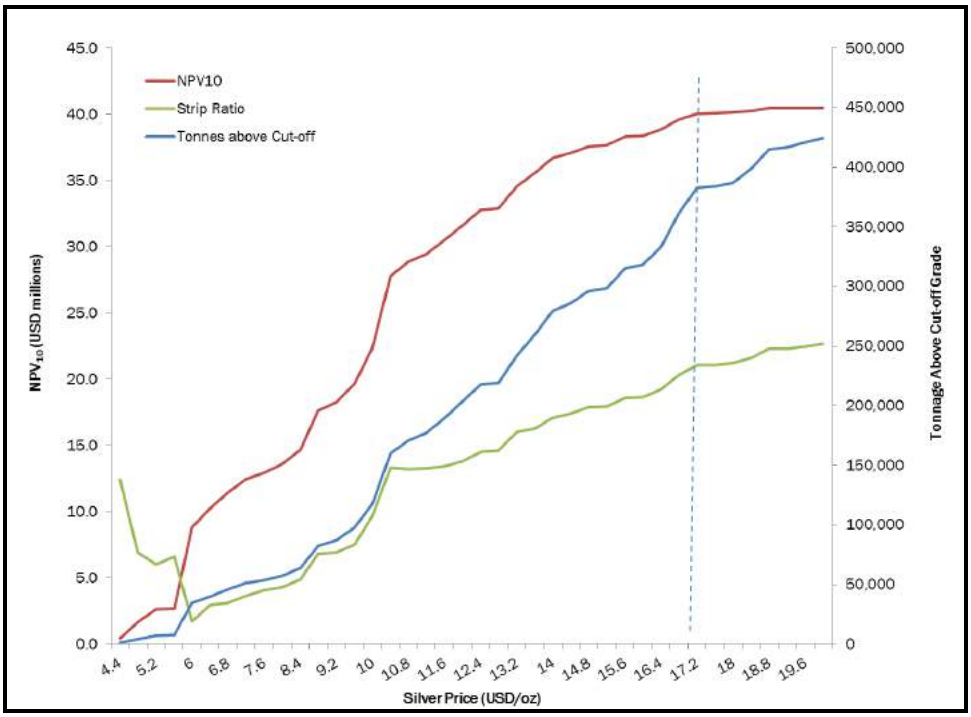
Parameter	Unit	Result
Overall Open Pit Slope Angle (Hanging Wall)	degrees	45
Overall Open Pit Slope Angle (Footwall)	degrees	50

Note that a transport cost of $1.40/t has been included to cover the cost of hauling the mineralised material from Mangazeisky North to Vertikalny. This equates to a cost of $0.06/t/km hauled.

Figure 24.16 illustrates the cumulative NPV at a discount rate of 10% (NPV_{10}), the tonnes above cut-off grade, and the waste stripping ratio for a set of nested economic shells that vary in price factor ranging between 0.2 ($4.00/troy oz) and 1.0 ($20.00/troy oz) in increments of 0.02. The selected pit shell was then developed into the final pit design, including ramps and other access constraints.

The ultimate pit limit shell selected showed a price factor of 1.0, in order to maximise the feed tonnes to the plant. As show in Figure 24.16, there is a possibility of slightly reducing the size of the pit by selecting a price factor of 0.86 ($17.20/troy oz) to limit the stripping ratio. However, this will reduce the tonnes above cut-off by as much as 10% because the relationship between tonnes above cut-off and silver price is almost linear at this price range.

Figure 24.16 Results from the Open Pit Optimisation of Mangazeisky North



The primary driver for pit optimisation is the metal price. Sensitivities were investigated by varying the metal price between $16.00/troy oz and $22.00/troy oz (Table 24.19). Base Case assumptions shown in Table 24.17 and Table 24.18 were used for all other parameters.

Table 24.19 Pit Optimisation – Sensitivity to Price

	Units	Silver Price ($/troy oz)			
		16.00	18.00	20.00	22.00
Rock	kt	5,037	6,803	7,218	8,355
Waste	kt	4,887	6,618	7,025	8,145
Mineralised Material	kt	150	185	193	210
Silver	t	108	126	130	139
	g/t	721	685	675	661
Strip Ratio	t:t	33	36	36	39
Free Cash Flow	$/t mineralised material	106	118	147	168

The results are equally insensitive to the choice of overall pit slope angle because the pit follows the footwall contact wherever possible. On the hanging wall side, the topography is favourable because the hillside slopes away to the east, which means that the strip ratio does not significantly increase with pit depth in the price range considered. This can be seen in Table 24.20, where the overall slope angle varied between −4° and +2° from the standard pit slope parameters for the $20/troy oz Base Case.

Table 24.20 Pit Optimisation – OSA

	Units	OSA Relative to $20.00/troy oz Base Case (degrees)			
		-4	-2	0	2
Rock	t	7,556	7,813	7,218	7,037
Waste	t	7,364	7,611	7,025	6,844
Mineralised Material	t	193	201	193	193
Silver	kg	130	134	130	130
	g/t	676	667	675	676
Strip Ratio	t:t	38	38	36	35
Free Cash Flow	$/t mineralised material	143	139	147	150

Open pit optimisation parameters such as royalties (6.5%) or refining cost ($0.05/troy oz), generally, have little to no impact on the open pit optimisation result, provided these parameters do not influence the revenue calculations by more than 5%. Any variation on these parameters would be minimal when compared to the input values used; therefore, in this instance, any variance between the actual financial model and optimisation calculations are inconsequential.

The most significant factor was found to be the method of applying the modifying factor for dilution. This can be demonstrated by first running the pit optimisation with global factors of 30% dilution and 95% mining recovery, and then re-running the pit optimisation with the diluted model. The results are shown in Figure 24.17 and summarized in Table 24.21.

Figure 24.17 Sensitivity of the Pit Optimisation at USD20/lb (MI Inventory) to Dilution Assumptions

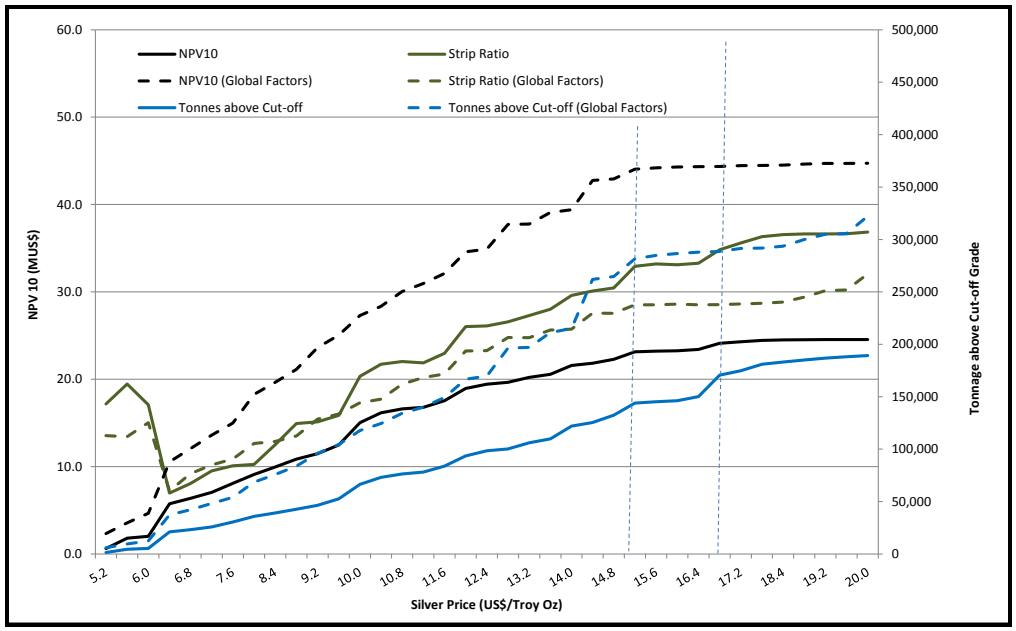


Table 24.21 Pit Optimisation – Comparison of Dilution Assumptions

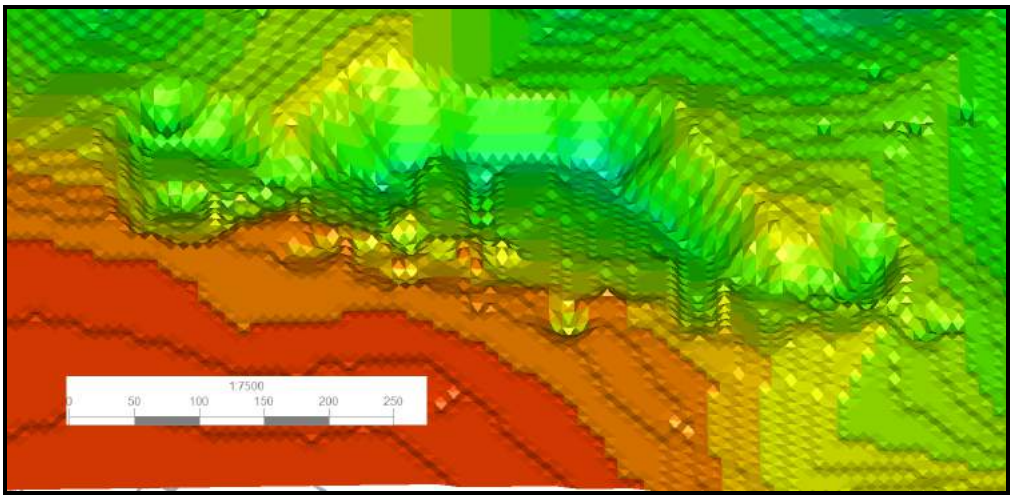
Parameter	Unit	Using Dilution Model	Using Global Factors	Variance (%)
Total Rock	Mt	7.22	10.8	33
Waste	Mt	7.03	10.4	33
Plant Feed	kt	193	331	42
Contained Ag	t	130	217	40
	g/t	675	653	3
Strip Ratio	t/t	36	31	16
NPV_{10}	$ millions	27	49	45

By using the dilution modelling methodology of DILUTMOD, the tonnage above cut-off grade reduced by 42% compared to that obtained with 30% global factors applied for dilution; however the average grade is increased from 653 to 675 g/t silver. The pit size (measured in terms of total rock) is reduced by 33% when using DILUTMOD. The reduction in pit size and increase in average grade translates into a 45% reduction in NPV_{10}.

The optimised pit shell selected for pit design and production scheduling is illustrated in Figure 24.18. The important features are:

- the long axis of the pit follows the strike of the deposit
- the western wall of the pit is defined by the footwall contact
- topography falls away to the east on the hanging wall side
- multiple pit exits can be developed on the eastern side.

Figure 24.18 Mangazeisky North Optimised Pit Shell at $20/troy oz



Source: Tetra Tech

MINE DEVELOPMENT

Mining will take place in three phases:

- Phase 1 – starter open pit trench mining
- Phase 2 – open pit bench/pushback operations
- Phase 3 – expansion to final pit limit.

It is envisioned that Mangazeisky North will be developed just prior to completion of the open pit operation at Vertikalny. There will be a short overlap (3 to 6 months) when the mining equipment from Vertikalny transitions to Mangazeisky North. During this time, the starter pit will be developed at Mangazeisky North so that a ROM stockpile can be built up to feed the centralised plant at Vertikalny.

Phase 1 – Open Pit Trench Mining

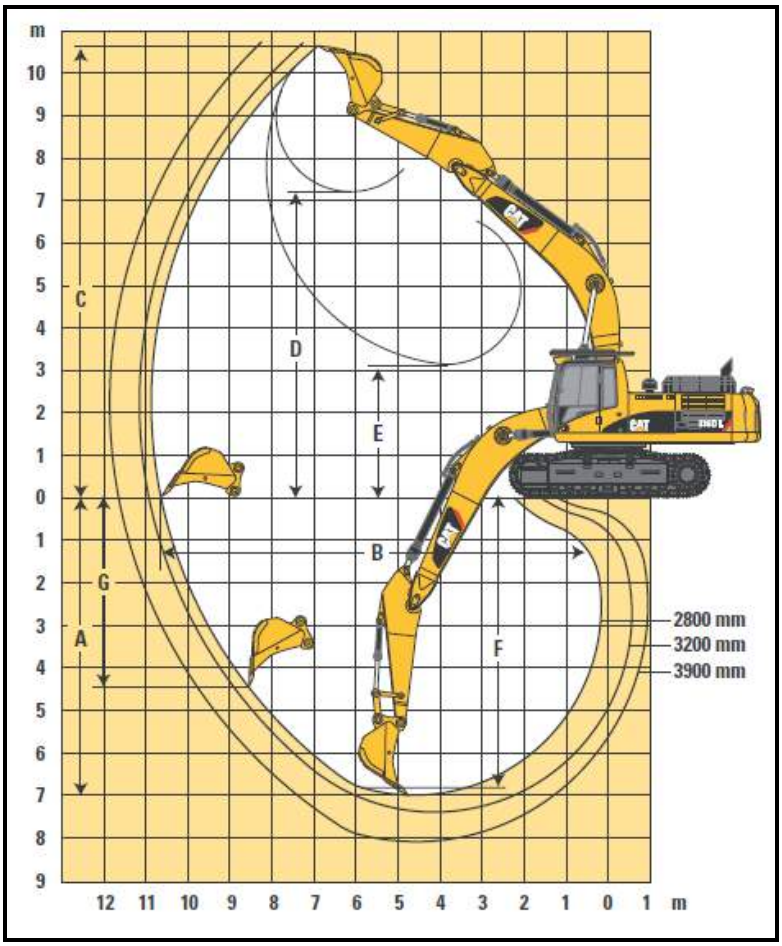
The initial phase of mining will minimise waste rock extraction, whilst targeting the higher silver grade outcrop areas located along the strike, using 5 m deep open pit trenches that will then expand laterally to allow mining of multiple benches. The proposed selection of 5 m bench height, mined in two flitches of 2.5 m, is deemed as acceptable for this level of study.

A small backhoe excavator (Figure 24.19) (CAT 336 with 1.6 m^3 bucket) will predominately be used for this work and for loading other mineralised areas. It will have a digging depth capability of approximately 6.5 m, depending on various factors such as rock density, bucket size, and boom.

A larger excavator (CAT 349 with 2.1 or 2.5 m^3 bucket) will predominately be used for waste mining on 5 m high benches. This machine can be used as a backhoe or a shovel.

Once the initial open pit trenches have been completed, mine development will commence with a series of expansions in waste, in order to develop a fully functional open pit operation.

Figure 24.19 Operating Range of the Caterpillar 336D



Source: Caterpillar

Phase 2 – Open Pit Pushback and Bench Mining

Drilling and Blasting

There are two options for drilling and blasting mineralised material.

Option 1

Option 1 is to choke blast the mineralised material with the waste and allow for the waste muck pile to move upward, with minimal disturbance (mixing of mineralized material and waste) during blasting. It is possible that the drill pattern can be varied across the shot, depending on whether it is waste or mineralisation, but this greatly complicates the initiation pattern and is not advisable.

The waste will be loaded from the high wall side in order to expose the contact with the mineralised material. The moderate dip of the orebody (50°) should mean that this is possible, allowing for the rill angle of the face and the flexibility of the hydraulic excavator (backhoe or face configuration) to accurately mine to a contact.

Once the waste has been cleared, the in-pit geologist will assist the excavator operator during the selective loading of the mineralised material. The mineralised

and waste contact material should be dumped separately on a low-grade, high-contamination, mineralisation stockpile.

Planning for increased fragmentation of the waste by lowering the powder factor will assist with the selecting the mineralised material during the mining process. The waste fragmentation should still allow for efficient loading with the proposed mining equipment.

Option 2

Option 2 is to attempt to drill and blast the waste rock areas on a bench separately. Once the waste is cleared away, the blast pattern for the mineralised material can be drilled and shot.

As with Option 1, by leaving blasted waste rock in front of the face to be shot, the subsequent blast can be "choked" to minimise the blast displacement in the horizontal direction. This will help to reduce the amount of mineralized material loss and waste dilution at the contacts.

Once the drilling of the mineralised material and the blast show have been completed, the peripheral waste around the mineralised material can be removed before loading mineralised material separately to ensure a more efficient loading control, and possibly allow for different (larger) equipment when loading waste.

This method will require further investigation to explore its practicality, as the average vein width is typically less than 3 m which means it is not practical to entirely blast the mineralised material from the waste when mining with a 5 m high bench and 4 inch (105 mm) holes. With this configuration, the burden between holes will be at least 1.9 m and the minimum depth of a shot is likely to be three rows or more.

Grade Control

Grade control is an important consideration when considering the blast patterns. Although a certain amount of additional information on the location of the contacts can be gathered from in-fill drilling with reverse circulation holes, the final definition of the dig lines will be guided by sampling the blast holes.

With both drilling and blasting Options 1 or 2, locating the contacts for mineralisation will not be easy with vertical blast holes that are spaced at the approximate average width of the vein. The grade control process will rely, to a large extent, on face mapping and visual material identification during loading.

Having a geologist on hand at all times, and only loading mineralised material in daylight hours, is advisable. This will be a limiting factor in winter when there will be as little as 5 hours of daylight and it will become necessary to use lighting plants.

Material Haulage

The mineralised material will be hauled from the pit to a temporary ROM stockpile located approximately 500 m from the nearest pit exit. The individual stockpiles will have a maximum capacity of 20,000 tonnes, which will act as a buffer between the mine and the plant.

The ROM stockpile will be reclaimed by a front-end loader and loaded into 20 or 30 t haul trucks for transportation to the centralised processing plant at Vertikalny. It is proposed that the Mangazeisky North and Vertikalny haul operation is sub-contracted out because the mine haul trucks are not suitable for the 12 km haul distance between the sites.

Once the trucks arrive at Vertikalny, there will be an option to either direct dump into the crusher feed bin, or to dump at one of the ROM stockpiles that will be located near to the crusher. It is proposed that a live stockpile capacity of at least 25,000 t is maintained at Vertikalny as a buffer to avoid delays in feeding the plant.

The ROM stockpiles will allow a certain amount of high grading in the first two years of operation by preferentially feeding higher grade material to the plant. This can be achieved if the ROM stockpiles are divided into low-grade, medium-grade, and high-grade stockpiles.

MINE SCHEDULING

Open pit designs were developed from the optimised pit limit and mining sequence. These were loaded into the NPV Scheduler software as pushbacks, which developed quarterly mining production schedules. The LOM for Mangazeisky North is two years, with production taking place over three mining phases, totalling approximately 193 kt of mineralised material at an average grade of 675 g/t silver.

The pre-production period will be approximately six months, which will allow the mining equipment to be progressively moved from Vertikalny to Mangazeisky North, the administration buildings and workshop to be built, and the access roads to the open pit area constructed. The access road from Vertikalny to Mangazeisky will need to be constructed prior to pre-production.

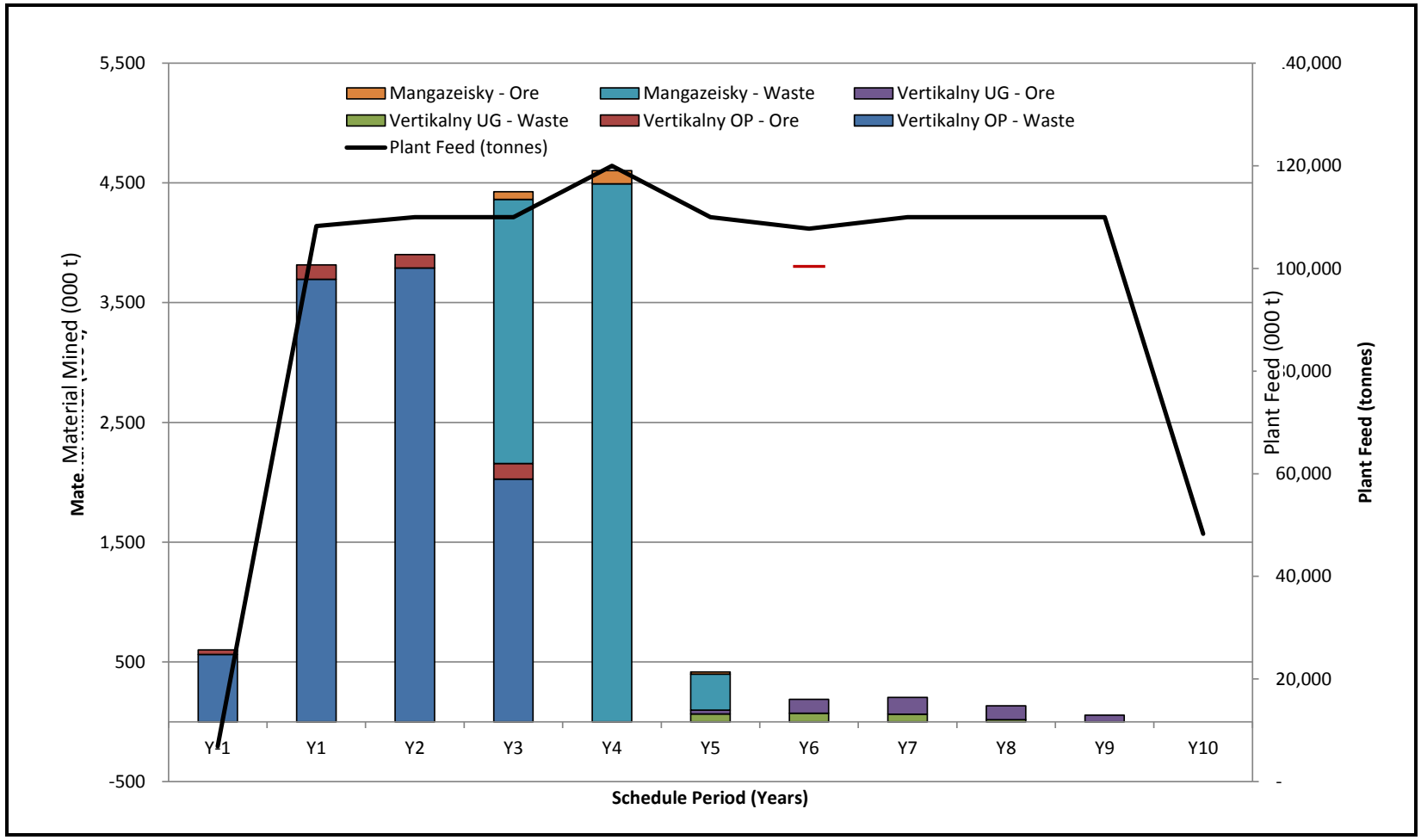
The open pit operations will commence with pre-stripping and open pit trenching of mineralised material that outcrops, or is near to surface. During the first three months of pre-production mining at Mangazeisky North, open pit mining will continue at Vertikalny; however, by this time the production rate at Vertikalny will have dropped to approximately 60,000 t/mo, as compared to over 380,000 t/mo during peak production rates. It will therefore be possible to move the bulk of the fleet to Mangazeisky North whilst still mining at Vertikalny.

The mine production and development schedules are detailed in Table 24.22 and, respectively. The latter summarises the production from Vertikalny (open pit and underground), as well as Mangazeisky North. Preference is given to open pit mining prior to underground mining due to the relative capital and operating costs.

Table 24.22 Mine Production Schedule for the Mangazeisky North Deposit

Activity	Schedule																			
	Year -1				Year 1				Year 2				Year 3				Year 4			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	
Pre-production																				
Establish Access Roads to Mangazeisky																				
Construct Local Infrastructure																				
Establish ROM Stockpiles and Dumps																				
Establish Mine Roads to Pit and Dumps																				
Production Build Up																				
Phase 1 – Open Pit Trenching																				
Full Production																				
Phase 2 – Full Production																				
Phase 3 – Expand to Pit Limit																				
Mine Production																				**Total**
Waste (kt)	-	318-	959	928	1,521	1,355	1,013	603	297	4	-	-	-	-	-	-	-	-	-	6,999
Low-grade Stockpile (kt)	-	-	6	11	333	668	24	60	46		-	-	-	-	-	-	-	-	-	1.147160
Mineralised Material (kt)	-	10-	28	27	28	28	27	28	19	1	-	-	-	-	-	-	-	-	-	193387
Mined Grade (g/t Ag)	-	650-	690	763	700	643	654	641	636	421	-	-	-	-	-	-	-	-	-	675531

Figure 24.20 Combined Mine Production Schedule for Vertikalny and Mangazeisky North



MINING EQUIPMENT

The equipment for the open pit operations at Mangazeisky North will be very similar to those used at Vertikalny. Where possible, equipment will be moved from Vertikalny to Mangazeisky North as open pit operations are curtailed at Vertikalny. The mine schedule allows a three to six month transition period, which will allow a gradual ramp up in production at Mangazeisky North.

The relatively short life of the open pit operations (four years) at Vertikalny means that there should be no need to purchase new equipment for Mangazeisky North, as the peak production rate at Mangazeisky North will be less than that of Vertikalny. Combining this with the expected shorter haul distance for waste and mineralised material means that the fleet requirements (Table 24.23) will reduce rather than increase.

The only exception is in the need to provide for haulage between the mines, as there are no plans to build a processing facility at Mangazeisky North. Instead, a contract haulier should be used to reduce capital costs.

Table 24.23 Mining Fleet for Mangazeisky North

Description	No. of Units per Period				
	Year –1			Year 2	Year 3
	Q2	Q3	Q4		
Drills (105 mm)	2	3	2	3	3
ANFO Truck	1	2	2	2	2
Excavator (CAT 336)	1	1	1	1	1
Excavator (CAT 349)	1	1	1	1	0
Haul Truck (30 t)	3	7	7	8	3
Track Dozer (D9)	1	1	1	1	1
Track Dozer (D8)	1	2	2	2	1
Road Building Truck (30t)	1	1	1	1	1
Grader	1	1	1	1	1
Fuel Tanker	1	1	1	1	1
Water Truck	1	1	1	1	1
Front-end Loader (CAT 950)	1	1	1	1	1
Service Truck	1	1	1	1	1
Pick Up – Light Duty Vehicle	5	10	10	10	5
Lighting Plant	4	4	4	4	2

Based on a typical life cycle of 35,000 hours for the excavators, 50,000 hours for the trucks, and 40,000 hours for ancillary equipment, it is expected that the need to replace equipment prior to completion of mining at Mangazeisky North will be minimal.

A suitable provision has been made in the mining sustaining capital to allow the major items of equipment to operate over the full life cycle of six to seven years. Once the production rate starts to decline from a peak of 3.7 Mt/a at Vertikalny in 2018 and 2019 to 1.9 Mt/a in 2020, it should be possible to put at least two trucks

and one excavator on standby to cover for maintenance on operational equipment until the operation at Mangazeisky North can be ramped up to 4.5 Mt/a in 2021

MINE INFRASTRUCTURE

The Project surface infrastructure is detailed in Section 18.0 and includes key mining surface infrastructure items that will be shared between Vertikalny and Mangazeisky North.

The following additional items will be required at Mangazeisky North:

- open pit mining communications and information technology (IT) systems
- a 12 km haul road between Mangazeisky North and Vertikalny
- ROM stockpile area (25 kt capacity)
- loading facility for haulage between mines
- dump areas prepared for waste dumps
- construction of pit viewpoints
- admin building for operations personnel
- workshop facilities with tyre bay (minor repairs)
- fuel and lube bays
- emergency aid point.

Wherever practical, the existing infrastructure at Vertikalny will be used to service Mangazeisky North. Only essential facilities for daily operations need to be accommodated at Mangazeisky North to enable it to work independently, if required.

Explosive Storage and Handling

The main blasting agent for the open pit will be ammonium nitrate/fuel oil (ANFO), which will be manufactured on site by a purpose built ANFO plant constructed near Vertikalny. The ammonium nitrate will be stored in a large shed in 25 kg bags. The shed will be able to store up to a one-year supply of ammonium nitrate, a maximum of 1,100 t during open pit operations. The fuel oil will be ordinary diesel stored on site in the main fuel depot, near the process plant, and brought to the ANFO plant in a dedicated fuel bowser.

Detonators, primers, and detonating cord will be stored in separate magazines on the surface in accordance with Russian explosive storage regulations. The detonator magazines will be re-supplied by air freight.

For the open pit, all explosives will be collected from the surface magazines. For the later phases of the Project, when the underground mine at Vertikalny is operational, the magazines will be established underground adjacent to the main adits and declines.

Fuel Storage and Distribution

The main fuel storage will be at a central fuel farm located close to Vertikalny's power plant. A smaller fuel farm will be established at Mangazeisky North to supply the mine equipment, to avoid hauling in fuel on a daily basis.

Mine Equipment Maintenance

A maintenance management system should be set up early in the Project to optimise equipment availability and utilisation. All major repairs and planned service maintenance will be carried out in the heavy mobile equipment workshop. This workshop should be fully equipped and capable of machine repair and manufacture of basic spare parts.

Drilling machines will be repaired in the drill shop, which will be equipped to re-sharpen tungsten carbide drill bits and integral steel chisel bits.

Health and Safety

A fully equipped medical centre staffed by qualified medical personnel will be available on site to treat minor injuries and ailments. The seriously injured and sick will be evacuated by helicopter to a hospital in Yakutsk.

A communication link will be established between the Mangazeisky North deposit and the Vertikalny camp. Future studies will consider the feasibility of a helicopter landing pad at Mangazeisky.

A porta cabin will be set aside as a mine rescue station and a mine rescue team established from suitably experienced employees.

MANPOWER REQUIREMENTS

The Mangazeisky mine will operate 24 h/d, 7 d/wk, 365 d/a. However, an allowance of 33 d/a lost due to severe weather has been included in the schedule.

Tetra Tech has assumed a two shift roster due to the remoteness of the Project and the limitations of surface infrastructure.

Table 24.24 indicates the typical mining personnel required without labour contingency. A 25% labour contingency factor has been included for mine personnel in the mine operating cost calculations to ensure that there are reasonable allowances for seven days working cover, shift changes, absenteeism, sickness, and other non-productive time (e.g. meal times and power outages).

It is assumed that overall management, catering, cleaning, laundry personnel, etc. will be accounted for in the G&A costs for the site.

Table 24.24 Open Pit Mining Department Personnel

Open Pit Labour	No. of Positions	No. of People/ Resources
Mechanic	1	4
Turner and Fitter	1	2
Field Mechanic	1	2
Mining Engineer	2	4
Geologist	2	4
Geology Technician	2	4
Surveyor	1	2
Surveying crew	1	2
Labourer	1	2
Drill Operator	3	12
Drilling Assistance	3	12
Backhoe Operator	2	8
Truck Driver	10	40
Truck Driver (Road Building)	1	2
Dozer Operator	3	12
Grader Operator	1	2
Loader Operator	1	2
Watercart driver	1	2
Blaster	1	2
Blasting Crew	2	4
Helper	2	8

Note: The total number of operators will vary during the life of the mine. The numbers shown in Table 24.24 are for the peak production rate.

24.4.3 RECOVERY METHODS

PROCESS FLOWSHEET CONSIDERATIONS

The high-level metallurgical test work completed on the mineralised material from the Mangazeisky North deposit indicates that the mineralisation is amenable to cyanidation. The test work also indicates that acceptable metallurgical recoveries can be achieved in a 24-hour leach retention time for the oxidised sections of the mineralisation.

It is assumed that the mineralisation is analogous to the Vertikalny mineralized material in terms physical parameters such as hardness and relative density.

The primary consideration for the process flowsheet development is the utilisation of the process plant that was designed to process the nearby Vertikalny material with minimal modifications. The process flowsheet for the Vertikalny deposit consists of a standard crushing and milling circuit, followed by gravity concentration and intensive cyanidation. The gravity tails will be processed by agitated tank leach with a 72-hour retention time. The leached slurry will go through a simple counter current decantation washing system and the solution will be processed by direct electro-winning to recover silver metal.

The modified flowsheet for the Mangazeisky North mineralisation consists of a standard crushing and milling circuit, followed by agitated tank leaching (with a 24-hour retention time), simple counter current decantation washing and direct electro-winning.

Tailings from the process plant will comprise filtered solids and a bleed stream from the depleted electro-winning solution and will be disposed of in the TMF.

Process Design Criteria

The process design criteria is presented in Table 24.25.

Table 24.25 Process Design Criteria

Description	Unit	Value	Source
General			
Type of Deposit	-	Silver Polymetallic	5
Plant Operating Schedule			
Shifts/Day	-	2	1
Hours/Shift	h	12	1
Hours/Day	h	24	6
Days/Year	d	365	6
Weeks/Year	wk	52	6
Operational Day/Year – Milling	d	336	1
Availability/Utilisation			
Crushing Plant Utilization	%	75	7
Crushing Plant Availability	%	85	6
Overall Annual Processing Rate	t/a (dry)	170,000	9
Daily Processing Rate	t/d	506	2
Operating Processing Rate	t/h	50	2
Milling Plant Utilization	%	92	7
Milling Plant Availability	%	91	6
Overall Annual Processing Rate	t/a (dry)	170,000	9
Daily Processing Rate	t/d	506	2
Operating Processing Rate	t/h	23	3
Design Processing Rate	t/h	31	2
Ore Characteristics – Design			
Ore Specific Gravity	-	2.50 – 2.65	4
Ore Bulk Density	t/m³	1.60	6
Ore Moisture Content	%	5	6
Bond Ball Mill Work Index	kWh/t	14.6	4
Head Grade - High	g/t	848	9
Head Grade - Lower	g/t	333	9
Head Grade - Average	g/t	531	9
Metallurgical Recovery			
Leach Dissolution	%	78.4	4
Electrowinning Efficiency	%	99.0	4

table continues...

Description	Unit	Value	Source
Silver Recovery	%	77.6	2
Silver Production	kg/a	70,064	2
	troy oz/a	2,252,607	2
Crushing			
Average Tonnage Rate, Operating Total ROM Feed	t/h	50	3
ROM Feed F_{80}	mm	250	6
Primary Crusher Type	-	Jaw (Metso C80 – Equivalent)	2,3,7
Number of Crushers	-	1	2,3,7
Crusher Feed Particle Size F_{95}	mm	300	6
Crusher Feed Particle Size F_{80}	mm	200	6
Crusher Product Particle Size, P_{80}	mm	50	6
Reduction Ratio	-	4.00	2
Closed Side Setting	mm	40	6
Secondary Crushing	-		
Secondary Screen	type	Horizontal Vibrating	6
Area	m^2	1.4	2
Number Needed	-	1	2
Deck Aperture Size	mm	20	6
Secondary Crusher	type	Fine Cone (Metso GP 100 - Equivalent)	6
Product Size (P_{80})	mm	13	6
Closed Side Setting	mm	13	8
Fine Ore Bin	-		
No of Discharge Conveyors	-	1	2,3,7
Product Size on Discharge Conveyor (Design) P_{80}	mm	13	2
Number of Feeders	-	1	2,3
Type of Feeders	-	Vibrating	2,3,7
Average Operating Tonnage Rate per Stream	t/h	23	3
Design Tonnage Rate	t/h	31	2,3,7
Grinding			
Primary Mill Type	-	Wet Overflow Ball Mill	5
Mill Size (diameter x length)	m x m	3.0 x 4.2	2,8
Motor Size	kW	475	2,8
Number of Mills		1	2
Solids/Pulp Density	%	65	6
Feed Size, F_{80}	µm	13,000	2,6,7
Product Size, P_{80}	µm	75	2,4
Classification Cyclone			
Number of Cyclones (Operating)	-	1	2,8
Number of Cyclones (Standby)	-	1	6,8
Number of Cyclones (Total)	-	2	2,6,8
Cyclone Size (diameter)	mm	508	2,8
Product Size P_{80}	µm	75	2,8

table continues...

Description	Unit	Value	Source
Dewatering Cyclones			
Number of Cyclones (Operating)	-	1	2,8
Number of Cyclones (Standby)	-	1	6,8
Number of Cyclones (Total)	-	2	2,6,8
Cyclone Size (diameter)	mm	508	2,8
Overflow Pulp Density (nominal)	% solids	0	6
Underflow Pulp Density	% solids	70	6
Leach Circuit			
Solids Feed Rate	t/h	23	3
Number of Parallel Trains	-	1	6
Feed Rate per Train	t/h	23	2
Pulp Density	% solids	45	2,3
Nominal Flow Rate	m^3/h	37	2,3
Processing Method	-	Agitated Tank Leach	5
Leach Tanks			
Type	-	Mechanical Agitation	6
Number of Tanks	per train	4	2
Each Operating Volume	m^3	242	2
Total Nominal Resident Time	h	24	4
Each Nominal Resident Time	h	6	2
Tank Dimensions	m x m	7.35 x 7.35	2
Washing Circuit			
Dewatering Cyclones			
Feed Rate	t/h	23	3
Number of Cyclones (Operating)		1	2,8
Number of Cyclones (Standby)		1	2,6,8
Number of Cyclones (Total)		2	2,8
Cyclone Size (diameter)	mm	508	2,8
Overflow Pulp Density (nominal)	% solids	5	2,8
Underflow Pulp Density	% solids	60	2,8
Clarifiers			
Clarifier Type	-	High-rate Clarifier	2
Number of Clarifiers	-	1	2,8
Clarifier Size (diameter)	m	9.00	2,8
Clarifier Underflow Solid Density	%	40 - 50	2,8
Overflow Pulp Density (nominal)	ppm	100	8
Slurry Transportation Method	-	Pumping	2,8
Filters			
Filter Type		Plate & Frame (Recessed Type)	8
Size	m x m	2 x 2	8
Number of Chambers	-	60.00	8
Filter Cake Moisture Content	%	15.00	6
Electrowinning			
Primary Electrowinning			
Type	-	EMEW Powder cell	8
Number of Cells	-	140	2,8

table continues...

Description	Unit	Value	Source
Cell (diameter)	mm	200	8
Operating Schedule	h/d	24	6
Type of Cathode	-	316SS	7,8
Type of Anode	-	316SS	7,8
Cathode Sludge Removal Method	-	Pressure Back Wash	7,8
Secondary Electrowinning			
Type	-	EMEW Polishing Cell	8
Number of Cells	-	80	2,8
Cell (diameter)	mm	200	8
Operating Schedule	h/d	24	6
Type of Cathode	-	316SS	7,8
Type of Anode	-	316SS	7,8
Sludge Filtration			
Filter Type	-	Plate & Frame, Pneumatic/ Hydraulic Closure	7,8
Capacity	l	85	2,8
Operating Hours	h/d	6	6
Filter Cake Moisture	%	20	6

Process Description

A crushing and milling circuit will be employed to produce a suitable feed for cyanide leaching at a size of 80% –75 µm. The mill product will be pumped to a cyclone separator which will size the material so that the fine overflow is fed to a conventional leaching circuit and the coarse underflow can be returned to the mill sump.

The cyclone overflow will be fed to a conventional leach circuit; thereafter the slurry product will be dewatered in a hydrocyclone with the cyclone underflow feeding to a plate and frame filter press for additional dewatering, and the overflow feeding to a clarifier. The overflow from the clarifier will feed to the direct electrowinning circuit, while the clarifier underflow will form part of the filter press feed.

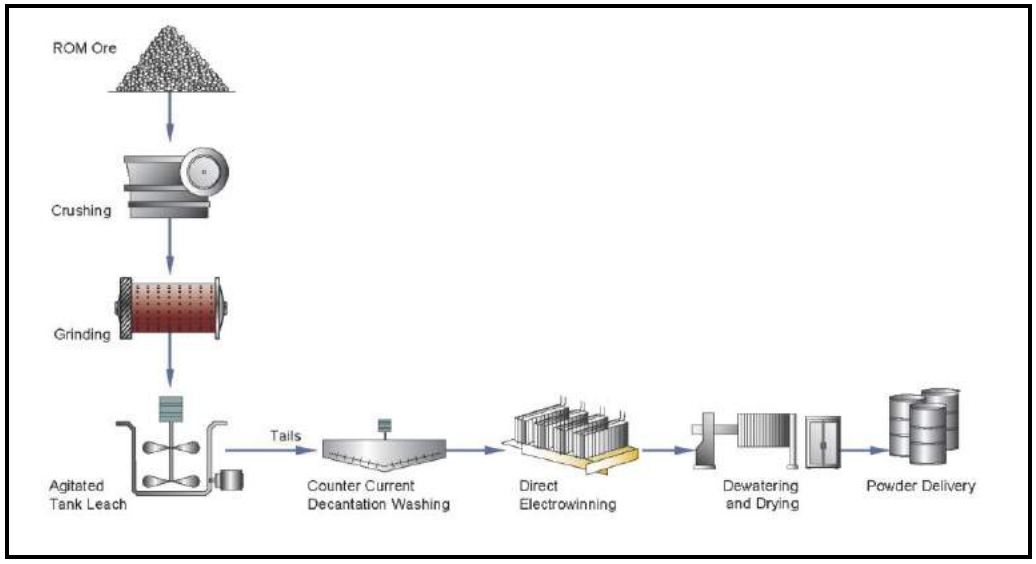
The solids within the filter press feed are leach tailings and will be dewatered and washed in the filter press. The filter wash cycle will ensure that liquid associated with the tailings, which contains silver and cyanide, has been rinsed from the solids tailings before the final tailings is discharged onto the tailings stockpile where it will be reclaimed for tailings dry stacking.

Direct electrowinning will be used to recover silver concentrations higher than 100 ppm from the leach solution. Owing to the high silver resource grades, it is possible to use direct electrowinning to recover the silver. In this technique, the high-grade silver-rich solution from the leaching process is passed directly to specially constructed electrowinning cells for direct recovery of silver onto cathodes which is then removed in a powder form. This method reduces the capital cost considerably and is easier to operate and maintain because of its modular design.

The silver powder from the direct electrowinning process can be transported off site to a suitable refinery.

The dry solids tailings produced from the selected flowsheet will be dry stacked into the TMF. A block flow diagram of the process plant is shown in Figure 24.21.

Figure 24.21 Block Flow Diagram of the Proposed Process Plant



Source: Tetra Tech

Crushing

ROM mineral with a maximum size of approximately 300 mm and 80% passing 200 mm will be trucked to the ROM pad, and the material will feed into the crushing and screening circuit. The crushing and screening unit will be arranged for closed circuit crushing. The ROM mineral will feed to a vibrating grizzly where the fine ore will be removed and passed directly to the secondary crushing circuit screen. The coarse fraction will be crushed in the primary jaw crusher. The product from the jaw crusher will be sent to the secondary crushing circuit screen.

The secondary crushing circuit will contain a secondary cone crusher operating in closed circuit with a single deck screen to produce a final screen product with a P_{80} of 13 mm, suitable for the ball mill feed. The screen undersize from the secondary crushing circuit will be conveyed to a crushed ore bin where it will be reclaimed and fed to the processing plant as required.

Milling

The crushed material will feed directly onto the ball mill via a feed conveyor belt and feed hopper. A single ball mill will be used to achieve the required grind size for cyanide leaching. The ball mill will be a standard open discharge mill operating in closed circuit with two stages of hydrocyclones in order to produce a ground product suitable for the downstream leaching process.

The estimated size of the ball mill is 3.0 m x 4.2 m installed with a 475 kW motor. The mill product slurry will be pumped to a two-stage cyclone system, where the primary cyclone will function as a classifier with a separation of size of P_{80} = 75 µm. The underflow from the primary cyclone will be recirculated back to the ball mill. The overflow from the primary cyclone will be densified at the second cyclone to approximately 45% solids, and gravitate to the cyanide leach tanks. The overflow

from the densifying cyclone will be pumped to the process water tank. It is anticipated that the solids content of this overflow will be nominally zero.

Leaching

Leaching of the ground slurry will be completed in agitated leach tanks, designed to give a total residence time of approximately 24 hours. This will be achieved by only using four of the leach tanks and bypassing the other two tanks of the Vertikalny leach circuit

At the end of the leaching, slurry from the last leach tank will be pumped to the dewatering circuit in order to separate the solids and solution for direct electrowinning.

Lime slurry and cyanide solution will be added to the leach tanks to maintain a pH of approximately 11.5 and free cyanide concentration at least 2, 000 ppm throughout the leaching. The lime slurry and cyanide solution will be prepared in the reagent preparation area and pumped into a ring main, where additions can be made to the leach tanks as required.

Dewatering, Filtration and Clarification

Solid-liquid separation of the leach product will be achieved using a combination of a dewatering hydrocyclone, a plate and frame filter press, and a lamella clarifier. The slurry from the last leach tank will be pumped through a dewatering hydrocyclone. The hydrocyclone overflow will be further clarified in a lamella clarifier in order to provide a suitable overflow solution for the direct electrowinning process.

The hydrocyclone underflow will pass into an agitated filter feed tank. From there, the slurry will be pumped to a plate and frame filter press. The filter press cycle will include a wash cycle to remove any silver containing solution from within the leached solid tailings. The wash water will also remove any cyanide that is associated with the filtered solids. The filter press filtrate will feed into the clarifier, with the filtrate produced during the filter cake wash returning to the water recycle circuit for re-use within the plant. The solid product from the filter press will be conveyed to the TMF.

Direct Electrowinning

The direct electrowinning process will employ patented emew® cell technology to recover silver, which can be directly smelted to produce doré silver bars or alternatively the precipitate can be shipped directly to the refinery. The direct electrowinning circuit will be operated in two stages, with both stages consisting of emew® cells.

The primary stage, consisting of powder cells, which reduce the silver solution from approximately 600 ppm to 50 ppm silver. The secondary stage, consisting of polishing cells, will reduce the solution to below 10 ppm silver prior to discharge.

Barren solution from the electrowinning process will return to the process water tank and subsequently used as dilution water for mill feed, or used for slurry density control purposes in the leach circuit.

PLANT SERVICES AND REAGENTS

Typical plant services include an air compressor to supply air to the agitated leach tanks for effective silver dissolution and for instrument air. It is expected that plant instrumentation will be minimal.

Water

Fresh water will be supplied to a dedicated process water tank and process water pumps will deliver water as required to the plant. Barren solution will be recycled as part of the process to prevent excess fresh water requirements; however, this will be limited to ensure that there is no build-up of deleterious elements within the slurry streams.

Cyanide and Lime

The principal reagents are cyanide and lime. Lime will be delivered to site in 1 t bulk bags as hydrated lime powder. The lime will be mixed in a dedicated agitated mixing tank, typically with a capacity for one day's consumption, and transferred to a storage tank where water will be added and the resulting lime slurry pumped into a ring main system delivering the lime to the leaching circuit to maintain the pH at 11.5 as required. The lime bags will be added to the mixing tank using a hoist and bag-breaker installed above the tank. Lime bags will be stored in a dedicated covered storage area.

Cyanide will be delivered to site as 1 t bulk bags containing sodium cyanide briquettes. The briquettes will be added to an agitated cyanide mixing tank, with the required volume of water, by hoisting the bag onto a bag-breaker situated above the tank and enclosed in a steel chute for dust control. The cyanide solution will be made up to approximately 25% concentration.

24.4.4 PROJECT INFRASTRUCTURE

OVERVIEW

The Mangazeisky North deposit will make use of infrastructure associated with the Vertikalny mine, including:

- airstrip
- off-site, on-site, and service road access
- TMF
- ROM pad and crushed ore stockpile
- borehole water supply and distribution system
- water treatment and distribution
- sewage collection and management
- surface water management
- process and ancillary facilities
- facilities for administration and an assay laboratory

- heavy and light vehicle workshops
- fuel storage and distribution
- explosives storage and handling
- emergency helipad
- camp accommodation
- mess and laundry facilities
- medical centre
- power supply and local distribution network
- communications infrastructure.

Changes to the infrastructure include:

- a haul road from the Mangazeisky North deposit to the Vertikalny complex
- a waste rock dump and temporary stockpile at the Mangazeisky North deposit site
- expansion of the TMF capacity at Vertikalny resulting in a likely footprint increase.

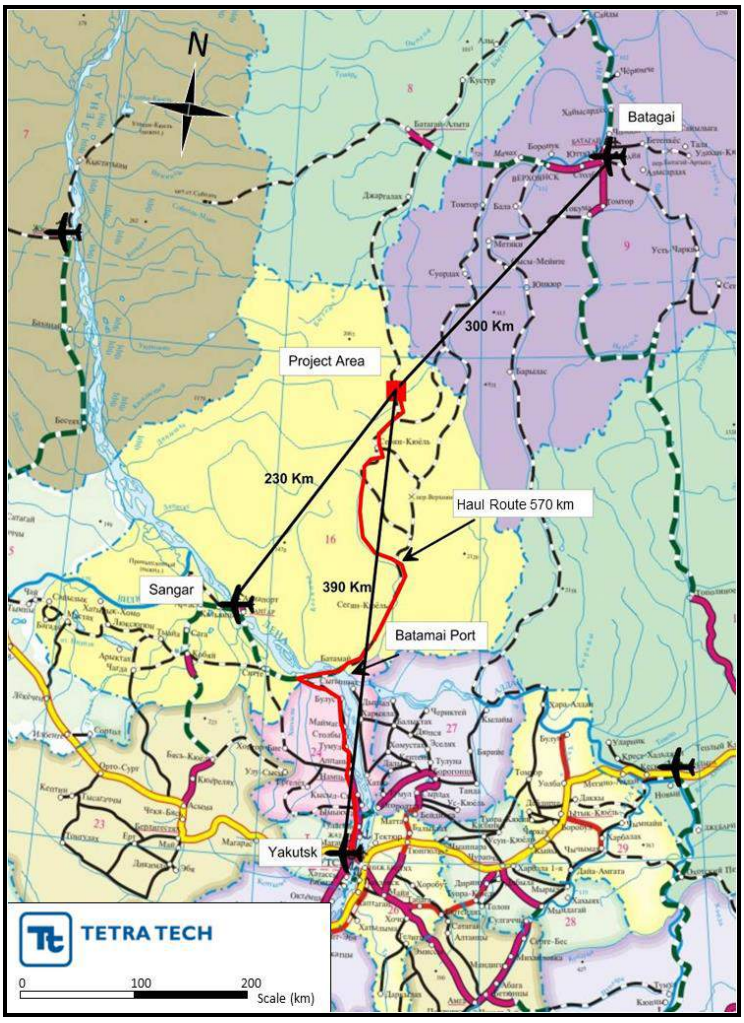
Logistical Infrastructure

The mine site is remote and relies on access by air transport throughout the year and an annual resupply mission during four months of the winter. Supplies and equipment is consolidated in warehouses at Yakutsk and at the regional ports, before being transported to site. Fortunately, Prognoz has refined the process after successfully completing a number of winter road resupply missions; including during construction.

The current haulage route is shown in Figure 24.22. The total distance by road from Yakutsk to the Property is 570 km.

During normal operation, fuel deliveries make up the bulk of the loads delivered to site, with typically just over 5,000 t making its way up the winter road to site in a season (Table 24.26).

Figure 24.22 Transport Routes



Note: Red outline = project area; black lines = existing gravel roads

Table 24.26 Annual Winter Road Transportation Tonnages by Item

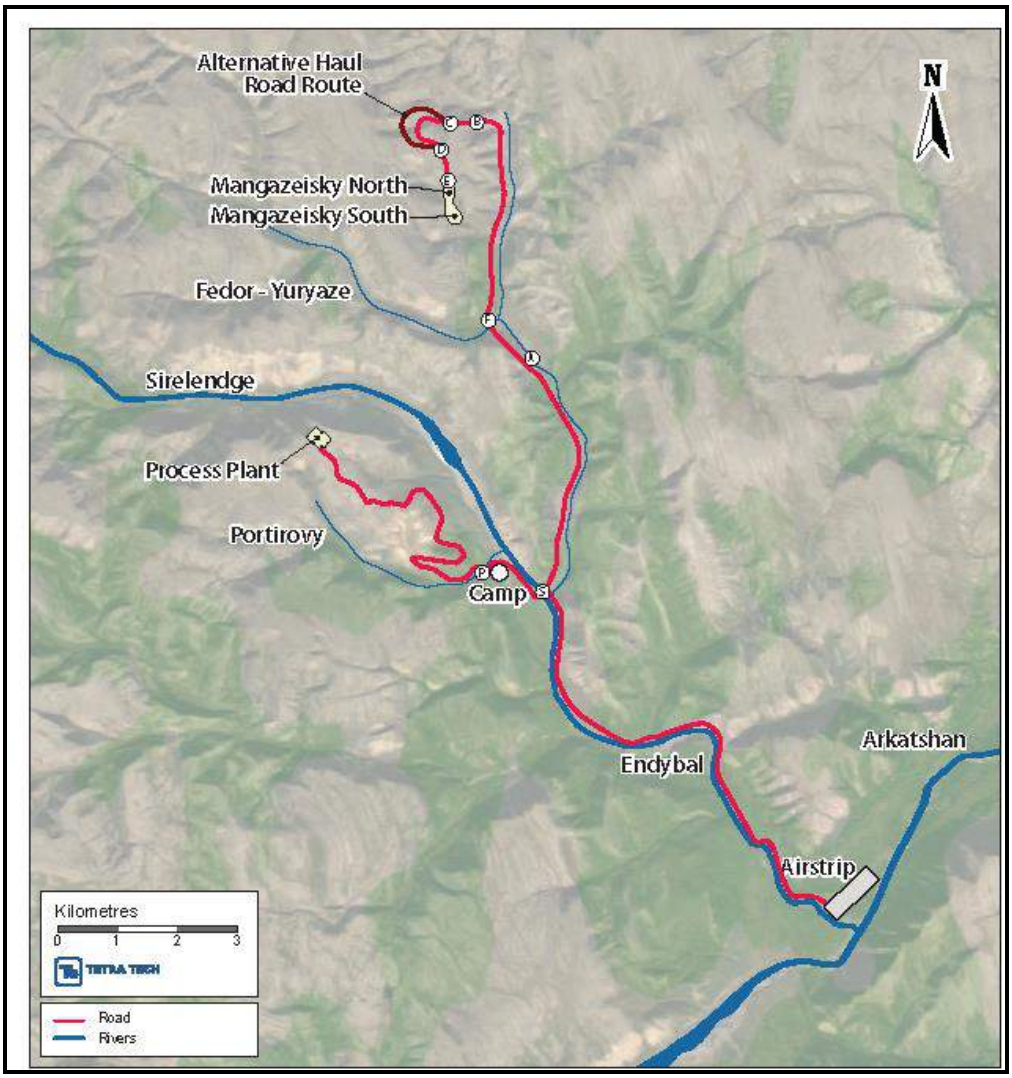
Item	Annual Winter Road Tonnages (t)
Mechanical Equipment	515
Electrical and Control Equipment	435
Piping	8
Valves	15
Total Equipment	**973**
Liners	10
Grinding Balls	100
Lime	100
Sodium Cyanide	500
Flocculent	5
Fuel	5,050
Camp and Exploration Consumables	2,800
Total Consumables	**8,565**
Total	**9,538**

SITE LAYOUT

Figure 24.23 is the Project layout, which shows the Mangazeisky North deposit in relation to the Vertikalny deposit and infrastructure.

Figure 18.4 and Figure 18.5 in Section 18.0 show the infrastructure layout at Vertikalny, the process plant layout, and the expanded TMF.

Figure 24.23 Mangazeisky Project Site Area



SITE ROADS

Haul Road between Mangazeisky North and Camp

Currently, there is an ungraded temporary rough road between the camp site (near the Sirelendge Valley) and the proposed Mangazeisky North deposit. This road can be travelled by all-terrain vehicles only. It will require extensive grading and road fill placement work in order to provide adequate support and trafficability for heavy dump trucks transporting material from Mangazeisky North to the process plant. Fortunately, an unlimited quantity of course-grained granular material is available in the valley floors that can be used for constructing a graded road for dump trucks and

other vehicles. It is anticipated that the existing mobile crusher and screen plant can be moved to the Fedor-Yuryaze Valley, in order to produce graded coarse materials for granular road fills required to provide a rideable surface on the top of the roughly-graded riverbed consisting of cobblestones and boulders.

Road Alignment

The suggested alignment of the haul road between the camp site (near the Sirelendge Valley) and the Mangazeisky North and South pits is shown in Figure 24.23. The total length of the road from the Porfirovy Valley (marked as P in Figure 24.23) to the Mangazeisky North deposit (marked as E in Figure 24.23) is approximately 12 km. The proposed haul road between these two points can be divided into sections.

- Section 1 (between points P and S) will be constructed between the Sirelendge Valley and the existing road leading to the process plant. This road section will be constructed next to the camp site at the junction of the Porfirovy Creek and the Sirelendge River; its length will be approximately 1,050 m.
- Section 2 (between points S and F) will be constructed within the Fedor-Yuryaze Valley. This section of road will be approximately 5.2 km long and the elevation difference between the two points (S and F) is approximately 100 m, representing an average longitudinal road grade of 1.9%.
- Section 3 (between points F and B) will be constructed within a side valley north of the Fedor-Yuryaze River. This section of road will be approximately 3.9 km long. The elevation difference between the two points (F and B) is approximately 120 m, representing an average longitudinal road grade of 3.1%.
- Section 4 (between points B and C) will be within a relatively narrow and shorter side valley. This section of road will be approximately 600 m long and the anticipated road grade is somewhat steeper at 3.4%.
- Section 5 (between points C and D) will be the steepest section of the road. This section will be approximately 1,000 m long and the anticipated road grade is 10.0%.

The road section proposed between points C and D is shown on the map as the broken dark green line (1 km long). The black broken line between the two points shows an alternative, longer alignment that would result in a longitudinal road grade of approximately 7%. As opposed to the easy road construction along the valley floors, construction of the road between points C and D will be somewhat more complicated and will require excavation into the side slopes of the valley. The final alignment will be controlled by the actual geological formation within those side slopes.

Design and Construction Considerations

All tree remains (roots, tree trunks, wood fragments) and any other organically contaminated soils will need to be removed from the areas of the proposed haul road.

The exposed subgrade (heterogeneous mix of sand, gravel, cobblestones and boulders along the rivers and creeks) will need to be graded and proof-rolled either by a heavy smooth roller or by a heavy, crawler mounted backhoe or bulldozer. All man-made granular fill that may potentially be needed to raise the grade in lower areas (to approximately 500 mm above the highest potential river level) will also need to be compacted by a roller or bulldozer. Boulders from the riverbed, up to 600 mm in size, may be incorporated within the road embankment provided that:

- they are placed only in the lower part of the embankment and are embedded in smaller-sized, "transition" rock fill material
- the maximum dimension of the largest rock fragments do not exceed 600 mm (24 inches)
- they are not located within 0.3 m of the final embankment side slopes
- they are not located within 0.7 m of the surface of the road grade.

The finished road surface should be sloped toward the road embankment's side slopes in order to provide efficient surface water drainage for the long term. Since the road will have no pavement, the surface should be sloped 4% toward the road edges in order to promote fast and efficient surface water drainage on the travelled portion of the road.

Once the properly crowned rock fill embankment is completed, two layers of surface course fill should be placed on the top of the coarse rock fill embankment. One of the most important requirements for gravel surfaced roads is the stability of the final surface layer, located immediately beneath the tires of the vehicles. For this purpose, a somewhat greater percentage of sand and silt is generally preferred within the dense-graded surface aggregate, in order to provide at least marginal cohesion within the surface fill. Considering the anticipated dump truck traffic on the haul road, the total thickness of the two surface courses shall be 450 mm.

Road Construction at River Crossings

The Sirelendge and Fedor-Yuryaze rivers meander in their respective valleys; therefore, a number of channel crossings may be required along the proposed haul road. Functionally, these crossings will need to allow for passage of the maximum amount of water that can reasonably be expected to occur within the lifetime of the road, and not degrade water quality or endanger the road itself or any downstream similar structures (if any).

There are three generally accepted methods used to cross flow channels (meandering river beds) on low-volume roads: bridges, fords, and culverts. Factors to consider when selecting the most appropriate crossing type are:

- **Bridges**: High-traffic volume, large and variable water volume, high debris potential in the riverbed, sensitive channel bottom and banks, significant fish resource, large elevation difference between channel and road grade (construction of bridges are not suggested anywhere at the Silver Bear site).
- **Culverts**: Medium to low water volume, medium to low debris potential, fish resource not significant, elevation difference between channel and road

grade less than 10 m, high traffic volume (may not be used at the Silver Bear site).

- **Fords**: Low to intermittent water flow, high debris potential, no significant fish resource, road grade can be brought down to channel bottom, low-traffic volume.

Out of the three alternatives, construction of "fords" are suggested along the proposed haul road in the river and creek valleys. Fords are a convenient way to provide economical, simple waterway crossings in areas subject to seasonal high runoff peaks, like the valleys around the Mangazeisky pit site during the spring/summer snow melting periods. Water and debris will simply flow/wash over the road structure at the crossings. After any water or debris has passed over the crossings, some clearing may be necessary to allow for subsequent vehicle passage.

Access from Airstrip to Camp

The airstrip is located approximately 10 km south of the Property, which includes the mine site and the new camp site (Figure 24.23). The access road to the new camp site runs along the Sirelendge River, which has seasonally higher water levels in the valley, but remains frozen during most of the year. The condition of the route is generally poor during the summer months and will require upgrading in order to provide acceptable driving conditions from the airstrip to the mine.

Large quantities of granular rock fragments including boulders, cobbles, and gravel-sized particles are available along the road alignment and may be used to upgrade the existing un-graded road into a gravel-surfaced access road to the mine.

Presently, vehicles drive along the gravelly, cobble bank of the shallow Sirelendge and cross the river approximately 1.5 km downstream from the new camp site. Although the granular soils along the river banks generally provide good riding conditions for all-terrain vehicles (ATVs), some upgrade (re-grading) of the surface along the road alignment may become necessary. Since the river valley is generally wide, there is no shortage of clean, inert, granular material on the valley floor for such a road upgrade along the entire alignment.

Internal Roads

The internal roads across the mine site will connect the new camp site; the process plant; the open pit; and the stockpile, tailings, waste rock, and other storage facilities. The approximate alignment of the main road is shown inFigure 18.4. The current road alignment between the camp and the process plant area. The first 30% (approximately) of the existing road follows the Porfirovy Creek Valley from the new camp site, before entering a narrow side valley and climbing up toward the mine facilities, located at elevations high above the valley floor of Porfirovy Creek.

The existing road upgrade, and the construction of any new road sections, will include the placement of acceptable granular fill materials over the prepared subgrade and to the specified width and thicknesses. The width of the road will depend on the specifications of the widest vehicle traveling on the road. At site, this is the KRAZ 65032 dump truck.

The current roads on site have been constructed by Prognoz personnel using on-site equipment. The proposed road upgrade and maintenance is not dissimilar to their current practice, and it is reasonable that future mine site road construction can be completed by the Prognoz construction team using mine site equipment, waste rock, and the on-site crusher to produce aggregate.

During periods of warm and dry weather, roadbeds will be regularly watered, as required, to keep dust levels to a minimum.

PROCESSING FACILITY

Process Plant Building

The Mangazeisky North mineralised material will be treated in the Vertikalny process plant. Minor changes to the facility such as cyclone reconfiguration, pump speed adjustment and in some cases minor pipeline replacement is anticipated to accommodate the new production rate and feed.

The main process plant building will be an 84 m by 24 m portal frame structure with an 84 m by 12 m lean-to structure attached to the southeast side of the main building. The building is situated in a northeast to southwest direction to take into account the prevailing north-easterly winds. Both the main portal frame and the lean-to are clad with Russian specified and supplied sandwich panels to provide insulation and weather protection.

The main 24-m-wide structure has an electric overhead travelling crane running the full length of the building, with a 20-t safe working load and a 5-t secondary hoist.

A 150 m^3, steel-constructed, mill feed bin will be attached to the main building at the southwest end of the building to provide buffer storage between the crushing and grinding operations. The bin will be attached to the building and insulated to allow heat from the building to thaw the mineralized material and prevent it from freezing solid within the bin.

The building houses the process equipment downstream of the mill feed bin, reagent mixing, compressor and blower house, tailings handling, small containerised warehousing and workshop, control room, motor control centres (MCCs), and office and change facility.

The office and change facility is a two-storey enclosed structure located in the lean-to section of the main building, between the reagent mixing and electrowinning process areas. It will be a steel structure with concrete block walls and poured concrete flooring. The concrete blocks will be produced locally on site.

Silver is precipitated from the pregnant solution in an electrowinning process to produce a high-grade silver slurry, which will be dried to produce silver powder. As a result, the area that contains this direct electrowinning equipment, and the adjacent silver room, will be secured and access controlled. The silver room will be located on the ground floor of the office structure and will contain the dewatering, drying and storage facilities, as well as an office for the foreman. The security office will control access to the silver room and will overlook the direct electrowinning area. A dedicated door will allow for secure vehicular access to the silver room for secure

loading of the transportation vehicle. The silver powder will be flown to Yakutsk for onward transportation to the refinery once a week; two weeks' worth of storage will be provided to allow for bad weather.

The remainder of the office complex will house process and administration offices on the first floor and change facilities on the ground floor.

On the northwest wall of the process plant, a two-level steel structure will support two MCCs on the first floor, allowing cable access from the ground floor underneath the MCCs, and the control room and laboratory on the second floor, overlooking the grinding and leach sections.

There will be three, 40-foot shipping containers located on the ground floor of the northwest wall of the process plant containing a basic workshop in one, and day to day maintenance stores in the other two.

A roll shutter door in the northern corner of the process plant on the northeast wall will allow vehicular access to the process plant laydown area. The truck bay and laydown area will be serviced by the electric overhead travelling crane. During the colder months, the truck bay and laydown area will be used to park some of the mine fleet in an emergency mine shutdown.

Separate, dedicated vehicular access to the reagent mixing facility will be located in the south corner of the building on the southwest wall. This access will be used to bring the shipping containers used for reagent storage into the reagent mixing and storage area for unpacking as required.

The cyanide mixing and storage room will be an enclosed, separately ventilated room, separated from the reagent mixing area and other plant areas by a concrete block wall. There will be no acid reagents or consumables in the vicinity of this area. Access to the cyanide mixing room will be from outside of the process plant via a dedicated door on the southeast wall to allow the cyanide containers to be brought into the room.

There will be vehicular access to the compressor house from inside the building to allow for maintenance. There is also pedestrian access to the office complex on the southeast wall, and outside to the crusher on the southwest wall, all at ground level.

The building will be heated by a 3.7-MW diesel fuelled boiler, housed in a separate building adjacent to the northwest wall of the process plant building. The boiler will be located next to the process plant power generators to allow for waste heat from the generators to be used, via a heat recovery system, to pre-heat the water prior to entering the boiler.

ROM Pad and Crushing

The ROM pad and crushing facility will be located outside at the southwest end of the process plant. The mine haul trucks will carry ROM material to the ROM pad, which is comprised of three graded piles. The mineralized material will then be fed into the crusher plant by a front-end loader. During periods of dry weather, as required, ROM pads and stockpiles will be sprayed with water to supress dust emissions.

The crusher facility will comprises two skid mounted modular crusher units and a fine ore bin feed conveyor. The skid mounted units will be bolted to a reinforced concrete slab.

The first skid mounted unit will contain a vibrating grizzly, feed hopper, vibrating feeder, jaw crusher, and discharge conveyor. The jaw crusher will operate in an open circuit and the product will be discharged into the second skid mounted modular crusher unit via the discharge conveyor.

The second unit will contain a feed hopper, double deck vibrating screen, cone crusher feed conveyor, cone crusher, cone crusher discharge conveyor, and screen discharge conveyor. The cone crusher and screen will operate in closed circuit with the screen undersize product reporting to the fine ore bin via the fine ore bin feed conveyor. The crushing plant will be able to crush at a faster rate than the process plant requires. A diverter chute will be installed to divert the crushed material when the fine ore bin is full. The crushed material will be stockpiled to give buffer storage and allow continued operation when the crushing plant is down for maintenance.

The crushing facility will be an outside operation. A heated, elevated, containerised operator's hut overlooking the skid mounted units will be located to the east of the crusher units between the crusher and the process plant, with good visibility of the ROM and front-end loader activity.

The stand-alone fine ore bin feed conveyor will be of covered construction to minimise dust.

The Mill

The layout of the plant was designed to keep the building size as small as possible whilst allowing for effective operation and maintenance of the process equipment.

The mineralized material will enter the process plant via a vibrating feeder at the base of the fine ore bin located in the western corner of the process plant, discharging onto the ball mill feed conveyor. The mineralized material will discharge from the ball mill feed conveyor into the ball mill feed chute where it will enter the ball mill alongside the mill classifying cyclone underflow product and process water.

The cyclone tower will be located adjacent to the mill foundation, housing the classification and dewatering cyclones, gravity trash screen, and gravity concentrator. On the ground floor level the associated tanks, pump boxes, and pumps will be installed as well as the inline leach reactor (ILR).

The leach circuit will be made up of six leach tanks. The ball mill will be sized to provide an upside in throughput of 25 to 30% given the current predicted mineralized material characteristics. As each of the six leach tanks represents 17% of the leach capacity, space for two additional leach tanks, one at the start of the leach circuit and one at the end of the circuit, will be required to match the upside in the ball mill. Having eight tanks installed will give a retention time of 120 hours, which will allow both the future fresh sulphide and oxide material to be processed in the same stream. The leach tanks will be 7.7 m in diameter by 7.8 m fabricated steel tanks designed to support the tank top walkways, agitators, gearboxes, and ventilation hoods.

The leach solution will report to the thickener via a clarifying cyclone. The thickener overflow will report to direct electrowinning and the thickener underflow will report to the filter presses for tails dewatering. The liquor from the filter press dewatering will report back to the thickener feed. The thickener will be located at the end of the leach circuit and will sit between the leach circuit, filter press, and electrowinning sections to keep the piping distances to a minimum. The filter cake will be sent to the TMF and the liquor will report back to the thickener.

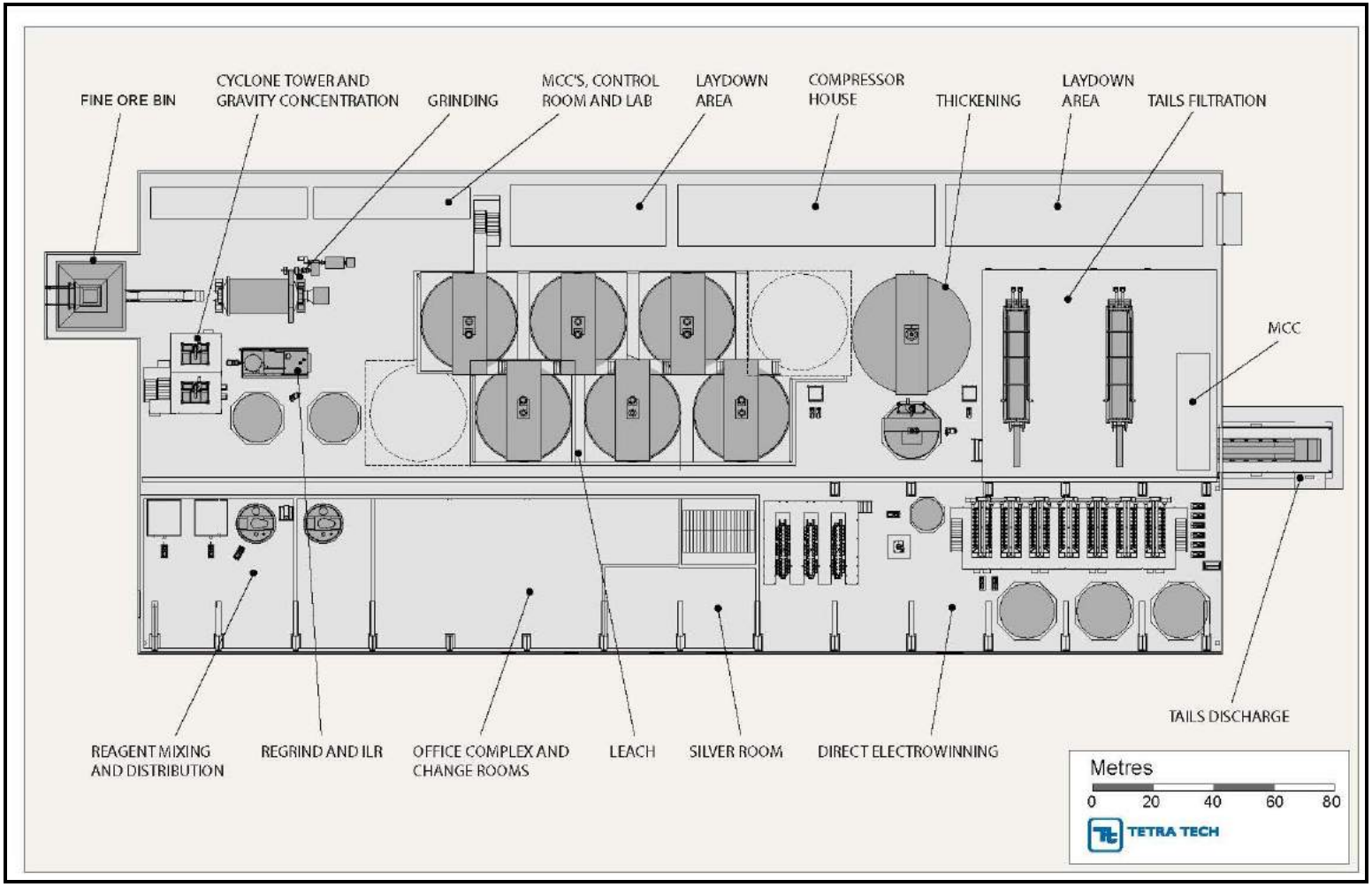
Piping will be designed to meet with GOST 32388-2013 and plastic piping will be designed to meet with GOST 18599-2001.

Tailings Handling

Dewatered tails will be released from each filter press onto a cake discharge conveyor running under each filter press. The cake discharge conveyor will discharge onto a common conveyor running perpendicular to the filter presses, which will transport the cake through a portal on the northeast side of the building to discharge to a stockpile.

After dumping the mineralized material on the ROM pad, every third truck will continue round the process plant building to be loaded with tails from the tails stockpile by a front-end loader. The truck will proceed to the TMF, 250 m to the east of the processing facility, and dump the tails in the TMF before proceeding back to the open pit.

Figure 24.24 Process Plant Layout



Plant Services

A compressor house will be located in the eastern corner of the building. The compressor house will contain instrument air compressors, an air receiver, and the leach blowers. Instrument and plant air will be distributed by a 100-mm diameter carbon steel ring main with 65 mm diameter stringers.

Process water will be pumped from the process water tank via a 100-mm ring main, supported at a high level around the plant, with 65-mm stringers dropping down to equipment and hose points. The return water will be fed back into the process water tank. Approximately 1 m^3/h of depleted electrolyte will be bled from the process for treatment and filtration to prevent the build-up of weak acid dissociable cyanide, base metals, and total dissolved solids in the recirculated process water.

A containerised treatment plant will be constructed on site and will be housed within the process building. The primary step in the process will be the removal of base metals in the form of metal precipitates. The barren solution discharge, approximately 15 m^3/h, will be conditioned with sulphuric acid and sodium hydrosulphide in order to precipitate the copper and zinc ions as sulphide metal complexes. The cyanide ions in the solution will be converted to hydrogen cyanide during this chemical reaction which dissolved instantaneously in the process water, as hydrogen cyanide is infinitely soluble. The slurry containing hydrogen cyanide solution and metal sulphides will be pumped into a conical bottomed, static settling tank and dosed with flocculants. The metal sulphide precipitates in the settling tank underflow will be pumped to the filter press feed tank within the process plant tailings disposal area and subsequently discharged to TMF.

The settling tank overflow will be conditioned in another tank with sodium hydroxide in order to regenerate the cyanide. It is anticipated that 90% of the cyanide solution, approximately 13.5 m^3/h, will be recirculated back to the process plant as plant water, reducing water demand and cyanide usage.

A bleed stream of 10% of the process solution, approximately 1.5 m^3/h, will be pumped through a sand filter followed by a series of carbon adsorption columns and cationic and anionic ion exchange (IX) columns in order to remove any remaining contaminants including residual cyanide. The clean water from the bleed stream will be recirculated back to process water tank, used for reagent make-up or in the gravity circuit. The treatment plant will run year round.

The potable water will be treated raw water. The potable water treatment plant will be a containerised unit, taking feed from the raw water tank, processing the water and feeding the potable water tank. From the potable water tank, potable water will be pumped round a ring main to the plant facilities, wash rooms and process equipment, namely the ACACIA reactor and gravity concentrator, requiring clean water.

The fire water tank will be located in the water treatment complex alongside the raw water tank and potable water tanks. The fire water tank will be fed from the raw water tank with fresh water. The fire water tank is the feed for the fire water system, which consists of a fire water ring main, fire water pump, jockey pump and backup pump. The fire water ring main will run through the garage warehouse building and

process plant building with stringers to provide fire hose points at the required locations.

Process Instrumentation and Controls

An uninterruptible power supply (UPS) will supply power to the control and instrumentation system. The control system is designed for semi-automation of the process plant. The control system architecture design consists of a central PLC and a supervisory control and data acquisition (SCADA) operator interface within the process plant, with distributed input/output (I/O) hardware at the various infrastructure sites

Processing Facility Closure Plan

Upon closure of the mine, a number of operations will be required to make the plant safe. Structures, buildings, and all equipment will remain in place. All tanks, pipes, and vessels containing potentially hazardous slurries and solutions, including, but not limited to, the leach tanks, electrowinning, reagent mixing and distribution, and pump boxes will be cleaned and flushed thereby neutralising the substances. Oils and coolants will be drained from machinery into sealed barrels. The coolants will be stored in the waste storage facility and oils will be burnt in the process plant boiler.

SITE FACILITIES

The following section provides an overview of site buildings and supporting facilities. An overall site plan, including arrangement of buildings on the site and layouts for each of the main buildings is presented inFigure 18.4.

Garage/Warehouse

Located to the east of, and running perpendicular to the main process plant building is the garage workshop building. The building itself will use an existing 40 m by 20 m by 8.4 m high warehouse building available on site, with a new 20 m by 20 m by 8.4 m high garage extension to make a new 60 m by 20 m building which will be clad with Russian specified and supplied sandwich panels to provide insulation and weather protection.

The new 20-m garage extension will house three vehicle bays, each with roller shutter door. The bays will be designated a vehicle wash down bay, a welding bay, and a repair bay with an under vehicle access pit. The welding and repair bays will be serviced by a 5-t gantry crane.

The 40-m workshop section of the building will contain a mezzanine floor for storage of lightweight spares. The ground floor of the workshop section will contain general workshop equipment such as hydraulic jacks, drills, sanding machines, pullers, and compressors. In addition to the general equipment there will be specialist sections for tire change and storage, axle maintenance, engine and gearbox maintenance, battery charging and brake maintenance, as well as a foreman's office and washroom facilities.

During periods of extreme weather when mining operations are shut down, the unused vehicle maintenance bays will be used to park some of the mining fleet to prevent them from getting too cold for restart.

Fire Station

The fire station will be located within the process plant complex. The building will be of steel construction and clad with Russian specified and supplied sandwich panels to provide insulation and weather protection.

The building will contain covered parking for the fire fighting vehicles with water tank refill equipment and roller shutter door, shift office, changing room, training room, equipment stores and a mess room.

Operation and Construction Camp Accommodation

The operational camp facilities will be located in the valley bottom next to the Sirelendge River. The operational camp will consist of two, two storey, 12 m by 49 m, 80 bed accommodation blocks including mess facilities, bath and laundry complex, warehousing for housekeeping maintenance spares, food, and consumable storage. Water will be supplied via a containerised potable water plant, from the raw water pump station. The camp will have its own 645-kW generator and fuel storage, used to provide power to both the camp and the raw water pump station.

The construction camp will be a 50-person, winter proof, containerised temporary camp with mess and laundry facilities, located next to the process plant.

There will be a medical centre located at the operational camp with a doctor on site. Injured and sick workers will be transported back to the camp in site vehicles for treatment.

Airstrip

The current airstrip allows access to site via helicopter or an Antonov AN2 fixed wing aircraft. The runway is approximately 800 m long and can be extended to 1,200 m. As neither of the aircraft mentioned can fly above 3,500 m in altitude, and the site airstrip does not have telemetry, lost flying days are estimated to be between 50 and 75% due to poor weather conditions. To mitigate for the risk of lost flying days, the shift rosters will be staggered and the secure silver store room will have the capacity to hold several week's production of silver powder.

Helicopter Pad

A helicopter landing zone has been designated at the camp area and clinic. Helicopters will be used for medical emergency evacuation.

Site Facilities Closure Plan

Buildings and equipment, including generators, will remain on site. Any oils, hydraulic fluid, coolant, and other hazardous substances will be drained from equipment and stored in sealed barrels. Oils will be used as fuel in the diesel boilers. All other fluids will be stored in the waste storage facility in sealed barrels. Any equipment with stored energy will be dismantled or made safe.

For a period of at least one year, a small team of personnel, accommodated in the local village of Sebyan-Kyel, will monitor the site on a weekly basis and take samples to ensure that the storage facilities and mine closure measures are effective.

REMOTE SITE FACILITIES

Offices

The Project will be remotely supported by management, administration and engineering teams based in an office located in Yakutsk. These functions do not need to be based at site incurring accommodation, travel, and rotational costs and will only fly out to site as required.

Refinery

Dried silver powder in sealed and tagged 30 l high-density polyethylene (HDPE) drums, flown from site, will be transported directly to a third party refinery from where doré metal will be sold directly. Once the silver metal powder arrives at the airstrip near Yakutsk, no handling (other than off-loading from the plane) or storage of silver metal by Prognoz will be required.

TAILINGS MANAGEMENT FACILITY

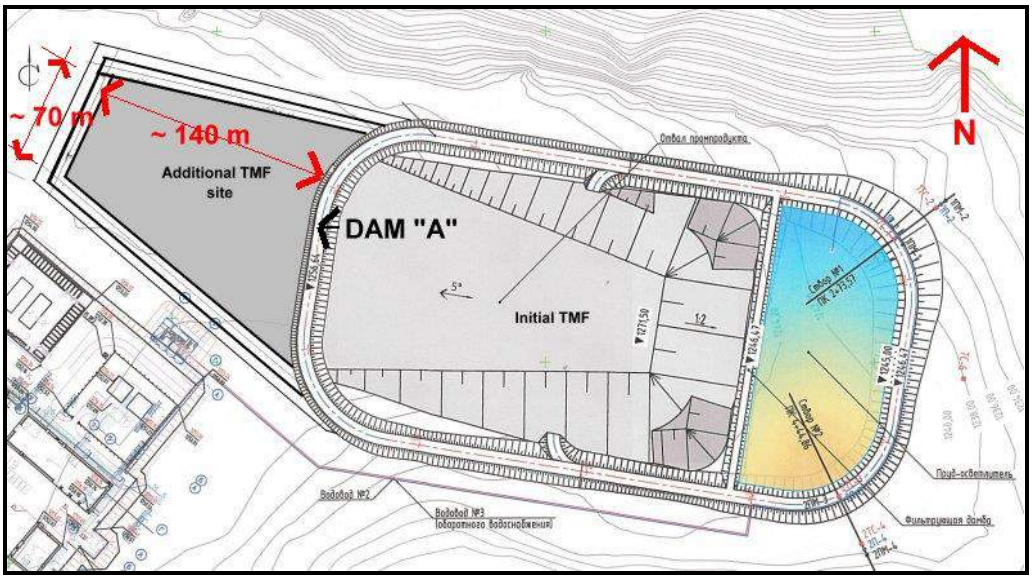
Approximately 386 kt of mineralised material from the Mangazeisky North open pit will be treated by the Vertikalny process plant, most of which will report to the TMF.

General Design Criteria

Based on details presented in the recently completed feasibility study (Tetra Tech 2016c), tailings from the Vertikalny pit (located in close vicinity of the process plant) will be deposited immediately adjacent to the process plant within a large, main tailings facility (Figure 18.4). The approximate footprint of the main tailings disposal area is 180 m x 270 m, while the adjacent water collection pond is 180 m x 90 m. According to the detailed design, the facility will be surrounded with a rock fill berm, and will be lined using a geosynthetic liner, installed between two layers of geotextile fabrics the base of the TMF and the upstream faces of all berms will be lined). According to the design drawings the height of the perimeter berms vary between 4 m and 9 m, and the maximum height of the tailings mass within the main TMF will be around 28 m.

The historic site selection study and the detailed design of the initial (main) tailings facility (Figure 24.25) was completed by SRK Consulting (UK) Ltd., in conjunction with the EMC Design Institute, based in St. Petersburg. The selected main TMF site is located in close vicinity of the process plant, at elevations ranging approximately from 1,238 to 1,248 masl. Overall, the ground surface in the area dips at an inclination of approximately 3% to the east, toward the water collection pond. To the immediate north, south, and east the land slopes away more steeply into the Sirelendge Valley below.

Figure 24.25 Suggested Additional TMF Area



Source: Adapted from Tetra Tech (2016c)

Based on the results of detailed soil explorations, the general ground conditions within and around the TMF site are favourable, with no weak or compressible soil present across the site. Ground conditions is described in terms of three soil and rock strata:

- Layer 1 – immediately beneath the surface there is a thin layer of organic topsoil with negligible thickness
- Layer 2 – weathered rock, in thickness varying between 0.5 m and 1.5 m, generally comprising cobbles with some finer soil infill (gravel, sand and silt)
- Layer 3 – bedrock (jointed and cracked near the surface but transitioning to fresh solid rock at depth).

No 'geological hazards', including sub-surface open cavities (karstic voids etc.), were encountered during the geotechnical investigations undertaken during the design of the Vertikalny TMF and process plant areas and, owing to its proximity, none are expected for the TMF extension. It is understood that the entire area of the mine development is underlain by permafrost and the thickness of the seasonally thawed ground zone is maximum 2 m only. It is also important to understand that the weathered rock, covering the surface throughout the site, is not susceptible to freeze-thaw action, and will remain stable throughout the years.

The tailings will be filtered within the plant at the end of the processing cycle and the dry tailings will be placed and compacted within an additional tailings area, constructed immediately adjacent to the main TMF, as shown on Figure 24.25.

The following assumptions have been made for the disposal of the tailings originating from the Mangazeisky North deposit:

- Similar to the material from the Vertikalny pit, all tailings from the Mangazeisky pits will be filtered at the process plant and transported by trucks to the nearby TMF.

- The deposited “filtered” tailings will have an estimated compacted dry density of 1.9 t/m^3
- The surrounding perimeter dams will be constructed from non-acid generating (NAG) waste rock and locally excavated weathered rock overburden

The conceptual design basis shown in Table 24.27 shall be considered during the detailed design of the additional TMF.

Table 24.27 General TMF Design Criteria

Item	Design Criteria
General	
Site Elevation	The geodetic elevation of the proposed additional TMF area varies between 1,250 and 1,255 masl.
Climate	The region where the TMF will be constructed is characterized by very cold winters and short and cold summers. The average annual temperature is estimated at -12.5°C. The total annual precipitation at the site varies between 203 to 337 mm. The majority of precipitation (60 to 70 per cent) occurs during the summer months.
Road	The proposed additional TMF will be constructed immediately adjacent to the process plant, where an already constructed haul road is available to facilitate the transportation of tailings from the plant to the TMF.
Water Management	
Site Water Management During Operation	All surface run-off water (assumed to be negligible in quantity) shall be collected in diversion ditches around the TMF and discharged to the nearby Sirelendge Valley All precipitation that may come into contact with the TMF shall be collected in the clarification pond and recycled to the process plant
Groundwater Flow Regime	The site is located in the continuous permafrost zone, on the top of a mountain, approximately 400 m above the floor elevation of the adjacent valley, and no groundwater flow is anticipated anywhere in the vicinity of the TMF area. It is evident, based on field observations that minor subsurface water seepage during the short thaw period may occur within the shallow (thin) weathered rock zone, above the solid bedrock mass; however, such seepage will be minimal and will have no impact on the stability and integrity of the proposed TMF.
TMF Design Criteria	
Total Tailings Production	386,728 t
Tailings Composition	Poorly graded silty sand (Percentages by mass: up to 30% fines and 70% sand - NSI Lab Testing Data)
Total Tailings Volume	Filtered, dry stacked tailings: 390,000 t → ~206,000 m^3 at 1.9 t/m^3 compacted dry density
Tailings Dam Dimensions and Criteria	3H : 1V upstream slope to facilitate liner installation 2.5H : 1V downstream to facilitate reclamation Maximum height of tailings within the TMF ~20 m
Seepage Control Measures	Fully lined dam (upstream face) and fully lined TMF area A minimum of 2% top surface grade to promote surface drainage after closure

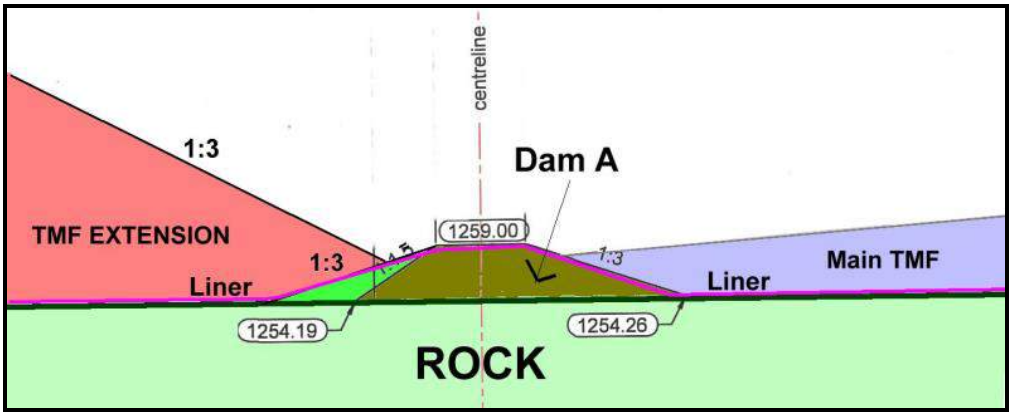
table continues...

Item	Design Criteria
Embankment Fill Material	Non-acid rock drainage (ARD) (NAG) mine waste rock and weathered rock overburden
Geological and Geotechnical Conditions	Thin, weathered rock overburden over solid bedrock. The weathered rock will be removed and used for berm construction.
Closure Requirements	
General	The TMF will be graded and capped at closure

As shown on Figure 24.25, the additional TMF will be the continuation of the main tailings facility. The berm between the two facilities, marked as “dam A” on Figure 24.26, will initially be constructed with an upstream slope of 1V:3H and a downstream slope of 1V:1.5H. The main tailings facility liner will cover that berm’s upstream face only. The TMF extension perimeter dams will be constructed from waste rock that have been shown to be benign; most probably sourced from the Vertikalny open pit.

The first step during the construction of the additional facility, must be the placement of additional rock fill to the downstream face of the berm (dam A), as shown on Figure 24.25 (light green zone). Once the berm’s new profile had been completed with 1V:3H slopes, the liner at the bottom of the new facility must be connected (jointed) to the main facility’s liner over the slopes and crest of the dividing berm. With this alternative, additional tailings (above the presently anticipated quantity) may also be placed over this berm in the future, and the two tailings mass may be combined into one continuous disposal facility.

Figure 24.26 Detailed View of Suggested Additional TMF Area



Waste Rock Dump, Stockpile and Overburden Storage Area

The three types of soil and rock streams that will require temporary storage or permanent disposal at selected locations across the mine site are: topsoil (generally weathered rock fragments), mineralized material, and waste rock. The selected locations for the temporary storage or permanent disposal of these materials are shown inFigure 18.4.

The selection of appropriate locations for the storage of these materials is one of the most important aspects of the mine layout design. The selection process is largely influenced by:

- environmental constraints/impacts
- material characteristics
- design volume of the mineralized material/rock streams
- storage capacity
- site availability
- capital, operating, and closure costs
- geotechnical, geological, hydrological and hydrogeological conditions
- ease of the day-to-day operation of mining and processing.

Other site selection considerations include:

- topographic boundaries
- location of ore bodies (open pit/underground shafts)
- surface water management and diversion
- available area and expansion potential
- proximity to existing infrastructure elements
- distance and elevation from the open pit and from the processing plant
- seepage control
- climate
- suitability of seepage and runoff collection pond(s)
- seismic conditions
- sub-grade soil and rock conditions.

Stockpile

Mineralized material from the Mangazeisky North open pit will be transported to the process plant by dump trucks. Prior to crushing, the mineralized material will be stockpiled in an area located immediately southwest of the plant. The stockpile site is generally flat and has firm ground, and comprises weathered rock fragments over shallow, solid bedrock. No stockpile instability is anticipated, as long as the side slopes of the individual piles are 1.5H:1V, or shallower.

The stockpiles will be located next to the process plant, on the top of the plateau; therefore, no runoff water is expected to come into contact with the dumps, and only a limited amount of precipitation will enter the voids of the mineralized material, which will eventually end up in the process plant. As a mitigating measure, contact water around the dumps will be collected in ditches and re-used in the process plant via the TMF. Such ditches may be formed around the individual stockpile heaps. Based on the available weather data, the anticipated amount of such water will

practically be negligible. Any water collected will be transported to the TMF via mobile water bowser.

Overburden Stockpile

Selected areas on the Mangazeisky North site, impacted by mine development, will require clearing of any soil overburden. Such overburden generally comprises layers or mixes of clay, silt, sand, gravel, cobble, and boulder sized rock fragments and soil particles. At the Project site, the overburden generally comprises gravel and cobble sized rock fragments, with limited amount of fine (silt and clay) particles.

Waste Rock Dump

Waste rock from the Mangazeisky North mine will be stockpiled adjacent to the open pit. The lithology is similar to the Vertikalny rock and therefore considered to be benign. Tetra Tech recommends that this is confirmed through geochemical laboratory testing.

Waste Rock Dump, Stockpile and Overburden Stockpile Closure Plan

The stockpile will be depleted during the ramp down of operations with any remaining mineralized material run through the process plant.

During normal operation there will be spare fleet hours for the dozers because during winter mobile equipment will be kept running to stop it from freezing. The dozers will be used for progressive closure of the waste dumps during the ramp down of mining operations. The waste rock will remain in place and vegetation will be planted on the top and slopes of the waste rock dump.

The material contained in the overburden storage area will be used in the remediation of the site on mine closure in areas such as the TMF.

Waste and Sewage

Regulations state that waste on site is not allowed to be disposed of by burning. Solid waste will be stored in the solid waste storage facility, located to the west of the processing facility, between the reagent storage and ROM pad. The solid waste storage facility will be lined and segregated to handle different waste streams.

Site Storage

Reagent Storage and Handling

Due to the accessibility to site and the constraints of the winter road only accessible for a short season, reagents will be stored in bulk with a minimum capacity of a nine-month supply.

Reagents will be transported to site in shipping containers and stored in a dedicated fenced secure storage area. The containers will sit on a lined pad that will contain drainage channels and sumps to catch surface runoff, which will be sent for treatment.

Explosives Storage and Handling

An explosives plant and storage facility will need to be constructed on site to meet the annual explosives and initiation product requirements. The explosives plant will be constructed from the specification and supervision of the explosives supplier. The location of the plant is south of the camp, down the Sirelendge River valley, an appropriate distance to maintain the recommended safety zone from the camp and occupied buildings. The plant is expected to have separate bunkered magazines for containerised storage of ammonium nitrate and initiation devices, office, workshop, and change room facilities.

The ammonium nitrate bags will be moved about the facility using a telehandler. The mobile manufacturing unit will be filled by driving onto a concrete apron adjacent to the store and the bags will be lifted over the truck and the ammonium nitrate discharged from the bottom of the bag. All waste will be adequately stored and disposed of in an appropriate manner in the explosives waste facility located further to the south down the valley.

The bunkered magazines will be placed a safe distance from the rest of the plant and from one another to ensure a safe and secure location for the high explosive and the initiation devices. The dimensions of the earth walls around the magazines will be provided by the explosives supplier; these are to contain any uncontrolled initiation and to minimise danger to plant personnel and infrastructure.

The explosives plant will be contained within a high-security fence. The access will via boom gate at the entrance which will be manned by security guards. Only limited personnel will be authorised to enter the facility and the materials made available will be controlled by the permanent security personnel. The components of the plant will be connected by a maintained gravel road.

Fuel Storage and Handling

Diesel and lubricants will be transported from a storage facility in Batamai during the winter road season. For construction and start-up, fuel bladder storage will be used. Each bladder can contain 250 m^3 of diesel. Permanent 12 m diameter by 12 m steel storage tanks will be constructed in the fuel farm at the process facility to store 6,000 m^3 of diesel, with bladder storage available for the remaining 400 m^3 of the nine-month supply volume.

Almost all of the diesel supply will be consumed by the process power station.

Mining equipment at Mangazeisky will be refuelled using a refuelling truck transporting diesel from the Vertikalny fuel farm.

Site Storage Closure Plan

The Mangazeisky North open pit phase will be followed by the Vertikalny underground phase. Closure planning is limited to the rehabilitation of the waste rock dumps adjacent to the open pit using mine earth moving equipment.

SITE SERVICES

Plant Site Surface Water Management

The anticipated amount of precipitation is negligible, and no runoff water will be flowing toward the plant area. During the very short thaw period, the negligible amount of precipitation within the small plant site will be collected in a perimeter ditch, excavated around the structures, and the collected water be conveyed to the nearby TMF.

Raw Water Supply

A raw water pump station, located in the Sirelendge River valley next to the camp, will provide the site raw water requirements from a groundwater borehole. A borehole pump located inside the station will pump ground water into a 25 m^3, heated and insulated tank located adjacent to the station structure. A horizontal multiple-stage centrifugal pump, sized for 45 m^3/h at 540 m of head, will pump water from the pump station tank to the raw water tank located at the processing facility.

The insulated pipe line between the pump station and the mill will be routed up the Porfirovy Valley alongside the access road before leaving the road to head north up the valley. The pipe line will meet up with the access road adjacent to the WRD and continue across the top of the hill to the process plant. The overall length of the pipe line will be approximately 4.5 km, and an elevation rise of approximately 400 m.

Process Water Tank

The process water tank will be located next to the cyclone tower and ball mill within the process plant. The location of the process water tank was chosen because of the proximity to the processes that require the largest quantity of water addition.

The process water tank will be topped up with fresh feed from the raw water tank and process return feed as barren solution from electrowinning. The process water tank is a 4 m diameter by 4 m fabricated steel tank with sloped tank bottom to allow for periodic cleaning of the settled solids, which will occur small quantities due to the nature of the recycled process water.

Potable Water Treatment

Raw water will be processed through a containerised potable water treatment plant to provide potable water. Two treatment plants will be installed on site, one located at the processing facility adjacent to the raw water tank, and one located at the camp to provide potable water for the camp.

The exact specification of the treatment required will be determined by the results of the groundwater water tests, but is likely to contain filtration, softening, and demineralization processes.

Assay and Metallurgical Laboratories

A metallurgical and a geological laboratory will be available on site.

The metallurgical laboratory will be a containerised laboratory located within the process plant and will handle approximately 10 samples per day. The metallurgical laboratory will include the following equipment:

- laboratory jaw crusher
- laboratory roll crusher
- riffles
- samplers
- sieve shaker
- laboratory ball mill
- bench flotation cell
- bottle roll equipment
- titration equipment
- laboratory pressure filter.

The geological laboratory will be located within the camp area, within close proximity to the core logging building. The geological laboratory will process the wet chemical assays from the mine, process plant, and exploration. Tetra Tech estimates that 12 grade control geological samples will be assayed daily.

Communications

The site wide data and voice telecommunications for the Project will be provided by a Russian cellular network provider. Silver Bear will install a mast on the hill to the south of the main pit and to this, the cell phone company will install their equipment and infrastructure.

The site local area network (LAN) will connect to the cellular network for datalink to the World Wide Web.

The communications systems for the Project will include:

- telephone system
- two separate LAN networks and datalink to the world wide web
- television surveillance system
- loudspeaker and alarm communications systems
- telecommunications cabling to all buildings

The LAN network is to include all cabling, equipment and infrastructure for connection to each building within the process plant area and to remote sites.

Cellular mobile telephones and communication equipment will provide the functions usually covered by site very-high frequency (VHF) radio. The cellular mast and transmission equipment will be installed and maintained by the cellular network provider.

Site Security

Pedestrian and vehicular access to the site will be controlled to provide security for personnel and property and to manage the risk of injury at the mining and processing operations.

The remote location of the site, and the site topography, eliminate the need for full perimeter fencing; however, high risk areas such as the explosives facility, reagent and cyanide storage and the power station will need to be locally fenced.

Whilst the majority of personnel on site will be operational staff, consideration must be given to the seasonal nomadic reindeer herders that pass through the area.

Operational Personnel Transport

Site personnel will be transported around site, between the camp and process facility, and between the camp and airstrip by two rough terrain personnel carriers, each with a capacity of 10 people excluding the driver.

During each shift change-over, 50 to 55 people will have to be transported to the facilities on the top of the hill which will require 3 to 4 round trips per shift. Mining, processing and engineering services shift change overs will be staggered by 30 minutes to accommodate the 30 minute round trip from the camp to the top of the hill.

Power Supply and Distribution

No additional power will be required over and above what is installed for the Vertikalny operation, to mine the Mangazeisky North deposit. The open pit will be mined using diesel powered equipment.

24.4.5 Environmental Studies, Permitting, and Social or Community Impact

Introduction

The Mangazeisky North deposit is 12 km from the Vertikalny mine site and currently accessible by all-terrain vehicles following river banks and temporary exploration access roads. As a result the environmental and social impact assessment and recommendations are based on the extensive work completed as part of the Vertikalny environmental and social impact assessment which is summarized in Section 20.0. A full list of environmental and social references is provided in Section 27.0.

The Project is likely to use the same transport options as Vertikalny. A transport management plan will be required to address transport safety and security aspects, as well as climatic conditions. Delivery of consumables and equipment to the site is expected to be via the winter road, which is usually open from December to April.

Extreme winter conditions prevail for approximately nine months of the year making the Mangazeisky North deposit a climate-sensitive operation.

The Mangazeisky North deposit is located in a non-industrial air shed; therefore, air quality conditions can be regarded as ambient with low, natural levels of gases from

industrial sources, transportation, etc. No significant issues in relation to air quality have been identified. Site operations will be subject to standard dust control procedures during dry conditions.

The Mangazeisky North deposit lies in a mountainous region, with elevations ranging from 800 to 1,400 masl. The main ridges have steep slopes (25 to 30°) and rounded crests that are 200 to 500 m above the valley floors. River valleys are steeply-incised with fast flowing streams, characterised by gravel and boulder beds under non-frozen conditions.

The Mangazeisky North deposit falls within the current exploration territory, which gives Silver Bear authority to conduct exploration activities and to use the subsoil for geological investigation. Flora and fauna surveys and related information are provided in detail in the ERM reports (ERM 2016) and summarised in Section 20.0.

Key Findings and Key Environmental Issues

The Mangazeisky North area is in a remote location with an extreme continental climate and steep terrain with incised valleys and rounded hills. The site is accessible via a winter road (December to April) and otherwise by air transport. Surface water sources are present through the river network; however, most surface water is tied up in permafrost. No areas with special conservation status have been identified within the exploration license area.

The Mangazeisky North deposit is part of the exploration license; however, no permit for surface operations has been obtained.

Expected key environmental issues and risks for development of the Mangazeisky North deposit include:

- coordination with and approval by YAKUTNEDRA
- reclamation of land disturbed during exploratory work, restoring it to a state suitable for further use and in compliance with landscape and recreational specifics of the territory
- payment of compensation for damages caused to hunting, agriculture, fisheries, and forestry
- source and sustainability of water supply for the Project given that the majority of fresh water is tied up in permafrost and observance of water protection zones (in accordance with the exploration permit)
- management of tailings and design of the TMF to take into account extreme climate conditions, remoteness of the site, design of solids (%), etc. to minimise environmental impact
- confirmation of any protected areas (local, regional and national) and their designation/protection status
- management of hazardous materials including transportation to site and storage (with respect to climatic conditions) using the winter road

- management of benign and hazardous wastes and potential reuse of inert materials for subsoil stabilisation, TMF embankments, levelling operations, etc.
  - noise and vibration are not expected to be significant issues given the remote location of the site
  - air quality (dust and gaseous emissions) are not considered to be significant given the non-industrial airshed and lack of point sources.

Waste, Tailings and Water Management

Mine Waste Management

The three types of soil and rock streams that will require temporary storage or permanent disposal at selected locations across the mine site are expected to be: topsoil (generally weathered rock fragments), mineralized material, and waste rock. The specific locations for the temporary storage or permanent disposal of these materials will need to be established through further detailed design. Locations and the selection of the preferred final designs will be influenced by:

- environmental constraints/impacts
- material characteristics
- design volume of the mineralized material/rock streams
- storage capacity
- site availability
- capital, operating, and closure costs
- geotechnical, geological, hydrological and hydrogeological conditions
- ease of the day-to-day operation of mining and processing.

Other site selection considerations include:

- topographic boundaries and restrictions due to the varied and steep topography
- location of ore bodies (open pit/underground shafts)
- surface water management and diversion
- available area and expansion potential
- proximity to existing infrastructure elements
- distance and elevation from the open pit and from the processing plant
- seepage control
- climate
- suitability of seepage and runoff collection pond(s)
- seismic conditions
- sub-grade soil and rock conditions.

Tailings Management

Tailings design will require:

- evaluation of alternatives for TMF location
- finalisation of the design of the TMF including material compaction characteristics to determine the optimal deposition method
- finalisation of geotechnical engineering, lining design, and slope stability assessment
- establishment of a piezometer network to establish groundwater characteristics for the water baseline
- clarification of environmental impacts

If the tailings material is identified as having the potential for leaching, the TMF will need to be designed as a fully-lined facility in accordance with best international practice. The design will also need to identify controls for seepage water and discharge design (unless operating as a zero discharge facility). The solid contents (%) of the tailings material will need to be confirmed to minimise environmental impact, take account of meteorological conditions including the potential for tailings to become airborne in periods of high wind and low humidity.

Health, Safety and Environmental Management and Monitoring

The Mangazeisky North deposit will require a full Health, Safety and Environment (HSE) management plan to include policies and procedures, organisational responsibilities, monitoring, and reporting lines. A fully-integrated program will be required to support the objective of a "zero-harm" goal for employees, contractors and visitors working on the Project as well as achieving environmental protection objectives.

The Mangazeisky North deposit will need to incorporate HSSE as key criteria in the design, constructability, and operability of each facility and major area with the requirement that all personnel must have completed site safety and environmental orientation. Tools will be implemented for performance tracking and accountability, including procedures for incident management.

Construction contractors will need to be capable of meeting performance objectives with respect to HSE and security. Hence, additional processes, policies, and procedures will be reviewed and approved in collaboration with the selected contractors to ensure compliance, including:

- pre-tender site safety audit checklists
- preventive maintenance procedures and inspections of tools and equipment
- documented plans for transporting heavy construction components and supplies
- pre-mobilization meeting with the Owner's team and contractors
- hazard identification and risk assessment workshops
- project safety management plan audits

- site safety and environmental orientation.

Further, a project health and safety management, environmental, communications, and security plan will need to be developed by the Project team and the construction contractors. This document will define the systems, procedures, and reporting frequency to be implemented and maintained throughout the Project life. These plans will be controlled and will act as live documents that will be actively used during execution of the Project. Emergency Response Plans (ERPs) will be integral elements of these plans to ensure appropriate actions and communications during non-routine events.

Water Management

Surface Water/Hydrology

The main water course is the Sirelendge River, which flows in a northwest to southeast direction into the Arkachan River. Two local water courses flow into Sirelendge River: the Porfirovy and Borisovsky streams. The entire hydrographical network connects to the basin of the Yana River, which flows north into the Laptev Sea. Surface water flow in the area ceases during the winter season due to the freezing temperatures, which cause the formation of ice along the river and stream beds.

Hydrogeology

In January 2016, SRK studied hydrogeological conditions, including the study of potential water flows into the pit and water supply availability from the permafrost for the Project. The study identified pump capacities for the open pit to manage surface water runoff at approximately 100 m^3/h up to 200 m^3/h with the 100-year return period storm event.

Pump tests along a borehole network were established along the Sirelendge River to establish flow rates and supply options with alternative scenarios utilising boreholes (groundwater) at the mine site. A number of scenarios were examined to account for a surface water contribution when the Sirelendge is frozen.

The study concluded that sufficient water would be available factoring in climate periods when the river is frozen. No longitudinal tests have been conducted.

Pit water will require pumping with additional standby pumps to cope with the 100-year return storm event and to evacuate water inflow within 24 hours. Water is available through the combination of boreholes and surface water depending on climatic conditions.

For the Mangazeisky North deposit, further studies will be needed to examine:

- number and location of boreholes for surface and groundwater supply
- longer-term pump and piezometer tests to monitor water levels
- design of pumps and standby equipment for pit water control
- site water management including storm water control and protection of soil/vegetation/biological resources from discharge water

PROJECT PERMITTING REQUIREMENTS

Mine Licenses

The Mangazeisky North deposit is situated in the central west portion of the Endybal Exploration Licence area (also known as the Mangazeisky Exploration License), number YaKU 12692 BP. The exploration licence covers a 35 km mineralised structural corridor approximately 400 km north of Yakutsk.

The exploration licence YaKU 12692 BP was granted to Prognoz on 24th September 2004 by ROSNEDRA, and was valid for an initial term of five years. On 18th February 2009, the exploration licence was extended by ROSNEDRA to 31st December 2011. Another extension was subsequently granted for one year until 31st December 2012. On 27th December 2012, a third extension was granted for four years until 31st December 2016. In September 2016, the exploration licence was extended to cover 556.45 km^2 to include a number of additional target deposits, including the Mangazeisky North deposit. A mining licence has been obtained for Vertikalny, which is valid through to 1st September 2033.

The Mangazeisky North deposit was first explored in 2013 with 11,200 m^3 of trenching. Mangazeisky North falls within the exploration licence; however, no surface mining licence has been granted as yet.

Whilst Silver Bear has access to exploration rights and hence the subsoil, there is no evidence of applicable surface rights. Any surface construction, for example for the installation of infrastructure, will require permit verification.

The validity date of the YAKU 12692 BP licence will need to be verified for Mangazeisky North. The approval to carry out surface construction will need to be verified and a permit in place. Any surface construction is at risk until these permits are in place—permits need to be negotiated with the regulators at an early stage in the process.

Environmental Permitting and Standards

The execution of the Mangazeisky North, and the associated permit and license applications required, will be subject to Russian federal and regional regulatory frameworks.

These frameworks include Gosudarstvennyy Standart (GOST), ROSNEDRA, and the Russian Construction Codes and Regulations (SNiP). Part of the application and approval process includes the submission of formal studies including TEO (feasibility study), Proekt (design documentation), and OVOS (environmental impact assessment). The TEO contains information on geology, mining, milling, tailings storage, infrastructure, civil defense measures, and environmental protection. Additional permits will be required for the air and water usage, earth works, site preparation, mill foundation, airstrip, explosive storage and usage, site roads and fuel tank construction, etc. Mangazeisky North is located within the Russian Federation; therefore, any transboundary or other trade restrictions and embargos will need to be addressed.

In addition, Silver Bear, as a TSX-listed public company, subscribes to the best practice international standards of:

- Equator Principles III risk management framework for determining, assessing and managing environmental and social risk in projects
- International Finance Corporation (IFC) Performance Standards on Environmental and Social Sustainability (IFC- PS 2012)
- International Labour Organization (ILO) Core Conventions
- International Cyanide Management Code
- World Bank Environmental, Health and Safety (EHS) Guidelines for Mining
- Voluntary Principles on Security and Human Rights

Silver Bear will need to demonstrate compliance with the Russian OVOS (environmental impact assessment) standard together with the relevant state permits and international standards applicable to open pit mining. The Proekt and OVOS will need to be prepared, submitted and approved by the Russian regulatory authorities. A schedule of state permits will be required with the reference number and validity dates.

Permitting Process

The simplified permitting process for mining Projects in Russia includes the following five stages:

- Obtain mining licence: Mining licenses are granted by the State Authority and outline the resources (quality and quantity of metals to be extracted) and specific conditions for exploration and development. A mining license holder is obliged to follow all the requirements specified in the license
- Confirm resources: Once the mining license is granted, exploration surveys and preliminary engineering studies must be conducted, as specified by the license. The outcomes of these surveys and studies are used to produce a TEO Konditsy Report (see above), which is a technical-economical study that specifies conditions for development of the deposit. The TEO needs to include the amount of resources to be extracted, required infrastructure, economic feasibility of the project, and the potential market overview. The TEO is subject to approval by the Russian Central Commission for Reserves, which allows the license holder to initiate the development of the technical design documentation
- Conduct surveys and studies: These are required to characterise the receiving environment and include geotechnical surveys, hydrology, hydrogeology and meteorological engineering surveys, geological engineering surveys (including surface and ground water surveys for water supply sources), environmental engineering surveys.
- Develop and receive approval of project design documentation: Includes the detailed design and applicable EIAs for the main components of the Project; only when a positive acceptance has been given from the State Environmental Review should the construction and other operational permits be obtained
- Obtain permits: at the State level, permits will be required as follows:

- Subsoil licence (geological study, exploration and recovery of underground fresh water for domestic use and technical water supply)
- Licence for hazardous wastes management
- Permit for emissions of hazardous (pollutant) substances to atmospheric air (except radioactive pollutants)
- Specific water use agreements for the use of surface or groundwater for the project depending on the source
- Approval of waste generation norms and waste disposal limits
- ESIA.

Silver Bear will be required to follow this five-stage process for the Mangazeisky North deposit and provide the correct level of design detail to the satisfaction of the Russian environmental regulators. A full schedule of the type and status of permits is recommended for Mangazeisky North.

Social Setting and Key Issues

The information provided in this section is based on ERM (2015) and other studies prepared by ERM (see Section 20.0)

Mine Closure

Remediation and Reclamation Requirements

Mine closure planning will need to take into account the following:

- remediation and reclamation of mine workings including pits, waste dumps, and TMF
- decommissioning and disposal of physical plant, housing, offices, and vehicles/plant, etc.
- re-vegetation of slopes (as far as is practically possible) with native plant species
- planning for any social closure impacts and retrenchment of workers as required
- payment of outstanding closure costs.

It is expected that the closure strategy for the Mangazeisky North deposit will tie into the plan for the Vertikalny deposit. Buildings and equipment, including generators, will remain on site. Any oils, hydraulic fluid, coolant, and other hazardous substances will be drained from equipment and stored in sealed barrels. Oils will be used as fuel in the diesel boilers. All other fluids will be stored in the waste storage facility in sealed barrels. Any equipment with stored energy will be dismantled or made safe. For a period of at least one year, a small team of personnel, accommodated in the local village of Sebyan-Kyel, will monitor the site on a weekly basis and take samples to ensure that the storage facilities and mine closure measures are effective. More detailed closure plans will need to be developed for the TMF, waste rock dumps, and soil storage areas.

24.4.6 CAPITAL AND OPERATING COST ESTIMATES

CAPITAL COSTS

Summary

The total estimated initial capital cost for the design, construction, installation, and commissioning of all facilities and equipment is $1.5 million. A summary breakdown of the capital cost is provided in Table 24.28.

Table 24.28 Capital Cost Summary

Area	Initial ($)	Sustaining ($)	Total ($)
Infrastructure	798,000	-	798,000
TMF	-	210,000	210,000
EPCM	120,000	-	120,000
Owner's Cost	250,000	-	250,000
Allowances	350,400	90,000	440,400
Closure Cost	-	330,000	330,000
Total	**1,518,400**	**630,000**	**2,148,400**

Note: EPCM – engineering, procurement and construction management

This estimate includes direct field costs required to execute the Project, plus indirect costs associated with design, construction and commissioning. This estimate is based on pricing as of Q1 2017, with no allowances for inflation or escalation. Elements of actual expenditures have been included, which further reduce the exposure to inflation or escalation. All currency in this capital cost estimate is expressed in US dollars, unless otherwise noted.

Basis of Estimate

Tetra Tech prepared a Class 5 capital cost estimate in accordance with the AACE International (AACE) Cost Estimate Classification System. The estimate accuracy interval is -40% to +40%. The estimate base date is Q4 2016.

The capital cost estimate is expressed in US dollars using an exchange rate of RUB66 to the US dollar throughout. Duties and taxes are included in this estimate as part of the project indirect costs.

The International System of Units (SI) measurement system is the standard measurements system used in this estimate.

Tetra Tech calculated unit rates for road construction based on costs calculated as part of the Vertikalny Feasibility Study (Tetra Tech 2016c) and applied these to the proposed haul road route. An allowance was made for earthworks along the route based on the QPs observations during his site visit.

Tetra Tech calculated the cost of additional tailings storage capacity based on the method and rates used in the Vertikalny Feasibility Study (Tetra Tech 2016c).

The activities to perform the majority of the mine closure will use personnel and machinery already accounted for in the operating and capital costs. Earth moving

and fabrication of physical barriers, making machinery safe, emptying vessels and moving waste materials to the waste dump will be systematically performed as final production ramps down at the end of mine life. Therefore the only Mangazeisky North closure cost allowance relates to the TMF.

Direct Costs

Mining

Mangazeisky North will use the Vertikalny mobile fleet and as such no allowance is made for capital expenditure.

Processing

Minor modification to the process plant configuration to accommodate the Mangazeisky North mineralised material and production rate will be made from the operating and maintenance budget.

Haul Road

The unit cost estimate of the haul road from the Mangazeisky North deposit to the Vertikalny camp is shown in Table 24.29. The road will be 9.8 m wide and the surface 0.45 m deep. Earthworks rates are assumed to be $11.00/m^3$.

Table 24.29 Haul Road Capital Cost

Area	Cost ($/km)
Survey Crew to Stake Out Road Alignment	400
Earthworks (cutting and filling) to Prepare Road Subgrade	4,850
Production and Placement of Surface Aggregates	48,500
Finish Grading	300
Drainage Work (ditches and river crossings)	1,400
Management and Supervision	8,950
Maintenance and Spares	2,100
Total	**66,500**

Tetra Tech estimated that the 12 km haul road will cost $798,000.

Waste Rock Dumps, Stockpiles, and Flood Protection Berms

A source of benign waste rock, no permanent stockpiles, and the year-round low to very low ambient temperature means that these material heaps will not require lining. Any minor earthworks, protections berms, and diversion trenches will be created as required by the Silver Bear site construction team. The cost of this is assumed to be part mine production cost.

Tailings Management Facility

Using the Vertikalny Feasibility Study (Tetra Tech 2016c) rates, Tetra Tech estimates that the capital cost required to provide additional storage space for the Mangazeisky North tailings will be $212,000.

Mine Closure

Once the mining operation has stopped and the remaining material in the stockpiles is processed, machinery and personnel will be reassigned to complete the earthworks required for mine closure. Machinery operating costs have taken into account the requirement to keep machinery running when not in active duty to prevent freezing. This time will be utilised for the mine closure activities.

An allowance of $330,000 has been made for TMF closure.

Indirect Costs

The indirect capital costs include project indirect costs, EPCM, Owner's cost, and allowances (which consists only of contingency allowance) and are shown in Table 24.30.

Table 24.30 Indirect Capital Cost Estimate

Area	Description	Cost ($)
Owner's cost	ESIA and Russian permit applications	250,000
EPCM	15% of direct capital cost	120,000
Allowances	Contingency	440,400
Total		**810,400**

A breakdown of the contingency is shown in Table 24.31.

Table 24.31 Allowances

Allowance	Description	Cost ($)
Haul Road	30% contingency	239,400
Owner's Cost	30% contingency	75,000
EPCM	30% contingency	36,000
TMF	Calculated contingency	90,000
Total		**440,400**

The estimated contingencies are allowances for undefined items incurred within the defined scope of work covered by the estimate, which cannot be explicitly foreseen or described at the time the estimate is completed due to the lack of complete accurate and detailed information. The contingency allowance is an integral part of the estimate; it is not to be considered as a compensating factor for estimating inaccuracy, nor is it intended to cover such items as any potential labour disputes, currency fluctuations, escalation, force majeure, or other uncontrolled risk factors. It should be assumed that the contingency amount will be spent over the engineering and construction period.

OPERATING COST ESTIMATE

Summary

The operating cost estimate for the Project consists of mining, processing, G&A (which includes TMF and site water management costs).

The total LOM average operating cost is estimated to be $150/t processed. The operating cost estimate is expressed in US dollars, unless specified. Table 24.32 summarises the unit operating costs of each area.

Table 24.32 LOM Averaged Operating Cost Summary

Area	Unit Cost ($/t processed)
Mining	77.77
Processing	29.80
G&A	42.00
Total	**149.57**

Note: Total may not add due to rounding.

Basis of Operating Cost Estimate

Labour

The permanent work force for the Project will be as local to Yakutsk as possible, and certainly from within Russia. From time to time, international specialist consultants will be engaged for short periods to resolve specific issues; however, their cost has been accounted for separately.

Personnel on the mine site will work on a four-week-on/four-week-off roster. The on- and off-cycles start when the workers report for duty at and when they are returned to the office in Yakutsk. In general, personnel are only paid during the on-cycle.

In accordance with Russian Federation employment practise, working hours and subsequent remuneration is controlled on a daily basis. The first eight hours are considered normal working hours and are included in the basic wage. The next two hours are paid at one and half the basic rate, and any subsequent hours are paid at double the basic rate. No one is allowed to work a shift of longer than 12 hours for more than two days in a row. The basic rate is based on the annual basic salary earned over 1,968 h/a (the so called norm hours).

Each employee is entitled to 52 days of annual leave of which up to 28 days can be taken during the on-cycle of the work rotation. The remainder is paid leave during the off-cycle. Public holidays falling on the on-cycle will be remunerated at double the basic rate, while those that fall on the off-cycle are unpaid.

Personnel working on the mine site generally fall into three categories: shift workers, day shift workers, and senior management. Personnel from the senior management group are not replaced when they are on the off-cycle, but rely on a senior subordinates or fellow managers to act in their stead. As a result, this group's roster is staggered to ensure maximum cover by senior personnel is maintained. In general,

rosters will be staggered to provide operational continuity and mitigate the risk of severe disruption during flight delays.

Technical services will be common to mining, processing, and infrastructure to minimise redundancy. This is particularly important when providing cover for annual leave and illness.

The cost of each role on the mine site has been calculated to include basic and overtime pay (including income tax and social insurance) and takes into account annual leave, leave replacement cost, public holidays, and the number of people that would have to be employed to cover the shift and roster requirements. For example, the basic annual salary (related to the norm hours) for a plant operator is RUB840,000, whereas the cost of one plant operator shift position is RUB5,375,233 and requires the employment of four people. Table 24.34 shows the list of roles used to estimate the operating cost for the Project.

Labour, tax and social insurance rates were provided by Prognoz.

Fuel and Power

Winterised diesel oil will be used for all internal combustion engines and boilers on site. Fuel will be source from any number of local supplier and transported by truck to site. In Q4 2015 Prognoz paid RUB61,700/t for fuel on site.

The power cost was then estimated using an assumed power cost of $0.21/kWh, which assumes on-site power generation as shown in Table 24.33.

Table 24.33 Power Generation Cost Estimate

Cost Component	Unit	Value
Generator Fuel Consumption	l/kWh	0.26
Fuel Price on Site	$/l	0.77
Power Generation Fuel Cost	$/kWh	0.20
Power Generation Non-fuel Cost	$/kWh	0.01
Total Power Generation Cost	**$/kWh**	**0.21**

Table 24.34 Personnel Roles

Band	Sub-grade	Role	Labour Type	Title	Typical Roles	2016 Salary incl. Tax and Insurance (RUB/a)	Overtime Rate (RUB/a)	Total Payroll (RUB/a)	Leave Replacement Cost (RUB/a)	Total Cost to Company (RUB/a)	Cost/Shift Position (RUB/a)	Cost/Day Shift Position (RUB/a)	Cost/ Management Position (RUB/a)
F	2	Policy Making	Top Management	C-Level Management	CEO	-	-	-	-	-	-	-	-
	1				CFO	-	-	-	-	-	-	-	-
E	3	Programming	Senior Management	General Manager	General Director	4,200,000	1,539,181	5,739,181	-	5,739,181	22,956,725	11,478,363	5,739,181
	2				Mine Manager/Operations Director	3,840,000	1,407,251	5,247,251	-	5,247,251	20,989,006	10,494,503	5,247,251
	1				Plant Manager	3,480,000	1,275,322	4,755,322	-	4,755,322	19,021,287	9,510,643	4,755,322
D	3	Interpretive	Middle Management	Department Manager, Superintendent	Senior Engineer	3,000,000	1,099,415	4,099,415	-	4,099,415	16,397,661	8,198,830	4,099,415
	2				Engineer	2,700,000	989,474	3,689,474	-	3,689,474	14,757,895	7,378,947	3,689,474
	1				Engineering supervisors/Junior Engineer	2,400,000	879,532	3,279,532	-	3,279,532	13,118,129	6,559,064	3,279,532
C	3	Routine	Skilled	Supervisor, Technician	Foreman/Shift Boss	1,800,000	659,649	2,459,649	419,940	2,879,589	11,518,357	5,759,178	2,879,589
	2				Senior Technician	1,500,000	549,708	2,049,708	349,950	2,399,658	9,598,631	4,799,315	2,399,658
	1				Technician	1,200,000	439,766	1,639,766	279,960	1,919,726	7,678,905	3,839,452	1,919,726
B	2	Automatic	Semi-skilled	Charge hand, Machinist	Operator, Cook, Security, Truck Driver	840,000	307,836	1,147,836	195,972	1,343,808	5,375,233	2,687,617	1,343,808
	1				Cook Assistant	780,000	285,848	1,065,848	181,974	1,247,822	4,991,288	2,495,644	1,247,822
A	2	Defined	Unskilled	Labourer	Cleaner, Laundry	660,000	241,871	901,871	153,978	1,055,849	4,223,398	2,111,699	1,055,849
	1					600,000	219,883	819,883	139,980	959,863	3,839,452	1,919,726	959,863

Open Pit Mining

The open pit operating costs have been estimated based on project-specific information provided by Silver Bear and are shown in Table 24.35.

Table 24.35 Average Open Pit Mining Unit Operating Costs

Item	Cost ($/t mined)
Fuel	0.80
Lube	0.18
Drill	0.10
Blast	0.22
Maintenance	0.68
Labour	0.92
Total	**2.90**

Process

The process operating cost includes the following estimated components:

- labour (operations and maintenance)
- reagents and consumables (plant)
- maintenance and spares
- power.

Reagent consumptions have been estimated based on the high-level test work on the Mangazeisky North mineralisation.

The annual labour cost of the processing plant for Mangazeisky North mineralisation is assumed at $527,076 based on the labour cost estimate for the Vertikalny Feasibility Study (Tetra Tech 2016c).

The annual maintenance and spare costs are estimated at 3% of the primary equipment in the plant area ($238,913) based on the Vertikalny Feasibility Study (Tetra Tech 2016c) plant equipment cost ($7.9 million)

The power cost has been estimated based on the process plant absorbed power consumption, assuming a unit power cost of $0.21/kWh.

The total plant operating cost is estimated at $29.80/t, to an accuracy of ±35%. A summary of the process plant operating costs are shown in Table 24.36.

Table 24.36 Process Operating Cost Summary

Description	Cost ($/t)
Labor	5.36
Reagents and Consumables	9.00
Maintenance and Spares	1.41
Power	14.03
Total	**29.80**

Tailings Management Facility

The operating expenditure for the TMF is estimated to be $124,000 over four years which equates to $0.30/t processed. This is included in the G&A operating cost estimate.

General and Administrative

With the scale of the overall operation unchanged from the Vertikalny open pit phase, it is assumed that the G&A cost of $42.00/t processed applies for the Mangazeisky North open pit phase.

24.4.7 ECONOMIC ANALYSIS

INTRODUCTION

This PEA of the Mangazeisky North deposit should not be considered to be a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. There is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Note that Inferred material is treated as waste and is given zero value in the pit optimisation, mine schedule and financial modelling.

Tetra Tech prepared an economic evaluation of the Mangazeisky North deposit based on a pre-tax financial model. The base case uses a variable silver price of $19.37/oz, $19.79/oz, $19.72/oz, during the first three years of production and $19.92/oz for the remainder of the mine life. The exchange rate applied in the base case is RUB66.00/USD1.00. All currency units are US dollars unless otherwise specified. By potentially combining the Vertikalny and North Mangazeisky deposits the Project is able to extend its mine life to 10-years and mine material to 1 Mt. The potential pre-tax financial parameters were calculated:

- 1.9-year payback period
- US$103.6 million NPV at a 5% discount rate.

Silver Bear commissioned PwC to prepare the tax component of the model for the post-tax economic evaluation for this PEA, with the inclusion of applicable income and mining taxes.

The following post-tax financial results were calculated:

- 1.9-year payback period
- US$87.1 million NPV at a 5% discount rate.

PRE-TAX ECONOMIC ANALYSIS

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.

The Mangazeisky North business case is presented as a possible integrated part of the Vertikalny mine plan. This would allow the Mangazeisky North satellite deposit to benefit from the use of existing capital equipment such as mobile equipment and a process plant, without having to carry the cost. The financial model reflects the updated mine plan, with the Mangazeisky North mine plan inserted between the Vertikalny open pit and underground phases, and only incremental capital costs related specifically to the satellite deposit. Where more than one mine is operated in one year, a weighted average unit cost is applied.

A variable silver price was used, as shown below. The average and weighted average (based on silver production) is $19.84/oz and $19.78/oz, respectively.

- Year 1 $19.37/oz
- Year 2 $19.79/oz
- Year 3 $19.72/oz
- Years 4 to 11 $19.92/oz

Payable silver values were calculated using the base case silver price. Silver payable values were calculated based on base case metal prices. Net invoice value was calculated each year by subtracting the applicable refining charges from the payable metal value. At-mine revenues were then estimated by subtracting transportation and insurance costs. Operating costs for mining, processing, and G&A were deducted from the at-mine revenues to derive annual operating cash flow.

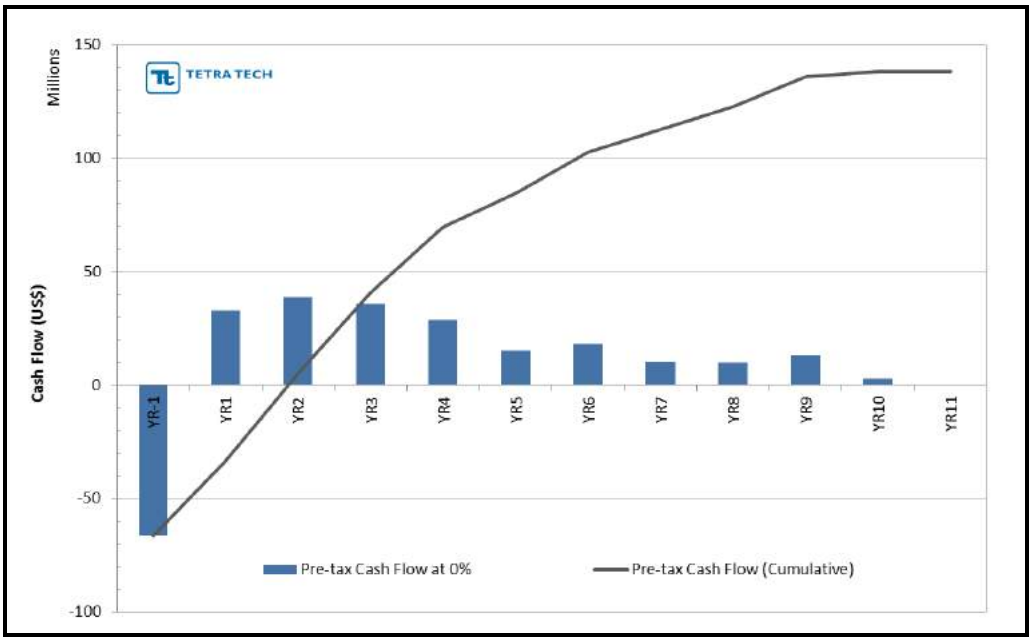
Initial and sustaining capital costs, as well as working capital, have been incorporated on a year-by-year basis over the LOM. The analysis assumes zero salvage value and mine reclamation costs, although an allowance has been made for closure costs. Capital expenditures were then deducted from the operating cash flow to determine the net cash flow before taxes.

As Mangazeisky North would utilize the infrastructure of the Vertikalny operation; the delineation of initial capital expenditures is arbitrary. In this context, initial capital is defined as costs accumulated prior to the processing of Mangazeisky North mineralized material. Sustaining capital includes any capital expenditures required during the production period. Initial and sustaining capital costs applied in the economic analysis are $65 million and $18.6 million, respectively.

Working capital is assumed to have a value equal to three months of operating cost and to be recovered at the end of the mine life.

The undiscounted annual net cash flow (NCF) and cumulative net cash flow (CNCF) are illustrated in Figure 24.27, representing the potential cash flow that could be generated by integrating the Mangazeisky North mined material into the Vertikalny mine plan and processing the material in the Vertikalny plant.

Figure 24.27 Undiscounted Annual and Cumulative Net Cash Flow



Summary of Financial Results

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 are presented in Table 24.37.

Table 24.37 Summary of Financial Results of the Combined Mine Plan

Item	Units	Value
Silver Price (LOM Weighted Average)	$/troy oz	19.78
Exchange Rate	RUB/USD	66.00
Total Capital Cost (LOM)	$ million	83.59
Mineralized Material (LOM)	kt	1,051
Silver Head Grade	g/t	784
Recovered Silver	koz (troy)	20.489
Unit Operating Costs	$/t processed	163
Pre-tax NCF	$ million	138
Pre-tax NPV @ 5% Discount Rate	$ million	104
Pre-tax-Payback	years	1.9

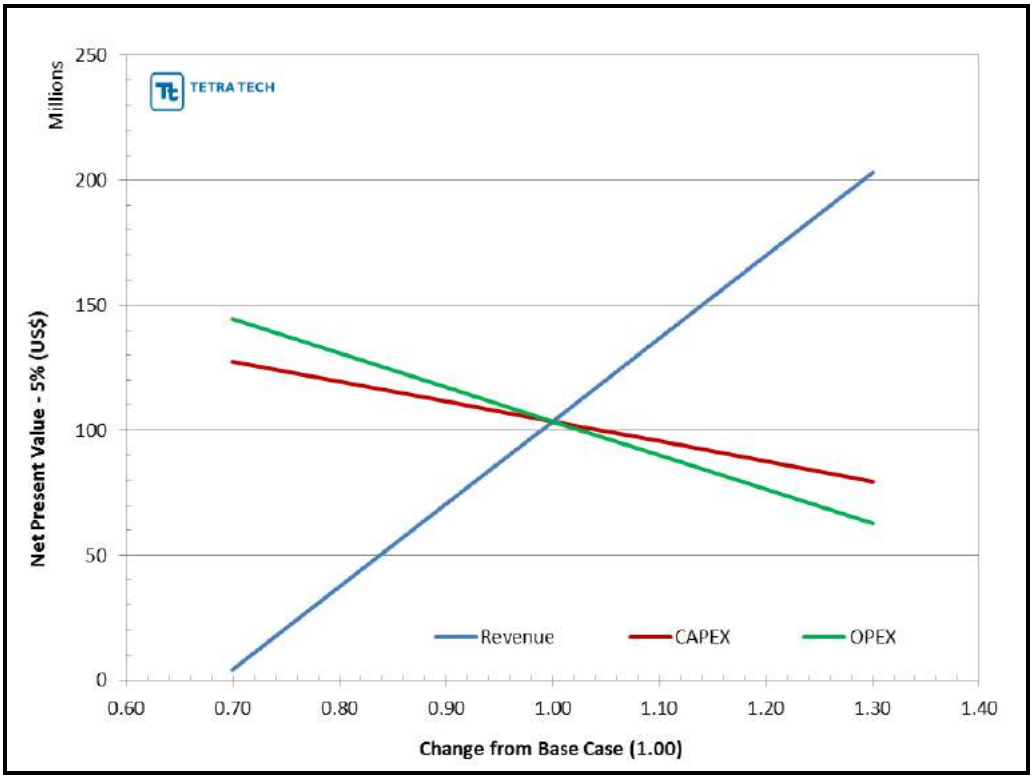
Sensitivity Analysis

Sensitivity of the Project's NPV and payback period 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 are the key variables investigated:

- capital costs
- operating costs
- revenue

As shown in Figure 24.28, the Project NPV, calculated at a 5% discount, is most sensitive to revenue factors such as silver price and metallurgical recovery.

Figure 24.28 Pre-tax NPV Sensitivity Analysis



Post-tax Economic Analysis

Silver Bear engaged PwC in Toronto, Canada to estimate the applicable Russian income and mining taxes for inclusion in the post-tax economic evaluation of the Project. PwC has estimated the applicable taxes in two cases. The first case does not account for the Far East Tax Incentives and the second case takes into account the Far East Tax Incentives.

Tax Assumptions

PwC prepared the tax estimate for inclusion in the financial model based on the following facts and assumptions, as applicable on 27 February 2017; the date on which the final draft of the model was prepared for inclusion in the financial analysis.

Profit Tax

- For the purposes of the profit tax calculation, net revenue (already net of off-site operating costs) less on-site operating costs, which are included in the financial model, were used as the Silver Bear's taxable income before various deductions.
- On-site and off-site operating costs were treated as tax deductible in the period in which they were incurred.
- Capital expenditures may be deducted for profit tax purposes through the depreciation of fixed assets based on their useful lives, which is determined based on the regulations issued by the Russian government. PwC has assumed that the fixed assets will have useful lives that span over the same period as the Project's life, which is eleven years as given in the financial model. Actual useful lives of Silver Bear's fixed assets, however, may vary from PwC's assumptions.
- Certain capital expenditures may be depreciated for profit tax purposes in the period the fixed assets are put into operation (called the "depreciation premium"). Depreciation premium may vary from 10 to 30% depending on the nature of the particular fixed asset and its useful life. For the purposes of computing depreciation premium, PwC has applied the rate of the depreciation premium in accordance with the information of the useful life of each group of fixed assets provided in the financial model.
- Sustaining capital costs provided in the financial model were treated as part of the capital expenses that are subject to depreciation for the profit tax purposes, with the assumption that these capital costs are generally related to the renovation of Silver Bear's fixed assets over their useful lives.
- PwC assumed that the working capital included as part of capital expense in the financial model would not be tax deductible for profit tax purposes. Also, royalties or other concessions that may not have been granted in respect of the Project have been ignored. Tax losses accumulated during a particular year may be carried forward and utilized against the taxable profits of Silver Bear with some limitations. According to the Russian tax legislation, from 2017 to 2020, the amount of prior-year losses recognised in the current period is limited to 50% of the current period tax base. From 2021, it will be allowed to recognise accumulated losses in full again. At the same time, the limitation on loss carry-forwards for a 10-year period is abolished;
- Any working capital return in the Project's life as given in the financial model was assumed to have no Russian tax implications, and hence was ignored. PwC assumed the best-case scenario for the application of the Far East Tax Incentives in the tax calculation. The law allows investors to use a reduced rate for profit tax purposes—zero rate for federal tax and a reduced rate for regional tax based on the following time frame:
  - 0 to 10% over the first five years from the year when the initial income from an investment project is received
  - 10 to 18% over the subsequent five years.

The specific regional rates are determined by the laws of the relevant Russian regions. There is a risk that the Yakutia Republic, where the Project is located, may

not apply the lowest regional tax rate. PwC applied a reduced regional tax rate to the first year in which Silver Bear generates a taxable profit base for purposes of the tax calculation, but can provide no assurance that this rate will be applicable when the Project enters commercial production.

Value Added Tax

- Under the Russian tax legislation, sales of precious metals mined by a taxpayer are subject to 0% output Value Added Tax (VAT) rate.
- Russian input VAT should be paid to the suppliers as part of the price of the goods or services. This amount can then be recovered from the Russian government. Accordingly, PwC did not include input VAT estimation in the tax calculation as it does not represent a cost to Silver Bear. Cash outflow related to payment and further recovery of input VAT does not usually last for more than six months, but such recovery may not apply to the future. A failure to recover VAT credits on a timely basis may have a material impact on the cash flow of the Project.

Mineral Resources Extraction Tax

- The tax rate of the Mineral Resources Extraction Tax (MRET) established for precious metals (other than gold) is 6.5%. This tax rate applies to the sales value of precious metals mined by the taxpayer during the tax period.
- A zero tax rate would apply to the losses of minerals that are incurred during the mining process within normal limits (the “normative losses”). The limits of losses are established annually by the public authorities to a particular field. Hence, these limits may change by the time the Project enters commercial production.
- For the purposes of the tax calculation in the financial model, PwC assumed the following scenario for the MRT regional tax incentive:
  - 0% of the standard rate is applicable for the first two years of the Project
  - 20% of the standard rate for years 3 and 4 of the Project
  - 40% of the standard rate for years 5 and 6 of the Project
  - 60% of the standard rate for year 7 and 8 of the Project
  - 80% of the standard rate for year 9 and 10 of the Project
  - 100% of the standard rate for the remaining life of the Project.

  PwC cannot provide assurance that these are the rates that will apply to the Project,

Property Tax

- Russian property tax is applicable to the average net book value of immovable property at a tax rate of 2.2%.
- All capital expenditures that were considered to be fixed assets and depreciated for profit tax purposes were assumed to be subject to property tax. Property tax is calculated based on these capital expenditures and PwC assumed that Silver Bear owns the fixed assets throughout the Project’s life as given in the financial model.

Taxation of Employees

- PwC assumed that both personal income tax and social insurance contributions relating to the employees have already been included in the Model as operating expenses. Therefore, PwC has not included these separately in the tax calculation.

Application of Far East Incentives

- PwC assumed that Silver Bear continues to be eligible for the Far East Incentives based on the letter from the Ministry of Economics of Yakutia Republic that was previously provided to PwC on file when performed the previous work in 2014 on the Vertikalny PEA for the same Project. PwC has also assumed that Silver Bear has or will make the corresponding application or submission of supporting documentation, as required, for this program. PwC was not provided with further information or subsequent determination regarding these incentives for purposes of the profit tax calculation. The receipt of any incentives will depend on future government approval in Russia.

Taxes and Post-tax Results

At the base case silver price and exchange rate used for this the preliminary economic assessment of the Mangazeisky North opportunity, the total estimated taxes payable on the Project profits over the 10-year LOM are $63.37 million without the Far East Tax Incentives, and $22.39 million with the Far East Tax Incentives. The components of the various taxes that will be payable and the post-tax results for the base case are shown in Table 24.38.

Table 24.38 Taxes and Post-tax Results

Tax Component	Unit	Option 1: Without Far East Tax Incentives	Option 2: With Far East Tax Incentives
LOM Weighted Average Silver Price	$	19.78	19.78
Exchange Rate	USD/RUB	66.00	66.00
Property Tax	$ million	12.07	12.07
MRET Tax	$ million	25.55	7.19
Profits Tax	$ million	25.75	3.13
Total Tax	$ million	63.37	22.39

24.4.8 Recommendations

Geology and Mineral Resource Estimate

Tetra Tech recommends the following two phases of work at Mangazeisky North:

- Phase 1 – strike extension drilling
- Phase 2 – Mineral Resource update.

Advancing to Phase 2 is contingent on the results of Phase 1.

Phase 1 - Infill and Strike Extension Drilling Programme

The Mangazeisky North deposit is open along strike to the north. It is recommended that Silver Bear drill a series of fences to the working progressively north along the strike of the deposit to build upon the Mineral Resource base.

Drilling Recommendations

Tetra Tech recommends a drilling campaign comprising nine drillholes, with an approximate total length of 400 m of drilling at Mangazeisky North. Figure 24.29 shows the proposed locations of drillholes.

Phase 1 – Cost Estimate

Tetra Tech estimates that the proposed drill programme will cost approximately CAD200,000 based on a drill program of 400 m, at an all-inclusive cost of CAD500/m drilled.

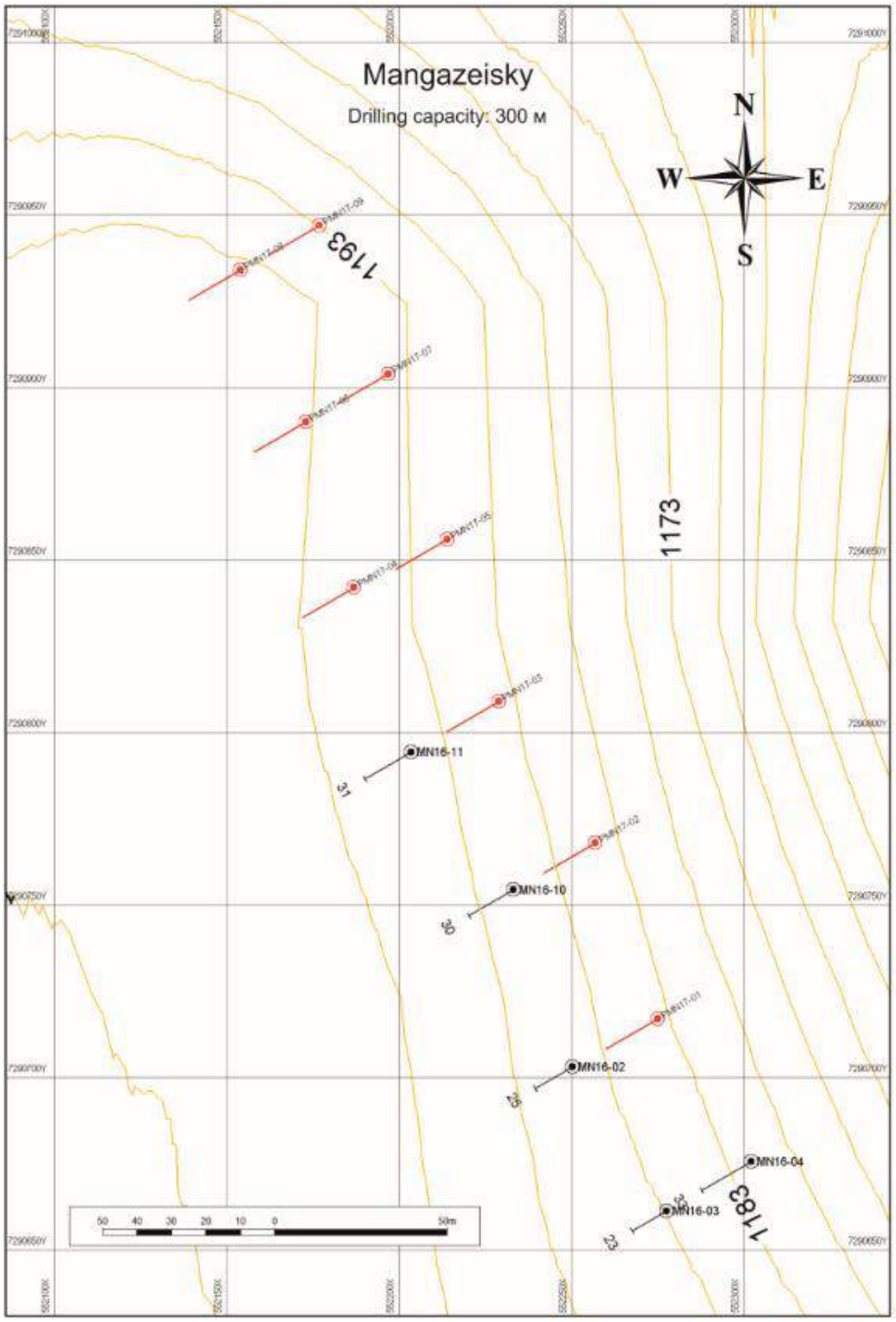
Phase 2 – Preliminary Economic Assessment

Tetra Tech recommends that Silver Bear complete a Mineral Resource update following the strike extension drilling. Subject to successful drilling results, this exercise will add to Silver Bear's Mineral Resource inventory and could support additional open pit optimisation studies.

Phase 2 – Resource Update Cost Estimate

Tetra Tech estimates that the Mineral Resource update could be completed for CAD40,000.

Figure 24.29 Proposed Drillhole Locations



Source: Prognoz

Mineral Processing

Based on the high level metallurgical test work completed, Tetra Tech recommends that further extensive metallurgical sampling and test work on the mineralized material from Mangazeisky North deposit should be completed.

The test work program is expected to include the following aspects as a minimum consideration:

- comminution test work (Bond Ball Mill index tests)
- detailed investigation of the preg-robbing behavior of the mineralization; for example, additional leach tests can be conducted with diesel oil as a potential additive to blind the preg-robbing carbon
- optimization of leach conditions for the sulphide mineralization with particular emphasis to the optimal grind size and the cyanide strength
- variability leach test work in order validate the silver recovery; a better geometallurgical understanding of the deposit will enable optimal and cost effective processing by way of better mine planning and scheduling
- A Quantitative Evaluation of Minerals by Scanning (QEMSCAN®) mineralogical investigation and silver deportment study in order understand the association and liberation characteristics of silver.

The approximate cost of the proposed test work program is estimated as CAD130,000.

Mining

It is anticipated that the ground conditions will be suitable for the proposed mining method; however, Tetra Tech recommends that detailed geotechnical investigations and analyses are undertaken for future studies.

The estimated cost for geotechnical drilling:

- footwall: 3 drillholes x 100 m at CAD500/m – CAD150,000
- hanging wall: 1 drillhole x 60 m at CAD500/m – CAD30,000

Environmental Studies, Permitting and Social or Community Impact

Recommendations for environmental and social or community impact are outlines in this section. The estimated cost for these recommendations is CAD300,000.

General

- A schedule of federal and state environmental permits required for construction and operations should be identified and prepared.
- Requirements for environmental and social impact assessment and relevant approvals from initial TEO to full ESIA should be identified.
- Terms of reference for an environmental, social and health baseline assessment in accordance with Russian Federation legal requirements and Equator Principles III/IFC 2012 guidelines should be prepared.

- ESIA scoping and a complete ESIA in accordance with national and international requirements should be conducted.

Environmental

- Environmental and social baselines studies are required to define the characteristics of the Managazeisky North area and in support of ESIA requirements. Baseline data often requires seasonal variations to be accounted for, therefore it would be prudent to commence the collection of such data early. The vicinity of and along potential haulage routes between the Mangazeisky area and the Vertikalny process plant should be considered.
- Further environmental studies are recommended to enhance the dataset characterizing the aquatic environment and confirm the source of elevated metals found in the local and regional waterways, including the potential impact of exploration and development activities that have already taken place. These studies should be used to inform the permitting process and to validate the effectiveness of environmental controls planned to protect the receiving environment.
- A detailed water balance should be developed to better characterize overall site water management intended for the Mangazeisky North deposit. This will confirm water supply through construction and operation, availability of reclaim water for the mill for operation, freshwater makeup supply and waste water sources and volumes. The water balance should be developed to characterize both the quality and quantity of water that will be managed site wide.
- River water quality and presence of aquatic species should be assessed, both for biodiversity value and presence of fish used for traditional fishing practices.
- Detailed environmental studies should be prepared in support of the TMF design (including ARD/metal leaching (ML) potential, water supply plan, etc.).
- Further study is recommended to characterize the predicted quality and quantity of water from the waste rock seepage and surface water run-off collection pond to confirm the assumed acceptability of release without additional management or treatment. This study should include further characterization of the waste rock that will be generated by the project and confirm that no further mitigation is required to protect the environment during operations and closure phases.
- A baseline study of biodiversity characteristics of the area should be conducted, including protected areas (not expected close to the site), species with conservation value (expected to be several), and species of value for traditional hunting and fishing activities.
- Further detailed management and monitoring programs should be established to support the construction, operations, and closure phases in accordance with Project standards; the plan should address:
  - air quality

- noise and vibration management
- water management
- transport and traffic management
- waste management
- management of hazardous materials
- biodiversity management
- environmental monitoring.

Social

- A Stakeholder Engagement Plan (SEP) should be prepared to actively engage with government authorities and community stakeholders as required by the Russian review and approval process. The SEP should include stakeholder mapping, engagement tools and frequency, address the indigenous people (Even) and include vulnerable groups such as the elderly, women or marginalised people.
- Further engagement with local indigenous communities should be conducted with regards to economic displacement through land use, potential impacts to forestry, hunting and fisheries activities together with any potential compensation claims. The social requirements of the International Finance Corporation's Performance Standard 7 (Indigenous Peoples) should be followed.
- Requirements for the compensation framework and calculated associated liabilities should be confirmed with respect to land take and impacts to grazing, reindeer migration or deer fawning lands.
- The potential for archaeological resources should be assessed, including tangible and intangible heritage; liaison with the relevant Ministry will be required to secure an approval letter for mining.
- A health profile assessment of the area should be conducted, including health service availability.

Closure and Remediation

- A detailed mine closure plan should be prepared, including remediation and reclamation together with liability costs for closure and any closure bonds required by the Russian government; the closure plan should include social planning including retrenchment of personnel.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 GEOLOGY

All of the exploration data for the Vertikalny, Mangazeisky North, and Nizhny Endybal deposits was reviewed and audited. In Tetra Tech's opinion, the exploration work carried out by Silver Bear has generally been completed using procedures that are consistent with recognised industry best practices, and the data is suitable for Mineral Resource estimation.

The 2015 infill drilling campaigns completed on the Vertikalny deposits provided further information in relation to the short-range variability of the grade distribution, and improved confidence in the grade continuity.

For the Mangazeisky North deposit, the parts of the deposit that were influenced by the 2015 infill drilling had sufficient data to allow the tonnage, grade, and density estimation to be completed with a suitable level of confidence for the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. This part of the deposit is therefore classified as Indicated Resource.

The Mangazeisky South deposit has sufficient data to allow the grade to be estimated, and to reasonably assume, but not verify, geological and grade continuity. Similarly, the 2011/2012 drilling at Nizhny Endybal provided adequate data to allow the grade to be interpolated, and reasonably assume, but not verify, geological and grade continuity. Both the Mangazeisky South and Nizhny Endybal Resource are classified in the Inferred category.

Tetra Tech is not aware of any material geological or resource risks associated with the Project.

25.2 MINERAL RESOURCE ESTIMATE

25.2.1 MANGAZEIKSY NORTH

Tetra Tech has reviewed and audited all of the exploration data for the Mangazeisky North deposits, and is of the opinion that the exploration work carried out by Silver Bear has generally been completed using procedures that are consistent with recognised industry best practices, and the data is suitable for resource estimation.

The parts of the deposit that were influenced by the recent infill drilling had sufficient data to allow the tonnage, grade, and density estimation to be completed with a suitable level of confidence for the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. This part of the deposit is therefore classified as Indicated Resource.

Outside of the area of influence of the recent infill drilling, geological and grade continuity can be assumed but not verified. This part of the deposit is classified as Inferred Resource.

In order to demonstrate that the deposit has reasonable prospects for economic extraction, a cut-off grade of 150 g/t was applied based on the following assumptions:

- silver price of $17.00/troy oz
- mining recovery 95%
- waste dilution 30%
- mining cost of $2.53/t
- processing cost of $52.00/t
- general and administration costs of $40.60/t
- royalty of 6.5%
- overall metal recovery of 88%.

Tetra Tech assumes that the deposits will be developed in tandem with the Vertikalny deposit, and will therefore share general and administrative costs.

The total Indicated Resource for the Mangazeisky North deposit is 334,000 t, with an average silver grade of 770 g/t. The total Inferred Resource for the Mangazeisky North deposit is 127,000 t, with an average silver grade of 560 g/t. There are no Measured Resources. The Tetra Tech is not aware of any material risks associated with the development of the Mangazeisky North deposits, or the wider Project.

25.2.2 VERTIKALNY

In Tetra Tech's opinion, the established understanding of the geological and grade continuity is sufficient to support the classification of the Mineral Resources as Inferred and Indicated, as presented in this report.

In order to demonstrate that the deposit has reasonable prospects for economic extraction a cut-off grade of 200 g/t silver has been applied for resources contained within an open pit shell at Vertikalny Central or within 80 m of surface at Vertikalny Northwest. A 350 g/t silver cut-off has been applied for the underground resource, based upon the following assumptions:

- silver price of USD17.00/troy oz
- Mining cost USD2.53/t open pit and USD70.00/t underground
- Processing cost of USD52.00/t
- General and administration costs of USD40.60/t.

At Vertikalny Central, a pit shell wireframe was used to constrain the open pit resource in order to demonstrate that the resource has reasonable prospects for economic extraction. The pit shell was developed using the following assumptions:

- silver price of $19.00/troy oz
- mining recovery 95%
- waste dilution 30%
- mining cost of USD2.53/t
- processing cost of USD52.00/t
- general and administration costs of USD40.60/t
- royalty of 6.5%
- overall silver recovery of 88%.

The total Indicated Resource for the Vertikalny Central deposit is 700,000 t, with an average silver grade of 1,227 g/t. There are no Measured Resources in either the Vertikalny Central or Northwest deposits. The total Inferred Resource for the Vertikalny Central deposit is 350,000 t, with an average silver grade of 786 g/t. The total Inferred Resource for the Vertikalny Northwest deposit is 200,000 t with an average silver grade of 476 g/t. There are no Indicated Resources within the Vertikalny Northwest deposit.

25.2.3 MANGAZEISKY SOUTH AND NIZHNY ENDYBAL

In Tetra Tech's opinion, the established understanding of the geological and grade continuity at Mangazeisky North in the areas that have been subject to infill drilling in 2015 is sufficient to support Indicate Resources. At Mangazeisky South and Nizhny Endybal the data is sufficient to support the classification of the Mineral Resources as Inferred.

In order to demonstrate that the deposits have reasonable prospects for economic extraction, a cut-off grade of 150 g/t was applied based on the following assumptions:

- forecasted silver price of $17/troy oz silver
- mining cost of $3.16/t
- processing cost of $50/t
- G&A of $12.50/t
- transport costs, from pit to process plant, of $3.60/t
- overall silver recovery of 80%.

Tetra Tech assumes that the deposits will be developed with the Vertikalny deposit, and will therefore share G&A costs.

The Mangazeisky South Mineral Resource is estimated at 60,000 t with an average silver grade of 246 g/t. There are no Measured Resources at Mangazeisky South at this stage.

The total Inferred Resource for the Nizhny Endybal deposit is 710,000 t, with an average silver grade of 316 g/t. There are no Measured or Indicated Resources at Nizhny Endybal.

25.3 METALLURGICAL TESTING AND RECOVERIES

The process design for the Project was completed based on the metallurgical test work, with the objective of developing a flowsheet that is simple as possible to supply, operate, and maintain, whilst maximizing silver recovery and minimizing power requirements.

Tetra Tech analysed the metallurgical test work results with the objective of identifying the optimal process design flowsheet. The metallurgical studies confirm that the ore is amenable to gravity concentration followed by cyanidation, and can be efficiently processed using a gravity and leach based flowsheet with direct electrowinning in order to produce silver powder. The presented process plant design is based on this principle; however, the installation of gravity circuit is deferred due to logistical issues. Hence, the design presented in this report is based on ROM Leach.

It is important to note that, an allowance for the construction of additional leach tanks has been made in order to process primary sulphide ore in the future.

The optimal grind size for primary grind is set at P_{80} = 75 µm for oxides and P_{80} = 25 µm for the primary ore type, based on the test work results.

The feasibility study process plant design is based on a 110,000 t/a capacity with a LOM average silver grade of 900 g/t, and is expected to provide a LOM silver recovery of 78.1% for oxide ore and 69.6% for primary ore.

25.4 MINERAL RESERVE ESTIMATES

The Mineral Reserve was estimated in accordance with CIM definitions and standards. According to the CIM Definition Standards for Mineral Resources and Mineral Reserves (prepared by the CIM Standing Committee on Reserve Definitions, Adopted by CIM Council on November 27, 2010), a Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a prefeasibility study. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined. The Mineral Reserve is inclusive of the reported Mineral Resource.

Proven and Probable Mineral Reserves for the Vertikalny deposit are presented in Table 25.1.

Table 25.1 Vertikalny Deposit Mineral Reserve Statement – April 30th 2017

Category	Cut-off Grade (g/t Ag)	Ore (t)	Ag Grade (g/t)	Ag Metal Content (koz)
Probable – Open Pit	240	400	1,083	13,934
Probable – Underground	450	458	569	8,375
Total Mineral Reserves	-	**858**	**809**	**22,309**

25.5 MINING METHODS

25.5.1 OPEN PIT

The integration of the open pit and underground mine design was the main task to optimise the mine boundaries and the overall open pit stripping ratio and the mine value.

The open pit mining operation will relatively be a small but high grade operation with a relatively high waste to ore strip ratio of 25:1. The selected optimal boundary is robust and no redesign may be required at slightly lower silver prices. If required the cut-off grade could be recalculated with the new economic parameters (within the acceptable change variation as per the sensitivity analyses results) and continue the operation within the designed open pits.

Accurate grade control, separate ore and waste blasting patterns, and selective ore excavation techniques in the mining operation will be the main key elements to minimise the dilution and to maintain a successful operation.

Based on the prepared open pit schedule, the first seven months of 2017 will be a critical period for the waste stripping operation where there is insufficient exposed ore for the direct delivery to the processing plant. The stockpile will be utilised to eliminate any ore delivery issues to the crusher, and should be maintained at a level of at least three months' supply to the plant.

The selected mining and ancillary equipment are suitable for the extraction of the orebody, waste stripping, and other earthmoving operations on site. However, during the harsh winter conditions, special care will be required to assure the safety of the personnel and the mining operation.

25.5.2 UNDERGROUND MINE

Tetra Tech recommends mechanised sub-level open stoping, with remote stope cleaning, as the safest and most productive underground mining method for the Vertikalny deposit. This method is preferred as it is particularly suited for narrow vein, steeply dipping, hard rock mineralisation geometries. The implementation of this method has become more practical with the advent of mining equipment technologies that have are compact and can be operated remotely. Mine simulations show that slightly higher dilution can be expected in in sub-level open stoping when compared to shrinkage stoping, but that this drawback is overshadowed by the benefit of lower underground unit operational costs.

All lateral development will be completed using an electrically-powered, single boom jumbo drill rig, with ventilation raises developed using a long-hole drill rig (stoping drill rig). The underground Mineral Reserve will be exploited in two distinct underground mining areas in the north and south, each with each own independent ramp access. This takes into account the stopes' proximity to surface and the open pit, and the operational flexibility provided by mining multiple deposits with a single fleet of mobile equipment. For the South Zone, adit accesses were designed for some of the sub-levels, with access from a surface road and from within the South Pit. As the

underground access ramps and adits will be tied into the open pit designs, the timing of their development will be contingent on the open pit mining schedule.

25.6 PROJECT INFRASTRUCTURE

The Project will require the development of a number of infrastructure items. The locations of project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and ensure efficient and convenient operation of the mine haul fleet.

Project infrastructure will include:

- airstrip
- off-site, on-site, and service road access
- mine haul roads
- TMF
- WRD, crushed ore stockpile, and ROM pad
- camp flood protection berms
- borehole water supply and distribution system
- water treatment and distribution
- sewage collection and management
- surface water management
- process and ancillary facilities, including:
  - process plant
  - crushing facility
  - reagent storage and handling
  - facilities for administration and an assay laboratory
  - heavy and light vehicle workshops
  - fuel storage and distribution
  - explosives storage and handling
  - emergency helipad
  - camp accommodation
  - mess and laundry facilities
  - medical centre
- power supply and local distribution network
- communications infrastructure.

Due to the remoteness of the mine site, Tetra Tech has purposefully steered away from an overly complex flow sheet in favour of technology that is simple and robust.

25.7 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Project is in progress with the Russian review and approval process. As part of this process, the environmental and social aspects of the Project are under evaluation by the Russian regulatory authorities. Currently, the mining part of the Project has received positive conclusions by the State Environmental Review; however, infrastructure and processing facilities are being revised based on comments from the regulators. There can be no guarantees that the Project will be approved under Russian law, the timeline for any decision, and if the Project is approved, that there won't be terms and conditions that will have a material impact on the feasibility of the Project.

The assessment of potential effects of the Project, the development of mitigating and management measures, and the establishment of monitoring programs embedded in the feasibility study are preliminary and are expected to evolve over time. The results of ongoing work, including ongoing environmental studies, stakeholder engagement, and permitting have the potential to influence the design, operations, and economic results of the Project.

The Project execution plan and schedule has been de-coupled from permitting activities. Construction activities are being undertaken without the necessary permits in place. Further, the commencement of mining required additional permits that may not be in place to support the currently planned schedule upon which the Feasibility Study is based.

In addition to real or perceived risks to safety and/or the environment, conducting these current and future activities without the necessary permits presents procedural risks that could result in regulatory action, including fines, Project re-design and/or schedule delays to the Project that could be material.

25.8 HYDROGEOLOGY AND WATER SUPPLY

As part of the Feasibility Study, site specific hydrogeological investigations have been carried out in the Verticalny deposit aiming to:

- Achieve sufficient understanding of the hydrogeology of the proposed mine and assess the dewatering requirements;
- Quantify the groundwater storage in the sub-permafrost aquifer, the availability of this to supply the required amount of water to the project.

The conclusions of the site investigations and data analysis are as follows:

- The mine (both the open pit and underground) will be located fully in the permafrost and therefore the groundwater inflow into the mine workings, if any, would be negligible.
- Surface water runoff during the warm season remains the main inflow component that requires active management, especially during exceptional storm events. For average conditions during the warm season, the required

pumping capacity to manage surface water runoff is about 100 m^3/h; however for exceptional storm event (100 year return period) the required pumping capacity is 200 m^3/h. In the freezing conditions (October to April) surface waters are frozen and precipitation falls in the form of snow, and so runoff does not occur.

- All four boreholes drilled in the Sirilendge river valley showed artesian flow, with water levels ranging from 1.1 m to 14 m above the ground level. Borehole GS15-05, which is located near the current camp, showed the highest water flow rate, indicating enough water for the project water supply. This borehole also showed the best water quality among all boreholes. Therefore, this borehole seems the most suitable for the project water supply.
- To support such observation, a numerical groundwater model was constructed and used to simulate long duration pumping from an imaginary well in GS15-05 borehole location. The model was first calibrated to reproduce the observations of the pressure recovery test that was carried out in this borehole during the investigation. Once the calibration was achieved the model was used in predictive mode to simulate various scenarios of pumping and duration. The groundwater model results suggest the existence of enough groundwater in the sub-permafrost aquifer to supply the required amount of water to the project. Nevertheless, since no long duration pumping tests were carried out as part of the Feasibility Study, the model results should be supported further with pumping tests and water quality analysis from the planned water supply wells in the spring of 2016.
- The total capital and operating cost for borehole water supply system for the project life (assumed 7 years) is estimated to around US$784,000. And the total capital and operating cost for the pit dewatering is estimated to around US$420,000 for the four years of mining.

25.9 CAPITAL AND OPERATING COST ESTIMATES

25.9.1 CAPITAL COST ESTIMATE

The total estimated initial capital cost for the design, construction, installation, and commissioning of all facilities and equipment is $49.9 million. A summary breakdown of the capital cost is provided in Table 25.2.

Table 25.2 Capital Cost Summary

Area	Initial ($)	Sustaining ($)	Total ($)
Mining	4,151,164	13,102,368	17,253,532
Processing	15,383,403	700,000	16,083,403
Infrastructure	4,584,021	-	4,584,021
Utilities	2,708,651	-	2,708,651
TMF	1,350,566	1,325,776	2,676,342
Site Facilities	7,221,388	-	7,221,388
Off-site Facilities	43,327	-	43,327
Project Indirects	16,388,012	1,016,409	17,404,421
EPCM	5,439,947	20,248	5,460,195
Owner's Cost	7,241,830	100,000	7,341830
Allowances	462,031	831,232	1,293,263
Total	**64,974,340**	**17,096,033**	**82,070,373**

This estimate includes direct field costs required to execute the Project, plus indirect costs associated with design, construction and commissioning. This estimate is based on pricing as of Q3 2015, with no allowances for inflation or escalation. Elements of actual expenditures until June 2017 have been included, which further reduce the exposure to inflation or escalation. All currencies in this capital cost estimate is expressed in US dollars, unless otherwise noted.

25.9.2 Operating Cost Estimate

The LOM operating cost estimate for the Project consists of mining, processing, and G&A costs (which includes TMF and site water management costs) and is estimated at $156.21/t processed. Table 25.3 summarises the unit operating costs of each area.

Table 25.3 LOM Averaged Operating Cost Summary

Area	Unit Cost ($/t processed)
Mining	53.71
Processing	60.58
G&A	41.92
Total	**156.21**

25.10 Economic Analysis

An economic evaluation was prepared for the Project based on a pre-tax financial model. The base case uses a variable silver price of $19.65/oz, $18.57/oz, $19.62/oz, $19.79/oz, $19.72/oz, and $19.92/oz during Q1 2018, Q2 2018, Q3/4 2018, 2019, 2020, and 2021, as well as the remaining project life, respectively, with a LOM weighted average silver price of US$19.76/oz. The exchange rate applied in the base case is RUB66.00/USD1.00. All currency units are US dollars unless

otherwise specified. For the 8-year LOM and 858,000 t Mineral Reserve, the following pre-tax financial parameters were calculated:

- 48.2% IRR
- 1.9-year payback on the $65 million initial capital
- $88 million NPV at a 5% discount rate.

Silver Bear commissioned PwC in Toronto, Canada, to prepare tax calculations for use in the post-tax economic evaluation of the Project with the inclusion of applicable Russian income and mining taxes. PwC estimated the applicable taxes in two cases. The first case does not account for the Far East Tax Incentives and the second case takes into account the Far East Tax Incentives (refer to Section 22.4 for further details). The following post-tax financial results were calculated for the base case without considering the Far East Tax Incentives:

- 28.4% IRR
- 2.4-year payback on the $65 million initial capital
- $46 million NPV at a 5% discount rate.

The following post-tax financial results were calculated for the base case when considering the Far East Tax Incentives:

- 44.3% IRR
- 2-year payback on the $65 million initial capital
- $77 million NPV at a 5% discount rate.

Sensitivity analyses were conducted to analyse the sensitivity of the Project merit measures (NPV, IRR, and payback periods) to the main inputs such as silver price, exchange rate, capital costs, and on-site operating costs. The Project's pre-tax NPV, calculated at a 5% discount rate, is most sensitive to silver price followed by exchange rate, on-site operating costs, and capital costs. The Project's pre-tax IRR is most sensitive to silver price followed by exchange rate, capital costs, and on-site operating costs. The payback period is most sensitive to silver price and exchange rate, followed by capital costs and on-site operating costs.

25.11 PROJECT EXECUTION PLAN

The Project is a greenfield project and will be facilitated using Owner-operated equipment and labour to mine the Vertikalny deposit.

The execution strategy for the successful monitoring and control of the Project reflects a modification to the traditional EPCM approach to project execution. Silver Bear will manage the completion of the Project with the input from various engineering companies, including a Russian-based company, who will complete the bulk of the detailed engineering, and international engineering companies, who will provide engineering review and specialist metallurgical input. Silver Bear will be responsible for the project management, procurement, and construction

management using in-house resources. Short lines of decision making will be used to execute this fast-track project.

The Silver Bear execution strategy will be built on a lean, fit-for-purpose approach that favours self-sufficiency in execution and operation.

The Project site near Yakutsk will be under the direct control of Silver Bear's construction manager who will report directly to Silver Bear's CEO. Earth and civil works will be executed by Silver Bear, while contracts will be issued for structural steel erection, mechanical and electrical installation, systems integration and specialist commissioning.

The project will reach mechanical completion in December 2016 followed by commissioning and ramp up in January 2017 to reach steady state production end of Q1 2017.

Tetra Tech concludes that the success of the project execution and future operation will rely on Silver Bear to manage the risk associated with winter road logistics. The current logistics system is sufficient for this scale of operation, and the operational team in Yakutsk has a proven track record of supporting the remote site.

26.0 RECOMMENDATIONS

26.1 GEOLOGY RECOMMENDATIONS

Tetra Tech recommends the following two phases of geology work:

- Phase 1 – infill drilling within the parts of the Vertikalny deposit constituting the first three years of production
- Phase 2 – infill drilling of high-grade minor parallel mineralised structures at the Vertikalny deposit.

Advancing to Phase 2 is not contingent on the results of Phase 1.

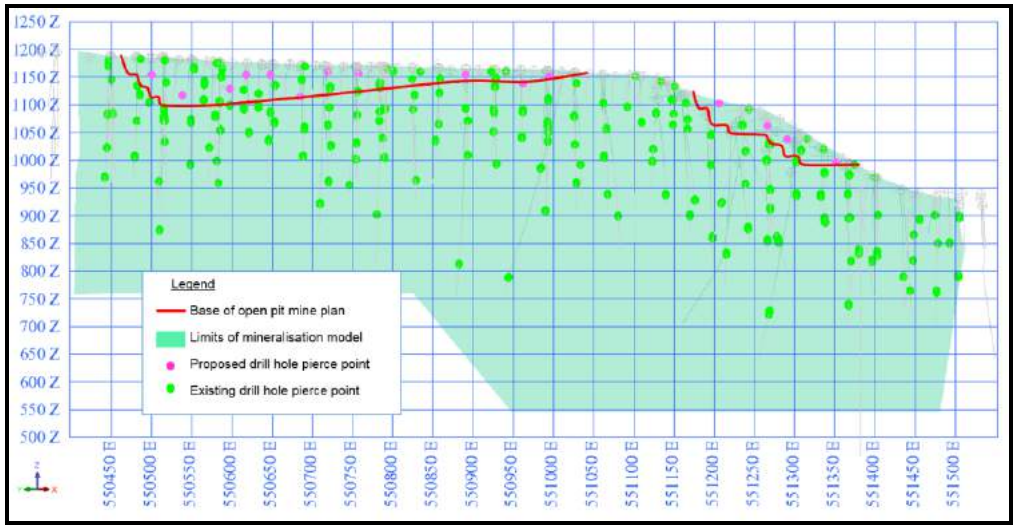
26.1.1 PHASE 1 – INFILL DRILLING FIRST THREE YEARS OF PRODUCTION

Infill drilling should be considered to further improve understanding of the short range variability in the grade continuity in the areas of the deposit that will constitute the initial three years of production. These areas are confined to the open pit element of the current mine plan. The increased understanding in these areas should provide sufficient support to allow the Mineral Resources to be locally classified as Measured.

PHASE 1 - DRILLING RECOMMENDATIONS

Tetra Tech recommends a drilling campaign comprising 15 drillholes, with an approximate total length of 750 m of drilling. Figure 26.1 illustrates the proposed locations of drillholes.

Figure 26.1 Proposed Drillhole Pierce Points



Source: Tetra Tech

PHASE 1 – COST ESTIMATE

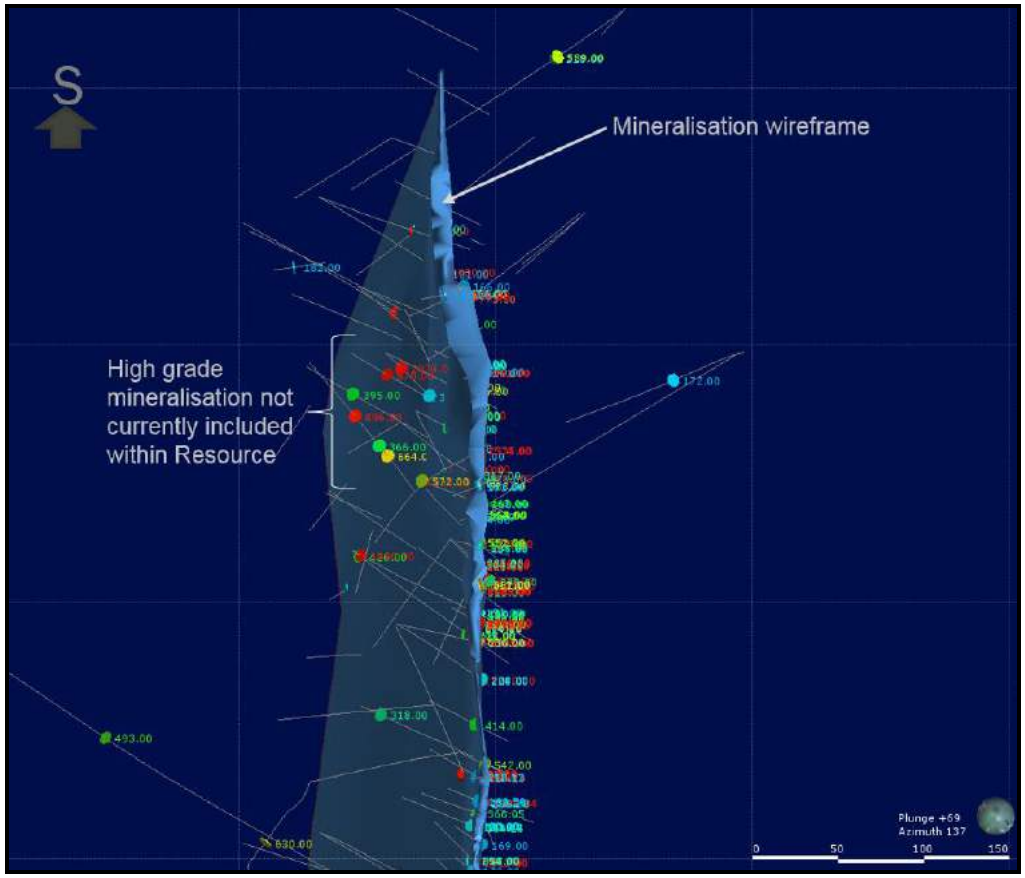
It is anticipated that the proposed drill programme will cost approximately CAD375,000 based on a drill program of 750 m, at an all-inclusive cost of CAD500/m drilled.

26.1.2 PHASE 2 – INFILL DRILLING HIGH GRADE MINOR PARALLEL MINERALISED STRUCTURES

The purpose of the second phase of infill drilling is to bring additional mineralisation into the Mineral Resource and more accurately delineate the geometry of the secondary structures.

It is proposed that four additional drillholes are advanced to intersect the parallel mineralised structures that are located within the southern open pit, as defined in the mine plan. It is anticipated that the additional data will allow the structures to be modelled adequately for Mineral Resource estimation.

Figure 26.2 3D Isometric View of the Mineralised Intercepts and Mineralisation Wireframe in the Southern Part of the Vertikalny Area



Source: Tetra Tech

PHASE 2 - DRILLING RECOMMENDATIONS

Tetra Tech recommends a small drilling campaign comprising 4 drillholes, with an approximate total length of 320 m of drilling.

PHASE 2 – COST ESTIMATE

It is anticipated that the Phase 2 drill programme will cost approximately CAD160,000, based upon a drill program of 320 m, at an all-inclusive cost of CAD500/m drilled.

26.1.3 GENERAL GEOLOGY RECOMMENDATIONS

It is recommended that additional sulphur analysis is completed at Vertikalny to expand the dataset available for modelling oxidation state. This information should be included, with an integer classification, within the geology table of the drillhole database.

Once the additional drilling and analysis is complete, it would be advisable to develop a geo-metallurgical block model that integrates metallurgical domaining with the Mineral Resource estimation to allow refined mine planning and process optimisation.

26.2 MINERAL PROCESSING AND METALLURGICAL TESTING

Based on the metallurgical test work and the process plant design completed for the feasibility study, Tetra Tech recommends the following.

- Further extensive metallurgical sampling and test work on the oxide ore should be completed in order to describe the ore variability in terms of hardness and silver recovery. A better geometallurgical understanding of the ore will enable optimal and cost effective processing by way of better mine planning and scheduling.
- A QEMSCAN® mineralogical investigation and silver deportment study for the primary ore should be completed in order understand the association and liberation characteristics of silver.
- Detailed metallurgical test work for the primary ore should be completed in order to optimize the processing options and conditions. The test work should include comminution and leach optimization tests, along with intensive leach of flotation concentrates with appropriate pre-treatment.
- The processing options for primary ore should be revisited based on the outcomes of mineralogical and metallurgical studies and the size of the primary ore reserve (the feasibility study flowsheet for primary ore is not optimal, but an acceptable solution based on the understanding that the primary ore constitutes only a minor proportion of the feasibility study ore Reserves).

26.3 MINING METHODS

The following are recommended for the mining operation:

- The detailed topography contour map of the open pit, WRD, and the road to the crusher be prepared to be used in the detail design and scheduling phase during the mining operation.

- Adopting an accurate and practical grade control operation with standard sampling and testing protocols with short turnarounds enabling the geology and mining departments to properly plan for the short term mining schedule.
- Increase the mined material and waste striping during the warmer weather periods, allowing a relaxed operation in the winter months without any interruption in supplying the ore tonnage to the process plant.
- Local safety bylaws be adopted for the personnel and equipment operation to prevent and minimise the risk of any injury or accident in the extreme weather condition.
- Personnel and equipment fatigue be monitored on regular basis and proper remedies be considered.
- Regular alcohol and drug tests be conducted in the start of the operation at every shift to minimise the risk of accidents.
- There is still significant potentially economical mineralisation remaining post the exploitation of the underground mine plan as designed. The remaining resource can and will be further explored.
- Fully mechanised sub level open stoping requires well trained personnel. Operational readiness should be implemented six months before the start of the underground operation.

26.4 PROJECT INFRASTRUCTURE

Tetra Tech recommends the following for the project infrastructure:

- Equipment for the processing plant is being procured from across the world. Political instability in certain regions may result in future sanctions or require rerouting of goods. A logistics plan should be in place to re-route equipment through unrestricted regions to ensure customs clearance.
- A maintenance camp is set up along the winter road at a point roughly half way along, in the region of Sebyan-Kyel, to provide first aid, breakdown and accident assistance to the convoys of trucks using the winter road, as well as providing accommodation and stores for maintenance equipment.
- Fuel is tested at every step of the supply chain to monitor quality and quantity. There are reports from other mine sites in the region of fuel being sold along the route and replaced with inferior grade fuel.
- Bulk storage of ammonium nitrate for an extended period of time can result in the granules caking up, inhibiting the take up of diesel, and resulting in a less efficient ANFO mixture. It is recommended that the ammonium nitrate is screened prior to loading onto the mobile manufacturing unit to screen out the caked material.
- The winter road season is dependent on the weather conditions and, as such, cannot be predicted accurately. To allow for this, a minimum of nine months of supplies should be stored on site, with a view to increasing this capacity to at least one year of supplies in the future. A trade-off study balancing risk versus cost of storage should be conducted.

26.5 TAILINGS MANAGEMENT FACILITY

SRK recommends the following for the TMF:

- Methods to minimize the volumes of the surface runoff water reporting to the clarification pond should be investigated. A potential solution is the formation of a "slop cell" in the centre of the dry stack to store frozen snow. This would reduce the size of the clarification pond and the requirement for water treatment costs.
- A thermal modelling of the dry stack should be performed to verify that the dry stack will be frozen with only the top 2-m thick layer being active. This may allow a steepening of the outer slopes and reduction in the footprint of the dry stack.
- The proposed closure plan should be refined to consider progressive rehabilitation of the dry stack, which could reduce the earthworks cost estimated for the post-closure period.
- Work is being carried out by Silver Bear to establish if a government environmental waste disposal levy of $7.88/t of tails will be enforced during deposition of tailings in the TMF. This environmental charge is preliminary and could change depending on the final waste classification.
- The impact of future Russian classification and permitting of the tailings material on the design and associated cost of construction of the TMF should be considered. The TMF design has considered, where possible, the impact of local environmental, best practice, and regulatory requirements; however, at the time of publishing this study the appropriate Russian application documents and associated regulatory approvals were not available to inform the work.

26.6 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

ERM provides the following recommendations as they relate to environmental, permitting, and regulatory aspects:

- The Project is in progress with the Russian review and approval process. As part of this process, the environmental and social aspects of the Project are under evaluation by the Russian regulatory authorities. Currently mining part of the Project has received positive conclusion by the State Environmental Review, however infrastructure and processing facilities are being revised based on comments from the regulators. There can be no guarantees that the Project will be approved under Russian law the timeline for any decision, and if the Project is approved, that there won't be terms and conditions that will have a material impact on the feasibility of the Project. ERM recommends that Silver Bear continue to actively engage with government authorities and community stakeholders as part of this process.
- Silver Bear currently accesses the camp and the mine site via land from the existing airstrip. A portion of the route taken traverses alongside the

Sirelendge River and crosses it in multiple locations. This is not a permitted road or activity. The Sirelendge River has a status of high fisheries significance. An upgraded road forms part of the Project as contemplated by the Feasibility Study but does not form part of the Russian Design Documentation submitted for permitting purposes. Due to the sensitivity of the adjacent river, using all-season access is not likely to be permitted by the fisheries authorities. ERM is of the view that access to the mine represents a potentially material risk to the Project.

- ERM recommends a detailed comparison between the Russian Design Documentation and the Feasibility Study is undertaken to establish if there are any discrepancies between the project as evaluated in this report and what is being reviewed and approved by the local government. Discrepancies have the potential to require amendments to the State Environmental Approvals that have been received.
- ERM notes that the Project execution plan and schedule has been de-coupled from permitting activities. Construction activities are being undertaken without the necessary permits in place to support the currently planned schedule upon which the Feasibility Study is based. In addition to real or perceived risks to safety and/or the environment, conducting these current and future activities without the necessary permits presents procedural risks that could result in regulatory action, including fines, Project re-design and/or schedule delays to the Project that could be material. ERM recommends that a detailed compliance assessment be undertaken and that Silver Bear work with regulatory authorities to address current and potential permitting/compliance gaps. ERM further recommends that any work currently underway, including construction, for which permits are not in place, ceases.
- Silver Bear has expressed its intention to comply with the Equator Principles and the IFC Performance Standards. ERM completed a gap assessment against these international frameworks for Silver Bear in November 2014. In 2016, the preliminary ESIA in line with IFC requirements was completed by ERM and relevant recommendations on further improvements of environmental and social performance were provided. ERM recommends that Silver Bear continue efforts to align the Project with these international frameworks as a means to inform project design and the development and implementation of project-specific environmental and social management and monitoring plans. An updated gap assessment is recommended to assess the current level of alignment of the Project as defined by the feasibility study. The more advanced the Project the more difficult it may become to fill gaps remaining.
- Further engagement with local indigenous communities with regard to land and resource restoration is recommended to ensure that social requirements of the IFC Performance Standards are met.
- Further environmental studies are recommended to enhance the dataset characterizing the aquatic environment and confirm the source of elevated metals found in the local and regional waterways, including the potential impact of exploration and development activities that have already taken place. These studies should be used to inform the permitting process and to

validate the effectiveness of environmental controls planned to protect the receiving environment.

- The current water balance developed for the Project should be expanded to include site-wide surface water sources, including contact water that collects in the seepage pond, water from the TMF, and open pit and underground mine water. This water balance should be developed to characterize both the quality and quantity of water that will be managed site wide. The current model was not used to predict water quality. The water balance should be used in conjunction with a site-wide water monitoring program to validate model predictions and confirm that environmental mitigation measures are effective and discharge can occur in accordance with anticipated permit conditions without further treatment.
- During normal operations, the TMF will be operated as a zero discharge facility, however, further study should be undertaken to define the conditions under which discharge to the environment is possible to assess the acceptability of this discharge. The Feasibility Study does not currently include an allowance for treatment of this potential waste. Further characterization of the tailings material and the potential for ARD/ML is required. The ARD/ML work completed to date is preliminary.
- A seepage and surface runoff collection pond will be constructed down gradient of the WRD that will act as a sediment control pond allowing settling of suspended solids prior to releasing water to be released to the environment. Further study is recommended to characterize the predicted quality and quantity of this waste water to confirm the assumed acceptability of release without additional management or treatment. This study should include further characterization of the waste rock that will be generated by the project and confirm that no further mitigation is required to protect the environment during operations and closure phases. The feasibility study does not currently include an allowance for further treatment of seepage that is released from this pond.
- The establishment of more detailed management and monitoring programs to validate that the project will need to be designed and implemented to support construction, operations, and closure phases of the project. Current plans are conceptual in nature. Potential costs associated with the implementation of these programs have been included as an allowance as part of the G&A costs presented in Section 21.0. ERM has not evaluated these costs and has not commented on their adequacy.
- The existing planning for closure and reclamation (as presented by others) is currently at a conceptual stage. Further planning and costing is required to support future stages of the Project and validate the assumptions made in the Study. Emphasis should be placed on defining the required technical studies needed to inform the evolution of the closure and reclamation plan over the life of the Project such that the end goals are met. Particular attention should be placed on infrastructure decommissioning, the closure of waste facilities and the management of water post-closure, including the potential need for water treatment.

26.7 SITE WATER MANAGEMENT

Results of the WBM show with a greater than 95% confidence that, given the WBM's inputs and assumptions, the monthly average groundwater demand will not exceed 250 m^3/d during the life of the Project. This groundwater demand is approximately the amount simulated by SRK in their groundwater flow model for Scenario 3 (260 m^3/d). However, their groundwater flow model was calibrated to one hour of recovery from a shut-in test conducted at borehole GS15-05, which is completed in a fault. Since a pumping test could not be conducted at this borehole, we recommend hydrogeological testing be conducted at this borehole and/or at new boreholes to reduce the risk of not having a reliable water supply for the Project.

26.8 MANGAZEISKY NORTH OPEN PIT OPPORTUNITY

Based on the favorable outcome of a preliminary economic assessment of the Mangazeisky North opportunity, Tetra Tech recommends that the economic potential of the deposit is further investigated in a more detailed study. Owing to the relatively simple scope of the project, it should be possible to progress the investigation directly to feasibility level. Although the current potential economic outcome seems favourable based on the Indicated material alone, the project would benefit from infill drilling to convert Inferred material to the Indicated category.

As part of the local permitting process, the project will have to undergo a formal Russian state approval process up to and including the project. The budget for the proposed feasibility studies is estimated at CAD350,000.

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27.4 ENVIRONMENTAL

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28.0 CERTIFICATES OF QUALIFIED PERSONS

28.1 JACQUES DU TOIT, CENG, PRENG, MSCENG, PMP

I, Jacques du Toit, CEng, PrEng, MScEng, PMP, of Swindon, Wiltshire, United Kingdom, do hereby certify:

- I am a Project Manager with Tetra Tech Canada Inc. with a business address at Ground Floor, Unit 2, Apple Walk, Kembrey Park, Swindon, SN2 8BL, United Kingdom.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of University of Stellenbosch, (MScEng, 2001). I am a registered Professional Engineer and a member in good standing of the Engineering Council of South Africa (20070153). I have project managed numerous feasibility studies in Africa, Russia and the CIS; all of them in base and precious metal mining. The most recent example is Kiaka Gold Feasibility Study for Volta Resources. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on 21st July 2014 for one day.
- I am responsible for Sections 1.1, 1.11.1, 1.11.3, 1.13, 1.15, 1.16, 1.17, 2.0, 3.2, 3.3, 3.4, 18.1, 18.2, 18.3, 18.5.3, 18.6, 18.7, 18.8, 18.9, 18.11, 18.13, 18.14, 18.15, 19.0, 20.7, 21.1 (except tailings capital costs), 21.2.1, 21.2.2, 21.2.3, 21.2.7, 24.1, 24.2, 24.1.1, 24.4.4, 24.4.5, 24.4.6, 24.4.7, 24.4.8 (environmental only), 25.6, 25.9, 25.11, 26.4, 26.8, and 28.1 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the following reports
  - "Preliminary Economic Assessment on the Mangazeisky Silver Project, Russia" dated 27th February 2014
  - "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Swindon, United Kingdom.

"Original document signed by
Jacques du Toit, CEng, PrEng, MScEng, PMP"

Jacques du Toit, CEng, PrEng, MScEng, PMP
Project Manager
Tetra Tech Canada Inc.

28.2 ROBERT DAVIES, BSC (HONS), CGEOL, EURGEOL, FGS

I, Robert Davies, BSc (Hons), CGeol, EurGeol, FGS, of Swindon, Wiltshire, United Kingdom, do hereby certify:

- I am a Senior Resource Geologist with Tetra Tech Canada Inc. with a business address at Ground Floor, Unit 2, Apple Walk, Kembrey Park, Swindon, SN2 8BL, United Kingdom.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of University of Liverpool, (BSc. Geology, 2004). I am a member in good standing and Chartered Geologist (CGeol) with the Geological Society of London (#1013629) and European Geologist (EurGeol) with the European Federation of Geologists (#965). My relevant experience includes twelve years of professional practise as a consulting geologist working in mineral exploration and resource estimation, most recently completing NI 43-101 compliant resource estimations for the Trun gold silver project in Bulgaria, the Ilovitza copper gold project in Macedonia and a JORC compliant resource estimate for the Zegen Gol, gold deposit in Eastern Russia. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was from the 23rd to 28th August 2015 12th to 15th September 2016 and between 26th June and 9th July 2016.
- I am responsible for Sections 1.2, 1.3, 1.4, 1.5, 1.7.1, 17.2, 3.1, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.1, 14.3, 23.0, 25.1, 25.2.2, 25.2.3, 26.1, 27.2, and 28.2 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the following reports
  - "Preliminary Economic Assessment on the Mangazeisky Silver Project, Russia" dated 27th February 2014
  - "Technical Report and Resource Estimate of the Mangazeisky Silver Project, Yakutia Russia" dated 10th April 2015
  - "Technical Report and Resource Estimate of the Mangazeisky Silver Project, Mangazeisky North Deposits, Yakutia Russia" dated 07th July 2015
  - "Technical Report and Resource Estimate of the Mangazeisky Silver Project, Mangazeisky North Deposits, Yakutia, Russia" dated 13th April 2016
  - "Technical Report and Resource Estimate of the Mangazeisky Silver Project, Vertikalny Deposits, Yakutia, Russia" dated 8th August 2016
  - "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Swindon, United Kingdom

"Original document signed by
Robert Davies, BSc (Hons), CGeol, EurGeol, FGS"

Robert Davies, BSc (Hons), CGeol, EurGeol, FGS
Senior Resource Geologist
Tetra Tech Canada Inc.

28.3 SABRY ABDEL HAFEZ, PHD, P.ENG.

I, Sabry Abdel Hafez, PhD, P.Eng., of Vancouver, British Columbia, Canada, do hereby certify:

- I am a Senior Mining Engineer with SGS Canada Inc. with a business address at Suite 203, 10 de la seigneurie E., Blainville, Quebec, J7C 3V5, V6B 1N5.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of Assiut University (BSc Mining Engineering, 1991; MSc Mining Engineering, 1996; PhD 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 includes more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. I have been involved in 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 1.14, 3.3, 22.0, 25.10, and 28.3 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Vancouver, British Columbia.

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

Sabry Abdel Hafez, PhD, P.Eng.
Senior Mining Engineer
SGS Canada Inc.

28.4 ARUNASALAM VATHAVOORAN, PHD, CENG, MIMMM

I, Arunasalam Vathavooran, PhD, CEng, MIMMM, of Swindon, Wiltshire, United Kingdom, do hereby certify:

- I am a Senior Metallurgical Engineer with Tetra Tech Canada Inc. located at Unit 2, Apple Walk, Kembrey Park, and Swindon, SN2 8BL, United Kingdom.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the University of Nottingham, United Kingdom (PhD Mineral Processing, 2006). I am a Chartered Engineer registered and in good standing of the Engineering Council (#579205), the Institute of Materials, Minerals and Mining (#444570), and the Society for Mining, Metallurgy, and Exploration (#4149245). I have practiced the profession of Metallurgical Engineering since my graduation and have been employed with Tetra Tech since 2011 as a Metallurgical Engineer, and from 2012 as a Senior Metallurgical Engineer. My relevant experience for 12 years in metallurgical engineering covers a wide range of commodities including precious metals and base metals. I have delivered detailed studies including preliminary economic assessments, prefeasibility studies and feasibility studies for mining projects located in Europe, Russia, North America, Asia and Africa.
- I have not completed a personal inspection of the Property.
- I am responsible for Section 1.6, 1.10, 13.0, 17.0, 21.2.5, 24.4.3, 24.4.8 (mineral processing only), 25.4, 26.2, 27.1, and 28.4 of the Technical Report.
- I am independent of Silver Bear Resources as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the following reports:
  - "Technical Report and Resource Estimate of the Mangazeisky Silver Project, Yakutia Russia" dated 10th April 2015
  - "Technical Report and Resource Estimate of the Mangazeisky Silver Project, Vertikalny Deposits, Yakutia, Russia" dated 8th August 2016
  - "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 28th day of August 2017 at Swindon, United Kingdom.

"Original document signed by
Arunasalam Vathavooran, PhD, CEng, MIMMM"

Arunasalam Vathavooran, PhD, CEng, MIMMM
Senior Metallurgical Engineer
Tetra Tech Canada Inc.

28.5 ANTON FERDINAND VON WIELLIGH, B.ENG., PR.ENG., ECSA, MSAIMM

I, Anton Ferdinand von Wielligh, B.Eng. (Hons), Pr.Eng., MSAIMM, of Perth, Western Australia do hereby certify:

- I am a Principal Mining Engineer with ABGM PTY LTD (Australia) with a business address at 5 Picotee Mews, Coogee, WA, 6166.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the University of Pretoria – South Africa, and hold a Bachelor's and Honours Degree in Mining Engineering. I am a registered Professional Engineer with the Engineering Council of South Africa (ECSA) and also a registered member with the South African Institute of Mining and Metallurgy (SAIMM). My relevant experience includes:
  - Participation as a mining engineer and as a professional engineer on various Pre-Feasibility and Feasibility studies and operating mines (in excess of 15 years study and practical experience)
  - Relevant and appropriate practical mining and mine planning experience with similar ore deposits (specifically modelled Nickel deposits) with similar structure, geometry and mineralogy.
- 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 1.8 (underground), 1.9.2, 15.0 (underground), 16.2 (except 16.2.3), 16.3 (underground), 21.2.4, 25.4 (underground), 25.5.2, 26.3 (underground), and 28.5 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the following reports
  - "Preliminary Economic Assessment on the Mangazeisky Silver Project, Russia" dated 27th February 2014
  - "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Perth, Western Australia.

"Original document signed by
Anton Ferdinand von Wielligh, BEng (Hons), PrEng, MSAIMM"

Anton Ferdinand von Wielligh, BEng (Hons), PrEng, MSAIMM
Principal Mining Engineer
ABGM PTY LTD (Australia)

28.6 MAX BROWN, BSC, MSC, CENG, MIMMM

I, Max Brown, BSc, MSc, CEng, MIMMM of Bridgend, Vale of Glamorgan, United Kingdom, do hereby certify:

- I am a Principal Consultant (Geotechnical Engineering) with SRK Consulting (UK) Limited with a business address at 5th Floor, Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, United Kingdom.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation", dated 21st August 2017 (the "Technical Report").
- I am a graduate of Brunel University, London, UK (BSc Geology, 1998) and the Camborne School of Mines, Cornwall, UK (MSc Mining Geology, 1999). I am a member in good standing of the Institution of Mining, Metallurgy and Materials (IOM3) and a Chartered Engineer (#99300622). I have practiced my profession continuously since 2000, I have been employed by SRK Consulting (UK) Limited since February 2006 during which time I have undertaken a significant number of open pit geotechnical projects. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property.
- I am responsible for Section 16.1.2, 16.2.3, and 28.6 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 28th day of August 2017 at Cardiff, United Kingdom.

"Original document signed by
Max Brown, BSc, MSc, CEng, MIMMM"

Max Brown, BSc, MSc, CEng, MIMMM
Principal Consultant (Geotechnical Engineering)
SRK Consulting (UK) Limited

28.7 RENE GHARAPETIAN, P.ENG.

I, Rene Gharapetian, P.Eng., of Toronto, Ontario, Canada, do hereby certify:

- I am an Independent Mining Consultant with a business address at Tetra Tech, 6835A Century Avenue, Mississauga, Ontario.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the University of Tehran (Technical Faculty) (BSc, Mine Engineering, 1990). I am a registered Professional Engineer in good standing with the Professional Engineers of Ontario (100055424). I have practiced my profession from graduation to present. My relevant experience includes pit optimisation, open pit design, production scheduling, mine fleet estimation. 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 1.9.1 (open pit), 16.1 (except 16.1.2, 16.1.5, 16.1.8 and 16.1.9), 16.3 (open pit), 25.5.1, 26.3 (open pit), and 28.7 of the Technical Report.
- I am independent of Silver Bear 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 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 28th day of August 2017 at Mississauga, Ontario.

"Original document signed and sealed by
Rene Gharapetian, P.Eng."

Rene Gharapetian, P.Eng.
Independent Mining Consultant

28.8 SAUNJAY DUGGAL, MSC, P.ENG.

I, Saunjay Duggal, MSc, P.Eng., of Mississauga, Ontario, Canada, do hereby certify:

- I am a Chief Engineer Civil/Structural with Tetra Tech Canada Inc. with a business address at 6835A Century Avenue, Mississauga, Ontario, L5N 2N2.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of National Institute of Technology, Calicut, India (Bachelor of Technology in Civil Engineering, 1992) and the University of Oklahoma, Norman, Oklahoma (Master of Science in Civil Engineering, 1996). I am a registered Professional Engineer in good standing with the Professional Engineers of Ontario (#100137875). My relevant experience includes preparation of detailed structural engineering design of buildings and appurtenances for mining facilities including preparation of associated construction documentation. Performing lead structural engineering role for feasibility study (including quantity estimate for concrete, steel and architectural items) of various mines. 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 18.5.1 and 28.8 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Mississauga, Ontario.

"Original document signed and sealed by
Saunjay Duggal, MSc, P.Eng."

Saunjay Duggal, MSc, P.Eng.
Chief Engineer Civil/Structural
Tetra Tech Canada Inc.

28.9 LASZLO BODI, MSC, CENG, P.ENG.

I, Laszlo Bodi, MSc, CEng, P.Eng., of Mississauga, Ontario, Canada, do hereby certify:

- I am a Principal Civil/Geotechnical Engineer with Tetra Tech Canada Inc. with a business address at 6835A Century Avenue, Mississauga, Ontario, L5N 7K2.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of Budapest University of Technology and Economics, Budapest, Hungary (B.Sc. Civil Engineering, 1977, M.Sc. Civil Engineering 1978). I am a member in good standing of the Professional Engineers of Ontario (#90235631), the Ontario Society of Professional Engineers (11386128) and the Canadian Geotechnical Society (#060398). My 37 years' experience covers a wide range of areas including detailed mining projects (waste disposal and infrastructure studies at scoping, prefeasibility and feasibility levels), foundation design and construction support for buildings, bridges, hydroelectric and nuclear power generating stations, pumping stations for TransCanada pipelines, transmission tower and pole foundations, transformer stations, large elevated storage tank and wind turbine foundations etc. I also have extensive experience in design and construction of seepage control systems (perimeter drainage and cut-off walls) around various waste disposal sites in Canada and around the World, in settlement analysis, stability analysis and design of road and railway embankments, pavement evaluations, and stability analysis and design of various concrete and earth-fill dams for mining applications. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My personal inspection of the Property was from the 10th to 12th of August 2015, and most recently from the 11th to 13th of October 2016.
- I am responsible for Sections 18.4, 18.5.2, 24.4.4 (haul road between North Mangazeisky and camp and tailings management facility), and 28.9 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the following reports:
  - "Preliminary Economic Assessment on the Mangazeisky Silver Project, Russia" dated 27th February 2014
  - "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Mississauga, Ontario.

"Original document signed and sealed by
Laszlo Bodi, MSc, CEng, P.Eng."

Laszlo Bodi, MSc, CEng, P.Eng.
Principal Geotechnical Engineer
Tetra Tech Canada Inc.

28.10 KRYSZTOF CZAJEWSKI, BSC, P.ENG.

I, Krzysztof Czajewski, BSc, P.Eng. of Cardiff, Wales, United Kingdom, do hereby certify:

- I am a Principal Consultant (Tailings Engineering) with SRK Consulting (UK) Limited with a business address at 5th Floor, Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the University of Saskatchewan, Canada (BSc Civil Engineering, 1989). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC) (#131396). I have practiced my profession continuously since 1989. I have been employed by SRK Consulting since February 2007 during which time I have undertaken numerous waste disposal engineering studies for various mining operations. 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 Section 1.11.2, 18.10, 21.1.3 (tailings capital costs only), 21.2.6, 26.5, and 28.10 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 28th day of Augusts 2017 at Cardiff, United Kingdom.

"Original document signed by
Krzysztof Czajewski, BSc, P.Eng."

Krzysztof Czajewski, BSc, P.Eng.
Principal Consultant (Tailings Engineering)
SRK Consulting (UK) Limited

28.11 DAMIAN HICKS, MIEAUST, CPENG

I, Damian Hicks, MIEAust, CPEng, of 60 Oakfield Road, London, United Kingdom, do hereby certify:

- At the time of writing this report I was an Associate with Tetra Tech Canada Inc. with a business address at 54A Topsfield Parade, London, United Kingdom.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the University of Queensland, Australia (BEng Electrical, 2006; BSc Physics, 2006). I am a registered Chartered Professional Engineer and a member in good standing of the Engineers Australia (#2790720). My relevant experience as a chartered electrical engineer includes project experience in multiple sectors, across multiple continents. I have been involved with several international projects, with experience in producing electrical and instrumentation scope for feasibility and EPCM projects in the mineral processing sector. 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 18.16, and 28.11 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at London, United Kingdom.

"Original document signed by
Damian Hicks, MIEAust, CPEng"

Damian Hicks, MIEAust CPEng

28.12 DEREK CHRISTOPHER CHUBB, P.ENG.

I, Derek Chubb, P.Eng., of Toronto, Ontario, Canada, do hereby certify:

- I am a Senior Partner with ERM Consultants Canada Inc. with a business address at 3300 Bloor Street West, Suite 2050, West Tower, Toronto, Ontario, M8X 2X2.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of McMaster University, Hamilton, Ontario, (Bachelor of Chemical Engineering, 1990). I am a Professional Engineer and a member in good standing of the Professional Engineers of Ontario (#90328121). My relevant experience includes:
  - 24 years working in the field of environmental management
  - review and report as a consultant to numerous operations and projects on matters related to environmental and socio-economic management
  - management of the design and implementation of environmental and socioeconomic baseline data programs, regulatory submissions including EIS documents, and project permitting support
  - senior management roles with accountability for environmental affairs at several mining companies including the VP Sustainability for a company seeking to develop a project in arctic conditions (Canada)
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.12, 20.1, 20.2, 20.3, 20.4, 20.5, 20.6, 25.7, 26.6, 27.4, and 28.12 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Toronto, Ontario.

"Original document signed and sealed by
Derek Chubb, P.Eng."

Derek Chubb, P.Eng.
Senior Partner
ERM Consultants Canada Inc.

28.13 GUY ROEMER, PE

I, Guy Roemer, PE, of Louisville, Colorado, United States, do hereby certify:

- I am an Environmental Engineer with Tetra Tech, with a business address at 363 Centennial Parkway, Suite 210, Louisville, Colorado, USA, 80027.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of Texas A&M University, College Station, Texas (BS Nuclear Engineering, 1995) and the University of New Mexico, Albuquerque, New Mexico (MS Nuclear Engineering, 1997). I am a Professional Engineer and member in good standing of the Professional Engineers of Colorado (#36810). My relevant experience includes 19 years of professional experience conducting site-wide and storage facility water balance models of mines using both spreadsheets and dynamic systems models, such as GoldSim. My experience also includes conducting groundwater flow and transport models of mine sites. 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 18.12, 26.7, 27.3, and 28.13 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Louisville, Colorado.

"Original document signed by Guy Roemer, PE"

Guy Roemer, PE
Environmental Engineer
Tetra Tech

28.14 HOUCYNE EL IDRYSY, PHD, CGEOL, FGS

I, Dr Houcyne El Idrysy, PhD, CGeol, FGS of Cardiff, Wales, UK, do hereby certify:

- I am a Principal Consultant (Hydrogeology) with SRK Consulting (UK) Limited with a business address at 5th Floor, Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the Université Libre de Bruxelles (ULB), Belgium (MSc Hydrology, 1995) and of Vrije Universiteit, Brussels (VUB), Belgium (PhD Hydrogeology, 2003). I am a Chartered Geologist and Fellow member in good standing of the Geological Society of London #1014745, and member of the International Mine Water Association (IMWA) and the International Association of Hydrogeologists. My relevant experience is in hydrogeology and mine water management. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on 15 September 2015 for one day.
- I am responsible for Section 24.3, 25.8, and 28.14 of the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the report titled "NI 43-101 Technical Report Feasibility Study Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 9th June 2016.
- 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 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 28th day of August 2017 at Cardiff, United Kingdom.

"Original document signed by
Houcyne El Idrysy, PhD, CGeol, FGS"

Houcyne El Idrysy, PhD, CGeol, FGS
Principal Consultant (Hydrogeology)
SRK Consulting (UK) Limited

28.15 JOSEPH HIRST, BSC (HONS), MSC, CGEOL, EURGEOL

I, Joseph Hirst, BSc (Hons), MSc, CGeol, EurGeol, of Swindon, Wiltshire, United Kingdom, do hereby certify:

- I am a Geologist with Tetra Tech Canada Inc. with a business address at Ground Floor, Unit 2, Apple Walk, Kembrey Park, Swindon, SN2 8BL, United Kingdom.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the University of Manchester, England (BSc (Hons.) Geology, 2001. I am a member in good standing and Chartered Geologist (CGeol) with the Geological Society of London (#1007756). My relevant experience includes 14 years of professional practice. I have been directly involved in mineral Resource estimations for 10 years, which recently includes, but it not limited to Sterzhnevoy, Porphivory on the Mangazeisky Property, and other precious metal deposits of a similar mineralization style. 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 1.7.3, 14.2, 25.2.1, 24.4.8 (geology), and 28.15 the Technical Report.
- I am independent of Silver Bear Resources Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property was as a Qualified Person for the following reports:
  - "Technical Report and Resource Estimate of the Mangazeisky Silver Project, Sterzhnevoy and Porhivory Deposits, Yakutia, Russia" dated 14th October 2016.
- 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 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 28th day of August 2017 at Swindon, United Kingdom.

"Original document signed by
Joseph Hirst, BSc (Hons), MSc, CGeol, EurGeol"

Joseph Hirst, BSc (Hons), MSc, CGeol, EurGeol
Geologist
Tetra Tech Canada Inc.

28.16 MATTHEW RANDALL, BSC, PHD, CENG, MIMMM

I, Matthew Randall, BSc (Hons), PhD, CEng, MIMMM, of Axbridge, United Kingdom, do hereby certify:

- I am a Director with Axe Valley Mining Consultants Ltd. located at 47 West Street, Axbridge, Somerset, BS26 2AA, United Kingdom.
- This certificate applies to the technical report entitled "NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation" dated 21st August 2017 (the "Technical Report").
- I am a graduate of the Camborne School of Mines, United Kingdom (BSc (Hons) Mining Engineering, 1978; PhD Mining Engineering, 1989). I am a Chartered Engineer registered and in good standing of the Engineering Council (Registration No. 45134) and the Institute of Materials, Minerals and Mining (Membership ID 458442). My relevant experience includes 35+ years in mining engineering in many aspects of open pit mine development, construction, and operations across several continents. My experience covers a wide variety of commodities including, gold, copper, platinum, zinc, iron ore, industrial minerals, diamonds, and coal. I have consulted on strategic evaluation and concept studies for mines, as well the preparation of Prefeasibility and Feasibility study reports.
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.8 (open pit), 1.9.1, 15.0 (open pit), 16.1.5, 16.1.8, 16.1.9, 24.4.2, 24.4.8 (mining), 25.4, and 28.16 of the Technical Report.
- I am independent of Silver Bear Resources as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this 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 contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 28th day of August 2017 at Axbridge, United Kingdom.

"Original document signed by
Matthew Randall, BSc (Hons), PhD, CEng, MIMMM"

Matthew Randall, BSc (Hons), PhD, CEng, MIMMM
Director
Axe Valley Mining Consultants Ltd.