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SILVER BEAR RESOURCES PLC

NI 43-101 TECHNICAL REPORT

MANGAZEISKY SILVER PROJECT MRE UPDATE AND STRATEGY RE-ASSESSMENT, REPUBLIC OF SAKHA (YAKUTIA), RUSSIAN FEDERATION

March 2021

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DATE AND SIGNATURE PAGE

The effective date of this report: 25th March, 2021.

This report titled "**Mangazeisky Silver Project MRE Update and Strategy Re-Assessment, Republic of Sakha (Yakutia), Russian Federation**" for Silver Bear Resources Plc., dated 25th March, 2021, was prepared and signed by the following author:

Steven James McRobbie {signed and sealed}

Steven James McRobbie, B.Sc. (Hons), M.Sc., ACSM, MAusIMM

Signing Date: 30th March, 2021

CERTIFICATES

CERTIFICATE OF AUTHOR

Steven James McRobbie

As author of this report entitled "Mangazeisky Silver Project MRE Update and Strategy Re-Assessment, Republic of Sakha (Yakutia), Russian Federation", with an effective date of 25th March, 2021 (the "Technical Report"), I, Steven James McRobbie, do hereby certify that:

i. I am a full-time employee of Wardell Armstrong Russia and employed as Regional Director, based at Office 5050, 21/5 Kuznetsky Most, Moscow, 107996, Russia;
ii. I graduated with a Bachelor of Science (Hons) Degree in Geology from the University of St Andrews, Scotland, in 1992 and thereafter graduated with a Master of Science Degree in Mining Geology from Camborne School of Mines, Camborne, Cornwall, UK, in 1994;
iii. I have worked as a geologist continuously for a total of 28 years since graduation;
iv. I am a registered member in good standing of the Australasian Institute of Mining and Metallurgy (Membership number 224976);
v. I have read the definition of "qualified person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "qualified person" for the purposes of the Instrument;
vi. I have not visited the property due to safety requirements enacted under pandemic conditions as defined by WHO due to Covid-19 and severely restricting movement of personnel as a result of Federal, Regional and site operational Health & Safety regulations that have been in effect in Russia since March 20th, 2020;
vii. I am responsible for the preparation of the Technical Report titled "Mangazeisky Silver Project MRE Update and Strategy Re-Assessment, Republic of Sakha (Yakutia), Russian Federation";
viii. I am independent of the issuer as defined in section 1.5 of the Instrument;
ix. I am independent of Silver Bear Resources Plc. as defined by Canadian NI 43-101 regulations and have provided consulting services to the companies;
x. I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form; and
xi. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of March 2021

Steven James McRobbie {signed and sealed}
Steven James McRobbie, B.Sc. (Hons), M.Sc., ACSM, MAusIMM

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SILVER BEAR RESOURCES PLC

NI 43-101 TECHNICAL REPORT ON THE MANGAZEISKY SILVER PROJECT MRE UPDATE AND STRATEGY RE-ASSESSMENT, REPUBLIC OF SAKHA (YAKUTIA), RUSSIAN FEDERATION

March 2021

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CONTENTS

TABLES

FIGURES

APPENDICES

1 EXECUTIVE SUMMARY (ITEM 1)

Silver Bear Resources PLC (SBR) has commissioned Wardell Armstrong International (WAI) to carry out an update of its mineral resource base and strategic re-assessment of the Mangazeisky Silver Project. The study has aimed to assess the combined potential of the Vertikalny and Mangazeisky North deposits and identify any strategic bottlenecks. The key elements included within the study are listed below:

- Mineral Resource Estimation;
- Hydrological and hydrogeological review;
- Mining geotechnical review;
- Open pit mining study;
- Underground mining study;
- Mine production scheduling;
- Mining capital and operating cost estimation;
- Mineral processing review; and,
- Financial analysis.

1.1 Vertikalny - Mineral Resource Estimate

The Mineral Resource Estimate was carried out with a 3D block modelling approach using Datamine Studio RM software. The effective date of the Mineral Resource Estimate is the 31st May 2019, the date of the limiting mine survey. In the opinion of WAI, the Mineral Resource Estimate reported herein is a reasonable representation of the mineral resources found in the Vertikalny Silver Project based on the current level of sampling.

WAI has been provided with exploration and grade control data for Vertikalny comprising all exploration carried out from 2005 to 2018 by CJSC Prognoz. Exploration data were imported and verified before geological and mineralisation envelopes were defined creating 3D wireframes based on a cut-off grade of 50g/t Ag representing the various mineralised zones at Vertikalny. In addition, digital terrain model (DTM) surfaces, surveys of mined-out areas, surfaces of overlapping sediments and boundaries of oxide and primary mineralisation were imported and/or created. Sample data were selected using the geological and mineralisation wireframes and selected samples were assessed for outliers before being composited to a length of 1.0m as the basis for geostatistical study.

The wireframe envelopes were used as the basis for a volumetric block model with a parent cell size of 10m x 10m x 10m and appropriate sub-celling to meet wireframe boundaries. Dynamic anisotropy was used to estimate dip and dip directions into each block of the model to control search ellipse orientation during grade estimation. Block model validation was carried out using visual, statistical and graphical checks between input composite sample data and estimated block grades.

Variogram models were constructed based on composite data and used Ordinary Kriging (OK) as the principal estimation methodology. Inverse Power Distance Cubed (IPD^2) was used for validation purposes.

The resultant estimated grades were validated against the input composite data and classification in accordance with the guidelines of the JORC Code (2012) and was carried out based on an assessment of geological and grade continuity and an assessment of assay data quality. Key drillhole spacing for the allocation of Mineral Resources stipulated Measured resources at 40m spacing, Indicated resources at 80m, and Inferred resources within greater than 80m. Mineral Resources (Table 1.1) were further limited based on an expectation of eventual economic extraction to an optimised open pit shell generated using appropriate economic and technical parameters. Underground Mineral Resources (Table 1.3) were allocated below the base of the optimised pit shell and above the Net Smelter Return cut-off value of $162.0/t.

Table 1.1: Mineral Resource Estimate. Vertikalny Project, Russia. 31st May 2019
(In Accordance with the Guidelines of the JORC Code (2012)) Potential Open Pit Resources

Ag Cut-off, g/t	Category	Tonnes, Kt	Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
50	Oxide							
	Measured	108.53	845.52	1.97	1.53	91,766	2,143	1,656
	Indicated	97.00	1,096.62	1.30	1.94	106,368	1,256	1,886
	Sub-Total M+I	205.53	964.03	1.65	1.72	198,133	3,399	3,542
	Primary							
	Measured	14.07	1,250.53	1.76	1.93	17,598	247	271
	Indicated	37.65	1,760.51	2.22	1.47	66,291	835	555
	Sub-Total M+I	51.73	1,621.77	2.09	1.60	83,889	1,082	826
	Oxide + Primary							
	Total M+I	257.25	1,096.28	1.74	1.70	282,022	4,481	4,368
100	Oxide							
	Measured	102.26	892.45	1.99	1.55	91,260	2,036	1,588
	Indicated	94.26	1,126.55	1.29	1.96	106,185	1,217	1,846
	Sub-Total M+I	196.51	1,004.73	1.66	1.75	197,445	3,253	3,434
	Primary							
	Measured	13.41	1,308.56	1.84	1.93	17,548	246	259
	Indicated	36.65	1,806.77	2.26	1.43	66,212	827	526
	Sub-Total M+I	50.06	1,673.30	2.14	1.57	83,761	1,073	785
	Oxide + Primary							
	Total M+I	246.57	1,140.46	1.75	1.71	281,205.34	4,325.70	4,218.76
200	Oxide							
	Measured	94.90	949.88	2.01	1.58	90,141	1,909	1,500
	Indicated	89.24	1,181.88	1.33	1.92	105,469	1,190	1,710
	Sub-Total M+I	184.14	1,062.32	1.68	1.74	195,610	3,099	3,211
	Primary							
	Measured	13.19	1,328.95	1.85	1.96	17,524	244	258
	Indicated	36.14	1,830.08	2.28	1.42	66,148	825	514
	Sub-Total M+I	49.33	1,696.13	2.17	1.56	83,672	1,069	772
	Oxide + Primary							
	Total M+I	233.47	1,196.24	1.79	1.71	279,281.95	4,168.20	3,982.53
300	Oxide							
	Measured	87.08	1,012.09	1.88	1.57	88,130	1,635	1,371
	Indicated	84.03	1,239.87	1.25	1.90	104,191	1,054	1,599
	Sub-Total M+I	171.11	1,123.96	1.57	1.74	192,321	2,689	2,971
	Primary							
	Measured	12.78	1,362.31	1.89	2.00	17,416	242	255
	Indicated	35.28	1,868.86	2.33	1.40	65,926	820	492
	Sub-Total M+I	48.06	1,734.12	2.21	1.56	83,342	1,062	748
	Oxide + Primary							
	Total M+I	219.17	1,257.75	1.71	1.70	275,662	3,715	3,718

Notes:
1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. Mineral Resources are not Ore Reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study.
3. Mineral resources include all potential mineable tonnage.
4. Mineral Resources are estimated as of 31 May 2019 based on an open pit mine survey of the same date.
5. Mineral Resources were constrained by an optimised pit shell using a NSR cut-off value of $172.78/t for oxide and $139.06/t for primary mineralisation.
6. Mineral Resources were constrained by an optimised pit shell based on economic and mining parameters provided by the Client and/or accepted by WAI.
7. This mineral resource estimate is not limited to any factors in terms of environmental, permitting, legal, title, taxation, socio-economic, market and other relevant factors.
8. The metal resources include all the in-situ metal disregard the metallurgical recovery factor.
9. All values in the tables have been rounded with relative accuracy of estimate. Numbers may not compute due to rounding.

Table 1.2: Mineral Resource Estimate. Vertikalny Project, Russia. 31st May 2019
(In Accordance with the Guidelines of the JORC Code (2012)) Potential Underground Resources

Ag Cut-off, g/t	Category	Tonnes, Kt	Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
50	Measured	0.52	383.12	2.52	0.55	199	13	3
	Indicated	419.06	463.13	1.12	2.59	194,076	4,675	10,847
	M+I	419.58	463.03	1.12	2.59	194,275	4,688	10,850
	Inferred	222.40	362.49	1.02	1.66	80,619	2,270	3,693
100	Measured	0.38	499.55	2.24	0.57	188	8	2
	Indicated	394.83	486.28	1.11	2.61	191,997	4,392	10,306
	M+I	395.20	486.29	1.11	2.61	192,185	4,401	10,308
	Inferred	214.55	372.81	1.02	1.62	79,985	2,178	3,465
200	Measured	0.36	515.71	2.32	0.58	185	8	2
	Indicated	328.27	555.26	1.16	2.52	182,275	3,806	8,267
	M+I	328.63	555.22	1.16	2.52	182,460	3,814	8,269
	Inferred	159.76	445.01	1.03	1.70	71,094	1,650	2,714
300	Measured	0.29	581.70	2.66	0.58	166	8	2
	Indicated	235.82	680.72	1.26	2.57	160,524	2,964	6,059
	M+I	236.10	680.60	1.26	2.57	160,690	2,972	6,061
	Inferred	109.42	538.93	1.26	1.75	58,970	1,378	1,919

Notes:
1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. Mineral Resources are not Ore Reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study.
3. Mineral resources include all potential mineable tonnage.
4. Mineral Resources are estimated as of 31 May 2019 based on an open pit mine survey of the same date.
5. Mineral Resources are located below an optimised pit and were evaluated based on an NSR cut-off value of $162.00/t for primary mineralisation.
6. Economic and mining parameters provided by the Client and/or accepted by WAI were incorporated in the calculation of NSR.
7. This mineral resource estimate is not limited to any factors in terms of environmental, permitting, legal, title, taxation, socio-economic, market and other relevant factors.
8. The metal resources include all the in-situ metal disregard the metallurgical recovery factor.
9. All values in the tables have been rounded with relative accuracy of estimate. Numbers may not compute due to rounding.

1.2 Mangazeisky North – Mineral Resource Estimate

The Mineral Resource Estimate was carried out with a 3D block modelling approach using Datamine Studio RM software. The effective date of the Mineral Resource Estimate is the 31st of May 2019. In the opinion of WAI, the Mineral Resource Estimate reported herein is a reasonable representation of the mineral resources found in the Mangazeisky North Silver Project based on the current level of sampling.

WAI has been provided with exploration data for Mangazeisky North comprising all exploration carried out since 2013 to 2016 by CJSC Prognoz. Exploration data were imported and verified before geological and mineralisation envelopes were defined creating 3D wireframes based on a cut-off grade of 50g/t Ag representing the various mineralised zones at Mangazeisky North. In addition, digital terrain model (DTM) surfaces and surfaces of overlapping sediments were imported and/or created. Sample data were selected using the geological and mineralisation wireframes and selected samples were assessed for outliers before being composited to a length of 1.0m as the basis for geostatistical study.

The wireframe envelopes were used as the basis for a volumetric block model with a parent cell size of 10m x 10m x 10m and appropriate sub-celling to meet wireframe boundaries. Dynamic anisotropy was used to estimate dip and dip directions into each block of the model to control search ellipse orientation during grade estimation. Block model validation was carried out using visual, statistical and graphical checks between input composite sample data and estimated block grades.

Variogram models were constructed based on composite data and used Ordinary Kriging (OK) as the principal estimation methodology. Inverse Power Distance Cubed (IPD2) was used for validation purposes. The resultant estimated grades were validated against the input composite data and classification in accordance with the guidelines of the JORC Code (2012) was carried out based on an assessment of geological and grade continuity and an assessment of assay data quality. Due to absence of data for definition oxide/primary boundary only Inferred Mineral Resources were classified at Mangazeisky North. Mineral Resources (Table 1.3) were further limited based on an expectation of eventual economic extraction to an optimised open pit shell generated using appropriate economic and technical parameters.

Table 1.3: Mineral Resource Estimate. North Mangazeiskiy Project, Russia. 31st of May 2019 (In Accordance with the Guidelines of the JORC Code (2012)) Potential Open Pit Resources

Ag Cut-off, g/t	Category	Tonnes, Kt	Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
50	Inferred	364.17	695.00	9.02	0.92	253,102	32,848	3,350
100	Inferred	354.94	711.24	9.25	0.94	252,446	32,819	3,335
200	Inferred	331.41	750.15	9.71	0.98	248,612	32,185	3,261
300	Inferred	309.87	784.56	10.20	0.99	243,111	31,604	3,073
400	Inferred	275.53	838.43	10.91	1.08	231,015	30,049	2,978

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. Mineral Resources are not Ore Reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study.
3. Mineral resources include all potential mineable tonnage.
4. Mineral Resources are estimated as of 31 May 2019.
5. Mineral Resources were constrained by conceptual optimum pit contours using NSR of $139.06/t for primary mineralisation.
6. All values in the tables have been rounded with relative accuracy of estimate. Numbers may not compute due to rounding.
7. Mineral Resources were constrained by an optimum pit shell based on the corresponding economic and mining parameters provided by the Client and/or accepted by WAI
8. The Northern Mangazeisky mineral resources were estimated in accordance with the guidelines of the JORC Code (2012) by Steven McRobbie, Independent Competent Person as defined by the JORC Code.
9. This mineral resource estimate is not limited to any factors in terms of environmental, permitting, legal, title, taxation, socio-economic, market and other relevant factors.
10. The metal resources include all the in-situ metal disregard the metallurgical recovery factor.

1.3 Hydrological & Hydrogeological Review

The Mangazeisky open pit, located in an interfluve area between creeks, is likely to encounter frozen groundwater and receive negligible groundwater inflow. Dewatering and drainage within the pit, using sump and perimeter collectors should be designed for a peak event representing a combined spring thaw and design storm event i.e., 1 in 100 year.

The southern end of the Vertikalny deposit is located on the flanks of the Porfirovy stream valley and this zone represents a different hydrogeological domain from the interfluve areas with much higher groundwater circulation and recharge from surface to depth. This means permafrost is likely to be thinner. Given the 300m depth of underground workings in Vertikalny Zone 1 in particular (south, river flank) and to a lesser extent in Zone 4 (interfluve) it is likely that free-flowing groundwater will be encountered in mid to lower levels of the underground mine. Across most of the underground sections (Zones 2 and 3), it is expected there will be negligible groundwater inflow because of permafrost.

Hydrogeological drilling is required to confirm permafrost conditions in Zones 1 and 4 and form the basis for an inflow model and dewatering plan. The hydrogeological wells should be tested to confirm hydraulic properties in sections using double packers so that isolated zones within and beneath the expected permafrost zones can be characterised. Wells should be drilled and tested throughout the full thickness of the proposed mine i.e., 300m.

Water supply for the mine, via a proposed water supply borehole near borehole GS15-05, should be tested by conducting a long-term pumping test i.e., 28 days and recovery phase to determine the storage and yield characteristics if this is to be used as supply well.

Surface water hydrology and the mine water balance have been reviewed and no particular additional comments over and above what has already been presented by SRK are raised.

1.4 Geotechnical Review

WAI has carried out a review of the geotechnical information provided by Silver Bear Resources (SBR) for the Vertikalny and Mangazeisky North deposits. The review has aimed to summarise the geotechnical parameters for use in mine optimisation and design. Information was drawn from the findings of the geotechnical study carried out by SRK consulting in late 2014. WAI has not carried out a site visit, nor has it carried out an independent review of the geotechnical data used in the SRK study.

1.5 NSR Model

A basic Net Smelter Return (NSR) calculation was performed which considered grade, metal price, metallurgical recovery, and metal payability. The payable metal includes the applicable concentrate and refining charges but does not include price participation or penalty element payments. The metal price assumptions were derived by WAI and approved by SBR. All metallurgical recoveries/costs used in the NSR calculation are based on data provided by SBR.

NSR factors were calculated and directly applied to each block within the Resource block models. This enabled the subsequent mine optimisation exercises to be carried out on the block NSR values. The NSR model forms a critical input into the development of the mining study and further detail regarding the NSR inputs must be understood to enhance the confidence of the study.

1.6 Open Pit Mining

WAI has carried out an open pit mining study to define a mineable tonnage estimate for the Vertikalny and Mangazeisky North deposits.

Open pit optimisation was carried out using the Datamine NPV Scheduler v4 (NPVS) software package. Pit optimisations were carried out on the Resource block models generated for the two deposits and driven on the calculated block NSR values. The optimisations included *Measured, Indicated* and *Inferred* resources.

Detailed mine designs were generated from the selected optimal shells using the Datamine Studio OP V2.4 general mine planning package. The designs were used to derive the mineable tonnage estimates and formed the basis for subsequent production scheduling.

A summary of the tonnages and grades contained within the Vertikalny and Mangazeisky North pit designs is provided in Table 1.4 below.

Table 1.4: Vertikalny Conceptual Pit Design Physicals (Dilution & Recovery Applied)

Parameter	Units	Vertikalny	Mangazeisky North
Oxide Material	kt	212	-
Ag Grade	*g/t*	800	-
Sulphide Material	kt	116	347
Ag Grade	*g/t*	846	570
Pb Grade	*%*	1.70	7.47
Zn Grade	*%*	1.66	0.82
Total Mineralised Tonnes	kt	329	347
Oxide Material (Below Cut-Off)	kt	45.0	--
Sulphide Material (Below Cut-Off)	kt	29.0	72.2
Waste	kt	11,000	8,540
Strip	t_W:t_O	33.7	24.8
Average NSR	US$/$t_{ore}$	382	245

Note:

- Mining Dilution of 30% and Mining Loss of 5% applied to **all** mineralised material.
- All figures rounded to 3SF, Pb/Zn grades rounded to 2DP
- Oxide material processed through oxide circuit; Pb/Zn are not recovered and are not reported.
- Strip ratio not inclusive of below cut-off material.
- Waste tonnes not inclusive of below cut-off material.
- Figures effective as of 01.06.19

It should be noted that 'minable tonnage estimates' are not Ore Reserves and are not demonstrative of technical and economic viability.

1.7 Underground Mining

WAI has carried out a mining study to define an underground mineable tonnage estimate for the Vertikalny deposit. The study has considered the volume of mineralised material below the generated Vertikalny pit designs.

Underground mineable tonnage estimates were prepared using the Vertikalny Resource block model. Stope optimisation was completed using the Mineable Shape Optimiser (MSO) module in the

Datamine Studio 5D Planner software package. The optimisations included *Measured, Indicated* and *Inferred* resources.

A summary of the tonnages and grades contained within the conceptual underground mine designs is provided in Table 1.5 below.

Table 1.5: Vertikalny Conceptual Underground Design Physicals (Dilution & Recovery Applied)

Parameter	Units	Value
Stope Mineralised Material	kt	609
Ag Grade	*g/t*	*462*
Pb Grade	*%*	*2.16*
Zn Grade	*%*	*1.68*
Development Mineralised Material	kt	232
Ag Grade	*g/t*	*263*
Pb Grade	*%*	*1.37*
Zn Grade	*%*	*1.26*

Note:
- Unplanned Dilution of 10% and Mining Loss of 10% applied to **stope** mineralised material.
- Development mineralised tonnes depleted from stope tonnes.
- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP
- Figures not representative of Ore Reserves (in accordance with JORC 2012)

1.8 Mine Production Schedule & Equipment Requirements

A combined open pit and underground production schedule was generated using the Geovia MineSched V9.2 mine scheduling software package. Effort was made to sequence the operations such that a steady flow of plant feed is maintained over the life-of-mine. Key points noted from the generated production schedule include:

- Overall mine life anticipated at 8 years;
- Mining in the Vertikalny open pit anticipated for completion in Q4 2021;
- Mining at Mangazeisky North anticipated to commence in Q3 2021 with production ceasing in Q3 2023: and,
- Underground pre-production development anticipated to start in Q2 2022 with stope production commencing in Q4 2023.

Open pit and underground mining equipment requirements were estimated on first principles analysis to achieve the generated production schedule. No ventilation studies were carried out for the underground mining operations and it is recommended that such studies be considered in more detailed engineering studies utilising the latest underground resource model.

1.9 Capital and Operating Costs – Mining

A mining cost model was developed to assess the open pit and underground mining capital and operating expenditures for the Mangazeisky Project. The cost estimates were developed by WAI based on data provided by SBR and WAI's internal cost database.

A summary of the costs is presented below:

Open Pit Capital Costs:	**US$2.53M**
Open Pit Operating Costs:	**US$2.17 /$t_{MINED}$**
Underground Capital Costs:	**US$23.33M**
Underground Operating Cost:	**US$40.56/$t_{ORE}$**

Total mining operating cost resulted in US$82.3m (or US$49.5/t ore mined) and capital cost of US$25.86m for both open pit and underground mining operations.

1.10 Mineral Processing

Silver production commenced in April 2018 and silver recovery has steadily improved from approximately 55-60% in 2018 to an average of 70.5% for the nine months to September 2019, although this is still someway off the design recovery for oxide ore of 85%. Silver was previously lost due to poor washing of the tailings filter cake, which has now reportedly been resolved. There is also an ongoing impact on recovery and costs due to primary/transition ore being included in the oxide feed as oxide resources are depleted. Due to SBR concerns with the original direct electrowinning process (high zinc and chloride levels in the feed solution), a Merrill Crowe circuit was constructed in April 2019 which can reportedly operate in parallel with the electrowinning circuit or in series to treat the electrowinning tails solution.

Current process plant throughput is slightly below the design of 110,000tpa (approximately 96,000tpa pro-rata from the September YTD number of 71,769t). The actual May 2019 YTD process operating cost reviewed was $74.9/t, significantly higher than the design of $47.9/t. This is mostly due to the impact of transition/sulphide ore in the feed blend with higher reagent consumptions, low activity lime and an incorrect design lime consumption of only 0.7kg/t used in the original feasibility study, compared to the testwork data of 20-30kg/t.

For the proposed processing of primary sulphide ore, a new flotation circuit is required for production of separate lead and zinc concentrates, with cyanide leaching of the lead flotation middlings as per the current plant. The annual throughput through the new flotation plant will also be increased to 180,000tpa. The capital cost for a brand-new plant of approximately $17.3M is considered reasonable, although this reduces to approximately $9M if the existing oxide circuit is used and the additional equipment retro-fitted (such as the flotation plant and additional crushing and grinding capacity for the higher throughput). The new plant is scheduled to be commissioned in June 2021 and, until then, the sulphide ore will be processed through the current plant with impact on recovery and costs.

The recoveries used in the optimisation and conceptual design studies are based on the ESTAGeo testwork results, with silver, lead and zinc recoveries of 85.4%, 65.9% and 82.2% respectively. Based on these results, the zinc concentrate at 42.4% Zn is considered to be saleable based on typical western smelter contracts. The lead concentrate at only 17.1% Pb is very low grade, but high in silver value at 10,215g/t Ag, according to the testwork results. This is therefore assumed to be most likely saleable to an Asian smelter.

The NSR terms for both concentrates have been provided by SBR for use in the pit optimisation studies (84% and 45% respectively for the lead and zinc concentrates).

The process operating cost for primary ore using the new flotation circuit has been estimated by SBR as US$46.3/t and is considered reasonable for use in the pit optimisation studies. This compares with the Tetra Tech design operating cost of US$121.8/t based on using the existing oxide plant (no flotation circuit), but with modifications for finer grinding, higher cyanide levels and additional leach residence time.

SBR has conducted ore sorter testwork on samples of oxide ore from current production. Based on these results, the current schedule assumes that approximately 270ktpa of ore will be mined with 180,000ktpa reporting to the flotation plant after crushing and ore sorting with 99% recovery of Ag, Pb and Zn to the flotation feed. This applies to both oxide and sulphide ore. The ore sorter is scheduled to be commissioned in April 2020.

1.11 Capital and Operating Costs – Processing

Total processing operating cost is estimated as US$68.3M. A summary of processing operating costs is shown in Table 1.6 below.

Table 1.6: Project Processing Opex Summary		
Ore Sorting Cost	US$ /t	2.25
Leach Plant (Current Plant)		
Unit Processing Cost (Oxides)	US$ /t	72.95
Unit Processing Cost (Sulphides)	US$ /t	123.71
Flotation Plant (New Plant)		
Unit Processing Cost (Sulphides)	US$ /t	47.18

Processing capital costs for construction of the new flotation plant have been estimated at US$17.3M. However, as most of required equipment is currently installed on the existing plant, the outstanding amount of capital costs has been estimated at approximately US$9.2M. In addition, US$2M has been allocated for the XRT sorter section.

1.12 Financial Analysis

Preliminary Economic Assessment of the Mangazeisky project has resulted in a positive NPV at various discount rates. The Project is mostly sensitive to changes in Silver prices. Break-even price of the Project has been estimated at US$14.11/oz, which is 21% lower than the base case silver price assumption.

Base case NPV @8.64% was estimated at US$46.51M (nominal values).

The financial analysis has been performed to reflect valuation as of the end of 2019 and does not include any sunk costs that have been previously invested in the project.

Overall capital cost of the project has been estimated at US$43M, and total operating costs of US$242.7M. The key project performance is shown in Table 1.7 below.

Table 1.7: Financial Project Summary

NPV @ Discount Rate of 8.64%	US$ M	46.51
Ag Break-even price	US$/oz	14.11
NPV @ Discount Rate of 10%	US$ M	43.87
NPV @ Discount Rate of 15%	US$ M	35.77
NPV @ Discount Rate of 20%	US$ M	29.60
IRR	%	N/A
Payback period of capital (Discounted, Cumulative)	date	Q3 2021

Current financial results have been derived from the production schedule that considers oxide material from stockpile No 5 to the amount of approximately 50kt.

2 INTRODUCTION (ITEM 2)

2.1 Terms of Reference and Reporting Aims

Silver Bear Resources plc (SBR) is listed on the Toronto Stock Exchange (TSX:SBR) and is the 100% owner of the 570km^2 Mangazeisky exploration licence containing the Vertikalny silver mine concession in the Republic of Sakha (Yakutia). Silver Bear was granted a 20-year Mining Licence for the Vertikalny deposit in September 2013 with first silver production on stream after commissioning in April 2018 stepping up to commercial production in July 2019. The current processing facility is set to be upgraded including new sorting facilities installed by June 2020. The Mangazeisky EL is valid until 2023.

This report was prepared as a National Instrument 43-101 (NI 43-101) Technical Report for Silver Bear Resources plc (SBR) by Wardell Armstrong International (Russia) Ltd. (WAI). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in WAI's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by SBR subject to the terms and conditions of its contract with WAI and relevant securities legislation. The contract permits SBR to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with SBR. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

The aims of this report are to:

- Provide an updated mineral resource estimate and a classification of resources in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, November 27, 2010 (CIM);
- Based on the updated resource estimate provide a Scoping Study level integrated mine design and schedule for the Vertikalny and North Mangazeisky open pits including transition to future Vertikalny underground production;
- Tailor the mine design and schedule to the increased production rates expected through upgrade to the sulphide process facility and installation of a new ore sorting system;
- Assess risks and opportunities arising from the plan for development.

In accordance with Article 7.1(1) (b) of Form 43-101F1 (2011) given that the Client has its properties as the subject of this report in a foreign jurisdiction, WAI has elected to report mineral resources according to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2012)].

2.2 Qualifications of Consultants

The Consultants preparing this technical report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in SBR. The Consultants are not insiders, associates, or affiliates of SBR. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between SBR and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individual, by virtue of his education, experience and professional association, is considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and is a member in good standing of appropriate professional institutions. The QPs is responsible for specific sections as follows:

- Steven McRobbie, Regional Director, Russia is the QP responsible for Sections 9, 10 and 12.

2.2.1 Details of Inspection

WAI consultants have not conducted a site visit to the Vertikalny Minesite nor Mangazeisky exploration area at the time of writing this report. WAI has had a history of involvement in the project since early 2018. It has not been possible to access the site due to international, regional and HSE policies of the operational site, namely:

- A ban on foreign citizens entering the Russian Federation for ordinary travel purposes since March 2020;
- A suspension of direct flights between UK and Russia for specific travel purposes since December 22, 2020 making travel for UK Citizens not resident in Russia for repatriation or emergency purposes only;
- Regional restrictions and a 14-day quarantine enforced by regional authorities in Yakutia for any citizens arriving from outside of the region for much of 2020;
- Operational policy of SBR insisting on a Covid negative test followed by a period of isolation for 10-14 days at a designated hotel in Yakutsk prior to any visitors or personnel travelling on the site.

The QP has examined flyover footage of the site area and videos/photographs of specific installations and areas such as the processing plant, open pit areas and stockpiles. This report is therefore prepared in lieu of a recent site inspection. Once international travel restrictions are lifted, expected in April 2021, a site inspection will be carried out by the QP.

2.3 Reliance on Other Experts (ITEM 3)

The Consultant's opinion contained herein is based on information provided to the Consultants by SBR Corporate in Moscow and Management at Vertikalny Minesite throughout the course of the investigations. WAI has relied upon the work of other consultants in the project areas in support of this Technical Report. The sources of information include data and reports supplied by SBR personnel as well as documents referenced in Section 22.

Historic information provided to WAI and used to prepare this report was acquired by SBR from a variety of sources that have had access to geologic, metallurgical, environmental and engineering studies and from predecessor companies. The predecessor company includes JSC Yanageologia.

2.3.1 Sources of Information and Extent of Reliance

Supporting information has been sourced from company reports generated by Tetra Tech, Hatch, ESTAGEO, Irgiredmet and from WAI's own archive. These documents are referenced in Section 22. WAI has not conducted any legal due diligence with regard to land tenure and ownership but has relied on documents and communications provided by SBR as issued by the Department of Subsoil Use for the Republic of Sakha in its technical review of land ownership and mineral tenure.

WAI also received historical information from maps, longitudinal and cross sections, data tables and documents prepared by SBR for statutory reporting (TEOs, 5G Reports, etc.). Documents used in the preparation of this report are assumed by the authors as accurate and complete in all aspects. Mineral title due diligence, Russian legal and regulatory compliance, and nature and extent of underlying agreements was not conducted by WAI. The authors rely on legal information provided by SBR and its subsidiaries, as well as documentation from the Russian Federal and Regional authorities presented in this report.

WAI has reviewed assay and geological results from SBR diamond drilling, trenching and reverse circulation drill campaigns conducted between 2009 and 2019. Assay and geological results represent:

- Vertikalny: A total of 304 diamond holes drilled for a running total of 44,060m and 210 grade control trenches. Maximum hole depth was 496m;
- North Mangazeisky: A total of 157 diamond holes drilled and 50 exploration trenches. Maximum hole depth was 122m.

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

2.4 Effective Date

The effective date for issue of this report is 25 March 2021. The effective date for reliance of information contained in this report is 28 May 2020 as no data or material information used in its compilation was considered after this date.

2.5 Terms and Units of Measurement

All currency amounts are stated in US dollars or Russian Rubles (₽) unless otherwise specified. The units of measure presented in this report are metric units except for bullion prices which are quoted in troy ounces (toz). Silver values are reported in in grams per tonne (g/t) or parts per million (ppm), respectively. Gold is also reported in grams per tonne (g/t). Tonnage is reported as metric tonnes (t), unless otherwise specified.

Mangazeisky is also referred in the literature as ‘Mangazeyskiy’ or Endybal

3 PROPERTY DESCRIPTION AND LOCATION (ITEM 4)

Information from this section is drawn from Tetra Tech (2017) and reliant thereupon the accuracy of this information.

3.1 Property Description and Location

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 centroid of which at approximately 65°40' south and 130°07' east. It lies approximately 400 km north of Yakutsk, capital city of the Sakha Republic, 300 km southwest of Batagai and approximately 230 km north of Sangary, a river port on the right bank of the Lena River (Figure 3.1)

Figure 3.1: Property Location Map (after Tetra Tech, 2017)

3.2 Licence Tenure

Silver Bear holds the mineral rights to the Property through its 100% interest in ZAO Prognoz. Silver Bear purchased ZAO Prognoz in 2004 from the National Resource Company. 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 3.1 as well as the surrounding Exploration License (Table 3.2).

Table 3.1: Mining License Coordinates		
Mining Licence YaKU 03626 BE		
Corner no	Northing Coordinate	Easting Coordinate
1	65˚41’15.917”	130˚01’55.381”
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˚36’46.221”	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”

Table 3.2: Exploration License Coordinates		
Mining Licence YaKU 03626 BE		
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”

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. Three extensions were granted until 31st December 2016. WAI understands that a further seven-year extension was granted until December 2023 with no minimum expenditure commitments.

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. 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 15,000m of drilling and 15,000m3 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 180,000tpa by the year 2023.

A summary of the terms of the licence agreements is presented in Table 3.3 below.

Table 3.3: Licence Details

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	28 September 2004	31 December 2023	150,242
Vertikalny Deposit	YaKU 03626 BE	Licence to Use Subsoil	13.55	31 August 2013	1 September 2033	110,771

3.3 Royalties, Agreements and Encumbrances

On 21st October 2004, Silver Bear completed an acquisition of all of the outstanding shares of ZAO 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 ZAO 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.

WAI is not aware of any liability in the form of royalties, financial encumbrances or any other debts/liabilities relating to other commercial activities carried out on the licence area but these may be applicable.

3.4 Environmental Liabilities and Permitting

WAI is not aware of any existing liabilities arising from previous industrial activity and land use and it is not part of the scope of this study to investigate historical impacts caused by project activities to date.

Baseline studies to fulfil environmental requirements for exploration activities revealed that concentrations of minerals in some surface water and sediment samples did exceed local regulatory standards in some cases, which were attributed to natural weathering processes across the Project affecting regional watersheds and to exploration activities in local waterways near the Vertikalny deposit area. It is assumed that the legacy of such emissions have been addressed where possible during exploration work and incorporated into the Environmental OVOS.

4 ACCESSIBILITY, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY (ITEM 5)

Information from this section is drawn from Tetra Tech (2017) and reliant thereupon the accuracy of this information.

Support for infrastructure development of Vertikalny was potentially available from the Regional Government of Yakutia as part of its “Scheme of Complex Development of Productive Forces, Transport and Power Industry of the Sakha Republic [Yakutia] by 2020”. WAI has not undertaken any investigation into tax breaks or other incentives available or taken up by SBR during development of Vertikalny.

4.1 Physiography

The Property lies in a mountainous region with elevations ranging from 800 to 1,400masl. The main ridges have steep slopes (25 to 30° and rounded crests that are 200 to 500 above the valley floors). The vegetation surrounding the Property is composed of ‘Taiga’ - primarily aspen, birch and fir trees in the lower parts of the valleys.

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. The nearest weather station to site is located at Verhojansk (National Oceanic and Atmospheric Administration (NOAA) Station ID RA24266; 67°33' North, 133°23' East, 137m). 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. Snow cover is formed around the end of September until mid-May. The area is subject to permafrost to 400m depth with seasonal thaw during the summer of the top 0.5-15m depth.

4.2 Operating Season

Operations and exploration occur all the year round. The exploration field season runs from May to October though drilling is carried out over the winter season when swampy Taiga is frozen.

4.3 Sufficiency of Surface Rights

SBR has industrial surface rights to carry out mining activities and construction on Vertikalny and right of access over Mangazeisky EL. WAI has not conducted an audit as to whether SBR has all the required permissions nor that permits are up to date and not in violation. WAI is also not aware of any third-party commercial rights over the property or any access rights to indigenous populations and activities. WAI has also not carried out any auditing of surface rights or mineral tenure as part of its scope and is not aware of any overlapping licences/resources for precious and base metals, industrial minerals or water resources owned by 3rd parties or on the State Reserves Balance. Local artisanal and alluvial operations (“artels”) may be active.

4.4 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 10km from the base camp. A flight by AN2 aircraft is typically two hours.

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

There is also a winter road for transport of all freight and supplies to the Property.

4.5 Infrastructure

4.5.1 Transport

The Project area is isolated and can be accessed by a winter road that is usable from mid-January until mid-April. Seven tonne all-terrain vehicles (ATVs) are used for transporting workers and materials to site. The main haul route runs north-south 370km to the port of Batamai on the Lena River then on an all-weather road an additional 200km down the Lena Valley to Yakutsk. The Lena River is navigable for barges up to 3,000t to Batamai and Sangar from June to September though there is no road access to the Property from May to December.

Regional airports are located at Sangar and Batagai, located 230km SW and 300km NE of the site respectively. During most of the year the Property is accessible primarily by helicopter or light fixed wing aircraft from Yakutsk, Batagai, or Sangar. Currently, AN-2 and AN-3 fixed wing aircraft are being used for small loads (800 to 900kg); MI-8 MTV and MI-26 helicopters are available for heavier loads (up to 1,800kg).

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.

4.5.2 Power

There is no access to the main power grid on the Property. Local supply with a capacity of 16MW comes from 12 diesel generating sites. The nearest power generator set to the Project site is at Sebyan-Kuel (375kW). It is planned by 2020 that the electrical generating capacity of Yakutia will be supplemented with a further 8,500MW from seven new power stations. The current status of connecting to this new grid is not known at the time of writing.

4.5.3 Water

Potential water sources include the Arkachan River located 10km from the Project, and the Endybal, Sirelendge, Fedor-Yuryage, and Mangazeisky creeks, which flow through the licence area. WAI understands that water resources have been developed through recent underground exploration and development and that Silver Bear has been working with regulatory authorities (YakutNedra) to put water resources on the State Balance and obtain relevant permitting for extraction for both process and potable water.

4.5.4 Labour

Given the relatively isolated location of the Property use of local resources is limited. There is no pool of local labour and all staff work on a rotational basis from Yakutsk and other parts of Russia. A regional administrative and support office is maintained in Yakutsk. Currently there is a compliment staff working on shift on site and additional staff supporting from Yakutsk. The site compliment of staff is expected to increase to accommodate construction and commissioning staff in 2020.

4.5.5 On-Site Infrastructure

The permanent camp, Hogan Camp, is comprised of one to two room cabins, huts and accommodation containers. There are several permanent structures for kitchen, ablution, warehousing and maintenance, and offices for mine and process administration. There are also buildings for core logging and sampling, sample preparation and sample storage, as well as sheltered core box storage.

5 HISTORY AND PREVIOUS WORK (ITEM 6)

Information from this section before 2016 is drawn from Tetra Tech (2017) and reliant thereupon the accuracy of this information.

The Deposit was initially discovered by Russian Cossacks in 1764. Soviet-era prospecting occurred during 1952 and 1953 and work focused on the Mikhailovsky and Kuzminsky zones, which are located 7.5km and 10km to the north of Vertikalny, respectively. This work included geological mapping (1:50,000), trenching, sampling, and the establishment of two short adits (32m) beneath the trenches. Work also included a topographic survey (1:2,000, 3km^2) and an induced polarisation (IP) survey (1:5,000, 1.7km^2). By 1960, the exploration work completed in the licence area had identified more than 160 anomalies within a north-south trend up to 20km in length. This trend is 2km wide in the north (Nuektame River) and up to 4.5 to 5.0km wide in the south (Endybal River).

In 1989, systematic prospecting and exploration resumed. From 1991 to 2003 JSC Yangeologia completed 151,452m^3 of trenching, 10.2-line kms of magnetic surveys, detailed geological mapping, soil geochemical surveys, and 10 diamond drillholes totalling 1,303m. 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, outlining over 30 mineralised structures containing potentially economic grades.

After the Russian Financial Crisis of 1998, the early 2000s experienced a rapid rise in foreign investment and the development of silver deposits in Far East Russia at Goltsovoye, Dukat with Pan American Silver, and acquisition of ZAO Prognoz by SBR in 2004. Metallurgical testwork was conducted on two samples and reported by Western Services (2004).

An historical Russian inventory of reserves and resources was compiled in 2000 and reviewed by JSC Yangeologia. NI 43-101 compliant estimates were produced 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).

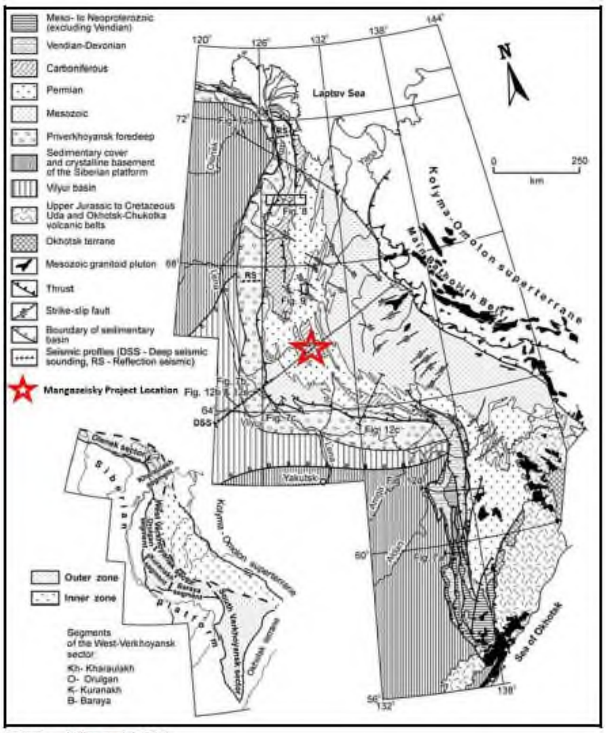
In September 2013 SBR was granted a 20-year Mining Licence for the Vertikalny deposit. Construction on Vertikalny commenced in early 2016 and first silver production was achieved on commissioning in April 2018. As of December 31 2018 a total of 594,921 ounces of silver was produced with sales of 433,095 ounces of silver totalling pre-commercial production revenue of US$6.4 million.

6 GEOLOGY AND MINERALISATION (ITEM 7)

Information from this section is drawn from Tetra Tech (2017) and reliant thereupon the accuracy of this information.

The Mangazeisky Exploration Licence area is located within the Verkhoyansk mobile belt of northeastern 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 terrains, which form most of the Russian Far East.

The belt extends for 2,000km from the Laptev Sea to the Sea of Okhotsk (Figure 6.1). The belt is made up of a rock package that is greater than seven km in thickness and is comprised of Late Precambrian to Triassic rocks deposited along the paleo-Pacific margin of the Siberian Craton. This margin developed because of rifting events which occurred in the Late Precambrian and again during the Late Devonian to Early Mississippian periods. Deformation events during the Late Jurassic to Early Cretaceous periods were accompanied by low-grade metamorphism in the internal parts of the belt and the emplacement of high-level granitic bodies. During the Tertiary period, strike-slip faulting occurred within the fold-and-thrust belt. The central part of the belt is dominated by a thick monotonous succession of Carboniferous and Permian turbidites which are metamorphosed to lower greenschist grade. Granodiorite and granite plutons intrude the core of the range and are associated with extensive precious metal-bearing quartz vein systems.



Source: Miller et al. (2002)

Figure 6.1: Regional Geology of the Property (after Tetra Tech, 2017)

At a district scale lithology and structure are dominated by three events influenced by shearing and overthrusting on the Nuektaminsky-Granichny Fault Zones:

1. Proto-mineralised layers of sandstone containing sulphide mineralisation;
2. Structural deformation
3. Intrusion of the Endybal Diatreme.

7 DEPOSIT TYPE (ITEM 8)

Information from this section is drawn from Tetra Tech (2017) and reliant thereupon the accuracy of this information.

The Property contains several explored areas that host more than 100 occurrences of mineralisation concentrated within a 35km long corridor (Figure 7.1).

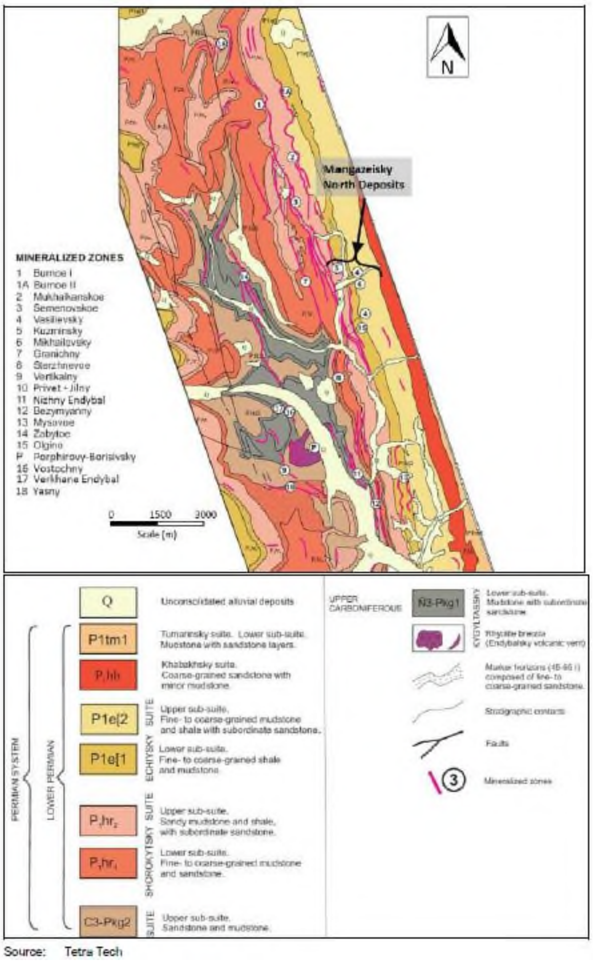


Figure 7.1: Mineralized Zones on the Property (after Tetra Tech, 2017)

Silver mineralization is epigenetic forming in a high-level low-sulphidation environment with meteoric dominated waters fuelled by an underlying porphyry intrusion. The mineralisation on the Property can be broadly classified into four different styles of occurrence:

- Strata-bound silver-bearing, quartz-carbonate-sulphide structures within sandstone with average grades greater than 900g/t silver and lead and zinc by-products. Examples of this are the Vasilievsky—Anglesite-Cerussite and Olgina—Mikhailovsky veins within the Mangazeisky North zone.
- Thick linear-type stockwork areas with carbonate-silver sulphosalt mineralisation. Examples of this occur in the Strezhevoy and Nizhny Endybal Zones.

- Narrow late-stage, steep dipping veins such as Vertikalny that cross-cut stratigraphy and feature grades in excess of 1,000g/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 marginal porphyry area associated with quartz, quartz-carbonate and quartz-sulphide veins and veinlets, hosted by extrusive rhyolite porphyry. Porfirovy is an example of this.

8 EXPLORATION (ITEM 9)

Information from this section is drawn from Tetra Tech (2017) and reliant thereupon the accuracy of this information.

Early exploration by ZAO 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. A summary of non-drilling exploration activities is presented in Table 8.1 below:

Table 8.1: Historic Exploration Activities at the Property (after Tetra Tech, 2017)

Year	Exploration Activities	Targets Explored
2004	No trench exploration was undertaken during 2004	-
2005	9,641m^3 of trenching	Vasilievsky, Milhailovsky, Sterzhnevoy, Nizhny, Endybal
2006	4,843m^3 of trenching and mapping	Nizhny, Endybal Vostochny, Sterzhnevoy, Vertikalny
2007	8,000m^3 of trenching	Vertikalny
2008	22,633m^3 of trenching. Mapping, lithochemical sampling, direct current induced polarisation/magnetotellurics and magnetic anomaly geophysical surveys.	Vertikalny, Zabyty, Zabyty-2, Kis-Kuel, Orogondia
2009	15,067m^3 of trenching. Lithochemical sampling, magnetic anomaly mapping	Nizhny, Endybal, Vertikalny, Kis-Kuel, Mukhalkan-Burney
2010	No exploration was undertaken in 2010.	-
2011-2012	1,600m^3 of trenching	Nizny, Endybal
2013	52 trenches at regular intervals with 474m of sampling	Magazeisky North and South
2014	19 trenches across multiple exploration targets	Vertikalny, Mangazeisky South, Porfirovy and Sterzhnevoy
2015	8 trenches for a total length of 593m	Porfirovy and Sterzhnevoy

9 DRILLING (ITEM 10)

Information from this section on programs before 2016 is drawn from Tetra Tech (2017) and reliant thereupon the accuracy of this information.

A total of 304 diamond holes have been drilled and considered for evaluation for a running total of 44,060m. The main drill campaigns at Vertikalny took place in 2005-2015, with no drilling in 2010, and consisted of diamond core drilling only. No Soviet-era drilling was considered for the evaluation.

In the majority of drillholes, the core was oriented at the commencement of every run to allow structural measurements to be made and all holes are subject to down-hole survey at generally 20.0m intervals. Data from HQ (63.5mm) and NQ (47.6mm) wireline diamond drillholes is used for interpretation and grade estimation. The predominate drilling diameter was of HQ size.

A total of 16 metallurgical holes for a running total of 2,786 l.m. were drilled either PQ or HQ diameter for technological testwork and ore-type definition in 2017.

A total of 19 advance grade control holes for a running 535m were drilled in 2018.

A total of 233 trenches for a running total of 5,667.87l.m. were sampled for a grade control in 2018-2019. The trenches have 10m spacing on each bench with the bench height of 5m. The grade control samples were collected from 5 benches with elevation from 1175m through to 1155m. This campaign was carried out at the Central part of Vertikalny.

WAI is not aware of any specific measures taken to reduce losses through drilling or that any drilling campaign suffered from poor recovery. Diamond drill recovery averages approximately 95% and are considered homogenous and acceptable for evaluation. No apparent relationship has been observed between sample recovery and grade.

10 SAMPLE PREPARATION, ANALYSIS AND SECURITY (ITEM 11)

A commentary on compliance relating to this section is presented in Section 1 of Appendix A in this report.

Prior to 2007 the sample preparation, analyses and security was conducted according to Russian State 'Gostandarts'. Since 2005, sampling has been carried out under SBR's Standard Operational Procedures using a combination of diamond core drillholes and surface trench channel samples.

10.1 Methodology

Diamond drilling was used to obtain predominantly 1.0m samples (minimum length 0.25m to a maximum of 3.00m) that were subsequently cut in half along its long axis, with half core used for primary analysis and the other half retained for reference purposes, to produce half core for sample preparation (crushing/pulverising) and a final sub-sample for laboratory analysis. Trenching was used to obtain predominately 1.0m samples (minimum length 0.10m to a maximum of 2.00m) cut by portable diamond saw and collected using hammer and chisel. The entire sample was taken for sample preparation (crushing/pulverising) to produce a final sub-sample for laboratory analysis.

Grade control (carried out from October 2018 to July 2019) sampling methods were not assessed as part of this study.

WAI understands sampling of dump stockpiles (six stockpiles in total) were taken at random mechanically from each 30t bucket at a temporary weighbridge facility where weight and moisture content were also measured. Four grab samples were taken of approximately 8kg each, representing 1 per mil of the load. Each sample was prepared and assayed according to RF protocol GOST 14180-80 "Ores and concentrates of non-ferrous metals. Methods of sampling and preparation of samples for chemical analysis and determination of moisture".

10.2 Security

Samples were transported to site sample preparation facilities. After preparation in the field, samples were packed into sealed bags and dispatched to the freight forwarders directly by the Company for dispatch direct to the laboratory. The laboratory is obliged to report on discrepancies in the state of the sample when checked in on arrival as part of its LIMS protocol.

The sample preparation facility, state of security and the laboratory has not been inspected by WAI at the time of writing this report.

10.3 Sample Preparation

Sample preparation for Vertikalny was carried out on site. The sample preparation flowsheet comprised:

- Two stage crushing to 85% passing 1mm;
- Split to 1kg sample;
- Submit for futher analysis.

Prior 2011 final milling and pulverising to 85% passing 75µm was carried out in Chemical Laboratory of State Enterprise Aldangeologia in Aldan (Russia) and later in ALS Chemex in Chita, Russia.

WAI is satisfied that sub-sampling quality control has been maintained through use of company SOP's being adopted to ensure consistency by following a standard set of practices throughout the process.

10.4 Quality Control Procedures

10.4.1 Introduction

Quality assurance and quality control (QA/QC) are the key components to verify the validity of sample collection, security, preparation, and analytical methods. The aim of the QA/QC programme is to quantify and monitor any errors and to provide information that might be used to improve sampling and analytical procedures in order to minimise any errors. A comprehensive QA/QC programme should monitor the accuracy, precision and contamination of each step through exploration from the sampling through the final assay value produced by the laboratory.

QA/QC programmes over the various exploration periods at Vertikalnoye have incorporated the inclusion of duplicate samples, certified reference materials, and blank samples inserted at differing ratios into the sample stream. The results of WAI analysis are summarised below.

10.4.2 WAI Procedures

For duplicate sample sets, the precision can be discussed in terms of the following statistical measures applied by WAI.

- Summary Statistics showing the mean, mode, standard error, range and standard deviation can be indictors if the data sets are in agreement.
- Rank HARD Plot which is the ranked half absolute relative difference, ranks all assay pairs in terms of precision levels measured as half of the absolute relative difference from the mean of the assay pairs (HARD), used to visualise relative precision levels and to determine the percentage of the assay pairs population occurring at a certain precision level (10%). Duplicates on Vertikalnoye include second core halves and/or repeatedly taken channel samples (so called field duplicates). In this case precision for 70% of samples should be within 10%. It should be noted that as the HARD statistic uses and absolute difference, a ranked HARD plot does not revel bias in duplicate data, only the relative magnitude of differences (i.e. precision). The HARD values are sorted from lowest to highest and ranked accordingly, with the rank expressed as a percentage. The ranked HARD plot is then generated by plotting the percent rank on

the X-axis against the HARD value on the Y-axis. A rank HARD plot is constructed that enables quick identification of the percentage of the sample pairs with a HARD value less than 10%.

- Correlation Plot is a simple plot of the value of the duplicate samples, assay 1 against assay 2. This plot allows an overall visualisation of precision and bias over selected grade ranges. Correlation coefficients are also good indicators to quantify the agreement between data sets. A correlation greater than 0.9 is generally described as strong, whereas a correlation less than 0.6 is generally described as weak.
- Thompson and Howarth Plot showing the mean relative percentage error of grouped assay pairs across the entire grade range, used to visualise precision levels by comparing against given control lines.

For certified reference materials (CRM), control charts such as Shewhart X (average) and R (range) charts are constructed for each element standard. The control charts plot process variability, with metal content on the Y-axis and sample number on the X-axis. The plotting of data on charts of this type allows for the easy recognition of samples that fall outside of the action limits applicable for each standard used. Warning and control limits are established at mean ±2 and ±3 standard deviation limits respectively. Any analysis beyond the ±3 standard deviation limit is considered as a failure.

10.5 Quality Control Analysis - Vertikalny

10.5.1 Exploration 2009 – 2019

During exploration activities in 2009-2019 (including samples from grade control trenches) blank samples and certified reference materials (CRM) were employed for QA/QC purposes, field duplicates of samples were used for internal control. Project geologists are in charge of control samples insertion into the samples stream. Field duplicates and blank samples were inserted before crushing, and CRMs were inserted after samples are ground, labelled and registered in a log.

10.5.1.1 Blanks

Barren material of host rocks was used as blank samples. It was reported that blank samples were inserted at 1:20 rate, CRMs – at 1:20 rate, and duplicates were also inserted at 1:20 rate. At the time of this report a total of 25,470 samples have been analysed and provided for review and the quality control samples provided consist of analysis for 985 internal reference materials (3.1%), 942 field duplicate samples (3.7%) and 1,152 blank samples (4.5%).

The results of the blank analysis for Ag are shown in Figure 10.1 with 123 samples showing marginal fails of >5.0g/t Ag with a maximum value of 290.5g/t.

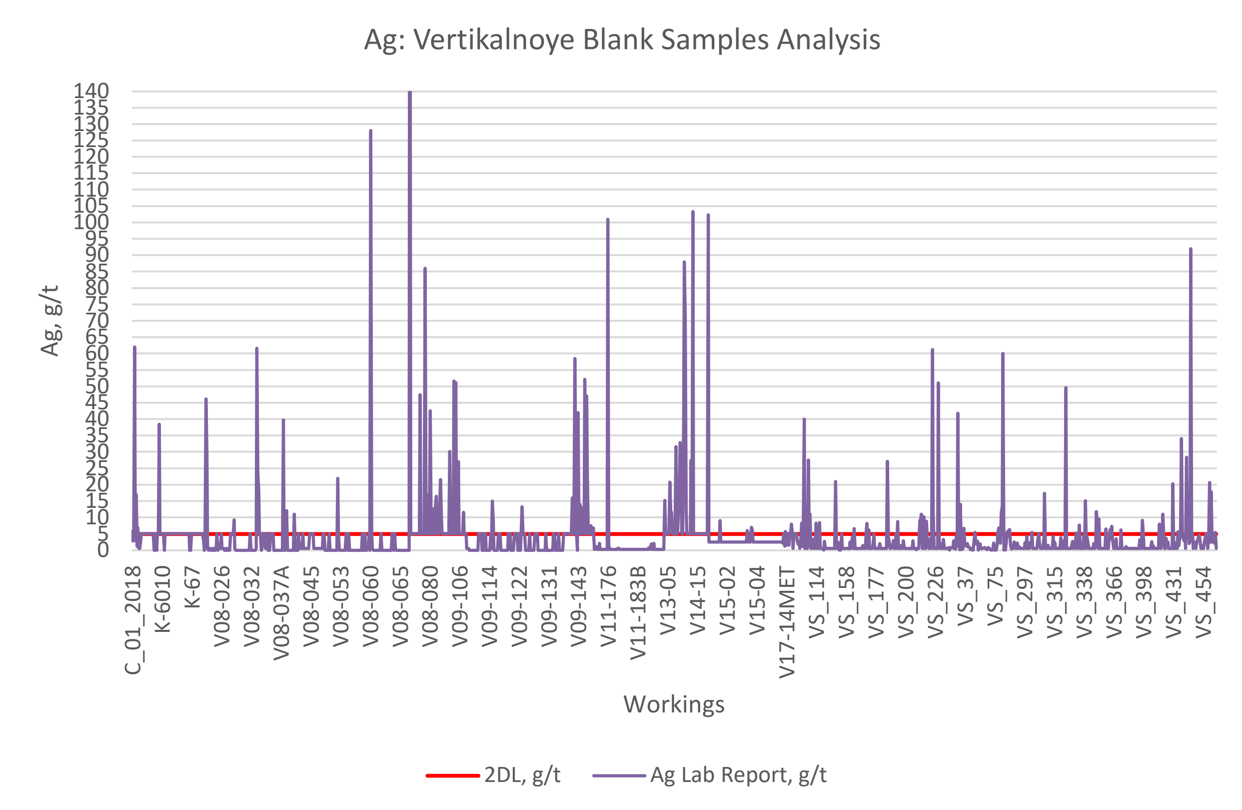


Figure 10.1: Blank Samples Analysed for Ag on Vertikalnoye

Out of 123 blank samples with overestimated grade, 17 samples had grade greater than critical 50g/t – COG for mineralisation delineation. Out of these 17 samples with grade >50g/t, 14 samples were from the intervals involved in Mineral Resource Estimate.

In the majority of cases, blank samples with grade >50g/t Ag are preceded by stream samples with high (and/or very high) grades – see Table 10.1.

Table 10.1: An Example of Blank Sample in the Interval of High Grade Stream Samples, Ag

Site	BH	From	To	Sample	Type of Sample	Ag Grade, g/t
Vertikalny	V08-066	111	111.9	21487	Core	145.5
Vertikalny	V08-066	111.9	113	21488	Core	1645.0
Vertikalny	V08-066	113	113.9	21489	Core	1439.5
Vertikalny	V08-066			21490	Blank	290.5
Vertikalny	V08-066	113.9	115.2	21491	Core	108.0

Blank samples data are summarised in Table 10.2. Contamination of blanks by previous sample material with high Ag grade occurs for 11% of mineralised intersections.

Table 10.2: Blank Samples Summary for Vertikalnoye

Indicator	Number of Samples	% of Total Number	% of Blanks in Mineralised Intersections	% of Blanks in Mineralised Intersections with Blanks
Total number of samples	25,470	100%		
Total number of blanks	1,152	5%		
Total number of samples in mineralised intersections	2,056	8%		
Total number of mineralised intersections (Ag>50g/t)	486			
Mineralised intersections with blanks:	131	0.51%	27%	
including:				
Blanks with Ag grade >5 g/t	57	0.22%	12%	44%
Blanks with Ag grade <5 g/t	74	0.29%	15%	56%
Blanks with Ag grade >50 g/t	14	0.05%	3%	11%

More than 10% of blanks in mineralised intersection showed a significant (>50g/t) Ag grade and this may pose a serious risk to the MRE.

Pb and Zn were detected in 467 blank samples. Out of them 55 samples returned Pb grade that was twice the accepted detection limit (0.02% Pb), and only 8 samples out of these 55 had Pb grade >0.25%. The results of blank samples analysis for Pb are presented on Figure 10.2.

In assays for Zn, 90 sampled returned Zn grade that was twice the accepted detection limit (0.02%Zn), 20 samples out of them had Zn grade >0.25%. The results of blank samples analysis for Zn are presented on Figure 10.3.

In general, the results of blank samples analysis for Pb and Zn might be considered satisfactory.

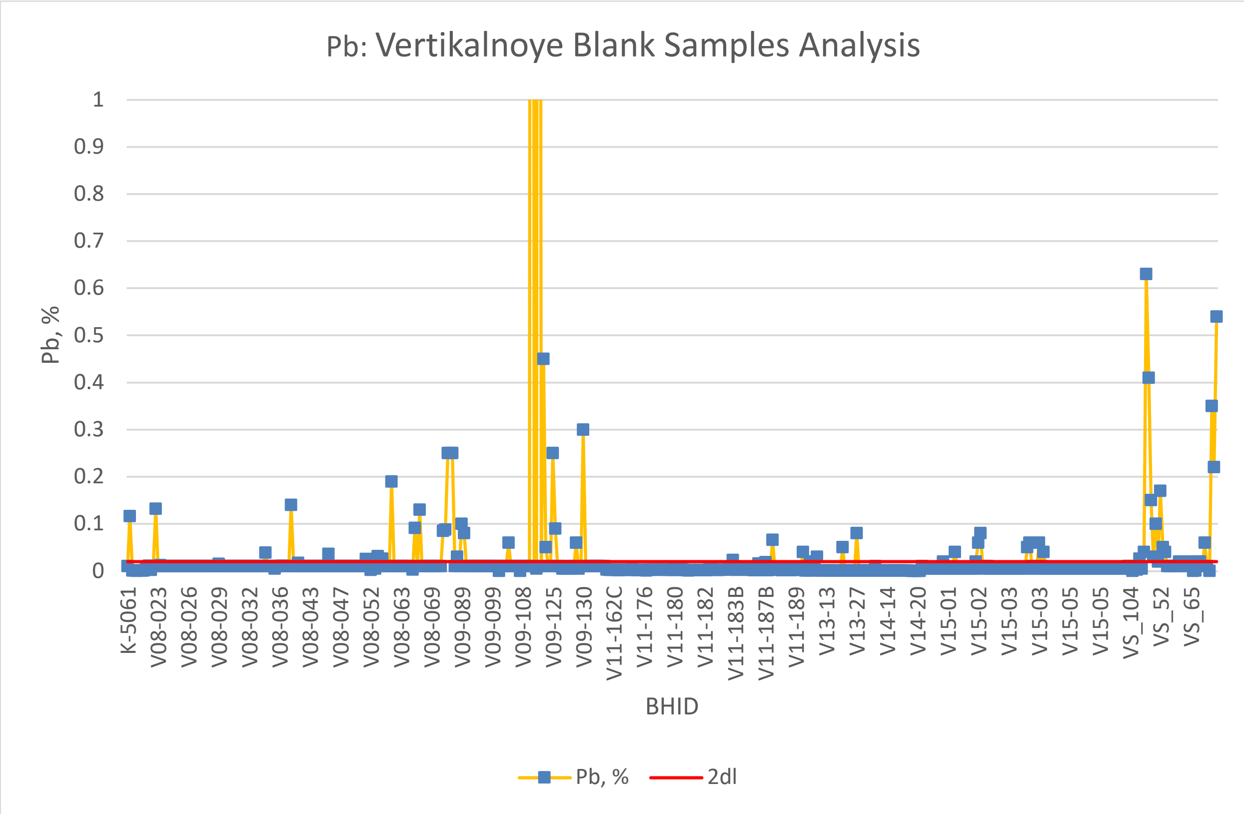


Figure 10.2: Blank Samples Analysed for Pb on Vertikalnoye

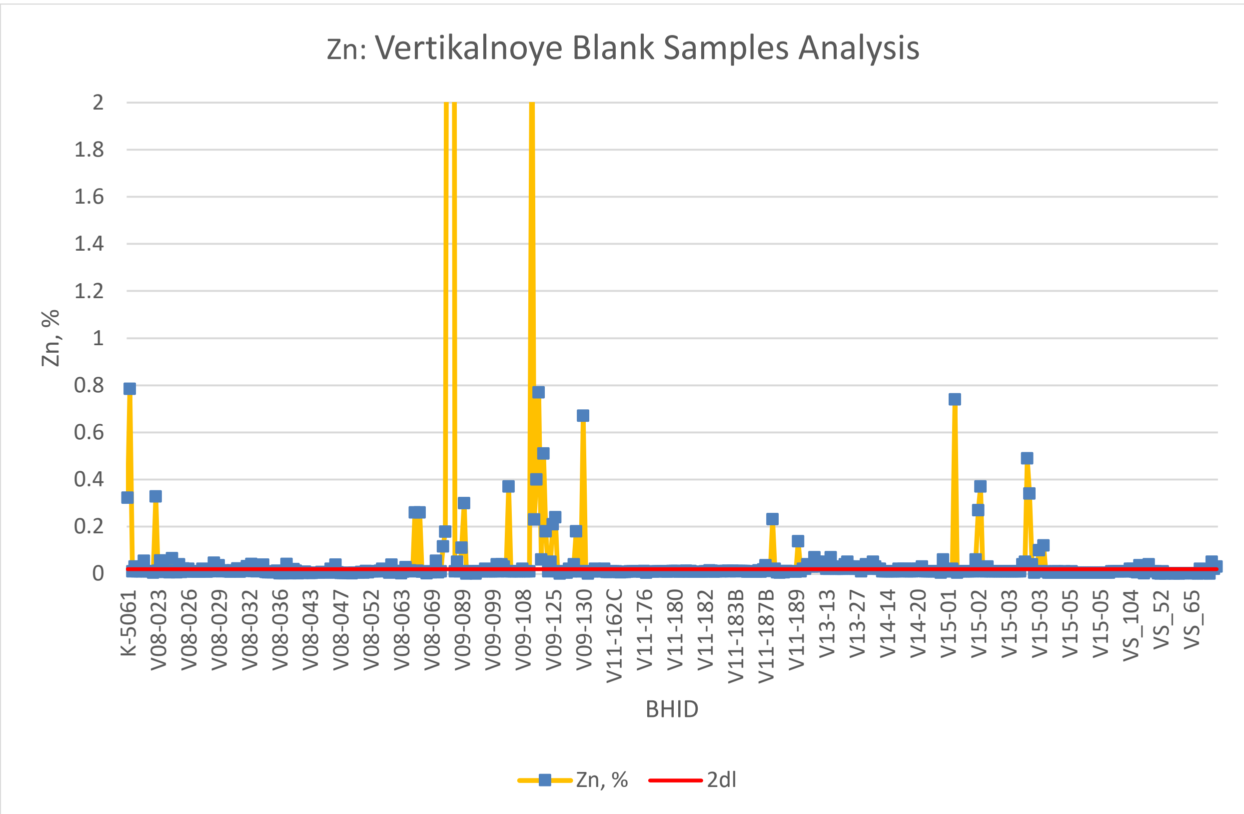


Figure 10.3: Blank Samples Analysed for Zn on Vertikalnoye

10.5.1.2 Certified Reference Materials (CRM)

Eighteen certified reference materials (CRMs) sourced from ORE Research & Exploration Pty Ltd, GEOSTATS Pty Ltd (Australia), STC Minstandard of St Petersburg, and Irgiredmet OJSC of Irkutsk (Table 10.3).

Table 10.3: List of Certified Reference Materials

№№	CRM	Manufacturer
1	OREAS 600	ORE Research & Exploration Pty Ltd, Australia
2	OREAS 605	
3	GBM 906-6	GEOSTATS Pty Ltd, Australia
4	GBM 913-13	
5	GBM 998-9	
6	GBM303-1	
7	GBM310-16	
8	GBM906-7	
9	GBM909-11	
10	GBM913-13	
11	GBM997-4	
12	СОП 01-2016 (SOP 01-2016)	Irgiredmet OJSC
13	СОП 02-2016 (SOP 02-2016)	
14	СОП 03-2016 (SOP 03-2016)	
15	MST SG 130i	STC Minstandard LLC, Russia
16	MST GS 161f	
17	MST SG 186	
18	MST SG 151h	

The recommended values and number of assays for each CRM are listed in Table 10.4. Laboratory certificates have been provided for all but one of the CRMs. CRM limits are provided as permitted allowed absolute error (based on >95% of samples being within that target) rather than the more usual standard deviation limits.

In general, a good precision of the results of laboratory assays for Ag and certified valued was noted. The highest deviations are typical for CRMs with low Ag grades (<5g/t) that are close to the assays' detection limits.

The majority of assay results beyond allowed error limits with meaningful zinc contents were shown for GBM 310-16 and GBM 909-11 CRMs generally returning lower Zn grades in comparison with CRMs.

Despite of this, WAI considers risk for MRE as insignificant.

Table 10.4: Summary of CRMs Data for Vertikalnoye

CRM	Metal, Unit	Grade	Standard Deviation	Expanded Uncertainty	Number of CRMs	Beyond Allowed Absolute Error	%ge of Satisfactory Assays
OREAS 600	Ag, g/t	24.8	1.01		3	0	100.0%
	Zn, %	0.255	0.008		NA		
	Pb, g/t	994	69		NA		
OREAS 605	Ag, g/t	972	27.8		1	0	100.0%
	Zn, %	0.216	0.009		1	0	100.0%
	Pb, g/t	1297	136		1	0	100.0%
GBM 906-6	Ag, g/t	389.7	21.1		311	4	98.7%
	Zn, g/t	210	14		151	32	78.8%
	Pb, g/t	290	14		151	27	82.1%
GBM 913-13	Ag, g/t	74,1	3.9		12	0	100.0%
	Zn, g/t	386	nr		12	3	75.0%
	Pb, g/t	125	nr		12	4	66.7%
GBM 998-9	Ag, g/t	101.2	4.8		156	11	92.9%
	Zn, g/t	27	10		89	very low grades	
	Pb, g/t	8	4		89	very low grades	
GBM303-1	Ag, g/t	1419.6	73.5		8	1	87.5%
	Zn, g/t	28750	1529		6	0	100.0%
	Pb, g/t	236561	14346		6	0	100.0%
GBM310-16	Ag, g/t	314.3	14.9		27	0	100.0%
	Zn, g/t	170201	6825		27	8	70.4%
	Pb, g/t	112603	5008		27	5	81.5%
GBM906-7	Ag, g/t	0.9	0.3		1	0	100.0%
	Zn, g/t	51	11		1	0	100.0%
	Pb, g/t	8	4		1	0	100.0%
GBM909-11	Ag, g/t	25.5	1.7		15	0	100.0%
	Zn, g/t	19486	591		15	6	60.0%
	Pb, g/t	2074	103		15	1	93.3%
GBM913-13	Ag, g/t	74.1	3.9		16	0	100.0%
	Zn, g/t	386	nr		16	0	100.0%
	Pb, g/t	125	nr		16	0	100.0%
GBM997-4	Ag, g/t	287.9	38.2		105	3	97.1%
	Zn, g/t	119	13		62	very low grades	
	Pb, g/t	159	17		62	very low grades	
СОП 01-2016 (SOP 01-2016)	Ag, g/t	3,21		+/- 0,28	38	16	57.9%
	Zn, %	0,129		+/- 0,007	11	2	81.8%
	Pb, %	0,083		+/- 0,004	11	4	63.6%
СОП 02-2016 (SOP 02-2016)	Ag, g/t	73,7		+/- 3,2	40	0	100.0%
	Zn, %	0,86		+/- 0,02	20	1	95.0%
	Pb, %	2,45		+/-0,09	20	1	95.0%
СОП 03-2016 (SOP 03-2016)	Ag, g/t	124,4		+/- 6,2	20	1	95.0%
	Zn, %	50,3		+/- 0,2	12	very low grades	
	Pb, %	1,37		+/-0,09	13	0	100.0%
MST SG 130i	Ag, g/t	171,8		+/-4,5	9	0	100.0%
MST GS 161f	Ag, g/t	1,49		NA	1		
MST SG 186	Ag, g/t	36		NA	32		
	Zn, %	0,0053		NA	10		
	Pb, %	0,035		NA	10		
MST SG 151h	Ag, g/t	78,3		+/-2.2	8	0	100.0%

There are no data on allowed absolute error for CRMs GS 161f (one sample) and MST SG 186 (32 samples) therefore results for these CRMs were not considered. The results of CRMs analyses are illustrated on Figure 10.4 to Figure 10.33.

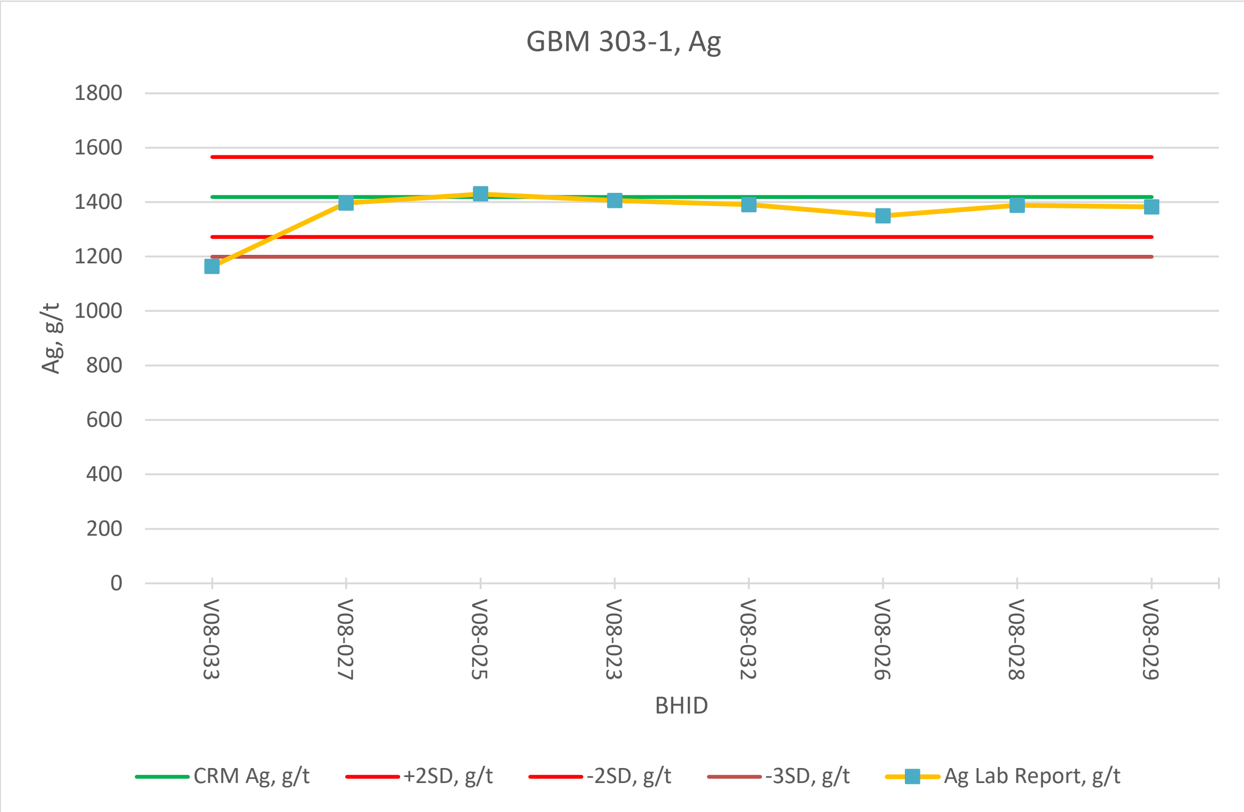


Figure 10.4: GBM 303-1, Ag, CRM Assaying Results

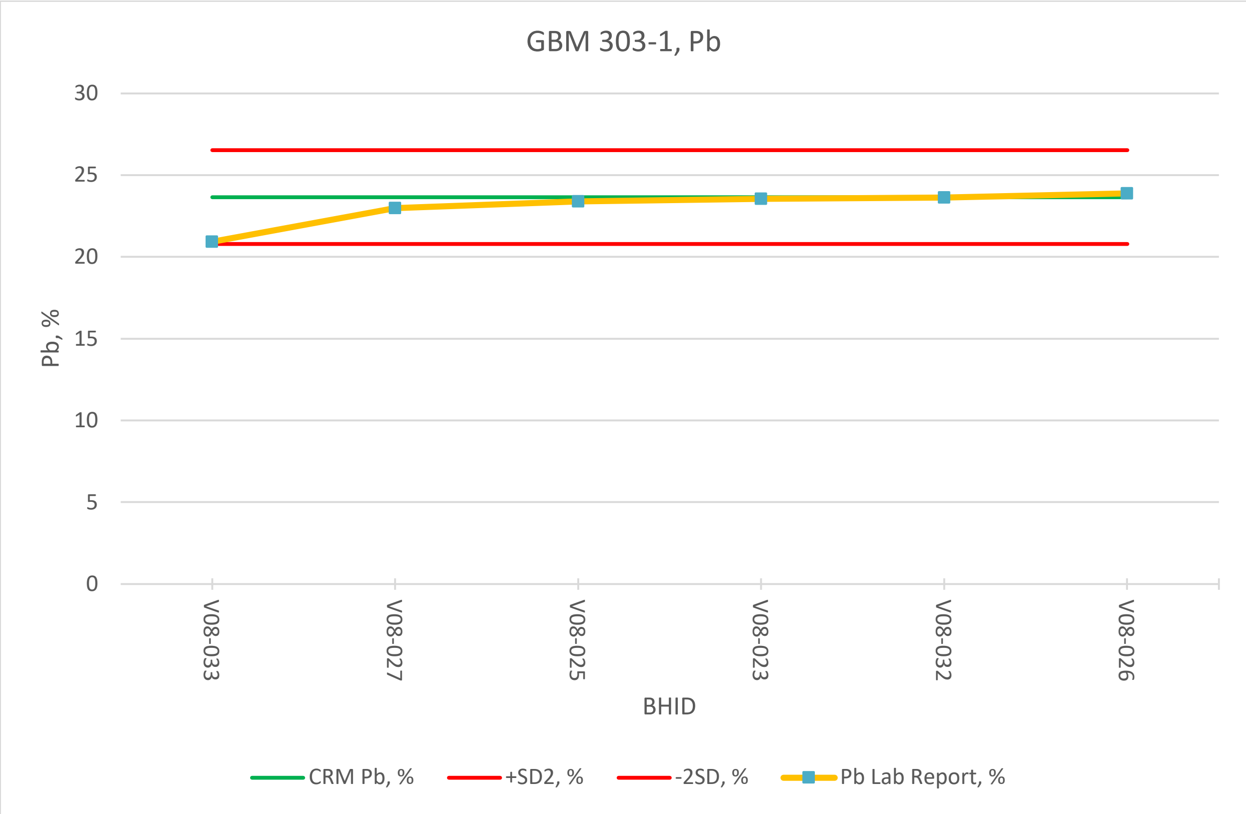


Figure 10.5: GBM 303-1, Pb, CRM Assaying Results

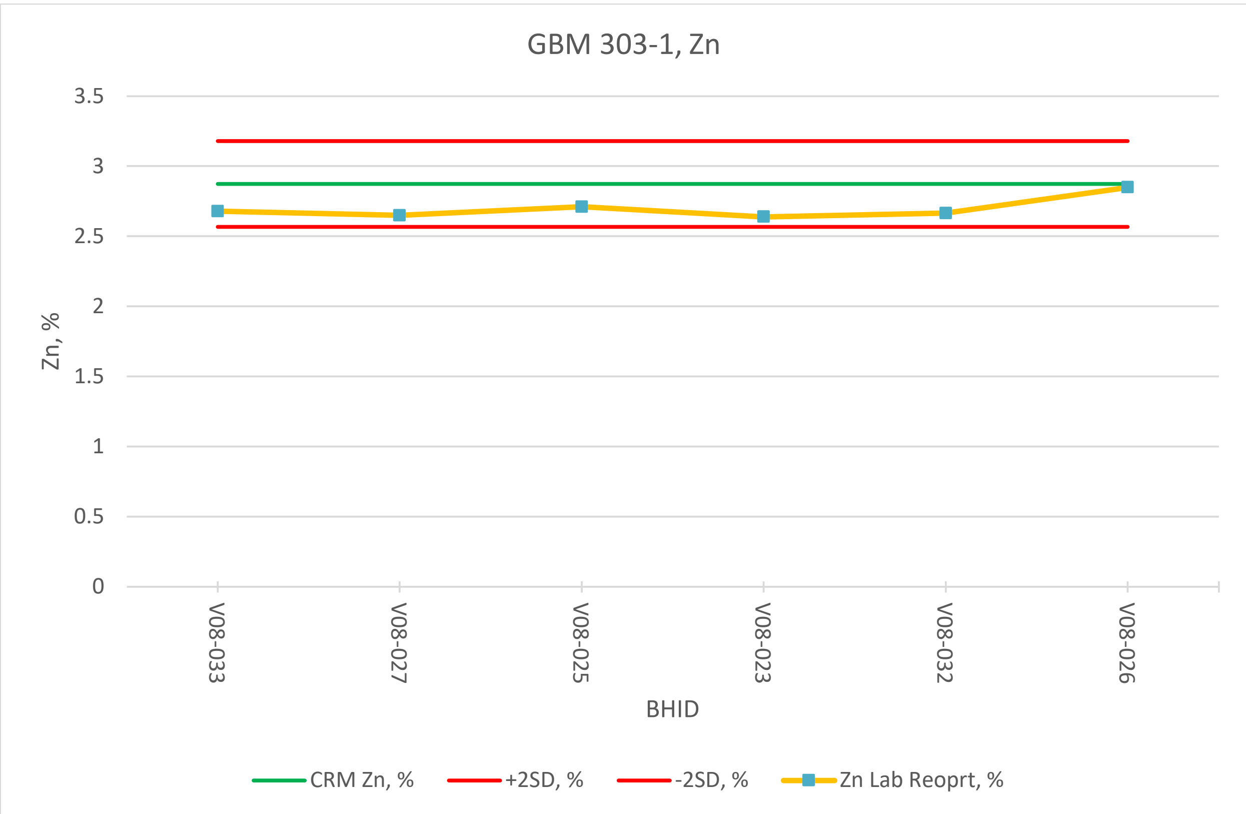


Figure 10.6: GBM 303-1, Zn, CRM Assaying Results

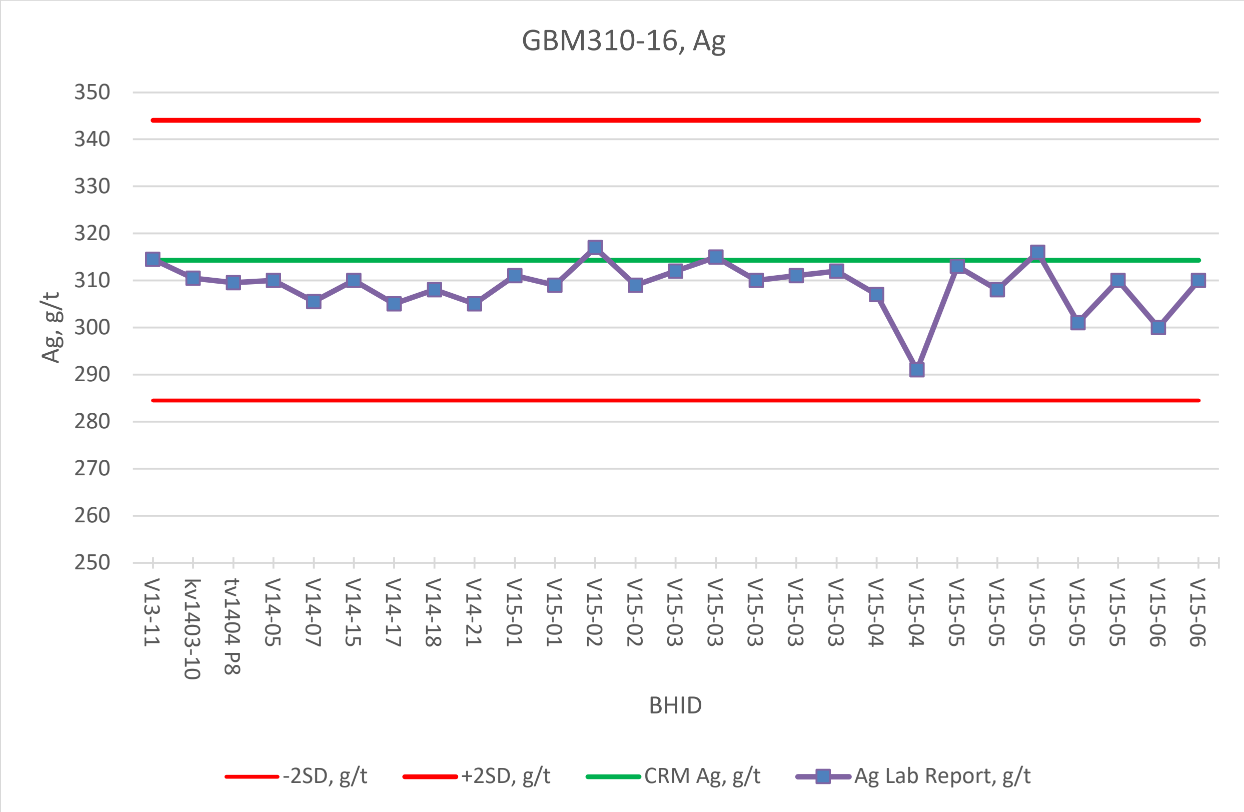


Figure 10.7: GBM 310-16, Ag, CRM Assaying Results

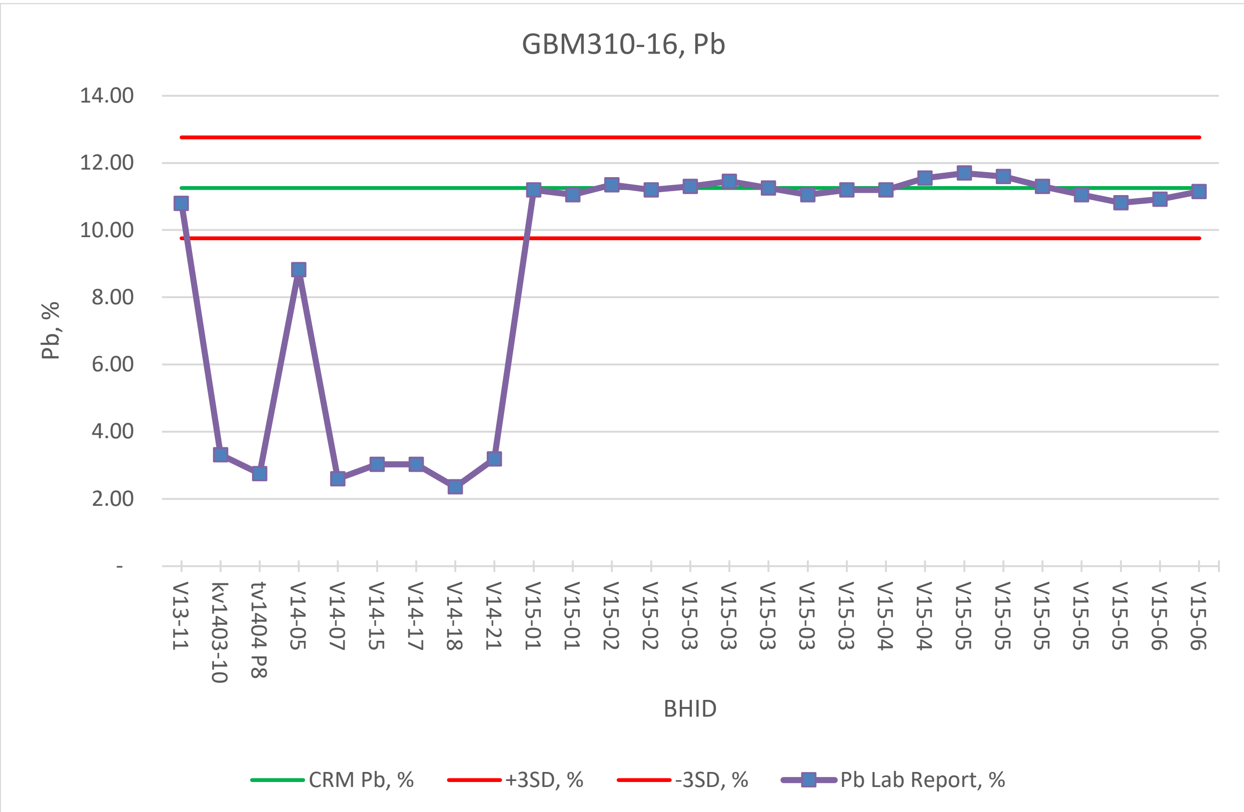


Figure 10.8: GBM 310-16, Pb, CRM Assaying Results

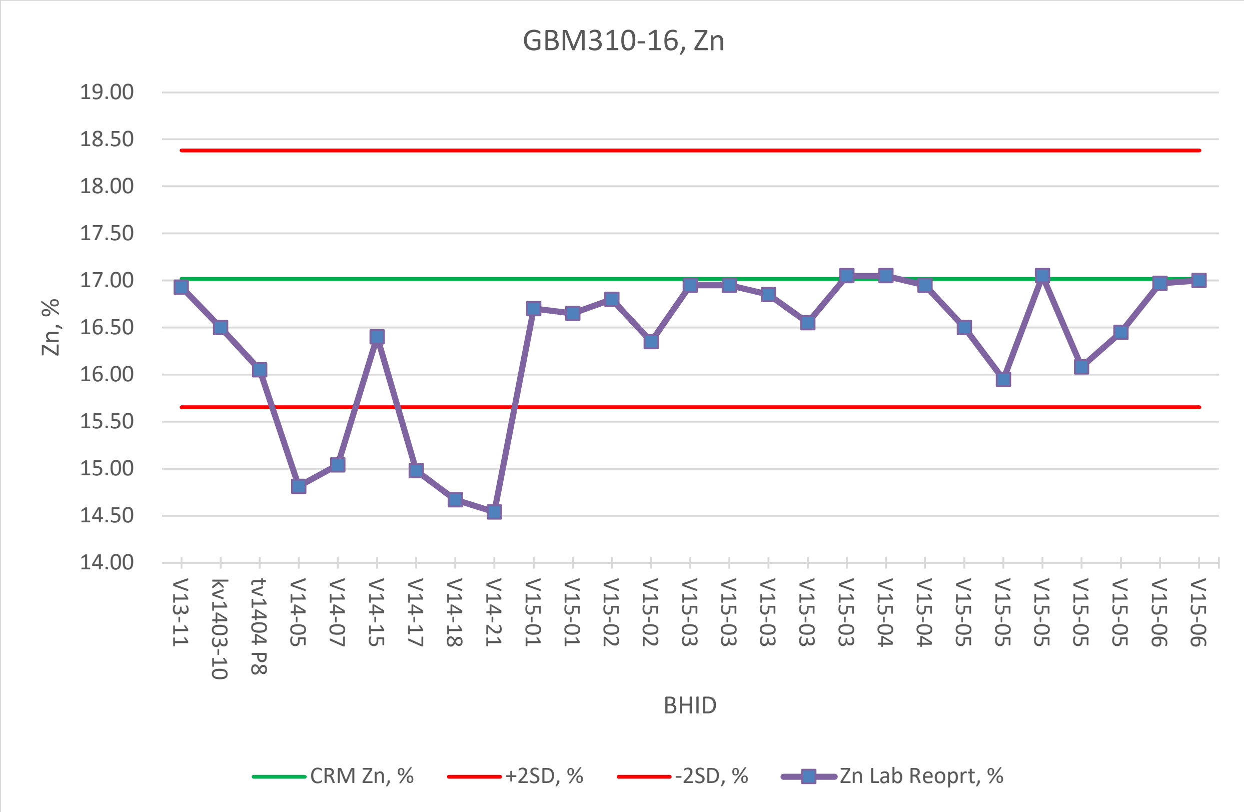


Figure 10.9: GBM 310-16, Zn, CRM Assaying Results

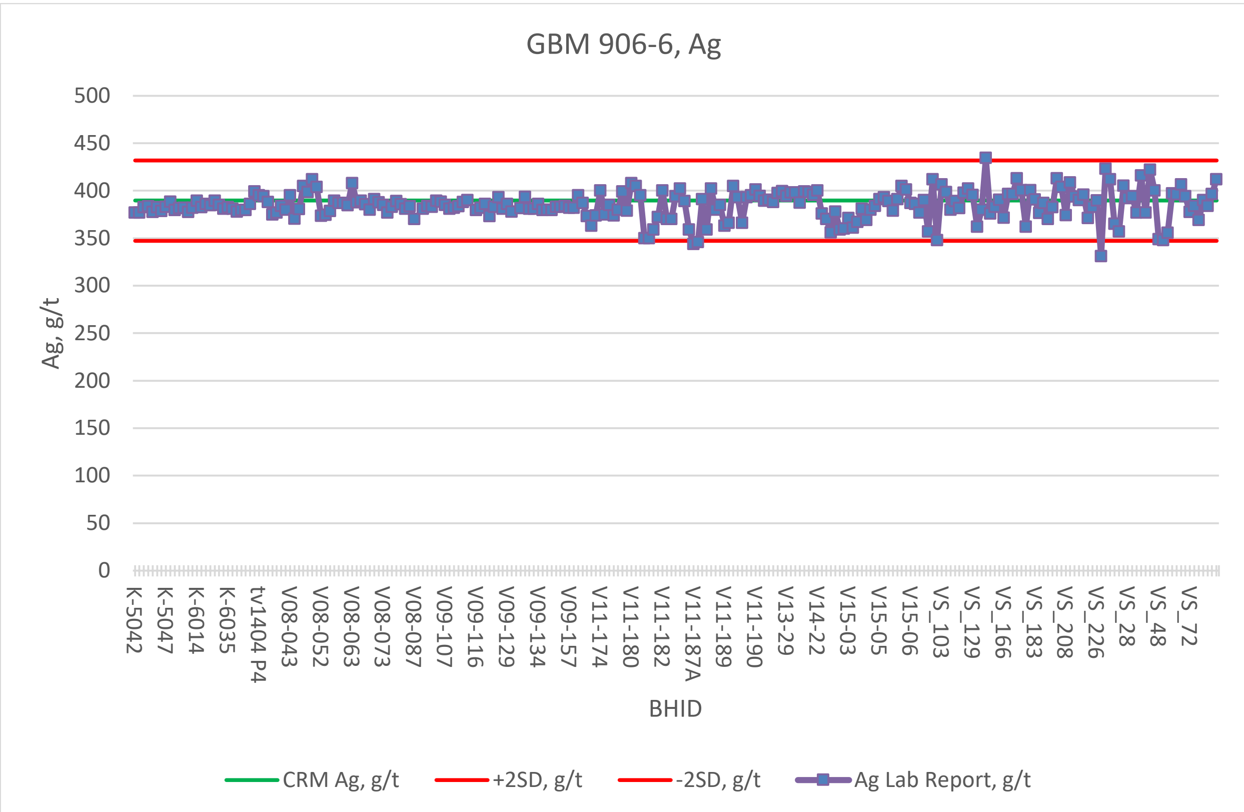


Figure 10.10: GBM 906-6, Ag, CRM Assaying Results

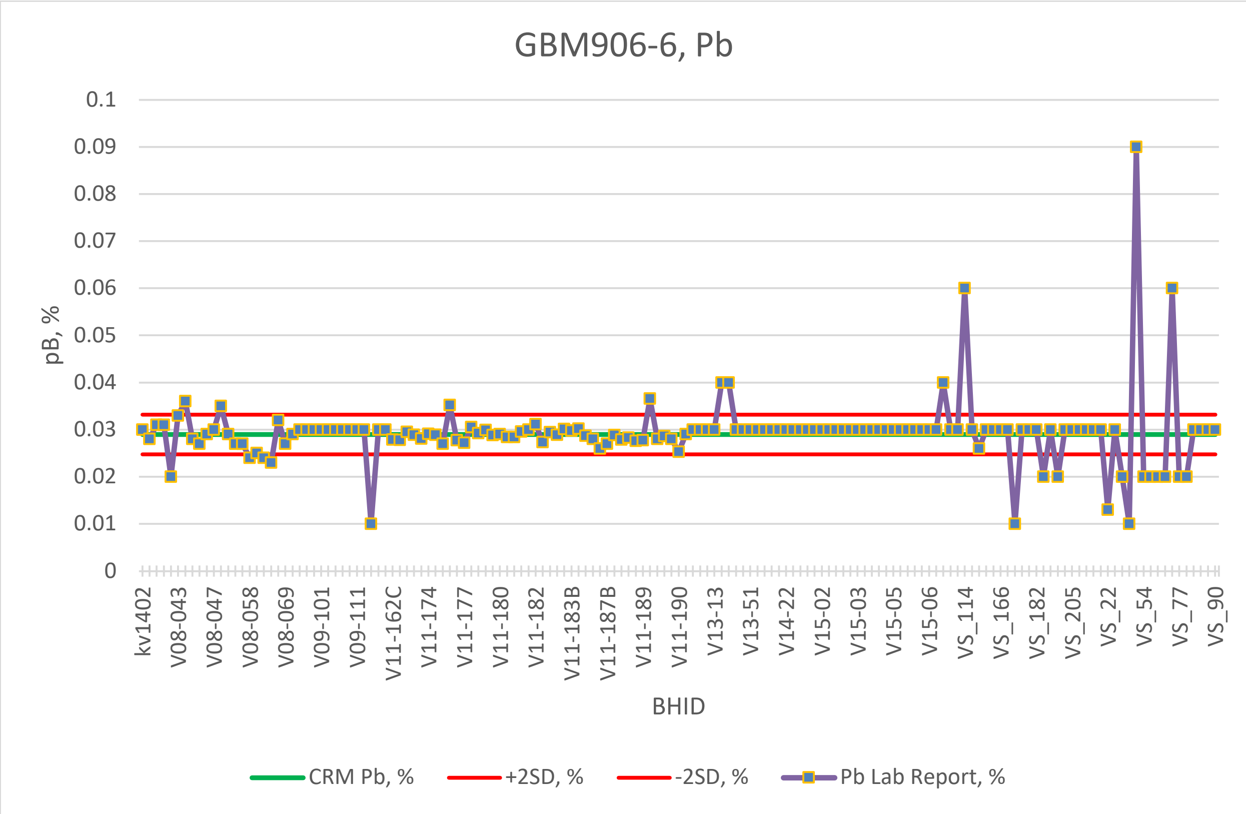


Figure 10.11: GBM 906-6, Pb, CRM Assaying Results

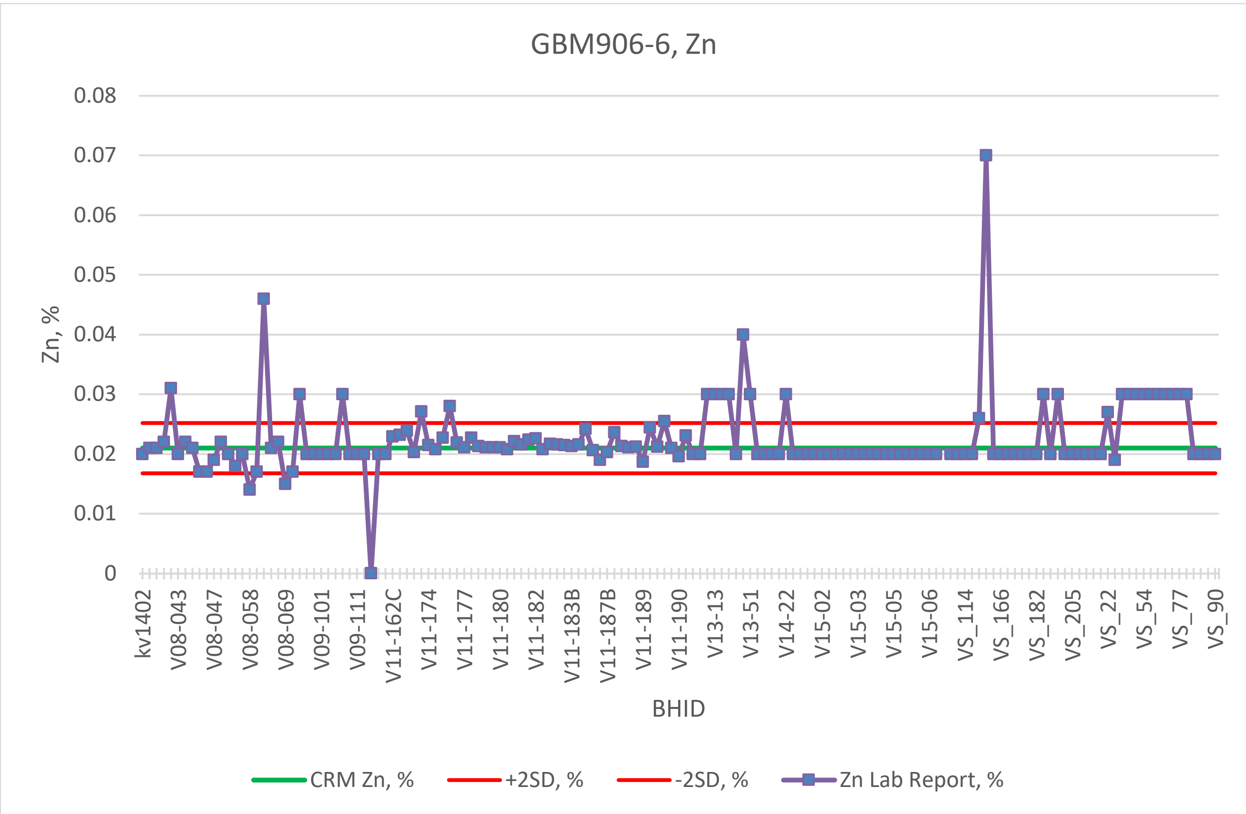


Figure 10.12: GBM 906-6, Zn, CRM Assaying Results

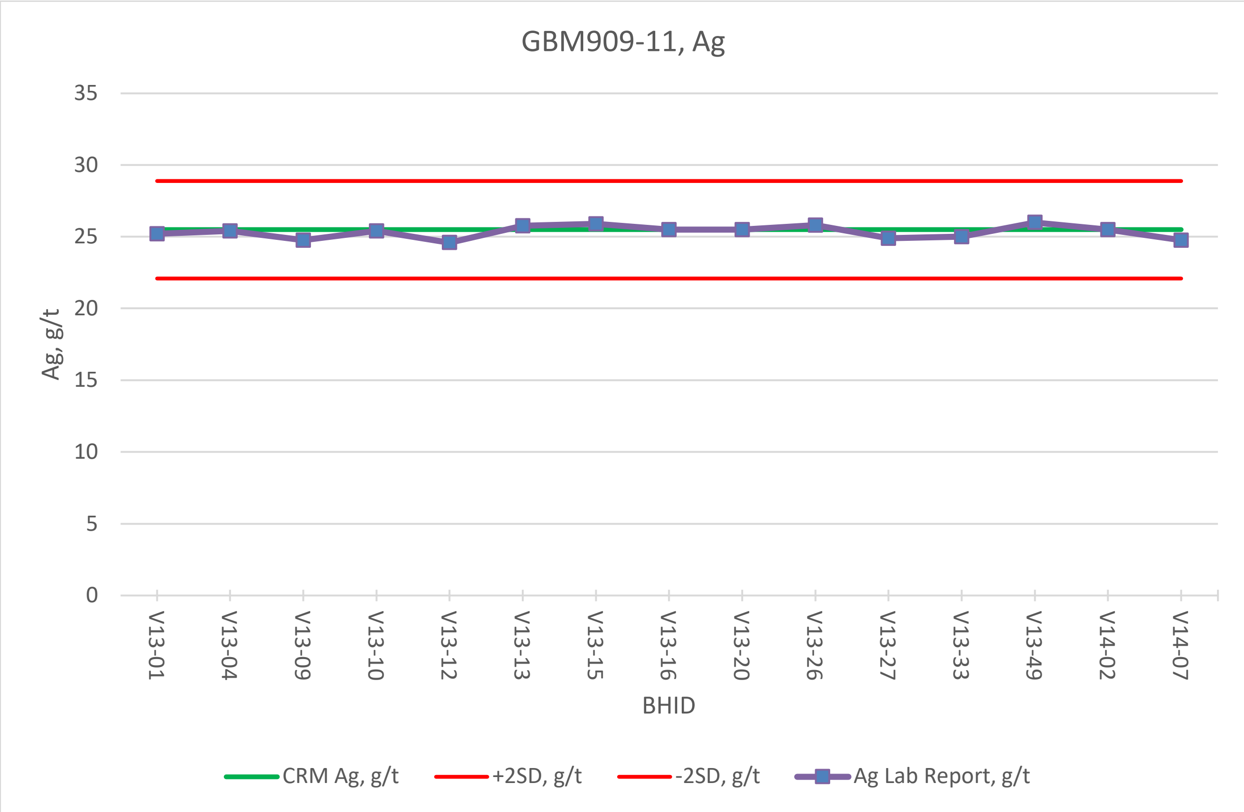


Figure 10.13: GBM 909-11, Ag, CRM Assaying Results

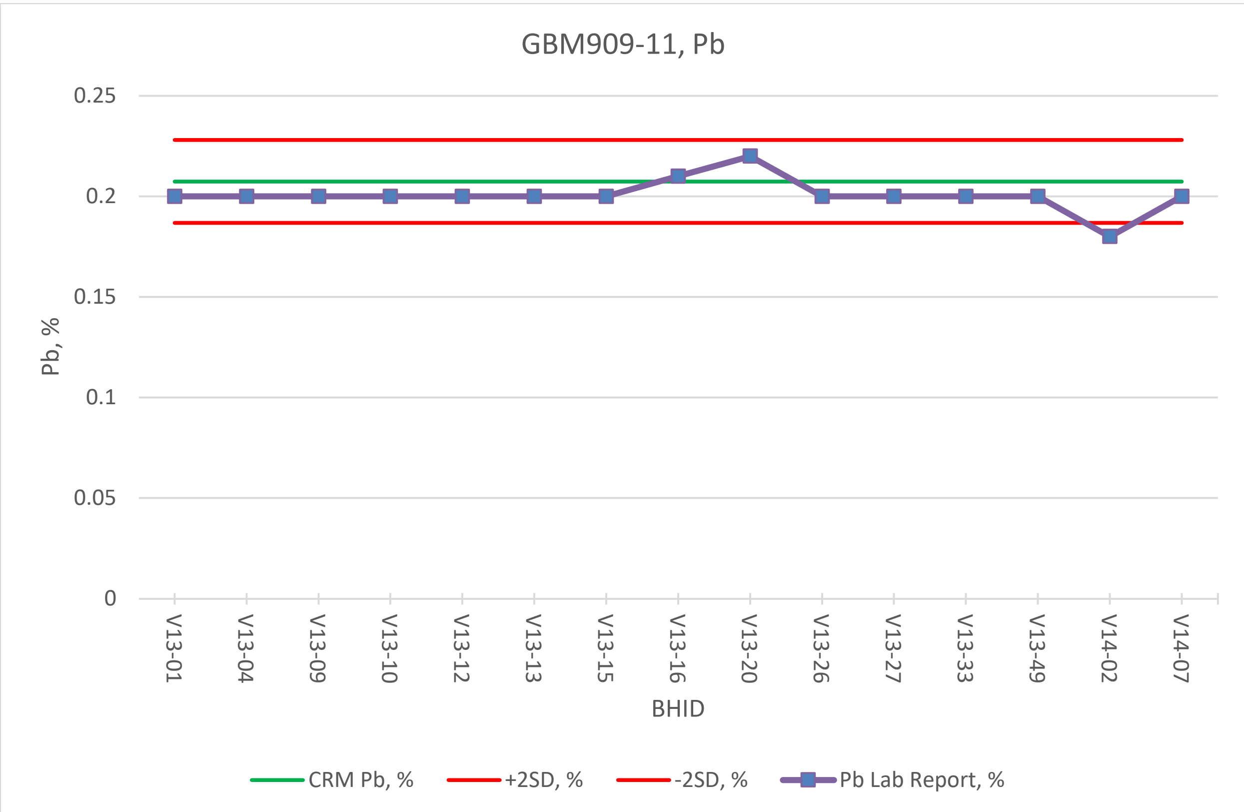


Figure 10.14: GBM 909-11, Pb, CRM Assaying Results

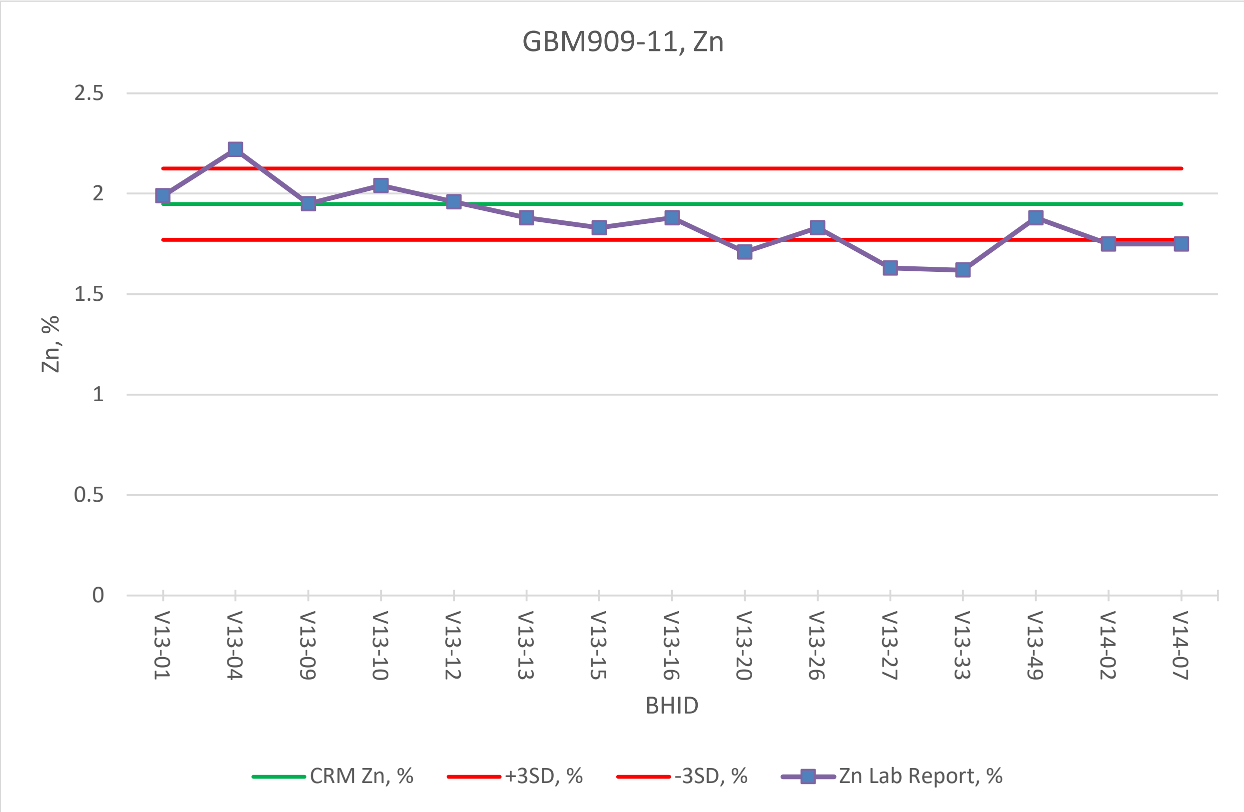


Figure 10.15: GBM 909-11, Zn, CRM Assaying Results

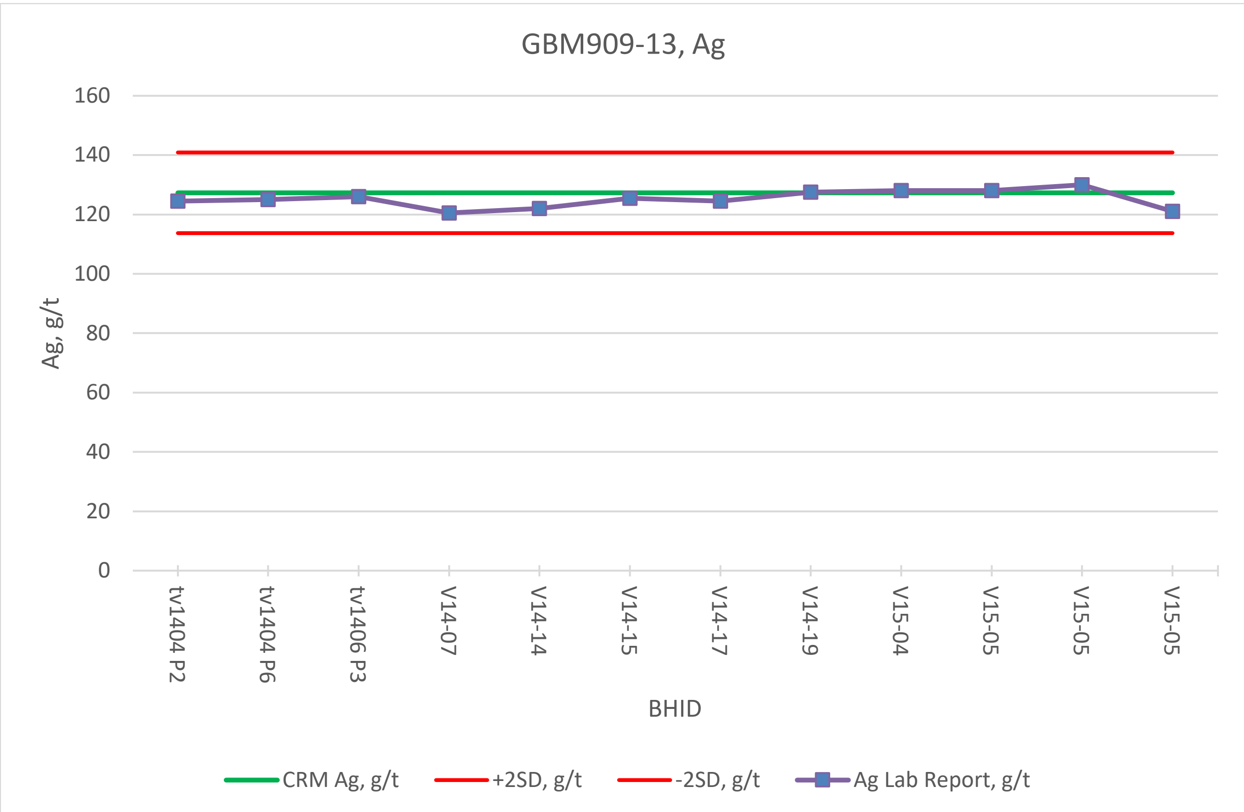


Figure 10.16: GBM 909-13, Ag, CRM Assaying Results

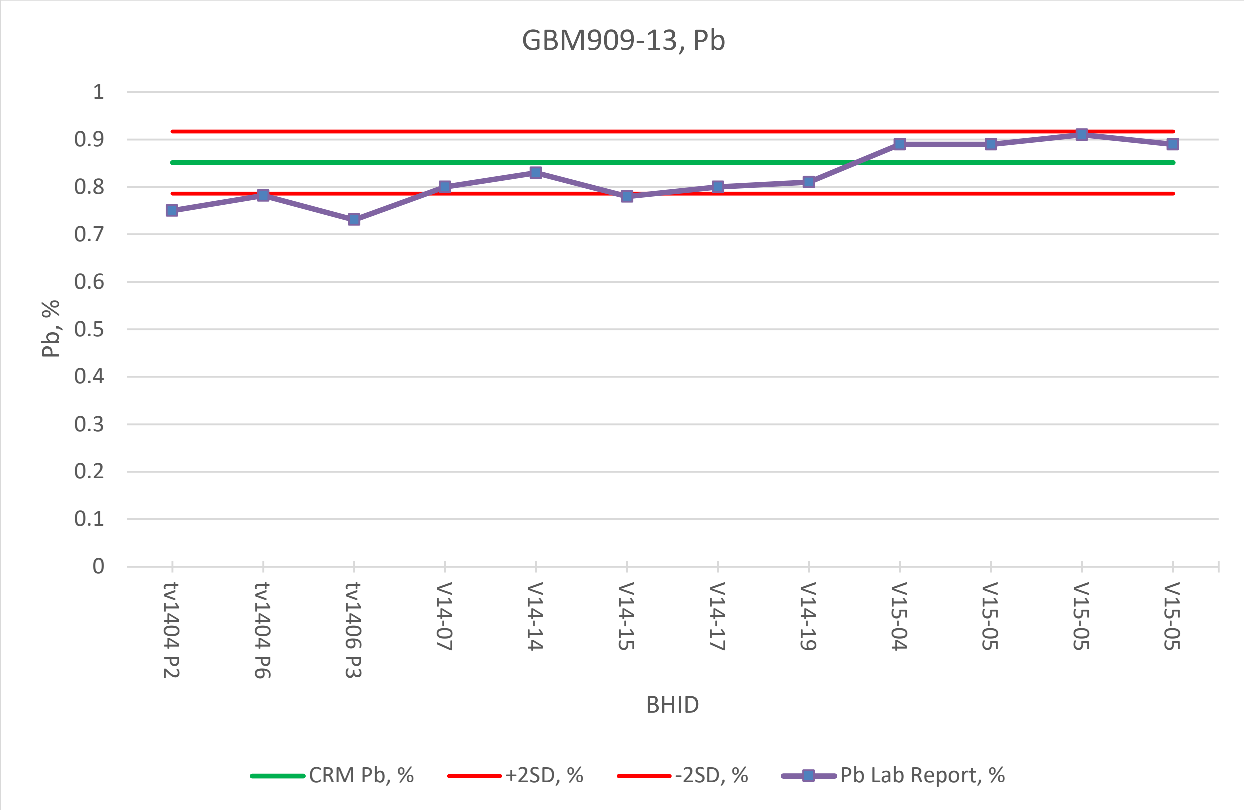


Figure 10.17: GBM 909-13, Pb, CRM Assaying Results

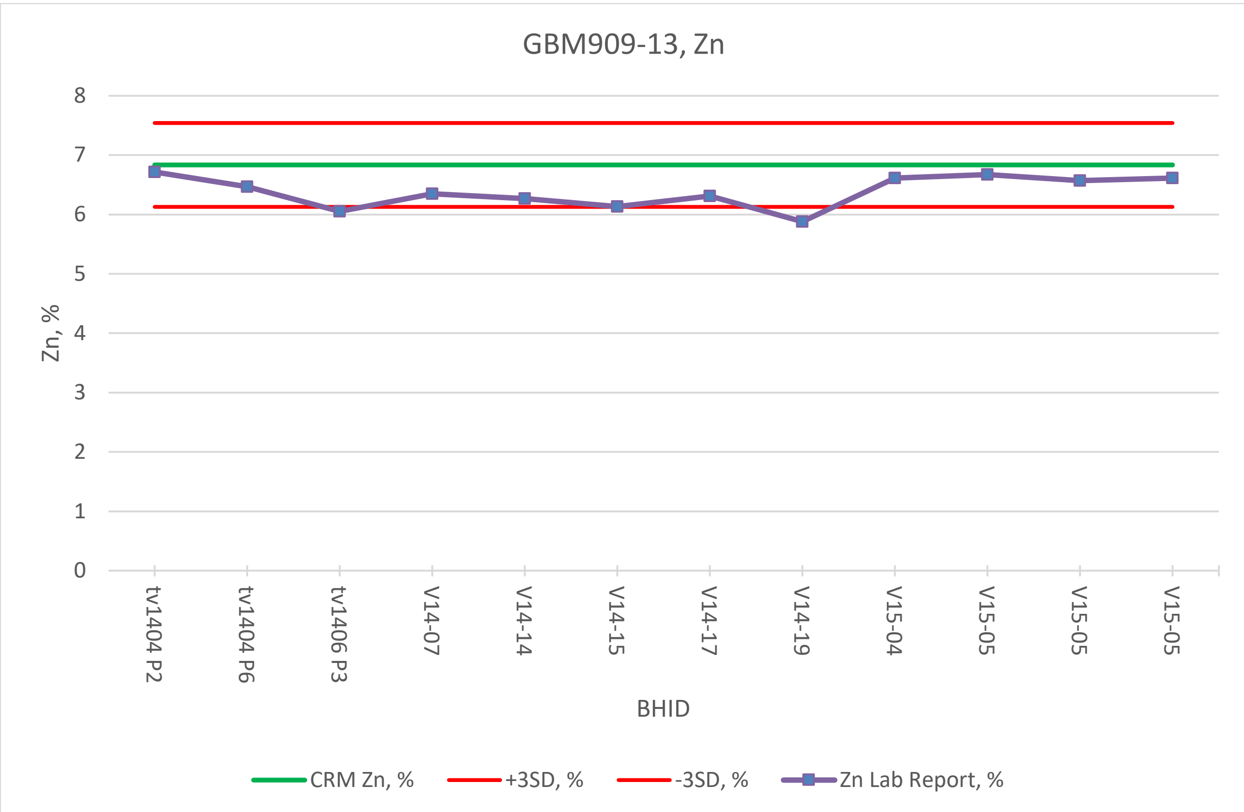


Figure 10.18: GBM 909-13, Zn, CRM Assaying Results

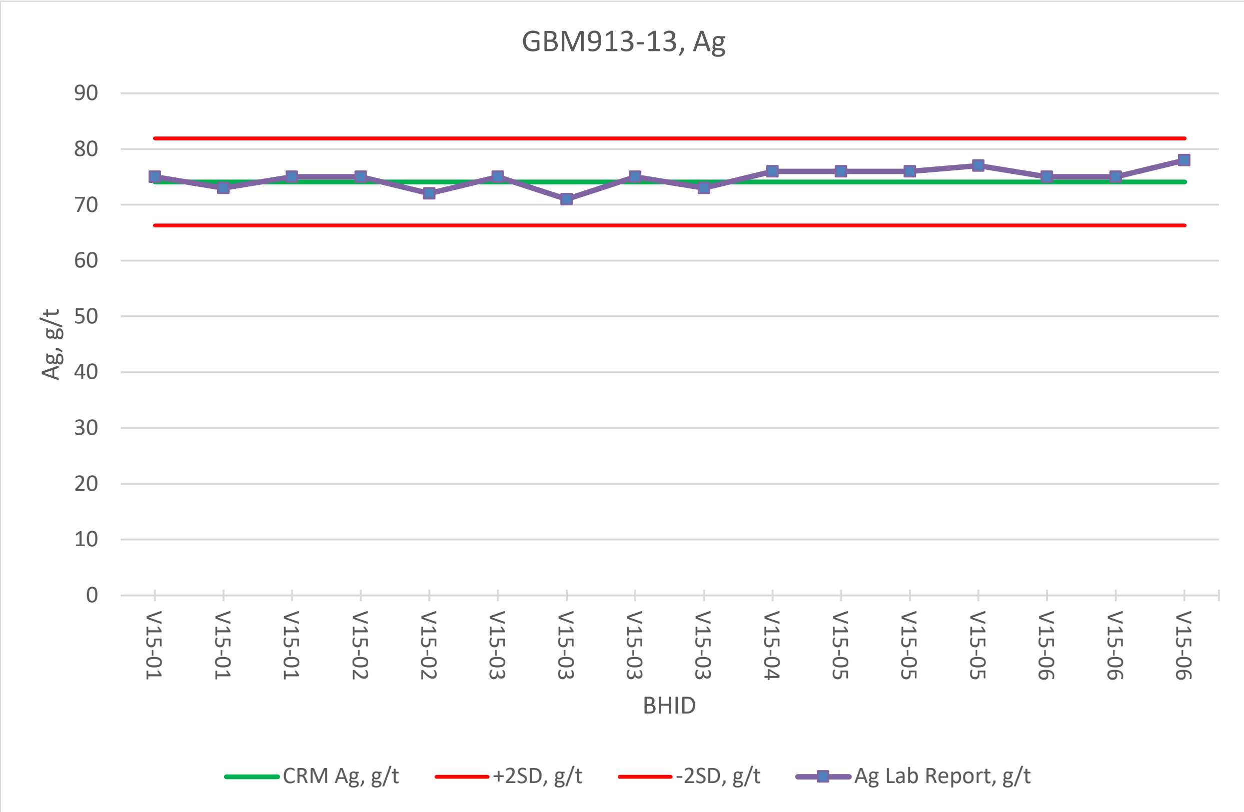


Figure 10.19: GBM 913-13, Ag, CRM Assaying Results

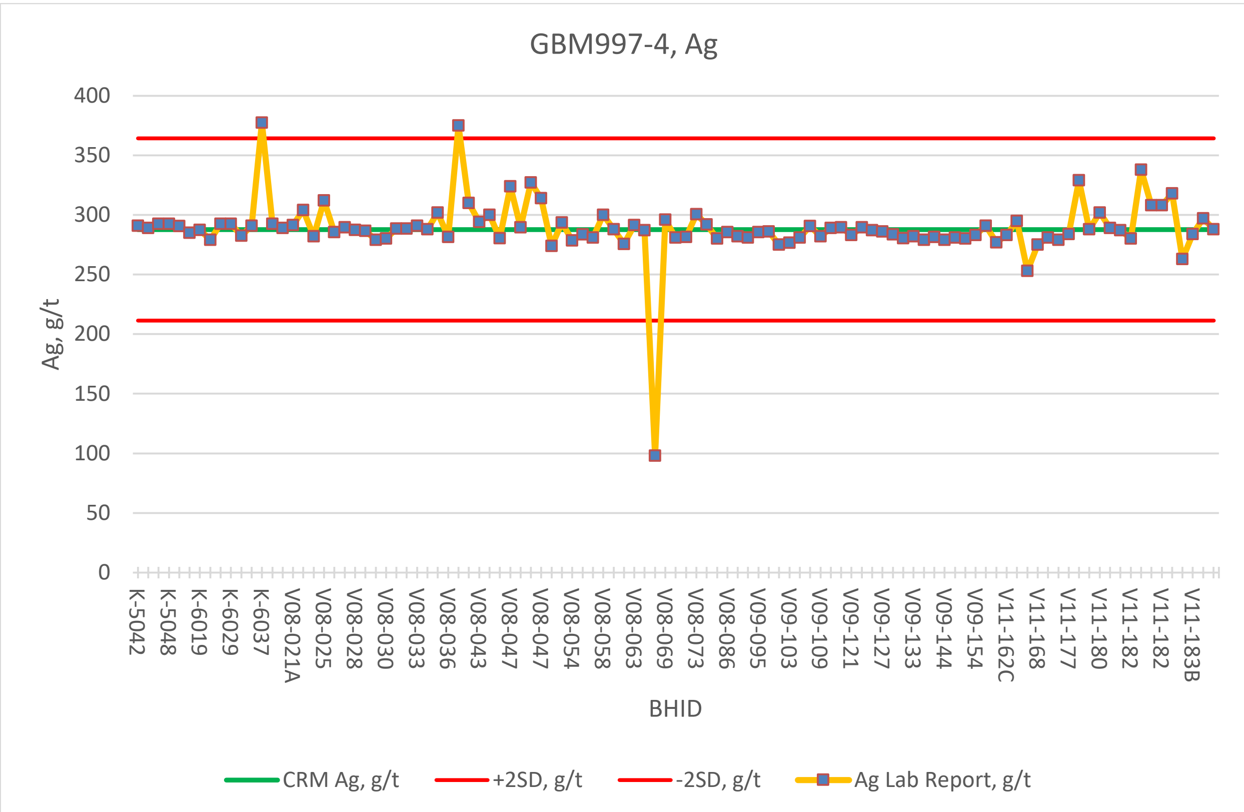


Figure 10.20: GBM 997-4, Ag, CRM Assaying Results

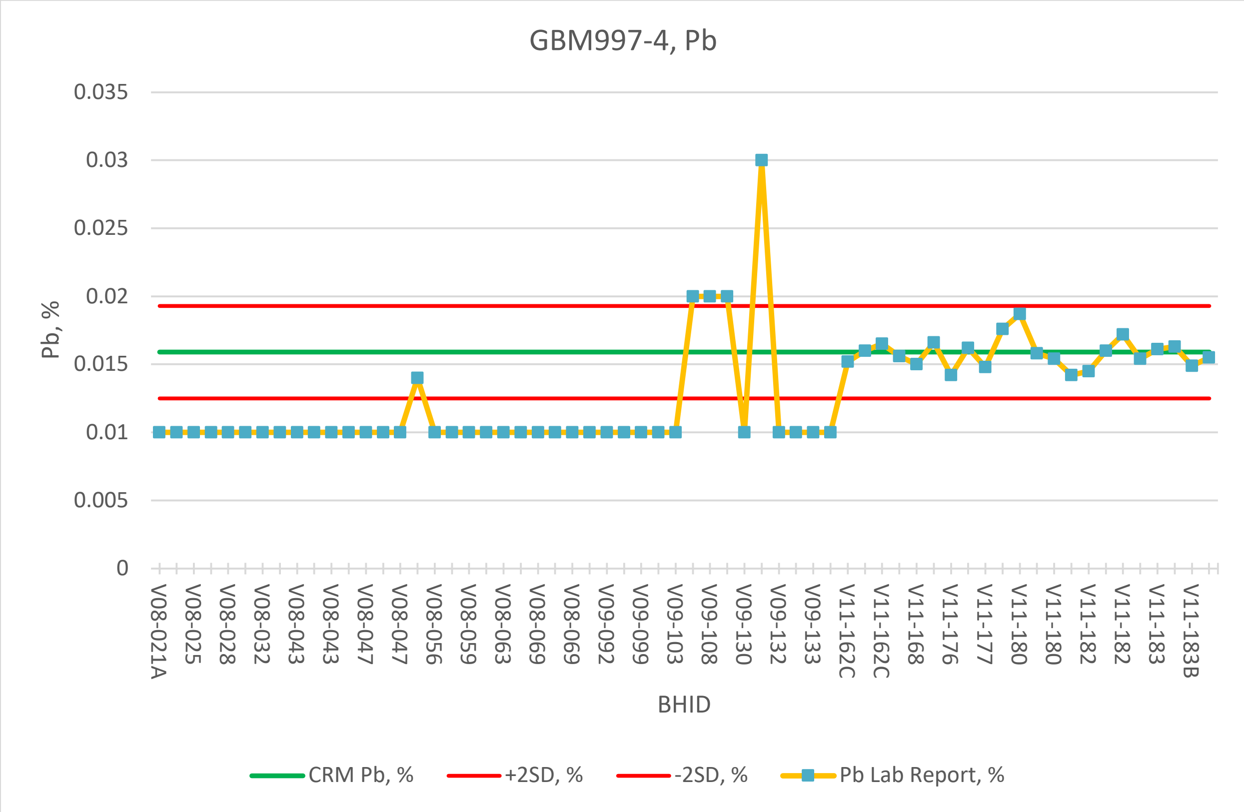


Figure 10.21: GBM 997-4, Pb, CRM Assaying Results

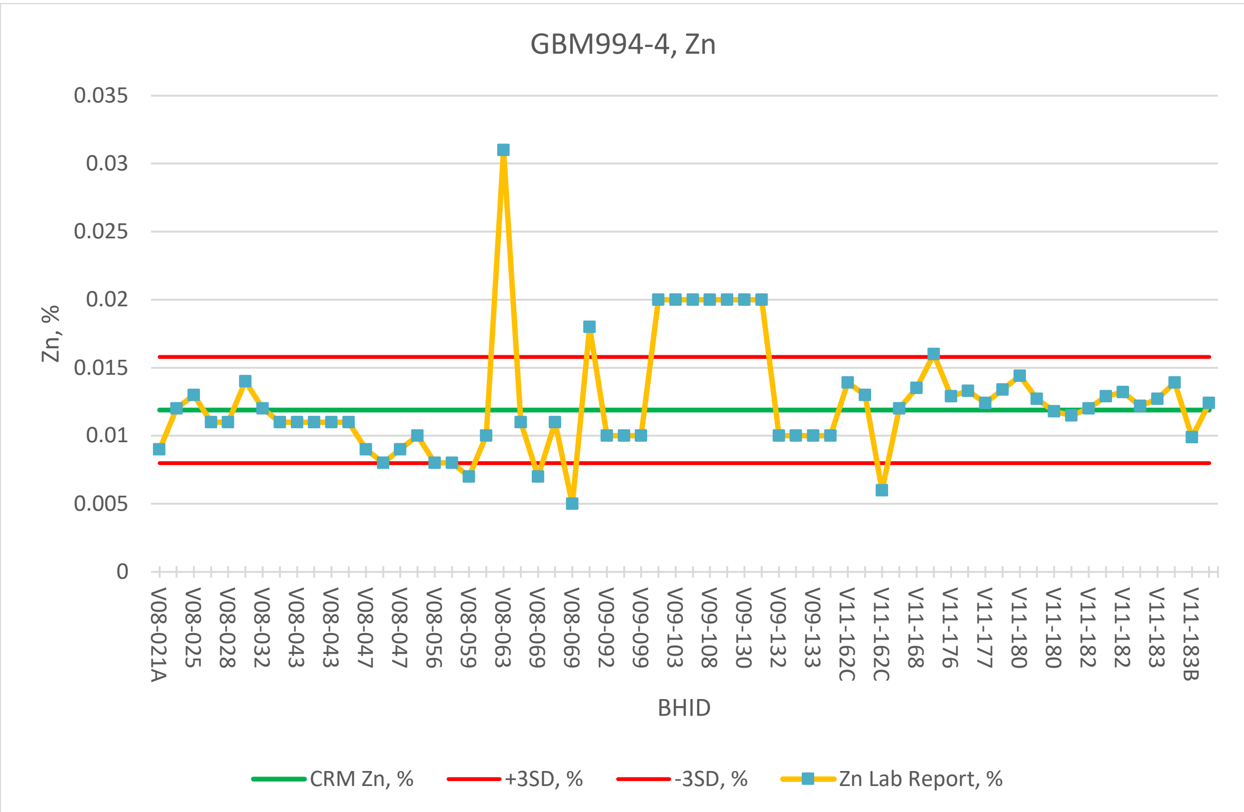


Figure 10.22: GBM 997-4, Zn, CRM Assaying Results

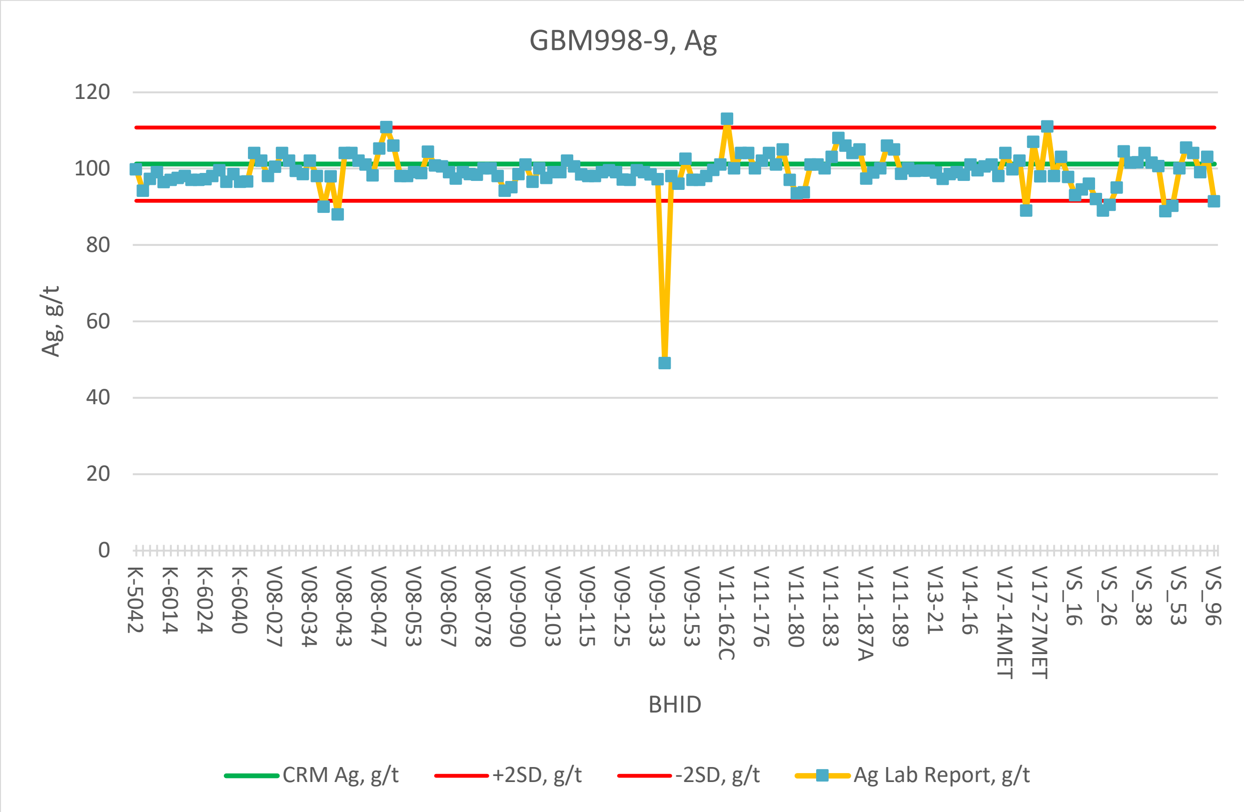


Figure 10.23: GBM 998-9, Ag, CRM Assaying Results

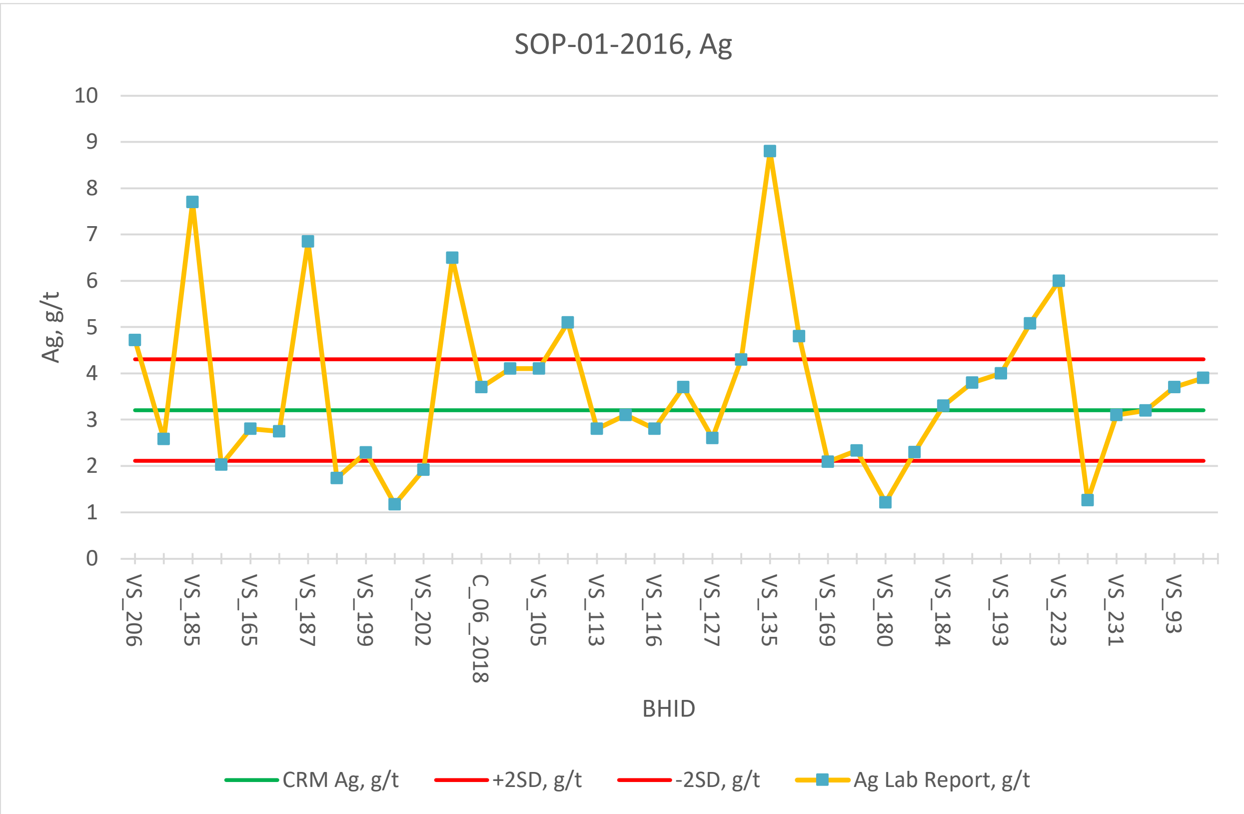


Figure 10.24: SOP-01-2016, Ag, CRM Assaying Results

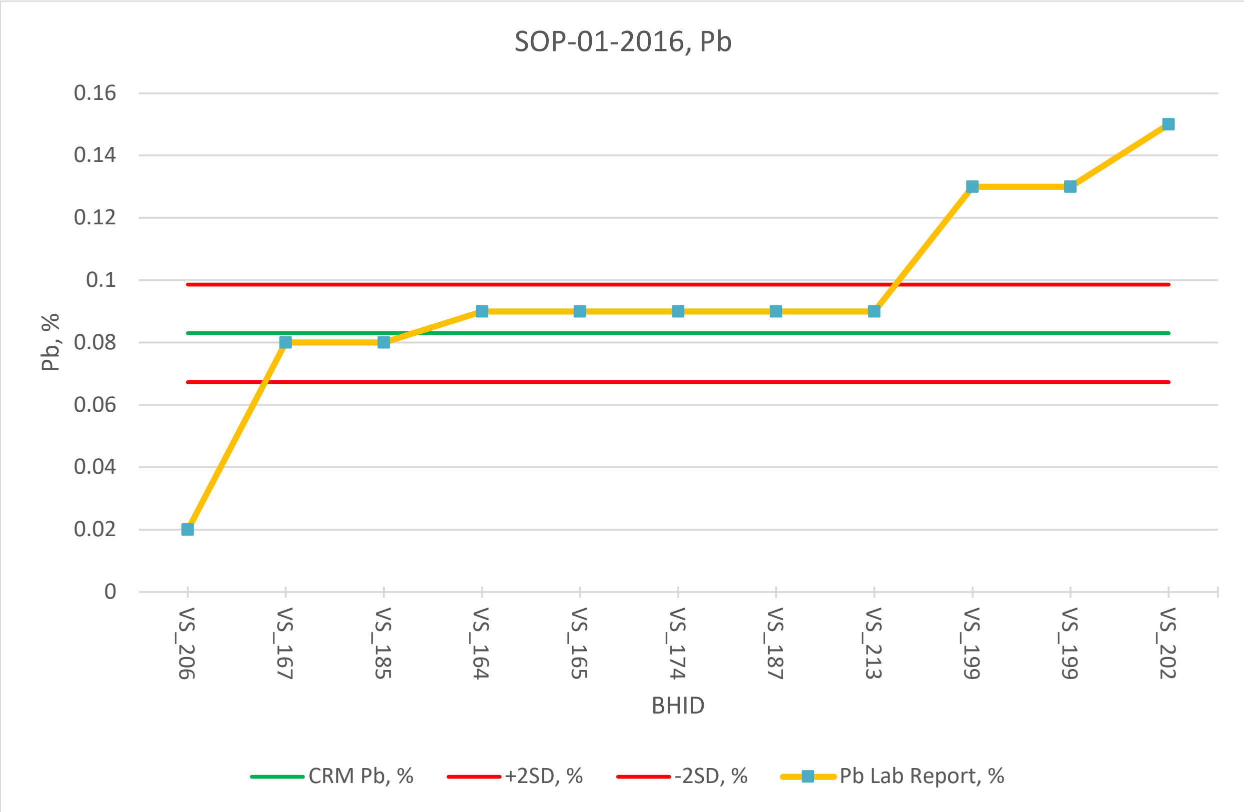


Figure 10.25: SOP-01-2016, Pb, CRM Assaying Results

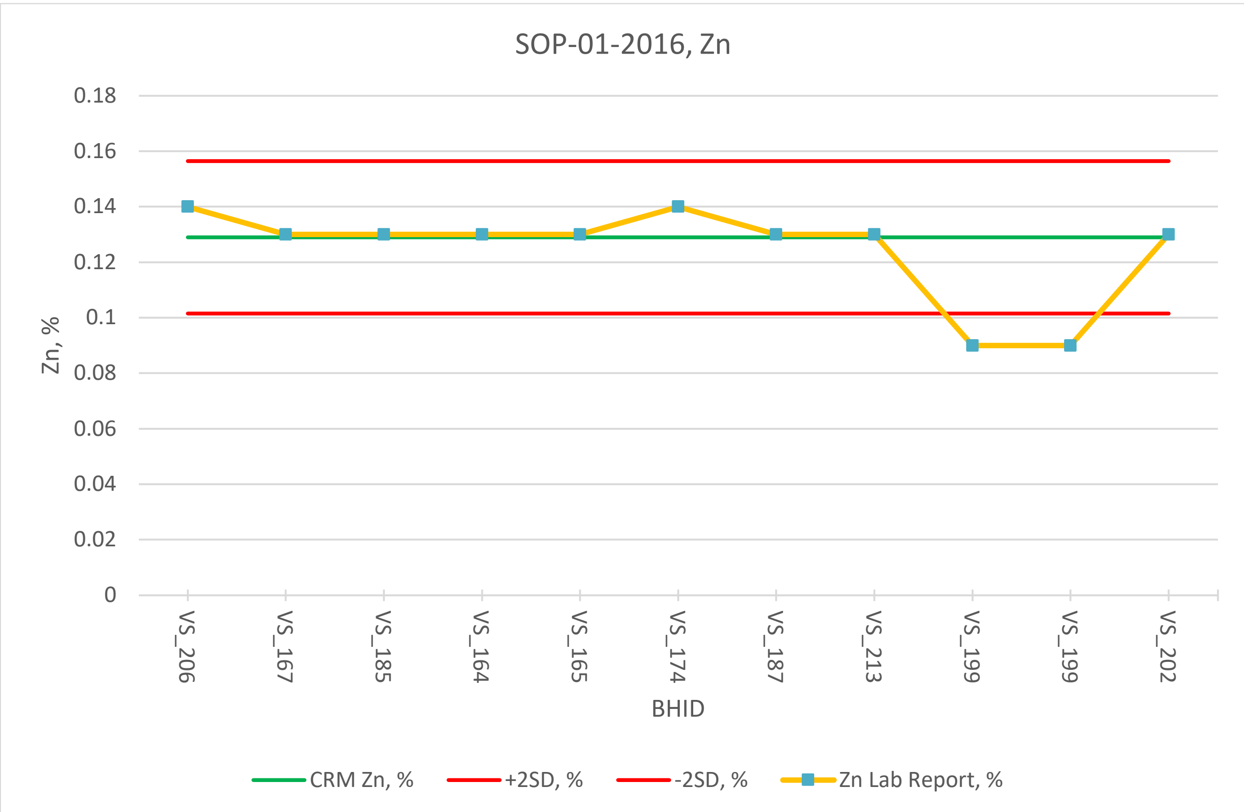


Figure 10.26: SOP-01-2016, Zn, CRM Assaying Results

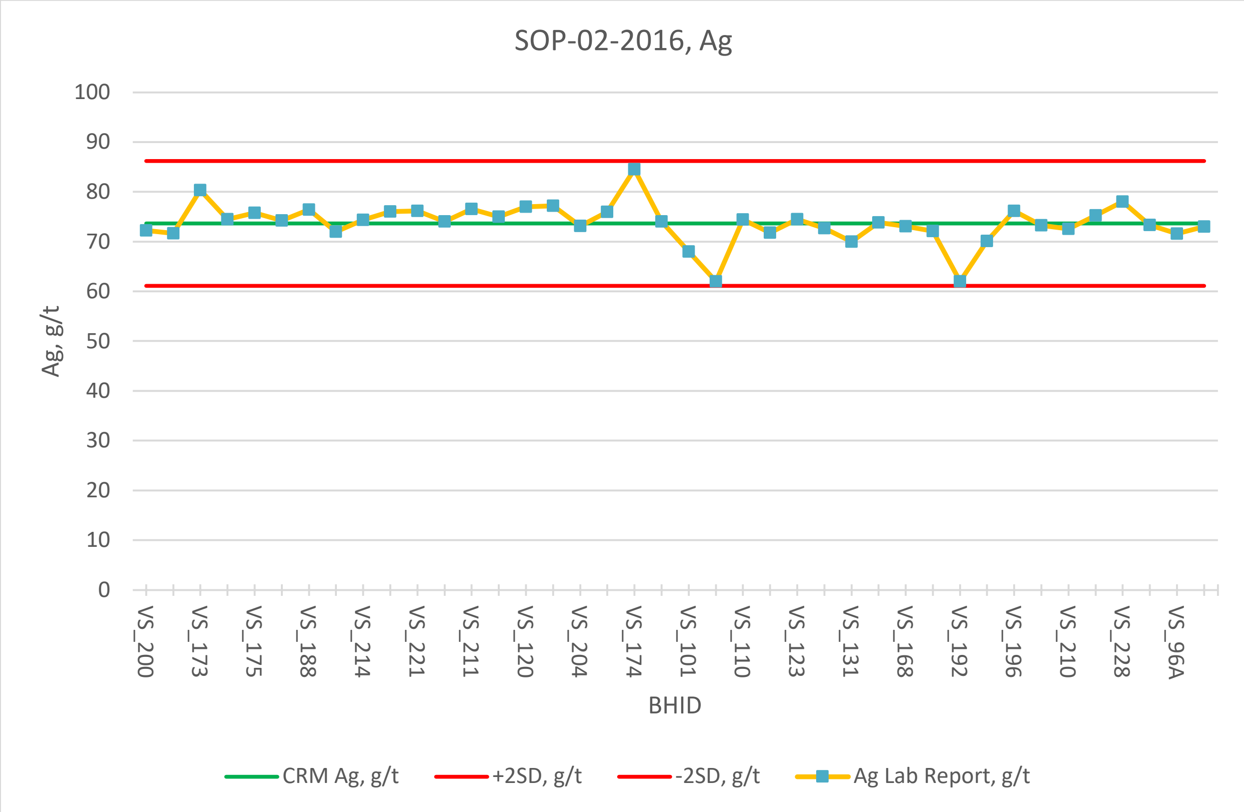


Figure 10.27: SOP-02-2016, Ag, CRM Assaying Results

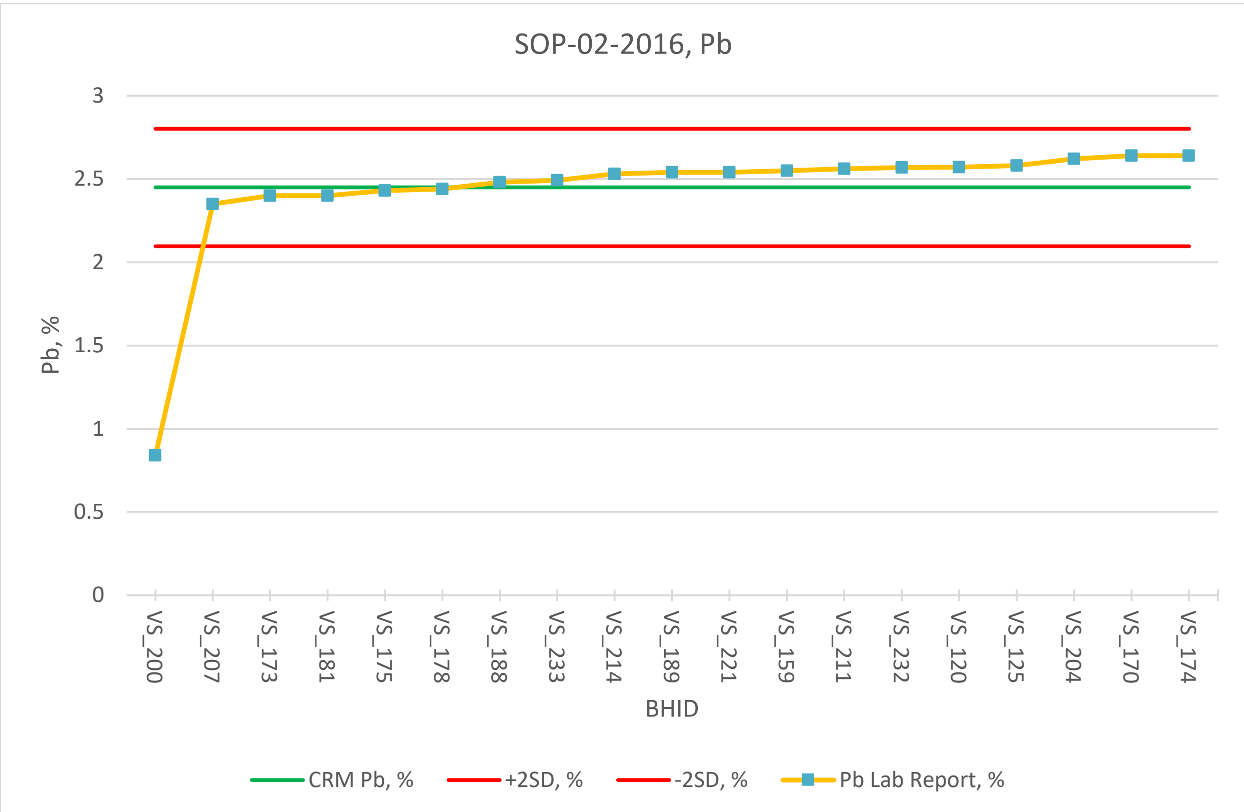


Figure 10.28: SOP-02-2016, Pb, CRM Assaying Results

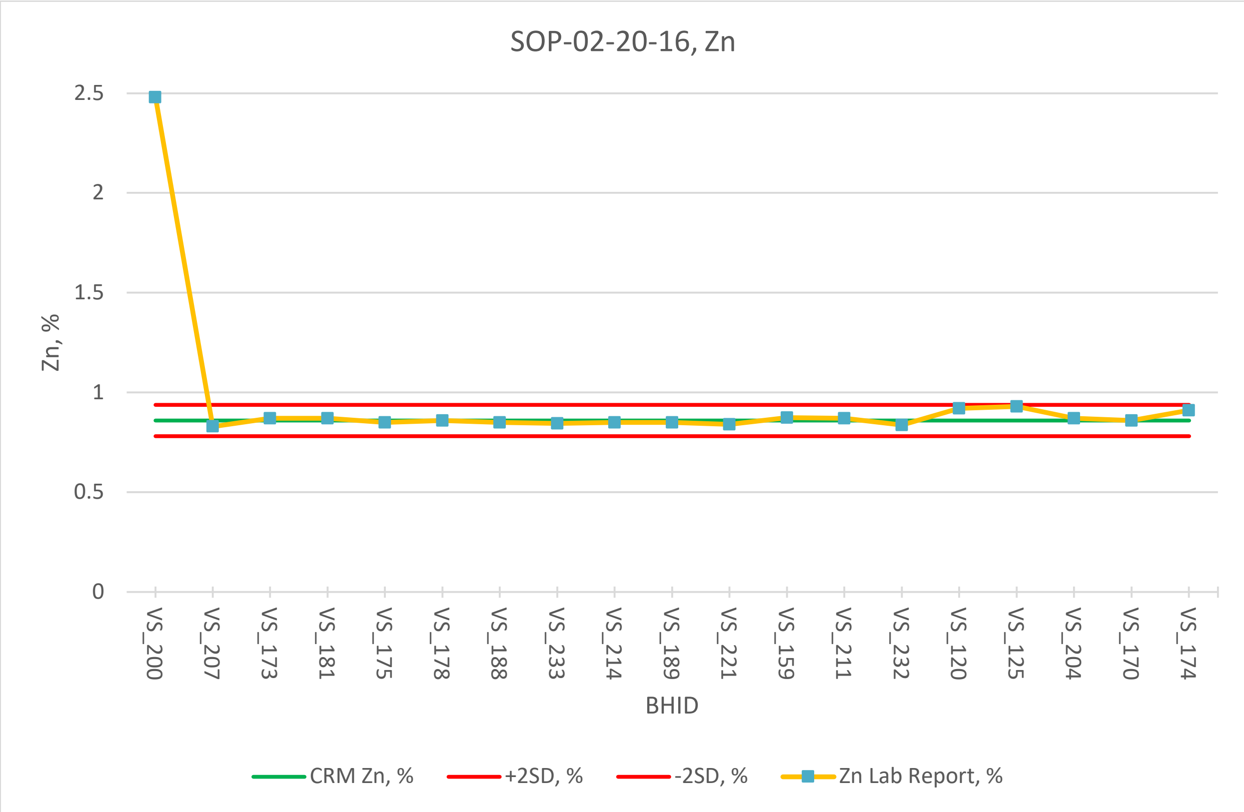


Figure 10.29: SOP-02-2016, Zn, CRM Assaying Results

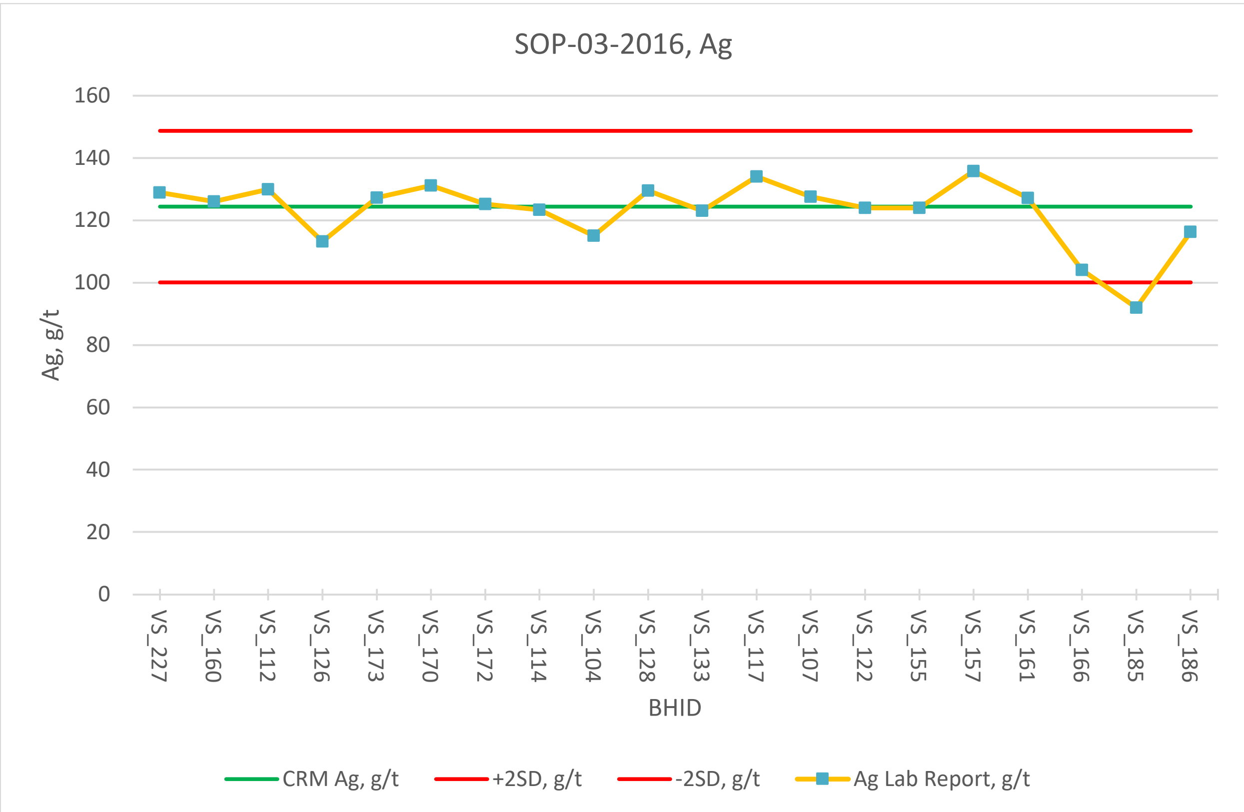


Figure 10.30: SOP-03-2016, Ag, CRM Assaying Results

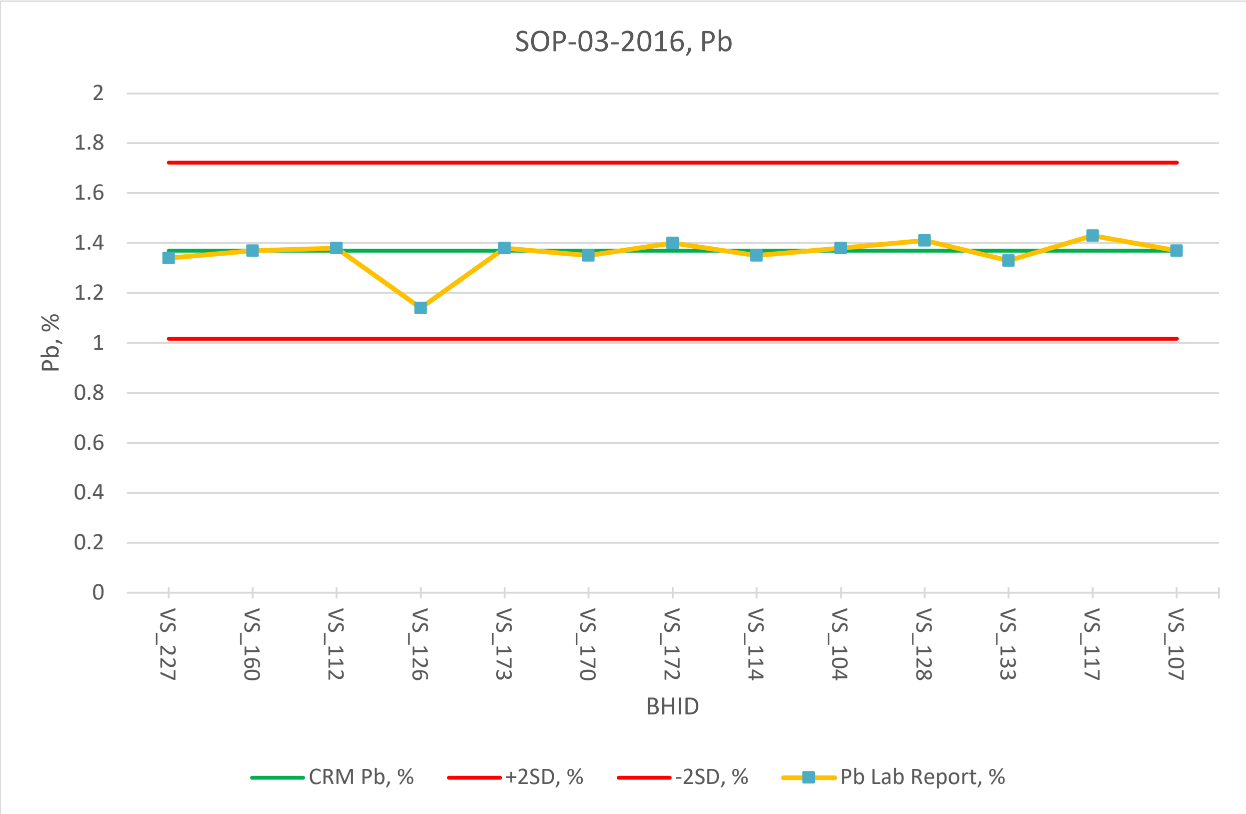


Figure 10.31: SOP-03-2016, Pb, CRM Assaying Results

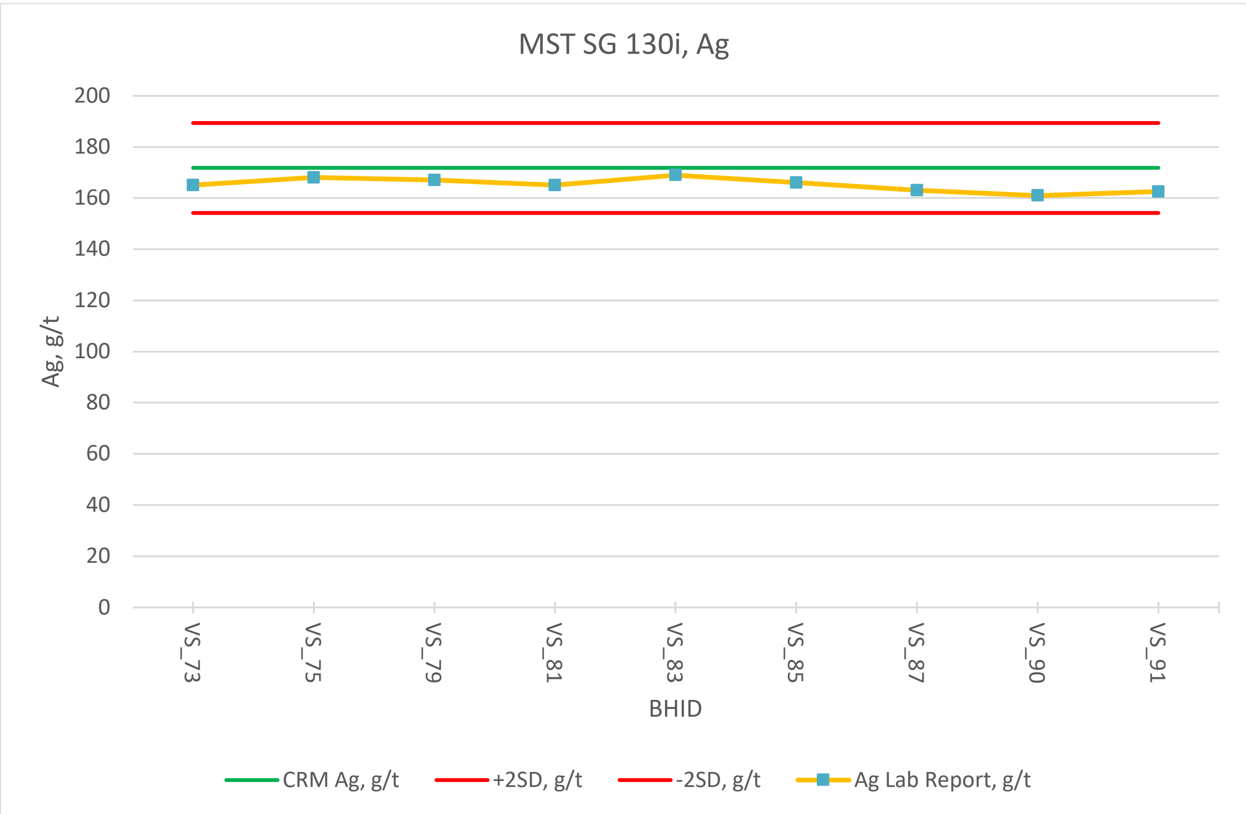


Figure 10.32: MST SG 130i, Ag, CRM Assaying Results

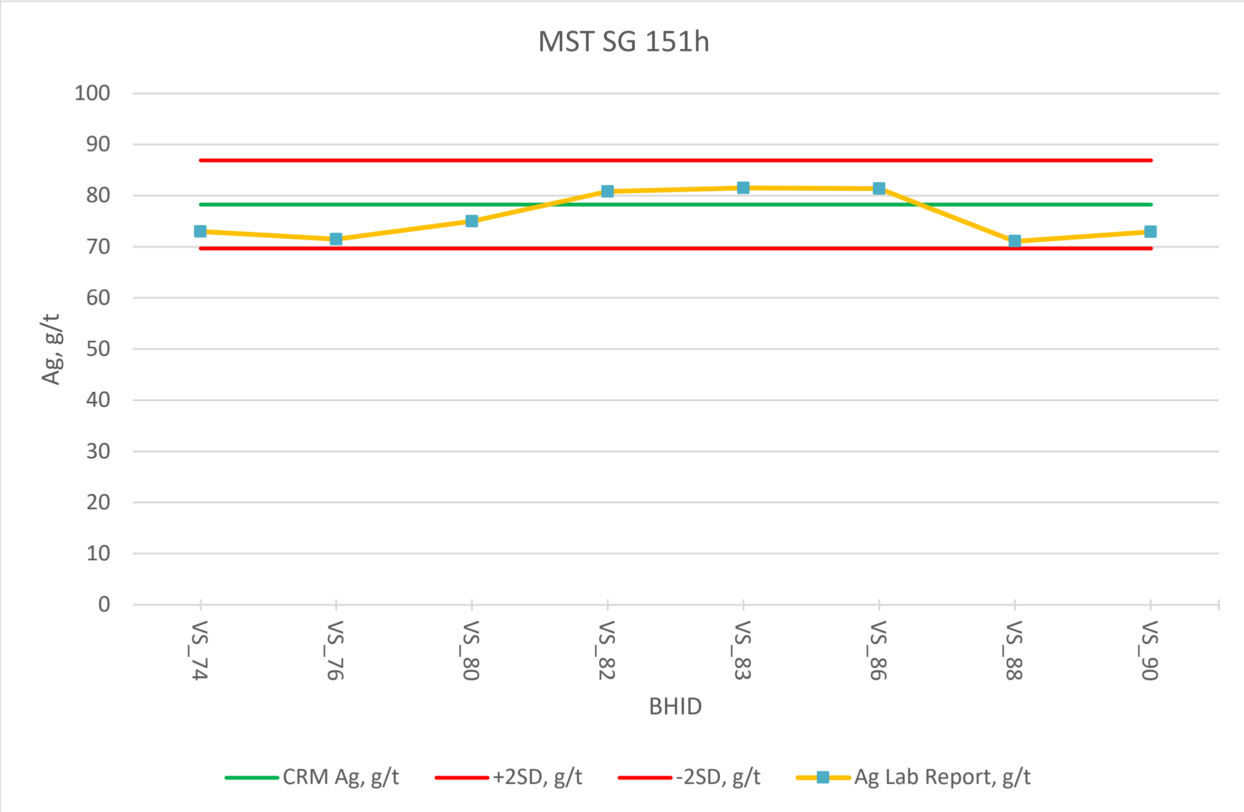


Figure 10.33: MST SG 151h, Ag, CRM Assaying Results

10.5.1.3 Field Duplicates

Data for 953 field duplicates representing second halves of core and/or additional/parallel channel samples from trenches were provided for the review. Initial grade for majority of samples (666) was less than 5g/t Ag.

The data show that HARD value for 70% of duplicates is less than 10% that is satisfactory for precision of initial samples and their field duplicates (Figure 10.34).

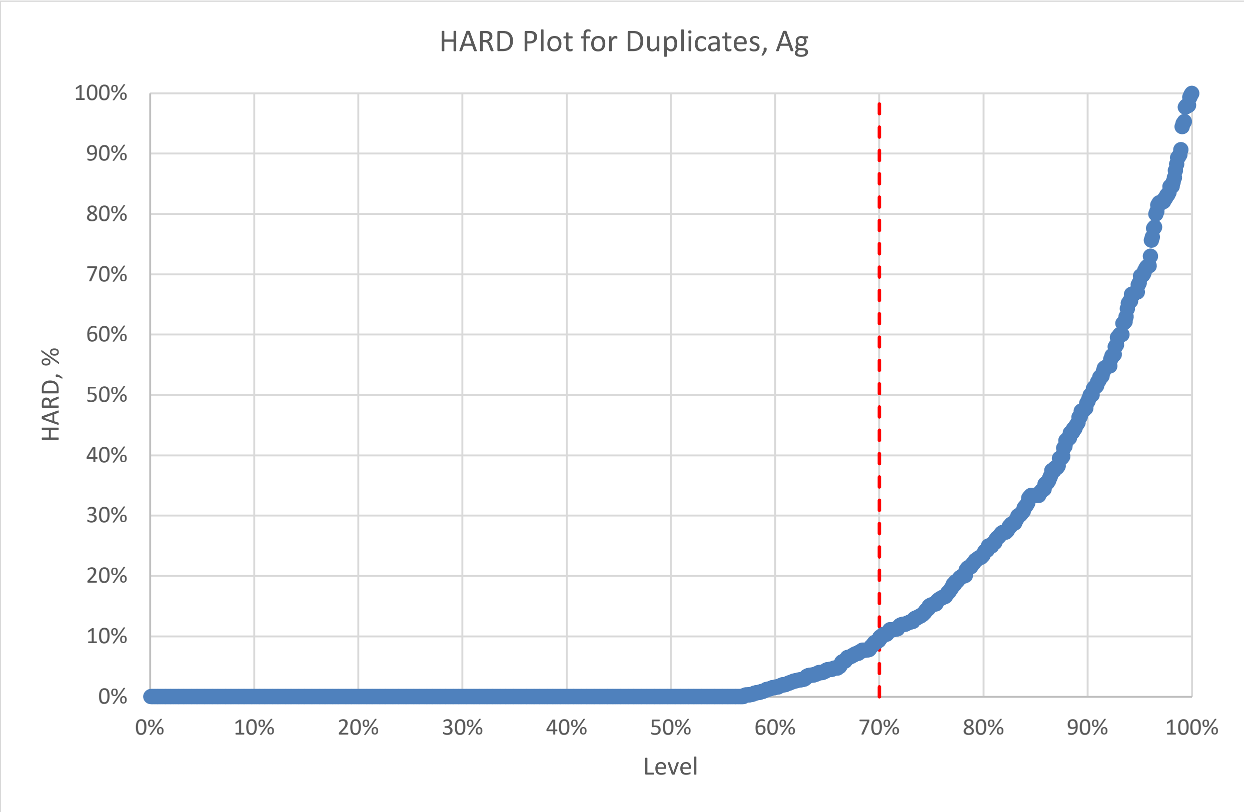


Figure 10.34: HARD Plot for Field Duplicates, Ag

Correlation plot for silver values in stream samples and their duplicates is shown in Figure 10.35.

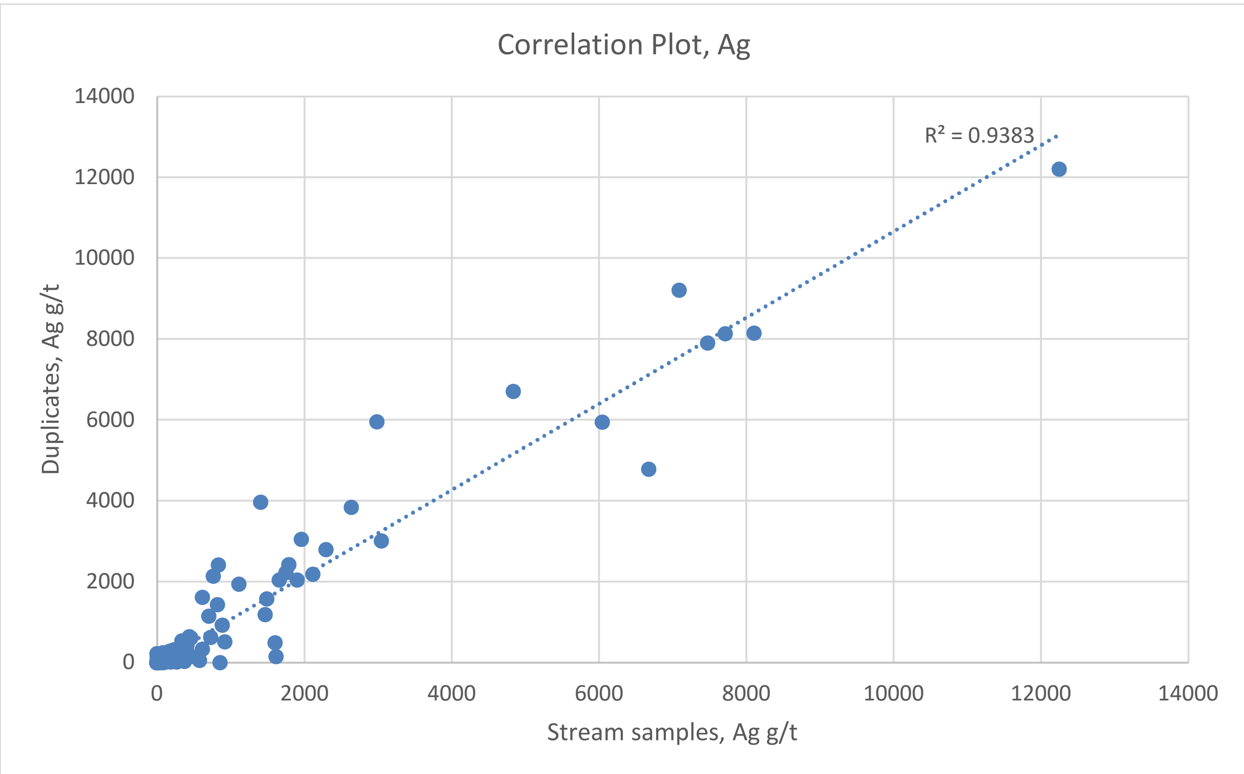


Figure 10.35: Field Duplicates Correlation Plot, Ag

Data for Pb and Zn were provided for 414 pulp duplicates. HARD value is within 10% of precision level for 71.2% and 72.4% samples for lead and zinc respectively. HARD plots for these metals are represented in Figure 10.36 and Figure 10.37.

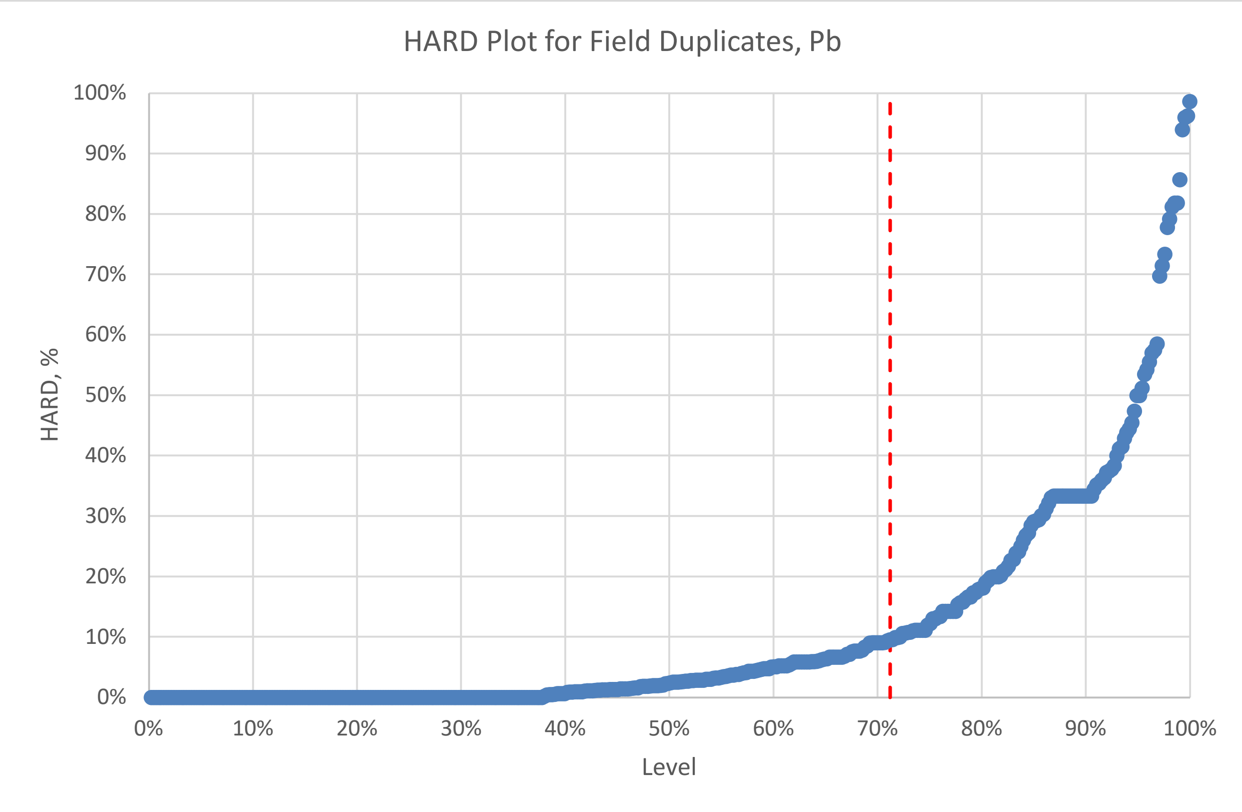


Figure 10.36: HARD Plot for Field Duplicates, Pb

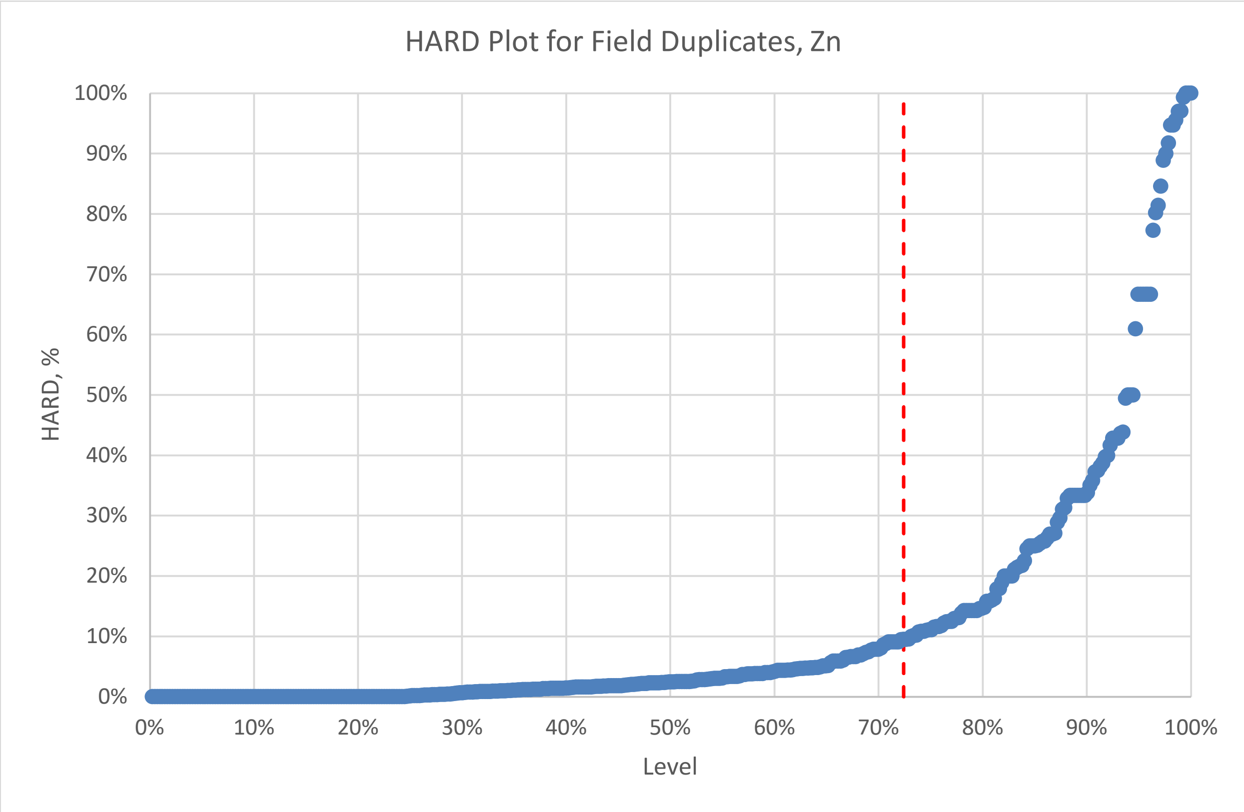


Figure 10.37: HARD Plot for Field Duplicates, Zn

Correlation plots for Pb and Zn for stream samples and duplicates are shown in Figure 10.38 and Figure **10.39**.

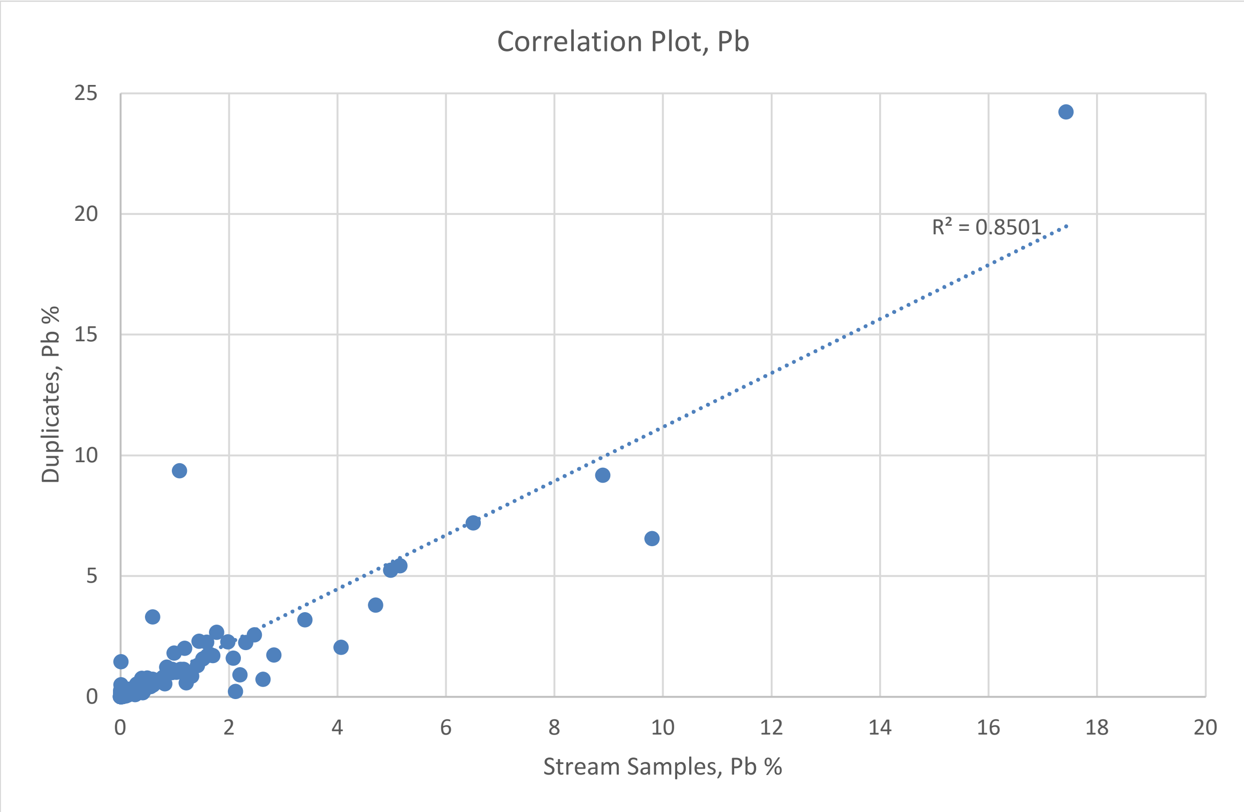


Figure 10.38: Correlation Plot for Field Duplicates, Pb.

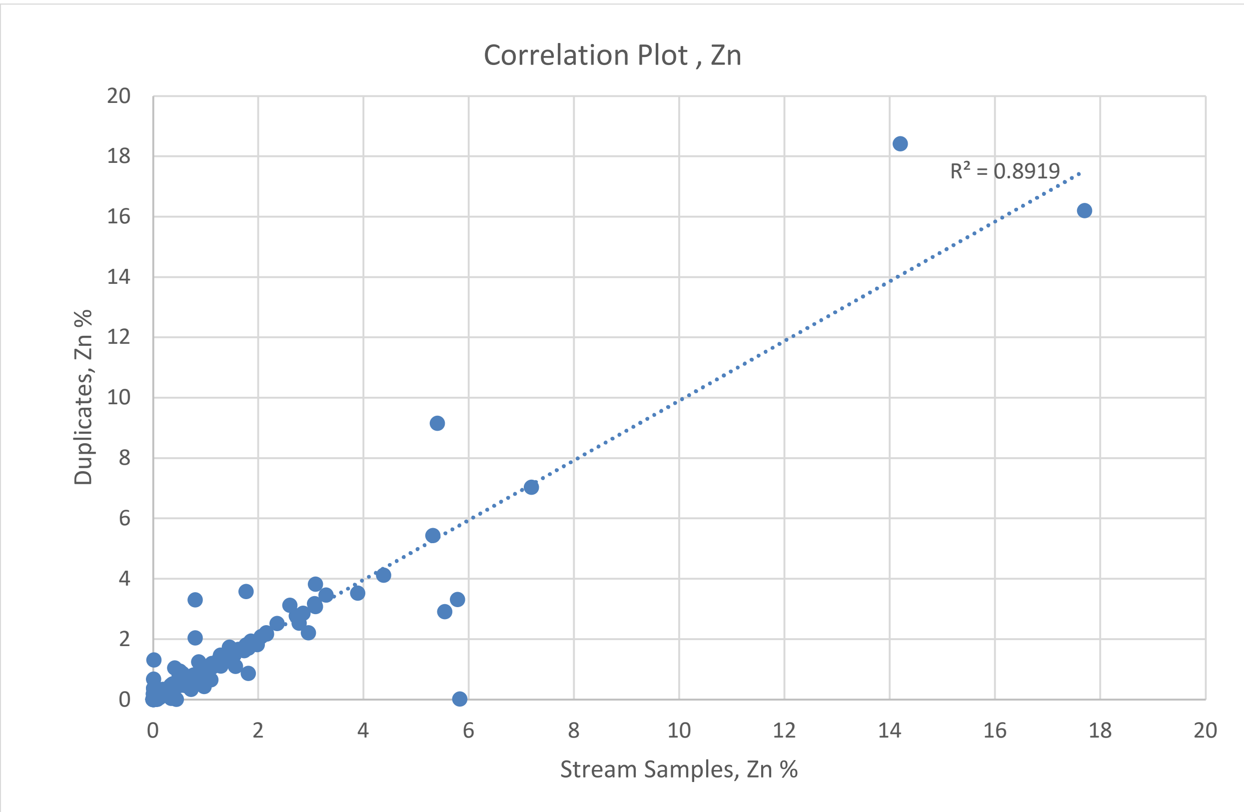


Figure 10.39: **Correlation Plot for Field Duplicates, Zn.**

10.5.2 Summary of QA/QC Risks

The WAI review of quality control data has identified a number of risks within the sample data. These risks are summarised in Table 10.5. It should be noted that Table 10.5 does not provide a quantitative risk assessment but gives an indication as to where WAI considers the risk lie within the sampling data.

A six-score classification has been employed where:

- 1 - 2 ('low' risk): Little or no perceived risk, or low uncertainty;
- 3 - 4 ('moderate' risk): Risk present which could lead to small material error in the resource model;
- 5 - 6 ('high' risk): This feature could lead to material error in the resource model (high uncertainty).

Table 10.5: Risk Matrix Vertikalnoye QA/QC Review

Sample Type	Risk	Comment
Blanks	5	Blanks assaying results for Ag show their possible contamination. Ag grade for more than 10% of blanks from ore sections was higher than 50g/t – cut-off grade for mineralisation delineation. In general, samples with higher silver grades are preceded by samples with high (more than 100g/t to first/several thousand g/t) grade of this metal. Zinc and lead blanks assaying results are satisfactory.
CRMs	2	CRM assaying results for Ag are satisfactory, there are some insignificant deviations for Zn and Pb assaying results.
Field Duplicates	2	Precision based on HARD data is at an acceptable level, more than 70% of samples are below error limit of 10%.

Total risk related to the quality of sampling, sample preparation and assaying is considered to be 'moderate' - risk present which could lead to small material error in the resource model. However, WAI would recommend that the QA/QC procedures to be improved by sampling and sample preparation of field duplicates as there is a risk of sample contamination.

10.6 Quality Control Analysis – Mangazeisky North

10.6.1 Exploration 2009 – 2016.

During exploration activities in 2009-2016 on Northern Mangazeisky blank samples and certified reference materials (CRM) were employed for QA/QC purposes, field duplicates of samples were used for internal control. Project geologists oversee control samples insertion into the samples stream. Field duplicates and blank samples were inserted before crushing, and CRMs were inserted after samples are ground, labelled and registered in a log.

At the time of this report a total of 3,446 samples (Table 10.6) have been analysed and provided for review and the quality control samples provided consist of analysis for 171 internal CRMs (4.9%), 159 field duplicate samples (4.6%), and 172 blank samples (5.0%).

Table 10.6: Summary Table of Control Samples

Type of Control Sample	Total	With Assay Results		
		Ag	Pb	Zn
Stream Samples	3,446	3,443	2,826	3,163
Blank Samples	172	172	83	83
Field Duplicate Samples	159	159	120	148
CRMs	171	171	159	160

10.6.1.1 Blanks

Barren material of host rocks was used as blank samples. It was reported that blank samples were inserted at 1:20 rate, CRMs – at 1:20 rate, and duplicates were also inserted at 1:20 rate.

The results of the blank analysis for Ag are shown in Figure 10.40 with 22 samples showing marginal fails of >5.0g/t Ag. Significant exceedances (>50.0g/t Ag) were identified for 9 samples with maximal Ag grade of 261.0g/t.

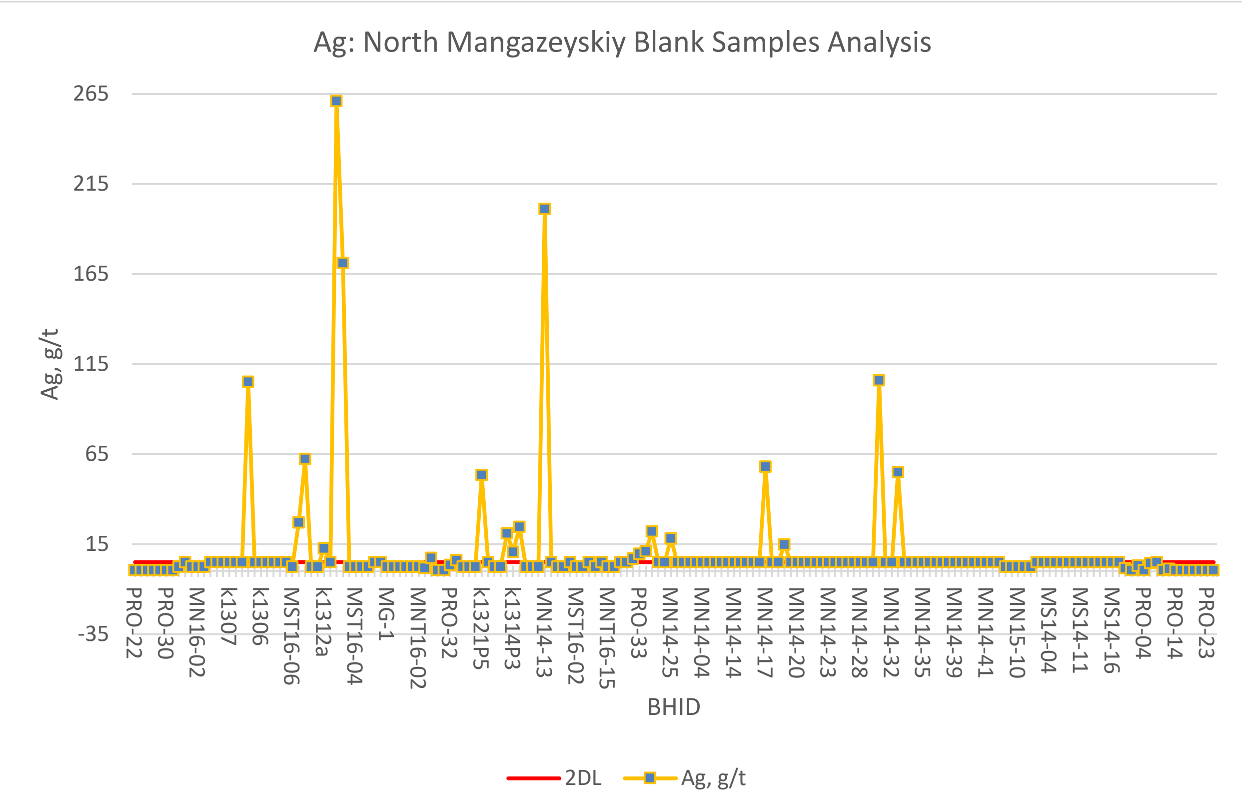


Figure 10.40: Blank Samples Analysed for Ag on North Mangazeyskiy

Pb and Zn were detected in 83 blank samples. Out of them 22 samples returned Pb grade that was twice the accepted detection limit (0.02% Pb), and only 16 samples out of these 55 had Pb grade >0.25%. The results of blank samples analysis for Pb are presented in Figure 10.41

In assays for Zn, 5 sampled returned Zn grade that was twice the accepted detection limit (0.02%Zn), 1 sample out of them had Zn grade >0.25%. The results of blank samples analysis for Zn are presented in Figure 10.42.

In general, the results of blank samples analysis indicate a potential contamination of samples during the sample preparation process.

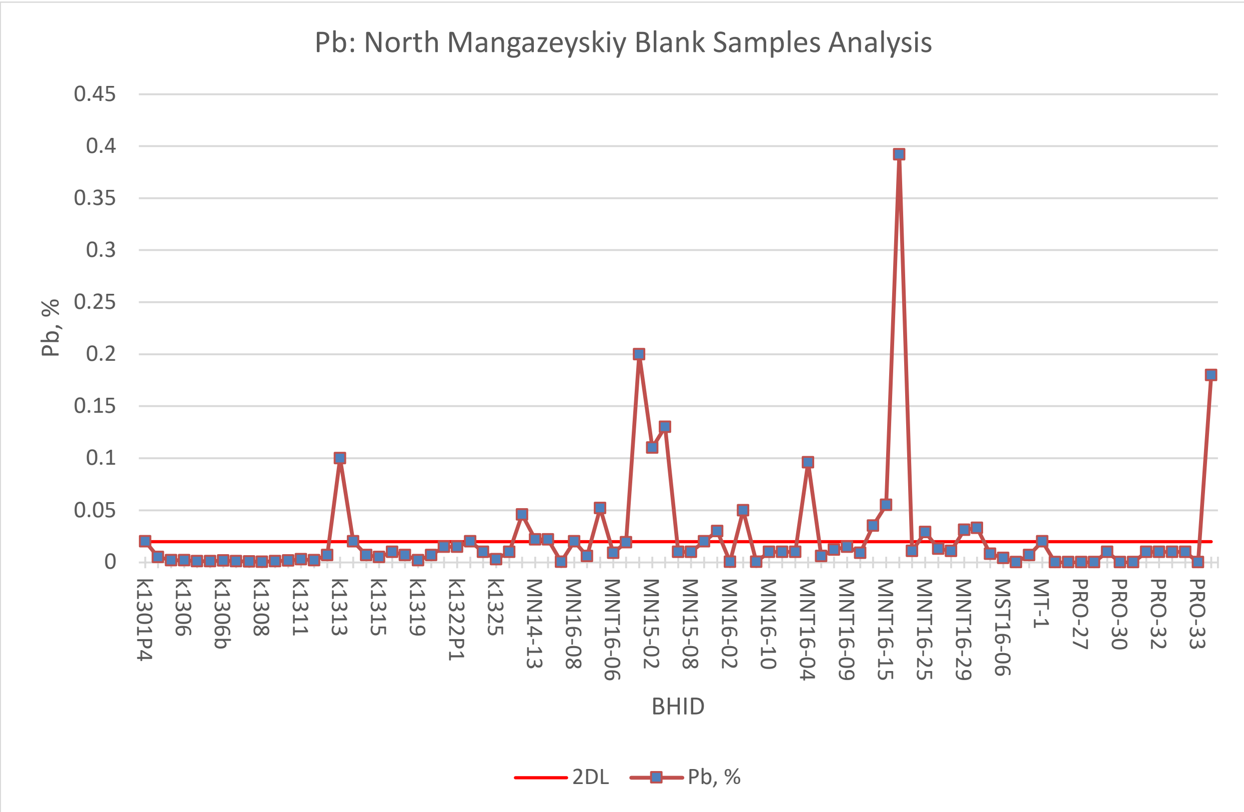


Figure 10.41: Blank Samples Analysed for Pb on North Mangazeyskiy

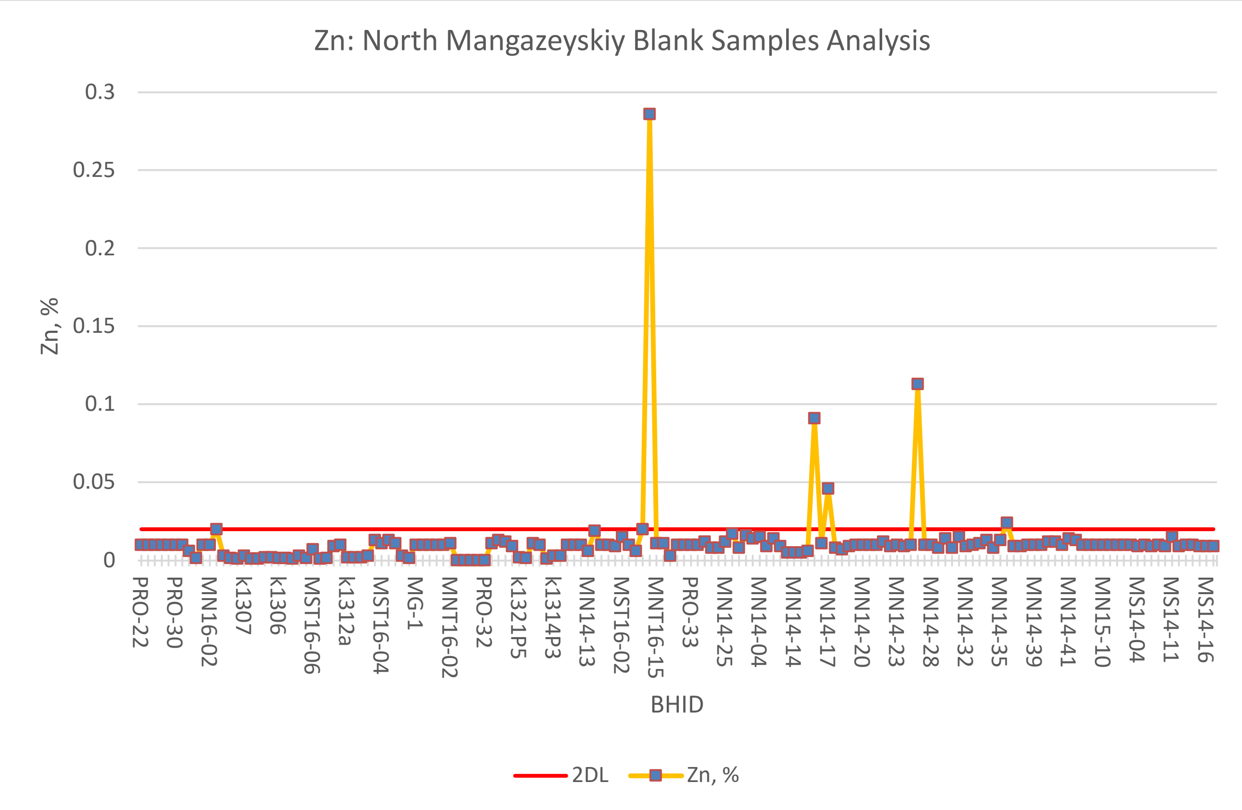


Figure 10.42: Blank Samples Analysed for Zn on North Mangazeyskiy

10.6.1.2 Certified Reference Materials (CRM)

Nine certified reference materials (CRMs) sourced from GEOSTATS Pty Ltd (Australia), STC Minstandard of St Petersburg, and Irgiredmet OJSC of Irkutst (Table 10.7).

Table 10.7: List of Certified Reference Materials		
№№	**CRM**	**Manufacturer**
1	GBM 906-6	GEOSTATS Pty Ltd, Australia
2	GBM 913-13	
3	GBM310-16	
4	GBM909-11	
5	GBM913-13	
6	СОП 01-2016 (SOP 01-2016)	Irgiredmet OJSC
7	СОП 02-2016 (SOP 02-2016)	
8	СОП 03-2016 (SOP 03-2016)	
9	MST SG 186	

The recommended values and number of assays for each CRM are listed in Table 10.8. Laboratory certificates have been provided for all but one of the CRMs. CRM limits are provided as permitted allowed absolute error (based on >95% of samples being within that target) rather than the more usual standard deviation limits.

In general, a good precision of the results of laboratory assays for Ag and certified valued was noted. The highest deviations are typical for CRMs with low Ag grades (<5g/t) that are close to the assays' detection limits.

The majority of assay results beyond allowed error limits with meaningful zinc contents were shown for GBM 310-16 and GBM 909-13 CRMs generally returning lower Zn grades in comparison with CRMs.

Despite of this, risk for MRE might be considered as insignificant.

Table 10.8: Summary of CRMs Data for North Mangazeyskiy

CRM	Metal, Unit	Grade	Standard Deviation	Expanded Uncertainty	Number of CRMs	Beyond Allowed Absolute Error	%% of Satisfactory Assays
GBM906-6	Ag, g/t	389.7	21.1		57	1	98.2%
	Zn, g/t	210	14		57	20	64.9%
	Pb, g/t	290	14		57	20	64.9%
GBM913-13	Ag, g/t	74,1	3.9		16	0	100.0%
GBM310-16	Ag, g/t	314.3	14.9		32	5	84.4%
	Zn, g/t	170201	6825		31	5	83.9%
	Pb, g/t	112603	5008		32	23	28.1%
GBM909-11	Ag, g/t	25.5	1.7		9	0	100.0%
GBM909-13	Ag, g/t	127.3	6.8		32	0	100.0%
	Zn, g/t	68362	2363		32	16	50.0%
	Pb, g/t	8513	327		26	17	34.6%
СОП 01-2016 (SOP 01-2016)	Ag, g/t	3,21		+/- 0,28	7	3	57.1%
	Zn, %	0,129		+/- 0,007	6	1	83.3%
	Pb, %	0,083		+/- 0,004	6	1	83.3%
СОП 02-2016 (SOP 02-2016)	Ag, g/t	73,7		+/- 3,2	6	0	100.0%
	Zn, %	0,86		+/- 0,02	3	0	100.0%
	Pb, %	2,45		+/-0,09	3	0	100.0%
СОП 03-2016 (SOP 03-2016)	Ag, g/t	124,4		+/- 6,2	3	0	100.0%
MST SG 186	Ag, g/t	36		n/d	32		
	Zn, %	0,0053		n/d	10		
	Pb, %	0,035		n/d	10		

There are no data on allowed absolute error for MST SG 186 (6 samples) therefore results for these CRMs were not considered. The results of CRMs analyses are illustrated on Figure 10.43 to Figure **10.60**.

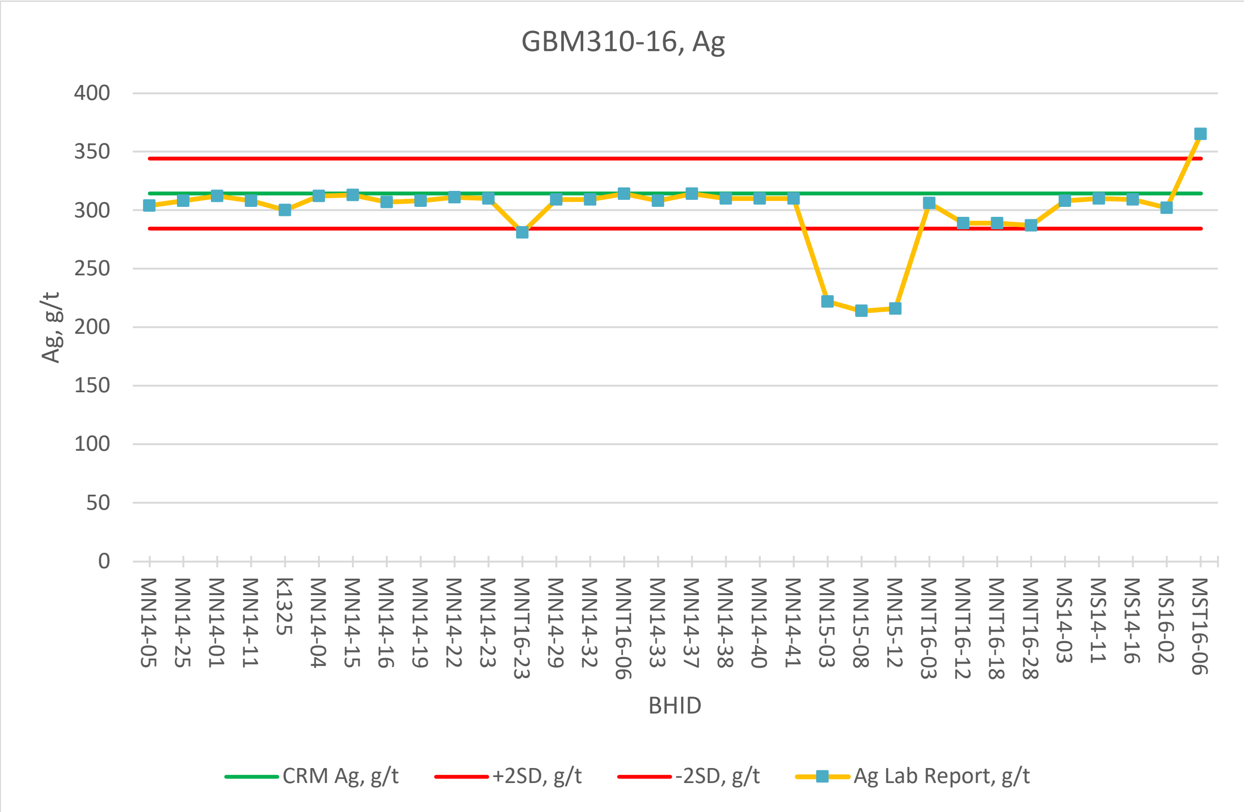


Figure 10.43: GBM 310-16, Ag, CRM Assaying Results

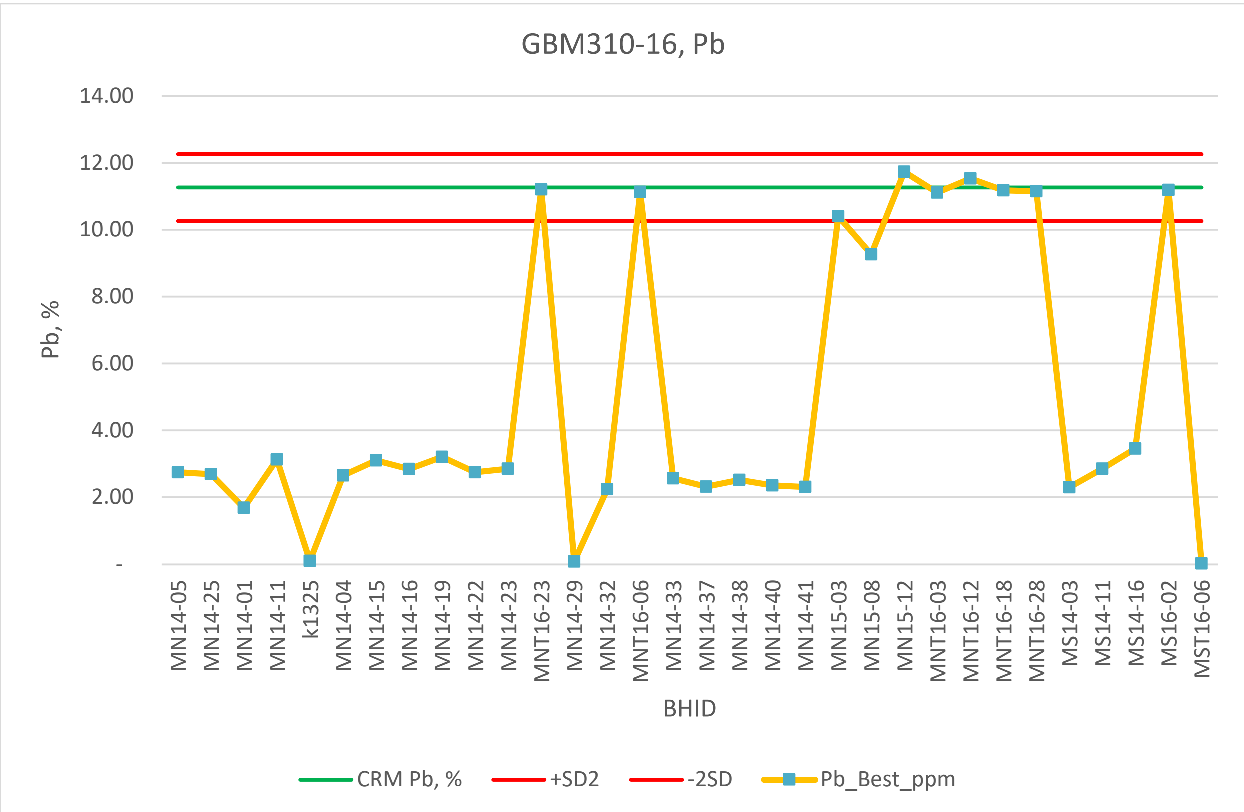


Figure 10.44: GBM 310-16, Pb, CRM Assaying Results

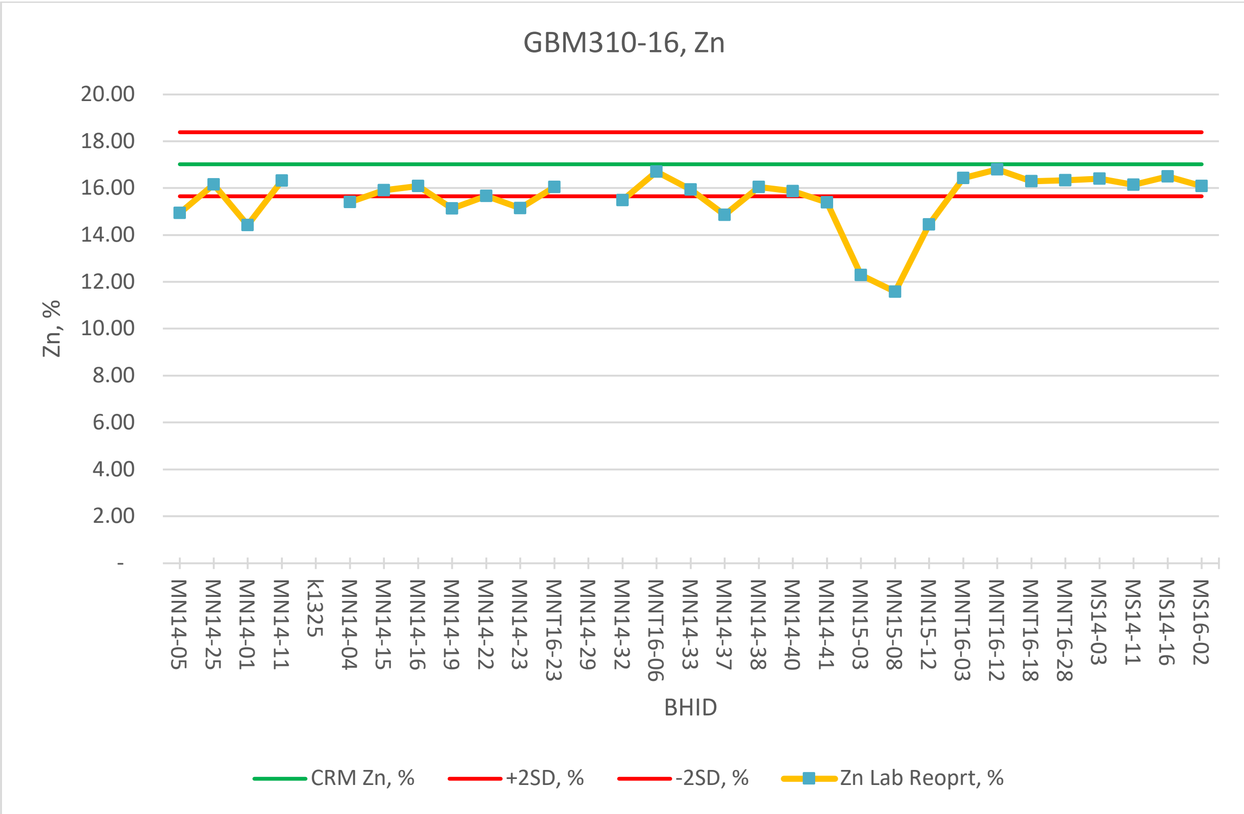


Figure 10.45: GBM 310-16, Zn, CRM Assaying Results

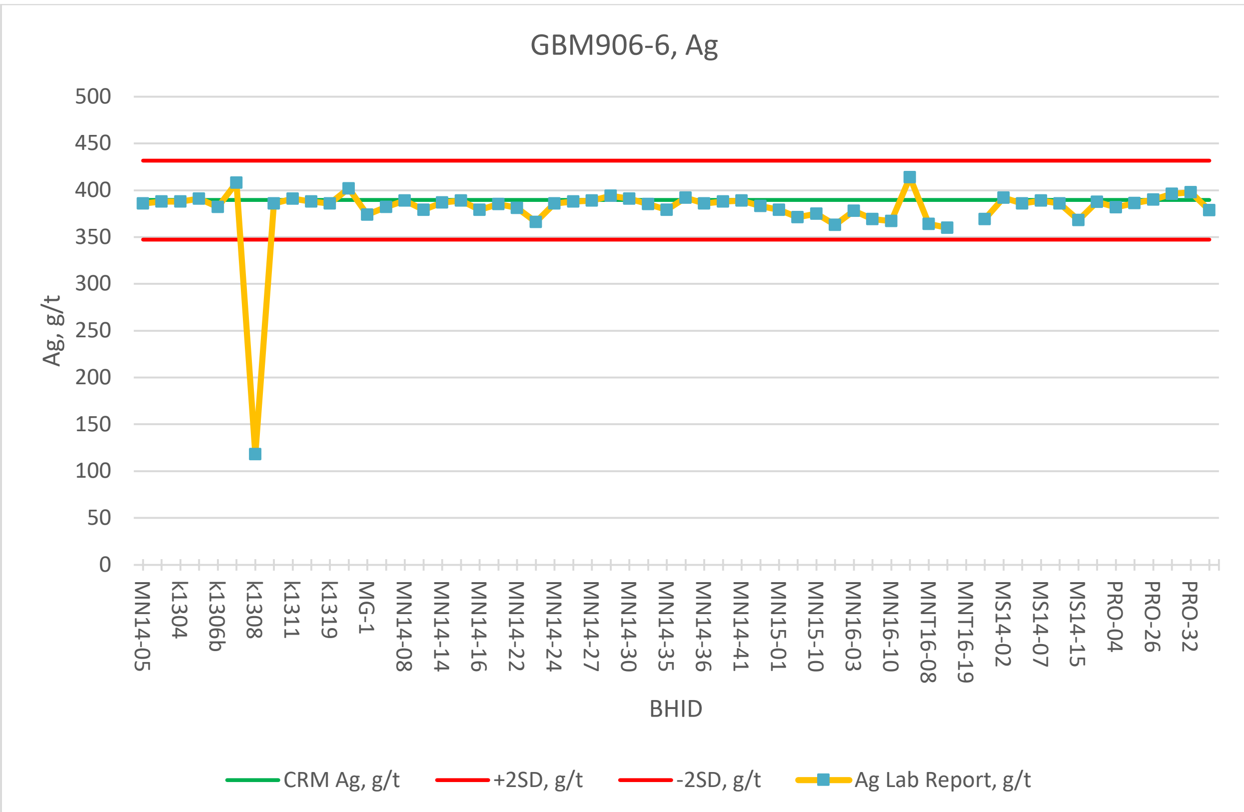


Figure 10.46: GBM 906-6, Ag, CRM Assaying Results

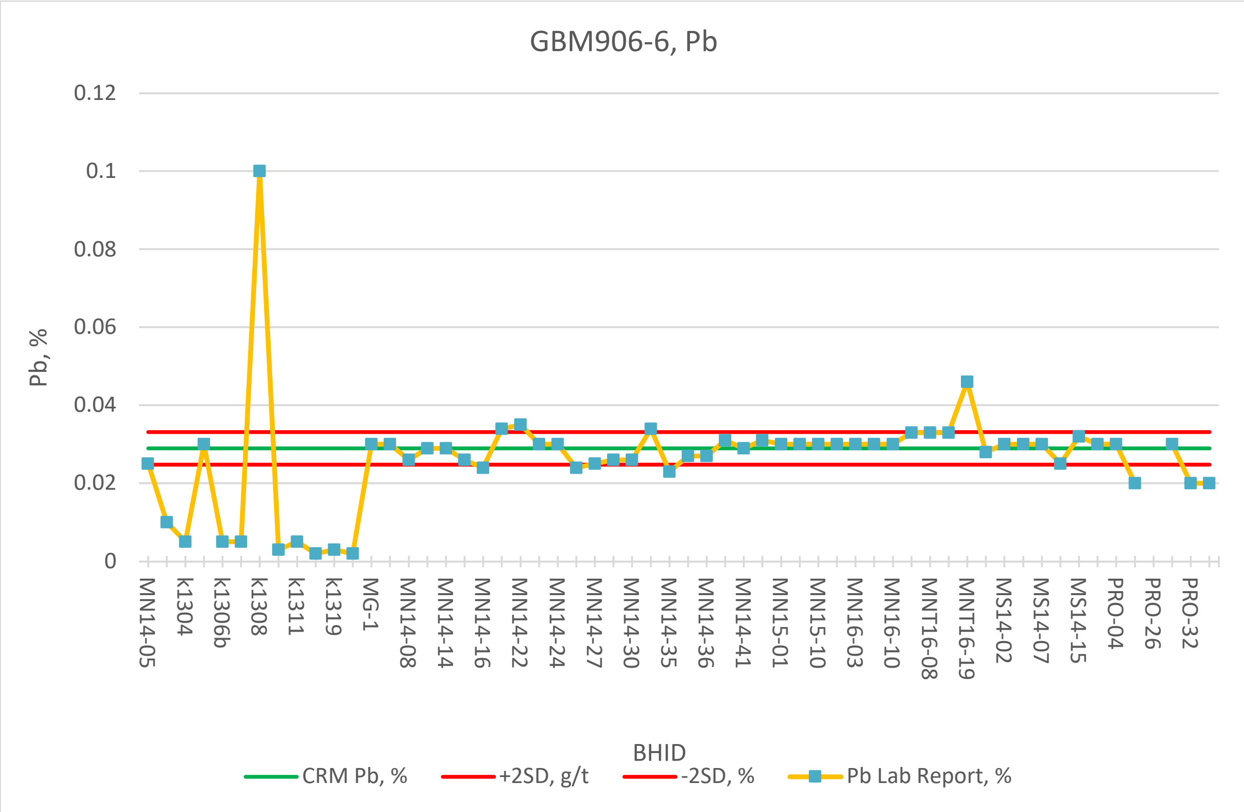


Figure 10.47: GBM 906-6, Pb, CRM Assaying Results

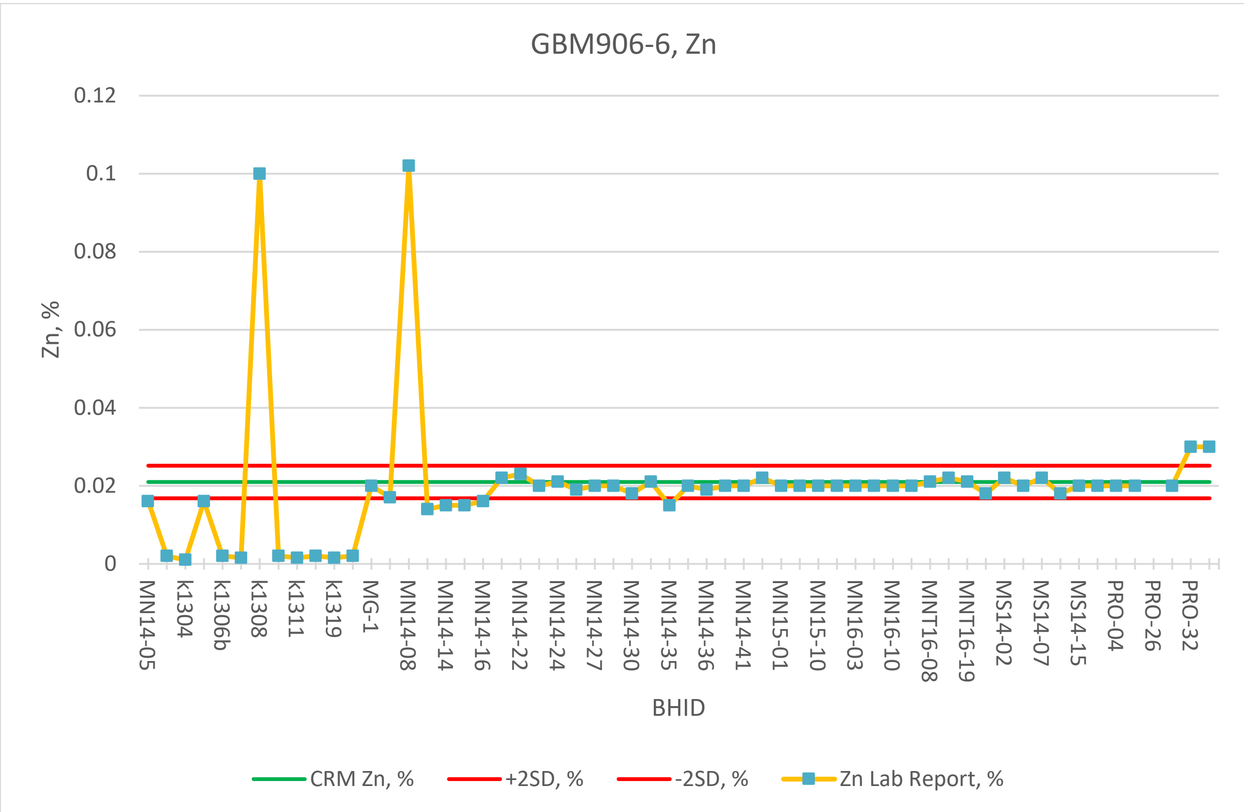


Figure 10.48: GBM 906-6, Zn, CRM Assaying Results

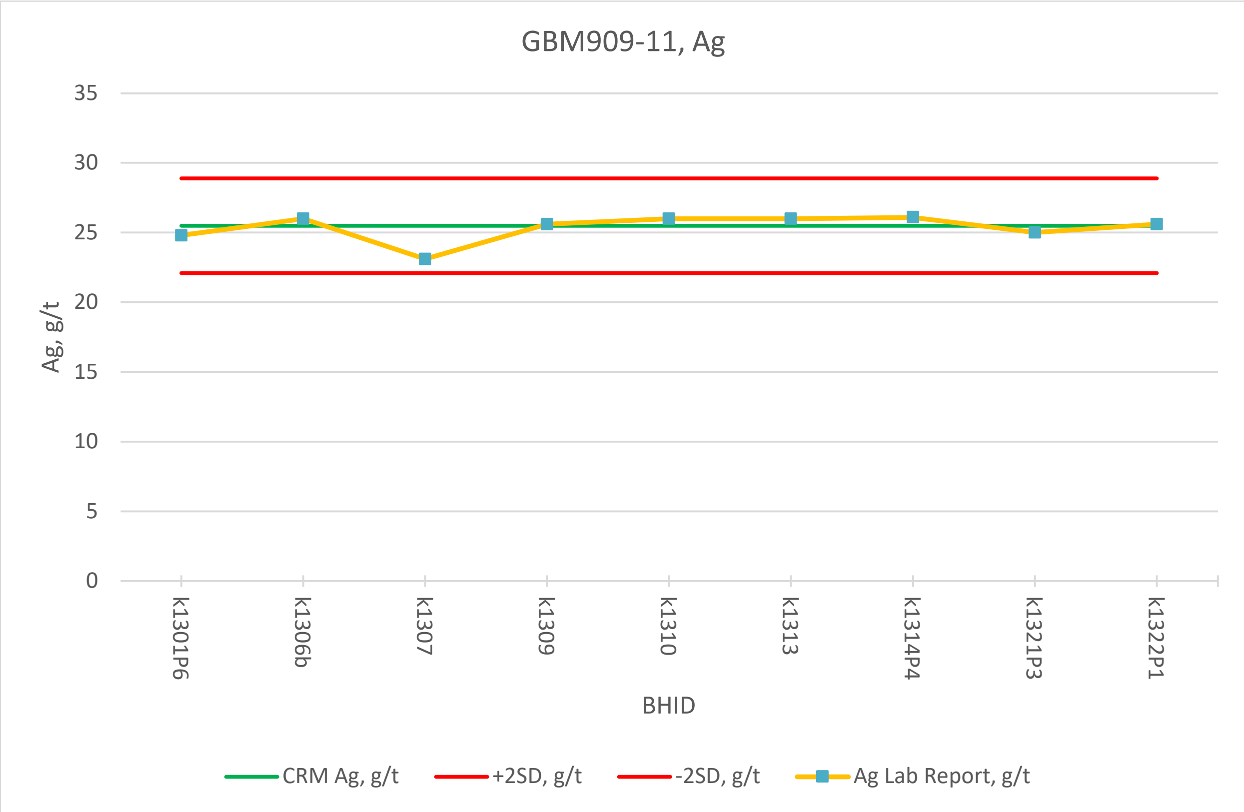


Figure 10.49: GBM 909-11, Ag, CRM Assaying Results

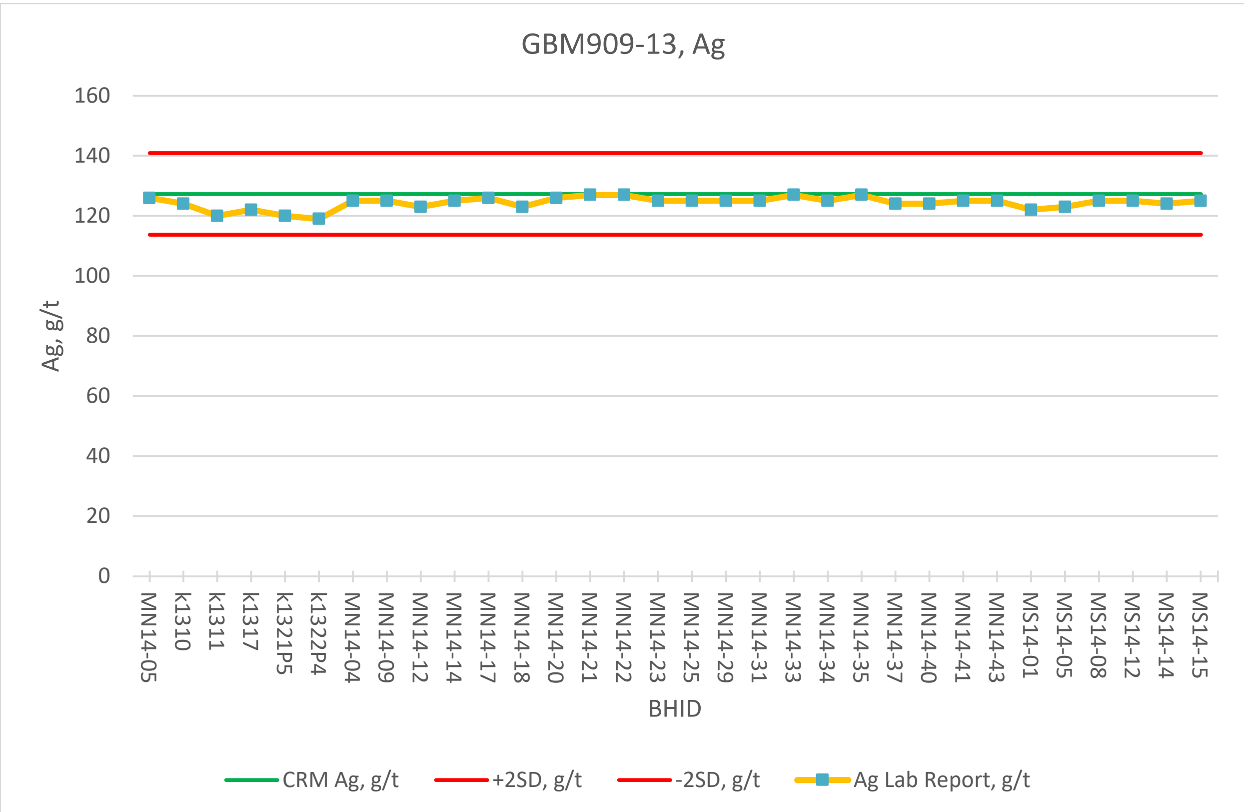


Figure 10.50: GBM 909-13, Ag, CRM Assaying Results

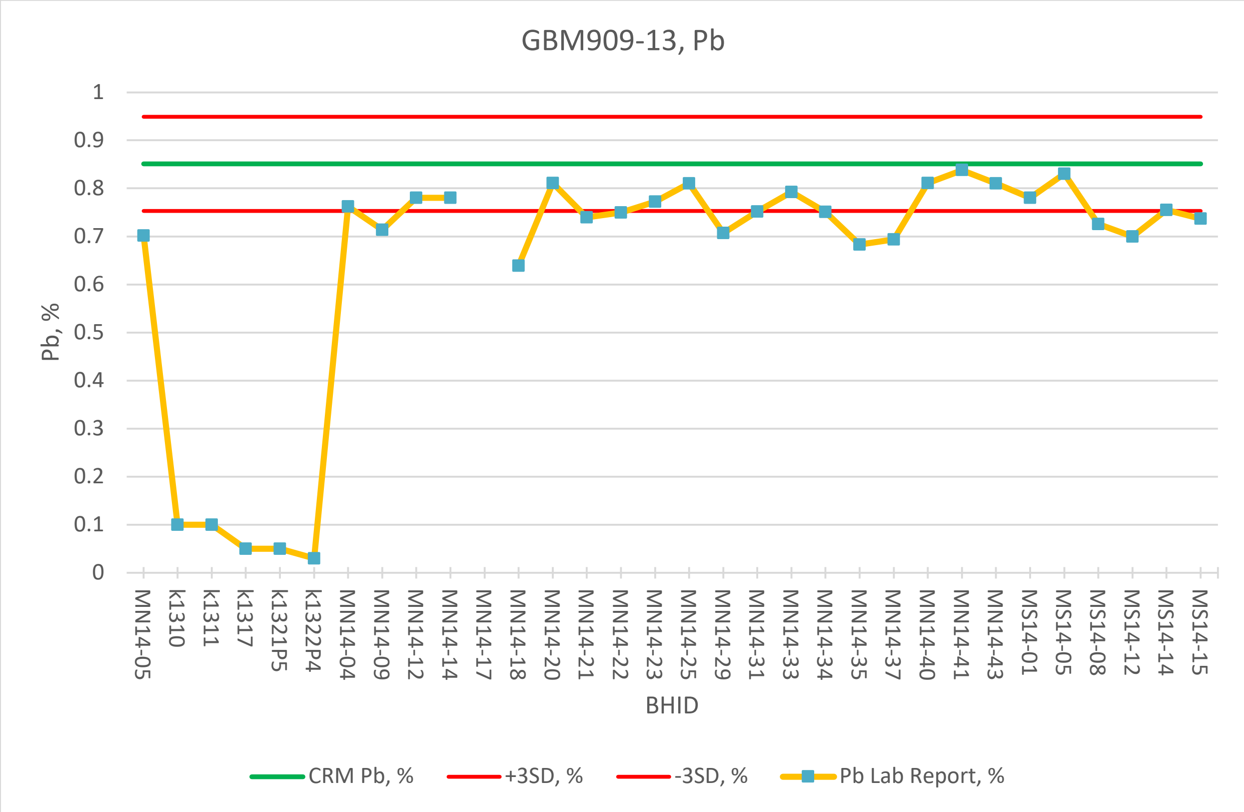


Figure 10.51: GBM 909-13, Pb, CRM Assaying Results

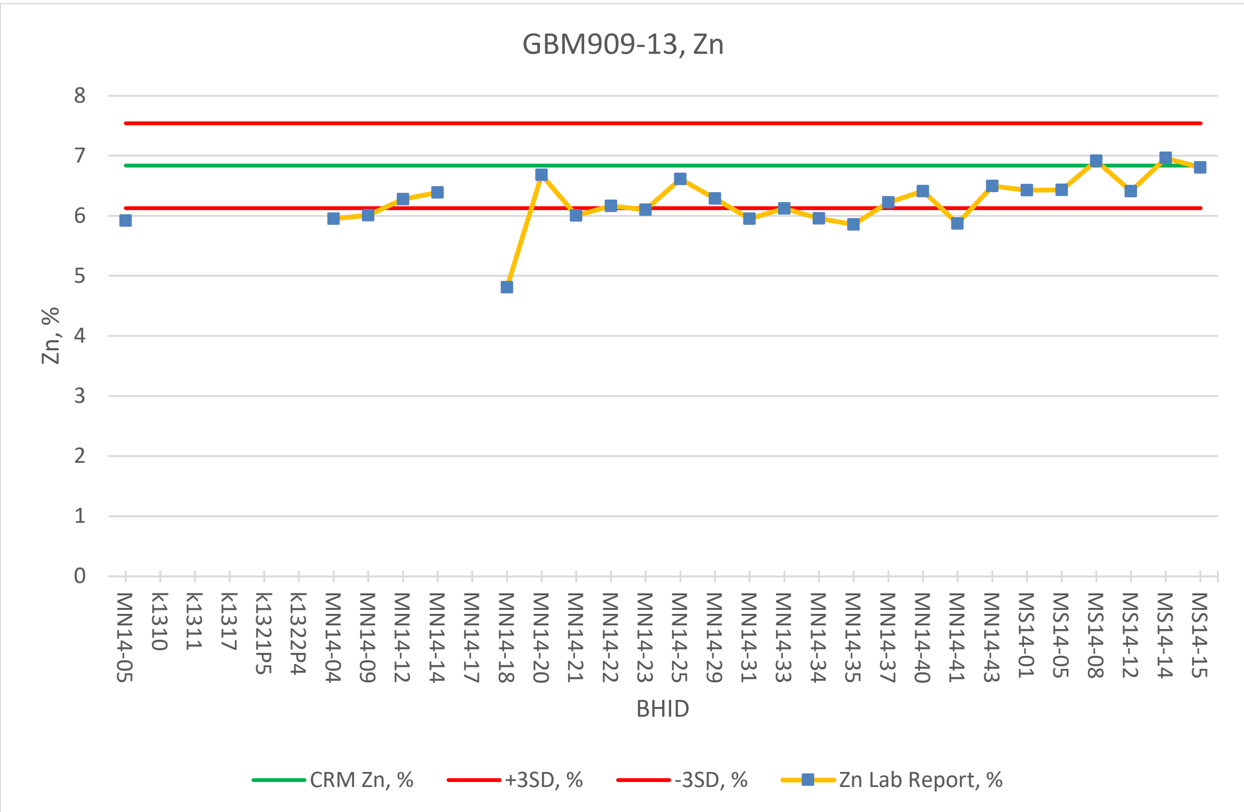


Figure 10.52: GBM 909-13, Zn, CRM Assaying Results

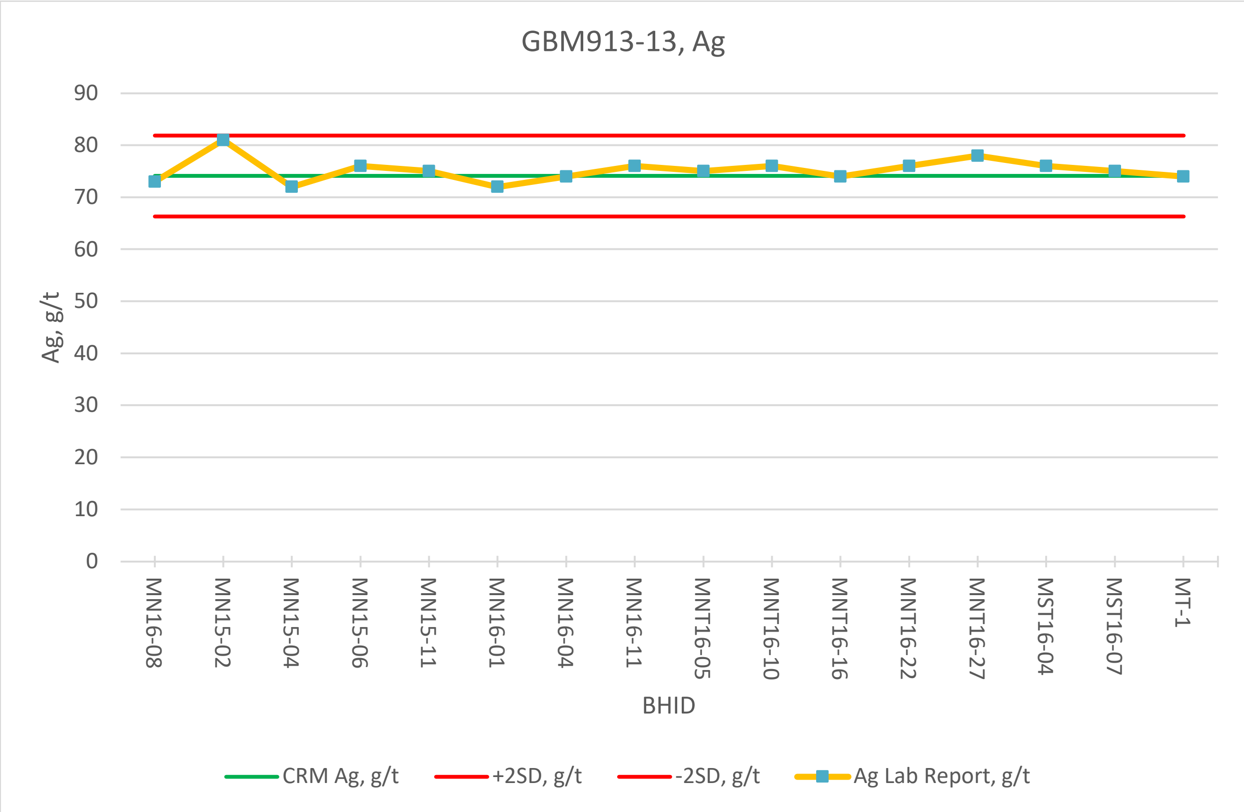


Figure 10.53: GBM 913-13, Ag, CRM Assaying Results

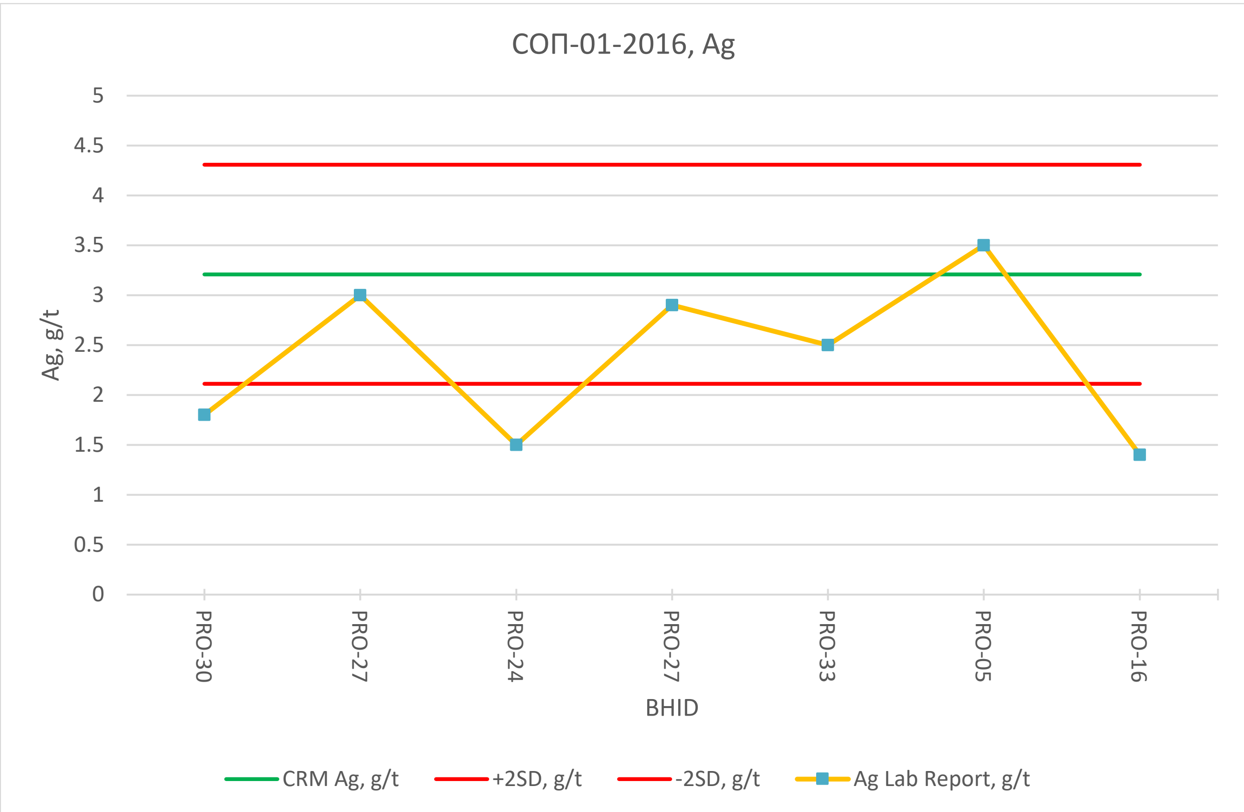


Figure 10.54: SOP-01-2016, Ag, CRM Assaying Results

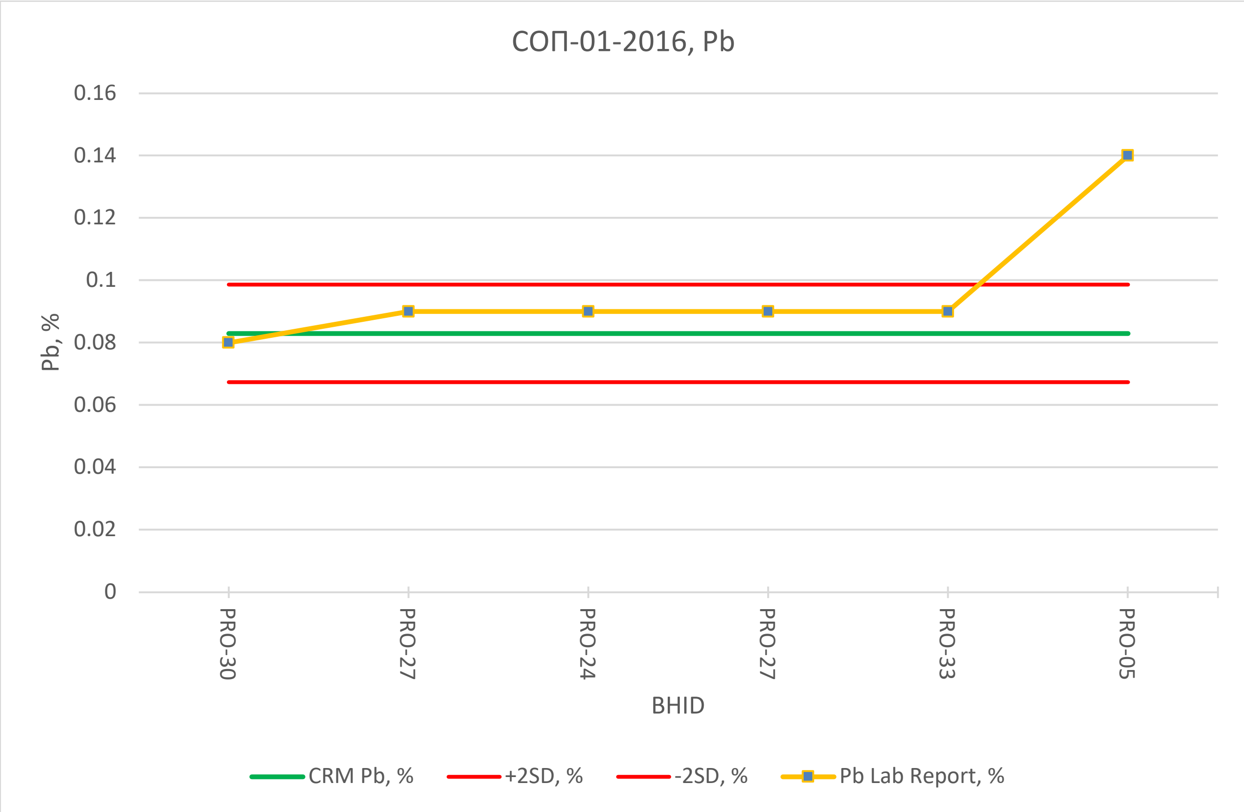


Figure 10.55: SOP-01-2016, Pb, CRM Assaying Results

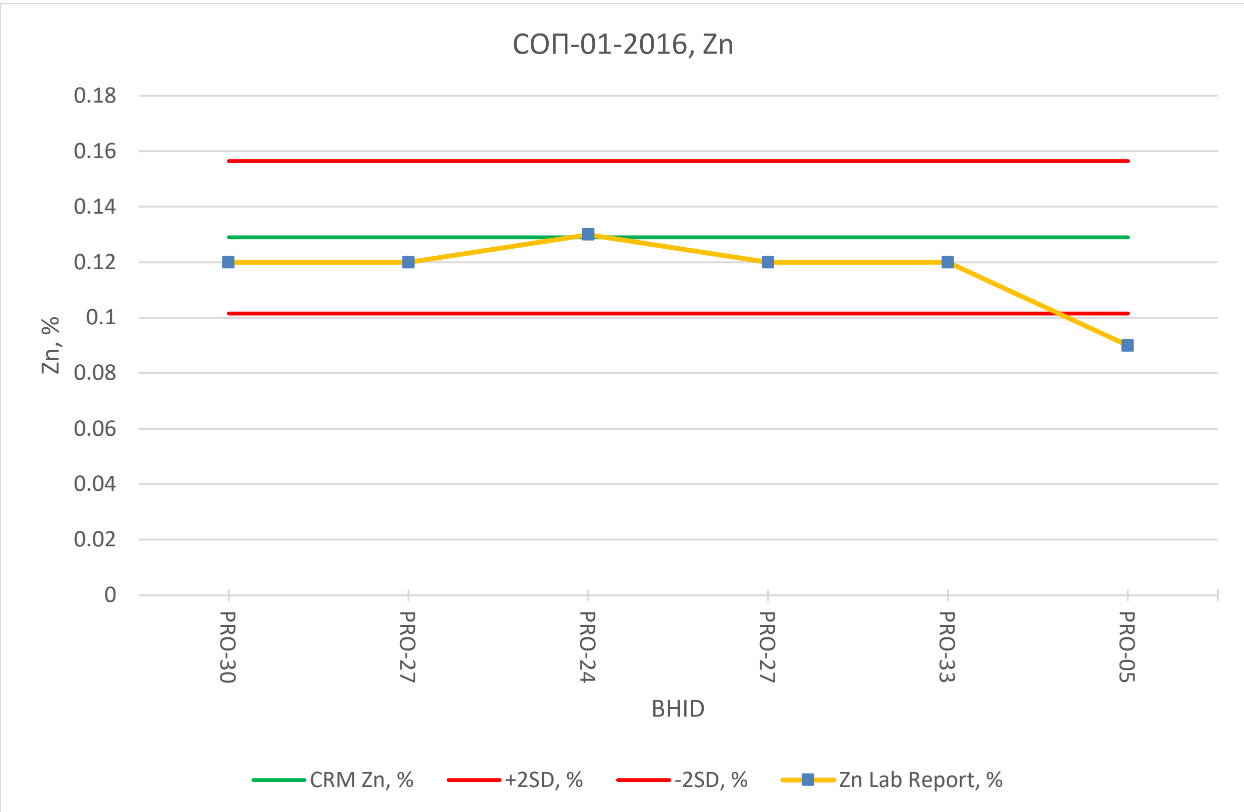


Figure 10.56: SOP-01-2016, Zn, CRM Assaying Results

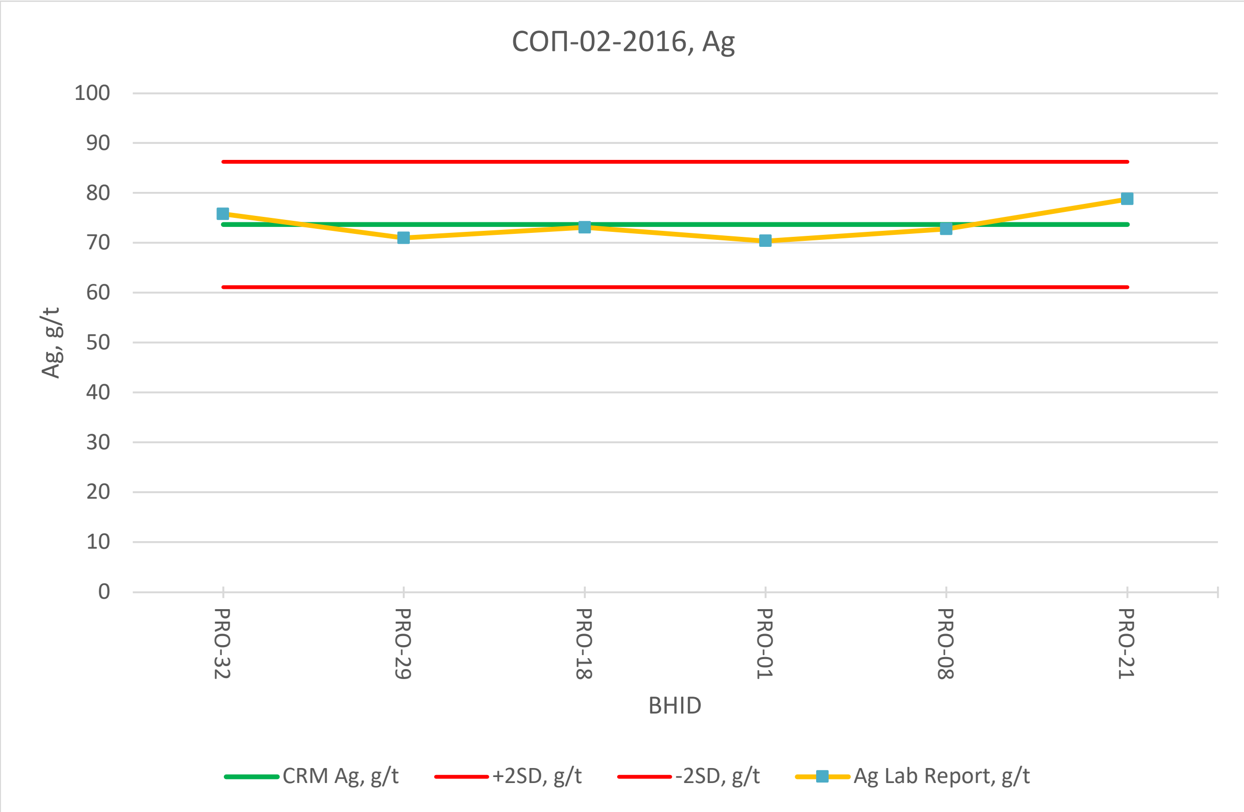


Figure 10.57: SOP-02-2016, Ag, CRM Assaying Results

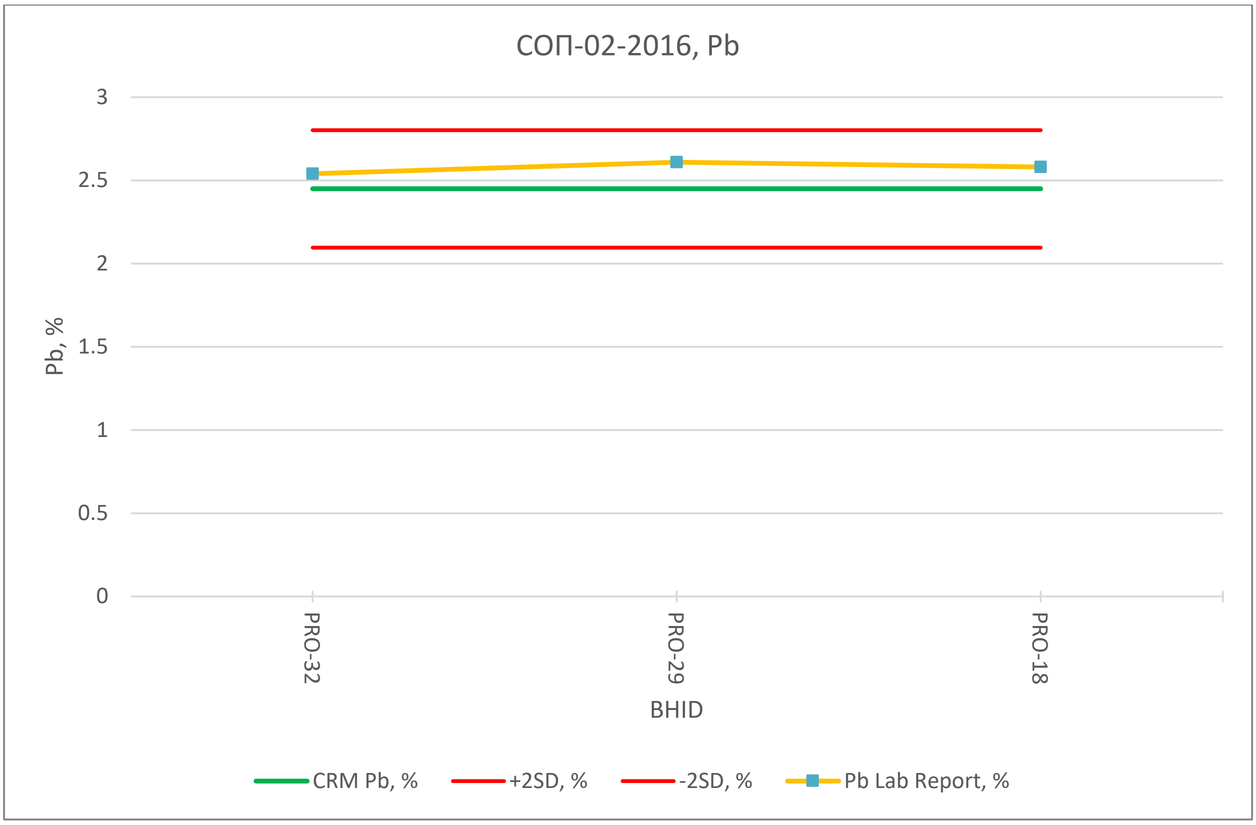


Figure 10.58: SOP-02-2016, Pb, CRM Assaying Results

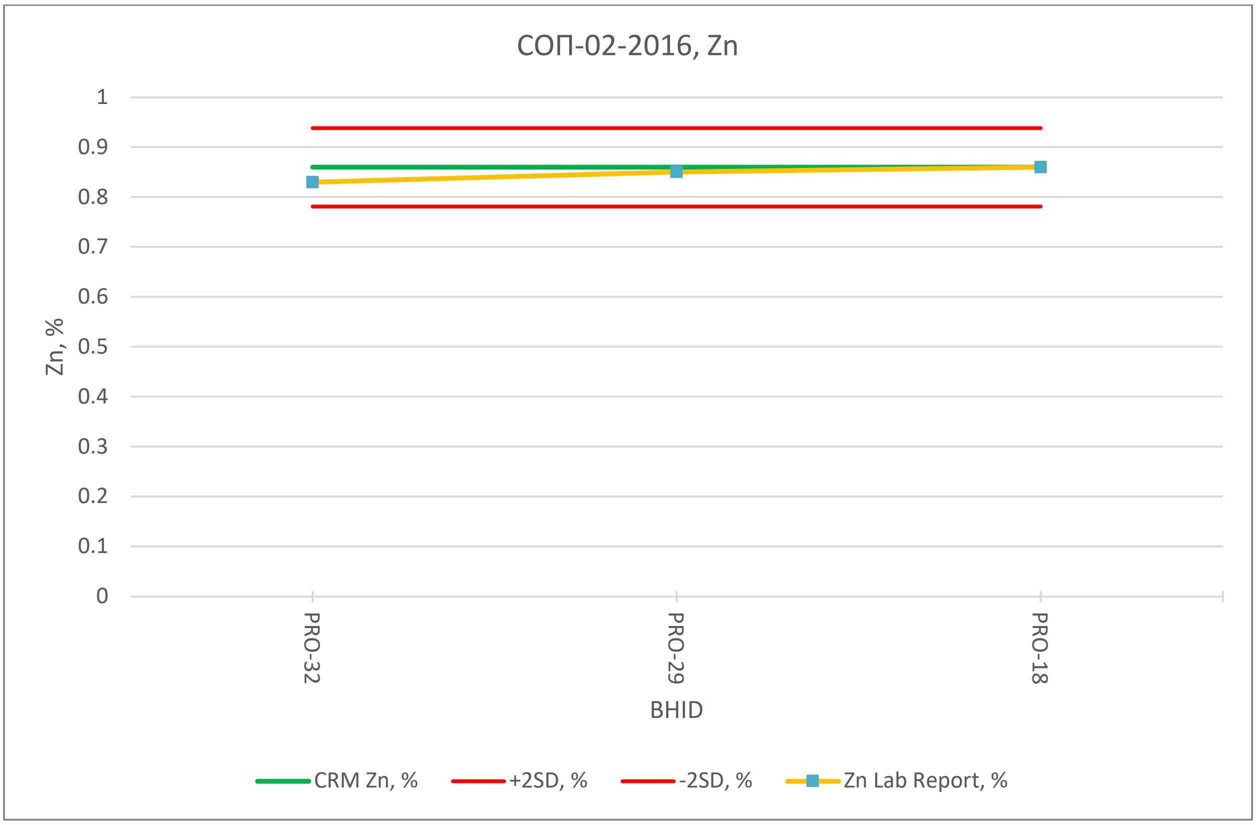


Figure 10.59: SOP-02-2016, Zn, CRM Assaying Results

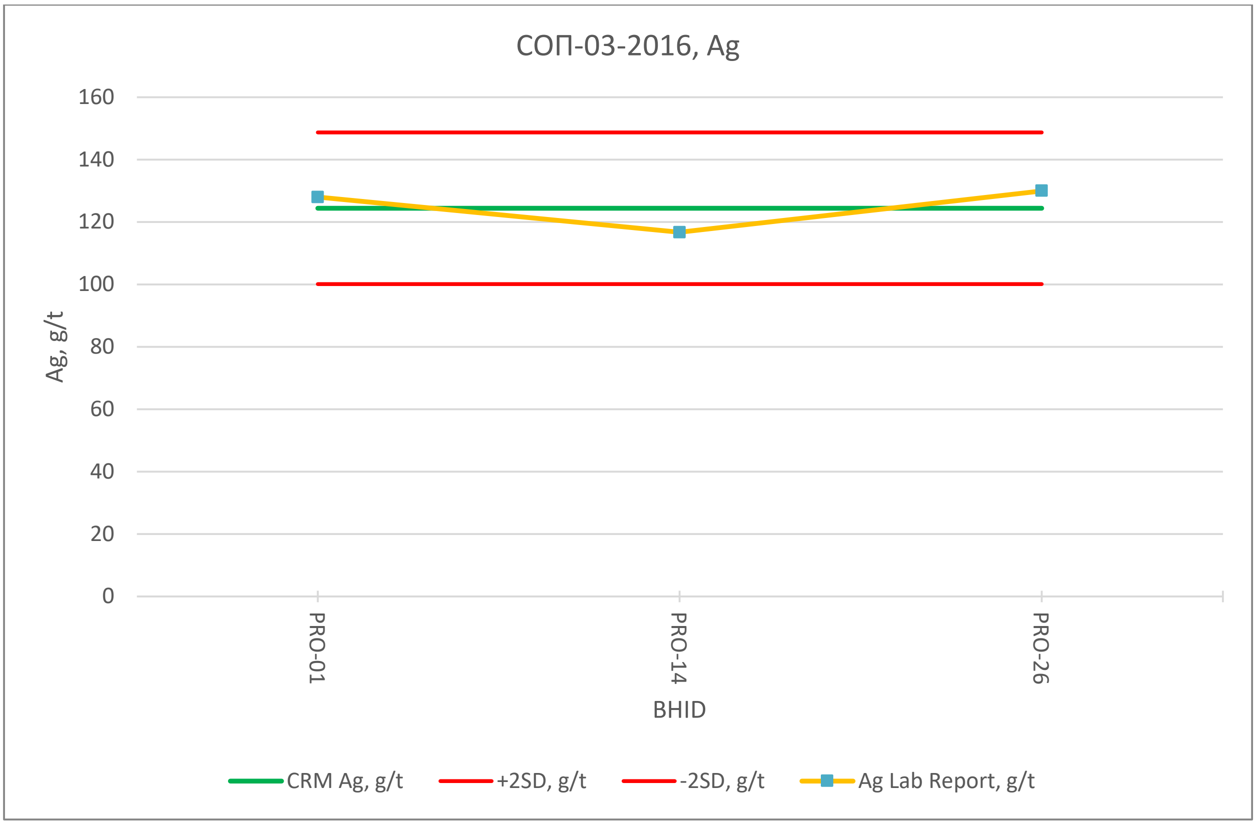


Figure 10.60: **SOP-03-2016, Ag, CRM Assaying Results**

10.6.1.3 Field Duplicates

Data for 159 field duplicates representing second halves of core and/or additional/parallel channel samples from trenches were provided for the review. Initial grade for majority of samples (111) was less than 5g/t Ag.

The data show that HARD value for 77% of duplicates is less than 10% that is satisfactory for precision of initial samples and their field duplicates (Figure 10.61).

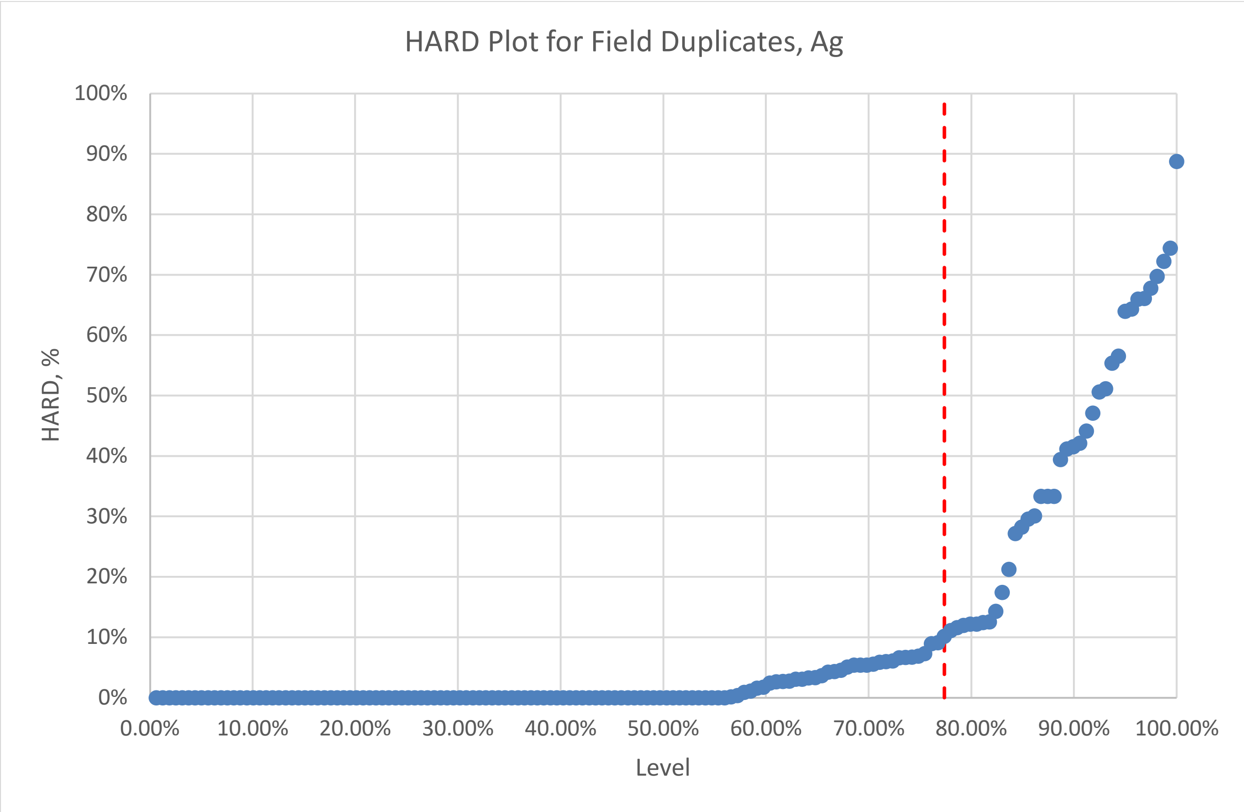


Figure 10.61: HARD Plot for Field Duplicates, Ag

Correlation plot for silver values in stream samples and their duplicates is shown in Figure 10.62.

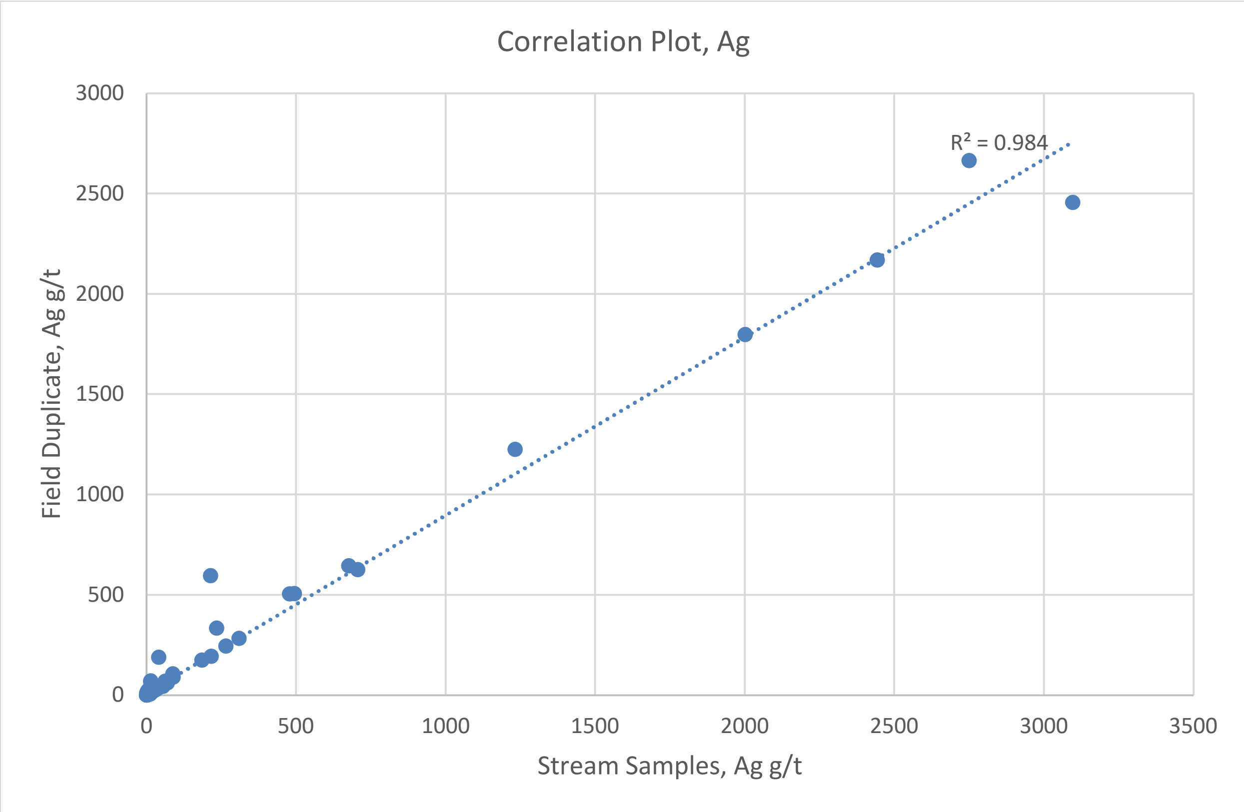


Figure 10.62: Field Duplicates Correlation Plot, Ag

Data for Pb and Zn were provided for 120 and 148 field duplicates, respectively. HARD value is within 10% of precision level for 73.3% and 77.7% samples for lead and zinc respectively. HARD plots for these metals are represented in Figure 10.63 and Figure **10.64**.

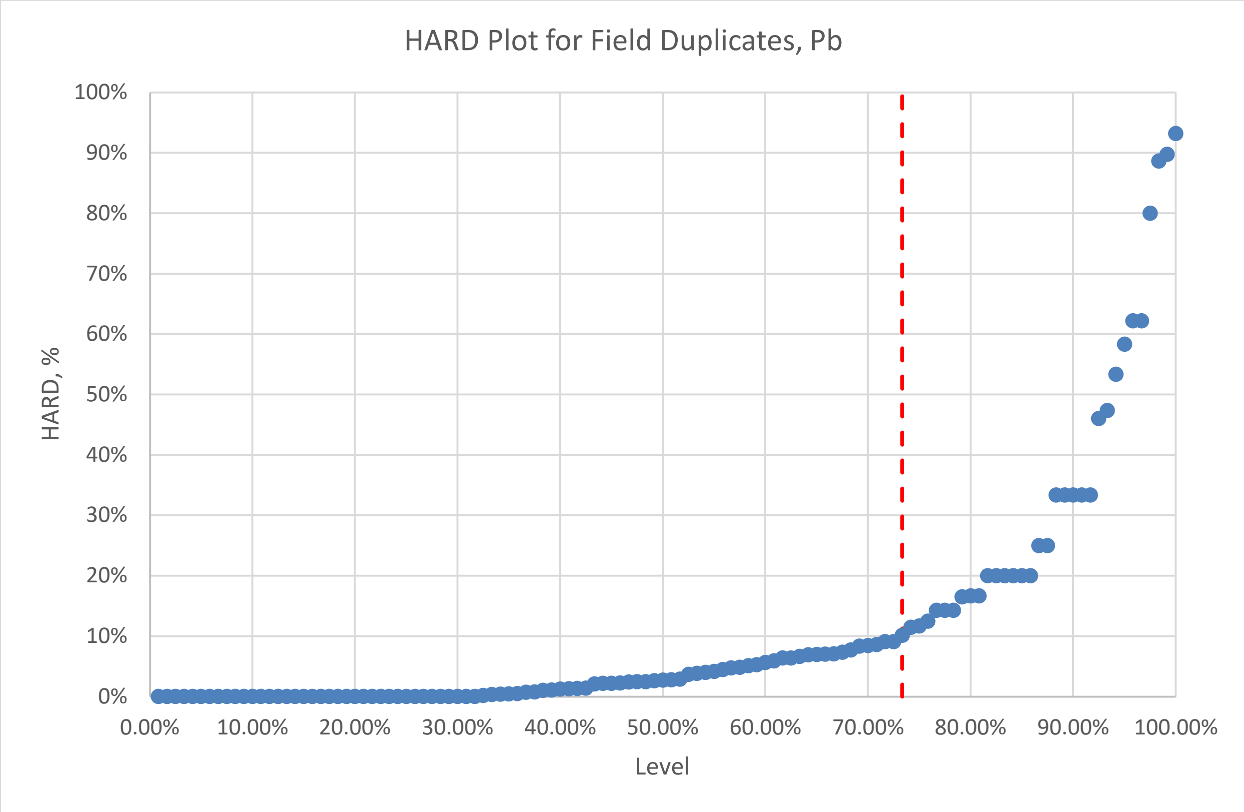


Figure 10.63: HARD Plot for Field Duplicates, Pb

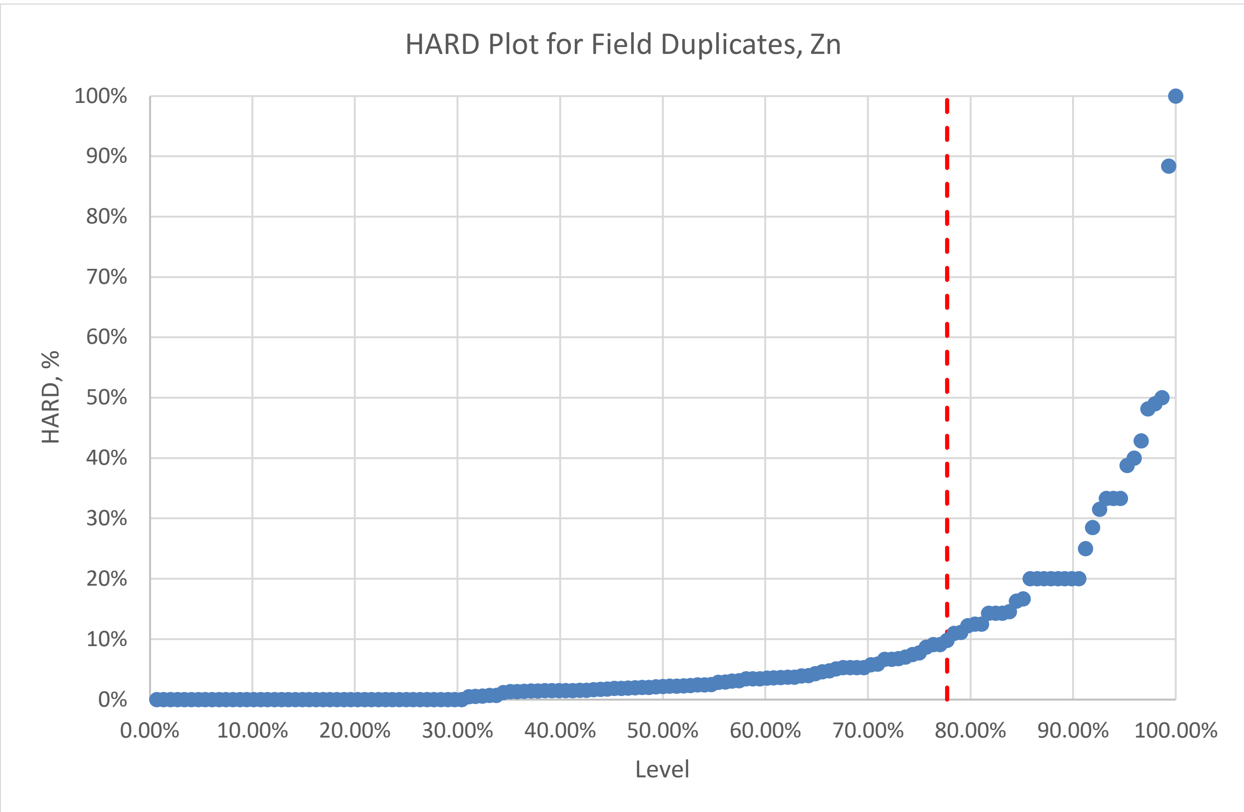


Figure 10.64: HARD Plot for Field Duplicates, Zn

Correlation plots for Pb and Zn for stream samples and duplicates are shown in Figure 10.65 and Figure 10.66.

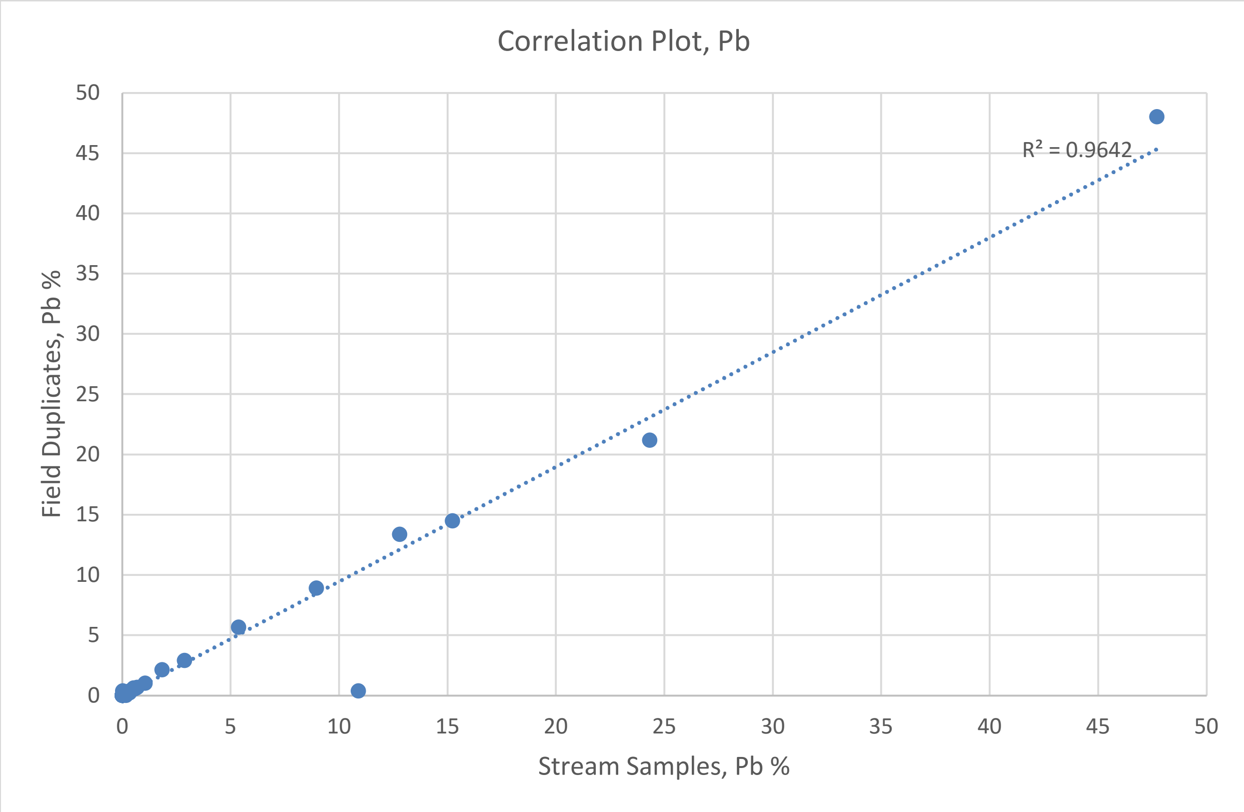


Figure 10.65: Correlation Plot for Field Duplicates, Pb

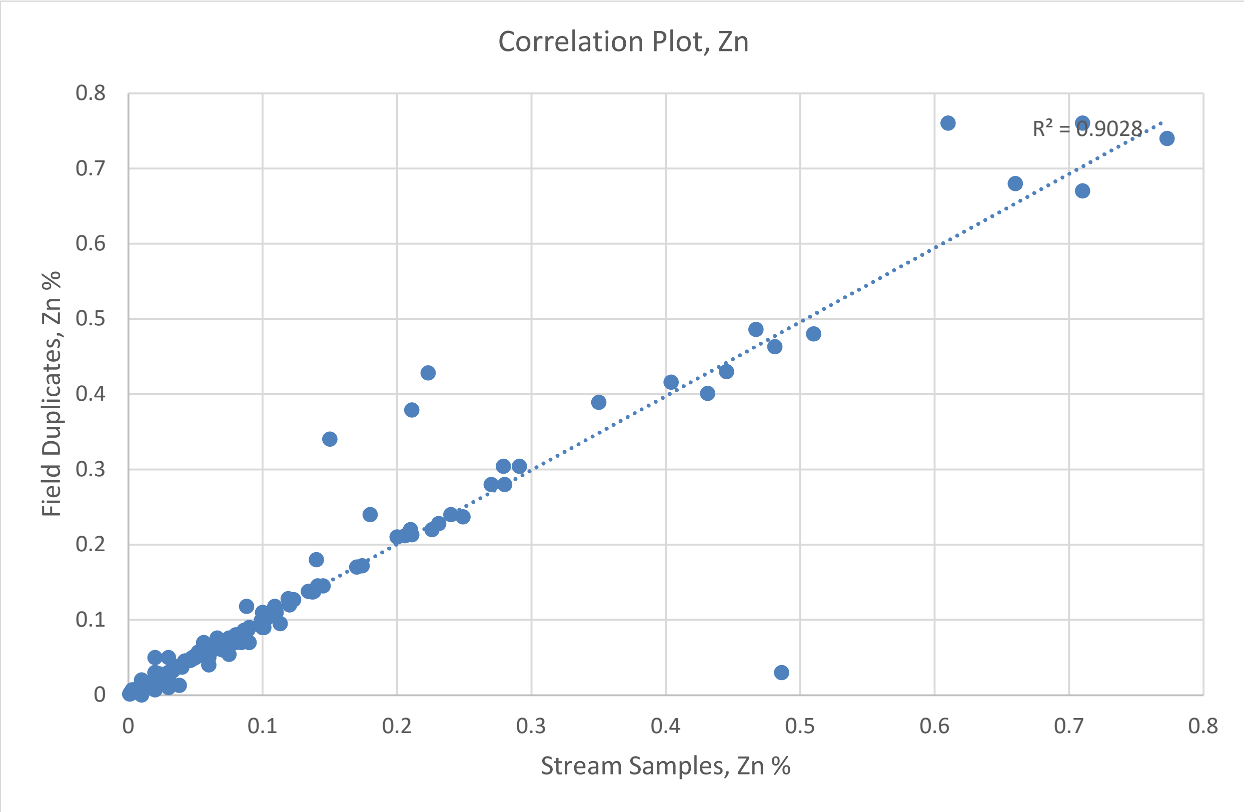


Figure 10.66: Correlation Plot for Field Duplicates, Zn

10.6.2 Summary of QA/QC Risks

The WAI review of quality control data has identified a number of risks within the sample data. These risks are summarised in Table 10.9. It should be noted that Table 10.9 does not provide a quantitative risk assessment but gives an indication as to where WAI considers the risk lie within the sampling data.

A six-score classification has been employed where:

- 1 - 2 ('low' risk): Little or no perceived risk, or low uncertainty;
- 3 - 4 ('moderate' risk): Risk present which could lead to small material error in the resource model;
- 5 - 6 ('high' risk): This feature could lead to material error in the resource model (high uncertainty).

Table 10.9: Risk Matrix Vertikalnoye QA/QC Review		
Sample Type	**Risk**	**Comment**
Blanks	3	Blanks assaying results for Ag show their possible contamination. Ag grade for more than 10% of blanks from ore sections was higher than 50 g/t – cut-off grade for mineralisation delineation. In general, samples with higher silver grades are preceded by samples with high (more than 100 g/t to first/several thousand g/t) grade of this metal. Zink and lead blanks assaying results are satisfactory.
CRMs	3	CRM assaying results for Ag are satisfactory, there are some insignificant deviations for Zn and Pb assaying results.
Field Duplicates	2	Precision based on HARD data is at an acceptable level, more than 70% of samples are below error limit of 10%.

Total risk related to the quality of sampling, sample preparation and assaying is considered to be ‘moderate’ - Risk present which could lead to small material error in the resource model. However, WAI would recommend that the QA/QC procedures to be improved by sampling and sample preparation of field duplicates as there is a risk of sample contamination.

11 DATA VERIFICATION (ITEM 12)

Commentary on this section is presented in Section 1 'Sampling Techniques and Data' in Appendices 1 and 2 of this report.

11.1 Procedures

WAI completed several checks on the raw data and data entry process to cover a minimum 5% of raw data and understands that recording of data and management of transfer of data from site has been supervised by qualified senior staff.

Logging data in the first instance was recorded by hand to form documentation for each hole that includes collar and down hole survey information and assay information once available. This information was subsequently transferred to an electronic database.

A review of collar locations in the field, review of core logging or review of data from primary assay sheets has not been made at time of writing this report. Significant intersections have not been verified by either independent or alternate company personnel.

No adjustments to assay data have been made.

11.2 Location, Spacing, Distribution and Orientation of Data

All data was supplied in the World Geodetic System 1984, Zone 36J Northern Hemisphere (UTM) and it is understood that. Collar positions for all holes were laid out by the on-site surveyor using a differential GPS and then checked again once drilling was completed. Downhole surveys were carried out for all of the diamond drillholes using Reflex Ez-Shot equipment over a nominal interval of 20m in general.

A topographic survey was conducted across the property in 2014. The survey was carried out using Topcon 5GR satellite receiver. The field data was processed using TOPCONTOOLS software package. This survey is used for the current Mineral Resource Estimate. The small differences between the GPS readings and the topographical survey data do not influence the interpreted mineralisation widths.

Data spacing is down to 40m x 40m in the central part of deposit with some area of infill drilling to 25m x 25m. On the flanks the data spacing is more generally between 80m x 80m. Trenching for grade control is developed every 10m on the each 5m bench. This spacing is sufficient to establish geological and mineralisation continuity appropriate for the reporting of Mineral Resources.

Mineral Resources are classified as Measured, Indicated and Inferred in accordance with the guidelines of the JORC Code (2012), and through geostatistical analysis considering the spatial distribution of sample data. Sample compositing was carried out as part of the mineral resource estimation process. The diamond drill and trench data spacing is deemed by the CP to be sufficient to imply/confirm geological and grade continuity, sufficient for the classification of Inferred resources only. The average length of the samples is 0.91m on Vertikalny and 0.85m for North Mangazeisky therefore the composite length of 1.0m was chosen for both datasets.

In general, drilling is carried out so that the intersections of holes with mineralised zones occurs at a high angle which results in limited sample bias. The general strike of mineralisation is to northwest at 310° with sub-vertical steeply dipping mineralisation zone hence drilling is generally inclined at –50-60° towards the strike of the zones. Intercepts are reported as apparent thicknesses except where otherwise stated.

11.3 Limitations

At the time of writing a site visit has not been carried out to verify standard operational procedures in grade control and exploration on site.

Independent verification of drill results has not been performed thus no twin drilling or direct field comparison of sample pairs has been carried out as part of WAI's terms of reference. This has not been felt necessary given adequacy of QA/QC analysis, repeatability of analyses using good industry practices over the course of the project and no reliance on Soviet-era data for evaluation. This situation may need to be reassessed for future exploration and evaluation on Mangazeisky and other deposits.

WAI has not had opportunity to analyse the paper trail and raw data supporting the grade, tonnage and ore processing characteristics of material from the five stockpiled areas on site, although this material is part of potential mineral inventory it is 'as mined' and not included in Mineral Resource Estimation.

11.4 Opinion on Data Adequacy

The quality control and assurance data reviewed by the CP indicates the assays are generally within expected limits. The CP is satisfied that data collection, security, spacing and orientation of sample collection is sufficient to support the Mineral Resource classification presented herein. For future exploration work a specific Zn-Pb-Zn CRM may be of benefit, such as OREAS 134a, to add to the CRM list to improve statistical analysis of the Pb/Zn relationship.

12 MINERAL PROCESSING AND METALLURGICAL TESTWORK (ITEM 13)

12.1 Procedures

The most recent testwork on the sulphide ores for the production of separate lead and zinc concentrates was reported by "NVP-ESTAGeo Centre" LLC in 2018 and the results have been used for pit optimisation in the current work, along with the NSR terms provided by SBR.

12.2 Historical Testwork

Historical testwork was completed by TSNIGRI in 2008 and GINTSVETMET in 2011. The main testwork programs for the Feasibility Study were conducted by SGS Vostok in 2014 and TOMS in 2015.

The SGS Vostok testwork program tested a composite sample from the Vertikalny Central zone representing higher-grade oxide ore to be mined in the early years of operation. The TOMS testwork program consisted of leach variability testwork for oxide, transition and primary ore samples, followed by leach optimisation and comminution testwork on a composite primary ore sample, again from the Vertikalny Central zone at greater drill hole depths.

12.2.1 Oxide Ore

Vertikalny ore is characterised as a polymetallic silver-lead-zinc partially oxidised ore, with acanthite as the most abundant silver mineral, but also metallic silver, silver chlorides, silver-rich tetrahedrite, silver-antimony-lead and silver-lead sulphosalts. Diagnostic leaching indicated that approximately 90% of the silver in the oxide sample is amenable to cyanidation at a grind size of 80% passing 75 microns. The ore is moderately hard with a Bond Work Index of 14.3kWh/t.

In summary, the Tetra Tech analysis of the testwork program results indicated that the Vertikalny oxide ore is amenable to standard agitated cyanide leaching, with **design silver recovery of 85%**, although this includes a gravity circuit recovering approximately 8% of the silver with cyanide leaching of the gravity tailings. Testwork clearly indicates that, without the gravity circuit, additional leach residence time with higher cyanide concentration and higher pH is required to maintain leach recovery. The leach residence time increases from approximately 72 hours to 96 hours with and without the gravity circuit respectively, with the pH increasing from 10.5 to 11.5 and the cyanide concentration from 2,000ppm to 5,000ppm respectively.

The TOMS whole ore leach variability results, using leach conditions of 2,000ppm cyanide, pH 11.5, 120 hours residence time and the grind size of 80% passing 75 microns indicated that the oxide sample recovery averaged 82.4%, while the transitional and primary sample recovery decreased significantly to an average of 44.4% and 28.2% respectively.

12.2.1.1 Direct Electrowinning

Due to the high silver head grades and the remote location of the deposit, Tetra Tech recommended the use of direct electrowinning for the cyanide leached solution, rather than by the conventional Merrill Crowe process. Testwork was conducted by Electrometals LLC in 2014 using a leach solution prepared by SGS Vostok that assayed 798ppm Ag. The results showed that the silver could be depleted to <5 ppm after 2 hours electrowinning. Copper was also depleted to low values, although depletion of the zinc was less efficient, decreasing from approximately 1,900ppm to 1,000ppm. The silver powder collected from the cathode was smelted to produce silver bullion assaying 99.9% Ag. Therefore, based on the completed direct electrowinning test work, Tetra Tech concluded that direct electrowinning technology could be effectively utilised, and this was incorporated into the process design with an assumed electrowinning efficiency of 99%.

12.2.2 Primary Ore

The primary ore composite tested by TOMS was collected from 27 samples over 11 separate drill holes and with an average silver head assay of 371g/t Ag. The primary ore is significantly harder with a Bond Work Index of 19.0kWh/t.

Initial whole ore leach tests using the same optimised conditions as for the oxide ore leach variability tests returned a low silver recovery of only 29.4%. Under optimised leach conditions obtained by using a finer grind of 80% passing 25 microns and increasing the cyanide concentration to 10,000ppm, then silver recovery of approximately 71% was obtained, with the leach kinetics being extremely slow. Tetra Tech then calculated a design silver recovery of 69.6%, assuming the same use of the direct electrowinning circuit.

Bulk flotation testwork recovered 93.6% of the silver to a concentrate assaying 2,333g/t Ag at a 15% mass pull to concentrate, but unfortunately intensive cyanidation of this concentrate recovered only 26.7% of the silver, even at a fine grind of 80% passing 25 microns and with a cyanide concentration of 30,000ppm.

Further evaluation of the flotation option was not considered by Tetra Tech due to the remoteness of the project and perceived potential difficulties in logistics, with the idea of keeping the operation as simple as possible. It is also stated in the feasibility study that only approximately 10% of the feasibility study ore reserves are primary ore, although this only includes Vertikalny. Therefore, Tetra Tech recommended use of the oxide plant design for sulphide ore processing, but with the necessary modifications to allow for the finer grind and longer leach residence time required at higher cyanide concentrations.

Subsequent to the Tetra Tech feasibility study, further work on the flotation option for primary ore was performed by "NVP-ESTAGeo Centre" LLC in 2018, particularly as the undeveloped Mangazeisky deposit is almost 100% primary ore.

This work focussed on producing separate lead and zinc concentrates with cyanide leaching of the lead circuit middlings. Locked cycle tests were conducted, and primary lead flotation was undertaken at pH 7-9 using A3418 collector and zinc sulphate to depress the sphalerite. A lead concentrate was produced, and tailings scavenged to produce a lead circuit middlings which was cyanide leached and the scavenger tailings which reported to the zinc circuit. Primary zinc flotation was conducted at approximately pH 12 using xanthate collector and copper sulphate for sphalerite activation to produce zinc concentrate. After scavenging the zinc rougher tailings a final tailing was produced and the scavenger concentrate recycled.

The results of this testwork are summarised in Table 12.1.

Table 12.1: Summary of Locked Cycle Flotation Testwork on Primary Ore

Products	Mass, %	Assays, %			Recovery, %		
		Ag, g /t	Pb	Zn	Ag	Pb	Zn
Flotation							
Pb Concentrate	4.54	10,215	17.1	4.4	66.0	65.9	4.6
Pb-Ag Middlings	6.84	2,357	3.6	5.6	23.0	21.0	8.8
Zn concentrate	8.50	400	0.4	42.3	4.8	3.1	82.2
Tailings	80.12	53.9	0.15	0.24	6.2	10.0	4.4
Initial Sample	**100.0**	**702.0**	**1.18**	**4.37**	**100.0**	**100.0**	**100.0**

Cyanide leach testwork on the lead middlings product indicated a silver recovery of 68.1% could be achieved. Allowing for direct electrowinning efficiency and solution losses, an overall design silver recovery of 85.4% was calculated for primary ore. This is considered reasonable for pit optimisation studies. The lead and zinc recoveries are **65.9%** and **82.2%** respectively, although the appropriate NSR terms must then be applied. SBR has used indicative metal recoveries in their forecast performance data and, while the silver and zinc recoveries are in line with the above testwork results, the lead recovery at approximately 80% is significantly higher than the 65.9% indicated and the latter has been used for the pit optimisation studies.

The chemical analysis of the concentrates is shown in Table 12.2.

Table 12.2: Analysis of Pb and Zn Concentrates

Element	Assay, %	
	Lead Concentrate	Zinc Concentrate
Ag, g/t	**10,215**	**400**
Pb	**17.06**	**0.43**
Zn	**4.38**	**42.27**
Fe	26.16	11.83
S	29.00	22.00
Cu	3.87	0.20
As	1.95	0.81
Cd	<0.02	0.18
Sb	1.01	0.06
In	<0.02	<0.02
Sn	0.19	0.11
SiO_2	6.53	9.22
NaO	<0.1	<0.1
MgO	0.31	0.55
Al_2O3	1.67	3.71
K_2O	0.87	1.40
CaO	0.26	0.62
TiO_2	0.11	0.19
P_2O5	0.03	0.06
MnO	0.97	1.22
Cl	0.06	0.04
Cr	<0.02	0.08

The lead concentrate at only 17% Pb is very low compared to typical lead concentrates grading 50% - 70% Pb. However, the silver content is very high at 10,215g/t Ag and so the concentrate is likely to be marketable to an Asian smelter. High levels of arsenic and antimony are indicated which could incur penalties. The copper and zinc in the lead concentrate are unlikely to be payable.

As advised by SBR, a Net Smelter Return (NSR) of **84%** for both the lead and silver has been used for the pit optimisation studies. In due course, a quotation should be sourced based on the concentrate analysis shown in Table 12.2. In addition, the concentrate should be assayed for cobalt, mercury and selenium which are also potential penalty elements.

The zinc concentrate assaying 42.2% Zn is likely to be marketable as a zinc concentrate to a western smelter, with a typical required minimum grade of approximately 45% Zn. High levels of arsenic and silica are indicated which could incur penalties.

Further discussion on concentrate quality and realisation of products is discussed in Section 18.1 of this report.

As advised by SBR, a Net Smelter Return (NSR) of **45%** for both the zinc and silver has been used for the pit optimisation studies.

12.3 Limitations

In due course, a quotation should be sourced based on the concentrate analysis shown in Table 12.2. In addition, the concentrate should be assayed for fluorine, mercury and selenium which are also potential penalty elements.

The figure of 45% for NSR recovery appears a little conservative but should be confirmed with an official quotation and full concentrate elemental analysis to determine the impact of any deleterious elements.

12.4 Opinion on Data Adequacy

It is WAI's opinion that the previous metallurgical testwork provided a scoping level of accuracy for the basis of developing the process flowsheet and 'reglament'.

13 MINERAL RESOURCE ESTIMATION (ITEM 14)

13.1 Mineral Resource Estimation - Vertikalny

13.1.1 General Methodology

The following sections describes the process of Mineral Resource estimation for the Vertikalny silver mine. The estimate has been prepared in accordance with the guidelines of the JORC Code (2012).

The Mineral Resource Estimate (MRE) was carried out using a 3D block modelling approach using Datamine Studio 3 software (Datamine). Exploration data were imported and verified before wireframe modelling. In addition, digital terrain model (DTM) surfaces, surveys of mined-out areas, surfaces of overlapping sediments and boundaries of oxide and primary mineralisation were imported and/or created. Sample data were selected using the geological and mineralisation wireframes and selected samples were assessed for outliers. The wireframe envelopes were used as the basis for a volumetric block model based on a parent cell size of 10m x 10m x 10m. Variogram models were constructed based on composite data and used for grade estimation by ordinary kriging and inverse distance weighting methods. The resultant estimated grades in the block model were validated against the input sample and composite data. Resource classification was undertaken in accordance with the guidelines of the JORC Code (2012) and incorporated an assessment of the geological continuity and complexity, data quality, spatial grade continuity and overall quality of the resource estimation. Mineral Resources were limited based on an expectation of eventual economic extraction by being constrained within an optimised open pit shell generated using Datamine's NPV Scheduler software and underground stopes generated using Datamine's Mineable Shape Optimiser in Studio 5D Planner and appropriate economic and technical parameters.

13.1.2 Software

The MRE has relied on several software packages for the various stages of the process. However, the main data preparation and validation, wireframe modelling, statistical and geostatistical analysis, block modelling, estimation and validation were performed in Datamine Studio 3 version 3.22.84.0 and Snowden Supervisor version 8.9.0.2.

13.1.3 Data Transformations

All data are stored using the same local co-ordinate system and the same unit convention based on the WGS84 system. Therefore, transformations of drillhole or other data were not required.

13.1.3.1 Sample Database

Sample data is contained in two databases. The first comprises the exploration database which includes all exploration drilling (drill core) from 2006 to 2015 and exploration trenching (also from 2006 to 2015). The second comprises the grade control trench sample database used for short-term mine planning using 10m spaced trenches (5m high benches).

The grade control database is from 2007 to 2018. The exploration and grade control databases were provided by the Client in Microsoft® Access and Excel format and consisted of the files shown in Table 13.1 and Table 13.2, respectively.

Table 13.1: Exploration Database Files

Collar File		Assay File		Survey File	
Column	**Explanation**	**Column***	**Explanation**	**Column**	**Explanation**
Project	Site	Project	Site	Project	Site
Hole	Working Number	Hole	Working Number	Hole	Working number
Length	Depth/length of working	From_m	Interval from	Depth	Measured depth
UTM_Grid	Coordinate system	To_m	Interval to	Dip	Dip angle
UTM_East	Collar easting	DHSample	Sample number	Measured_Azimuth	Working azimuth
UTM_North	Collar northing	Sample_Type	Sample type	**Lithology File**	
UTM_Elevation	Collar elevation	Pimary_Sample	Original sample number for duplicate sampling	Project	Site
Azimuth	Azimuth of drilling	Au_OL_ppm	Au, g/t	Hole	Working Number
Dip	Angle of drilling	Ag_OL_ppm	Agg/t	From_m	Interval from
Hole_Type	Type of working	Cu_OL_pct	Cu, %	To_m	Interval to
Drill_Rig	Drill rig model	Pb_OL_pct	Pb, %	Lith1	Code of rock
Timestamp	Completion date	Zn_OL_pct	Zn, %	Lith1_Oxidation	Degree of oxidation
* assays for 32 elements are not included in the estimate					

Table 13.2: Grade Control Database Files

Collar File		Assay File		Survey File	
Column	**Explanation**	**Column***	**Explanation**	**Column**	**Explanation**
Project	Site	Project	Site	Hole_id	Working number
Hole	Working Number	Trench	Working Number	From	Measured depth
Length	Depth/length of working	Sample	Sample number	Azimuth	Working azimuth
UTM_Grid	Coordinate system	From_m	Interval from	Dip	Dip angle
UTM_East	Collar easting	To_m	Interval to	**Lithology File**	
UTM_North	Collar northing	Length	Sample length	Project	Site
UTM_Elevation	Collar elevation	Mass_sample	Sample weight	Trench	Номер working number
Azimuth	Azimuth of drilling	Ag, g/t	Ag grade	From_m	Interval from
Dip	Angle of drilling	Cu, %	Cu grade	To_m	Interval to
End	Data closed	Pb, %	Pb grade	Litocod	Code of rock
		Zn, %	Zn grade	Sample_Type	Sample type
		Sample_Type	Sample type		
* assays for the key elements using AA and ICP					

13.1.3.2 Database Review

A review of the sample databases was undertaken by WAI. The database includes data for core drillholes and trenches which were carried out during exploration campaigns and grade control trenches. The drilling and trenching was carried out in 2006-2018. The number of assayed samples split by type of developments and periods are shown in Table 13.3.

Table 13.3: **Assays Performed by BH Type and Periods**

Year	Type	Number of Assays			Comments
		Ag	Pb	Zn	
2006-2009	Trench	1,851	1,818	1,818	
2007	Drillhole	3,271	3,271	3,271	
2008	Drillhole	4,500	4,454	4,453	
2009	Drillhole	2,650	1,968	1,968	
2011	Drillhole	704	704	704	
2012	Drillhole	120	120	120	
2013	Drillhole	525	525	525	
2014	Drillhole	436	436	436	
2014	Trench	144	144	144	
2015	Drillhole	1,001	1,001	1,001	
2017	Drillhole	352			Metallurgical Holes
2018	Drillhole	174	4	4	Grade Control
2018	Trench	4,058	1,015	1,015	Grade Control
Total		**19,786**	**15,460**	**15,459**	

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.

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 0.25g, 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 0.4g, followed by ICP finish (laboratory code ME-OG62).

Where values of silver exceeded the upper detection limit for ME-OG62 (1,500g/t), a 50g 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 50g sample (laboratory code Au-AA24).

No replacement was done for samples with absent assay data or with zero assay value. The detection limit data was replaced with half of detection limit value for such samples.

13.1.3.3 Database Import

The database was imported by WAI into Datamine© software and desurveyed using the HOLES3D process. Where minor validation errors were discovered in terms of overlapping intervals these were subsequently corrected by WAI. The location of the drillholes / trench samples contained in the database is shown in Figure 13.1 while the location of the open pit is shown in Figure 13.2.

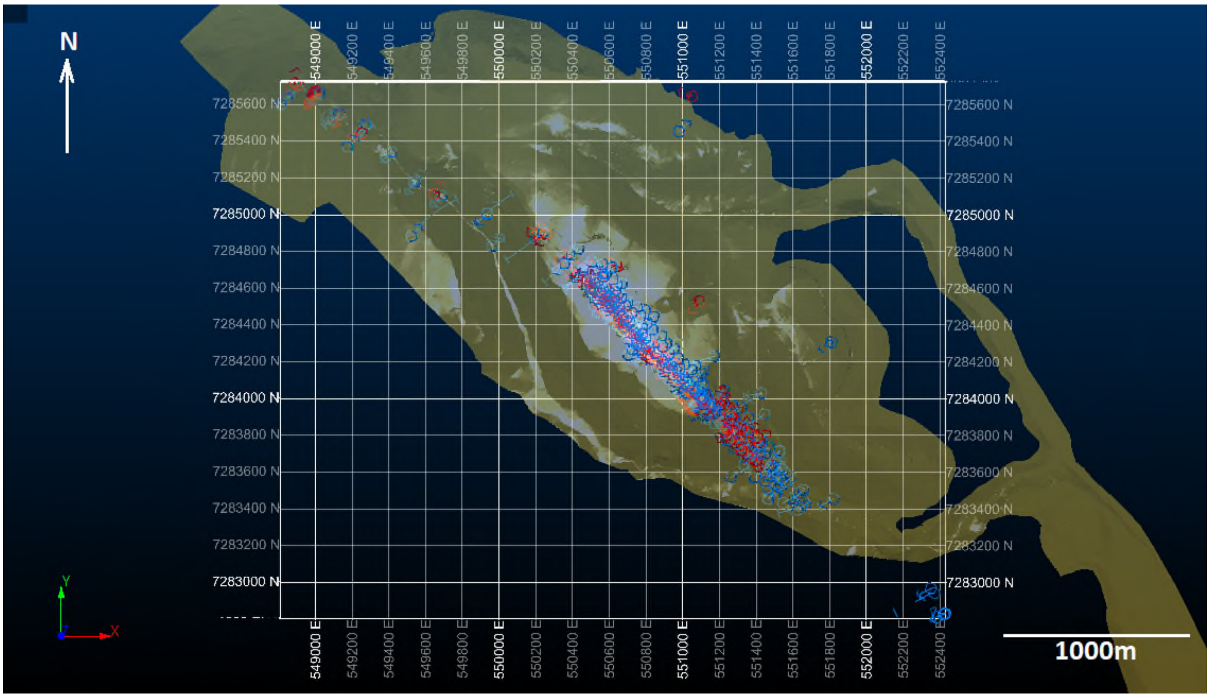


Figure 13.1: Location of Drillholes (blue) and Trenches (red) at Vertikalny

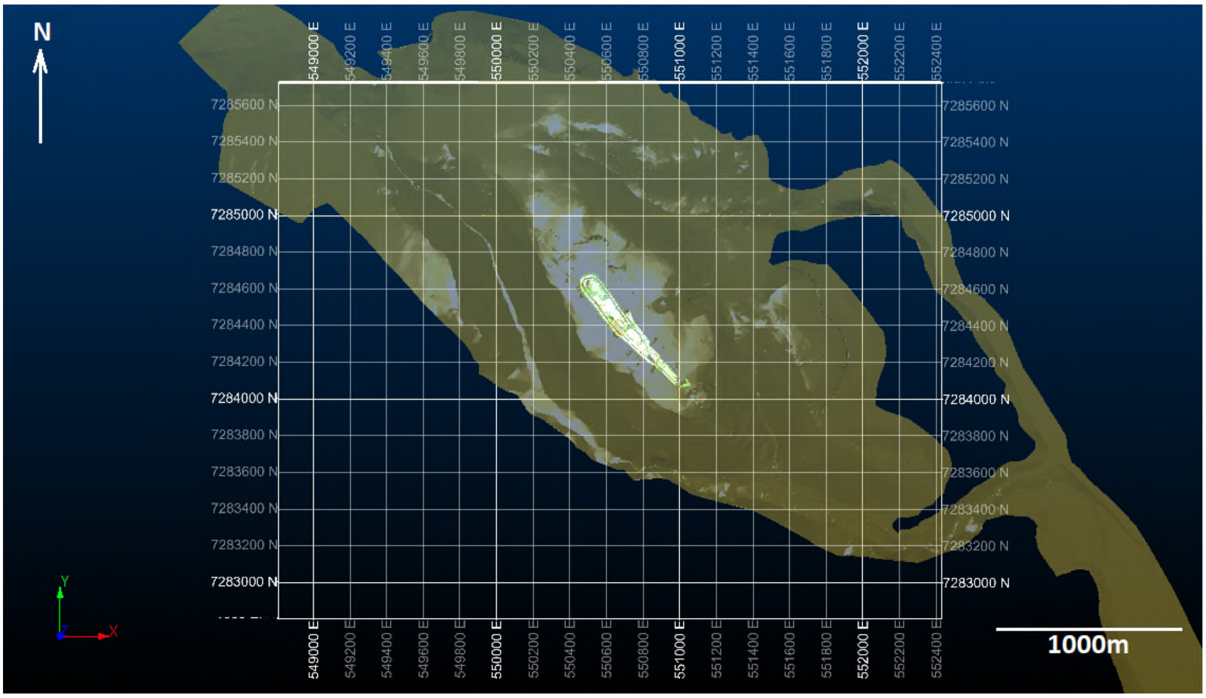


Figure 13.2: Location of Open Pit at Vertikalny Central Area as of May 2019

13.1.3.4 Data Verification

Data verification was undertaken by WAI following import of the database. A summary of the data verification procedures is detailed below:

- Comparison of historical drillhole logs with the drillhole database;
- Comparison of geological cross sections with the drillhole database;
- Check the presence of blank duplicate and Certified Reference Material in the database;
- Verification that collar coordinates coincide with topographical surfaces;
- Verification that downhole survey azimuth and inclination values display consistency;
- Evaluation of minimum and maximum grade values;
- Evaluation of minimum and maximum sample lengths;
- Assessing for inconsistencies in spelling or coding (typographic and case sensitive errors);
- Ensuring full data entry and that a specific data type (collar, survey, lithology and assay) is not missing and assessing for sample gaps or overlaps;
- Copper and gold were not considered by WAI in the MRE as the reported values are not considered to have economic potential;
- A statistical analysis of grades from the different sample types (drillholes, exploration trenches and grade control trenches) was undertaken by WAI and is summarised in the following section.

13.1.3.5 Final Database

A summary of the exploration database for Vertikalny is shown in Table 13.4. The database contains data for surface core drillholes, exploration trenches and grade control trenches.

Table 13.4: Final Database

Type of Working	Number	Total Length (m)
Drillholes – exploration	304	44,059.82
Trenches – exploration	76	2,380.88
Trenches – grade control	210	4,383.26
Total	**590**	**50,823.96**

13.1.4 Geological Interpretation and Wireframe Modelling

13.1.4.1 Introduction

CJSC Prognoz has provided a topographical pit survey DTM as on May of 2019. Topographical survey DTM in AutoCAD format prior start of mining was also provided to WAI.

The summarised results of metallurgical mapping to assess oxide/primary mineralisation boundary was also provided as a vertical long section through Vertikalny deposit.

Also, WAI has modelled a DTM of the overburden material using geological logging data from drillholes.

13.1.4.2 Geological Interpretation

The Vertikalny deposit consists of a hydrothermal vein type deposit containing silver, lead and zinc mineralisation in economic quantities with minor copper and gold. Mineralisation is strongly structurally controlled and is hosted within a main fault structure which strikes northwest and extends for 3.5km. Three main zones (Zones 1 to 3) are found within the overall structure. The zones dip sub-vertically and mineralisation has been defined to a depth of 800m. The thickness of the zones is generally less than 4m. Zone 1 comprises the central area (current open pit) whilst Zone 2 and Zone 3 comprise the south-eastern and north-western areas, respectively. Some additional minor mineralised structures (Zones 4 to 9) propagate from Zones 1 and Zone 2, however the tonnages contained in these propagating structures are less significant.

13.1.4.3 Mineralisation Wireframe modelling

The wireframes were constructed using a cut-off grade of 50g/t Ag. This cut-off is considered by WAI to reflect a “natural” cut-off grade for the deposit and corresponds to an inflexion in the population of Ag grades as shown in Figure 13.3.

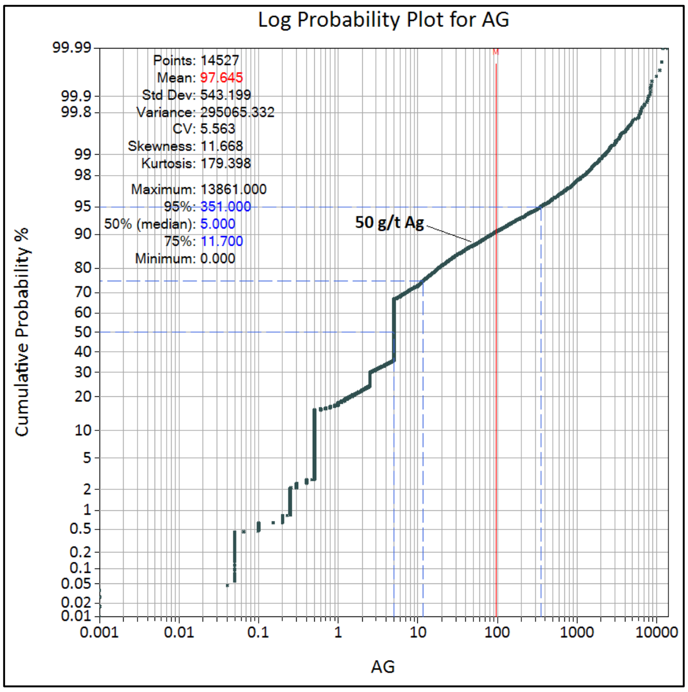


Figure 13.3: Log Probability Plot of Ag grades for Sample Data

Wireframes of the mineralisation contained within the nine structural zones were produced by WAI using the exploration database and grade control database to guide the interpretation.

A minimum sample thickness (interval) of 1m and a maximum waste interval of 3m was used by WAI during construction of the mineralised zones. In order to maintain mineralised continuity, and/or to avoid unnecessary splitting of the mineralised intervals, there was some flexibility permitted in the parameters during wireframe modelling.

The nine mineralised zones defined by WAI at Vertikalny (Figure) including three largest zones – Zone 1 (central area), Zone 2 – (south-east area) and Zone 3 (north-west area). The remained zones are being apophasis of the Zones 1 and 2 have a short strike length and traced in 2-3 up to 5 neighbouring exploration profiles. The general mineralisation strike is to north-west at 320-325° with sub-vertical dip. A plan view showing the location of the zones within the main fault structure is shown in Figure 13.4. An isometric view showing the zones of central area in more detail is shown in Figure 13.6.

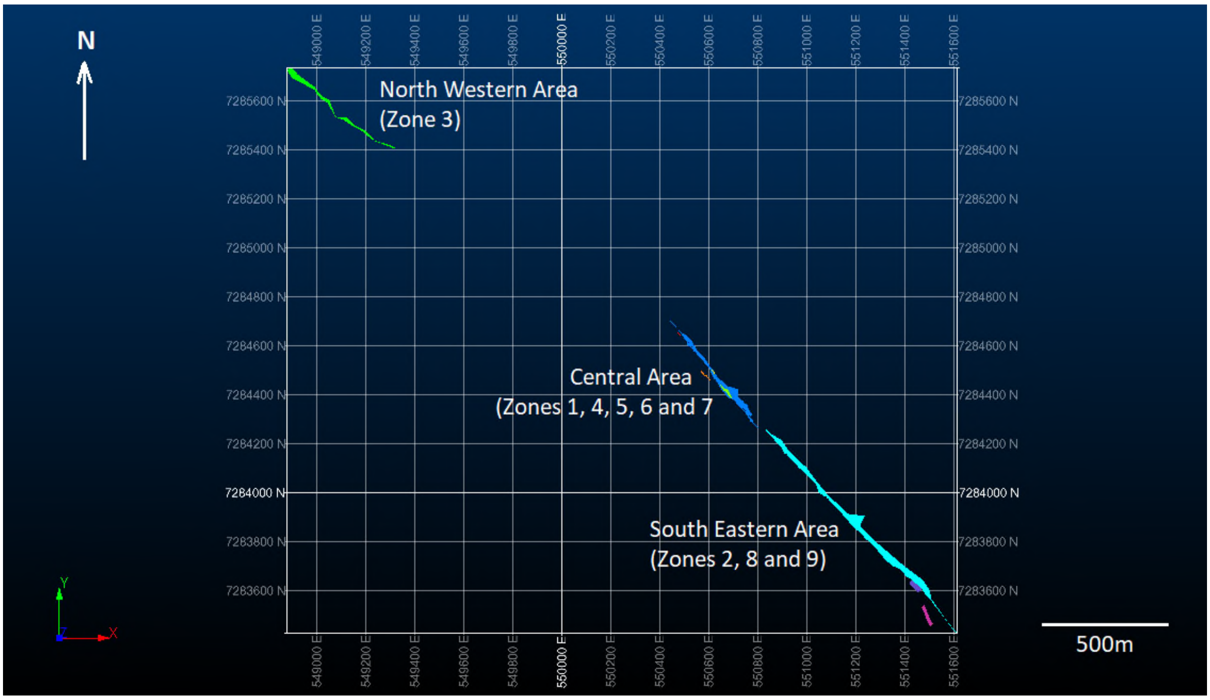


Figure 13.4: Plan View Showing Location of Mineralised Zones

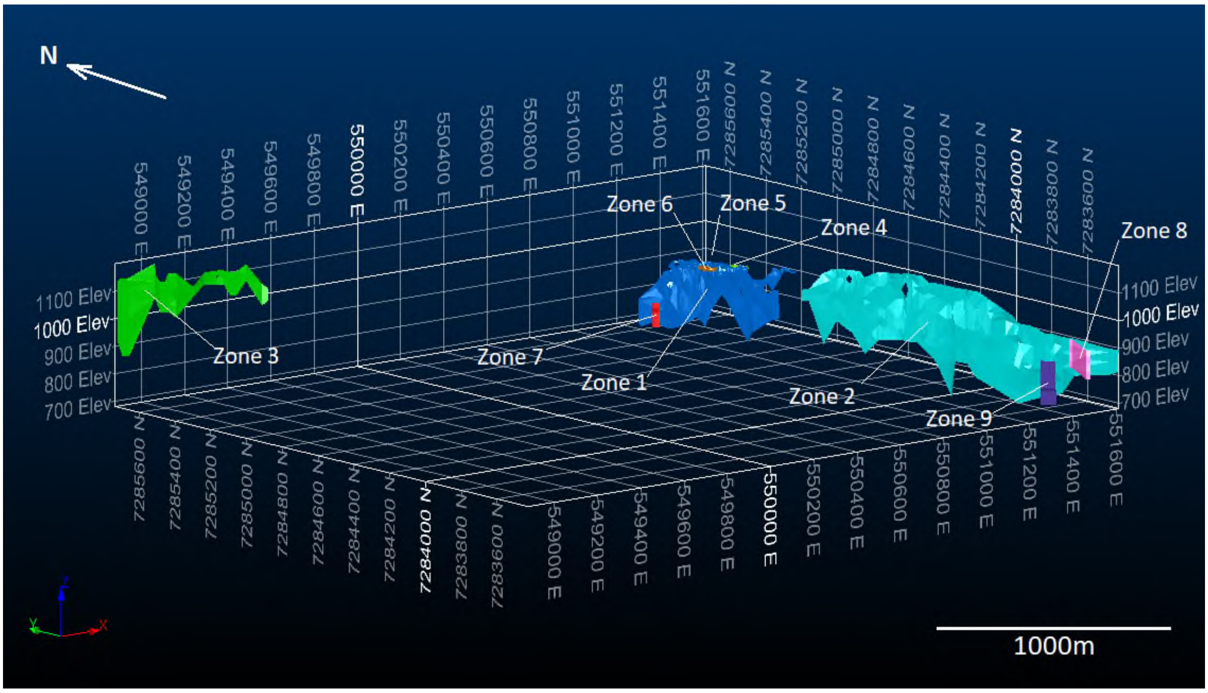


Figure 13.5: Isometric View of Mineralised Zones

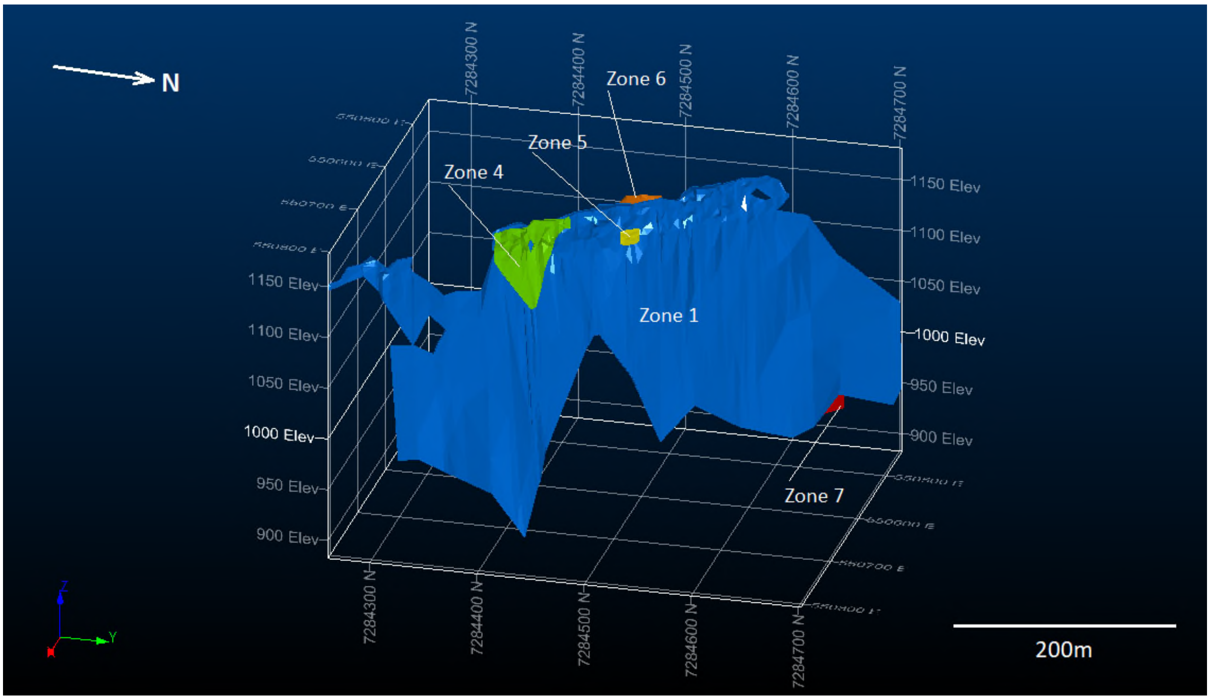


Figure 13.6: Isometric View of Central Area Only Showing Mineralised Zones

WAI considers the cut-off grade parameters used to be appropriate for the mineralisation at Vertikalny and are also appropriate for an open pit mining scenario. WAI considers that sufficient continuity of mineralisation is exhibited at this cut-off upon which to define the mineralised zones.

i) Oxidation

Oxide and primary mineralization is present at Vertikalny. A semi-oxide (mixed) type of mineralization was also distinguished, however, direct cyanide leaching of this mineralisation is characterized by generally low silver recoveries, similar to the primary mineralisation. As a result, all semi-oxidised mineralisation is therefore considered as primary.

The degree of oxidation can be determined visually during geological logging of mine workings. To confirm the identified types of mineralisation, additional phase analyses were carried out to assay for total sulphur and sulphur sulfide. The degree of oxidation was determined based on the sulphur sulphide to sulphur total proportion:

- < 50% sulphur sulphide – oxide ores; and
- ≥50% sulphur sulphide – primary ores (including semi-oxide).

In 2014-2015, the degree of oxidation was determined from proportion of iron oxide and iron total:

- 90% iron oxide to iron total – oxide ores;
- < 90% - semi-oxide and primary ores.

Additional studies on flotation concentration following a single processing flowsheet were carried out in 2017-2018 on samples taken based on visual assessment of the degree of oxidation.

Based on the oxidation data, geological-metallurgical mapping was undertaken by the Client to determine the boundaries of the oxidation zone. The results were represented as a vertical section at Vertikalny. The zone of oxidation is seen to have a complicated morphology. The bulk of oxide mineralisation is confined to near-surface areas, although the depth of the oxidation zone is occasionally over 100m below the surface. At the same time, primary ores locally outcrop. The greatest depth of the oxidation zone is confined to the center of the deposit.

A wireframe solid depicting the zones of oxidation was created by WAI and is shown in Figure 13.7.

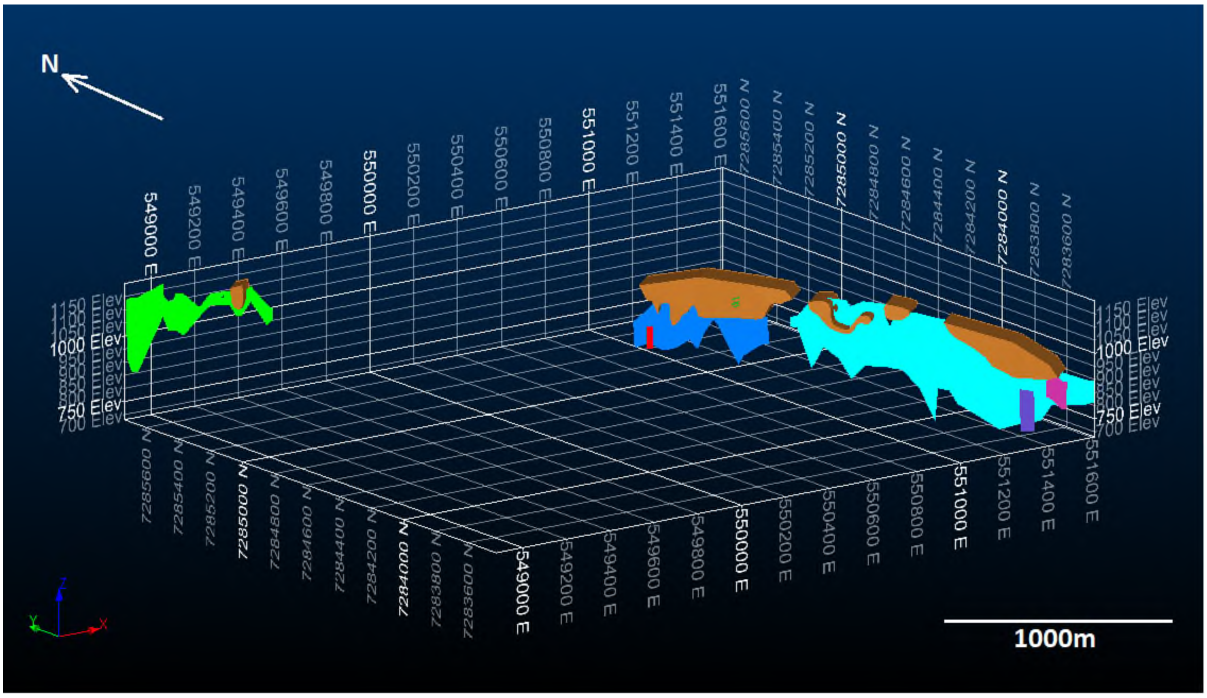


Figure 13.7: Modelled Zones of Oxidation at Vertikalny

A statistical analysis was undertaken by WAI to compare the oxide and sulphide grades to assess the need for separate domaining. Log probability plots for silver, lead and zinc were produced by WAI and are shown in Figure 13.8. A slightly higher-grade population for silver is potentially seen to be associated with the oxide mineralisation, while slightly higher zinc grades appear to be associated with the primary mineralisation. The lead grades appear consistent between the oxide and primary. Overall, the grade populations observed in the oxide and primary mineralisation are considered to be relatively similar, however due to the slight differences seen in the silver and zinc grades, WAI has elected to consider the oxide and sulphide mineralisation as separate domains.

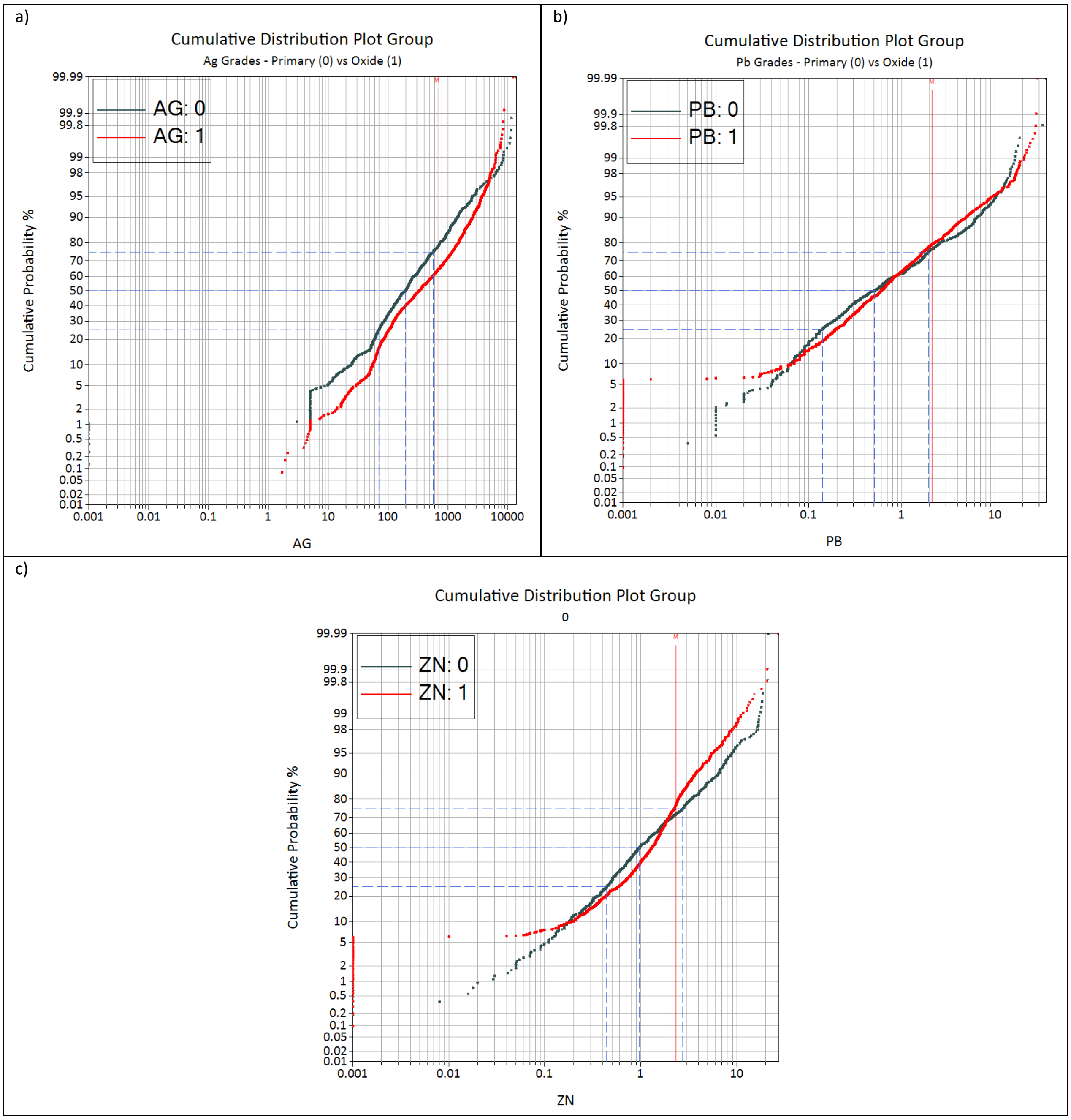


Figure 13.8: Log Probability Plots Comparing Grades for Oxide and Primary Mineralisation for a) Ag, b) Pb and c) Zn

ii) Lithology

As it was mentioned above, mineralisation of Vertikalny is associated with steeply dipping mineralized tectonic zones of north-west strike. The zones are composed of quartz-carbonate-sulphide material. The host rock is represented by interbedding of aleurolite, sandstone and argillite. The sub-surface area is covered by diluvial sediment with thickness of the overburden material of first meters.

A wireframe surface of the overlying sediments based on the drillhole logging data was constructed by WAI and incorporated in the resource model. No further domaining based on lithology was undertaken by WAI.

13.1.5 Drillhole Data Processing

Drillhole samples from the verified database were selected within the mineralised zone wireframes and were further sub-divided based on oxide/primary mineralisation types. To preserve the integrity of the assay sample lengths, the drillhole files containing only assay data were used (rather than assay and lithology combined). The final selected samples were coded by the principal domains and formed the basis of the Mineral Resource Estimate. A summary of the sample data contained in each domain is shown in Table 13.5.

Table 13.5: Sample Data Contained in Individual Wireframe Zones

Zone	Type	Workings*	Samples**	Total (m)	Ave Length (m)
1	Oxide	214	993	976.30	0.98
1	Primary	43	200	148.60	0.74
2	Oxide	42	130	105.90	0.81
2	Primary	112	499	439.39	0.88
3	Oxide	1	1	1.40	1.40
3	Primary	17	63	59.35	0.94
4	Oxide	21	87	71.50	0.82
4	Primary	5	17	16.00	0.94
5	Oxide	6	11	11.30	1.03
6	Oxide	18	32	30.80	0.96
7	Primary	1	4	4.30	1.08
8	Primary	2	9	5.05	0.56
9	Primary	4	10	7.10	0.71
Total for Oxide		**302**	**1,254**	**1,197.2**	**0.95**
Total for Primary		**184**	**802**	**679.79**	**0.85**
Total		**486**	**2,056**	**1,876.99**	**0.91**

* the total number of workings is 590, some workings do not access the mineralization; moreover, some workings intersect more than one mineralised zone
** not all samples contain recorded assay values

A statistical analysis of Ag, Pb and Zn grades by domain is shown in Table 13.6.

Table 13.6: Statistical Analysis of Selected Samples

Type	ZONE	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag (g/t)								
Oxide	1	972	1.7	12,247.70	1,021.54	2,104,823	1,451	1.42
	2	129	5	7,476.00	604.55	1,363,705	1,168	1.93
	3	1	224	224.00	224.00	-	-	-
	4	87	4.55	3,530.00	547.62	520,060	721	1.32
	5	11	55.24	530.00	237.24	16,345	128	0.54
	6	32	35	3,054.00	436.45	416,248	645	1.48
	7	-	-	-	-	-	-	-
	8	-	-	-	-	-	-	-
	9	-	-	-	-	-	-	-
Primary	1	191	5	13,861.00	1,256.66	6,752,610	2,599	2.07
	2	484	0	7,147.00	461.85	618,495	786	1.70
	3	61	5	2,768.00	452.75	347,265	589	1.30
	4	16	80	3,991.73	842.16	1,076,388	1,037	1.23
	5	-	-	-	-	-	-	-
	6	-	-	-	-	-	-	-
	7	4	3	1,590.00	746.75	394,521	628	0.84
	8	8	106.77	589.00	260.85	37,577	194	0.74
	9	10	87	769.50	209.99	37,181	193	0.92
Pb (%)								
Oxide	1	804	0	28.29	2.02	15.79	3.97	1.97
	2	116	0.045	22.00	1.63	9.43	3.07	1.88
	3	1	1.4	1.40	1.40	-	-	-
	4	74	0	27.70	1.42	11.99	3.46	2.43
	5	7	0	3.22	1.00	1.67	1.29	1.29
	6	28	0	18.90	3.71	34.42	5.87	1.58
	7	-	-	-	-	-	-	-
	8	-	-	-	-	-	-	-
	9	-	-	-	-	-	-	-
Primary	1	143	0.01	35.60	1.92	27.21	5.22	2.71
	2	321	0.01	15.98	1.86	8.08	2.84	1.53
	3	44	0.005	16.50	4.81	23.01	4.80	1.00
	4	16	0	7.67	1.28	4.27	2.07	1.61
	5	-	-	-	-	-	-	-
	6	-	-	-	-	-	-	-
	7	4	0.01	0.19	0.13	0.00	0.07	0.55
	8	8	0.359	14.85	4.55	37.36	6.11	1.34
	9	5	0.07	4.48	1.17	2.80	1.67	1.43
Zn (%)								
Oxide	1	804	0	13.61	1.82	3.42	1.85	1.01
	2	116	0.06	27.22	1.67	17.73	4.21	2.53
	3	1	0.37	0.37	0.37	-	-	-
	4	74	0	14.59	2.63	9.57	3.09	1.17
	5	7	0	2.48	1.18	0.53	0.72	0.61
	6	28	0	3.89	1.67	1.30	1.14	0.68
	7	-	-	-	-	-	-	-
	8	-	-	-	-	-	-	-
	9	-	-	-	-	-	-	-
Primary	1	143	0.016	20.90	2.08	10.19	3.19	1.53
	2	321	0.03	21.22	2.39	10.12	3.18	1.33
	3	44	0.029	18.10	2.78	26.35	5.13	1.85
	4	16	0	17.70	3.40	29.79	5.46	1.61
	5	-	-	-	-	-	-	-
	6	-	-	-	-	-	-	-
	7	4	0.008	0.47	0.28	0.03	0.17	0.61
	8	8	0.38	4.86	2.85	2.77	1.66	0.58
	9	5	0.19	3.14	0.96	1.24	1.11	1.16

13.1.5.1 Compositing

A histogram of the lengths of the selected samples which contain Ag values is shown in Figure 13.9. The majority of sample lengths are 1m or less with relatively few samples greater than 1m. A 1m composite interval was therefore selected by WAI. Compositing was carried out within each domain and composites were coded by these domains. A minimum composite interval of 0.20m was used by WAI to prevent excessively small composites being generated. Composites less than this length were rejected. Only relatively few samples are greater than 1m, therefore WAI considers that decompositing of these samples to 1m length will not have a significant impact on the MRE.

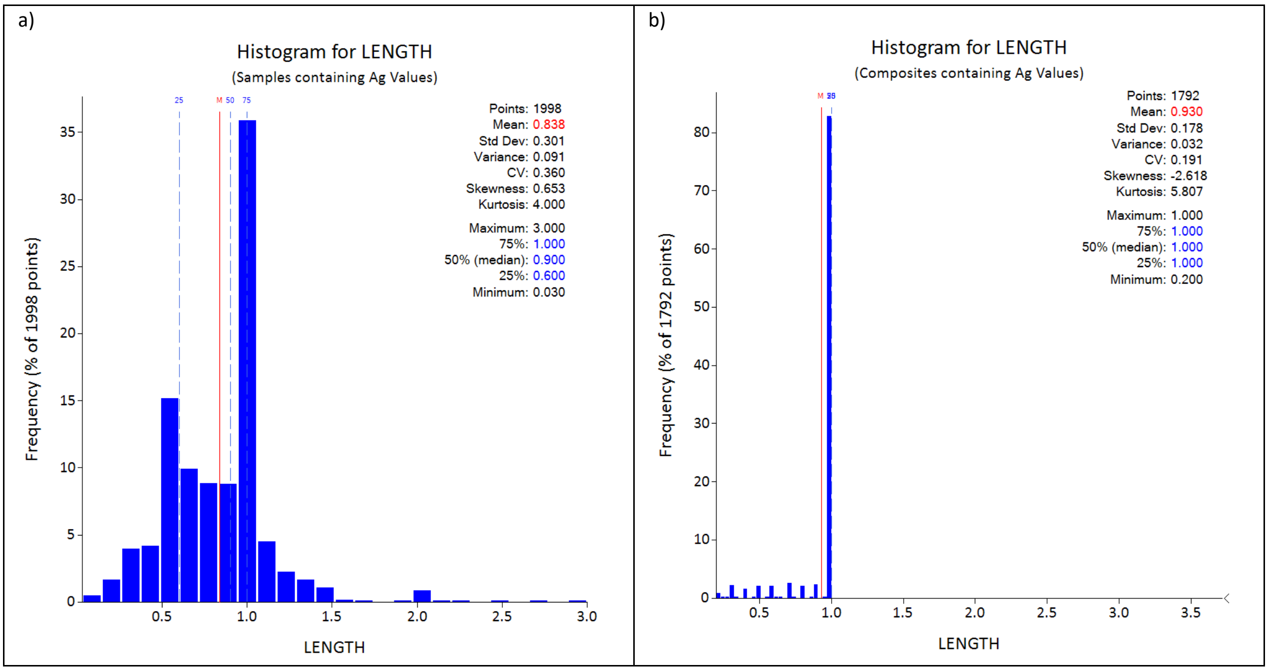


Figure 13.9: a) Histogram of Lengths of Selected Samples, b) Histogram of Composite Lengths

13.1.5.2 Statistical Analysis by Sample Type

Statistical analysis of the grades for drillholes exploration trenches and grade control trenches for Vertikalny is in Table 13.7. The average grade of silver and lead from grade control trenches is higher than the grade from exploration workings. The average zinc grade is in general the same in grade control and exploration developments.

Table 13.7: Statistical Analysis of Composites for Various Types of Workings

Type of Working	Metal	Qty of composites	Grade		
			Min	Max	Average
Exploration drillholes	Ag (g/t)	866	3	11,832.50	664.11
Exploration trenches	Ag (g/t)	127	23.05	3,800.11	473.71
Grade control trenches	Ag (g/t)	799	4.2	8,801.00	950.49
Exploration drillholes	Pb (%)	620	0.01	26.30	1.68
Exploration trenches	Pb (%)	111	0	19.83	1.89
Grade control trenches	Pb (%)	689	0	28.29	2.12
Exploration drillholes	Zn (%)	620	0.01	20.75	2.25
Exploration trenches	Zn (%)	111	0	21.18	0.96
Grade control trenches	Zn (%)	689	0	17.70	1.79

WAI has carried out statistical analysis of the grades from drillholes, exploration trenches and grade control trenches located within the area of the open pit is shown in Table 13.8. The average silver grades from the exploration drillholes and grade control trenches are almost identical, while lower silver grades report from the exploration trenches, however these are based on the fewest number of samples. The average lead grade is generally higher in the grade control trenches while the average zinc grades are slightly higher in the exploration drillholes. Overall, no significant bias is evident between the different sample types.

Table 13.8: Statistical Analysis of Composites for Various Types of Workings within the Open Pit

Type of Working	Metal	Qty of composites	Grade		
			Min	Max	Ave
Exploration drillholes	Ag (g/t)	72	5	4,920.72	925.78
Exploration trenches	Ag (g/t)	35	25.85	2,574.04	561.72
Grade control trenches	Ag (g/t)	721	4.2	8,801.00	929.36
Exploration drillholes	Pb (%)	45	0.055	18.90	1.40
Exploration trenches	Pb (%)	25	0	19.83	1.64
Grade control trenches	Pb (%)	611	0	24.89	2.01
Exploration drillholes	Zn (%)	45	0.35	8.03	2.41
Exploration trenches	Zn (%)	25	0	1.75	0.62
Grade control trenches	Zn (%)	611	0	17.70	1.82

In general, it can be expected that silver grade will decrease with the depth while lead and zinc grade will be on the same level.

13.1.5.3 Top Cutting

Top cuts were applied to the composites to ensure that anomalously high-grade samples did not bias the grade estimation of the domain. Where outliers were identified, the grade of these composites was reduced to the top cut level. A summary of the top cut levels is shown in Table 13.9. The number of samples which were capped is shown in brackets.

Table 13.9: Top Cut Levels

Type	ZONE	Ag (g/t)	Pb (%)	Zn (%)
Oxide	1	None	20 [6]	None
	2	4,000 [2]	15 [2]	15 [3]
	3	None	None	None
	4	None	8 [1]	None
	5	None	None	None
	6	2,000 [1]	17 [2]	None
	7	-	-	-
	8	-	-	-
	9	-	-	-
Primary	1	10,000 [4]	20 [2]	None
	2	4,000 [1]	15 [1]	15 [2]
	3	2,000 [1]	14 [2]	5 [4]
	4	None	None	15 [2]
	5	-	-	-
	6	-	-	-
	7	None	None	None
	8	None	None	None
	9	None	None	None

NB - Number of capped samples shown in brackets

The need for top cutting and the selection of the top cut values was assessed by WAI using quantile analysis of grades and probability plots and are discussed in the following sections.

i) Quantile Analysis

Quantile analysis is a recognized rule of thumb to analyze the outliers and determine the appropriate top cutting value. The quantile analysis provides for the samples to be ordered by grades and then the grade values are determined for the first 10% samples, then 20%, 30% etc. The topmost quantile is also checked in percentiles, since it is often required to be analyzed in more detail. Checks on increased quantity and proportion of metal in each quantile and percentile provides an indication if outlier values are present. In general, if the upper quantile (90-100%) contains more than 25-30% of the accumulated metal, then top cutting may be required. If the top 2 or 3 percentiles contain more than 10% of the total accumulated metal, it is recommended that either top cutting be carried out or these values should be isolated as separate high-grade zones. The quantile analysis results for Zone 1 show that 45.47% of Ag metal is contained in the top quantile whilst the accumulated metal in the top percentile exceeds 9%. WAI therefore considers that there is a need to top cut these outlier composites. The results of all quantile analysis are contained in Appendix 1.

ii) Probability Plots

Probability plots were used by WAI to further assess the presence of outlier grades and to select appropriate top cut values. Example log probability plots showing the top cut levels selected for Ag in Zone 1, Zone 2, Zone 3 and Zone 6 are shown in Figure 13.10, Figure 13.11 and Figure 13.12, respectively.

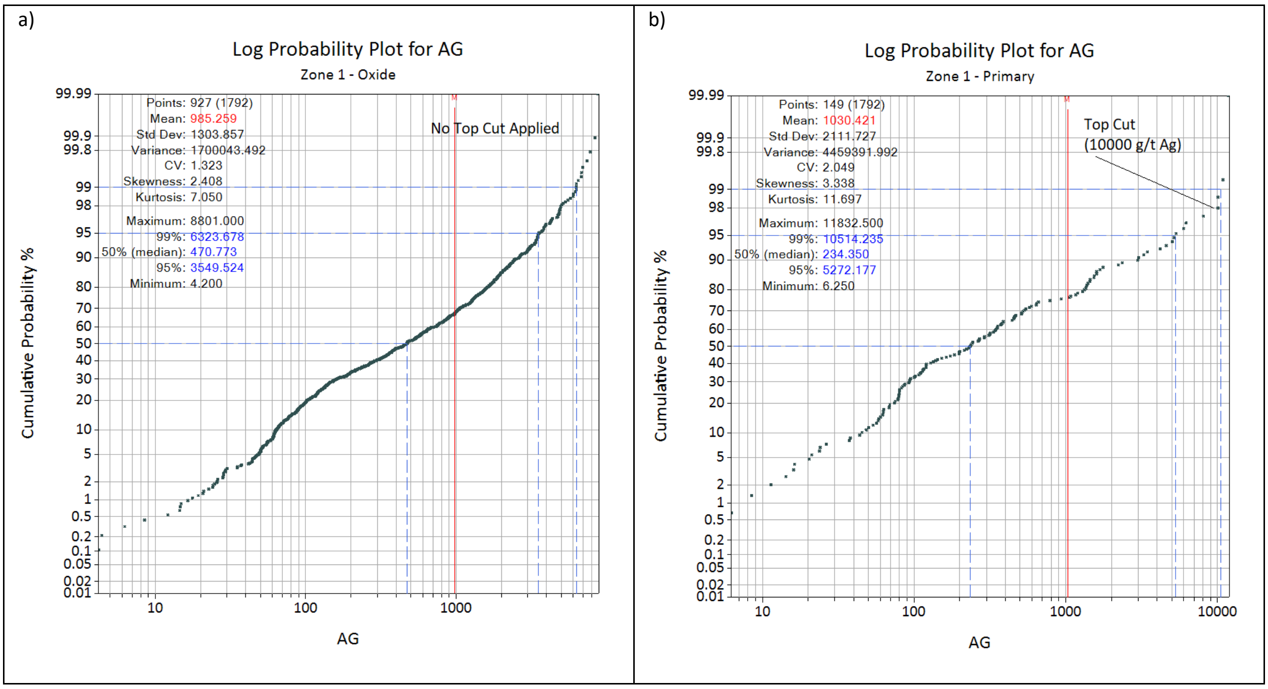


Figure 13.10: Log Probability Plots Showing Top Cut Levels for Ag for Zone 1 - a) Oxide, b) Primary

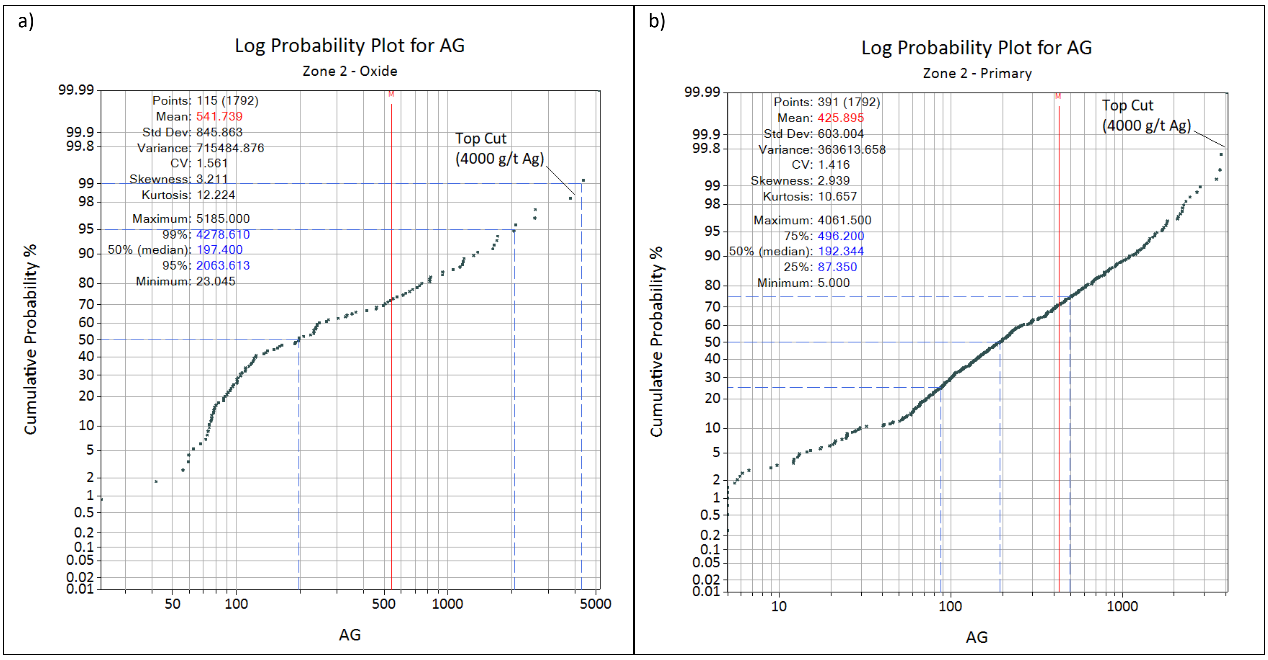


Figure 13.11: Log Probability Plots Showing Top Cut Levels for Ag for Zone 2 - a) Oxide, b) Primary

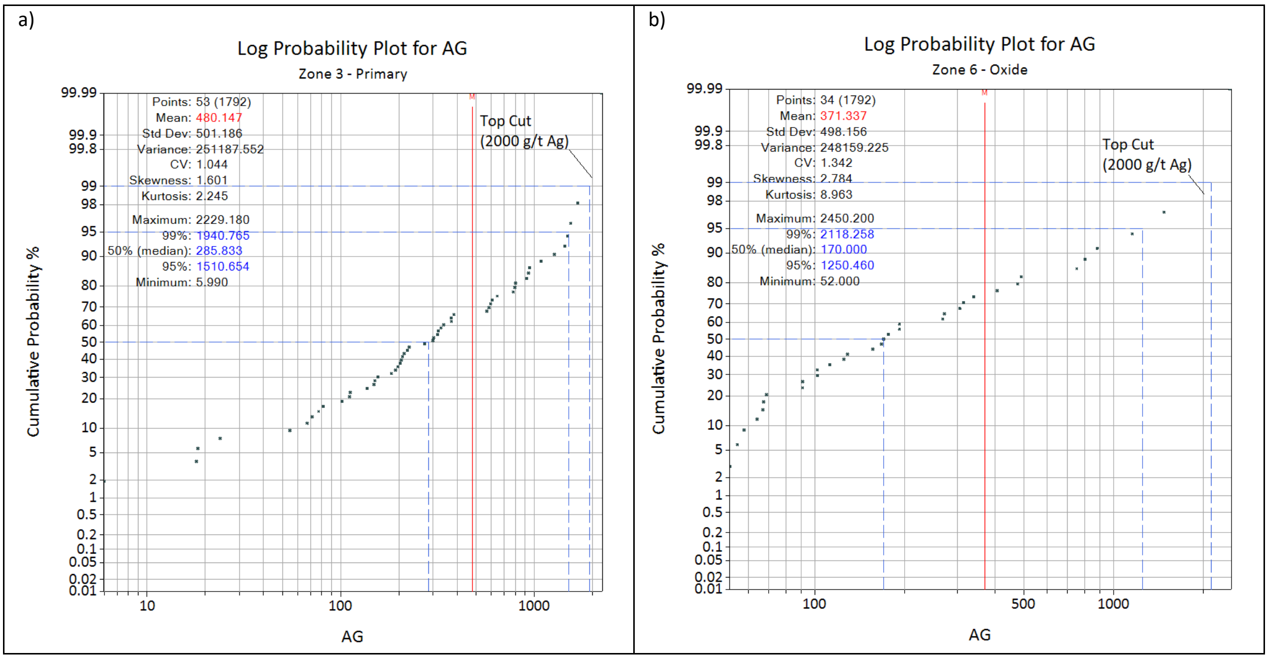


Figure 13.12: Log Probability Plots Showing Top Cut Levels for Ag for: a) Zone 3 - Primary, b) Zone 6 - Oxide

13.1.5.4 Final Composites

Statistical analysis by domain of the final composites (after top cutting) is shown in Table 13.10. Overall, no significant effect on the mean grade is observed as a result of compositing or top cutting.

Table 13.10: Statistical Analysis of Composites

Type	ZONE	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
				Ag (g/t)				
Oxide	1	927	4.2	8,801.00	985.26	1,698,210	1,303	1.32
	2	115	23.045	4,000.00	528.28	602,620	776	1.47
	3	2	224	224.00	224.00	-	-	-
	4	75	4.55	2,727.12	497.26	283,525	532	1.07
	5	13	80.74	530.00	257.10	18,908	138	0.53
	6	34	52	2,000.00	358.10	191,593	438	1.22
	7	-	-	-	-	-	-	-
	8	-	-	-	-	-	-	-
	9	-	-	-	-	-	-	-
Primary	1	150	6.25	10,000.00	1,004.53	4,029,859	2,007	2.00
	2	405	0	4,000.00	411.02	355,101	596	1.45
	3	53	5.99	2,000.00	475.82	232,294	482	1.01
	4	15	80	2,839.94	896.32	801,608	895	1.00
	5	-	-	-	-	-	-	-
	6	-	-	-	-	-	-	-
	7	5	3	1,590.00	646.16	356,090	597	0.92
	8	5	106.77	589.00	276.59	32,891	181	0.66
	9	8	87	769.50	226.74	45,073	212	0.94
				Pb (%)				
Oxide	1	754	0	20.00	1.89	11.70	3.42	1.81
	2	106	0.06	15.00	1.66	7.47	2.73	1.65
	3	2	1.4	1.40	1.40	-	-	-
	4	64	0	8.00	1.15	2.42	1.56	1.36
	5	9	0.01	3.22	1.19	1.82	1.35	1.13
	6	32	0	17.00	3.13	28.34	5.32	1.70
	7	-	-	-	-	-	-	-
	8	-	-	-	-	-	-	-
	9	-	-	-	-	-	-	-
Primary	1	111	0.01	20.00	1.47	13.11	3.62	2.46
	2	271	0.01	15.00	1.76	5.85	2.42	1.37
	3	42	0.005	14.00	4.42	18.17	4.26	0.96
	4	15	0	6.43	1.22	2.87	1.69	1.39
	5	-	-	-	-	-	-	-
	6	-	-	-	-	-	-	-
	7	5	0.01	0.19	0.13	0.00	0.06	0.48
	8	5	0.359	14.85	5.32	28.11	5.30	1.00
	9	5	0.07	4.48	1.17	2.80	1.67	1.43
				Zn (%)				
Oxide	1	754	0	13.26	1.77	2.71	1.65	0.93
	2	106	0.0616	15.00	1.39	8.51	2.92	2.10
	3	2	0.37	0.37	0.37	-	-	-
	4	64	0	12.78	2.63	8.51	2.92	1.11
	5	9	0.416	2.48	1.28	0.36	0.60	0.47
	6	32	0	3.89	1.63	1.17	1.08	0.66
	7	-	-	-	-	-	-	-
	8	-	-	-	-	-	-	-
	9	-	-	-	-	-	-	-
Primary	1	111	0.017	12.14	1.90	6.61	2.57	1.35
	2	271	0.034	15.00	2.22	6.97	2.64	1.19
	3	42	0.029	5.00	1.31	1.78	1.34	1.02
	4	15	0	15.00	3.88	27.06	5.20	1.34
	5	-	-	-	-	-	-	-
	6	-	-	-	-	-	-	-
	7	5	0.008	0.47	0.28	0.02	0.15	0.55
	8	5	0.38	4.53	2.93	2.76	1.66	0.57
	9	5	0.19	3.14	0.96	1.24	1.11	1.16

13.1.6 Variography

The top-cut composites were used for modelling of experimental semi-variograms. To provide sufficient sample pairs, WAI elected to combine the oxide and primary mineralisation during the variogram analysis. Robust variogram models were produced for Ag at Zone 1 and Zone 2. Robust variogram models were also produced for Pb and Zn at Zone 2. Due to a low number of composites, and/or their irregular spacing, it was not possible to model robust variograms for the remainder of the zones and metals. Examples of the along strike and down-dip modelled variograms for Ag at Zone 2 is shown in Figure 13.13 and Figure 13.14. The parameters of all modelled variograms are presented in Table 13.11.

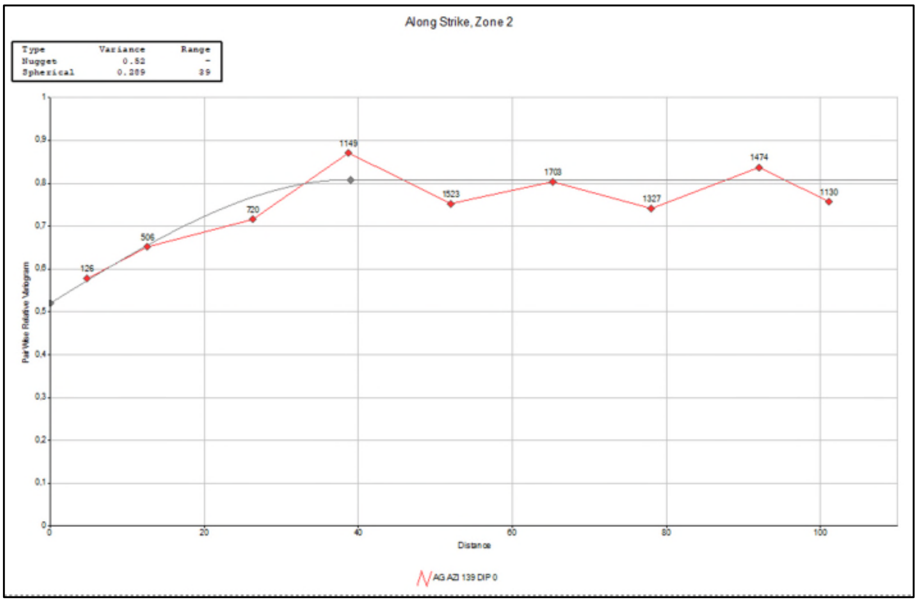


Figure 13.13: Ag Modelled Variogram, Zone 2, Along Strike

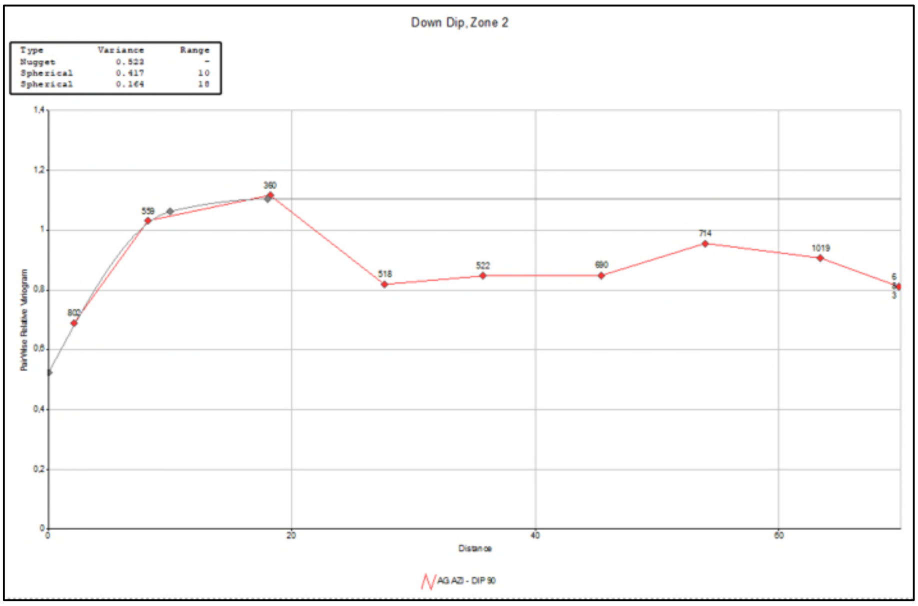


Figure 13.14: Ag Modelled Variogram, Zone 2, Down-Dip

Table 13.11: Parameters of Modelled Variograms

	Along Strike				Down-Dip				Across Strike			
Parameter	**Ag**	**Ag**	**Pb**	**Zn**	**Ag**	**Ag**	**Pb**	**Zn**	**Ag**	**Ag**	**Pb**	**Zn**
Zone	1	2	2	2	2	1	2	2	1	2	2	2
File	z1wcomp	z2tcomp	z2tcomp	z2tcomp	z2tcomp	z1tcomp	z2tcomp	z2tcomp	z1tcomp	z2tcomp	z2tcomp	z2tcomp
Lag	14	13	18	16	9	8	20	20	2	2	3	3
Nlag	8	8	8	8	8	10	8	8	8	8	6	6
HorAng	20	50	30	50	50	20	60	50	30	30	60	50
VerAng	20	50	30	50	50	20	60	50	30	30	60	50
CylRad	50	80	20	20	80	20	90	40	50	50	90	40
Ang1	139	139	139	139	49	49	49	49	49	49	49	49
Ax1	3	3	3	3	3	3	3	3	3	3	3	3
Ang2	-	-	-	-	90	90	90	90	-	-	-	-
Ax2	-	-	-	-	1	1	1	1	-	-	-	-
VarType	RV	RV	RV	RV	RV	RV	RV	RV	RV	RV	RV	RV
MoRefNo	2	3	8	11	4	5	9	12	6	7	10	13
Nugget	0.03	0.52	0.36	0.191	0.52	0.509	0.485	0.035	0.325	0.256	0.267	0.133
R1	19.9	38.7	91.2	49.5	9.6	10.2	43.3	71.2	2.5	2.2	3.3	6.2
C1	0.362	0.289	0.48	0.324	0.42	0.231	0.165	0.354	0.228	0.359	0.325	0.411
S1	0.391	0.809	0.84	0.515	0.94	0.74	0.651	0.39	0.553	0.615	0.592	0.544
R2	55.8	-	-	-	18.3	31.7	65.6	99.7	4.2	6	5.9	-
C2	0.044	-	-	-	0.16	0.114	0.11	0.494	0.355	0.369	0.317	-
S2	0.436	-	-	-	1.1	0.854	0.761	0.884	0.907	0.984	0.909	-

The large range for silver from Zone 1 is associated with strike direction. For Zone 2 the ranges along strike and down dip are similar and have 38.7 and 31.7m. The range for lead is 91.2m along strike and 65.6m down dip. For zinc down dip range is 99.7m whereas along strike is 49.5m. The across strike ranges for all metals are similar with the length being around the first meters. The nugget value is relatively high with covariance from 0.2 to 0.5.

13.1.7 Block Modelling

The block model was constructed using Datamine with a parent cell size of 10m x 10m x 10m (along strike, across strike and vertical), sub-celling was allowed down to 1.0m x 1.0m x 2.0m. The block model was created within the individual zone wireframes. The block model also reflects the DTM surface before mining and depleted volume as of May 2019. In addition, the model comprises oxide and primary ores, also outlines the blocks corresponding to unconsolidated sediments overlying the bedrock. No rotation has been applied to the model. A summary of the parameters used in the model prototype is shown in Table 13.12.

Table 13.12: Block Model Prototype

Parameters		Direction	Size
Model Origin		X	548,685
		Y	7,283,257
		Z	667
Model Parameters	Parent Block Size	X	10
		Y	10
		Z	10
	Number of Blocks	X	667
		Y	330
		Z	269

The block model with outlined oxide and primary mineralisation is shown in Figure 13.15.

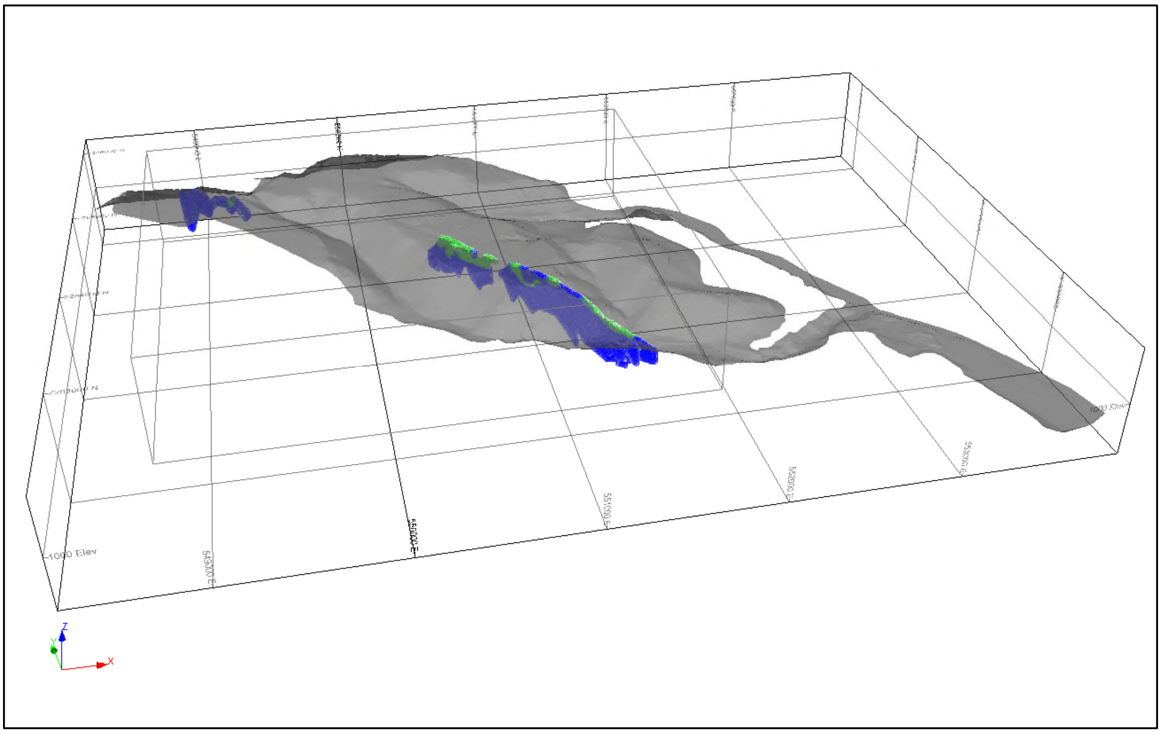


Figure 13.15: Block Model of Mineralisation - Green: oxide, Blue: primary

Parameters of dynamic anisotropy showing the true dip angle and azimuth were interpolated into the blocks of each individual zone of mineralisation. In order to produce the points with true dip angle and azimuth WAI modelled wireframes corresponding with the axial surfaces of mineralized zones. Points with true dip angles and azimuth corresponded with the centers of triangles of these wireframes.

An example of the points used for dynamic anisotropy for Zone 1 is shown in Figure 13.16.

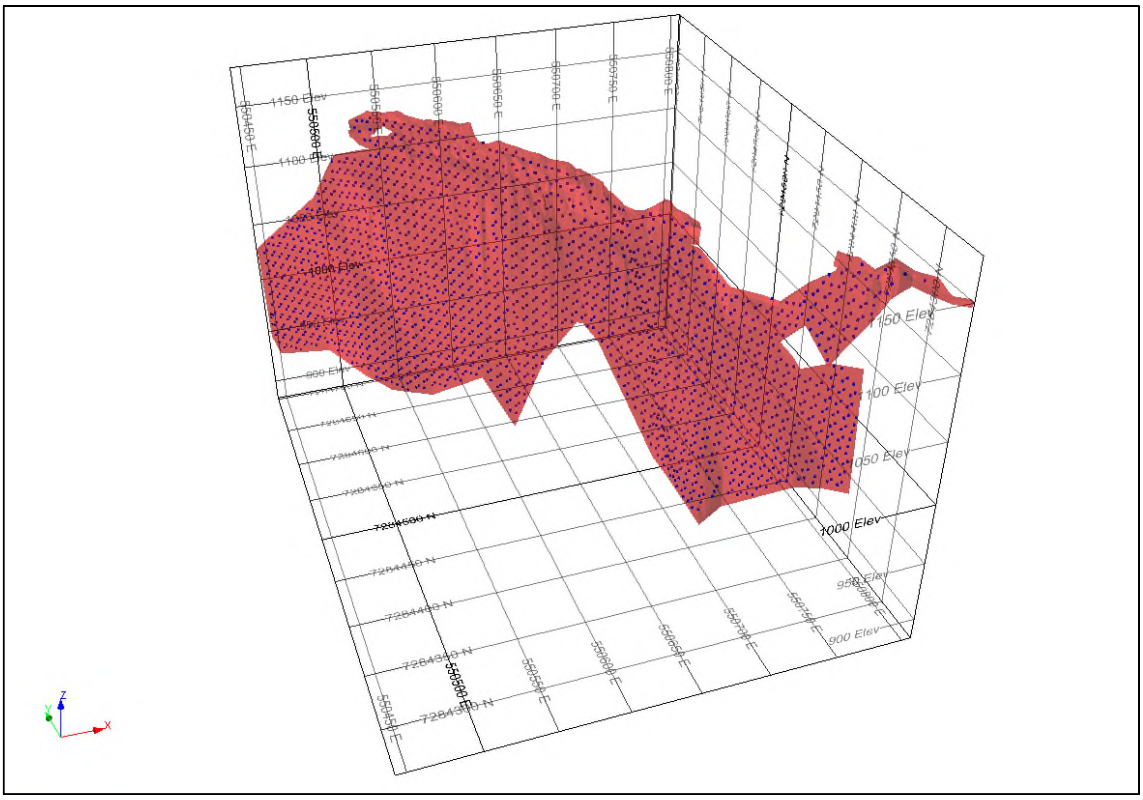


Figure 13.16: Wireframe Model of Zone 1 with Points Used to Determine Dynamic Anisotropy

13.1.8 Density

Density of rocks and ores was studied on 173 samples taken from the core of the 2004-2012 drillholes. It was determined on site and field duplicates were analyzed in State Unitary Mining and Geological Laboratory Yakutskgeology, Republic of Sakha (Yakutia). The summarized data on 144 samples with assays and referenced to the drillholes depths are shown in Table 13.13.

Table 13.13: Density Data for Samples taken in 2004-2012			
Type of Ore	**Average Density for 144 determinations (g/cm^3)**		
	In-House Lab	**Yakutskgeology**	**Average**
Primary + mixed	3,575	3,594	3,584
Oxide	3,125	3,206	3,166

In 2012, a total of 88 samples were taken for primary ores in Drillhole V12-198A of 74m deep to determine the density; the average value amounted to 3.50t/m^3.

As part of the processing studies of ores undertaken by TOMS Engineering LLC in 2015 a total of 53 samples were taken to determine the ore density. The laboratory testwork resulted in the following density values:

- Oxide ores – 3.17g/cm^3
- Mixed ores – 3.38g/cm^3

- Primary ores – 3.59g/cm^3

Investigation of the correlation relationship between the grades of elements of interest (silver, lead, and zinc) with regard to all the previous studies showed a weak dependence between the metal/s grades and density (the correlation coefficient is 0.08 to 0.19). No tendency to decrease/increase in density with depth was determined.

Determination of natural moisture content was carried out both at exploration and development of the deposit. The average value of moisture content based on the mining data from June to December 2018 was 5.6%.

Currently, ZAO Prognoz is using the following density values for development of Vertikalny:

- Oxide mineralisation – 3.13t/m^3
- Primary and mixed mineralisation – 3.56t/m^3
- Host rocks – 2.75t/m^3

The mixed zone at Vertikalny is not significant, therefore no separate mixed zone has been included by WAI in the resource model. The MRE is based on the ZAO Prognoz values for density.

13.1.9 Grade Estimation

Grade estimation was performed only on mineralised material defined within each mineralised zone with oxide and sulphide mineralisation estimated separately. The domains were treated as hard boundaries and composites from an adjacent domain could not be used in the grade estimation of another domain. Ordinary Kriging (OK) and inverse distance weighting to power 3 (IDW^3) estimations were undertaken.

13.1.9.1 Grade Estimation Plan

Grade estimation was undertaken for Ag, Pb and Zn. The estimates were run in a nine-pass plan, with each consecutive pass using progressively larger search radii to enable the estimation of blocks un-estimated on the previous pass. The search parameters were derived from the variography. The first search distances corresponded to the distance at 1/3rd of the variogram range, the second search corresponded to the distance at 2/3^{rds} of the variogram range with the third search distance up to the variogram range. The remaining searches were used to ensure that all blocks contained within the domains were estimated.

The OK method was used as the principal estimation method for all domains. Variogram model parameters for Zone 1 were used for the estimation of Ag for all domains in which no suitable variograms could be derived. Variogram model parameters for Zone 2 were used for the estimation of Pb and Zn all domains in which no suitable variograms could be derived. Sample weighting during grade estimation was determined by variogram model parameters. The IDW^3 method was also used for all domains as a secondary (check) estimation method.

Grade estimation was carried out using a parent block size of 10m x 10m x 10m. Sub-cells received the same grade as the parent cell. Block discretisation was set to 3 x 3 x 3 to estimate block grades. Search ellipse orientations were controlled by dynamic anisotropy. A summary of the grade estimation plan is shown in Table 13.14.

Table 13.14: Vertikalny Grade Estimation Plan

Zone	Metal	Search	Search Distance (m)			Composites		Minimum Octants
			Down Dip	Along Strike	Across Strike	Minimum	Maximum	
Zone 1 and Zones 3 to 9	Ag	1st	6.1	18.6	1.4	2	8	2
		2nd	12.2	37.2	2.8	2	8	2
		3rd	18.3	55.8	4.2	2	8	2
		4th	36.6	111.6	8.4	2	8	2
		5th	73.2	223.2	16.8	2	8	2
		6th	109.8	334.8	25.2	2	8	2
		7th	146.4	446.4	33.6	2	8	1
		8th	292.8	892.8	67.2	1	15	1
		9th	549	1674	126	1	15	1
Zone 2	Ag	1st	10.6	12.9	2.0	2	8	2
		2nd	21.1	25.8	4.0	2	8	2
		3rd	31.7	38.7	6	2	8	2
		4th	63.4	77.4	12	2	8	2
		5th	126.8	154.8	24	2	8	2
		6th	190.2	232.2	36	2	8	2
		7th	253.6	309.6	48	2	8	1
		8th	507.2	619.2	96	1	15	1
		9th	951	1161	180	1	15	1
All Zones	Pb	1st	21.8	30.0	2.0	2	8	2
		2nd	43.7	60.0	3.9	2	8	2
		3rd	65.8	90	5.9	2	8	2
		4th	131	180	11.8	2	8	2
		5th	262	360	23.6	2	8	2
		6th	393	540	35.4	2	8	2
		7th	524	720	47.2	2	8	1
		8th	1048	1440	94.4	1	15	1
		9th	1965	2700	177	1	15	1
All Zones	Zn	1st	33.2	16.5	2.1	2	8	2
		2nd	66.5	33.0	4.1	2	8	2
		3rd	99.7	49.5	6.2	2	8	2
		4th	199.4	99	12.4	2	8	2
		5th	398.8	198	24.8	2	8	2
		6th	598.2	297	37.2	2	8	2
		7th	797.6	396	49.6	2	8	1
		8th	1595.2	792	99.2	1	15	1
		9th	2991	1485	186	1	15	1

Note – Maximum (MAXKEY) of 4 composites per drillhole

13.1.9.2 Validation of Grade Estimate

Following grade estimation, a statistical and visual assessment of the block model was undertaken to 1) assess successful application of the estimation passes 2) to ensure that as far as the data allowed, all blocks within mineralisation domains were estimated and 3) the model estimates performed as expected. The model validation methods carried out included:

- On-screen visual assessment of composite and block model grades;
- SWATH plot (model grade profile) analysis; and
- Mean grade check.

i) On-Screen Check

An on-screen visual assessment of drill hole, composite and block model grades was carried out as shown in Figure 13.17. Visually the model was considered to spatially reflect the composite grades.

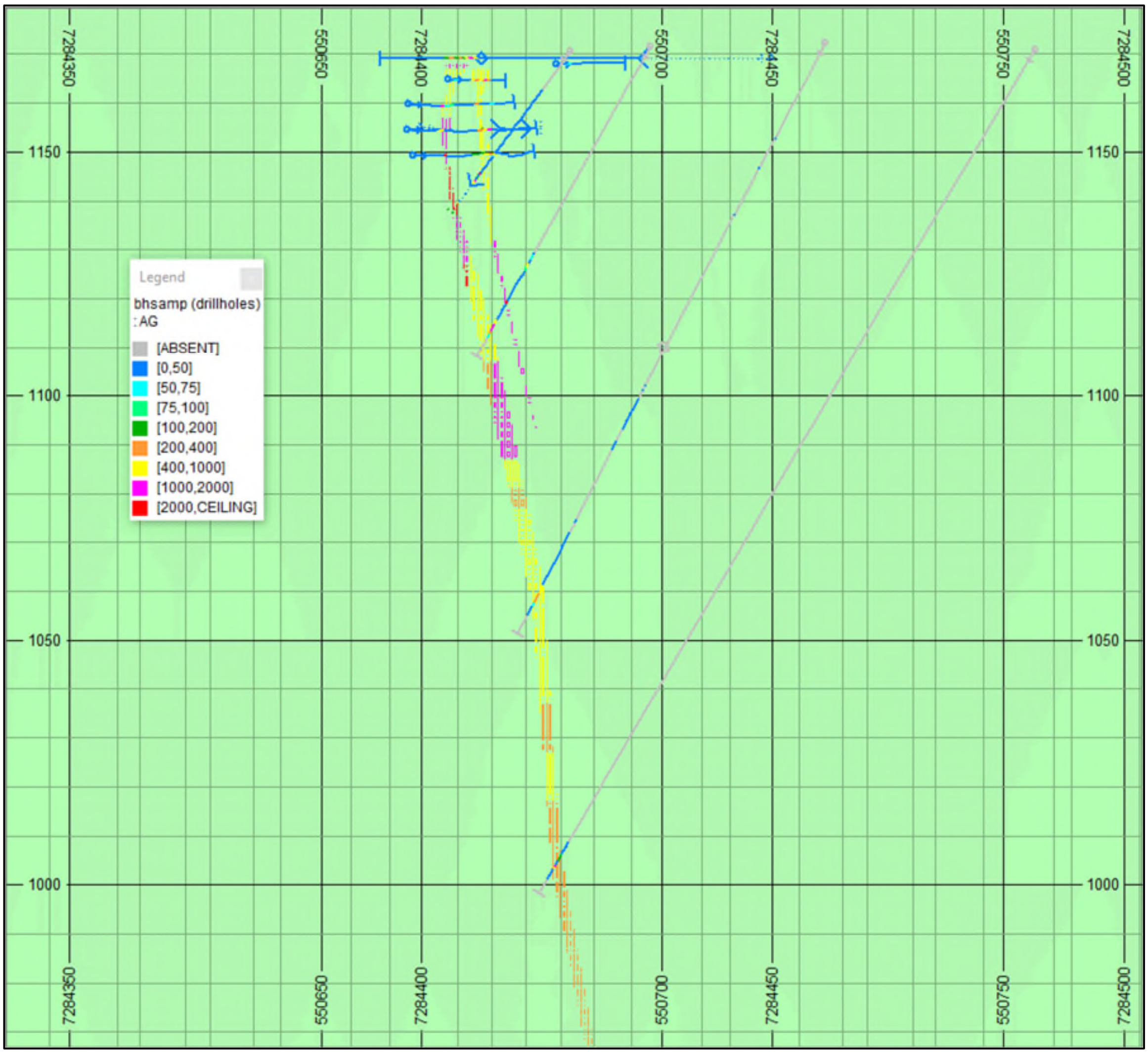


Figure 13.17: Example Cross-Section Comparing Drillhole and Block Model Ag Grades

ii) SWATH Analysis

Swath plots were generated from the model by averaging composites and blocks along panels. Swath plots were generated for all estimation methods and should exhibit a close relationship to the composite data upon which the estimation is based. An example Swath analysis for Ag for the primary mineralisation at Zone 2 is shown in Figure 13.18. The relationship between composite and block grades across the model is considered by WAI to be acceptable. Some deviations between the composite and estimated block grade occur at the edges of the deposit where reduced tonnages accentuate the differences in grade. Differences in grade also become more apparent in lower grade areas. These lower grade areas are typically where the density of drilling decreases and a few composites can have a disproportionate effect on the estimated grades.

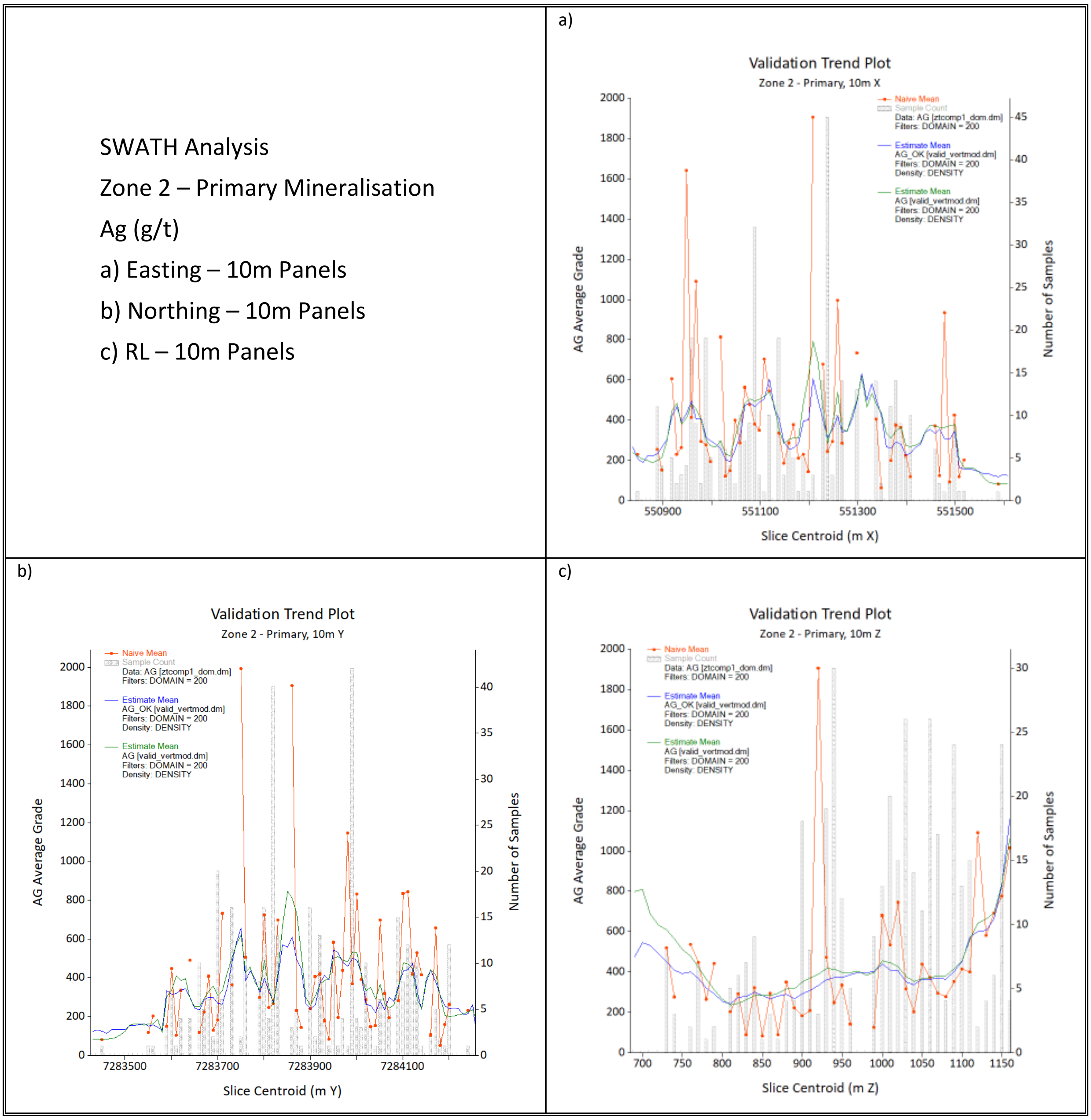


Figure 13.18: Example SWATH Analysis for Zone 2 - Primary Mineralisation

iii) Mean Grade Check

Statistical analysis of the block model was carried out for comparison against the composited drillhole data. This analysis provides a check on the reproduction of the mean grades of the composite data against the model over the global domain. Typically, the mean grade of each domain should not be significantly greater or less than the composites from which it has been derived. A comparison of the mean block model grades and mean composite grades for all domains is shown in Table 13.15. Where discrepancies between the composite mean grades and block model mean grades were observed, these were checked by WAI and seen to result from the spatial distribution of the data rather than errors in the grade estimation. Overall, WAI considers the composite grades and block model grades to be sufficiently comparable.

Table 13.15: Comparison of Composite and Block Model Average Grades

Type	ZONE	Tonnes (t)	Composites		Block Model
			No. of Samples	Mean	Mean
Ag (g/t)					
Oxide	1	298,217	927	985.26	955.03
	2	129,757	115	528.28	425.64
	3	8,313	2	224.00	224.00
	4	15,331	75	497.26	650.28
	5	908	13	257.10	365.57
	6	3,925	34	358.10	374.32
	7	-	-	-	-
	8	-	-	-	-
	9	-	-	-	-
Primary	1	379,435	150	1,004.53	660.38
	2	1,385,502	405	411.02	363.44
	3	371,952	53	475.82	485.34
	4	4,931	15	896.32	1,141.79
	5	-	-	-	-
	6	-	-	-	-
	7	14,617	5	646.16	610.17
	8	54,472	5	276.59	270.17
	9	34,941	8	226.74	241.96
Pb (%)					
Oxide	1	298,217	754	1.89	1.63
	2	129,757	106	1.66	1.27
	3	8,313	2	1.40	1.40
	4	15,331	64	1.15	1.21
	5	908	9	1.19	1.11
	6	3,925	32	3.13	2.89
	7	-	-	-	-
	8	-	-	-	-
	9	-	-	-	-
Primary	1	379,435	111	1.47	1.40
	2	1,385,502	271	1.76	1.95
	3	371,952	42	4.42	4.80
	4	4,931	15	1.22	1.88
	5	-	-	-	-
	6	-	-	-	-
	7	14,617	5	0.13	1.12
	8	54,472	5	5.32	5.46
	9	34,941	5	1.17	1.46
Zn (%)					
Oxide	1	298,217	754	1.77	1.73
	2	129,757	106	1.39	2.44
	3	8,313	2	0.37	0.37
	4	15,331	64	2.63	3.09
	5	908	9	1.28	1.15
	6	3,925	32	1.63	1.86
	7	-	-	-	-
	8	-	-	-	-
	9	-	-	-	-
Primary	1	379,435	111	1.90	2.02
	2	1,385,502	271	2.22	2.34
	3	371,952	42	1.31	1.36
	4	4,931	15	3.88	5.72
	5	-	-	-	-
	6	-	-	-	-
	7	14,617	5	0.28	1.50
	8	54,472	5	2.93	2.20
	9	34,941	5	0.96	1.95

Note – Block model grades based on OK estimates

iv) Validation Summary

The comparison of composite and block model average grades shows significant difference between for Zone 1. Average silver grade for block model is 660.38g/t whereas average composite grade gives 1,004.53g/t.

The detailed analysis of data for primary mineralisation at Zone 1 shows the predominant locations of the high-grade intersections (i.e. above 1,000g/t) occurs on the relatively restricted area in the upper part of mineralization nearby oxide/primary mineralization boundary (Figure 13.19).

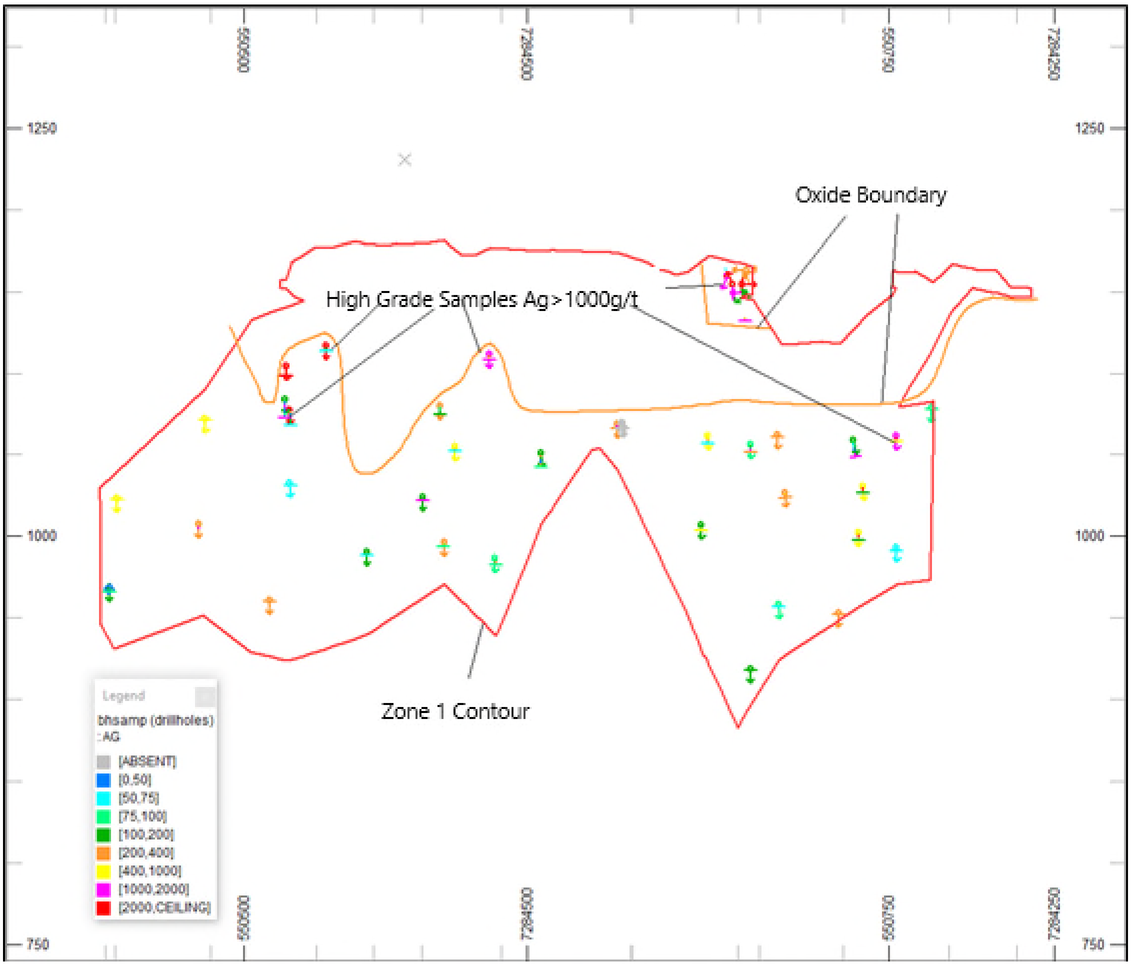


Figure 13.19: Location of the High Grade Silver Composites (>1000g/t) for Primary mineralisation, Zone 1

At the same time, the majority of the drillholes below the oxide boundary have grades of 350-400g/t. During grade interpolation into the block model the influence of the 'rich' samples is blocked by nearest relatively low-grade intersections.

Globally no indications of significant over or under estimation were apparent in the model nor were any obvious interpolation issues identified. From the perspective of conformance of the average model grade to the input data, WAI considers the model to be a satisfactory representation of the sample data used and an indication that the grade interpolation performed as expected. The Mineral Resource Estimate was based upon the OK grade estimation.

13.1.10 Selective Mining Units

No selective mining unit was applied at the resource stage. A minimum block size of 1m x 1m x 1m was however applied during block model construction. Mining selectivity, including mining dilution (planned and unplanned) and mining losses was incorporated during the mining study.

13.1.11 Depletion of Mined-Out Resources

Mineral resources were depleted by WAI based on an open pit mine survey supplied by the Client and dated 31 May 2019.

13.1.12 Reconciliation

CJSC Prognoz provided grade control and actual mining data for the period from November 2018 to July 2019 inclusive. In addition, the open pit survey data as of late October 2018 and late July 2019 was also provided. The grade control data was used by WAI for comparison with the WAI model. The WAI model was limited to the open pit surfaces as at October 2018 and July 2019 and the results of the comparison is given in Table 13.16.

Table 13.16: Block Model vs Grade Control Data from October 2018 to July 2019 – Vertikalny

Source	Ore, t	Grade, g/t	Silver, kg
Grade control model	66,339.90	877.83	58,235.46
WAI model	61,024.72	996.78	60,828.42
Absolute difference	**5,315.18**	**-118.95**	**-2,592.95**
Relative difference, %	**109%**	**88%**	**96%**

Overall, the grade control model and the WAI model compare well with slightly higher tonnes and lower grades reporting from the grade control model. The difference in contained silver metal between the two models is approximately 4%.

13.1.13 Mineral Resource Classification

The Mineral Resource classification for the Vertikalny deposit was undertaken by WAI in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2012)]. The principles governing the operation and application of the JORC Code are Transparency, Materiality and Competence:

- Transparency requires that the reader of a Public Report is provided with sufficient information, the presentation of which is clear and unambiguous, to understand the report and not be misled by this information.
- Materiality requires that a Public Report contains all the relevant information that investors and their professional advisers would reasonably require, and reasonably expect to find in the report, for the purpose of making a reasoned and balanced judgement regarding the Exploration Results, Mineral Resources or Ore Reserves being reported.
- Competence requires that the Public Report be based on work that is the responsibility of suitably qualified and experienced persons who are subject to an enforceable professional code of ethics.

13.1.13.1 Considerations for Vertikalny Resource Classification

To classify the Vertikalny deposit, WAI has taken into account the following indicators:

- Geological Continuity and Complexity;
- QAQC Results - Quality of Data;
- Spatial Grade Continuity - Results of Geostatistical Analysis; and
- Quality of Block Model.

WAI considers that silver, lead and zinc mineral resources can be classified as Measured, Indicated and Inferred.

ii) Geological Continuity and Complexity

With the current drill hole/trench spacing, geological continuity between exploration profiles both along strike and down dip is evident. The current drill hole spacing allows for interpretation of continuous zones of mineralisation based on the cut-off grade of 50g/t Ag.

iii) Data Quality

QA/QC results of exploration data show acceptable results when measuring accuracy, precision and contamination. This data can be used for estimation of mineral resources.

iv) Spatial Grade Continuity

An assessment of spatial grade continuity is important when assigning classification to a Mineral Resource. The confidence that can be placed in the variogram parameters is a major consideration when determining classification. The data used in geostatistical analysis resulted in reasonably robust along strike and down dip variogram structures for silver, lead, and zinc allowing the determination of the most appropriate search parameters.

v) Block Model Veracity

Validation of the block model has shown the estimated grades to be a good reflection of the input composite grades. Visual and statistical checks reveal no evidence of major under or over estimation.

13.1.13.2 Final Classification

WAI considers that the Vertikalny Mine has been sufficiently explored to assign Measured, Indicated, and Inferred Mineral Resources as defined by JORC Code (2012).

Based on the geostatistical studies, and achieved drillhole spacing, the following criteria was used to define resource categories at Vertikalny.

- ***Measured Mineral Resources*** – belong to the interpreted principal mineralised zone, based on a drill grid of 40m by 40m along strike and down dip, where grade continuity is confirmed.
- ***Indicated Mineral Resources*** – belong to the interpreted principal mineralised zone, based on a drill grid of 80m by 80m along strike and down dip; the grade continuity can be confirmed.
- ***Inferred Mineral Resources*** – belong to the interpreted principal mineralised zone, based on a drill grid of >80m by 80m along strike and down dip; the grade continuity can be confirmed.

An isometric view of the block model Mineral Resource classification is shown in Figure 13.20.

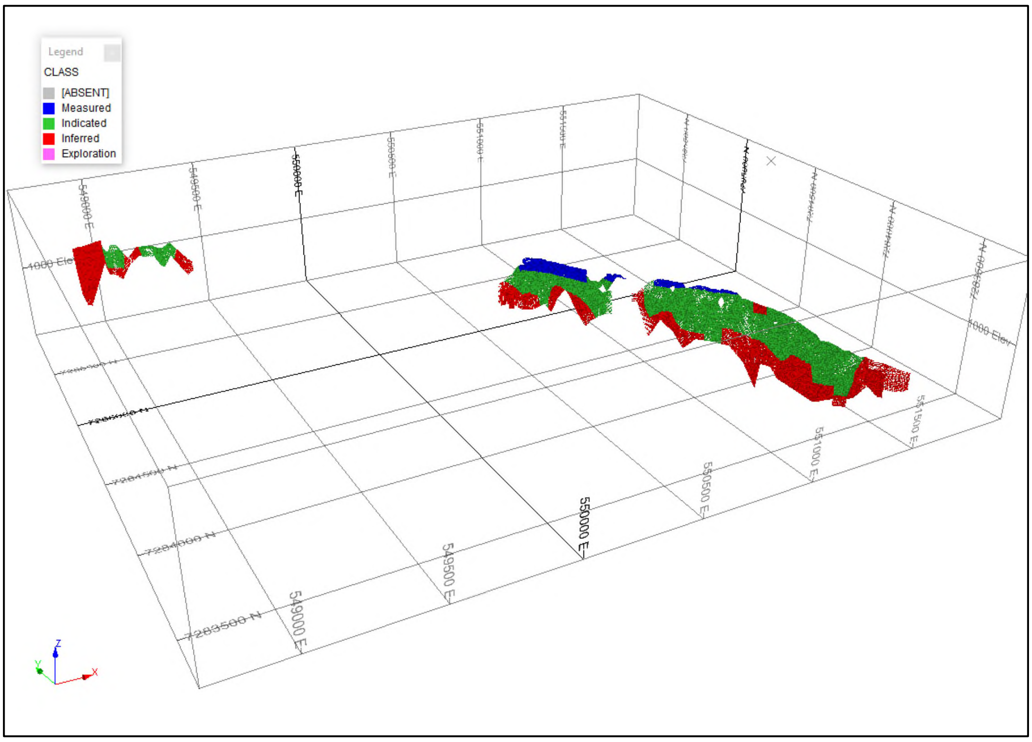


Figure 13.20: Unconstrained Block Model Classification

13.1.14 Mineralised Inventory

A mineral inventory includes all mineralisation contained at a deposit and has not been limited by a cut-off grade or optimised pit shell. A mineral inventory therefore does **not** reflect a Mineral Resource Estimate in accordance with the guidelines of the JORC Code (2012) but does however provide an indication of the total mineralisation contained in a deposit that has potential to be economic in the future. The mineralised inventory for Vertikalny is presented in Table 13.17 and contains all mineralisation contained within the mineralised zones and depleted to an open pit mine survey dated 31 May 2019.

Table 13.17: Mineral Inventory at Vertikalny within Wireframe Models (Depleted as of 31 May 2019)

Zone	Class	Volume, m^3, 000	Tonnage, kt	Grade			Contained Metal		
				Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
Oxide									
1	Measured	29.83	93.36	869.67	2.04	1.50	81,191	1,903	1,400
2	Measured	4.61	14.42	519.94	1.55	0.41	7,499	224	59
4	Measured	1.90	5.95	756.17	1.16	3.13	4,502	69	187
5	Measured	0.14	0.44	345.53	1.42	1.25	152	6	6
Total Measured		**36.48**	**114.18**	**817.54**	**1.93**	**1.45**	**93,344**	**93,344**	**2,202**
1	Indicated	37.99	118.92	1 018.32	1.23	1.95	121,100	1,459	2,314
2	Indicated	35.70	111.73	378.78	1.10	2.52	42,321	1,224	2,815
3	Indicated	2.66	8.31	224.00	1.40	0.37	1,862	116	31
4	Indicated	0.21	0.65	1 105.82	0.58	2.26	720	4	15
Total Indicated		**76.55**	**239.61**	**692.79**	**1.17**	**2.16**	**166,003**	**166,003**	**2,802**
Measured + Indicated		**113.03**	**353.79**	**733.05**	**1.41**	**1.93**	**259,347**	**259,347**	**5,004**
2	Inferred	0.15	0.47	160.37	2.09	1.13	75	10	5
Total Inferred		**0.15**	**0.47**	**160.37**	**2.09**	**1.13**	**74.79**	**75**	**10**
Primary									
1	Measured	1.25	4.46	1 627.58	1.56	1.44	7,260	69	64
2	Measured	4.27	15.19	689.26	1.59	0.62	10,470	242	94
4	Measured	0.69	2.46	1 208.06	2.04	6.75	2,972	50	166
Total Measured		**6.21**	**22.11**	**936.28**	**1.64**	**1.46**	**20,702**	**20,702**	**362**
1	Indicated	63.45	225.89	778.21	1.44	2.18	175,786	3,260	4,928
2	Indicated	246.18	876.40	359.01	1.97	2.62	314,633	17,246	22,999
3	Indicated	23.72	84.45	403.22	4.67	1.64	34,052	3,942	1,382
9	Indicated	6.23	22.17	237.90	1.40	1.51	5,274	310	335
Total Indicated		**339.58**	**1,208.90**	**438.20**	**2.05**	**2.45**	**529,745**	**529,745**	**24,758**
Measured + Indicated		**345.79**	**1,231.02**	**447.15**	**2.04**	**2.43**	**550,447**	**550,447**	**25,119**
1	Inferred	41.44	147.51	382.34	1.17	1.53	56,398	1,727	2,250
2	Inferred	137.95	491.08	347.13	1.80	1.70	170,471	8,822	8,327
3	Inferred	80.76	287.50	506.92	4.42	1.14	145,740	12,720	3,281
7	Inferred	4.11	14.62	589.82	1.10	1.57	8,622	161	229
8	Inferred	15.30	54.47	270.73	5.49	2.21	14,747	2,993	1,204
9	Inferred	3.59	12.77	279.87	1.28	2.20	3,575	164	282
Total Inferred		**283.13**	**1,007.96**	**396.40**	**2.64**	**1.55**	**399,553**	**399,553**	**26,588**

Note – A mineralised inventory is **not** a Mineral Resource Estimate as the potential for economic extraction has not been demonstrated

13.1.15 Reasonable Prospects for Economic Extraction

For a deposit, or portion of a deposit, to be classified as a Mineral Resource there must be reasonable prospects for eventual economic extraction (the JORC Code [2012]). The model classified as described above was therefore further limited by economic parameters as described in this section.

The prospects for eventual economic extraction were tested by running an open pit optimisation using Datamine's NPV Scheduler software and using the parameters listed in Table 13.18.

Table 13.18: Optimisation Parameters for Constraining Open Pit Mineral Resources

Parameter	Unit	Value	Comments
Annual production rate – Mining and Processing	kt	115	SBR data
Operational costs for:			SBR data
Ore mining	US$/t	*2.53*	SBR data
Oxide ore processing	US$/t	*72.91*	SBR data
Primary ore processing	US$/t	*46.97*	SBR data
G&A	US$/t	*60*	SBR data
Metal Recovery	%	95	Tetra Tech data
Dilution	%	30	Tetra Tech data
Discount rate	%	8	WAI Estimate
Slope angle	Hanging wall	56	SRK data
Slope angle	Foot wall	48	SRK data
Note – Processing cost includes cost processing cost itself and G&A cost			

Parameters used to constrain Mineral Resources for underground mining are given in Table 13.19.

Table 13.19: Parameters used to Constrain Underground Mineral Resources

Parameter	Unit	Value	Comments
Operational costs for:			
Ore mining	US$/t of ore	55	SBR data
Processing of primary ore (tonnage of oxide ores is insignificant, the major type of mineralisation for underground is primary ore)	US$/t of ore	46.97	SBR data
G&A	US$/t of ore	60	SBR data
NSR	US$/t of ore	162	WAI estimate

The NSR calculation is shown in Table 13.20 below.

Table 13.20: Data for NSR Calculation

Parameter	Unit	SULPHIDE			OXIDE	Comment
		Zn Conctrate	Lead Concentrate	Pb/Ag Middlings		
Metal Prices						1.15x spot prices 27.08.19
Ag	US$/tOz	20.42	20.42	20.42	20.42	
Pb	US$/t	2,379.35	2,379.35	2,379.35	2,379.35	
Zn	US$/t	2,589.80	2,589.80	2,589.80	2,589.80	
Mill Recovery						SBR
Ag	%	4.7	65.0	15.6	85	
Pb	%	0	65.9	0	0	
Zn	%	82.2	0	0	0	
Concentrate Assay						SBR
Ag	g/t	Variable	Variable			
Pb	%	0.00	17.1			
Zn	%	42.3	0.00			
Moisture Content	%	0	0			Assumed 0%
Smelter Payment						SBR - Pb/Zn payability
Ag Payability	%	45	84	98	98	WAI Estimate - Ag Payability
Pb Payability	%	0	84	0	0	WAI Estimate - Deductions
Zn Payability	%	45	0	0	0	
Ag Deductions	g/t	0	0	0	0	
Pb Deductions	%	0	0	0	0	
Zn Deductions	%	0	0	0	0	
Treatment Charge/Refining Charge						SBR
Transport	US$/tconc	274.9	274.9	0	0	
Treatment	US$/tconc	0	0	0	0	
Refining (Ag)	US$/tOz	0.48	0.48	0.48	0.48	
NSR Factor						
Ag	US$/g/$t_{ore}$	0.46				
Pb	US$/%/$t_{ore}$	2.58				
Zn	US$/%/$t_{ore}$	4.24				

NSR cut-off values were used to evaluate the Mineral Resources based on mineralisation type and open pit/underground mining methods as shown in Table 13.21. It should be noted that the amount of oxide mineralisation for underground mining is insignificant and therefore only primary mineralisation has been considered for underground mining. The higher NSR cut-off value for open pit mining of oxide compared to primary is due to a higher processing cost of oxide.

Table 13.21: NSR COG for Open Pit and Underground Mining

Method	Mineralisation Type	Unit	NSR
Open pit mining	Oxide	US$/t	172.78
Open pit mining	Primary	US$/t	139.06
Underground mining	Primary	US$/t	162.00

Open pit Mineral Resources limited by the optimised pit shell are shown in Figure 13.21.

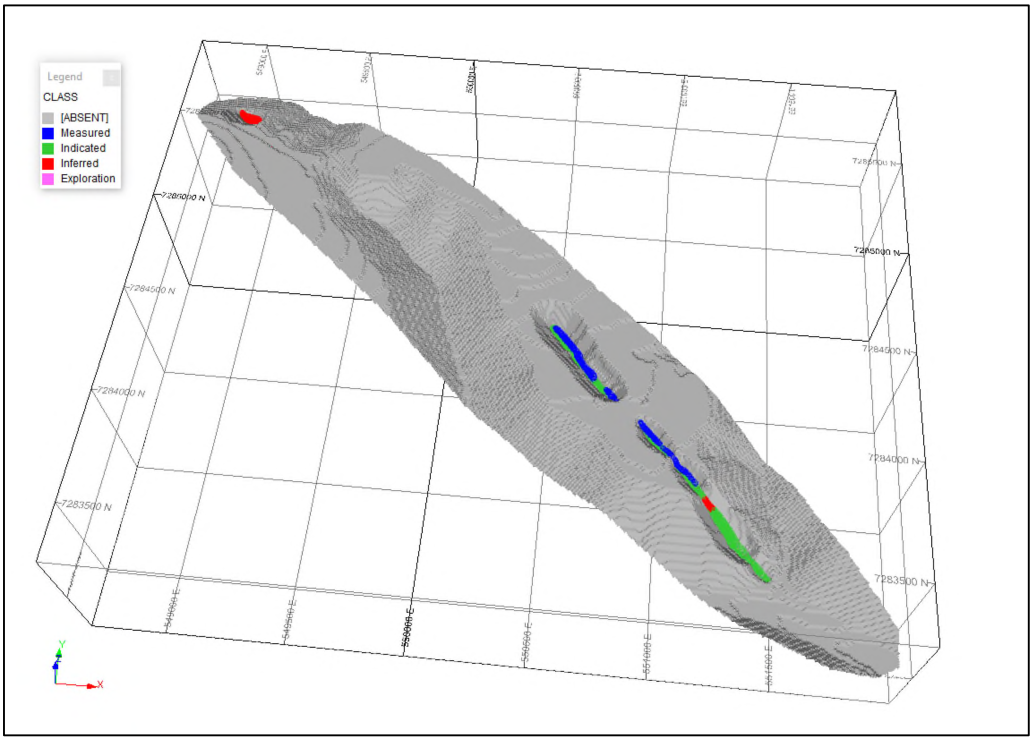


Figure 13.21: Mineral Resources for Open Pit Mining

Underground Mineral Resources located below the base of the optimised pit shell and above the NSR cut-off value of $130/t are shown in Figure 13.22.

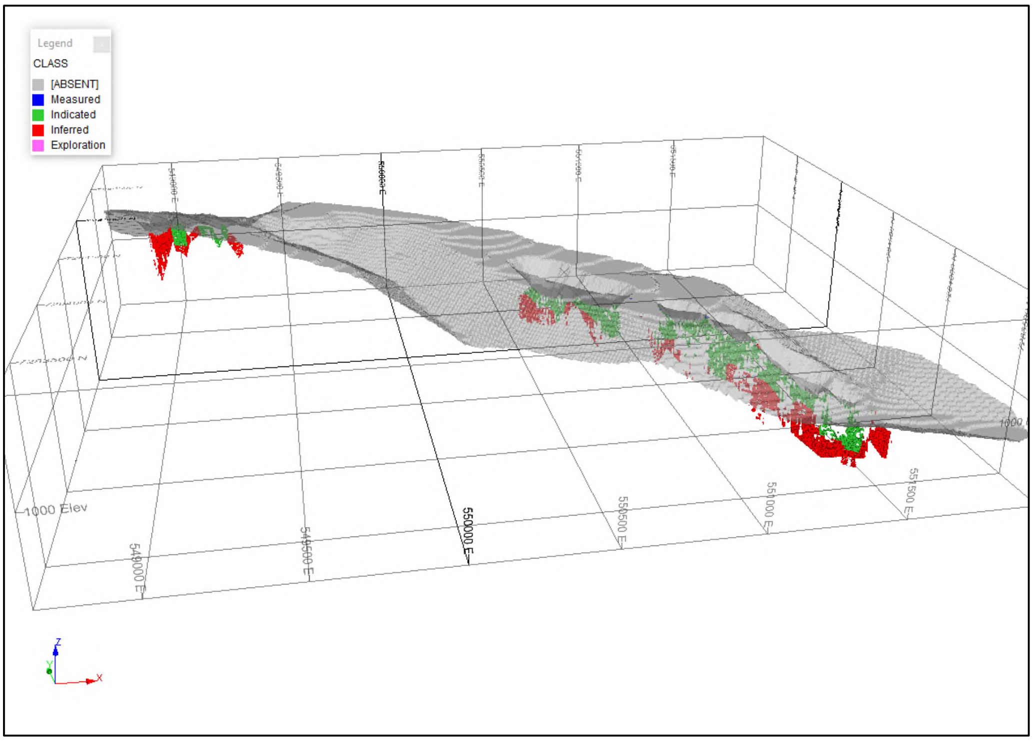


Figure 13.22: Mineral Resources for Underground Mining

13.1.16 Mineral Resource Statement

The Mineral Resource estimate for the Vertikalny deposit is classified in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2012)].

The stated Mineral Resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the author. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this Mineral Resource Estimate currently.

The effective date of the Mineral Resource Estimate is 31st of May 2019.

The Mineral Resource statement for the open pit resources at Vertikalny is shown in Table 13.22.

The Mineral Resource statement for the underground resources at Vertikalny are shown in Table 13.23.

Table 13.22: Mineral Resource Estimate. Vertikalny Project, Russia. 31st May 2019 (In Accordance with the Guidelines of the JORC Code (2012)) Potential Open Pit Resources

Ag Cut-off, g/t	Category	Tonnes, Kt	Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
50	**Oxide**							
	Measured	108.53	845.52	1.97	1.53	91,766	2,143	1,656
	Indicated	97.00	1 096.62	1.30	1.94	106,368	1,256	1,886
	Sub-Total M+I	***205.53***	***964.03***	***1.65***	***1.72***	***198,133***	***3,399***	***3,542***
	Primary							
	Measured	14.07	1 250.53	1.76	1.93	17,598	247	271
	Indicated	37.65	1 760.51	2.22	1.47	66,291	835	555
	Sub-Total M+I	***51.73***	***1 621.77***	***2.09***	***1.60***	***83,889***	***1,082***	***826***
	Oxide + Primary							
	Total M+I	***257.25***	***1 096.28***	***1.74***	***1.70***	***282,022***	***4,481***	***4,368***
100	**Oxide**							
	Measured	102.26	892.45	1.99	1.55	91,260	2,036	1,588
	Indicated	94.26	1 126.55	1.29	1.96	106,185	1,217	1,846
	Sub-Total M+I	***196.51***	***1 004.73***	***1.66***	***1.75***	***197,445***	***3,253***	***3,434***
	Primary							
	Measured	13.41	1 308.56	1.84	1.93	17,548	246	259
	Indicated	36.65	1 806.77	2.26	1.43	66,212	827	526
	Sub-Total M+I	***50.06***	***1 673.30***	***2.14***	***1.57***	***83,761***	***1,073***	***785***
	Oxide + Primary							
	Total M+I	***246.57***	***1,140.46***	***1.75***	***1.71***	***281,205.34***	***4,325.70***	***4,218.76***
200	**Oxide**							
	Measured	94.90	949.88	2.01	1.58	90,141	1,909	1,500
	Indicated	89.24	1 181.88	1.33	1.92	105,469	1,190	1,710
	Sub-Total M+I	***184.14***	***1 062.32***	***1.68***	***1.74***	***195,610***	***3,099***	***3,211***
	Primary							
	Measured	13.19	1 328.95	1.85	1.96	17,524	244	258
	Indicated	36.14	1 830.08	2.28	1.42	66,148	825	514
	Sub-Total M+I	***49.33***	***1 696.13***	***2.17***	***1.56***	***83,672***	***1,069***	***772***
	Oxide + Primary							
	Total M+I	***233.47***	***1,196.24***	***1.79***	***1.71***	***279,281.95***	***4,168.20***	***3,982.53***
300	**Oxide**							
	Measured	87.08	1 012.09	1.88	1.57	88,130	1,635	1,371
	Indicated	84.03	1 239.87	1.25	1.90	104,191	1,054	1,599
	Sub-Total M+I	***171.11***	***1 123.96***	***1.57***	***1.74***	***192,321***	***2,689***	***2,971***
	Primary							
	Measured	12.78	1 362.31	1.89	2.00	17,416	242	255
	Indicated	35.28	1 868.86	2.33	1.40	65,926	820	492
	Sub-Total M+I	***48.06***	***1 734.12***	***2.21***	***1.56***	***83,342***	***1,062***	***748***
	Oxide + Primary							
	Total M+I	***219.17***	***1,257.75***	***1.71***	***1.70***	***275,662***	***3,715***	***3,718***

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. Mineral Resources are not Ore Reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study.
3. Mineral resources include all potential mineable tonnage.
4. Mineral Resources are estimated as of 31 May 2019 based on an open pit mine survey of the same date.
5. Mineral Resources were constrained by an optimised pit shell using a NSR cut-off value of $172.78/t for oxide and $139.06/t for primary mineralisation.
6. Mineral Resources were constrained by an optimised pit shell based on economic and mining parameters provided by the Client and/or accepted by WAI.
7. This mineral resource estimate is not limited to any factors in terms of environmental, permitting, legal, title, taxation, socio-economic, market and other relevant factors.
8. The metal resources include all the in-situ metal disregard the metallurgical recovery factor.
9. All values in the tables have been rounded with relative accuracy of estimate. Numbers may not compute due to rounding.

Table 13.23: Mineral Resource Estimate. Vertikalny Project, Russia. 31st May 2019 (In Accordance with the Guidelines of the JORC Code (2012)) Potential Underground Resources

Ag Cut-off, g/t	Category	Tonnes, Kt	Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
50	Measured	0.52	383.12	2.52	0.55	199	13	3
	Indicated	419.06	463.13	1.12	2.59	194,076	4,675	10,847
	M+I	***419.58***	***463.03***	***1.12***	***2.59***	***194,275***	***4,688***	***10,850***
	Inferred	222.40	362.49	1.02	1.66	80,619	2,270	3,693
100	Measured	0.38	499.55	2.24	0.57	188	8	2
	Indicated	394.83	486.28	1.11	2.61	191,997	4,392	10,306
	M+I	***395.20***	***486.29***	***1.11***	***2.61***	***192,185***	***4,401***	***10,308***
	Inferred	214.55	372.81	1.02	1.62	79,985	2,178	3,465
200	Measured	0.36	515.71	2.32	0.58	185	8	2
	Indicated	328.27	555.26	1.16	2.52	182,275	3,806	8,267
	M+I	***328.63***	***555.22***	***1.16***	***2.52***	***182,460***	***3,814***	***8,269***
	Inferred	159.76	445.01	1.03	1.70	71,094	1,650	2,714
300	Measured	0.29	581.70	2.66	0.58	166	8	2
	Indicated	235.82	680.72	1.26	2.57	160,524	2,964	6,059
	M+I	***236.10***	***680.60***	***1.26***	***2.57***	***160,690***	***2,972***	***6,061***
	Inferred	109.42	538.93	1.26	1.75	58,970	1,378	1,919

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. Mineral Resources are not Ore Reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study.
3. Mineral resources include all potential mineable tonnage.
4. Mineral Resources are estimated as of 31 May 2019 based on an open pit mine survey of the same date.
5. Mineral Resources are located below an optimised pit and were evaluated based on an NSR cut-off value of $162.00/t for primary mineralisation.
6. Economic and mining parameters provided by the Client and/or accepted by WAI were incorporated in the calculation of NSR.
7. This mineral resource estimate is not limited to any factors in terms of environmental, permitting, legal, title, taxation, socio-economic, market and other relevant factors.
8. The metal resources include all the in-situ metal disregard the metallurgical recovery factor.
9. All values in the tables have been rounded with relative accuracy of estimate. Numbers may not compute due to rounding.

13.1.16.1 Comparison to Previous Mineral Resource Estimates

A mineral resource estimate was undertaken by OREALL in 2019 as part of a TEO study of cut-off criteria. The estimation was carried out using geological blocks for 50, 75, 150, and 250g/t Ag COG. Mineral resources were estimated by OREALL for both open pit and underground mining scenarios. It is understood that the estimate by OREALL was not signed off as being in accordance with any international reporting standards e.g. JORC. The most suitable option for comparison is using a 50g/t Ag cut-off grade as WAI used the same cut-off grade to model the mineralised wireframes.

The comparison included mined-out material as this was included in the OREALL estimate. The WAI estimate used the optimised open pit shell from the MRE. The results of comparison are shown in Table 13.24. The two estimates are considered comparable.

Table 13.24: OREALL MRE (2019) vs WAI MRE (2019)
(Cut-Off Grade of 50g/t Ag)

Source	Mineral resources	Ore (kt)	Grade (g/t)	Silver (kg)
OREALL	Within the open pit shell	726	705	511,503
OREALL	Below the open pit shell	1,858	397	738,091
OREALL	**Total**	**2,583**	**484**	**1,249,594**
WAI	Within the open pit shell	733	794	582,197
WAI	Below the open pit shell	1,974	371	732,053
WAI	**Total**	**2,707**	**485**	**1,314,250**
Difference (%)		**+5%**	**0%**	**+5%**

13.2 Mineral Resources Estimate – North Mangazeisky

13.2.1 General Methodology

The following section describes the process of Mineral Resource estimation of the North Mangazeyskiy silver deposit. The estimate has been carried out in accordance with the guidelines of the JORC Code (2012).

The Mineral Resource Estimate (MRE) was carried out with a 3D block modelling approach using Datamine Studio 3 software (Datamine). Exploration data were imported and verified before being used for modelling mineraliseв wireframes. Besides, digital surface models, mining boundaries, overburden surface, and contours/boundaries of oxide and primary material were imported and/or created. Sample data were selected within mineralisation wireframes and their populations were assessed for outliers. The wireframe envelopes were used as the basis for a volumetric block model based on a parent cell size of 10m x 10m x 10m. Variogram models were constructed based on composite data and used for grade interpolation using Ordinary Kriging (OK) and Inverse Power of Distance methods. The resultant estimated grades were validated against the input samples and composites. The mineralisation was classified in accordance with the guidelines of the JORC Code (2012) and based on an assessment of geological and silver grade continuity of the mineralised zones. Mineral Resources were defined according to the expectation of eventual economic extraction by being constrained within an optimised open pit shell generated using NPV Scheduler and underground stopes optimised using Mineable Shape Optimiser module of Datamine Studio 5D Planner, based on appropriate economic and technical parameters.

13.2.2 Data Transformations and Software

13.2.2.1 Data Transformations

All data are stored using the same local co-ordinate system and the same unit convention based on the WGS84 system. Therefore, transformations of drillhole or other data were not required.

13.2.2.2 Software

The MRE has relied on several software packages for the various stages of the process. However, the main data preparation and validation, wireframe modelling, statistical and geostatistical analysis, block modelling, estimation and validation were performed in Datamine Studio 3 version 3.22.84.0.

13.2.3 Database

13.2.3.1 Exploration Database

i) Input Data

The structure of North Mangazeysky database is similar to that of Vertikalnoye. Exploration database for the period from 2004 to 2016 was supplied by the Client in MS Access and Excel format with separate files for collar/trench starting point, downhole survey for drill holes and bearing/dip for trenches, and assay. An excel file was also provided with codes of lithologies and petrography for both ore and waste, together with their oxidation degree. The relevant imported data in each of these files are listed in Table 13.25.

Table 13.25: Information in Exploration Database Files

Collar File		Assay File		Survey File	
Column	**Explanation**	**Column***	**Explanation**	**Column**	**Explanation**
Project	Exploration area	Project	Exploration area	Project	Exploration area
Hole	drill hole/trench ID No.	Hole	ID No. of drill hole/trench	Hole	drill hole/trench ID No.
Length	Depth/length of drill hole/trench	From_m	Interval from	Depth	Глубина замера
UTM_Grid	Coordinate system	To_m	Interval to	Dip	Inclination angle
UTM_East	Collar easting	DHSample	Sample No.	Measured_Azimuth	Bearing of drill hole/trench
UTM_North	Collar northing	Sample_Type	Sample type	**Lithology file**	
UTM_Elevation	Collar elevation	Pimary_Sample	Original sample No. for duplicates	Project	Exploration area
Azimuth	Bearing of drill hole/trench	Au_OL_ppm	Au, g/t	Hole	drill hole/trench ID No.
Dip	Inclination angle	Ag_OL_ppm	Ag, g/t	From_m	Interval from
Hole_Type	Type of drill hole/trench	Cu_OL_pct	Cu, %	To_m	Interval to
Drill_Rig	Drill rig details	Pb_OL_pct	Pb, %	Lith1	Rock code
Timestamp	Closure date	Zn_OL_pct	Zn, %	Lith1_Oxidation	Oxidation degree
		* assay data for 32 elements are not included into the estimate			

13.2.3.2 Database Summary

A summary of the exploration database for North Mangazeysky is shown in Table 13.26. The database includes the surface diamond drill holes and trenches completed as part of the geological exploration phase. The trenches were excavated during the period from 2004 to 2015, the drilling was undertaken between 2005 and 2016. The locations of drill hole collars by years and trenches completed at the exploration phase is shown on Figure 13.23.

Table 13.26: Summary of Database

Exploration types	Number	Total length, m
Drill holes	157	7,096.80
Trenches	50	566.60
Total	**207**	**7,663.40**

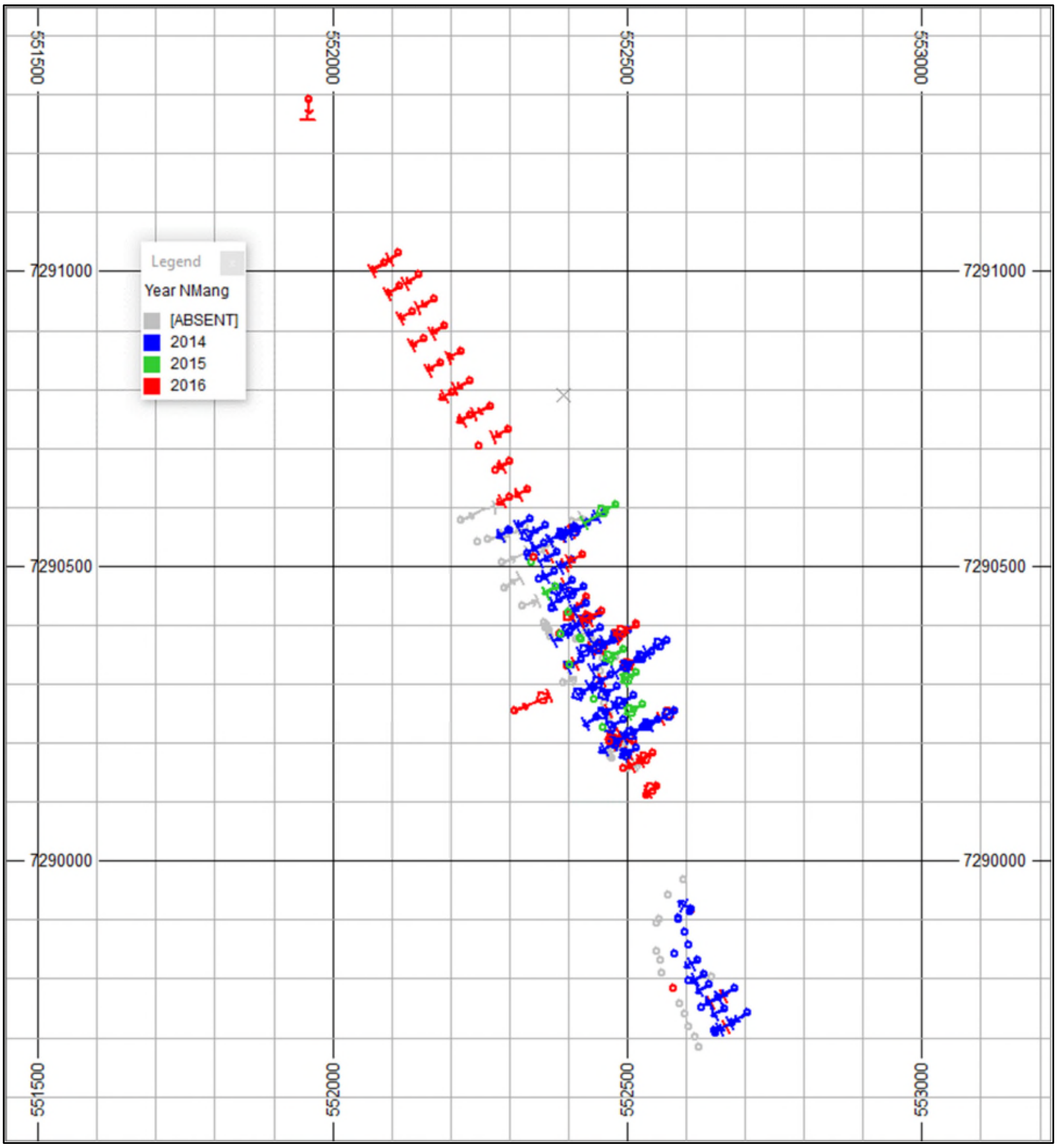


Figure 13.23: Locations of drill hole collars and trenches completed at the exploration phase. Trenches as shown in grey and the drill holes are shown according to the legend.

ii) Database Processing

The individual geological exploration and grade control database files were imported into Datamine. The data from the files then were desurveyed in accordance with the coordinates, downhole survey, assay data and lithologies. Verification was carried out during the desurveying process to ensure that no duplicate or overlapping samples were included in the final database.

Collar locations were checked against the current or pre-mining topographic surfaces and were found to be consistent. Deviation of downhole surveys was checked to ensure that no significant deviations were recorded.

Distribution of samples, where assay detected silver grades, between the exploration types is shown in Table 13.27.

Table 13.27: Distribution of Samples between Exploration Types		
Exploration types	**No. of samples**	**% of total No. of samples**
Drill holes	2,514	83%
Trenches	513	17%
Total	**3,027**	**100%**

13.2.4 Wireframe Modelling

13.2.4.1 Introduction

Prognoz CJSC provided topographical survey in AutoCAD format, which was then used to create a digital terrain model (DTM). In addition, WAI also modelled the overburden based on drill hole logging data.

WAI made an attempt to model the boundary of the oxide zone based on trench and drill hole geological logging data. However, the provided data contained contradictory information, where mineralised intervals in adjacent holes/trenches were different mineralisation types, and intervals within one mineralised intersection were often assigned different oxidation degree (from primary to oxide material types).

13.2.4.2 Mineralised Wireframe Modelling

The mineralised wireframe modelling for Mangazeysky was based on the same cutoff parameters as for Vertikalny:

- Cut-off grade – 50g/t Ag;
- Minimum mineralised interval included into wireframe model – 1m;
- Maximum waste interval included into the mineral wireframe – 3m.

It should be noted that both the thickness and the grade of the mineralisation both at Vertikalnoye and Mangazeyskoye has a significantly variable nature, and in order to maintain continuity and consistency of mineralisation and in order to maintain mineralised continuity, and/or to avoid a redundant splitting of mineralised intervals, there was some flexibility permitted in the parameters listed above.

As a result, a total of 17 individual mineralised zones were modelled at Mangazeysky (Figure 13.24), including three major zones – Zone 1 and Zone 3 in the central part and Zone 17 in the southeastern part of the deposit. The other zones have insignificant extent and are intersected by holes/trenches in 1-3 up to 4 exploration profiles. Minor mineralisation zones are located mainly above main zone 1 and also below this zone. The mineralised zones have north-west strike (bearing 330-340°), dip angle is 30-40° northeast.

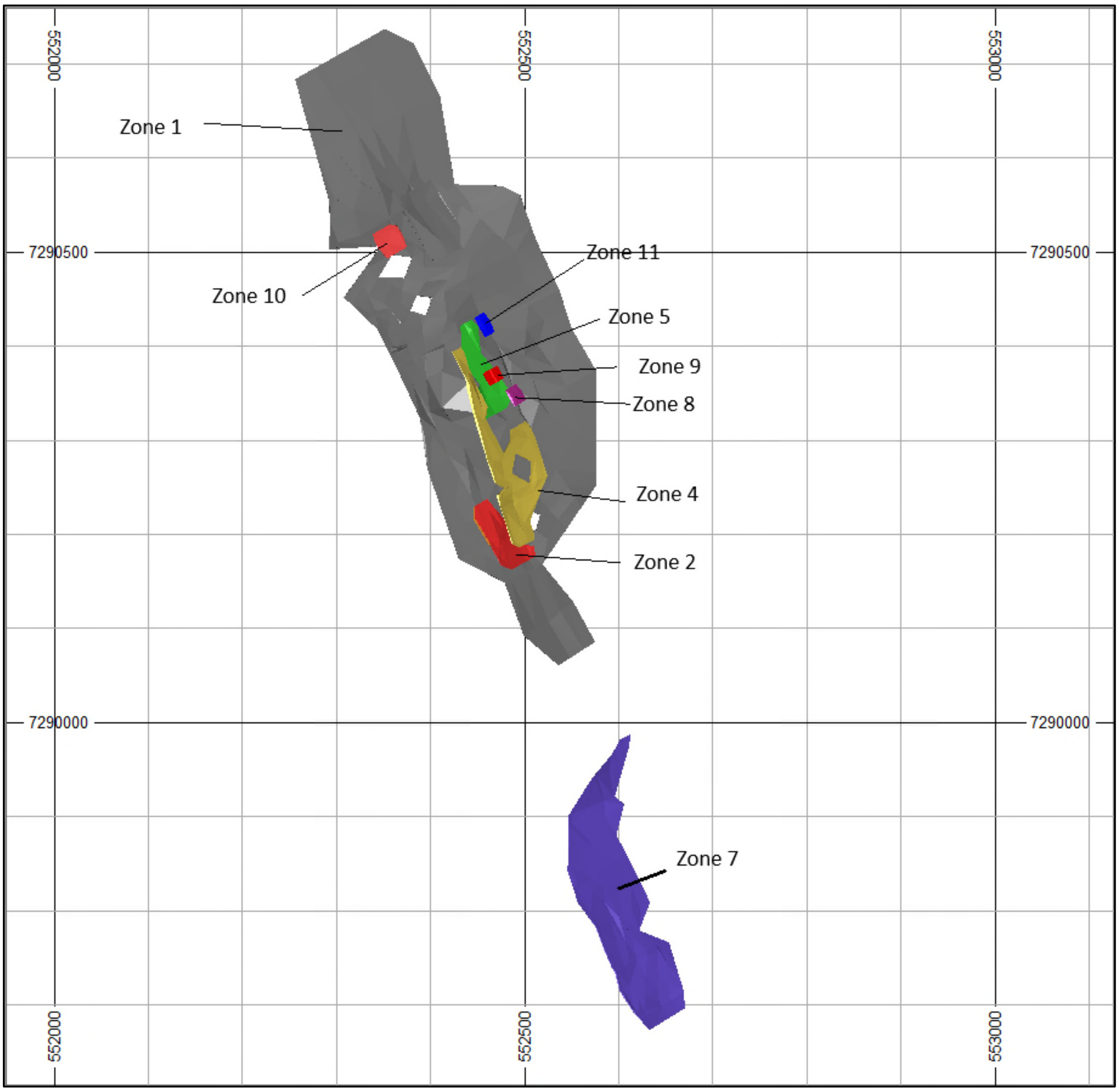


Figure 13.24: Mineralized Zone Wireframe Models for Northern Mangazeisky. Some zones are below Zone 1

Modeling made it clear that the location of drillhole collars and/or deviation survey data need to be refined for some close drillholes since there is an abrupt change in the mineralized occurrence at a relatively short distance (Figure 13.25)

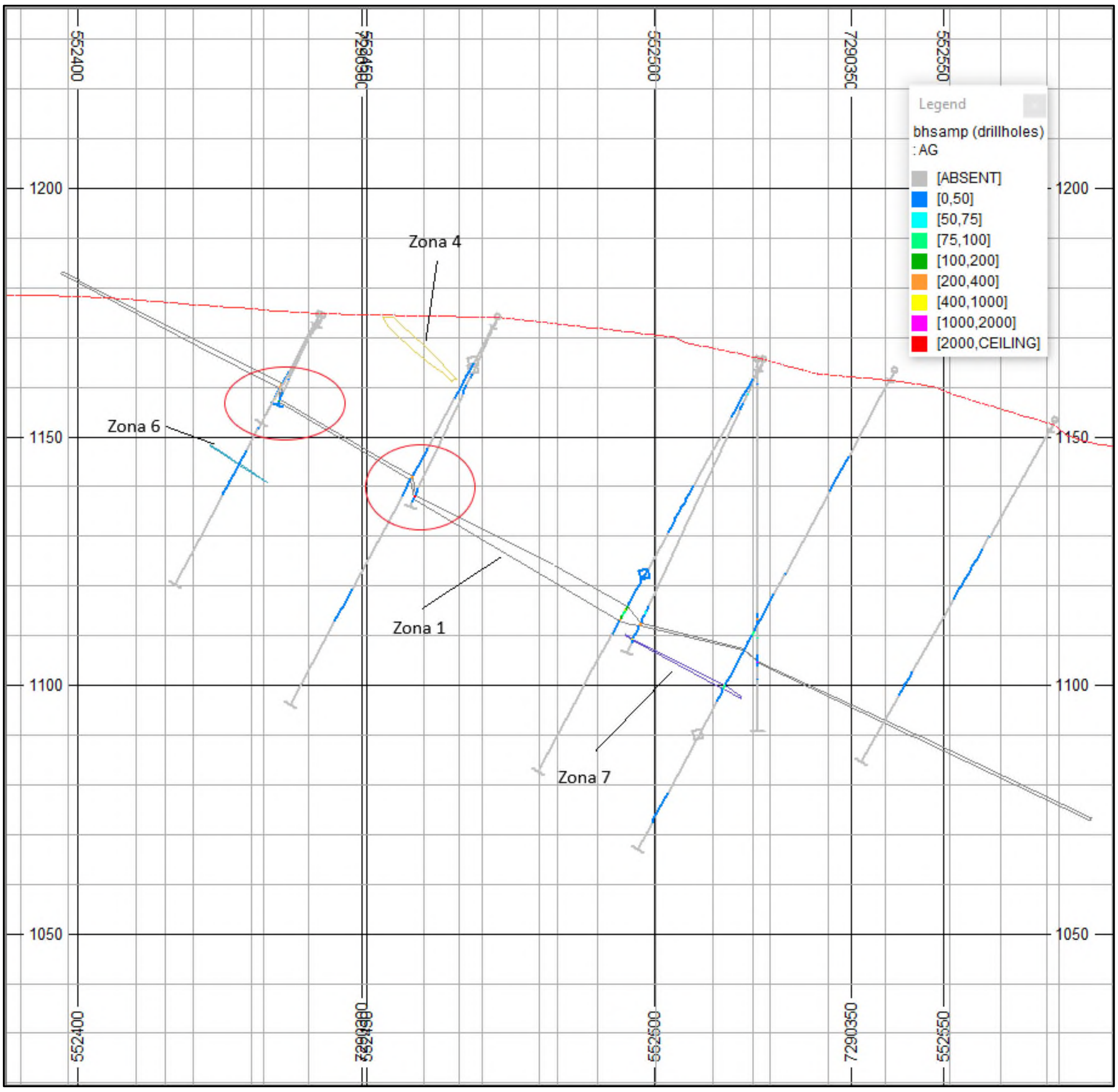


Figure 13.25: Section crossing the Mineralized Wireframes. Highlighted areas with abrupt changes in the mineralized occurrence in the near holes

13.2.5 Statistical Analysis and Variogram Modelling

13.2.5.1 General Statistics

WAI has coded individual wireframes for different zones and completed a general statistical analysis on the number of drillholes/trenches, samples, and composites for individual zones as summarised in Table 13.28. The average length of the samples is 0.58m therefore the composite length of 1.0m was chosen for North Mangazeysky (Table 13.28).

Table 13.28: Statistical Data for Individual Wireframe Zone

Zone	Exploration type	Number of			Total, m	Average length, m
		Drill holes/trenches	samples	composites		
1	Trenches	10	30	26	24.90	0.83
1	Drill holes	80	207	143	109.95	0.53
2	Trenches	4	13	8	7.45	0.57
2	Drill holes	6	8	7	3.70	0.46
3	Drill holes	3	4	4	2.85	0.71
4	Drill holes	5	10	8	6.55	0.66
4	Trenches	12	41	30	26.40	0.64
5	Drill holes	6	11	8	6.00	0.55
6	Drill holes	2	3	2	0.90	0.30
7	Drill holes	2	2	2	1.20	0.60
8	Trenches	1	4	4	4.00	1.00
9	Drill holes	2	3	3	2.50	0.83
10	Drill holes	1	2	2	1.60	0.80
11	Drill holes	1	2	2	2.00	1.00
12	Drill holes	1	2	1	0.40	0.20
13	Drill holes	2	2	2	0.60	0.30
14	Drill holes	7	17	13	10.25	0.60
15	Drill holes	1	3	2	2.30	0.77
16	Drill holes	1	2	1	0.30	0.15
17	Trenches	11	24	17	15.10	0.63
17	Drill holes	20	30	24	15.13	0.50
Trenches total		**38**	**112**	**85**	**77.85**	**0.70**
Drill holes total		**140**	**308**	**224**	**166.23**	**0.54**
Trenches/drill holes total		**178**	**420**	**309**	**244.08**	**0.58**
- the total number of drill holes/trenches is 207, however, some of these did not hit mineralisation, and some of the drill holes/trenches intersect more than one zone.						

The general statistics for composites within mineralised wireframes are presented in Table 13.29. The average copper grade is very low and is close to the detection limit of most of the analytical methods. The average lead grade is significantly higher than at Vertikalnoye, while the average zinc grade is more than two times lower than at Vertikalnoye.

Table 13.29: General Statistics for Composites Inside Wireframe

Metal	Composite No.	Minimum	Maximum	Count	Mean	Variance	Standard Deviation	Standard Error	Variance factor
Ag	309	0.0005	3,410.00	191,293.73	619.07	516,866.31	718.93	40.90	233%
Pb	309	0.005	48.02	1,598.73	5.17	66.02	8.13	0.46	3%
Zn	309	0.00185	37.06	200.14	0.65	14.47	3.80	0.22	1%
Cu	309	0	0.09	3.84	0.01	0.00021	0.01	0.0008	0%

Statistical parameters for the main metals (Ag, Pb and Zn) within the wireframes of individual mineralised zones are given in Table 13.30.

Table 13.30: Statistical Parameters for Composites within Individual Zones

Zone	Metal	Composite No.	Minimum	Maximum	Mean	Variance	Standard Deviation	Standard Error	Variance factor
1	AG	169	0.0005	3,410.00	653.79	570,470.41	755.29	58.10	116%
1	PB	169	0.005	48.02	6.77	87.46	9.35	0.72	138%
1	ZN	169	0.002	32.05	0.62	12.19	3.49	0.27	564%
2	AG	15	61	1,670.24	511.54	266,482.98	516.22	133.29	101%
2	PB	15	0.027	8.43	1.80	8.82	2.97	0.77	165%
2	ZN	15	0.01	1.22	0.20	0.09	0.30	0.08	152%
3	AG	4	80.9	419.00	219.23	15,248.10	123.48	61.74	56%
3	PB	4	1.314	8.81	4.37	7.66	2.77	1.38	63%
3	ZN	4	0.085	0.17	0.13	0.00	0.03	0.02	27%
4	AG	38	42.28	2,166.00	539.06	285,648.67	534.46	86.70	99%
4	PB	38	0.01	4.29	0.48	1.01	1.00	0.16	208%
4	ZN	38	0.00185	0.33	0.05	0.00	0.07	0.01	148%
5	AG	8	98.7	803.35	303.73	48,162.31	219.46	77.59	72%
5	PB	8	0.06	12.46	3.97	12.66	3.56	1.26	90%
5	ZN	8	0.039	0.31	0.15	0.01	0.09	0.03	60%
6	AG	2	1360	2,698.00	2 029.00	447,561.00	669.00	473.05	33%
6	PB	2	21.746	30.00	25.87	17.03	4.13	2.92	16%
6	ZN	2	0.602	0.75	0.67	0.01	0.07	0.05	11%
7	AG	2	97.2	771.00	434.10	113,501.61	336.90	238.22	78%
7	PB	2	1.264	23.60	12.43	124.72	11.17	7.90	90%
7	ZN	2	0.085	0.09	0.09	0.00001	0.00350	0.00247	4%
8	AG	4	79.6	334.00	235.65	9,868.37	99.34	49.67	42%
8	PB	4	0.1	0.10	0.10				
8	ZN	4	0.01	0.05	0.02	0.00	0.02	0.01	71%
9	AG	3	79.6	144.20	108.93	713.13	26.70	15.42	25%
9	PB	3	1.724	4.99	3.34	1.78	1.33	0.77	40%
9	ZN	3	0.096	0.13	0.11	0.00	0.02	0.01	13%
10	AG	2	61.5	274.65	168.08	11,358.23	106.58	75.36	63%
10	PB	2	0.266	5.52	2.89	6.89	2.63	1.86	91%
10	ZN	2	0.2148	0.22	0.22	0.00	0.00	0.00	1%
11	AG	2	62.5	3,150.00	1,606.25	2,383,164.06	1,543.75	1,091.60	96%
11	PB	2	1.238	21.82	11.53	105.85	10.29	7.28	89%
11	ZN	2	0.111	0.68	0.40	0.08	0.28	0.20	72%
12	AG	1	237.6	237.60	237.60				
12	PB	1	1.2	1.20	1.20				
12	ZN	1	0.28	0.28	0.28				
13	AG	2	77	2,380.90	1,228.95	1,326,988.80	1,151.95	814.55	94%
13	PB	2	0.08	1.70	0.89	0.66	0.81	0.57	91%
13	ZN	2	0.366	37.06	18.71	336.61	18.35	12.97	98%
14	AG	13	109	1,619.20	472.68	186,468.96	431.82	119.77	91%
14	PB	13	0.22	22.50	6.78	34.99	5.92	1.64	87%
14	ZN	13	0.0678	32.98	2.74	76.22	8.73	2.42	318%
15	AG	2	140	1,332.50	736.25	355,514.06	596.25	421.61	81%
15	PB	2	0.229	13.28	6.76	42.61	6.53	4.62	97%
15	ZN	2	0.1104	0.12	0.12	0.00	0.01	0.00	5%
16	AG	1	1968	1,968.00	1,968.00				
16	PB	1	15.169	15.17	15.17				
16	ZN	1	0.469	0.47	0.47				
17	AG	41	55.9	3,035.20	666.20	569,064.77	754.36	117.81	113%
17	PB	41	0.01	30.00	3.03	30.84	5.55	0.87	183%
17	ZN	41	0.007	2.05	0.29	0.16	0.40	0.06	139%

13.2.5.2 Comparison of Statistics between Drill Holes and Trenches

Grade statistics for main metals for both drill holes and trenches are given in Table 13.31 below. The average silver grade for trenches is higher than that for the drill holes. However, the number of composites in drill hole is almost three times higher than in trenches. At the same time, the maximum silver grades for both trenches and drill holes are comparable. The average grades of lead and zinc are significantly lower in trenches than in drill holes and maximum grades of these metals in trenches have very low values.

Table 13.31: Statistics of Composites separately for Drill Holes and Trenches

Drill hole/trench	Metal	No. of composites	Grade		
			Min	Max	Av
Drill holes	AG	224	2.10	3,410.00	550.06
Drill holes	PB	224	0.01	48.02	7.10
Drill holes	ZN	224	0.016	37.06	0.89
Trenches	AG	85	0.0005	3,283.00	800.93
Trenches	PB	85	0.005	0.10	0.09
Trenches	ZN	85	0.002	0.07	0.02

13.2.5.3 Top Cutting

Simalarly to Vertikalny, the need for top cutting and top cut values in the composites were analyzed for individual zones of North Mangazeysky using a quantile / decile analysis of grades and probability plots.

It should be noted that only three zones (Zones 1, 4 and 17) had populations of composites where their total number exceeded 30 values and top cut analysis was undertaken for these three zones only. The populations for other mineralised zones are insufficient for such analysis.

Table 13.32 shows a quantile analysis of silver grades for zones 1, 4 and 17. Each of the percentiles in the upper quantile (90-100%) for Zone 1 contains less than 6% of accumulated metal. Upper quantile for Zone 4 contains only 4 samples, where the accumulated metal contained is close to 30%. In WAI opinion, top cutting is not required for these zones.

The estimated limit value for silver for Zone 17 is 2,000g/t, however, these samples are spatially located in the near-surface area and are concentrated in one area (Figure 13.26). WAI believes that this can indicate the peculiarities of the developed mineralization in Zone 17 and no top cutting is required.

Examples of probability plots for Zones 1, 4 and 17 for silver are shown in Figure 13.27, Figure 13.28, and Figure 13.29.

Table 13.32: Quantile Analysis of Silver Grades for Individual Zones

Zone	Q%_from	Q%_to	Qty of samples	Ave	Min	Max	Accumulated metal	Accumulated metal (%)
1	0	10	17	45.5	2.1	71.1	773.57	0.69
1	10	20	17	88.13	74.9	105.06	1 498.27	1.34
1	20	30	17	137.24	106	167.5	2 333.13	2.09
1	30	40	17	198.37	170.17	230	3 372.30	3.02
1	40	50	17	265.75	241.33	303	4 517.83	4.04
1	50	60	17	390.55	326	463.72	6 639.35	5.94
1	60	70	17	551.12	464.2	665	9 369.05	8.38
1	70	80	17	960.61	703.7	1 195.50	16 330.36	14.61
1	80	90	17	1 389.22	1 197.35	1 650.00	23 616.77	21.13
1	90	100	18	2 408.00	1 653.00	3 410.00	43 344.00	38.77
1	90	91	1	1 653.00	1 653.00	1 653.00	1 653.00	1.48
1	91	92	2	1 747.60	1 745.20	1 750.00	3 495.20	3.13
1	92	93	2	1 888.00	1 880.00	1 896.00	3 776.00	3.38
1	93	94	2	2 089.60	2 010.00	2 169.20	4 179.20	3.74
1	94	95	2	2 293.50	2 262.00	2 325.00	4 587.00	4.1
1	95	96	1	2 450.00	2 450.00	2 450.00	2 450.00	2.19
1	96	97	2	2 517.50	2 455.00	2 580.00	5 035.00	4.5
1	97	98	2	2 620.30	2 620.00	2 620.60	5 240.60	4.69
1	98	99	2	3 117.50	3 015.00	3 220.00	6 235.00	5.58
1	99	100	2	3 346.50	3 283.00	3 410.00	6 693.00	5.99
1	0	100	171	653.77	2.1	3 410.00	111 794.63	100
4	0	10	3	53.73	42.28	65.7	161.18	0.85
4	10	20	4	101.42	80.6	111	405.67	2.13
4	20	30	4	119.69	112.5	123.8	478.76	2.51
4	30	40	4	146.85	126	180.4	587.4	3.08
4	40	50	4	221.74	186	268.54	886.94	4.65
4	50	60	3	331.43	271	368.28	994.28	5.21
4	60	70	4	403.9	370	444.8	1 615.60	8.47
4	70	80	4	724.38	484	926.44	2 897.52	15.2
4	80	90	4	1 134.25	952.5	1 231.00	4 537.01	23.79
4	90	100	4	1 625.77	1 364.00	2 166.00	6 503.06	34.11
4	92	93	1	1 364.00	1 364.00	1 364.00	1 364.00	7.15
4	94	95	1	1 461.26	1 461.26	1 461.26	1 461.26	7.66
4	97	98	1	1 511.80	1 511.80	1 511.80	1 511.80	7.93
4	99	100	1	2 166.00	2 166.00	2 166.00	2 166.00	11.36
4	0	100	38	501.77	42.28	2 166.00	19 067.42	100
17	0	10	4	60.37	55.9	62.88	241.48	0.88
17	10	20	4	81.49	65.44	91	325.94	1.19
17	20	30	4	126.5	115	146	506	1.85
17	30	40	4	197.42	190	213.68	789.68	2.89
17	40	50	4	278.97	221.87	314	1 115.87	4.09
17	50	60	4	359.82	320	385	1 439.28	5.27
17	60	70	4	533.25	398	591	2 133.00	7.81
17	70	80	4	779.5	696.8	874	3 118.00	11.42
17	80	90	4	1 540.90	1 155.60	1 800.00	6 163.60	22.57
17	90	100	5	2 296.26	1 999.10	3 035.20	11 481.30	42.03
17	91	92	1	1 999.10	1 999.10	1 999.10	1 999.10	7.32
17	93	94	1	2 061.50	2 061.50	2 061.50	2 061.50	7.55
17	95	96	1	2 090.50	2 090.50	2 090.50	2 090.50	7.65
17	97	98	1	2 295.00	2 295.00	2 295.00	2 295.00	8.4
17	99	100	1	3 035.20	3 035.20	3 035.20	3 035.20	11.11
17	0	100	41	666.2	55.9	3 035.20	27 314.15	100

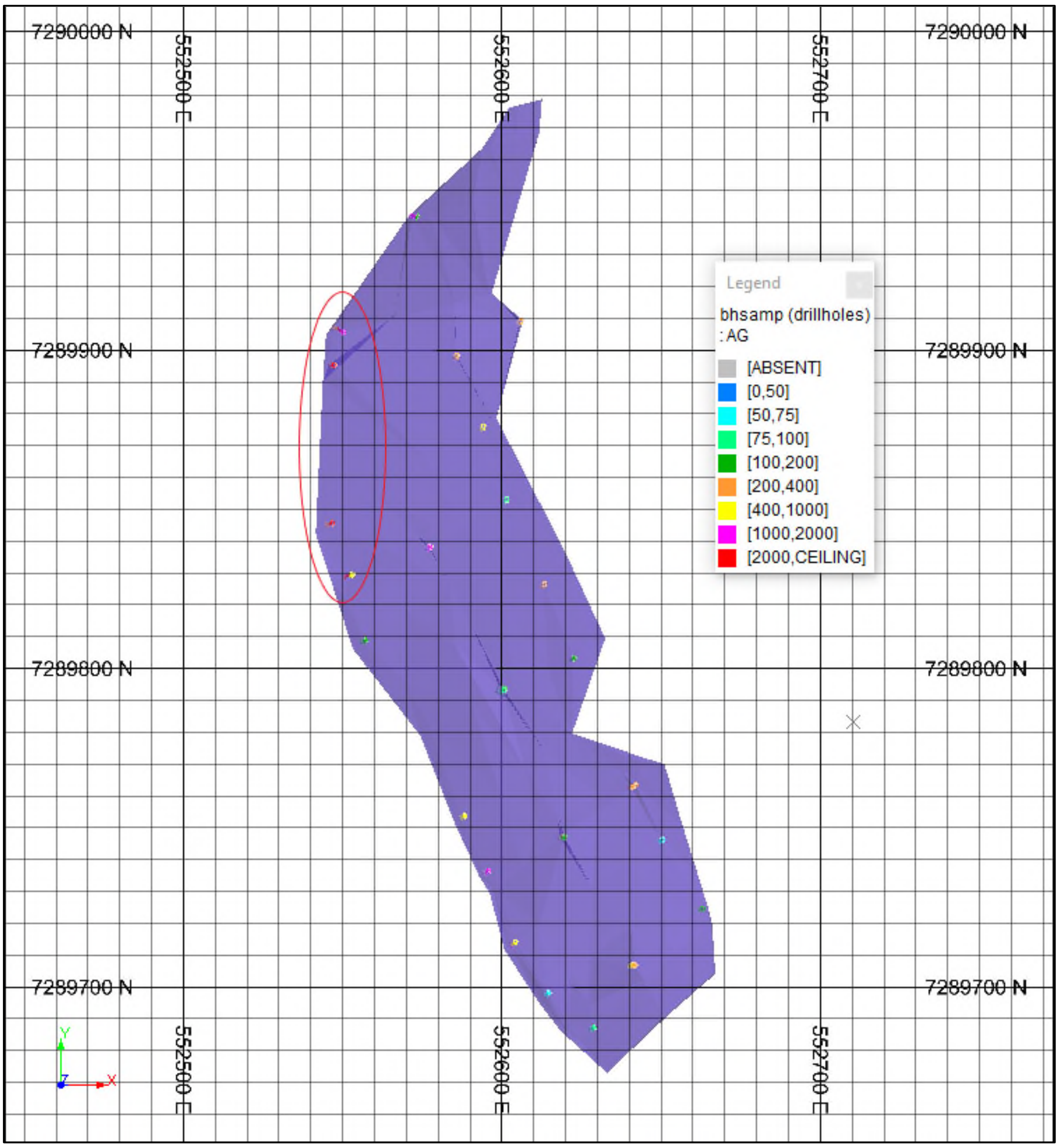


Figure 13.26: Trench Location with High Grade of Silver, Zone 17.

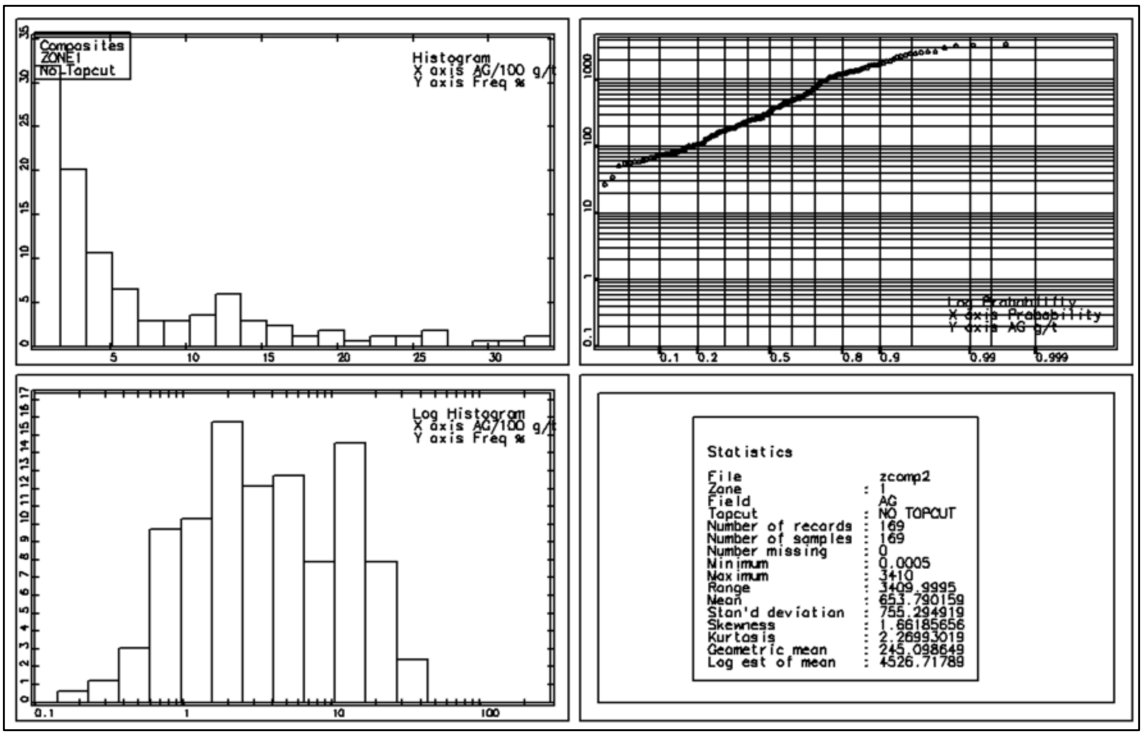


Figure 13.27: Statistical Plots for Silver, Zone 1

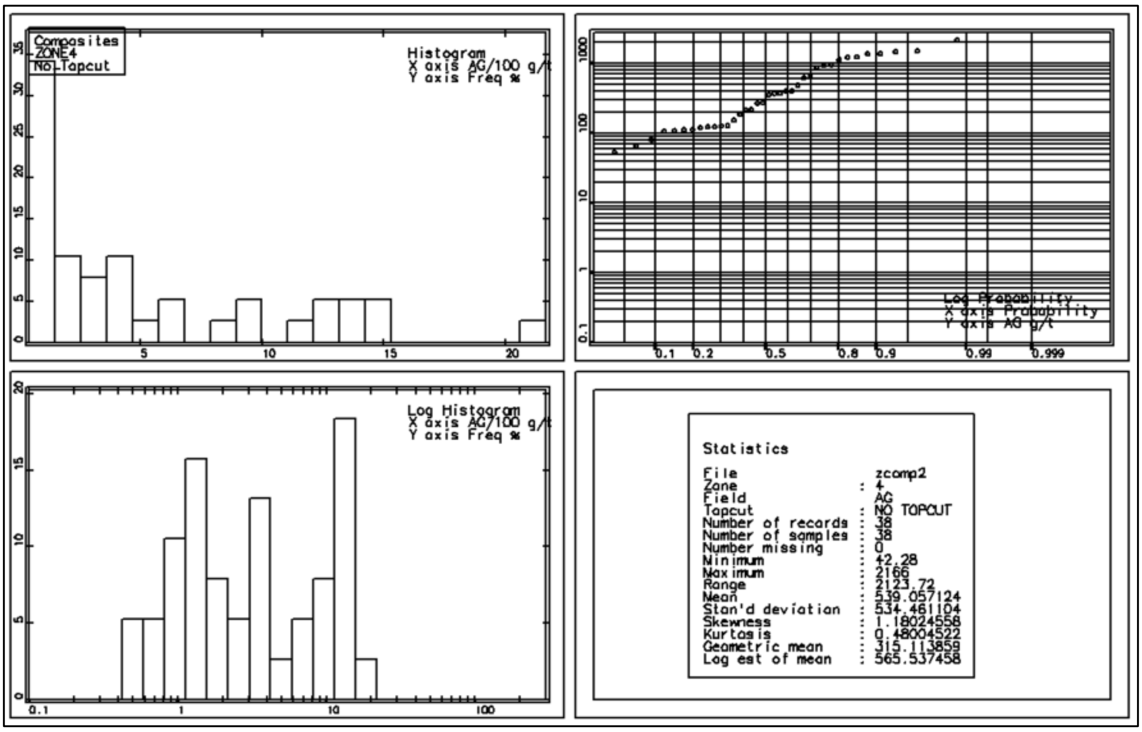


Figure 13.28: Statistical Plots for Silver, Zone 4

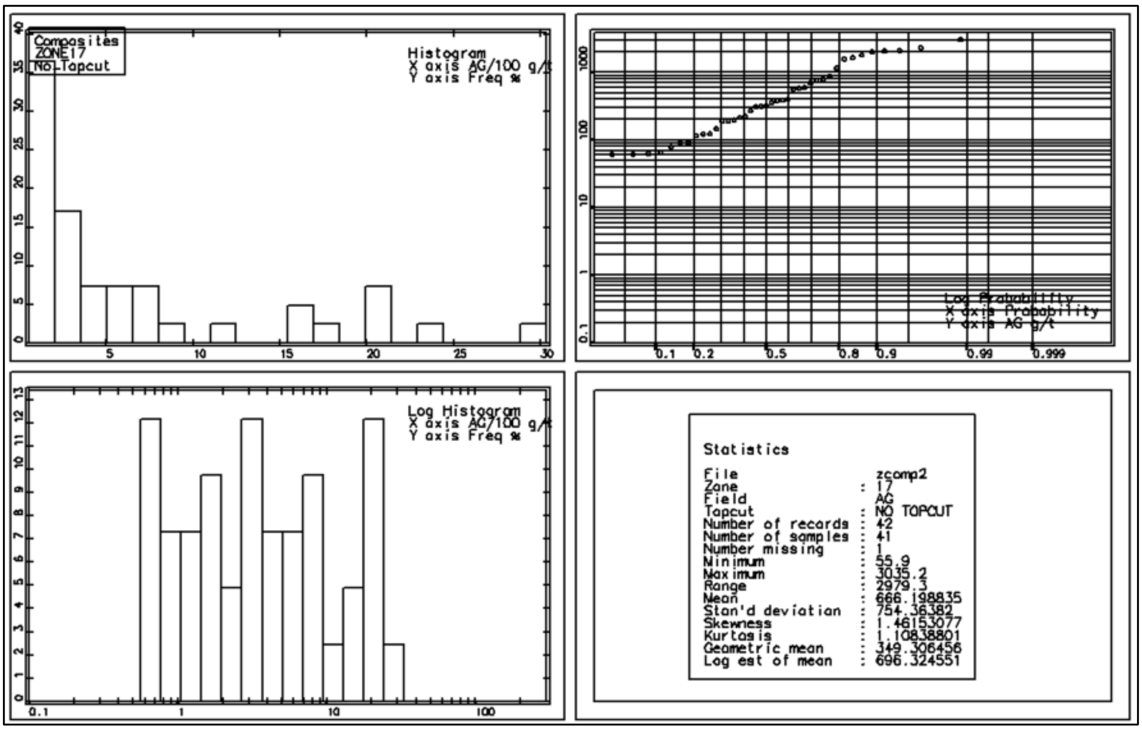


Figure 13.29: Statistical Plots for Silver, Zone 17

13.2.5.4 Variogram Modelling

The top-cut composites were used for modelling of experimental semi-variograms. Robust variogram models were produced for silver for Zone 1 which is the largest zone at North Mangazeisky. Due to a low number of composites, and/or their irregular spacing, it was impossible to model robust variograms for the remainder of the zones and metals. An example of the along strike, down-dip and across the strike modelled variogram for silver for Zone 1 is shown in Figure 13.30, Figure 13.31 and Figure 13.32. The parameters of the modelled variograms are presented in Table 13.33.

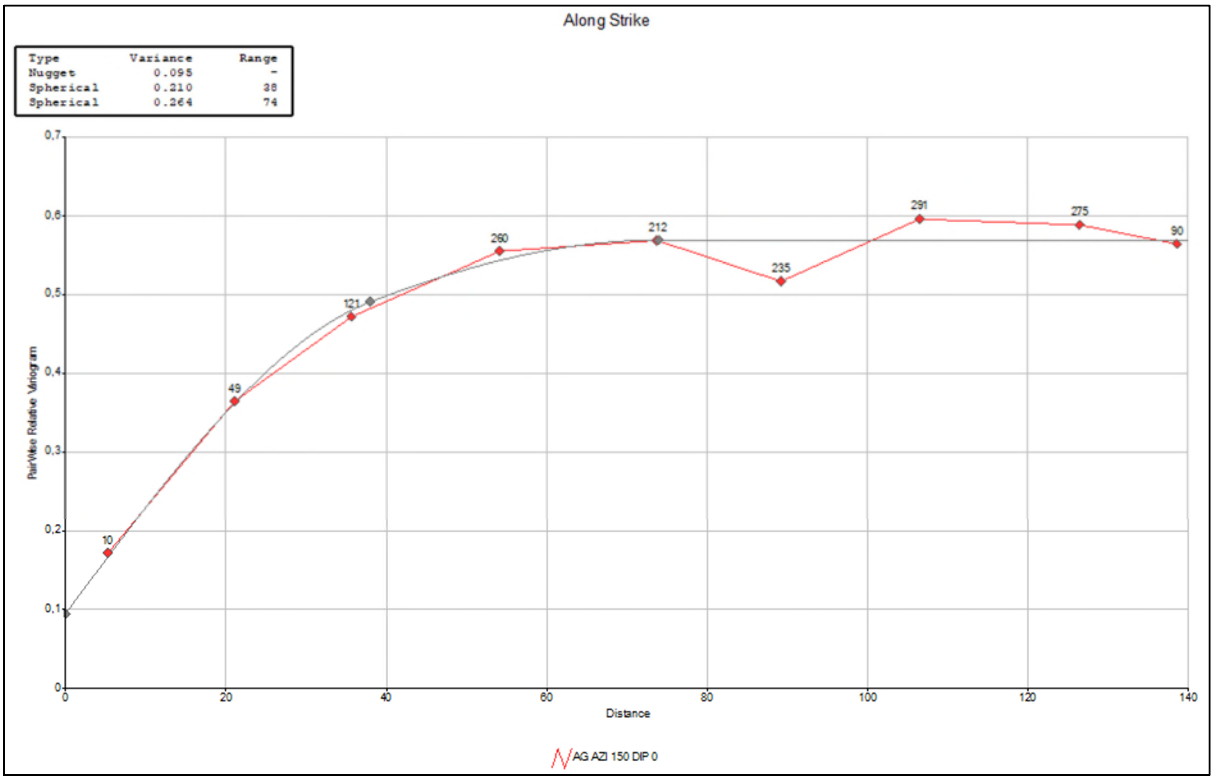


Figure 13.30: Ag Modelled Variogram, Zone 1, Along the Strike

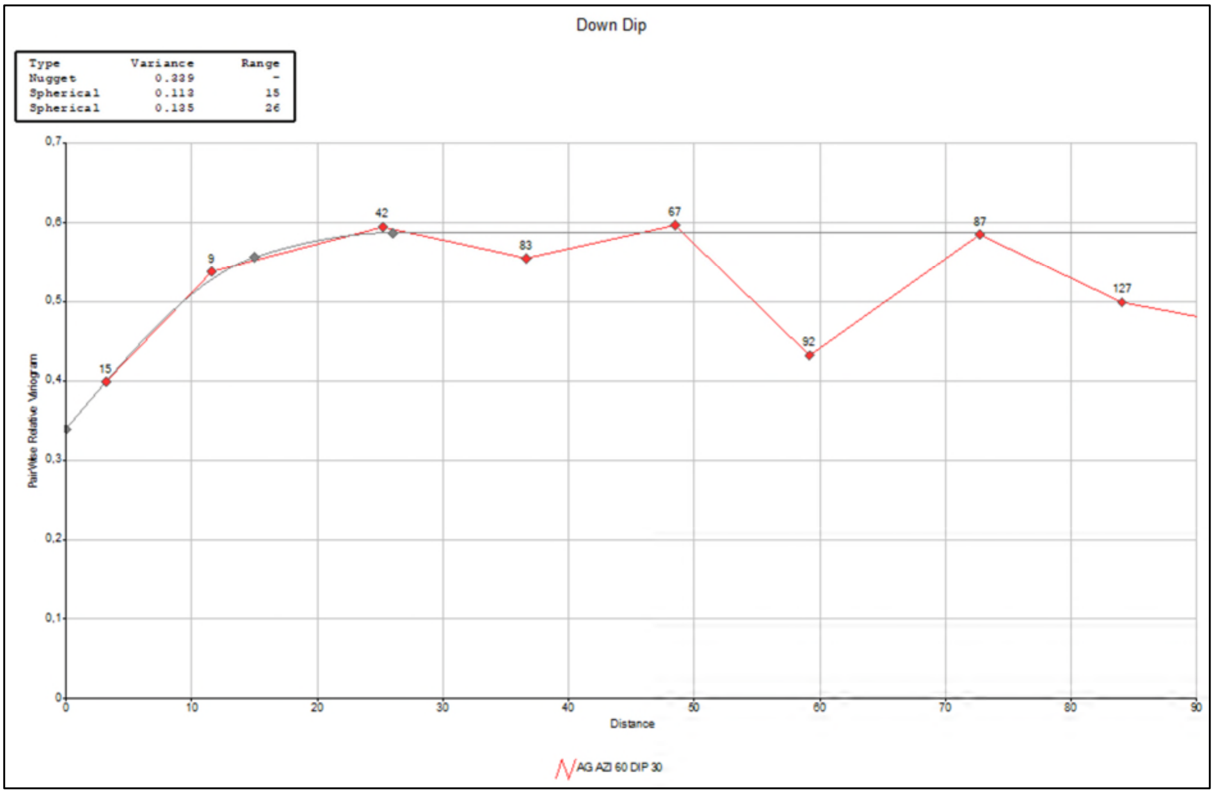


Figure 13.31: Ag Modelled Variogram, Zone 1, Down-Dip

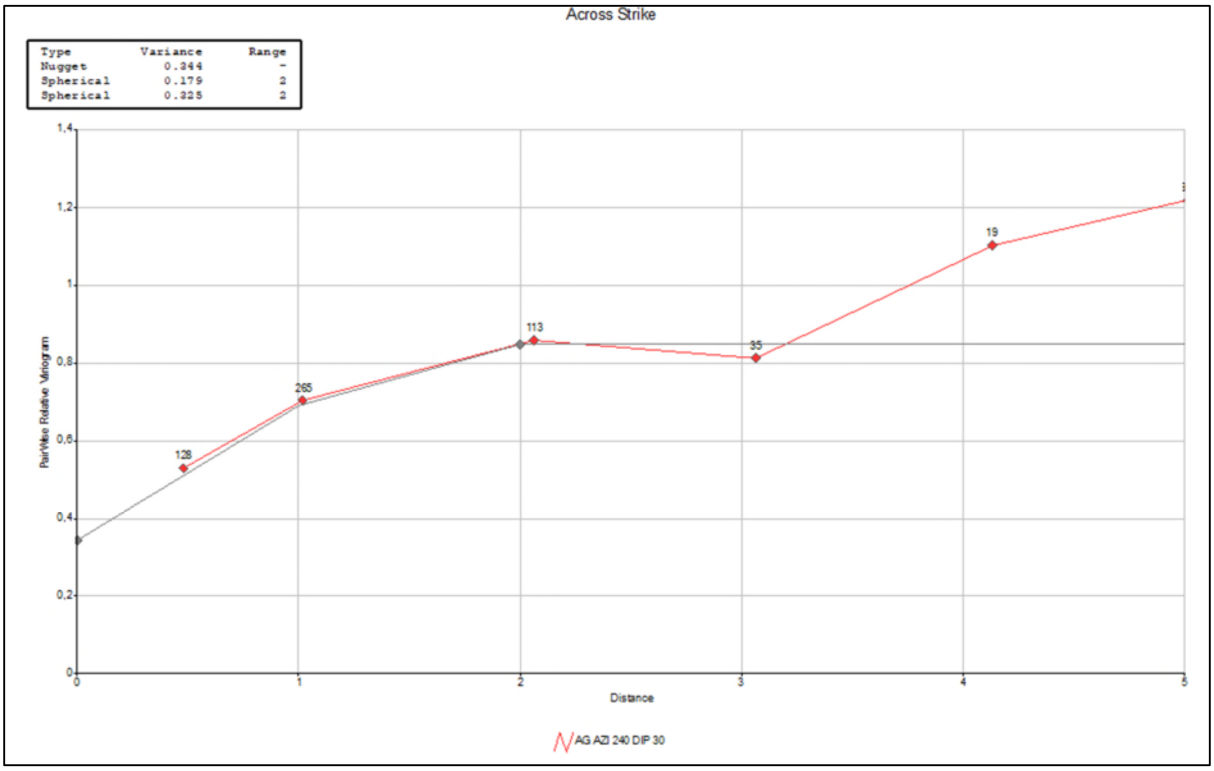


Figure 13.32: Ag Modelled Variogram, Zone 1, Aross the Strike

Table 13.33: Parameters of Modelled Variograms for Silver, Zone 1

Parameters	Along the Strike	Down-Dip	Across the Strike
Zone	1	1	1
File	z1wcomp1	z1wcomp1	z1tcomp1
Lag	18	12	1
Nlag	8	6	6
HorAng	60	50	40
VerAng	60	50	40
CylRad	100	80	50
Ang1	150	60	240
Ax1	3	3	3
Ang2		30	30
Ax2		1	1
VarType	RV	RV	RV
MoRefNo	1	4	3
Nugget	0.095	0.339	0.344
R1	38.1	15.3	1.7
C1	0.21	0.113	0.179
S1	0.305	0.452	0.524
R2	74	26.1	2.1
C2	0.264	0.135	0.325
S2	0.569	0.587	0.849

13.2.6 Block Modelling

13.2.6.1 Block Model Prototype

The block model was constructed using Datamine with a parent cell size of 10m x 10m x 10m (along strike, across strike and vertical), sub-celling was allowed down to 1.0m x 1.0m x 2.0m. The block model was created within the individual zone wireframes. The block model also reflects the DTM surface, also outlines the blocks corresponding to unconsolidated sediments overlaying the bedrock. No rotation has been applied to the model. A summary of the parameters used in the model prototype is shown in Table 13.34. The block model relative to the surface with outlined oxide and primary mineralization is shown in Figure 13.33.

<table>
<tr><th colspan="4">Table 13.34: Block Model Prototype</th></tr>
<tr><th colspan="2">Parameters</th><th>Direction</th><th>Size</th></tr>
<tr><td colspan="2" rowspan="3">Model Origin</td><td>X</td><td>552,065</td></tr>
<tr><td>Y</td><td>7,289,495</td></tr>
<tr><td>Z</td><td>1,062</td></tr>
<tr><td rowspan="6">Model Parameters</td><td rowspan="3">Parent Block Size</td><td>X</td><td>10</td></tr>
<tr><td>Y</td><td>10</td></tr>
<tr><td>Z</td><td>10</td></tr>
<tr><td rowspan="3">Number of Blocks</td><td>X</td><td>86</td></tr>
<tr><td>Y</td><td>137</td></tr>
<tr><td>Z</td><td>16</td></tr>
</table>

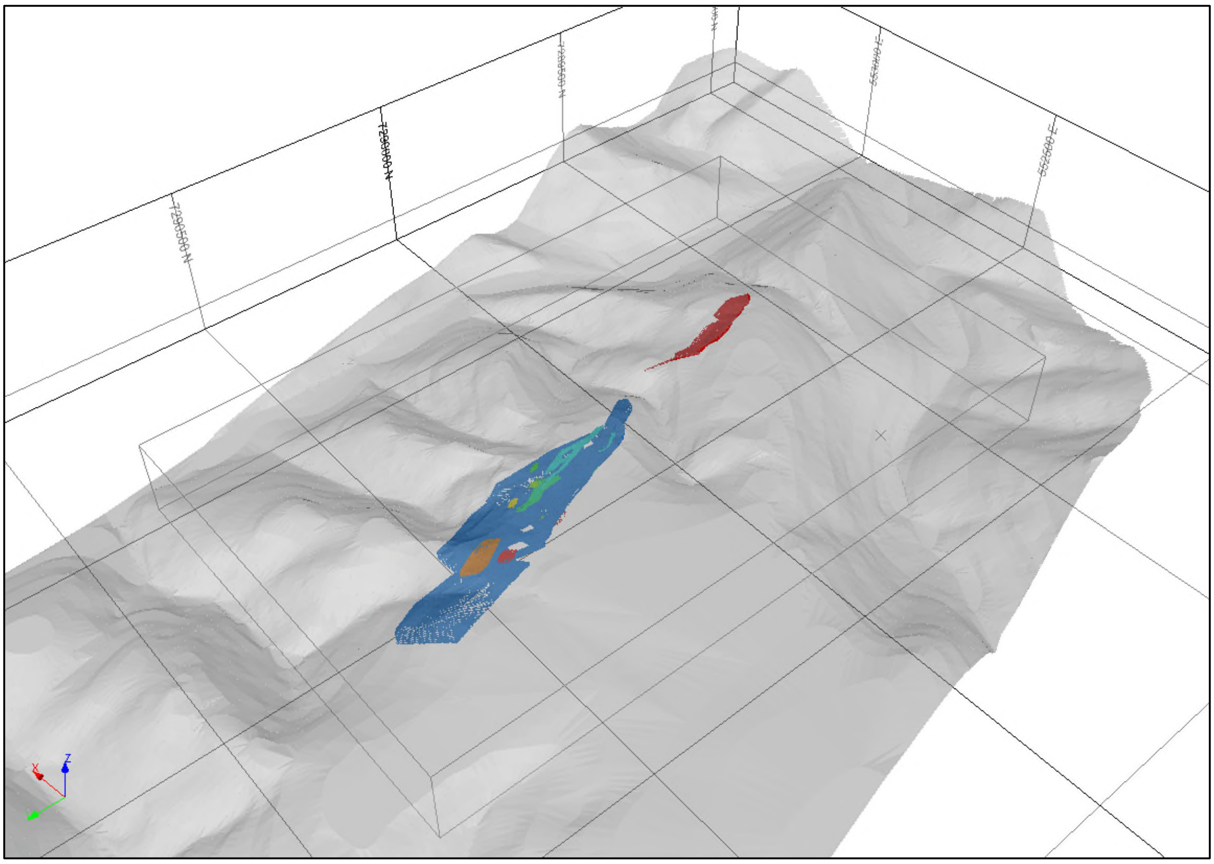


Figure 13.33: Block Model of Northern Mangazeisky Mineralization Relative to Surface

13.2.6.2 Dynamic Anisotropy

Parameters of dynamic anisotropy showing the true dip angle and azimuth were interpolated into the blocks of each individual zone of mineralization. In order to produce the points with true dip angle and azimuth WAI modelled wireframes corresponding with the axial surfaces of mineralized zones. Points with true dip angles and azimuth corresponded with the centers of triangles of these wireframes.

An example of location of points of dynamic anisotropy for Zone 1 is shown in Figure 13.34.

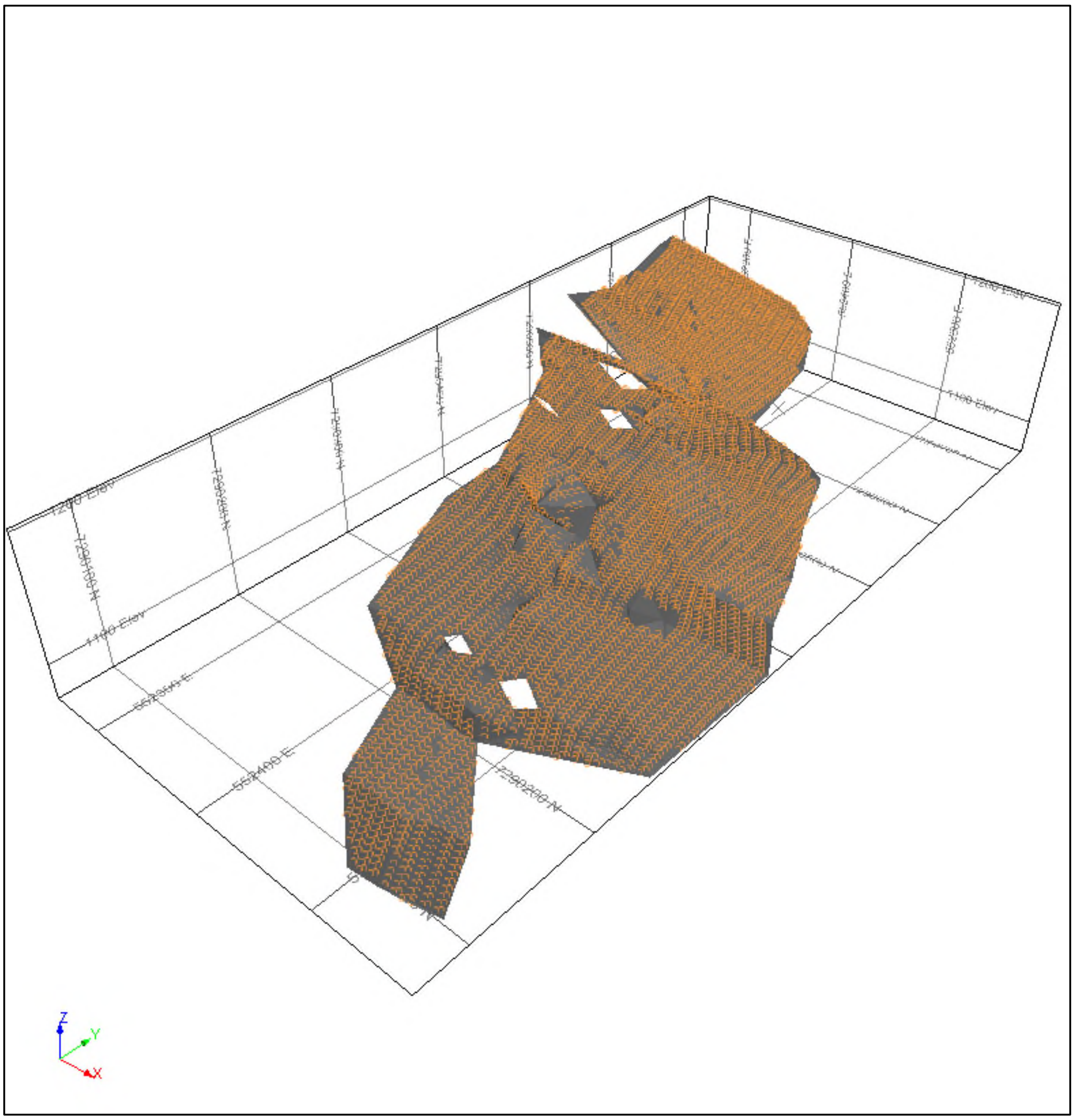


Figure 13.34: Wireframe Model of Zone 1 with Points Used to Determine Dynamic Anisotropy

13.2.6.3 Density

CJSC Prognoz provided the density data on rocks and ores determined from the drillcore of 2014. A total of 68 samples from 33 drillholes was taken. The summary data on the density determination for individual zones of mineralization and host rocks are given in Table 13.35.

Table 13.35: Data to Determine Density		
Zone	Number of samples	Average, t/m^3
1	35	3.54
2	3	2.66
5	2	2.61
6	1	2.73
10	1	3.36
14	4	3.19
15	1	2.69
16	1	2.64
Total for the mineralized material	**48**	**2.93**
Host rocks	**20**	**2.70**

The largest number of samples used to determine the density value was in Zone 1, the average density was $3.54t/m^3$. Given that there is not enough data to identify the zone of oxidation, all mineralization in North Mangazeysky was assigned to primary ore. The final density values for the estimation of mineral resources were accepted by analogy with the Vertikalny deposit and amounted to:

- All mineralization (without division into primary and oxide ores) – $3.56t/m^3$
- Host rocks – $2.75t/m^3$

13.2.6.4 Grade Interpolation

WAI has used Ordinary Kriging (OK) as the principal interpolation method and Inverse Power Distance Cubed (IPD3) as the secondary method for silver, lead, and zinc. Zonal control and dynamic anisotropy were used for grade interpolation. Eight estimation passes were run with each one using a consecutively larger ellipsoid to ensure that all blocks were estimated.

The grade interpolation plan is presented in Table 13.36.

Table 13.36: Plan of Grade Interpolation	
Run 1 (strike x downdip x cross-strike)	1/3 x 1/3 x 1/3 radii
Run 2 (strike x downdip x cross-strike)	1 x 1 x 1 radii
Run 3 (strike x downdip x cross-strike)	2 x 2 x 2 radii
Run 4 (strike x downdip x cross-strike)	4 x 4 x 4 radii
Run 5 (strike x downdip x cross-strike)	6 x 6 x 6 radii
Run 6 (strike x downdip x cross-strike)	8 x 8 x 8 radii
Run 7 (strike x downdip x cross-strike)	16 x 16 x 16 radii
Run 8 (strike x downdip x cross-strike)	30 x 30 x 30 radii
Min comp no (run 1/2/3/4/5/6/7/8/9)	2/2/2/2/2/2/1/1
Max comp no (run 1/2/3/4/5/6/7/8/9)	8/8/8/8/8/8/15/15
Min Octan no (run 1/2/3/4/5/6)	2/2/2/2/2/1/1/1
Max comp no from 1 hole	4/4/4/4/4/4/4/4
Note –	
1) Dynamic Anisotropy used for search ellipsoid orientation	

The size of search ellipsoid for silver in Zone 1 was used for all metals and zones as shown in (Table 13.37).

Table 13.37: Search Ellipsoid				
Metal	Zone	Radii, m		
		Along the Strike	Down-Dip	Across the Strike
All	All	74	26.1	2.1

13.2.6.5 Model Validation

Following grade estimation, a statistical and visual assessment of the block model was undertaken:

1. To assess successful application of the estimation passes;
2. To ensure that as far as the data allowed, all blocks within mineralisation domains were estimated; and
3. To ensure the model estimates performed as expected.

The model validation methods carried out included global statistical grade validation, a visual assessment of grades, and swath plot (model grade profile) analysis.

i) Statistical Comparison

Statistical analysis of the block model was carried out to compare the interpolation results against composite and initial sample data. This analysis provides a check on the reproducibility of the mean grade of the composite and initial sample data against the model over individual mineralized zones. Typically, the mean grade of the block model should not be significantly greater/lower than that of the composites from which it has been derived.

WAI has carried out a comparison between interpolated grades in the block model (BM), grade in the initial samples, and 1.0m composites used for interpolation. Global comparison was only undertaken for silver grades for each individual zone (Table 13.38).

Table 13.38: Global Comparison of Ag Grades in Block Model, Samples, and Composites for Individual Mineralized Zones within Wireframes

Zone	Volume (,000m^3)	Tonnes (Kt)	Qty of composites	Average Ag Grade, g/t		
				Sample	Composite	Block Model
1	106.38	378.71	171	614.21	609.40	636.54
2	1.27	4.53	15	657.91	633.87	378.38
3	0.75	2.67	4	232.93	232.93	218.58
4	6.42	22.87	38	471.20	476.75	326.26
5	2.51	8.92	8	321.62	328.82	284.16
6	0.25	0.89	2	1,799.56	1,742.29	1,961.68
7	0.32	1.15	2	265.65	265.65	367.34
8	0.48	1.71	4	235.65	235.65	234.89
9	0.28	0.98	3	109.18	110.42	100.99
10	0.60	2.12	2	194.72	194.72	168.74
11	0.28	1.01	2	1,606.25	1,606.25	1,604.67
12	0.03	0.10	1	237.60	237.60	237.60
13	0.03	0.10	2	1,228.95	1,228.95	1,101.26
14	3.70	13.17	13	428.98	428.98	502.99
15	0.58	2.08	2	736.74	736.25	734.45
16	0.26	0.92	1	1,968.00	1,968.00	1,968.00
17	9.89	35.20	41	728.55	722.77	536.94

ii) Visual Comparison

A visual comparison of composite grades and block grade was completed in cross section and in plan. An example of visual comparison of silver grade in the block model and composite within drillholes is presented in Figure 13.35. Visually the model was generally considered to reflect the composite grades.

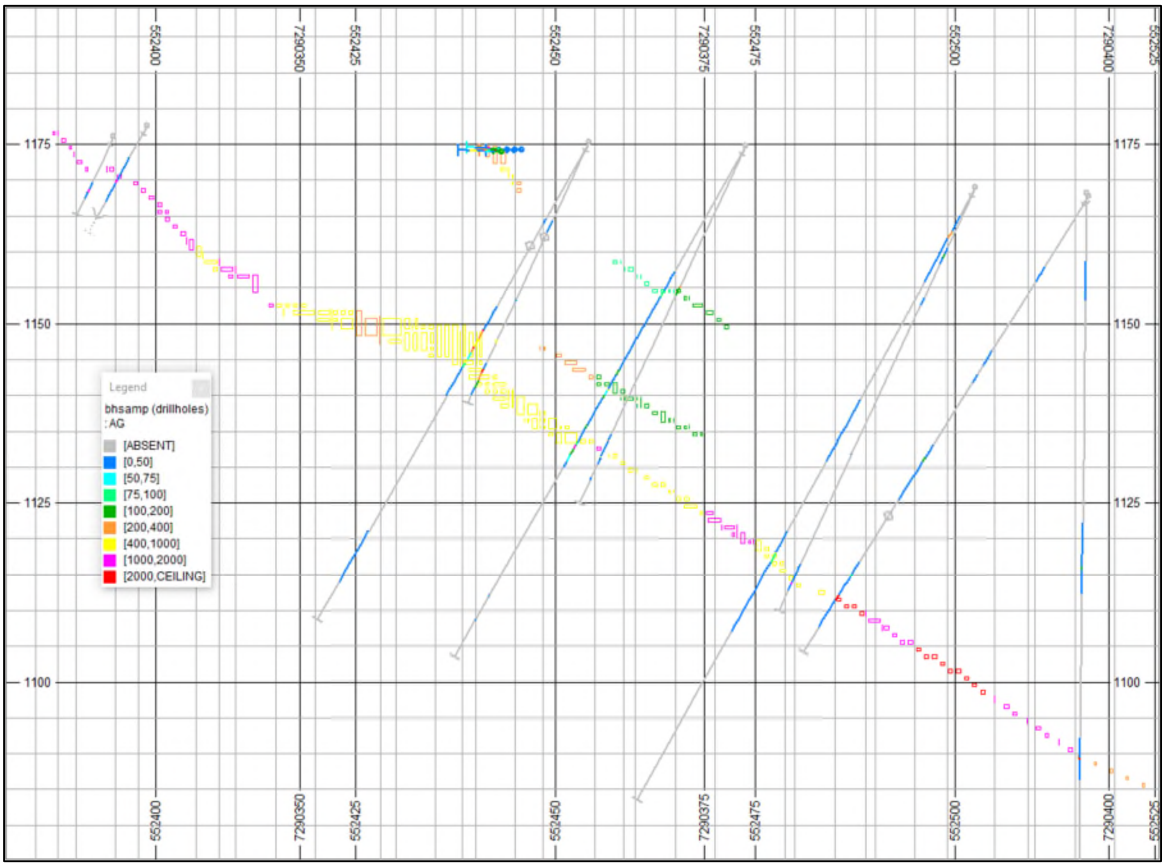


Figure 13.35: Block Model Grades vs Original Samples

iii) Local Comparison (SWATH Plot)

Swath plots were generated to compare the average block model grade and grade in the composite data (example is given in Figure 13.36). A series of 100m slices from south to north and horizons in 50m bottom-upwards were used to assess the average grade for the block model and for composite data. A generally close relationship was observed between composite and block grade across the model. Some deviations between the composite and estimated block grade occur at the edges of the deposit where reduced tonnages accentuate the differences in grade. Differences in grade also become more apparent in lower grade areas. These lower grade areas are typically where the density of drilling decreases and a few composites can have a disproportionate effect on the estimated grades.

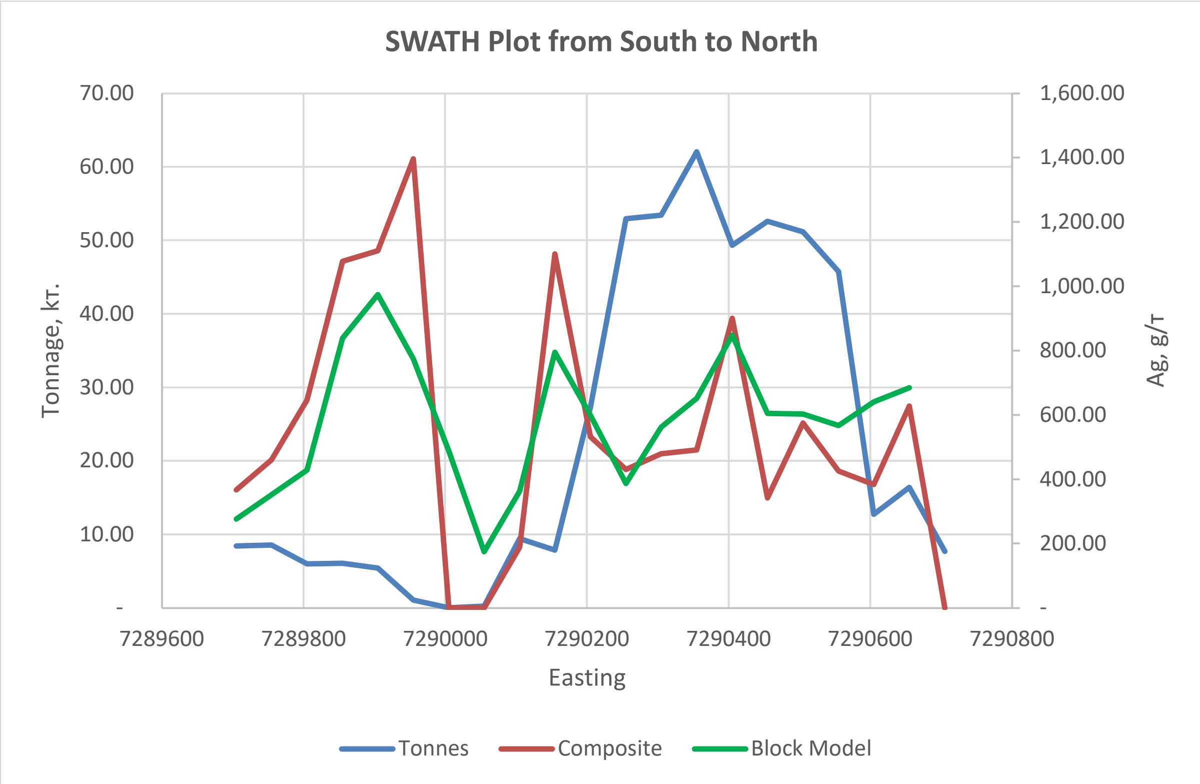


Figure 13.36: SWATH Plot for Ag, looking from South to North

iv) Validation Summary

Globally no indications of significant over or under estimation are apparent in the model nor were any obvious interpolation issues identified. From the perspective of conformance of the average model grade to the input data, WAI considers the model to be a satisfactory representation of the sample data used and an indication that the grade interpolation has performed as expected. In terms of conformance to the drill hole composite data, WAI considers the OK interpolation method to most closely represent the drillhole data. The Mineral Resource Estimate is therefore based upon the OK grade estimation for all zones.

As a general comment, the validations only determine whether the grade interpolation has performed as expected. Acceptable validation results do not necessarily mean the model is correct or derived from the right estimation approach. It only means the model is a reasonable representation of the data used and the estimation method applied.

13.2.6.6 Mineral Resource Classification

The North Mangazeisky mineral resources are classified in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [the JORC Code (2012)].

i) Considerations for Mangazeisky Resource Classification

To classify the Northern Mangazeisky deposit, WAI has taken into account the following indicators:

- Geological Continuity and Complexity;
- QA/QC Results - Quality of Data;
- Spatial Grade Continuity - Results of Geostatistical Analysis; and
- Quality of Block Model.

Since it is impossible to delineate and determine the geometry of oxide and primary mineralization at North Mangazeisky, WAI believes that the silver, lead, and zinc resources can only be classified as Inferred.

Geological Continuity and Complexity:
With the current drill hole/trench spacing, geological continuity between exploration profiles both along strike and down dip is seen. The current drill hole spacing allows for interpretation of continuous zones of mineralisation based on the cut-off grades of 50g/t Ag. At the same time, the submitted data is insufficient to delineate mineralization of different types – oxide and primary.

Data Quality:
QA/QC results of exploration data show acceptable results when measuring accuracy, precision and contamination. This data can be used for estimation of mineral resources.

Spatial Grade Continuity:
An assessment of spatial grade continuity is important when assigning classification to a Mineral Resource. The confidence that can be placed in the variogram parameters is a major consideration when determining classification. The data used in geostatistical analysis resulted in reasonably robust along strike and down dip variogram structures for silver. However, no variograms could have been created for lead and zinc.

Block Model Veracity:
Validation of the block model has shown the estimated grades to be a good reflection of the input composite grades. Visual and statistical checks reveal no evidence of major under or over estimation.

ii) Final Classification

WAI considers that the Northern Mangazeisky Mine has been sufficiently explored to assign Inferred Mineral Resources as defined by JORC Code (2012).

13.2.6.7 Mineralised Inventory at North Mangazeisky

WAI estimated the mineralization within the mineralized wireframes modelled at 50g/t Ag COG. It should be noted that this estimation of mineralization is not mineral resources in accordance with the guidelines of the JORC Code (2012) since it **is not** limited to the optimum open pit and underground contours. It has been included within the report to allow comparison with previous resource estimates. The Mineralised Inventory at North Mangazeisky is presented in Table 13.39.

Table 13.39: Mineral Inventory at North Mangazeisky within Wireframe Models

Zone	Volume, m3, 000	Tonnage, kt	Ordinary Kriging						IDW					
			Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t	Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
1	105.65	376.10	640.70	9.24	0.84	240,971	34,747	3,158	619.29	8.91	0.92	232,917	33,509	3,478
2	1.27	4.53	388.36	3.78	0.30	1,758.6	171.1	13.5	406.43	3.73	0.32	1,840.5	169.0	14.4
3	0.75	2.67	215.12	4.25	0.13	573.6	113.4	3.5	226.68	4.46	0.13	604.4	118.8	3.4
4	6.42	22.87	333.52	0.97	0.09	7,626.3	220.7	19.8	331.89	0.96	0.09	7,588.9	219.8	20.2
5	2.51	8.92	286.66	3.93	0.14	2,557.4	350.6	12.8	283.71	3.85	0.15	2,531.1	343.5	13.0
6	0.25	0.89	1,935.00	26.45	0.66	1,715.3	234.5	5.9	1,906.10	26.63	0.66	1,689.7	236.1	5.9
7	0.32	1.15	367.40	10.22	0.09	422.5	117.5	1.0	345.49	9.49	0.09	397.3	109.2	1.0
8	0.48	1.71	235.05	0.10	0.02	402.5	1.7	0.3	248.60	0.10	0.02	425.7	1.7	0.3
9	0.28	0.98	101.00	2.91	0.11	99.2	28.6	1.1	105.86	3.16	0.11	104.0	31.1	1.1
10	0.60	2.12	168.74	2.91	0.22	357.4	61.6	4.6	169.01	2.91	0.22	358.0	61.7	4.6
11	0.28	1.01	1,604.43	11.51	0.39	1,616.4	116.0	4.0	1,569.40	11.28	0.39	1,581.1	113.7	3.9
12	0.03	0.10	237.60	1.20	0.28	24.5	1.2	0.3	237.60	1.20	0.28	24.5	1.2	0.3
13	0.03	0.10	1,298.66	0.84	19.82	124.8	0.8	19.1	1,305.90	0.84	19.94	125.5	0.8	19.2
14	3.70	13.17	484.94	6.41	3.28	6,387.6	844.3	432.0	483.14	6.36	3.25	6,363.9	838.0	428.2
15	0.58	2.08	734.58	6.74	0.12	1,527.2	140.1	2.4	730.07	6.69	0.12	1,517.8	139.1	2.4
16	0.26	0.92	1,968.00	15.17	0.47	1,814.6	139.9	4.3	1,968.00	15.17	0.47	1,814.6	139.9	4.3
17	9.89	35.20	545.72	3.27	0.34	19,210.1	1,150.0	119.1	533.73	3.37	0.35	18,788.0	1,185.4	122.6
Total	**133.29**	**474.52**	**605.23**	**8.10**	**0.80**	**287,189**	**38,439**	**3,802**	**587.28**	**7.84**	**0.68**	**278,672**	**37,218**	**4,122**

13.2.6.8 Reasonable Prospects of Economic Extraction

Parameters for constraining of mineral resources at Morth Mangazeisky were similar to that for the open pit optimization at Vertikalny, except for the following:

- Oxide mineralization was not delineated due to the lack of data;
- The accepted overall slope angle was 45 ° due to a limited geotechnical dataset.

The mineral resources for open pit mining constrained to the open pit shell are illustrated in Figure 13.37.

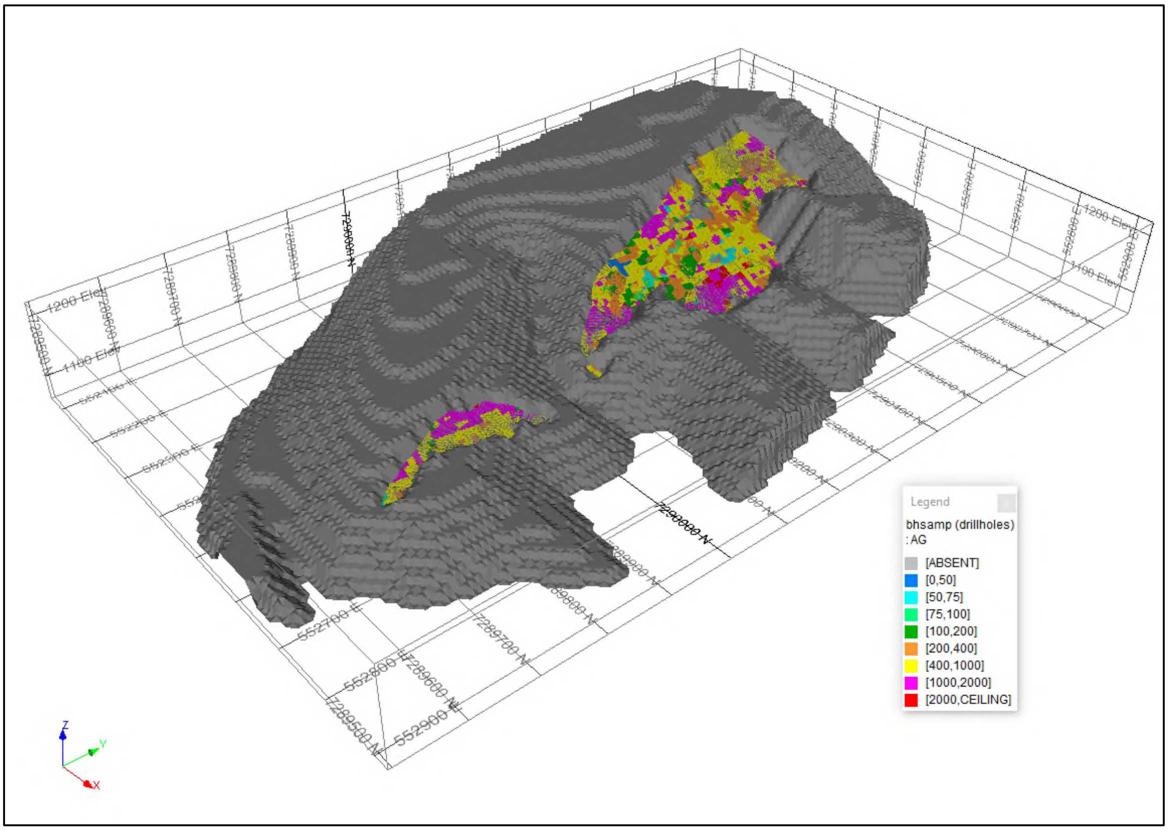


Figure 13.37: Mineral Resources for Open Pit Mining

13.2.7 Mineral Resource Statement for North Mangazeisky

The North Mangazeisky mineral resources have been estimated in accordance with the guidelines of the JORC Code (2012) as seen in Table 13.40.

WAI is not aware, at the time of preparing this report, of any modifying factors such as environmental, permitting, legal, title, taxation, socioeconomic, marketing, and political or other relevant issues that may materially affect the Mineral Resource estimate herein; nor that the Mineral Resource estimate may be affected by mining, metallurgical, infrastructure or other relevant factors.

Table 13.40: Mineral Resource Estimate. North Mangazeiskiy Project, Russia. 31st of May 2019 (In Accordance with the Guidelines of the JORC Code (2012)) Potential Open Pit Resources

Ag Cut-off, g/t	Category	Tonnes, Kt	Ag, g/t	Pb, %	Zn, %	Ag, kg	Pb, t	Zn, t
50	Inferred	364.17	695.00	9.02	0.92	253,102	32,848	3,350
100	Inferred	354.94	711.24	9.25	0.94	252,446	32,819	3,335
200	Inferred	331.41	750.15	9.71	0.98	248,612	32,185	3,261
300	Inferred	309.87	784.56	10.20	0.99	243,111	31,604	3,073
400	Inferred	275.53	838.43	10.91	1.08	231,015	30,049	2,978

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. Mineral Resources are not Ore Reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study.
3. Mineral resources include all potential mineable tonnage.
4. Mineral Resources are estimated as of 31 May 2019.
5. Mineral Resources were constrained by conceptual optimum pit contours using NSR and in accordance with the parameters presented in Table 13.21.
6. All values in the tables have been rounded with relative accuracy of estimate. Numbers may not compute due to rounding.
7. Mineral Resources were constrained by an optimum pit shell based on the corresponding economic and mining parameters provided by the Client and/or accepted by WAI
8. The North Mangazeisky mineral resources were estimated in accordance with the guidelines of the JORC Code (2012) by Steven McRobbie, Independent Competent Person as defined by the JORC Code.
9. This mineral resource estimate is not limited to any factors in terms of environmental, permitting, legal, title, taxation, socio-economic, market and other relevant factors.
10. The metal resources include all the in-situ metal disregard the metallurgical recovery factor.
11. The Russian version of this report uses the following JORC terms in Russian:

Ore Reserves		***Mineral Resources***		
извлекаемые запасы		*минеральные ресурсы*		
Proven	***Probable***	***Measured***	***Indicated***	***Inferred***
доказанные	*вероятные*	*измеренные*	*исчисленные*	*предполагаемые*

The words "ore", "mineralized" and "mineable tonnage" in this Russian version are used as "natural mineralized material" without reference to the profitability and technical feasibility of its mining and processing.

13.2.8 WAI MRE vs. Tetra Tech MRE

Tetra Tech (TT) estimated mineral resources of North Mangazeisky in 2017. Mineralized wireframe models were developed and samples within the wireframes were taken followed by compositing of 0.4m. The undertaken statistical analysis did not identify silver outliers for top-cutting. The variogram models were created in three directions with the following search radii:

- Along the strike – 95m;
- Down-dip – 45m;
- Across the strike – 15m.

The density values were interpolated to the block model using the Inverse Power Distance Squared; the blocks without the estimated density values were assigned with 3.18 t/m3. Ordinary kriging was used to interpolate grades to the block model; several estimation passes were run with each one using a consecutively larger ellipsoid.

The following parameters were used to determine the potential for economic extraction of mineralization:

- Silver price – 17 US$/oz;
- Losses – 5%;
- Dilution – 30%;
- Operational costs:
  - For mining – 2.53 US$/t ore
  - For processing – 52 US$/t ore;
  - G&A – 40.60 US$/t ore;
- Royalty – 6.5%;
- Overall recovery – 88%.

Based on these parameters TT concluded that the 150g/t Ag cut-off grade shall be applied to the mineralization to estimate mineral resources (Table 13.41).

Table 13.41: Mineral Resource Estimation, Tetra Tech, 2017

Category	Tonnage, kt	Ag, g/t	Ag, kg
Indicated	334	770	257,180
Inferred	127	560	71,120
Total	**461**	**712**	**328,300**

Location of the TT and WAI mineralized wireframes is shown in Figure 13.38. The TT mineral resources were not constrained to the optimum RF1 pit shell. It should be noted that the TT model was extrapolated for a significant distance downdip from the workings at the deposit owing to wider drill spacing and assumption of greater continuity of mineralisation. The additional drill results incorporated in the WAI MRE have enabled greater definition of the resource model albeit more conservative in response to greater discontinuity. In this regard, it is not conducive to undertake direct comparison of the TT and WAI mineral resources.

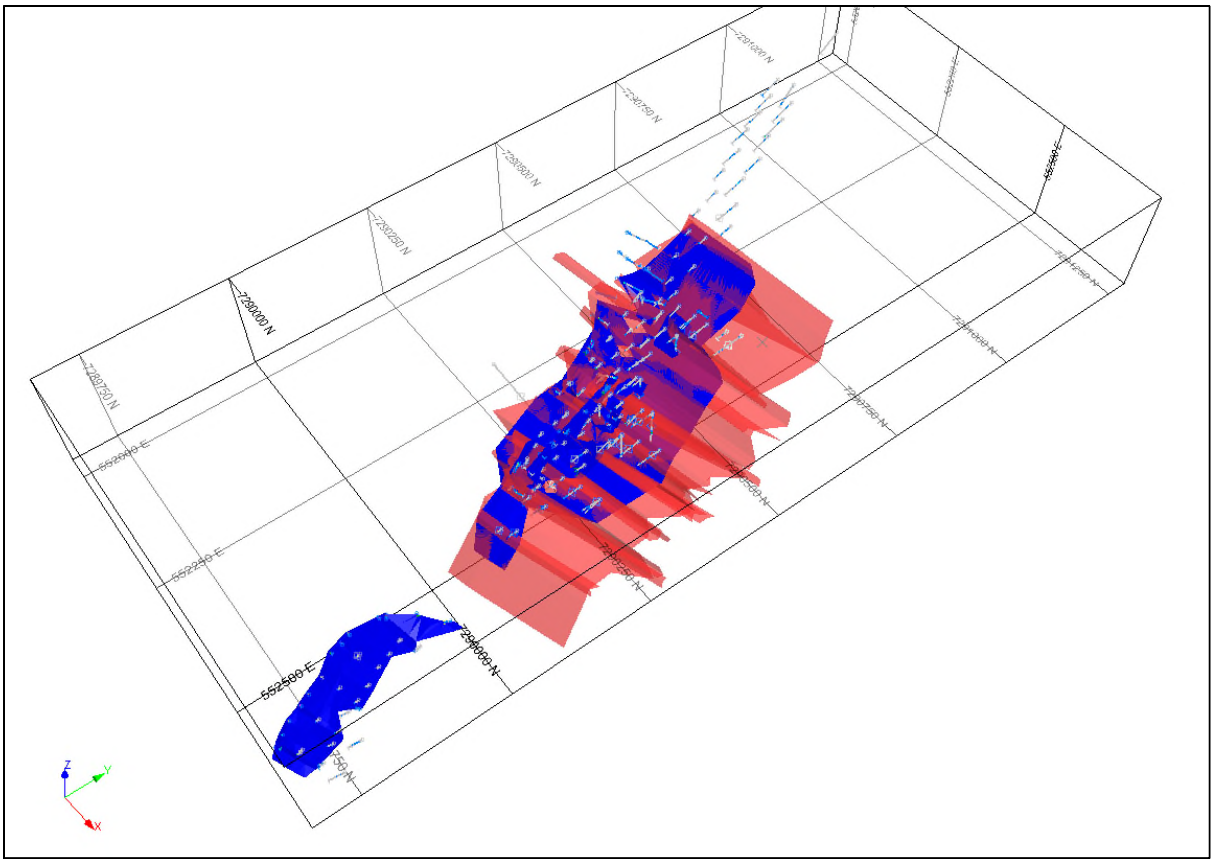


Figure 13.38: Wireframe Models of TT (red) and WAI (blue) with workings at Northern Mangazeisky

14 MINERAL RESERVE ESTIMATE (ITEM 15)

Estimation of mineral reserves has not formed part of this study and are not reported here.

It should be noted that 'minable tonnage estimates' are not Ore Reserves and are not demonstrative of technical and economic viability. The study was carried out to assess the potential of the Mangazeisky Silver Project as whole and identify any strategic bottlenecks.

The use of 'minable tonnage estimate' or minable inventory and its relationship to Mineral Resource Estimates is discussed further in Section 14.5.

15 MINING METHODS (ITEM 16)

15.1 Mining Methods

WAI has carried out a scoping level open pit mining study to define a mineable tonnage estimate for the Vertikalny and Mangazeisky North deposits. The Vertikalny deposit is currently being extracted by open pit mining techniques, whereas the Mangazeisky north deposit is greenfield and has yet to be mined.

WAI has also carried out a mining study to define an underground mineable tonnage estimate for the Vertikalny deposit. The study has considered the volume of mineralised material below the generated Vertikalny pit designs. The study is based on applying a stope optimiser to the mineable tonnage estimate and assessment of supporting development/infrastructure and constitutes only a high-level conceptual design given that 'minable tonnage estimates' are not Ore Reserves and are not demonstrative of technical and economic viability.

15.2 Hydrology and Hydrogeology

15.2.1 Introduction

This assessment considers hydrogeological modifying factors relating to mining of the open pits at the Vertikalny deposit and subsequent open pit mining of the Mangazeisky North deposit. The review is based on information provided by the client. Hydrogeological modifiers associated with underground mining (Vertikalny) are also considered at a development parameter level only.

The assessment is based on a review of the completed works, available designs and WAI's own mine pre-design opinion. Project technical and economic factors are considered, environmental and social (E&S) assessment has been excluded from the scope, however any significant hydrogeological factors affecting E&S are noted for consideration in the next project phase. Hydrogeology may affect pit shell design, feasibility, mining parameters and production scheduling if significant groundwater control is required. The performance of the mine has varied from the feasibility benchmarks primarily because of geological (resource) in-situ variability, ore processing costs, mining costs – however this is predominantly due to the variation in ROM production rather than the intrinsic cost of mining, and administrative and infrastructure costs. The role that mine-water management has played (if any) on affecting mining costs is examined below.

15.2.2 Hydrogeology

15.2.2.1 General

The Mangazeisky silver deposit, comprising multiple targets along a N-S striking orebody is within the Endybal River basin, a tributary of the Arkachan River. Six named rivers and smaller streams are noted within the Licence area. These streams are classified as sixth and lower order watercourses (with overall drainage to the Yana River): Feodor-Yureghe River, Sirilendzhe River, Mangazeyka River, Porfirovy Creek (adjacent to the southern termination of the Vertikalnoye pit and approximately 50m below the underground mine portal), Borisovsky Creek and Nameless Creek. The rivers are typically upland type characterised by low salinity, soft, weakly alkaline quality not exceeding 30m cross-sectional width at maximum spate condition. The creeks are typically ephemeral in the order of less than 2m width. Natural geochemical parameters in the surface water result in exceedances of regulatory Maximum Permissible Concentrations (MPC) for lead, zinc, aluminium, nickel and cadmium in a number of samples reflective of the mineralisation of the region.

The mining operations target the Vertikalny vein's Central and Northwest Zones of mineralisation, situated in the Mangazeisky licence area. The Central Zone extends for some 1,600m along strike, Northwest Zone extends approximately 900m along strike. The process plant site is located in the Porfirovy stream valley. The project water supply is direct run-of-river abstraction principally from the Arkachan river, with additional summer flow supplements withdrawn from local creeks (Endybal and Mangazeisky) as necessary (ERM, 2014).

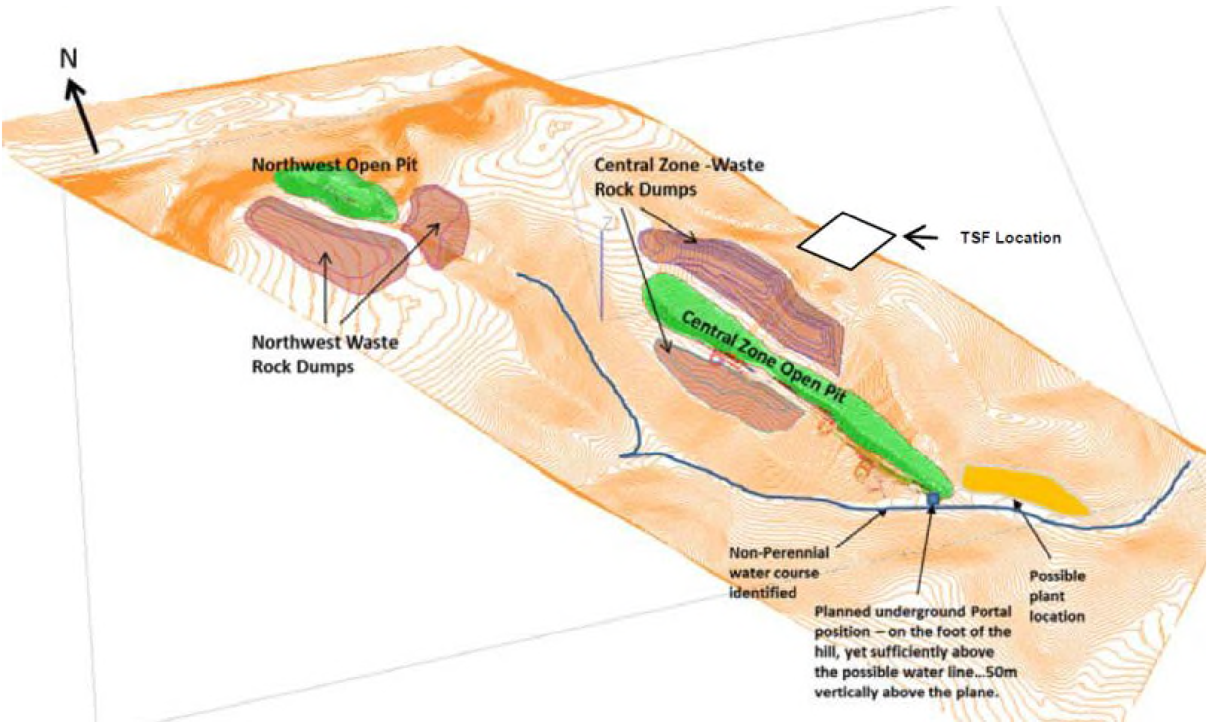


Figure 15.1: Approximate Mine Layout Sand Topographic Relationship (ERM, 2014).

The climate is extreme continental arctic with average snow cover days of 240 per year and low precipitation average – 320mm of which a third is as snow. Spring thaw occurs in early May and snow begins to settle in September with permafrost prevalent across the terrain. The area has very low wind activity and precipitation is anticipated to be negligible given the prevailing summer temperatures do not generally exceed 13˚C (ERM, 2014).

15.2.2.2 Hydrogeological Description

The project is at the edge of the Siberian platform in the interfluve of the Nuektame and Arkachan rivers. The site is within the West-Verkhoyansk hydrogeological massif. Fractured aquifers within sandstones, siltstones, conglomerates and shales (Carboniferous - Permian age) are understood to be modified by faulting, structural blocking and compartments and metamorphic texture and facies controls. Overprinting the lithology and aquifer characteristics is a layered permafrost system comprising an upper active zone where annually porewater freezes and thaws, and an underlying permafrost zone in which porewater is permanently frozen. Below the permafrost at depth, groundwater becomes unfrozen again. Recharge of meteoric and seasonally warm melt water through 'talik' and colluvial materials, also possibly through preferential flow networks in fracture zones can be important controls over the hydrogeological water balance and flow mechanisms, potentially resulting in deeper groundwater occurrence than otherwise suggested by the nominal thickness of permafrost present.

The depth to a permanent groundwater water table is reported to be 300 to 500m (ERM, 2014) depending on location and aquifer type. The overall groundwater system is consistent with typical Siberian groundwater regimes with deep, confined groundwater held within generally reducing permeability fractured rocks at depth and a dynamic near surface (active zone or supra-permafrost) system which cycles significant quantities of groundwater through 'talik' and alluvial water 'beqaring' zones with baseflow and spring discharge. Groundwater is reported to be a bicarbonate-sodium type with low mineralization. Recharge rates are reported to be 1 L/sec per 1km^2 (8.7E^{-5}m/day). The total spring discharge rate was estimated at 36m^3/second. Hydrometerological surveys carried out in September 2015 representing the annual lowest flow period are shown in Figure 15.2.

Hydrographic Monitoring Point 1. Porfirovy Creek. Flow 0.11m^3/sec

Hydrographic Monitoring Point 2. Borisovsky Creek. Flow 0.01m^3/sec

Hydrographic Monitoring Point 3. Sirilendzhe River. 25km downstream Flow 3.08m^3/sec

Figure 15.2: Project Surface Water Systems (photos and flow records courtesy of Nerungristroyresearch, Vol. 3 Book 1 (Hydrometeorology), April 2016)

15.2.2.3 Sources of Information

The principal source of hydrogeological information has been an SRK study included within the Tetra Tech 2017 competent persons technical report. The objective of the SRK work was to develop an understanding of mine hydrogeology, assess dewatering requirements and assess the usability of a sub-permafrost aquifer to supply the mine with water.

The overall information and data sources reviewed includes:

- Tetra Tech, 2017. NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation Document No. 1454430200-REP-R0006-02.
- SRK Consulting (UK) Limited, 2016, Project No.U6065 Appendix K of the NI 43-101 Technical Report: Hydrogeology.
- ERM, October 2014, Scoping Report, Mangazeisky Project: Environmental and Social Impact Assessment, Project №0264539
- Nerungristroyresearch, Vol. 3 Book 1 (Hydrometeorology), April 2016. Technical report on engineering and hydrometeorological research. Ref. 497-75/14-IGM.

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- Non-orientated core logging sheets for geotechnical borelogs 14B – 191 (Filepath: N:\RU\RU10139 - Mangazeiskiy silver project\02 - Data\Incoming\IN ENGLISH\SBR-WAI DATA ROOM\документация скважин\Геотех-кая Док-ция Geotechnical Log V11-191.xls inter alia)
- Nerungristroyresearch, Vol. 2, Book 3, Part 1, (Geophysics and Geological Survey), April 2016. (Filepath N:\RU\RU10139 - Mangazeiskiy silver project\02 - Data\Incoming\IN ENGLISH\SBR-WAI DATA ROOM\ Технический_проект_Вертикальное\ вертикальное_огр\ изыскания\ИГИ\ 497-75-14-ИГИ-Книга 3_изм2_Часть_1.pdf)
- Nerungristroyresearch, Vol. 2, Book 1, (Geological Engineering), April 2016. (Filepath N:\RU\RU10139 - Mangazeiskiy silver project\02 - Data\Incoming\Report on Geology\TOM 2

15.2.3 Pit Geometries and Interaction with Groundwater

The deposit has an exceptionally narrow geometry and necessitates a pit design and mining method that is highly optimised to minimise mine wastes and control grades. The pit design and optimisation has been based on SRK geotechnical studies and slope configuration results (Tetra Tech, 2017, Appendix B). Sensitivity analysis on the selected pit shells (base case) shows the overall financial model is relatively insensitive to mining and processing costs and most sensitive to grade control (Tetra Tech, 2017). Consequently, the steep (vertical) mineralisation promotes a constrained mining method to access ore and maintain integrity of the mine structures. Calculated overall strip ratio for the operation is 25. Overall slope angle is defined by bench and berm geometry and inter-ramp angles (IRAs) which, in line with kinematic and rock fall analysis has resulted in a maximum IRA of 56° recommended for the hanging wall and 48° for the footwall. Steeper slopes will start to undercut the bedding on an inter-ramp scale (instability).

The deposit will be mined in a north zone (Mangazeisky) by open pit, and a central zone approximately 6km south-southwest named Vertikalny which will be mined by open pit and underground methods. Vertikalny will comprise a sequence of four individual pits developed along strike of the mineralised vein with underground mining commencing beneath the main part of the central zone with the final underground drive (Zone 4) extending northwards.

The Mangazeisky pit has an overall strike length of 650m, maximum width of 250m, and a pit floor elevation of 1084m. Vertikalny will be developed as four pits along strike, the pits range from Pit 2 (smallest) with dimensions of 50m width and 120m length to the largest (Pit 4) in the northwest which is 145m width and 530m length. The respective floor elevations of these pits are 1117m Above Datum (AD) and 1094mAD.

Underground mining will occur through sub-level open stoping, with remote stope cleaning extending below the open pits with mine access portals located above invert levels of the surface water systems and open water accumulations in the pits. The underground mine will comprise 25m vertically spaced mining levels, the planned underground mining depths are approximately 950mAD in underground zones 2, 3 and 1 which correspond to pits 4, 3&2 and 1 respectively. Zone 4 is a northward extension beyond the Vertikalny open pit footprint. Zone 1 in the southern section of the deposit is the deepest underground section and is planned to extend to 700 mAD. Generally, the maximum depth of the underground section is approximately a 150m deeper than the base level of the overlying open pit. In zones 1 and 4 of Vertikalny, the maximum depth of the underground mine below ground surface is approximately 300m. Given the long-term tendency for permafrost thickness reduction, the lower levels of these zone should conservatively be assumed to be in sub-permafrost (free-flowing water) conditions.

15.2.4 Groundwater Control and Management

SRK prepared an open pit and underground geotechnical study in support of the TetraTech NI 43-101 study (Figure 15.3) in which it was noted that ground conditions *"are generally good with no special measures required for orebody extraction"*. SRK also completed a feasibility-level hydrogeology and water supply study for the Mangaziesky Project with two main objectives to assess the potential inflows of water into the mine and evaluate the water supply potential of the sub-permafrost aquifer.

Site investigations focused on the hydrogeology of the mine location and the sub-permafrost aquifer along the main Sirilendzhe River, where the permafrost layer is expected to be thinner and the potential for water supply from the sub-permafrost aquifer higher. SRK noted with respect to hydrogeology that the open pits and underground mines are *"entirely located in the permafrost; therefore, groundwater inflow into the mine workings, if any, will be negligible."* SRK appraised the surface water (precipitation) based inflows to the pits and deduced a pumping capacity to deal with average flow of 100m^3/hour would also need to be able manage 200m^3/hour inflows for exceptional (1 in 100 year) storm events. It was noted that the Siberian conditions mean that water is unfrozen only in late spring and summer (April to October).

SRK also investigated the permafrost distribution within the proposed mine site using two deep boreholes (VG-2 and G-1) which were equipped with thermistors. Temperature measurements were taken downhole several times and whilst the boreholes did not traverse the full thickness of the permafrost, extrapolation was possible.

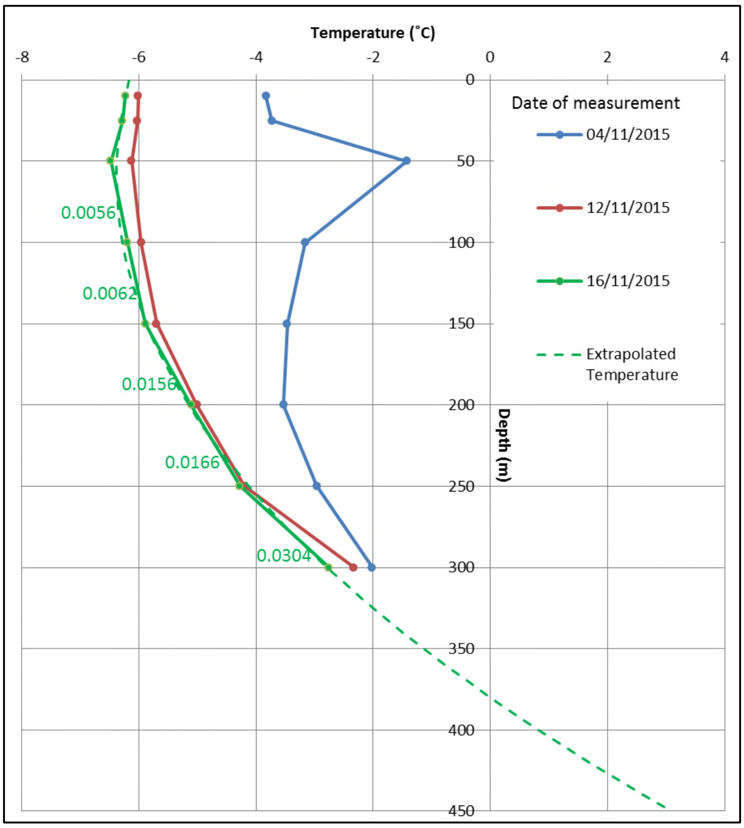


Figure 15.3: **Underground Mine Layout (Tetra Tech, 2017)**

The depth to the bottom of the permafrost was assessed by SRK who identified this could be 380-400m in the interfluve area of the mine and between 157 – 220m deep in the area of the stream valleys.

15.2.5 Groundwater Supply

Surface water is unable to provide a reliable water supply source to the mine due to strongly seasonal hydrographic variations and prolonged freezing periods. SRK, 2016 evaluated a sub-permafrost groundwater source of water:

> "For the purpose of the water supply investigation, four boreholes were drilled along the Sirilendzhe River, which traversed the full thickness of the permafrost layer and reached the aquifer beneath. The boreholes showed artesian flow, with water levels ranging from 1.1m to 14m above the ground level. Pumping tests were completed on three of the four boreholes, to estimate the hydraulic parameters of the aquifer. The fourth borehole, which is drilled near the current camp at the junction of Porfirovy Stream and Sirilendzhe River and labelled GS15-05, was not pumped because the artesian water outflow was higher than the capacity of the pump available on site at the time. In addition to the high-water flow, this borehole also showed the best water quality among all four boreholes, and seems the most suitable for the Project water supply. Therefore, the location of this borehole is recommended by SRK as the most appropriate site for water supply well installation."

SRK undertook modelling (including development of a finite element numerical model) based on the limited field data available. The modelling used hydraulic conductivity and specific storage properties from the results of pumping and recovery tests conducted in the hydrogeological boreholes. Values used in the model appear to be realistic and there is a variability of an order of magnitude (K = 0.78m/d) to account for higher flow in a fault zone. Modelling results indicated a sustainable supply of water for the life of mine for three different groundwater pumping scenarios, wherein rates and duration of pumping were altered to match the annual demand requirements (± input for 4 months from surface flow when available).

- The assumption that the underground mine will be located fully in the permafrost zone and groundwater inflows into the mine workings, if any, would be negligible (SRK 2016) appears to have been disregarded by the mine designers and contradicts a statement in the Geotechnical report (Appendix B) which states some of the underground workings may be in the sub-permafrost water bearing zone. Tetra Tech 2017 assumes the underground mine will need a drainage system comprising collection sumps in each underground mine situated at the lowest adit level receiving uncaptured drainage from the levels above. Drainage from ramps, raises, drain holes and stopes is designed to report to the sumps. The gradients of the main drives and levels of the underground development are designed to facilitate gravity drainage from the mine towards adits to avoid flooding.
- The potential for underground mine inflow needs to be confirmed and re-appraised using suitably conservative assumptions.

Surface water hydrology and the mine water balance have been reviewed and no additional comments over and above what has already been presented by SRK are raised.

15.3 Geotech

15.3.1 Introduction

WAI has carried out a review of the geotechnical information provided by Silver Bear Resources (SBR) for the Vertikalny and Mangazeisky North deposits.

Information was collected from the findings of the geotechnical study carried out by SRK Consulting (SRK)[1] in late 2014 for the Vertikalny deposit. The review has aimed to summarise the geotechnical parameters for use in mine optimisation and design in support of the strategic review for the Mangazeisky Silver Project.

WAI has not carried out a site visit, nor has it carried out an independent review of the geotechnical data used in the SRK study.

[1] SRK Consulting (UK) Limited, 2015. Geotechnical Feasibility Study Report on Open Pit and Underground Mining for the Vertikalny Deposit

15.3.2 Vertikalny Deposit

15.3.2.1 Geotechnical Data Collection

A geotechnical drilling campaign was initiated in late 2014 in support of the SRK geotechnical study. The campaign included the drilling and geotechnical logging of eight diamond cored boreholes for open pit analysis. Additional geotechnical data was gathered from several previous exploration and resource drilling campaigns and used to substantiate the SRK study.

15.3.2.2 Rock Mass Characterisation

15.3.2.3 Lithological Description

The Vertikalny rock mass is overlain by thin layer of overburden and highly weathered rock; generally, less than 10m in thickness. Beneath this zone, the rock mass is primarily composed of alternating sandstone and sandy-siltstone sequences. The sandstone sequences are reported to be unweathered and have well-defined bedding planes. A cross-section prepared by EMC Mining[2] which provides an indicative representation of the Vertikalny rock mass is presented in Figure 15.4, below.

[2] EMC Mining, 06/2015 - «Проект строительства горноперерабатывающего комплекса на базе месторождения «Вертикальное» - Площадка №1. Карьер и отвалы - Геологический разрез по линии 10700

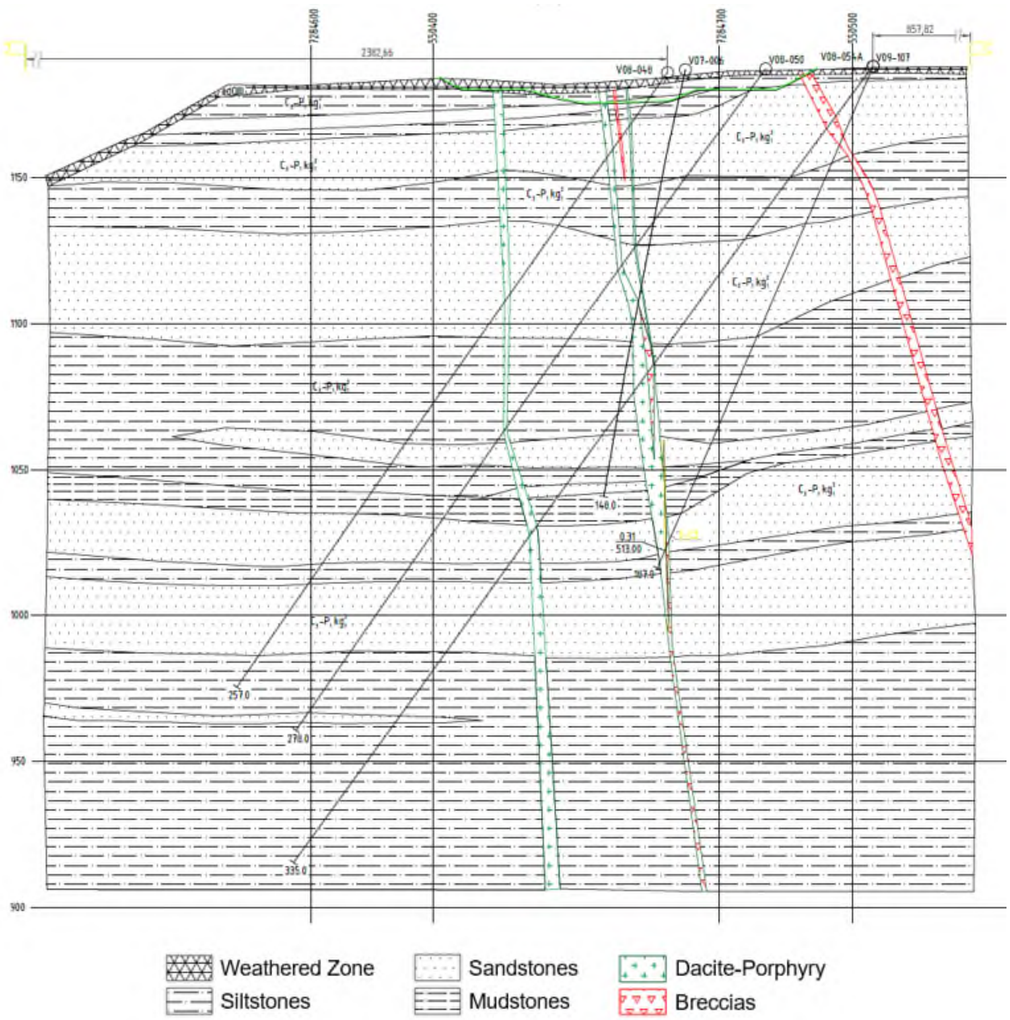


Figure 15.4: Vertikalny Rock Mass Composition

15.3.2.4 Geotechnical Domains

The SRK study notes that the major lithologies have minor variations in rock mass characteristics between one another. Consequently, the geotechnical domains were defined according to the to the mining domains:

- Hanging wall;
- Footwall; and
- Mineralised zone.

SRK generated three-dimensional models of each domain which were used to perform statistical analyses on the key geotechnical parameters. A cross section representing the three domains is presented in Figure 15.5, below.

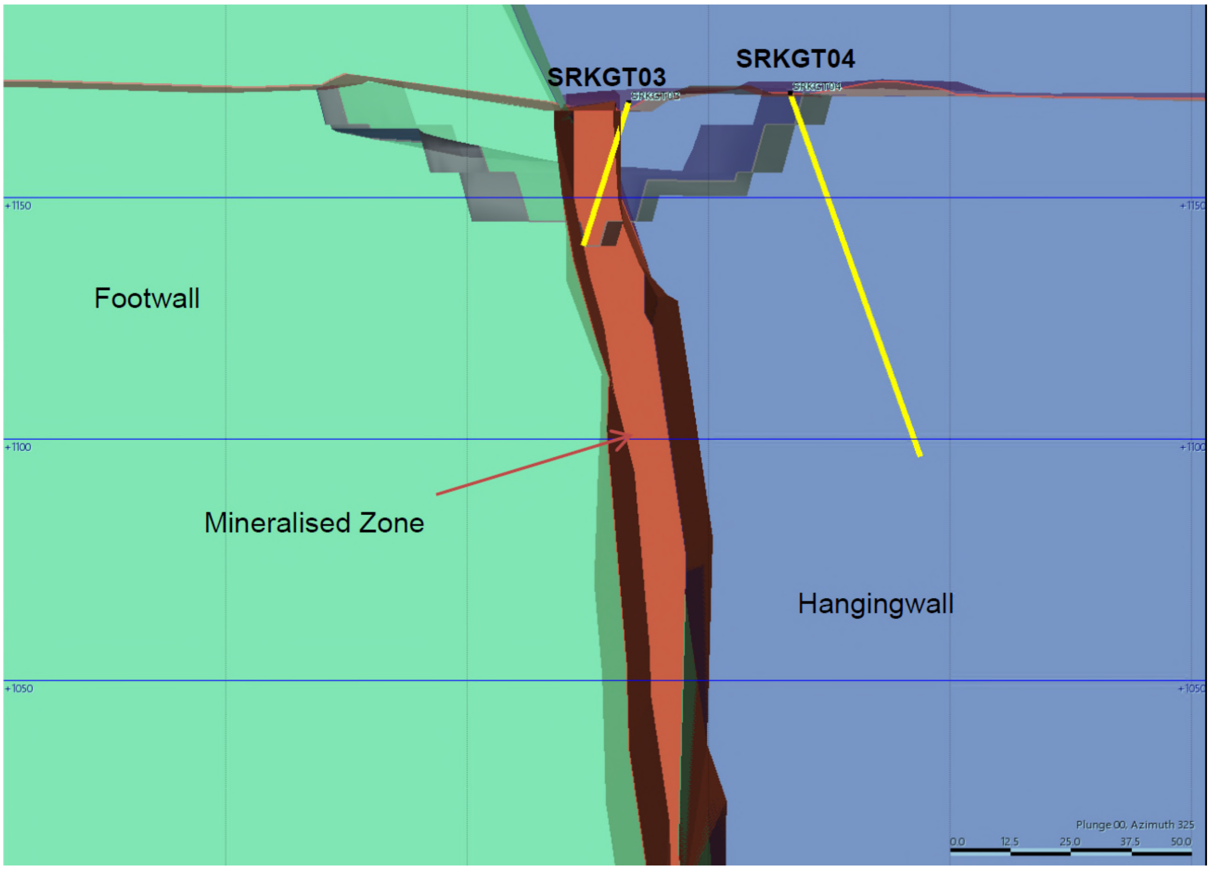


Figure 15.5: Vertikalny Geotechnical Domain Cross-Section (SRK Geotechnical Study)

15.3.2.5 Rock Mass Classification

Open pit and underground rock mass classification was carried out using the RMR_{89}[3] and Barton's Q[4] classification system, respectively. Detail regarding the methodologies and results of the rock mass classification exercise may be found in the SRK geotechnical study.

15.3.2.6 Major Structural Features

WAI is unaware of the availability of any large-scale three dimensional structural/fault models for the Vertikalny deposit. Regional geological maps indicate a series of steeply dipping structures which strike sub-parallel to the mineralisation. A geological map (modified after EMC Mining[5]) indicating these features relative to the Vertikalny deposit is presented in Figure 15.6, below.

[3] Bieniawski, Z.T. 1989. Engineering rock mass classifications. New York: Wiley
[4] Barton, N.R., Lien, R. and Lunde, J. 1974. Engineering classification of rock masses for the design of tunnel support. Rock Mech. 6(4), 189-239.
[5] EMC Mining. 06/2015 - «Проект строительства горноперерабатывающего комплекса на базе месторождения «Вертикальное» - Площадка №1. Карьер и отвалы -Геологический разрез по линии 10700 Масштаб 1:1000

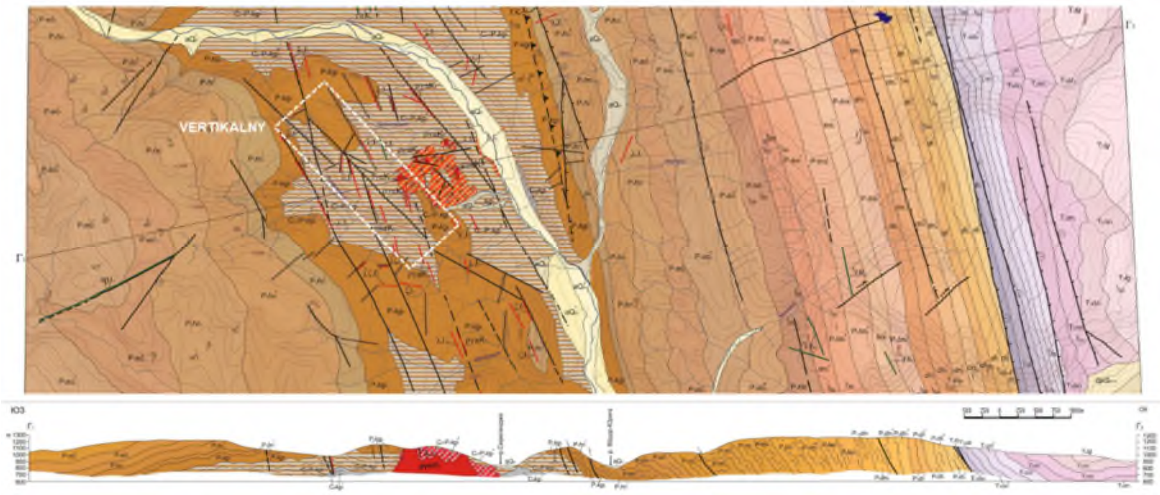


Figure 15.6: Regional Geological Map

SBR have not indicated the presence of any major structural features intersecting the current open pit. Any features that are intersected are assumed to be mapped and managed operationally. An understanding of the location and engineering properties of these features is essential in identifying any potential instabilities within the open pit and future underground operations.

15.3.2.7 Groundwater Conditions

The Mangazeisky Project area has permafrost layer of 300m to 400m in thickness. Groundwater inflows are not considered to play a major role in open pit or underground stability. WAI notes that localised thawing of the rock mass may occur during excavation of the open pit and potential underground workings.

15.3.2.8 Open Pit Geotechnical Review

15.3.2.9 Kinematic Analysis

SRK carried out a detailed kinematic stability analysis to determine the appropriate berm width and bench face angles for the given structural conditions. Kinematic analysis of wedge, planar and toppling type failures were assessed for the hanging wall and footwall rock masses.

Analysis of the footwall rock mass suggests that bench scale planar instabilities are likely to exist. Inter-ramp angles (IRA) were set at 48° to avoid undercutting the bedding and minimise potential multi-bench instabilities. The hanging wall was noted to have favourable structural geometries and able to support a steeper IRA of 56°. No special measures were note for the excavation of the relatively shallow overburden and weathered rock zone.

Additional detail regarding the methodologies and results of the kinematic analysis may be found in the SRK geotechnical study.

15.3.2.10 Numerical Analysis

SRK carried out an assessment of open pit slope stability using the RocScience Phase2 finite element (FE) modelling software package. The software allows for the calculation of the Strength Reduction Factor (SRF); a measure broadly equivalent to Factor of Safety (FOS). Modelling was carried out on the deepest section of the proposed pit to produce the lowest SRF (FOS). Mine geometries were defined using the slope parameters identified in kinematic analysis. The cross-section tested in FE modelling is presented in Figure 15.7, below.

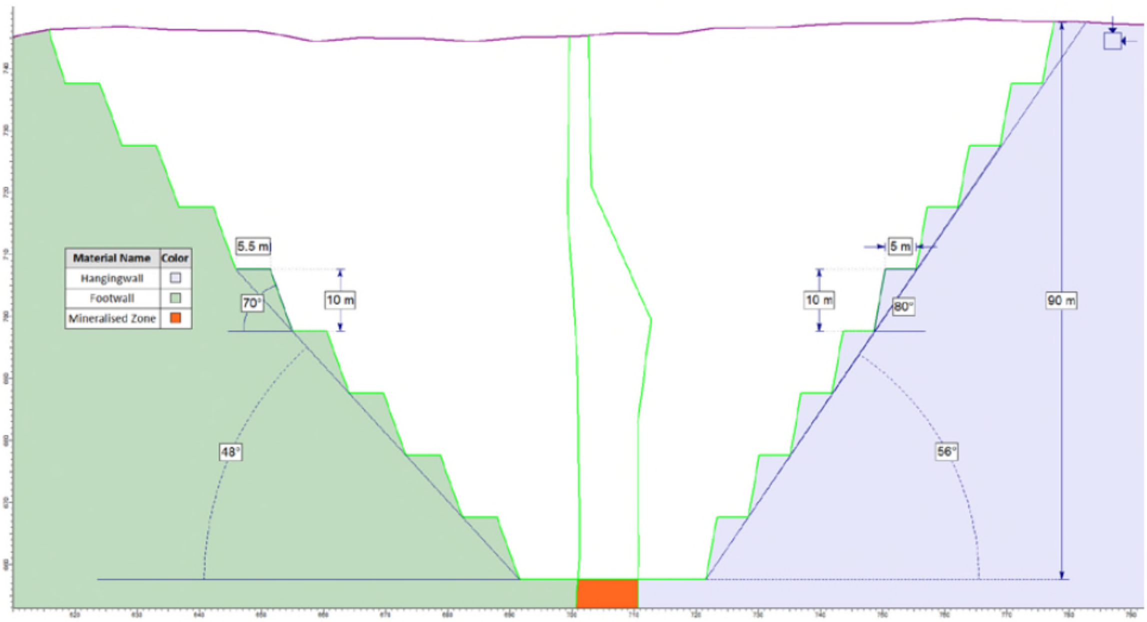


Figure 15.7: Finite Element Modelling Slope Geometry

Rock mass strength was modelled by domain using the Hoek-Brown strength criterion. The results of the FE stability modelling are presented in Table 15.1, below.

Table 15.1: Finite Element Stability Analysis Results

Domain	Strength Reduction Factor	Probability of Overall Slope Failure
Hanging Wall	2.31	0.02%
Footwall	2.39	0.11%

The results clearly indicate that the rock mass can support the prosed mining geometries. Further detail regarding the inputs and methodologies used in the FE modelling may be found in the SRK geotechnical report.

15.3.2.11 Pit Slope Design Criteria

The recommended pit design parameters identified in the SRK study are summarised in Table 15.2, below.

Table 15.2: Pit Design Parameters

Domain	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	Inter-Ramp Angle (°)
Hanging Wall	10	80	5	56
Footwall	10	70	5.5	48

15.3.3 Underground Geotechnical Review

15.3.3.1 Mining Method

Previous studies have suggested the application of several underground mining methods for the Vertiklany deposit. The two main candidates include:

- Shrinkage stoping (SRK geotechnical study); and,
- Sublevel longhole open stoping (Tetra Tech technical report6)

WAI propose to maintain the mining methodology outlined by Tetra Tech; mechanised sub-level open stoping. The method offers favourable results in safety, cost and dilution control. Stopes will be extracted in a retreat, top-down sequence, with adequate in-situ rock pillars left unmined for localised and regional stability.

15.3.3.2 Stope Wall Stability

The SRK study analysed a range of empirically derived stope dimensions determined through the Mathews (Mathews *et al*. 1981[7] and updated by Potvin 1988[8]) stability graph method. The stope dimensions proposed by Tetra Tech, and utilised by WAI, are as follows:

- Strike length: 10m
- Wall height: 25m
- Span: 4m

[6] Tetra Tech 2017. NI 43-101 Technical Report, Mangazeisky Silver Project, Republic of Sakha (Yakutia), Russian Federation

[7] Mathews, K.E., Hoek, E., Wyllie, D.C. and Stewart, S.B.V. 1981. Prediction of stable excavations for mining at depths below 1000m in hard rock. CANMET Report. DSS Serial No. OSQ80-00081, DSS File No. 17SQ. 23440-0-9020 Ottawa. Dept. Energy, Mines and Resources.

[8] Potvin, Y. 1988. Empirical open stope design in Canada. PhD thesis. Vancouver. Dept Mining & Minerals Processing, Univ British Columbia.

Based on SRK's stope stability results, these dimensions plot within the 'stable' zone of the stability graph. A plot of the design surfaces on the stability graph is presented in Figure 15.8, below.

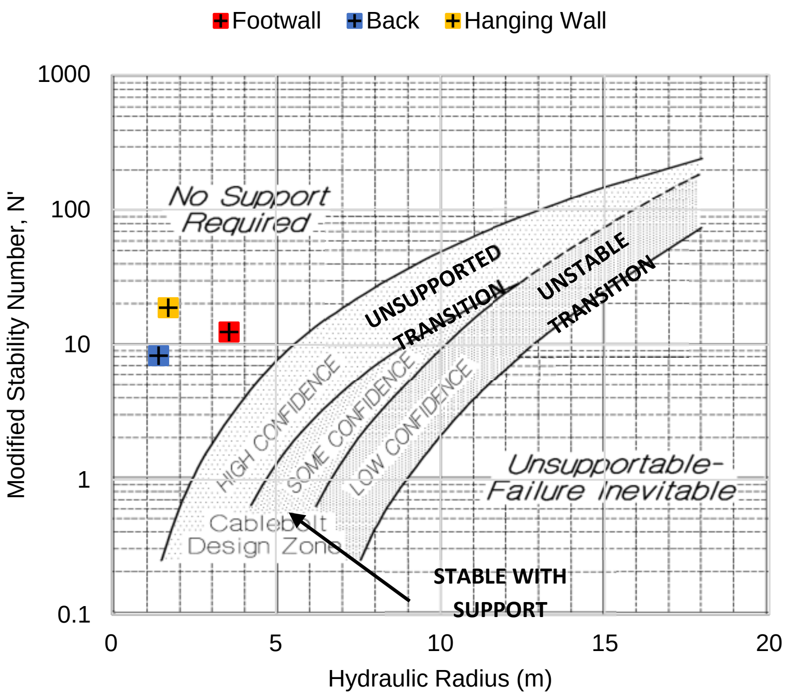


Figure 15.8: Stability Graph for Proposed Open Stop Dimensions

For the given wall height and maximum stope span, stope strike lengths may extend up to 20m before the footwall design surface plots within the 'unsupported transitional' zone of the stability graph. This indicates the approximate spacing at which in-situ rock pillars (rib pillars) would be required to maintain stability. Stope pillar dimensions and spacings have not been defined in the study.

Detail regarding the methodologies and results of the stope stability analysis may be found in the SRK geotechnical study.

15.3.3.3 Crown Pillar Stability

The empirical Scaled Span method (Carter 2014[9]) was used by SRK to assess the required crown pillar dimensions to promote safe workings between the open pit and underground operations. The method draws from a crown pillar database containing over 500 case records with 70 analysed failures.

The SRK study modelled various pillar thicknesses for both shrinkage and open stoping. WAI has utilised the shrinkage stoping results as the modelled pillar spans of 3m closely match the maximum proposed stope span of 4m. A crown pillar thickness of 15m was selected as it provides a good factor

[9] Carter, T.G., 2014. Guidelines for use of the Scaled Span Method for Surface Crown Pillar Stability Assessment. Golder Associates, Toronto, Canada.

of safety and low probability of failure. This figure is comparable to the dimensions utilised in the Tetra Tech design work which range from 10m to 15m.

Detail regarding the methodologies and results of the crown pillar stability analysis may be found in the SRK geotechnical study.

15.3.3.4 Ground Support

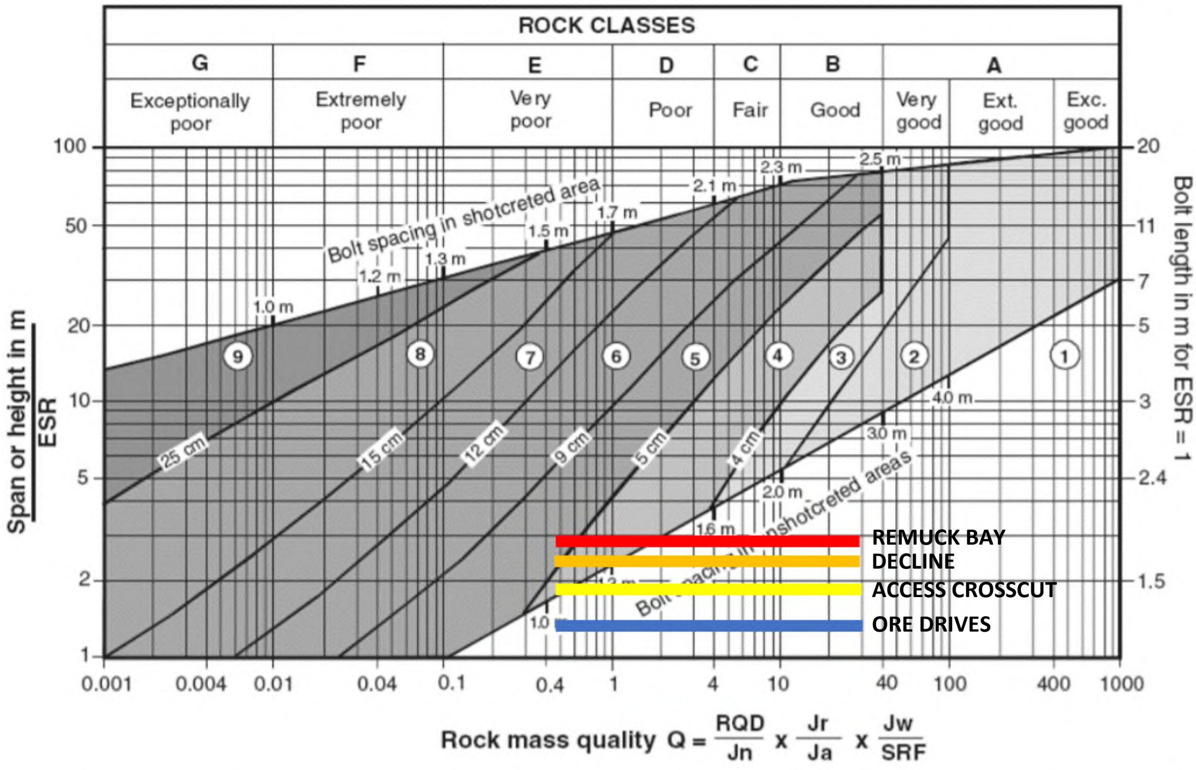
The SRK study estimated ground support requirements by use of Barton's Q system. A set of fixed excavation spans were tested against a range of rock mass Q values and assessed on Barton's Q support chart. The Q values tested by SRK include the 10^{th}, 50^{th} and 90^{th} percentile Q values generated from logging. The excavations categories assessed included access crosscuts, ore drives and undercut drives.

WAI notes that the maximum excavation span tested by SRK was 2.5m. This differs from excavation spans recommended by Tetra Tech; summarised in Table 15.3, below.

Table 15.3: Q Parameters (Derived from footwall Q' values)

Excavation Category	Span (m)	Height (m)
Access Decline	3.8	3.2
Remuck Bay	4.5	4.5
On-Vein Drive	3.2	3.0
Level Access Drive	3.0	3.0

WAI has compared the spans presented in Table 15.3 against the rock mass parameters utilised by SRK. An updated Q support chart is presented in Figure 15.9, below.



REINFORCEMENT CATEGORIES:

1) Unsupported
2) Spot bolting
3) Systematic bolting
4) Systematic bolting, (and unreinforced shotcrete, 4 – 10 cm)
5) Fibre reinforced shotcrete and bolting, 5 – 9 cm
6) Fibre reinforced shotcrete and bolting, 9 – 12 cm
7) Fibre reinforced shotcrete and bolting, 12 -15 cm
8) Fibre reinforced shotcrete, >15 cm, reinforced ribs of concrete shotcrete and bolting
9) Cast concrete lining

Figure 15.9: Q Support Chart (UG Development)

Most of the excavation categories plot within the unsupported zone of the Q support chart for the given range of Q values. A combination of systematic bolting and shotcreting may be required for excavations located in poorer rock mass conditions and must be assessed on an operational basis. For the purposes of ground support cost estimation, WAI have assumed that 20% of the excavations will require support.

15.3.4 Mangazeisky North Deposit

15.3.4.1 Geotechnical Data

Limited geotechnical data is available for the Mangazeisky North deposit. Rock mass strength parameters have been assumed equivalent to those at the Vertikalny deposit.

15.3.4.2 Rock Mass Structure

The Mangazeisky North rock mass consists of interbedded siltstone, sandstone and argillite. Geological descriptions suggest that the area is dominated by a north-north west south-south east striking anticlinal fold. Bedding planes and mineralisation are noted to dip between 20 and 40° towards the East. A generalised cross-section through the Mangazeisky North deposit is presented in Figure 15.10, below.

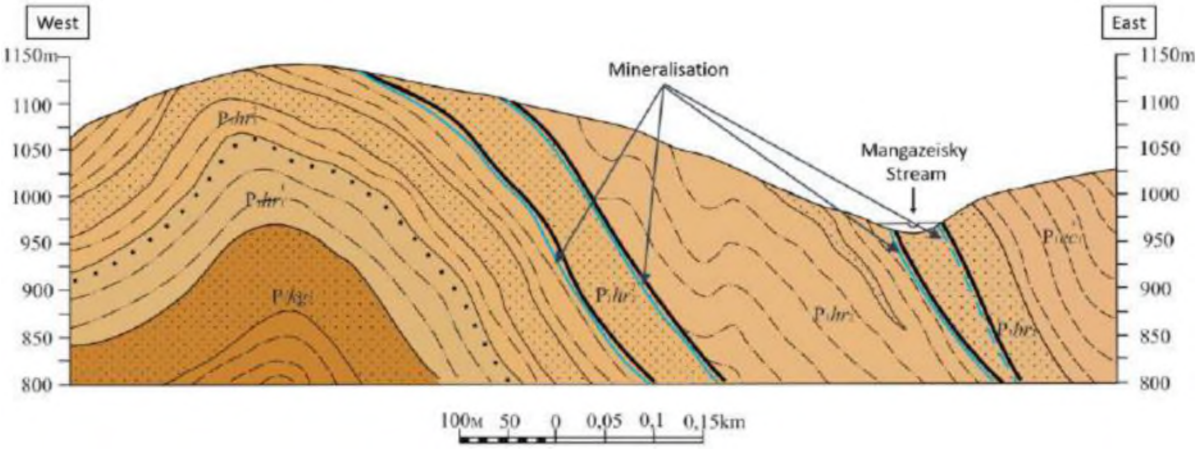


Figure 15.10: Mangazeiksy North Rock Mass Cross-Section

15.3.4.3 Proposed Pit Design Criteria

A summary of the design criteria proposed by WAI for pit optimisation and design is provided in Table 15.4, below.

Table 15.4: Pit Design Parameters			
Bench Height	**Bench Face Angle**	**Berm Width**	**Inter-Ramp Angle**
(m)	**(°)**	**(m)**	**(°)**
10	70	6.4	45

These parameters are based on a standard WAI base case and have not been determined from geotechnical analysis. The parameters may not present an optimal set of criteria and should be treated as indicative only.

15.4 Net Smelter Return Model

The Vertiklany and Mangazeisky North deposits are polymetallic with the main elements being silver, lead and zinc.

The current ore processing circuit is optimised for oxide mineralisation only and produces silver as the sole product. A key strategic consideration is the potential implementation of a flotation plant capable of processing the sulphide mineralisation. Three products would be produced from such a plant:

- Zinc concentrate;
- Lead concentrate; and
- Silver (from Lead/Silver middlings).

A basic net smelter return (NSR) calculation was performed which considered grade, metal price, metallurgical recovery, and metal payability. The payable metal includes the applicable concentrate and refining charges but does not include price participation or penalty element payments. The metal price assumptions were derived by WAI and approved by SBR. All metallurgical recoveries/costs used in the NSR calculation are based on data provided by SBR.

WAI notes that only the sulphide blocks consider the value contributions of each payable element. This is based on the premise that most of the sulphide blocks will be processed through a flotation plant; following depletion of the oxide blocks which form a relatively contiguous volume within the current Vertikalny pit. Oxide blocks only considered the value contribution of silver.

The NSR model forms a critical input into the development of this mining study and further detail regarding the NSR inputs must be understood to enhance the confidence of the study.

15.4.1 NSR Factors

NSR factors were calculated and directly applied to each block within the Resource block models enabling the subsequent mine optimisation exercises to be carried out on the block NSR values. The inputs and calculations used to derive the NSR factors are presented below.

SULPHIDE NSR ASSESSMENT

Feed

Parcel	1000	kg
Ag	1000	g/t
Pb	2.03	%
Zn	1.73	%

Metal Prices — **SP ANGEL (27.08.19)**

Ag	17.76	US$/tOz
Pb	2,069	US$/t
Zn	2,252	US$/t

Charges

Transport	274.9	US$/tconc
Treatment	0	US$/tconc
Refining	0.4	US$/tOz

ZINC CONCENTRATE

Mill Recovery

Zn	82.2	%
Ag	4.7	%

Contained Metal

Zn	14.2	kg
Ag	47.0	g

Concentrate

Zn	42.3	%
Ag	1398	g/t
Mass	33.6	kg

	Zn	Ag
Deductions	0 %	0 g
Payability	45 %	45 %
Value	**14.41 US$/tORE**	**12.08 US$/tORE**
Transport Cost	9.24 US$/tORE	0.00 US$/tORE
Treatment Cost	0.00 US$/tORE	0.00 US$/tORE
Refining Cost	0.00 US$/tORE	0.60 US$/tORE
Total Costs	**9.24 US$/tORE**	**0.60 US$/tORE**
Value (Less: Total Costs)	**5.17 US$/tORE**	**11.47 US$/tORE**
Ore:Concentrate	29.75	29.75
Conc. Value	153.77 US$/tCONC	341.25 US$/tCONC
Feed Grade	1.73 %	1000 g/t
NSR Factor	2.99 US$ / % / t	0.011 US$ / g / t
NSR for 1t of Ore from Zn Conc.	**16.64 US$/tORE**	

LEAD CONCENTRATE

Mill Recovery

Pb	65.9	%
Ag	65.0	%

Contained Metal

Pb	13.4	kg
Ag	650.0	g

Concentrate

Pb	17.1	%
Ag	8309	g/t
Mass	78.2	kg

	Pb	Ag
Deductions	0 %	0 g
Payability	84 %	84 %
Value	**23.25 US$/tORE**	**311.76 US$/tORE**
Transport Cost	21.51 US$/tORE	0.00 US$/tORE
Treatment Cost	0.00 US$/tORE	0.00 US$/tORE
Refining Cost	0.00 US$/tORE	8.36 US$/tORE
Total Costs	**21.51 US$/tORE**	**8.36 US$/tORE**
Value (Less: Total Costs)	**1.74 US$/tORE**	**303.41 US$/tORE**
Ore:Concentrate	12.78	12.78
Conc. Value	22.29 US$/tCONC	3878.27 US$/tCONC
Feed Grade	2.03 %	1000 g/t
NSR Factor	0.86 US$ / % / t	0.303 US$ / g / t
NSR for 1t of Ore from Pb Conc.	**305.15 US$/tORE**	

LEAD/SILVER MIDDLINGS

Mill Recovery		
Ag	15.6	%
Contained Metal		
Ag	156	g
Payability	98	%
Value	**87.29**	**US$/tORE**
Refining Cost	**2.01**	**US$/tORE**
NSR for 1t of Ore from Ag/Pb M	**85.29**	**US$/tORE**
NSR Factor	**0.09**	**US$ / g / t**

SULPHIDE NSR FACTORS

Ag	**0.40**	**US$ / g / t**
Pb	**0.86**	**US$ / % / t**
Zn	**2.99**	**US$ / % / t**

Total NSR	**407.08 US$/tORE**

NOTE: Concentrate assumed at 0% moisture.

OXIDE	NSR ASSESSMENT		
Feed			
Parcel		1000	kg
Ag		1000	g/t
Pb		2.03	%
Zn		1.73	%
SILVER PRECIPITATE			
Mill Recovery			
	Ag	85	%
Contained Metal			
	Ag	850	g
Payability		98	%
Value		475.64	US$/$t_{ORE}$
Refining Cost		10.93	US$/$t_{ORE}$
Value (Less:Costs)		**464.71**	**US$/$t_{ORE}$**
OXIDE NSR FACTOR		**0.46**	**US$ / g / t**

15.5 Mineable Inventories

Table 15.5 to Table 15.7 summarise the mineable inventories for all areas.

Table 15.5: Vertikalny Open Pit

Rock Type	Economic Cut-Off	Classification	Tonnage
Oxide Material	Above Cut-Off NSR>= 117.00 US$/t	Measured	58,850
		Indicated	113,178
		Inferred	-
	Total		172,028
	Below Cut-Off NSR<117.00 US$/t	Measured	16,587
		Indicated	19,847
		Inferred	-
	Total Oxide	Measured	75,436
		Indicated	133,025
		Inferred	-
Sulphide Material	Above Cut-Off NSR>= 113.06 US$/t	Measured	11,405
		Indicated	75,378
		Inferred	7,443
	Below Cut-Off NSR<113.06 US$/t	Measured	1,748
		Indicated	21,319
		Inferred	454
	Total Sulphide	Measured	13,153
		Indicated	96,697
		Inferred	7,897
Total Above Cut-Off			258,811
Total Mineable Inventory			**326,208**

Table 15.6: Vertikalny Underground Material				
Rock Type	**Economic Cut-Off**	**Classification**	**Mineralised Tonnage**	**Mineralised Tonnage with Planned Dilution**
Stope	Stope Cut-Off NSR>= 142 US$/t	Measured	-	-
		Indicated	255,966	291,124
		Inferred	287,080	326,512
		Sub-Total	**543,046**	**617,636**
On-Vein Drive	No Cut-Off	Measured	-	-
		Indicated	67,366	117,610
		Inferred	65,239	113,897
		Sub-Total	**132,604**	**231,507**
Total		Measured	-	-
		Indicated	323,332	408,735
		Inferred	352,319	440,409
		Total	**675,650**	**849,144**

* Unplanned dilution (10%) and mining recovery (90%) not applied to stopes
* Planned dilution (waste within stopes and on-vein drives) added to mineralised tonnes on pro rata basis.

Table 15.7: Mangazeisky North Open Pit			
Sulphide Material	Above Cut-Off NSR>= 113.06 US$/t	Measured	-
		Indicated	-
		Inferred	280,805
	Below Cut-Off NSR<113.06 US$/t	Measured	-
		Indicated	-
		Inferred	58,463
		Measured	-
		Indicated	-
	Total	**Inferred**	**339,268**

The mineable inventories represent all resources that have the potential to be economic in the future as upside in the scoping study for long term financial forecasting. WAI has and based conceptual open pit designs and a combined conceptual design on the in-pit and underground inventories.

The in-pit MRE is based on a set of cost parameters supplied by the Client which align with its actual current costs of production, G&A ($60/t) and oxide processing costs ($72.91/t). The wireframe resource model was done at 50g/t Ag COG and includes mineralisation with grade between 75-240g/t Ag (and which is the subject of using XRT separation) so all potentially economic mineralisation is captured. These resources are at a satisfactory level of confidence that best reflect the economic conditions under the set of parameters given by SBR and, as there has been no addition of evaluation data since, best reflect the current state of SBR's resources.

The mineable inventory (tonnage) estimate is based on a more optimistic set of cost parameters developed downstream to the MRE and considered for the future conceptual design. The latest mineable inventory estimates utilise optimistic optimisation cost parameters; G&A at $40/t and oxide processing at $50/t. The main differences as a result of the different optimisation parameters are that the volumes reported in the MRE (lower estimate) are based on a physically smaller set of pit shells and higher cut-off grades due to the higher optimisation costs.

Consequently, the MRE and mineable inventory estimates for Vertikalny cannot be directly compared but Table 15.5 lends to a broad comparison. The entire open-pit inventory for Vertikalny (green) numbers adds up, bar rounding, to the entire inventory included in the Financial Model (Appendix C) without including any inventory from stockpiles. The (red) numbers in Table 15.5 above the NSR cut-off we get (258kt) effectively correlates with the larger optimization shell at 50g/t Ag CoG in the MRE (Table 13.27), which would be expected with the larger pit shell used for the mineable inventory.

Considering Vertikalny underground, the MRE represents a set of underground operating parameters applied to block grades below the open pit shell and classified accordingly as potentially economic. It does not include a design or development whereas the mineable inventory does incorporate a stope optimiser to simulate stoping and considers development. The MRE uses a higher cut-off and break even reflecting the current operating costs and G&A. The mineable inventory is based on a break-even for the stopes using the more optimistic operating costs.

In Table 15.6 attached splitting the inventory into M & I + Inf, the indicated material at 300g/t cut-off grade roughly corresponds with the indicated for **undiluted** mineralisation NSR>$124/t (256kt orange) which is a reasonable approximation to the MRE (236kt M&I shown in Table 13.28). The main difference is the mineable inventory also includes development at zero cut-off (231kt), inferred material and planned loss and dilution – hence the higher tonnage, given rounding in calculations, approximating to 840kt.

WAI does not see any need to adjust in-pit or underground parameters for the MRE to reflect the more optimistic parameters as the original conditions supplied by SBR in November 2019 still best reflect the operating conditions of the mine and does not exclude material critical to the project assumptions. The schedule and combined design is conceptual and not based on reserves.

15.6 Open Pit Optimisation

15.6.1 Overview

WAI has carried out open pit optimisation for the Vertikalny and Mangazeisky North deposits using the Datamine NPV Scheduler v4 (NPVS) software package.

The pit optimisations were carried out on the resource block models generated for the two deposits and driven on the calculated block NSR values. Optimisations were driven on *Measured, Indicated* and *Inferred* resources.

NPVS utilises the Lerchs-Grossmann (LG) algorithm to produce a pit shell yielding the highest undiscounted profit; subject to a fixed set of selling prices (NSR values), mining costs, processing costs and slope angle constraints. NPVS provides the ability to parametrise the commodity selling price (NSR values) and run successive applications of the LG algorithm to generate a sequence of nested pit shells; commonly known as LG phases.

15.6.2 Vertikalny Deposit

15.6.2.1 Optimisation Parameters

A breakdown of the costs and parameters used in the Vertikalny deposit pit optimisation are presented in Table 15.8, below.

Table 15.8: Optimisation Input Parameters

Parameter	Unit	Value	Source
Milling Rate	ktpa	180	SBR – Planned rate
Discount Rate	%	8	WAI Estimate
Mining Cost	US$/t	2.53	SBR Estimate
Processing Cost			
Oxides	US$/t ore	50.00	SBR Estimate
Sulphides	US$/t ore	46.97	SBR Estimate
G&A	US$/t ore	40.00	SBR Estimate
Mining Recovery	%	95%	Tetra Tech
Mining Dilution	%	30%	Tetra Tech
Slope Angles			
Hanging Wall	°	56	SRK
Footwall	°	48	SRK

WAI has not carried out an independent review of the optimisation parameters. All optimisation cost parameters were provided by SBR.

15.6.2.2 NSR Cut-Off Calculation

NPVS was used to calculate a marginal NSR cut-off using the parameters presented in the section above. The marginal NSR cut-off grade is the NSR value at which the revenue generated from a block is equal to the cost of processing it. The calculated cut-offs per rock type are as follows:

Oxide Material = $117.00/t
Sulphide Material = $113.06/t

NPVS uses the calculated marginal cut-offs to delineate ore and waste blocks within the block model. Waste blocks are assigned a net value equal to the cost of mining the block as waste, whereas ore blocks are assigned a net value equal to the revenues generated from the block, less the associated costs of production. The resulting 'net value' model is used by NPVS to determine the optimal mining envelopes; details of which are presented in the following sections.

15.6.2.3 Optimal Shell

A summary of the in-situ tonnages and grades contained within the selected optimal pit shell is provided in Table 15.9, below.

Table 15.9: Vertikalny In-situ Pit Shell Physicals

Parameter	Units	Value
Oxide Material	kt	205
Ag Grade	*g/t*	*973*
Sulphide Material	kt	158
Ag Grade	*g/t*	*1,040*
Pb Grade	*%*	*2.31*
Zn Grade	*%*	*2.29*
Total Mineralised Tonnes (Oxide + Sulphide)	kt	362
Oxide Material (Below Cut-Off) *(NSR<117.0 US$/t)*	Kt	36.8
Sulphide Material (Below Cut-Off) *(NSR<113.06 US$/t)*	kt	26.4
Waste	kt	8,300
Strip	$t_W:t_O$	23.1

Note:
- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Oxide material processed through oxide circuit; as such Pb/Zn are not recovered and are not reported.
- Tonnage and grade figures may not reconcile due to rounding.
- Mining dilution and recovery **not** applied.
- Figures effective as of 01.11.19.
- Strip ratio inclusive of below cut-off material:
  Strip Ratio = (Waste + Oxide Material Below Cut-off + Sulphide Material Below Cut-off) / Total Mineralised Tonnes

15.6.3 Mangazeisky North Deposit

15.6.3.1 Optimisation Parameters

A breakdown of the costs and parameters used in the Mangazeisky North deposit pit optimisation are presented in Table 15.10, below.

Table 15.10: Optimisation Input Parameters

Parameter	Unit	Value	Source
Milling Rate	ktpa	115	Current Rate
Discount Rate	%	8	WAI Estimate
Mining Cost	US$/t	2.53	SBR
Processing Cost			
Sulphides	US$/t ore	46.97	SBR
G&A	US$/t ore	60.00	SBR
Mining Recovery	%	95%	Tetra Tech
Mining Dilution	%	30%	Tetra Tech
Slope Angles	°	45	WAI Estimate

Note:
- Only sulphide processing costs applied as no oxide material modelled in resource model.

WAI has not carried out an independent review of the optimisation parameters. All optimisation cost parameters were provided by SBR.

15.6.3.2 NSR Cut-Off Calculation

The calculated NSR cut-off for the Mangazeiksy North deposit is summarised below.

Sulphide Material = $113.06/t

15.6.3.3 Optimal Shell

A summary of the in-situ tonnages and grades contained within the Mangazeisky North optimal shell is provided in Table 15.11, below.

Table 15.11: Mangazeisky North In-situ Pit Shell Physicals

Parameter	Units	Pit Shell 38
Sulphide Material	kt	311
Ag Grade	*g/t*	*775*
Pb Grade	*%*	*10.07*
Zn Grade	*%*	*0.98*
Sulphide Material Below Cut-Off *(NSR<113.06 US$/t)*	kt	40.0
Waste	kt	8,890
Strip	t_W:t_O	28.7

NOTE:
- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Tonnage and grade figures may not reconcile due to rounding.
- Mining dilution and recovery **not** applied.
- Strip ratio inclusive of below cut-off material:
  Strip Ratio = (Waste + Sulphide Material Below Cut-off) / Total Mineralised Tonnes.

15.7 Open Pit Design

15.7.1 Vertikalny Conceptual Pit Design

15.7.1.1 Pit Design Parameters

A summary of the parameters used in the Vertikalny pit designs is presented in Table 15.12, below.

Table 15.12: Vertikalny Open Pit Design Parameters

Parameter	Units	Value	Source
Bench Height	m	20	SBR
Bench Face Angle	°	70	SBR
Berm Width	m	11	SBR
Ramp Width (Single/Double)	m	12.5/17.016	SBR
Ramp Gradient	%	8	SBR
Min. Working Width Final Benches	m	16	SBR

15.7.1.2 Pit Design

Two individual pits have been designed along the strike of the Vertikalny deposit; in-line with the selected optimal pit shell. The portion of the pit shell extracting material from the south-eastern extent of the mineralised zone intercepts a hillside. A conceptual cut & fill (CAF) road has been designed along the hillside to provide an initial indication of the access requirements to this area.

Plan, sectional and isometric views of the generated pit designs are presented in Figure 15.11 to Figure 15.13 below.

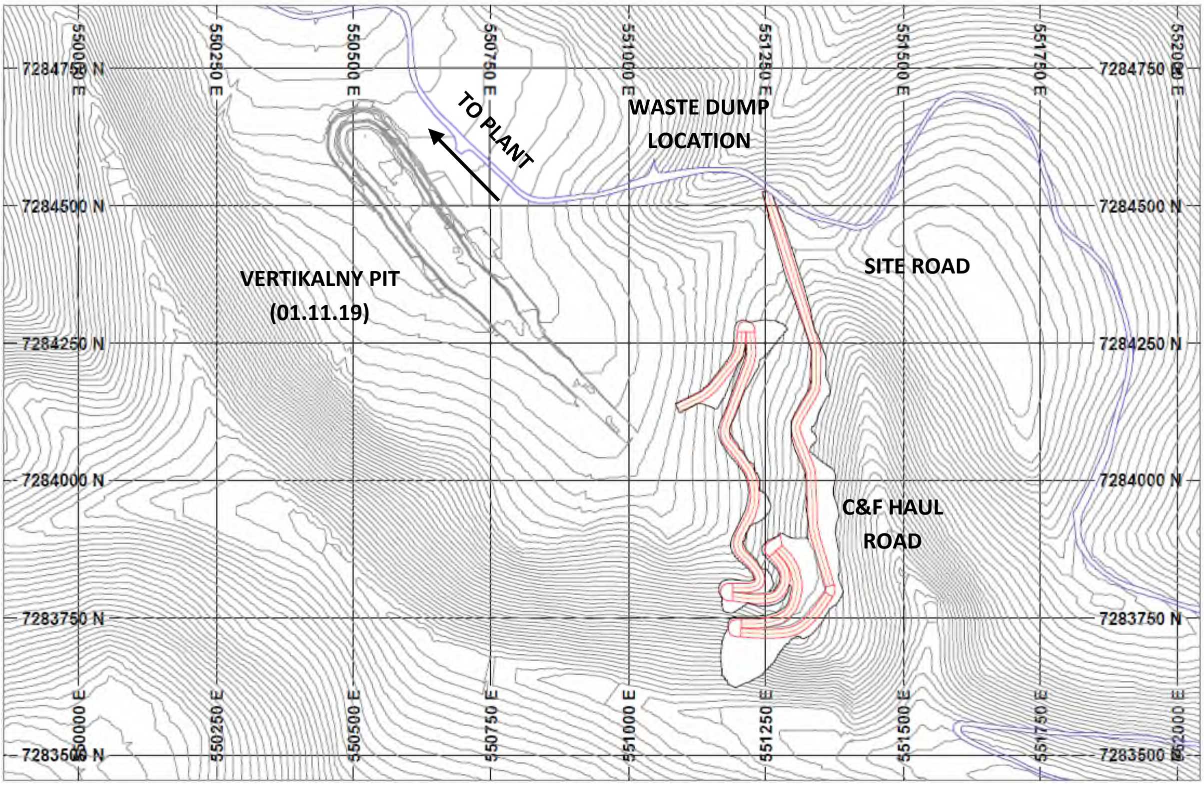


Figure 15.11: Vertikalny Cut & Fill Road

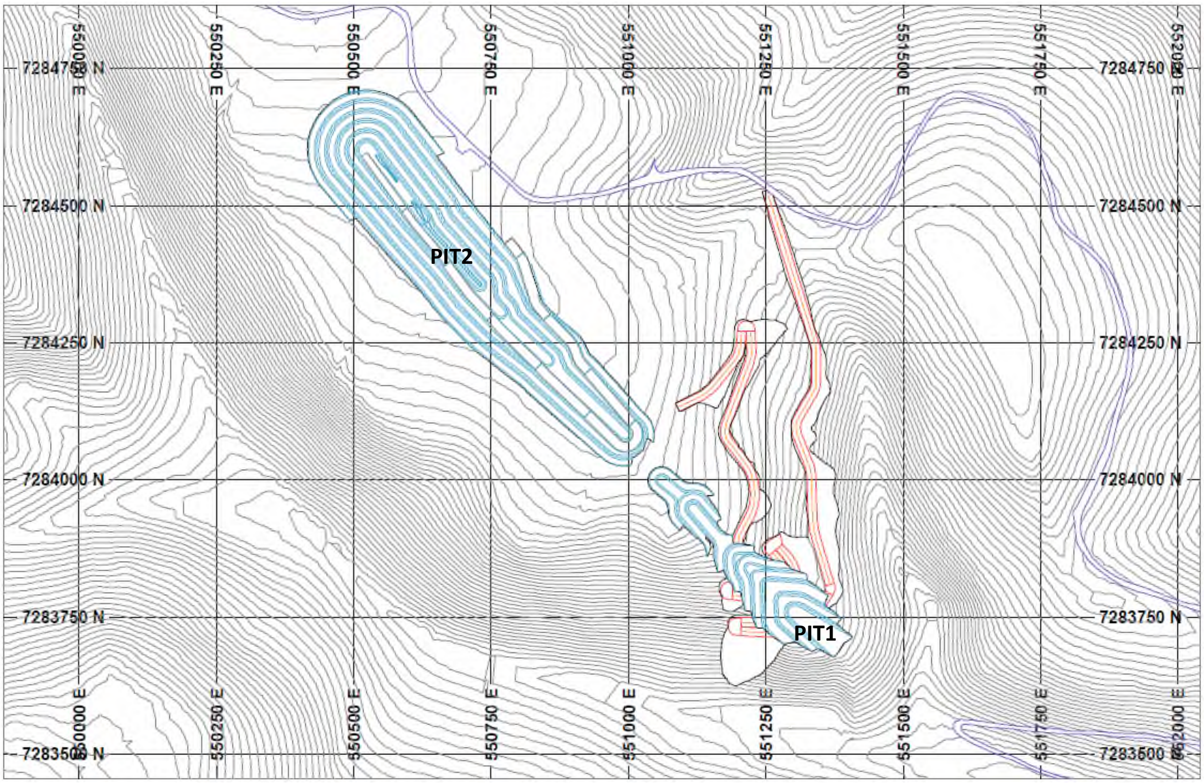


Figure 15.12: Vertikalny Conceptual Pit Design

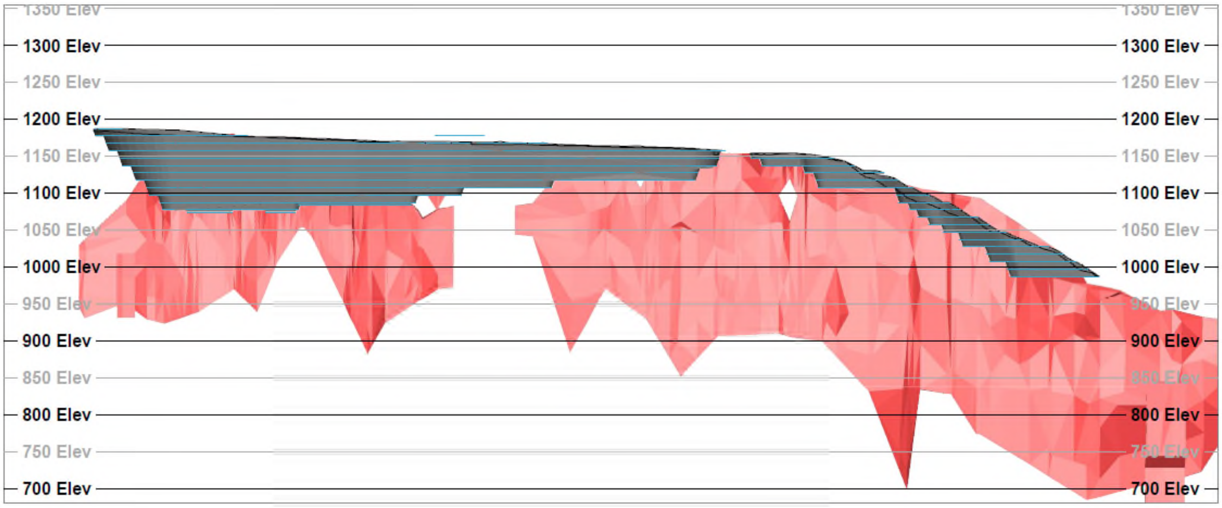


Figure 15.13: Vertikalny Conceptual Pit Design -Sectional View

The volume of cut material required to prepare the CAF road is estimated at 169,000m^3. CAF road designs are conceptual only and may not be representative of the final access requirements.

A summary of tonnages and grades contained within the conceptual pit designs is provided in Table 15.13, below.

Table 15.13: Vertikalny Conceptual Pit Design Physicals (Dilution & Recovery Applied)		
Parameter	**Units**	**Value**
Oxide Material	kt	212
Ag Grade	*g/t*	*800*
Sulphide Material	kt	116
Ag Grade	*g/t*	*846*
Pb Grade	*%*	*1.70*
Zn Grade	*%*	*1.66*
Total Mineralised Tonnes (Oxide + Sulphide)	kt	329
Oxide Material Below Cut-Off *(NSR<117.00 US$/t)*	kt	45.0
Sulphide Material Below Cut-Off *(NSR<113.06 US$/t)*	kt	29.0
Waste	kt	11,000
Strip	$t_w:t_o$	33.7

Note:
- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Oxide material processed through oxide circuit; as such Pb/Zn are not recovered and are not reported.
- Volume, tonnage and grade figures may not reconcile due to rounding.
- Mining dilution (30%) and mining recovery (95%) applied.
- Strip ratio inclusive of below cut-off material:
  Strip Ratio = (Waste + Oxide Material Below Cut-off + Sulphide Material Below Cut-off) / Total Mineralised Tonnes
- Figures effective as of 01.11.19
- Figures not representative of Ore Reserves (in accordance with JORC 2012)

WAI has not prepared a waste dump design as part of this study. It is assumed that the current waste dump footprint may be extended to accommodate any additional waste material. Waste disposal strategies should be examined in greater detail in further engineering studies. The pit physicals are based on the topographic surface as of November 2019.

15.7.2 Mangazeisky Conceptual Pit Design

15.7.2.1 Pit Design Parameters

A summary of the parameters used in the Mangazeisky North pit design is provided in Table 15.14, below.

Table 15.14: Mangazeisky North Open Pit Design Parameters

Parameter	Units	Value	Source
Bench Height	m	10	WAI Estimate
Bench Face Angle	°	70	WAI Estimate
Berm Width	m	6.4	WAI Estimate
Ramp Width	m	16	Tetra Tech
Min Ramp Width	m	10	Tetra Tech
Min. Working Width Final Benches	m	<10	Tetra Tech

15.7.2.2 Conceptual Pit Design

The Mangazeisky North deposit is situated some 6.5km NNW of the Vertikalny Pit. The pit shell is located on the brow of a hill approximately 130m above the valley floor. A conceptual cut & fill road was designed along the hillside to provide an indication of pit access requirements.

Plan, sectional and isometric views of the generate pit designs are presented in Figure 15.14 and Figure 15.16, below.

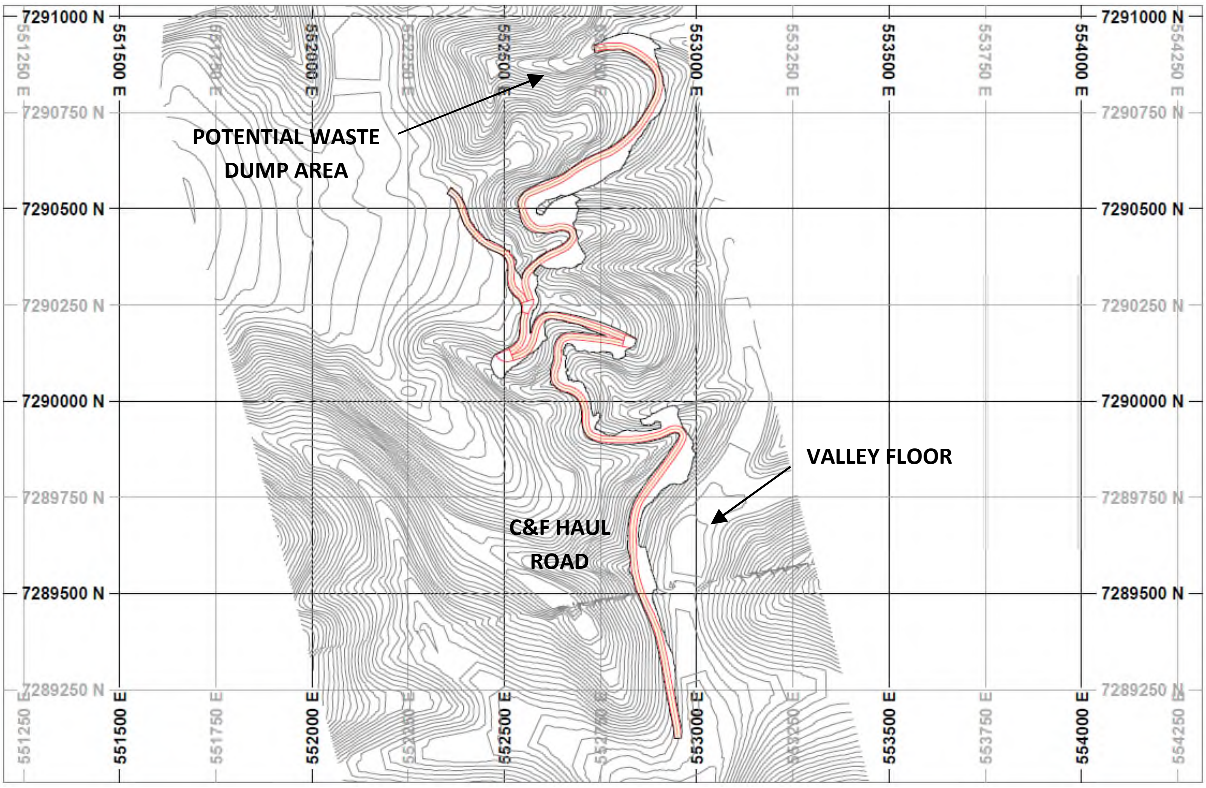


Figure 15.14: Mangazeisky Cut & Fill Road

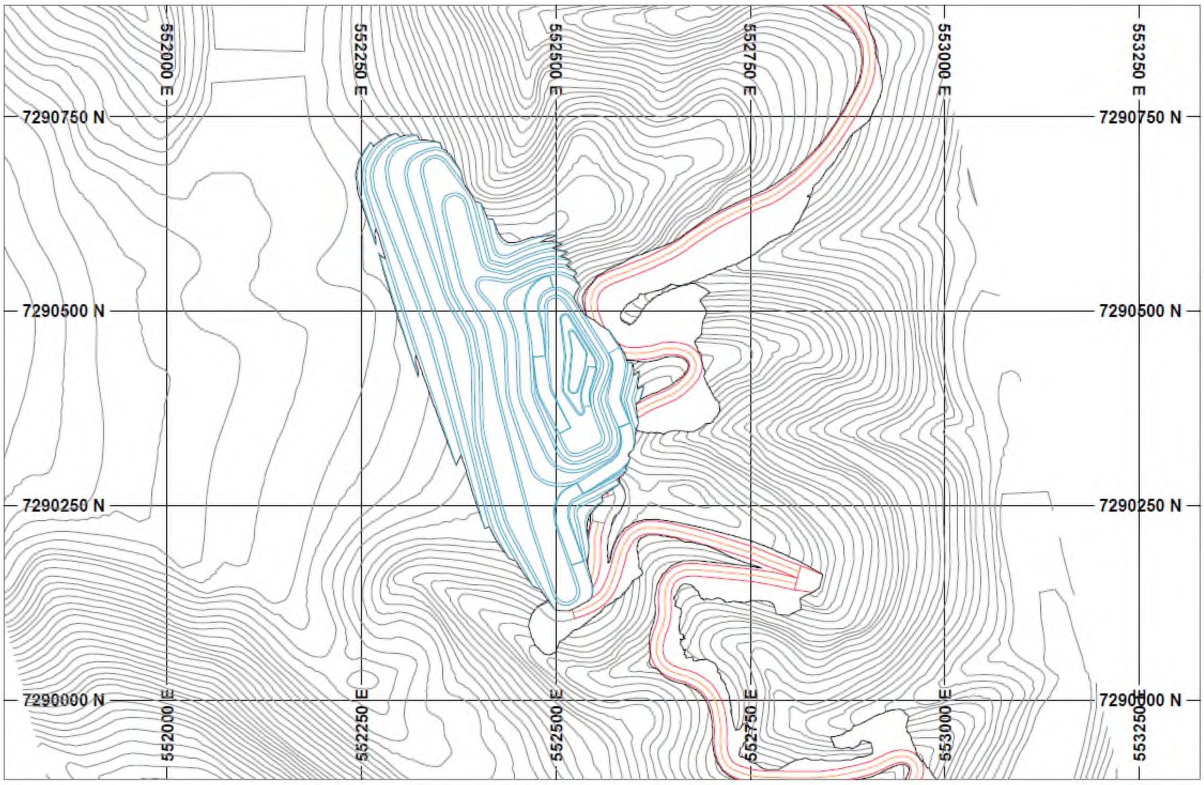


Figure 15.15: Mangazeisky North Conceptual Pit Design

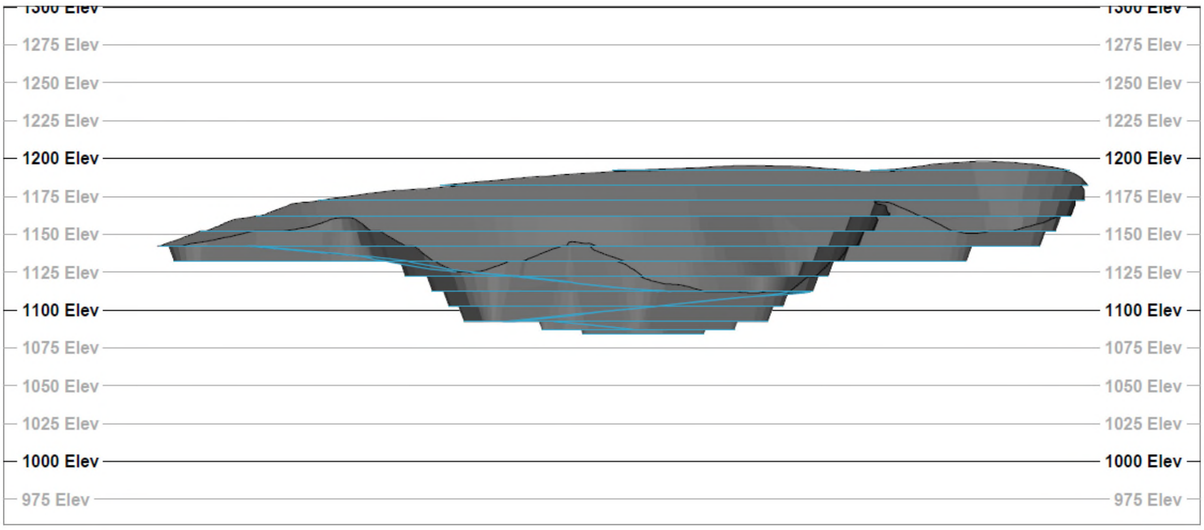


Figure 15.16: Mangazeisky North Conceptual Pit Design – Section View

The volume of cut material required to prepare the CAF road is estimated at 175,000m^3. CAF road designs are conceptual only and may not be representative of the final access requirements.

A summary of tonnages and grades contained within the conceptual pit design is provided in Table 15.15, below

Table 15.15: Mangazeisky Conceptual Pit Design Physicals (Dilution & Recovery Applied)

Parameter	Units	Value
Sulphide Material	kt	347
Ag Grade	*g/t*	*570*
Pb Grade	*%*	*7.47*
Zn Grade	*%*	*0.82*
Sulphide Material Below Cut-Off *(NSR<113.06 US$/t)*	kt	72.2
Waste	kt	8,540
Strip	$t_W:t_O$	24.8

Note:
- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Volume, tonnage and grade figures may not reconcile due to rounding.
- Mining dilution (30%) and mining recovery (95%) applied.
- Strip ratio inclusive of below cut-off material:
  Strip Ratio = (Waste + Sulphide Material Below Cut-off) / Total Mineralised Tonnes
- Figures not representative of Ore Reserves (in accordance with JORC 2012)

WAI has not prepared a waste dump design as part of this study. Waste disposal strategies should be evaluated in greater detail in further engineering studies.

15.8 Underground Mining

15.8.1 Underground Mining Method

WAI propose to maintain the mining methodology outlined by Tetra Tech; mechanised sub-level open stoping (SLOS). The method offers favourable results in safety, cost and dilution control as outlined by Tetra Tech. Stopes will be extracted in a retreat, top-down sequence, with adequate in-situ rock pillars left unmined for localised and regional stability.

15.8.2 NSR Cut-Off

The NSR of each potential mining block was evaluated against a break-even economic cut-off value. The economic cut-off considers the cost of mining, processing and the general and administrative costs.

Mining blocks with an average NSR value above the economic cut-off, that have defined access, and are not isolated (i.e. mining blocks that do not pay for the development of those blocks) are included in the mine design. Mining blocks that do not meet the criteria above are disregarded.

A summary of the parameters used in the calculation of the breakeven NSR cut-off is provided in Table 15.16, below.

Table 15.16: NSR Cut-Off Parameters			
Parameters	**Units**	**Value**	**Comment**
Mining Cost	US$/$t_{ore}$	55.00	TetraTech - Calculated operating cost
Processing Cost	US$/$t_{ore}$	46.97	SBR
G&A	US$/$t_{ore}$	40.00	SBR
NSR Cut-Off	**US$/$t_{ore}$**	**142.00**	

Only sulphide processing costs have been considered as most of the potential stope material will be situated within primary mineralisation.

15.8.3 Stope Optimisation

15.8.3.1 Optimisation Parameters

Underground mineable tonnage estimates were prepared using the Vertikalny Resource block model as the basis for stope optimisation.

Stope shapes were defined using the Datamine Mineable Shape Optimiser (MSO) module. MSO generates a set of practical stope shapes around a geological block model in accordance with a supplied cut-off grade and a set of geometrical constraints. A summary of the input parameters used in stope optimisation is provided in Table 15.17, below.

Table 15.17: Stope Optimisation Parameters			
Parameters	**Units**	**Value**	**Comment**
NSR Cut-Off	US$/$t_{ore}$	142.00	
Level Intervals	m	25	TetraTech
Stope Strike Length	m	10	TetraTech
Minimum Mining Width	m	1.3	TetraTech
Maximum Mining Width	m	4	TetraTech

15.8.3.2 Optimisation Results

A summary of the in-situ stope tonnages and grades is provided in Table 15.18, below.

Table 15.18: Vertikalny In-situ Stope Tonnages & Grade		
Parameter	**Units**	**Value**
Mineralised Material	kt	655
Ag	g/t	569
Pb	%	2.64
Zn	%	2.09
NSR	US$/t	236

Note:
- Figures rounded to 3SF, Pb/Zn grades rounded to 2DP
- The generated stopes contain 92.5kt of waste material which would need to be mined (representative of planned dilution)
- Unplanned dilution and recovery factors (pillar losses, mining recovery etc.) have not been applied.

WAI notes that 2.0% of the stope mineralised tonnes are classified as oxide material. A summary of in-situ stope tonnage resource classification split is presented in Table 15.19, below.

Table 15.19: Vertikalny UG Resource Class Proportions	
Parameter	**Value**
Measured	0%
Indicated	48%
Inferred	52%

The locations of the planned underground mining zones and sectional views of the generate stopes are presented in Figure 15.17 and Figure 15.18, below.

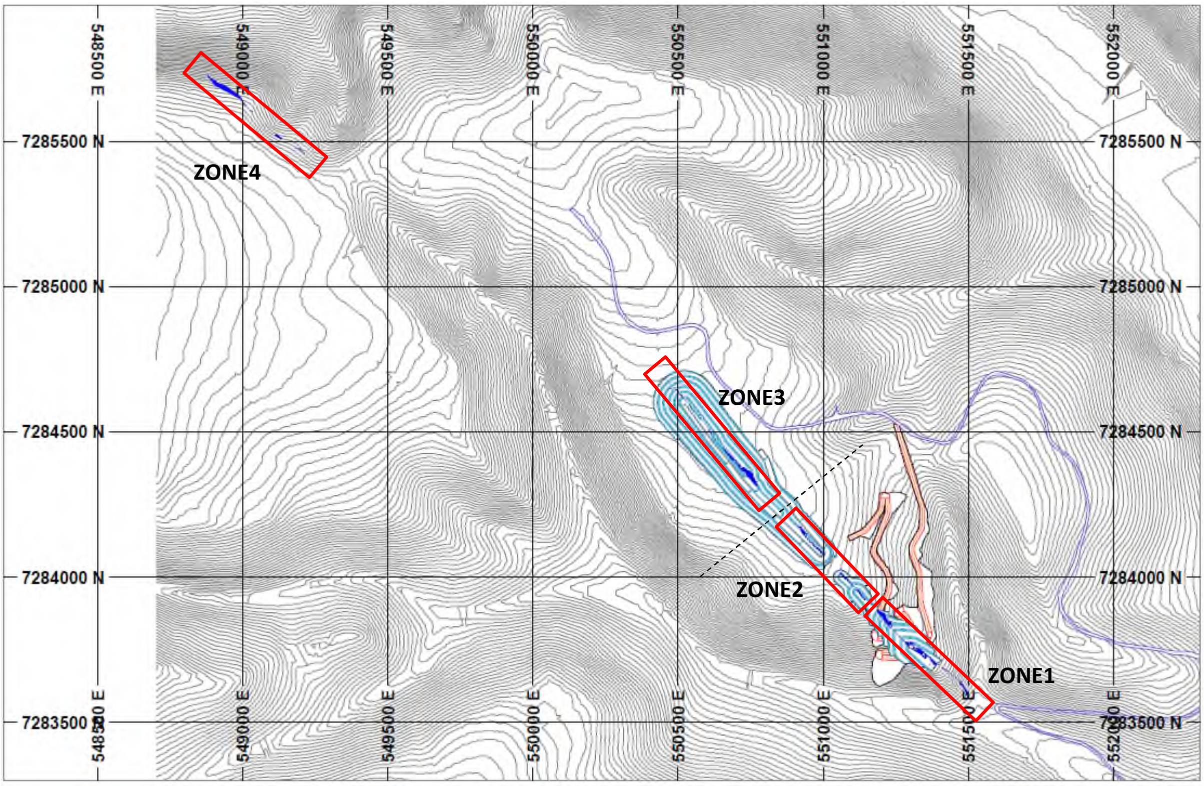


Figure 15.17: Planned Underground Mining Zones

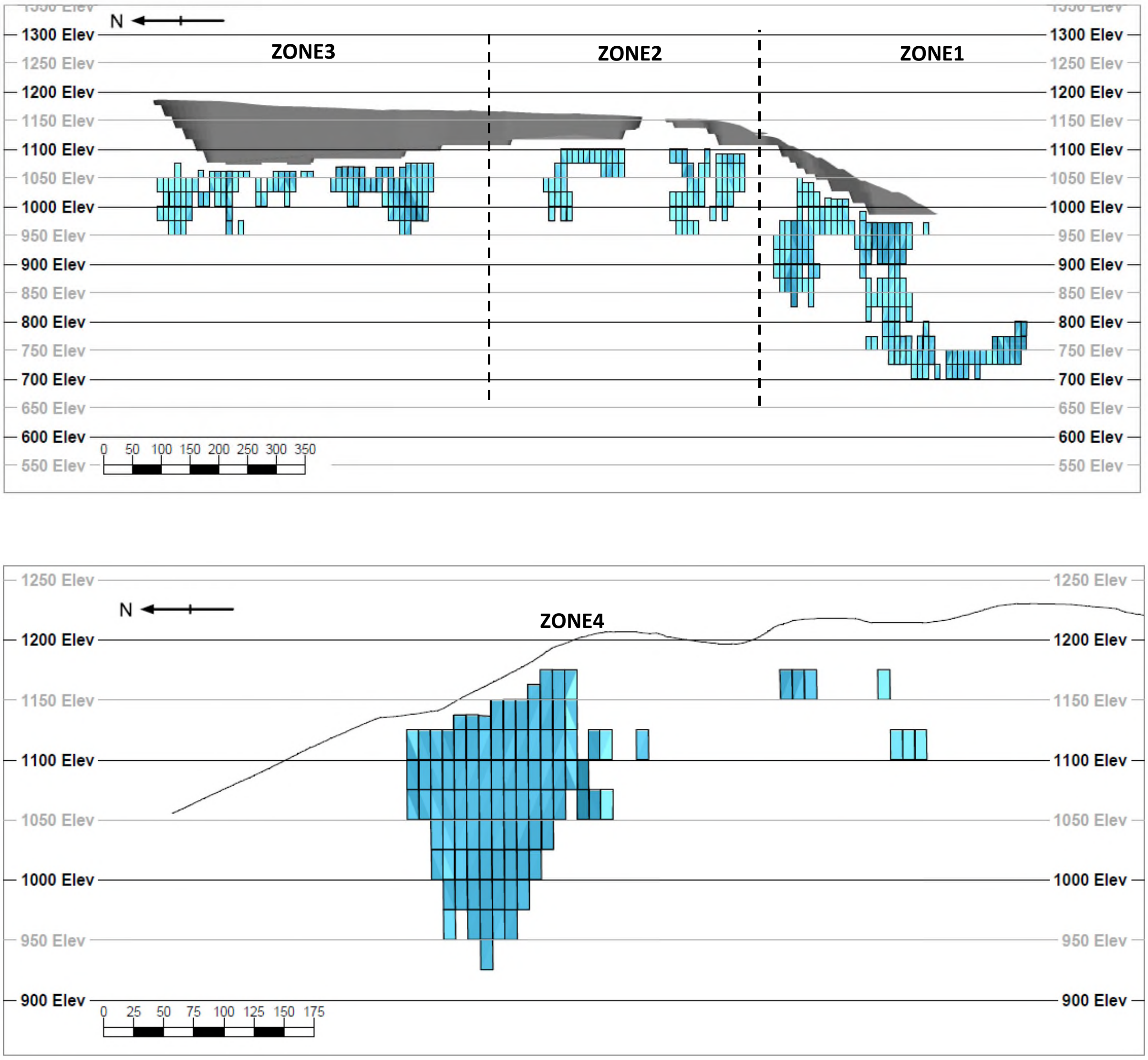


Figure 15.18: Vertikalny Stopes

15.8.4 Underground Mine Design

15.8.4.1 Design Parameters

A total of four underground mining zones were designed in line with the stope zones presented in Section 15.8.3.3.2 (Figure 15.17). The following excavations types were included in the underground development designs:

- Main decline (access);
- Level access drives (drives from the decline to access the ore drives);
- Ventilation drives (ventilation tunnels connecting the waste access crosscuts to the ventilation raises);
- Ventilation raises;
- Remuck bays (stockpile bays 7.5m long); and

- Ore drives (excavations developed along the strike of the mineralised vein to provide access for slot raise and stope drilling).

WAI has maintained the underground mine design parameters implemented within the Tetra Tech study. All mine development was positioned within the footwall of the deposit. The parameters used in underground mine design are summarised in Table 15.20 and Table 15.21, below.

Table 15.20: Underground Design Parameters

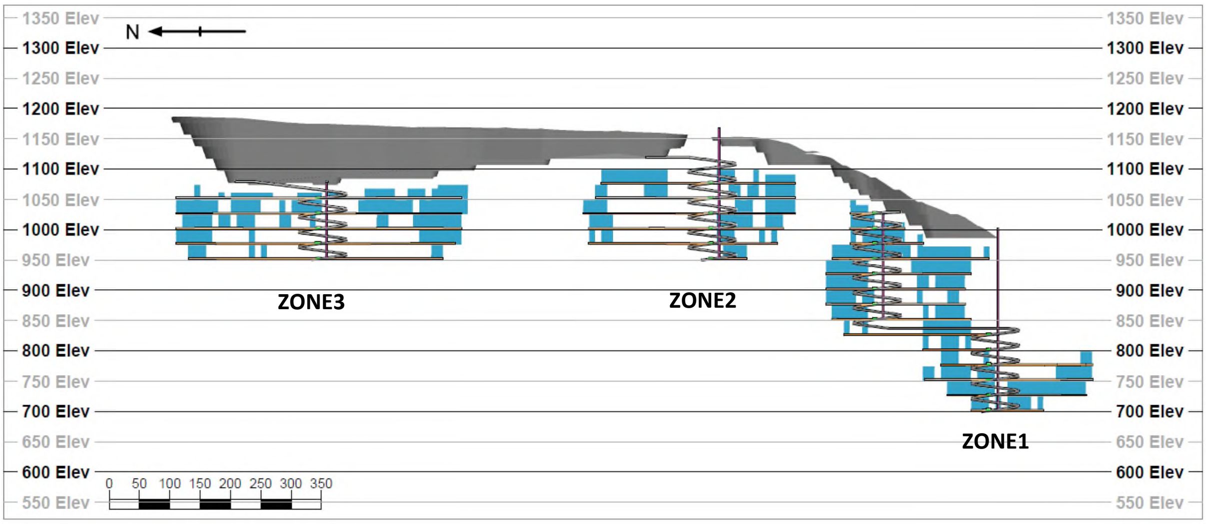
Parameter	Units	Footwall	Source
Level Spacing	m	25	Tetra Tech
Minimum Crown Pillar Depth	m	15	SRK
Decline Gradient	1:N	1:8	WAI Estimate
Decline Turn Radius	m	20	WAI Estimate

Table 15.21: Development Dimensions

Development Class	Dimensions	Area	Source
	mW x mH	m^2	
Decline	3.8 x 3.2 (Arch)	11.71	Tetra Tech
Ventilation & Access Drive	3.0 x 3.0 (Arch)	8.55	Tetra Tech
Remuck Bay	4.5 x 4.5 (Arch)	19.80	Tetra Tech
On Vein Drive	3.2 x 3.0 (Arch)	9.15	Tetra Tech
Ventilation Raise	3.0m (Diameter)	7.07	Tetra Tech

15.8.4.2 Conceptual Underground Mine Designs

Sectional and isometric views of the generated underground mine designs are presented in Figure 15.19**,** Figure 15.20 and Figure 15.21 below.



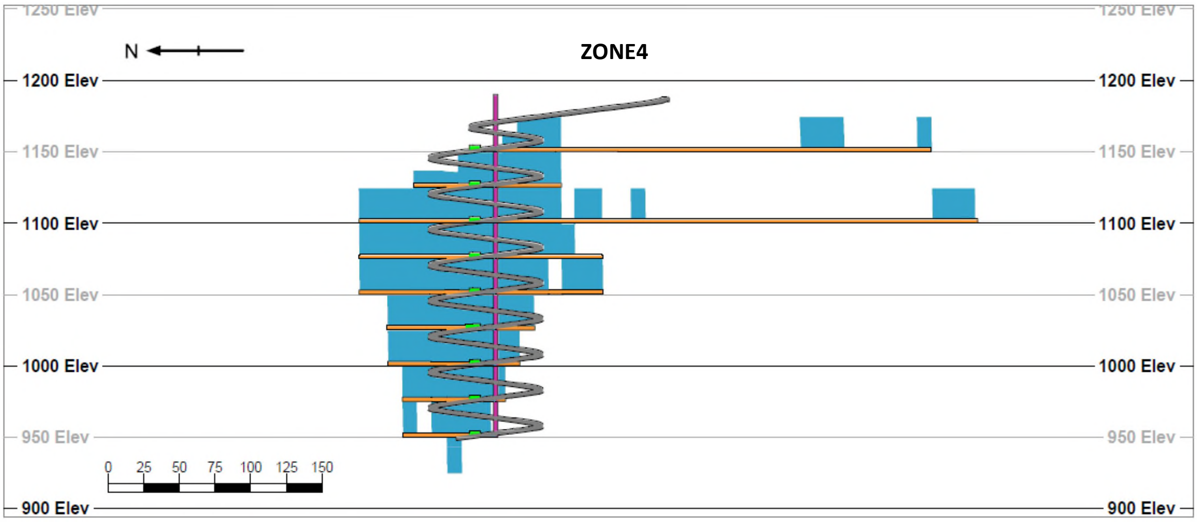


Figure 15.19: Vertikalny Conceptual Underground Mine Design – Sectional View

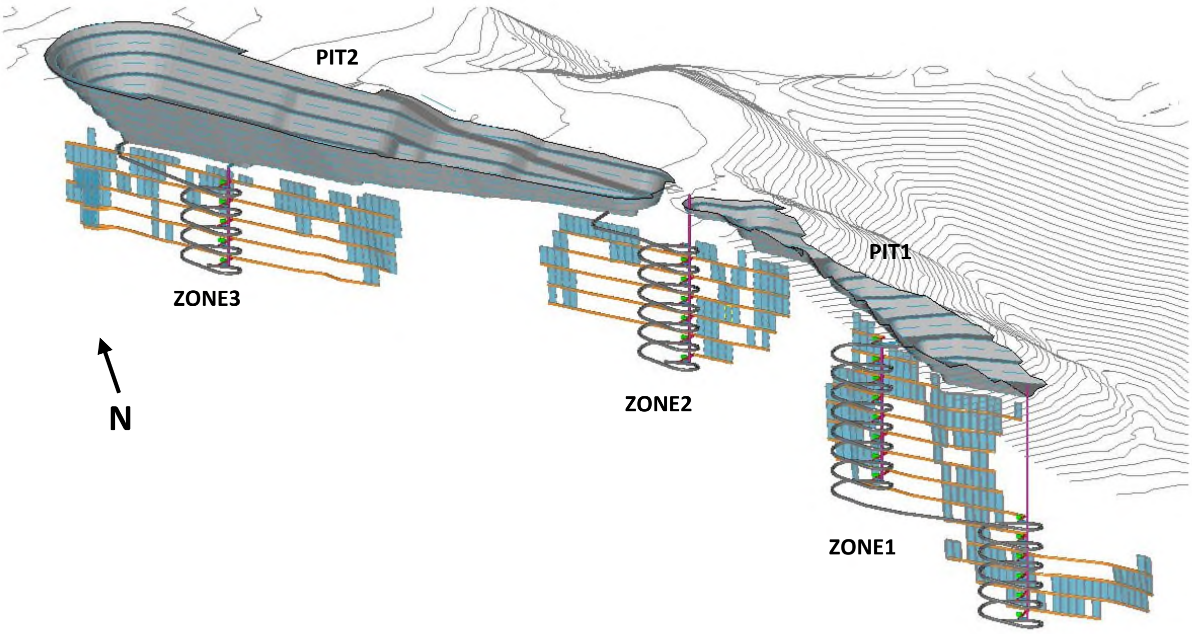


Figure 15.20: Vertikalny Underground Zone 1-3 Isometric View

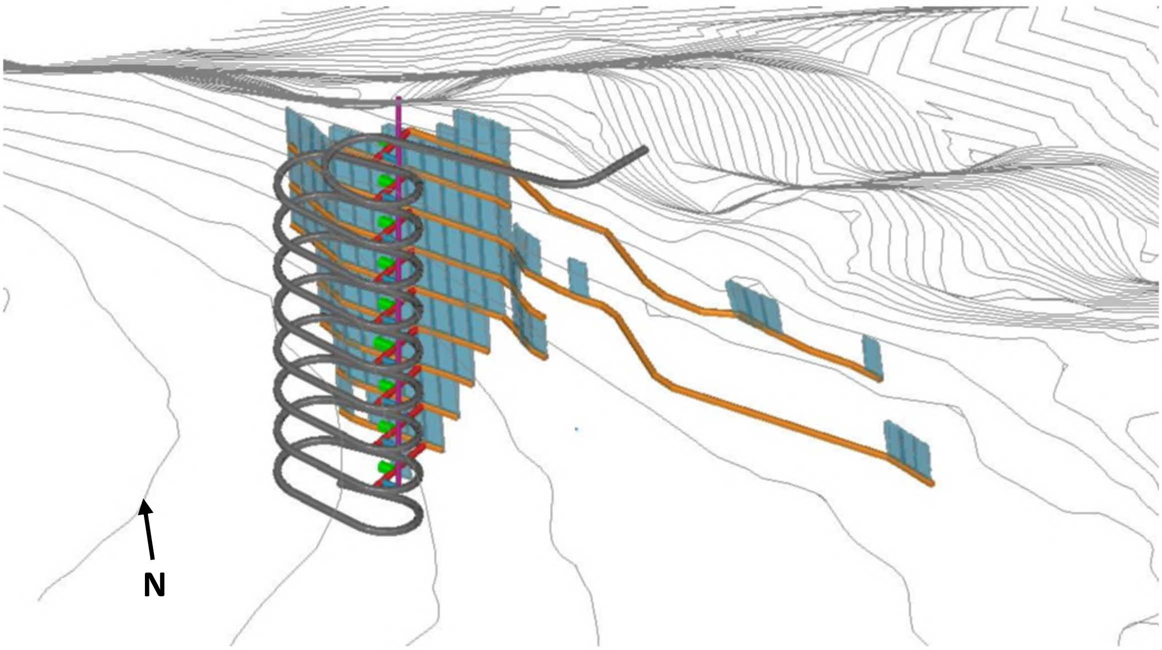


Figure 15.21: Vertikalny Underground Zone 4 Isometric View

A summary of the tonnages and grades contained within the conceptual underground mine designs is provided in Table 15.22**,** below.

Table 15.22: Vertikalny Conceptual Underground Design Physicals (Dilution & Recovery Applied)

Parameter	Units	Value
Stope Mineralised Material	kt	609
Ag Grade	*g/t*	*462*
Pb Grade	*%*	*2.16*
Zn Grade	*%*	*1.68*
Development Mineralised Material *(On-Vein Drives Only)*	kt	232
Ag Grade	*g/t*	*263*
Pb Grade	*%*	*1.37*
Zn Grade	*%*	*1.26*

Note:
- Unplanned Dilution of 10% and Mining Loss of 10% applied to **stope** mineralised material.
- Development mineralised tonnes depleted from stope tonnes.
- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP
- Figures not representative of Ore Reserves (in accordance with JORC 2012)

15.9 Mine Production Scheduling and Equipment Requirements

Mine production scheduling was carried out using the Geovia MineSched mine scheduling software package. A combined open pit and underground production schedule was generated utilising the mine designs for both the Vertikalny and Mangazeisky North deposits. A scheduling block model was prepared in which the mineralised material was split by cut-off grade (i.e., above/below) and rock type (i.e., oxide/fresh).

15.9.1 Combined Production Schedule

The schedule was prepared on the premise that a flotation circuit will be implemented to process the sulphide feed following depletion of the oxides contained within the Vertikalny open pit. A flotation plant is anticipated to be available as of mid-2021. Any sulphide feed produced before this is assumed to be processed through the current leach circuit.

An ore sorter will be available on site as of Q2 2020. A summary of the ore sorting parameters is provided below:

Mass Recovery = 66% *(Source: SBR)*
Ag Recovery = 99% *(Source: SBR)*
Pb Reocvery = 99% *(Source: SBR)*
Zn Recovery = 99% *(Source: SBR)*

SBR have indicated that due to the installation of the new ore sorter, below cut-off material will be incorporated into the plant feed. Consequently, WAI has incorporated this approach in subsequent scheduling.

The targeted processing plant throughput rates (post ore sorter) are summarised below:

1. Oxide: 115ktpa (Current plant throughput rate)
2. Sulphide: 180ktpa (SBR flotation plant capacity estimate)

Underground production is scheduled to coincide with the depletion of the open pits; thereby, maintaining a steady throughput of mineralised material to the plant. A steady state stope production rate of 340tpd has been applied.

Results of the production schedule are summarised in Table 15.23 to Table 15.28 below. WAI notes that scheduling has been carried out on a quarterly basis but has been reported annually.

Table 15.23: Vertikalny Open Pit Physicals

Parameter	Units	2019	2020	2021	2022	2023	2024	2025	2026	Total
Oxide (NSR>=117 US$/t)	**kt**	**15.9**	**87.6**	**109**	**-**	**-**	**-**	**-**	**-**	**212**
Ag	*g/t*	*716*	*789*	*821*	-	-	-	-	-	***800***
Oxide (NSR<117 US$/t)	**kt**	**4.10**	**25.6**	**15.3**	**-**	**-**	**-**	**-**	**-**	**45.0**
Ag	*g/t*	*92*	*100*	*114*	-	-	-	-	-	***104***
Sulphide (NSR>=113.06 US$/t)	**Kt**	**3.45**	**47.2**	**65.7**	**-**	**-**	**-**	**-**	**-**	**116**
Ag	*g/t*	*814*	*959*	*767*	-	-	-	-	-	***846***
Pb	*%*	*0.95*	*1.65*	*1.79*	-	-	-	-	-	***1.70***
Zn		*2.37*	*1.46*	*1.76*	-	-	-	-	-	***1.66***
Sulphide (NSR<113.06 US$/t)	**kt**	**0.150**	**15.3**	**13.6**	**-**	**-**	**-**	**-**	**-**	**29.0**
Ag	*g/t*	*136*	*147*	*114*	-	-	-	-	-	***131***
Pb	*%*	*0.32*	*0.81*	*1.19*	-	-	-	-	-	***0.98***
Zn	*%*	*0.34*	*1.04*	*1.72*	-	-	-	-	-	***1.36***
Total Mineralised Material	**kt**	**23.6**	**176**	**204**	**-**	**-**	**-**	**-**	**-**	**403**
Waste	**Kt**	**383**	**5,530**	**5,080**	**-**	**-**	**-**	**-**	**-**	**11,000**

Notes:

- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Tonnage and grade figures may not reconcile due to rounding.
- Mining **dilution (30%)** and mining **recovery (95%)** applied.
- Figures not representative of Ore Reserves (in accordance with JORC 2012)
- Figures effective as of 01.11.19

Table 15.24: Mangazeisky North Open Pit Physicals

Parameter	Units	2019	2020	2021	2022	2023	2024	2025	2026	Total
Sulphide (NSR>=113.06 US$/t)	**Kt**	**-**	**-**	**32.1**	**199**	**115**	**-**	**-**	**-**	**347**
Ag	g/t	-	-	*507*	*554*	*617*	-	-	-	***570***
Pb	%	-	-	*4.84*	*6.35*	*10.16*	-	-	-	***7.47***
Zn		-	-	*0.08*	*0.40*	*1.75*	-	-	-	***0.82***
Sulphide (NSR<113.06 US$/t)	**kt**	**-**	**-**	**5.78**	**44.6**	**21.8**	**-**	**-**	**-**	**72.2**
Ag	g/t	-	-	*161*	*125*	*128*	-	-	-	***129***
Pb	%	-	-	*0.55*	*1.51*	*1.33*	-	-	-	***1.38***
Zn	%	-	-	*0.01*	*0.16*	*0.90*	-	-	-	***0.37***
Total Mineralised Material	**kt**	**-**	**-**	**37.9**	**244**	**137**	**-**	**-**	**-**	**419**
Waste	**Kt**	**-**	**-**	**1,070**	**4,680**	**2,790**	**-**	**-**	**-**	**8,540**

Notes:

- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Tonnage and grade figures may not reconcile due to rounding.
- Mining **dilution (30%)** and mining **recovery (95%)** applied.
- Figures not representative of Ore Reserves (in accordance with JORC 2012)

Table 15.25: Vertikalny UG Physicals

Parameter	Units	2019	2020	2021	2022	2023	2024	2025	2026	Total
Waste Development	**kt**	**-**	**-**	**-**	**55.4**	**81.4**	**92.8**	**54.7**	**-**	**284**
Vein Drive Mineralised Material	**kt**	**-**	**-**	**-**	**17.5**	**89.3**	**82.0**	**40.2**	**2.59**	**232**
Ag	g/t	-	-	-	*281*	*269*	*231*	*306*	*239*	***263***
Pb	%	-	-	-	*1.34*	*1.17*	*1.35*	*1.88*	*1.13*	***1.37***
Zn	%	-	-	-	*2.35*	*1.53*	*0.84*	*1.07*	*0.72*	***1.26***
Stope Mineralised Material	**kt**	**-**	**-**	**-**	**-**	**43.3**	**172**	**233**	**160**	**609**
Ag	g/t	-	-	-	-	*457*	*452*	*466*	*468*	***462***
Pb	%	-	-	-	-	*2.39*	*1.65*	*1.51*	*3.60*	***2.16***
Zn	%	-	-	-	-	*2.95*	*2.50*	*1.35*	*0.92*	***1.68***
Total Mineralised Material	**kt**	**-**	**-**	**-**	**17.5**	**133**	**254**	**273**	**163**	**840**
Ag	%	-	-	-	*281*	*331*	*381*	*442*	*465*	***407***
Pb	%	-	-	-	*1.34*	*1.57*	*1.56*	*1.57*	*3.56*	***1.95***
Zn	%	-	-	-	*2.35*	*1.99*	*1.97*	*1.31*	*0.92*	***1.56***

Notes:

- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Tonnage and grade figures may not reconcile due to rounding.
- Mining **dilution (10%)** and mining **recovery (90%)** applied to stope tonnes.
- On-vein drive mineralised material depleted from stope tonnes.
- Figures not representative of Ore Reserves (in accordance with JORC 2012)

Table 15.26: Stockpile Balance (Closing Balance)

Parameter	Units	2019	2020	2021	2022	2023	2024	2025	2026	
Oxide Stockpile*	kt	*45.2*	-	-	-	-	-	-	-	
Sulphide Stockpile	kt	3.60	13.8	52.0	41.6	38.9	19.2	19.8	-	

* Out-of-balance, sub grade material

Table 15.27: Ore Feed (Through Sorter from Q2 2020)

Parameter	Units	2019	2020	2021	2022	2023	2024	2025	2026	Total
LEACH PLANT (CURRENT)										
Oxide Feed	**kt**	**20.0**	**113**	**124**	**-**	**-**	**-**	**-**	**-**	**257**
Ag	g/t	588	633	734	-	-	-	-	-	**678**
Sulphide Feed	**kt**	**-**	**52.3**	**31.9**	**-**	**-**	**-**	**-**	**-**	**84.2**
Ag	%	-	799	514	-	-	-	-	-	**691**
Sulphide + Oxide Feed	**kt**	**20.0**	**165**	**156**	**-**	**-**	**-**	**-**	**-**	**342**
FLOTATION PLANT										
Sulphide Feed	**Kt**	**-**	**-**	**47.2**	**272**	**272**	**274**	**272**	**183**	**1,320**
Ag	g/t	-	-	587	507	460	379	439	448	**452**
Pb	%	-	-	2.99	4.89	5.61	1.53	1.53	3.39	**3.37**
Zn	%	-	-	0.99	0.53	1.77	2.02	1.30	0.93	**1.33**

Notes:

- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Dilution and recovery applied.

Table 15.28: Process Plant Feed

Parameter	Units	2019	2020	2021	2022	2023	2024	2025	2026	Total
LEACH PLANT (CURRENT)										
Oxide Feed	**kt**	**20.0**	**84.8**	**82.0**	**-**	**-**	**-**	**-**	**-**	**187**
Ag	g/t	588	838	1,100	-	-	-	-	-	**927**
Sulphide Feed	**kt**	**-**	**34.5**	**21.1**	**-**	**-**	**-**	**-**	**-**	**55.6**
Ag	g/t	-	1,200	770	-	-	-	-	-	**1,040**
Sulphide + Oxide Feed	**kt**	**20.0**	**119**	**103**	**-**	**-**	**-**	**-**	**-**	**242**
Sulphide in Blend	%	0%	29%	20%	-	-	-	-	-	**23%**
FLOTATION PLANT-										
Sulphide Feed	**kt**	**-**	**-**	**31.1**	**179**	**180**	**181**	**180**	**121**	**872**
Ag	g/t	-	-	881	761	690	568	659	673	**677**
Pb	%	-	-	4.48	7.33	8.42	2.29	2.30	5.09	**5.06**
Zn	%	-	-	1.49	0.80	2.66	3.03	1.96	1.39	**1.99**

Notes:

- All figures rounded to 3SF. Pb/Zn grades rounded to 2DP.
- Dilution and recovery applied.

15.9.2 Development Profile

Horizontal and vertical development rates of 140m/mo (Terta Tech estimate), and 1.5m/d (WAI estimate), were applied in scheduling, respectively. The development advance rates were used in MineSched to generate the development schedule. Development was scheduled sufficiently in advance to maintain steady state stope production. A summary of the development meterage by development type is provided in Table 15.29, below.

Table 15.29: Underground Development Schedule

Development	Unit	2019	2020	2021	2022	2023	2024	2025	2026	TOTAL
Access Decline	m	-	-	-	1,487	2,192	2,343	1,389	-	7,411
Level Access Drive	m	-	-	-	193	328	395	293	-	1,208
On-Vein Drive	m	-	-	-	622	3,193	2,985	1,395	97	8,292
Remuck Bays	m	-	-	-	36	55	74	44	-	209
Vent Connection	m	-	-	-	69	75	79	51	-	273
Ventilation Raise	m	-	-	-	175	261	450	175	-	1,061
TOTAL	**m**	-	-	-	2,582	6,104	6,326	3,347	97	18,454

15.9.3 Open Pit Equipment Requirements

Mine equipment requirements were estimated to achieve the open pit production schedule presented in Table 15.30. Equipment requirement estimates for drilling, loading and hauling were calculated from first principles analysis. Key considerations made in estimation include:

- Utilisation of similar specification equipment to that currently available on site;
- Application of the current blast design parameters;
- Estimates of the annual haulage distances to the waste rock dump (WRD) and run-of-mine (ROM) pad; and,
- Application of suitable productivity/utilisation factors and working hours.

The ancillary equipment requirements were estimated based on previous experience of similar projects and approximate working hours required. A summary of the estimated major fleet requirements is provided in Table 15.30, below.

Table 15.30: Estimated Equipment Requirements

TYPE	MODEL	QTY
Excavator	CAT 336 DL (Ore)	1
	CAT 349 DL (Waste)	2
Haul Trucks	SCANIA G440	8
Production Drills	Sunward SWDE-120 (Or equiv.)	3
Wheel Loader	CAT 950GC	2
Motor Grader	CAT14M (Or equiv.)	1
Tracked Dozer	D9R	1
Fuel Tank	8000L	2
Water Tank	6000L	1
Lube/Shop Truck	-	1

All mining equipment currently deployed on site is owned and operated by SBR. A summary of the existing major mining equipment is provided in Table 15.31, below.

Table 15.31: Existing Mining Equipment on Site

TYPE	MODEL	QTY
Excavator	CAT 336 DL (Ore)	1
	CAT 349 DL (Waste)	1
Haul Trucks	SCANIA G440	8
Tracked Dozer	CAT D9R	2
Production Drills	Sunward SWDE-120	1
	URB-2A2 (URAL 4320 Chassis)	1
Wheel Loader	CAT 950GC	2
Motor Grader	SEM-922	1
Fuel Tanker	8000L	1
Water Tanker	6000L	1

Comparison of Table 15.30 and Table 15.31 indicates that the following additional items of equipment will be required:

1x CAT349 (Waste rock excavator)
1x Production Drill (Atlas Copco ROCL or equivalent)
1x 8000L Fuel Tanker
1x Auxiliary Lube/Shop truck

These additional items will be required as of 2020 of the production schedule, indicating an effective working life of four years before the cessation of open pit production in 2023. WAI has treated the equipment as leased over this period in order to save on the capital cost requirements of purchasing new equipment. It is assumed Scania trucks will be replaced near-end of operational life and retained for spares/cover for downtime/maintenance. Operating costs for these additional items include a mark-up factor of 25% to account for leasing.

15.9.4 Underground Equipment Requirements

Mine equipment requirements were estimated to achieve the underground production schedule presented in Table 15.32. In addition to the mobile equipment, fixed infrastructure crucial to the operation of the underground workings were also considered. A summary of the underground equipment requirements is provided in Table 15.32, below.

Table 15.32: Underground Equipment Requirements

TYPE	QTY
Mobile Equipment	
Development Jumbo – Single Boom	4
Production Drill	2
Load Haul Dump – 1.5m^3	4
Underground Haul Truck – 20t	4
Raise Bore	1
Explosives Truck	1
Small Motor Grader	1
Fuel & Lube Truck	1
Water Truck (Dust suppression)	1
Underground 4x4	6
Scissor Lift	1
Fixed Infrastructure	
Primary Fan	4
Secondary Fans & Starters	16
Compressors	4
Main Pump	4
Face Pump	21
Jumbo Boxes	21

WAI notes that raise boring equipment was treated as leased in this study due to the high purchase price, life of the operation and anticipated workload. Operating costs include a mark-up factor of 50% to account for leasing.

Ventilation and fixed infrastructure requirements were not calculated in this study. Provision was made for these items based on data from similar projects and the number of underground mining zones in operation at single point in time. Detailed ventilation and infrastructural studies should be carried out in further studies.

15.10 Risks

The key mining risks associated with the Mangazeisky Silver project are summarised in the points below:

- The derived 'mineable tonnage' estimates for the Vertikalny and Mangazeisky North deposits are not representative of Ore Reserves. Sufficiently detailed modifying factors were not applied, nor was economic viability demonstrated to a suitable degree of confidence.

- The Mangazeisky North deposit is comprised of Inferred Resources only. Further infill drilling is required to upgrade geological and metallurgical confidence. This is essential to progress the deposit to a more advanced stage of design and planning. The Mangazeisky North deposit provides an essential source of sulphide feed and provides the necessary time to develop a potential underground mine at the Vertikalny deposit following depletion of the Vertikalny open pit.
- WAI is unaware of the presence of any detailed geotechnical data and analysis for the Mangazeisky North deposit. The conceptual pit design was based on a set of design criteria derived from analogous projects. Additional geotechnical data and analysis is required to define a set of site-specific design criteria to mitigate the risks associated with geotechnically sub-optimal pit designs.
- WAI's production schedule indicates that a shortage of oxide feed from the Vertikalny open pit will occur between Q3 2020 and Q1 2021. During this period, the oxide feed shortage will be substituted with sulphide material. The main risks associated with processing sulphide material through the current processing plant include significantly higher processing costs and reduced metal recoveries.
- The Vertikalny conceptual open pit design includes a significant amount of waste material due to the implementation of SBR's pit design criteria which utilise wide benches, shallow haul roads and minimum pit bottom width requirements. A significant amount of waste development is required in order to maintain steady production (combined oxide and sulphide). SBR have indicated that additional equipment is being brought to site to address the increased waste mining volumes. Should mining productivity or equipment capacity be lower than required, ore production may be adversely impacted and exacerbate the oxide feed gap.
- Low-grade stockpiled (stockpile no.5) oxide material may offer an opportunity to address the oxide feed gap indicated in the production schedule. The material composition and metallurgical characteristics of this stockpile are unknown and require further sampling and testing before being considered a viable source of feed to bridge the oxide production gap. Initial scheduling results indicate that the oxide deficit could potentially be reduced by half when incorporating the low-grade stockpile into the production schedule (assuming stockpile material suitable for plant feed).
- Construction of a flotation plant is anticipated for completion by mid-2021. The generated production schedule assumes that production will seamlessly transition between the current (oxide) plant and new flotation plant in Q4 2021. It is assumed that the flotation plant will require no ramp-up period and be able to accept sulphide material at the stated capacity of 180ktpa (as indicated by SBR). Should a ramp-up period be required, actual metal production may be lower than that indicated in the production schedule; therefore, adversely impacting project economics.
- Further geotechnical data and analysis is required to refine the underground geotechnical design criteria as derived for the Vertikalny deposit by SRK Consulting in 2014. Particular attention should be given to the identification of any potential large-scale structural features that may pose a risk to underground excavations.

- Underground development dimensions used in the Vertikalny underground mine design were based on the design parameters outlined in the Tetra Tech study (dated 21-08-17). The Tetra Tech study assumed a steady state underground production rate of 110ktpa. The production rate target used by WAI in underground scheduling was 272ktpa. This is due to the higher capacity of the new flotation plant (180ktpa) and the presence of an upstream ore sorter which rejects approximately 33% of ROM plant feed. Underground development dimensions must be re-evaluated to accommodate the potentially larger equipment required to achieve the higher production rates.
- Mining capital and operating cost estimates are based on a Preliminary Economic Assessment (PEA) level of confidence (±45%). The study offers a valuable view in determining the merits of pursuing further engineering studies but should not be the sole reference for the purposes of economic decision making. Enhanced engineering costs estimates should be prepared as part of a more detailed study aligned with the preparation of an Ore Reserve estimate.

16 RECOVERY METHODS (ITEM 17)

16.1 Introduction

Wardell Armstrong International was requested to undertake a Strategic Review of current operations at SBR. The main issue from a processing perspective is the amount of primary sulphides that require processing and the potential options for doing this. The process plant as currently configured was designed to operate on oxides only. This review mainly references actual SBR operating data as provided by SBR and the Tetra Tech (TT) NI 43-101 Feasibility Study report, dated 9th June 2016. The main oxide ore zones currently being mined and processed are from the Vertikalny Central and Northwest zones. These were drilled most recently in 2013/2014 and current mining is by open pit. Additional ore zones drilled in 2015 but not yet mined include Mangazeisky North and South zones, which are predominantly primary sulphide ore. It appears that these zones have not yet been tested, with primary ore testing restricted to the deeper parts of the Vertikalny Central zone.

EMC Mining developed the detailed design documentation for the plant based on the conceptual circuit originally developed by Tetra Tech and this documentation has been generally reviewed. In addition, Benitex developed the design documentation for the recently constructed Merrill Crowe plant. A recent site visit report by Benitex on the status of the overall plant and the Merrill Crowe plant in particular was also reviewed.

16.2 Process Design

16.2.1 Oxide Ore

The process design is based upon the original Tetra Tech design in the feasibility study but with some modifications introduced by SBR. EMC Mining developed the final process design and detailed design documentation for construction.

The original Tetra Tech design was based on the processing of oxide ore only, but with recommendations to modify the plant for processing sulphide ore. The plant was designed for a throughput of 110,000 tonnes per annum (tpa) and a plant availability of 91% for an operating throughput rate of 15tph. Design silver head grade was 772g/t Ag. First production of silver was achieved in April 2018.

Comminution is achieved using conventional two-stage crushing with a jaw and cone crusher and milling is achieved in a single ball mill equipped with a 500-kW motor. The grind size required is 80% passing 75 microns.

A gravity circuit was incorporated in the original design using a Knelson concentrator with regrinding and intensive cyanide leaching of the concentrate. However, the gravity circuit was not subsequently installed by SBR.

The grinding circuit incorporated two-stage hydrocyclones (classification and dewatering cyclones) but the dewatering cyclone was replaced with a dedicated pre-leach thickener, to achieve a nominal 50% solids pulp density required for leaching.

The original leach circuit required six tanks for a design residence time of 72 hours. However, with the exclusion of the gravity circuit and the testwork indicating the subsequent slow leach kinetics, an additional two leach tanks were installed by SBR to provide the increased design residence time of 96 hours.

The original design dewatered the final leach tailings slurry in a hydrocyclone with the overflow clarified in a high-rate clarifier (lamellar thickener) to produce a suitable solution for the direct electrowinning process. The clarifier and hydrocyclone underflows were then filtered in plate and frame filter presses. The filtrate solution was recycled to the plant as process water and the filtered solids disposed in a dedicated Dry Stack Tailings Facility.

This circuit was subsequently modified by removing the dewatering cyclone and clarifier and filtering the leach tailings directly in the filter presses, but now using two stages of filter presses to obtain solution suitable for direct electrowinning.

The direct electrowinning process uses patented emew® cell technology to recover the silver from solution, with the resulting silver precipitate shipped directly to a refinery (or can be smelted on-site).

The primary stage, consisting of 140 emew® powder cells, each 200 mm in diameter, reduces the silver solution from approximately 800ppm to 50ppm silver. The secondary stage, consisting of 80 emew® polishing cells, each 200mm in diameter, reduces the solution to below 10ppm silver prior to discharge. The entire direct electrowinning plant is supplied as a modular turnkey package plant by Electrometals. The barren solution is returned to the process water tank. The design should incorporate a 1% bleed of solution to avoid a build-up of base metals, such as zinc, in the solution. It is not known if this was incorporated into the final design.

Figure 16.1 shows the current schematic flowsheet for the plant including the changes as outlined above.

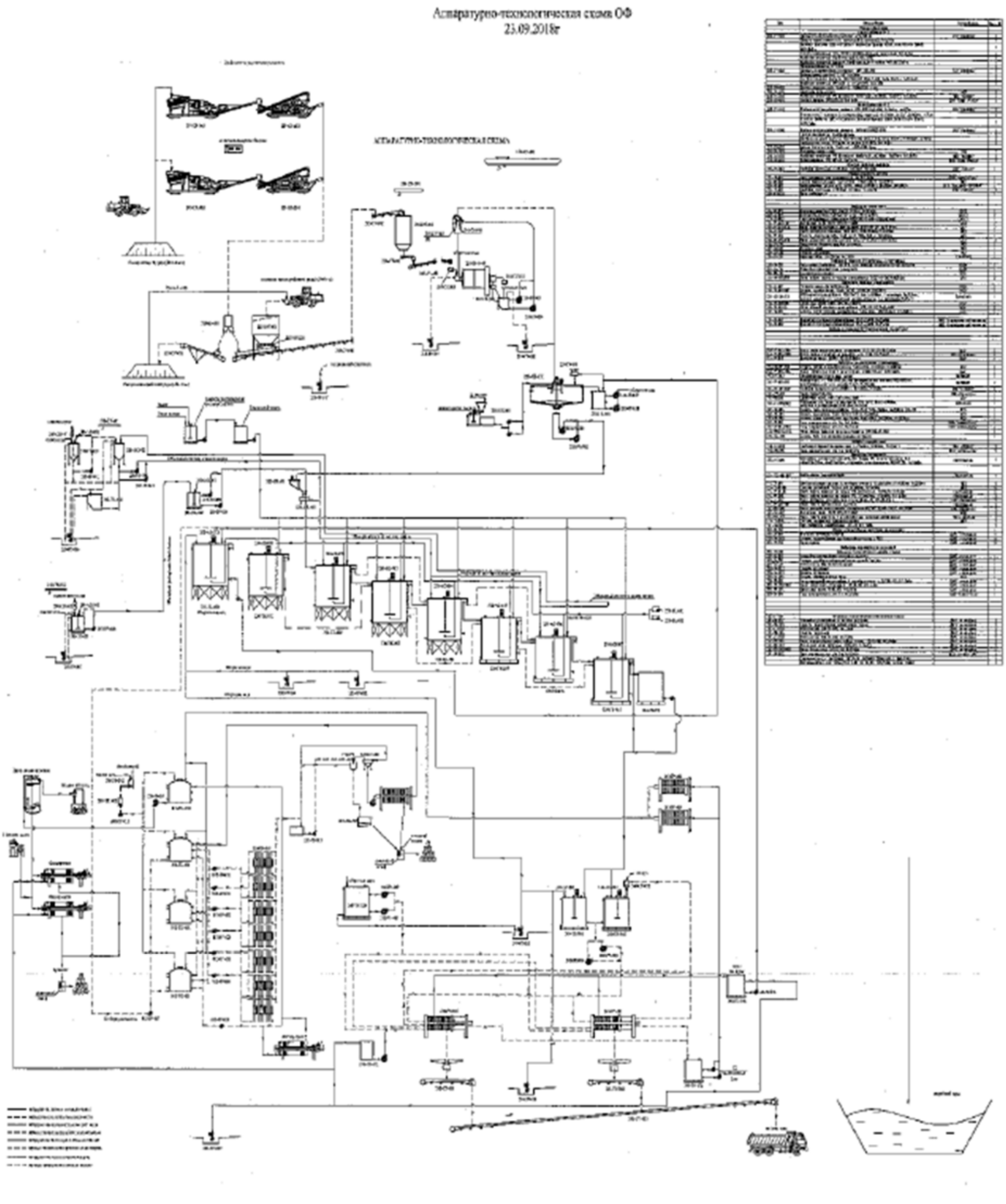


Figure 16.1: Schematic Flowsheet for Oxide Ore

16.2.1.1 Current Problems & New Merrill Crowe Circuit

A significant issue with SBR was the operation and performance of the direct electrowinning process. Issues include corrosion due to the chloride content in solution and excessive levels of base metals, in particular zinc. In fact, a new Merrill Crowe circuit was installed by Benitex in April 2019 (not shown in the flowsheet above). A representative from Benitex also conducted a site visit in April 2019 and commented that, at that time, there were issues with non-delivery and/or poor performance of some of the equipment and incorrect installation of some of the pipework. Some of these issues, including training of personnel, were rectified during the site visit, with others remaining to be completed. It was also recommended that cyanide solution be added after the deaeration tower to control the copper content in solution.

SBR report that the Merrill Crowe circuit is operating well and recovering 98-99% of the silver in solution. The circuit is flexible and operates either in parallel with the direct electrowinning circuit or in series by treating the electrowinning barren solution. It is the intention that the Merrill Crowe circuit will eventually operate directly as a replacement for the direct electrowinning circuit. The resulting silver-rich powder has approximately 70% silver content and is refined off-site, although it is recommended that silver bullion be produced on-site.

Other issues mentioned in the Benitex report include the following:

- Lack of instrumentation and automatic control in the milling circuit;
- Incorrect water distribution around the whole plant;
- Pre-leach thickener acting as a bottleneck, lack of instrumentation and control;
- Inefficient slurry mixing in the agitated leach tanks resulting in short-circuiting, and elevated temperatures attributed to oxidation of sulphides;
- Low silver recovery compared to design and the conclusion that up to 20% of the ore was primary sulphide ore;
- Low activity of received lime (55.8%);
- Manual dosing of lime from ring main system results in inefficient dosing;
- Incorrect cyanide make-up procedures and inefficient manual dosing;
- Insufficient water washing (time and volume) of the filtered solids resulting in 19.1% silver recovery loss in the solids reporting to tailings.

The main issues to be noted from the above observations are the high silver recovery loss of 19.1% estimated from insufficient washing of the filter cake, higher cyanide and lime consumptions from inefficient preparation and dosing and the inclusion of primary sulphide ore with the oxides that lowers recovery and increases reagent consumptions. Some of these issues were reportedly addressed during or soon after the Benitex site visit.

16.2.2 Primary Ore

The proposed process design for treating primary sulphide ore includes a new flotation circuit for the production of separate lead and zinc concentrates. The lead flotation middlings are cyanide leached as per the current flowsheet to produce a silver-rich powder for transport to the refinery. The design allows for increased throughput to 180,000 tpa with harder ore and therefore includes additional crushing and milling capacity in the form of a second identical primary and crushing circuit and ball mill.

The schematic flowsheet is shown in Figure 16.2.

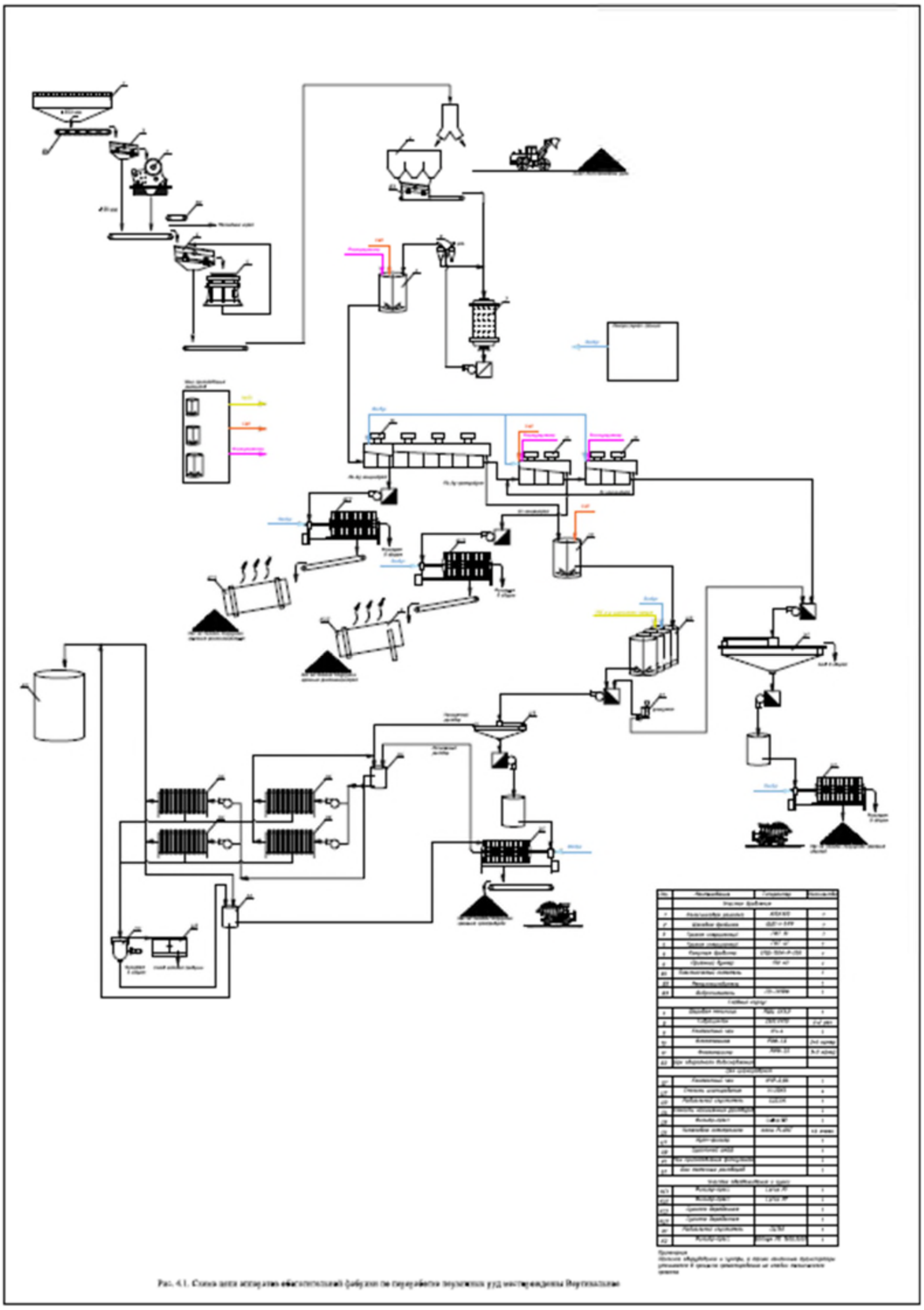


Figure 16.2: **Schematic Process Flowsheet for Primary Sulphide Ore**

16.3 Operating Performance

The mine achieved first silver production on oxide ore from open pit operations in April 2018. SBR has provided operating and cost data up to and including July 2019 for when this report was initially prepared.

For July 2019 YTD, SBR processed 55,184t at an average head grade of 672g/t Ag. Subsequently, in an update to this report and according to the SBR website, for the nine-month period to September 2019, 71,769t were processed at an average grade of 670g/t Ag for a silver recovery of 70.5%. Pro-rata, this is equivalent to approximately 96,000tpa, slightly less than the design of 110,000tpa.

Figure 16.3 below plots the final silver recovery (allowing for refinery adjustments) since operations commenced, from the original production data supplied by SBR to July 2019.

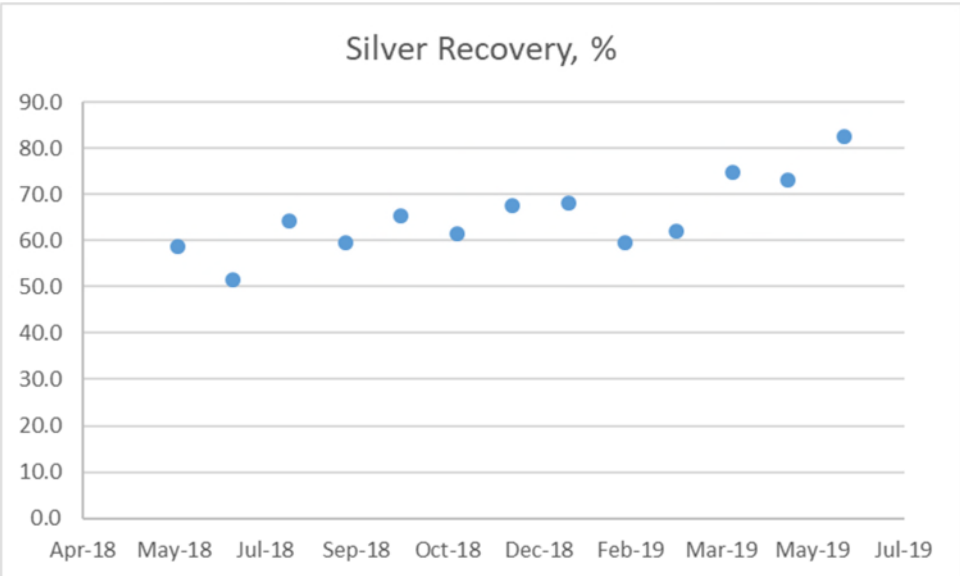


Figure 16.3: Final Silver Recovery

Allowing for initial commissioning, it can be seen that silver recoveries remained generally in the range of 50 – 70% until April 2019, when recoveries sharply improved, approaching the design recovery of 85%. This coincided with a decrease in silver head grade to an average of 485g/t Ag for April – June 2019, as normally lower head grades will give lower recoveries and vice-versa. It is believed that, following the Benitex site visit and remedial measures to improve the washing of the tailings filter cake, where significant silver losses were occurring, this resulted in the improvement in silver recovery. Further measures to improve recovery included the addition of the Merrill Crowe circuit to re-process the barren solution from the direct electrowinning circuit.

However, in an update to this report and reviewing the SBR website, the silver recovery for the nine months to September 2019 is stated as 70.5%, so this is still some way short of the design of 85%.

It is believed that inclusion of primary sulphide ore in the plant feed blend has significantly contributed to the lower-than-design recoveries. SBR indicated that approximately 5-15% of the ore may be primary sulphide ore, although this is likely to be higher, and the reported cyanide concentration of 5,000 ppm (compared to the design for oxides of 2,000 ppm) also reflects this.

The design operating cost for oxide ore from the Tetra Tech feasibility study is **US$47.9/t**. Power is the main contributor at $23.4/t, followed by reagents at $14.0/t, labour at $8.3/t and maintenance at $2.2/t. However, May 2019 YTD actual process costs are reported by SBR to be approximately **$74.9/t**, with the reagents cost at approximately $28/t, i.e. double the design. Some of the increase in unit costs can also be attributed to the lower actual throughput compared to design.

The main reagents consumed are cyanide, lime and steel balls and the design consumption rates are 4.6kg/t, 0.7kg/t and 0.7kg/t respectively. Actual June 2019 YTD consumptions are 5.9 kg/t, 23.9kg/t and 0.99kg/t respectively. The cyanide and steel ball consumptions show moderate increases compared to design, most likely a reflection of the sulphide ore content. The lime consumption, however, is significantly higher than design and it appears that the design value of 0.7 kg/t is incorrect based on the latest testwork.

Reviewing an SGS testwork report from 2014, lime consumptions in the bottle roll tests conducted varied from approximately 20 – 30kg/t. Even allowing for typical actual field consumptions to be 2-3 times lower than the testwork results, the design figure of 0.7kg/t is clearly too low. Design lime consumption should be approximately 15kg/t maximum, so actual consumption is still higher than this value. This is probably a reflection of the low as-delivered lime activity and inefficient dosing, as outlined in the Benitex report.

Sales and refinery costs are reported as approximately $3.2/t for May 2019 YTD.

16.4 Ore Sorting

Testwork has been conducted on the use of ore sorting to provide an upgraded feed to the flotation plant and to reject a low-grade tailings stream, allowing the mining of an increased throughput of 270ktpa to provide 180ktpa as feed for the new flotation plant.

16.4.1 Testwork

A summary of the testwork results has been provided by SBR. The testwork was conducted by GeoTestService (GST) on two samples, a low-grade oxide sample (GTS1) and a current production sample (GTS2). Although no sorter testwork has been performed on primary sulphide ore, SBR reports that they expect results to be very similar due to the similar mineralogy. However, this does present a small risk that performance with primary ore may not be the same as for the oxide ore tested.

Testwork was conducted on three different size fractions, -100+60mm, -60+30mm and -30+15mm. The -15mm, at 28.8% of the feed mass, is too fine for ore sorting and will be fed direct to the flotation plant. The results from testing each of the three size fractions were broadly similar and, in summary, combining the results, the average stage sorter mass recovery to the “accepts” fraction was 22.8%

and therefore 45% of the total ROM feed (including the -15mm fraction) will report to the flotation plant.

The average Ag, Pb and Zn recoveries to the flotation plant feed were 99%, 99% and 69.7% respectively. A significant upgrade in head assay also results, the Ag assay increasing from 690g/t to 1,518g/t, the Pb assay from 1.06% to 2.25% and the Zn assay from 1.61% to 2.48%. This should result in better flotation recoveries.

16.4.2 Processing Schedule

The latest mining schedule is shown below in Figure 16.4.

5. ORE SORTER FEED										
CURRENT PLANT										
Oxide	t	20,039	113,151	124,243	-	-	-	-	-	257,434
Ag	g/t	588	633	734	-	-	-	-	-	678
Sulphide	t	0.03	52,253	31,947	-	-	-	-	-	84,200
Ag	g/t	786	799	514	-	-	-	-	-	691
Oxide + Sulphide	t	20,039	165,405	156,190	-	-	-	-	-	341,634
FLOTATION PLANT										
Sulphide	t	-	-	47,152	271,817	272,394	273,860	272,493	182,754	1,320,470
Pb	%	-	-	2.99	4.89	5.61	1.53	1.53	3.39	3.37
Ag	g/t	-	-	587	507	460	379	439	448	452
Zn	%	-	-	0.99	0.53	1.77	2.02	1.30	0.93	1.33

6. PROCESS PLANT FEED										
Mass Recovery			0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66
Ag Recovery			0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99
Pb Recovery			0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99
Zn Recovery			0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99
CURRENT PLANT										
Oxide	t	20,039	84,844	82,001	-	-	-	-	-	186,884
Ag	g/t	588	838	1,101	-	-	-	-	-	927
Sulphide	t	0	34,487	21,085	-	-	-	-	-	55,572
Ag	g/t	786	1,199	770	-	-	-	-	-	1,036
Oxide+Sulphide	t	20,039	119,331	103,085	-	-	-	-	-	242,456
% Sulphide in Blend		-	29%	20%	0%	0%	0%	0%	0%	23%
FLOTATION PLANT										
Sulphide		-	-	31,121	179,399	179,780	180,747	179,845	120,618	871,510
Ag		-	-	881	761	690	568	659	673	677
Pb		-	-	4.48	7.33	8.42	2.29	2.30	5.09	5.06
Zn		-	-	1.49	0.80	2.66	3.03	1.96	1.39	1.99

Figure 16.4: Mining Schedule

The schedule indicates that ore sorting will be applied for the whole of 2020. However, SBR report that the sorter is expected to be commissioned towards the end of April 2020 (equipment is on site and installation has started). Sulphide ore will continue to be processed through the current plant in 2020 and most of 2021, until the new flotation plant is commissioned, reported by SBR to be expected in June 2021.

The tonnes processed through the current plant after ore sorting in 2020 and 2021 of 119kt and 103kt respectively should be achievable with continued optimisation, as SBR report that a throughput of 10,000tpd is now considered normal since further de-bottlenecking was completed in September 2019 (the plant design for oxides is 110ktpa, although harder sulphide ore is now in the blend (29% and 21% respectively for 2020 and 2021). However, there is still a risk that this throughput may not be achieved depending on the hardness and actual blend of sulphide ore. In addition, 31kt of sulphide ore is due to be processed through the new flotation plant in 2021.

From 2022 onwards, the ore feed is 100% sulphide ore through the new flotation plant, maintaining capacity at 180ktpa. At this rate, approximately 270ktpa of ROM feed is scheduled to be fed to the primary crusher. After primary crushing, the product is screened to remove the -15mm fraction (28.8%) that reports direct to the flotation plant. The remaining 71.2% reports to the ore sorter. Based on the original testwork, the tailings stream from the sorter is rejected (77.2% of sorter feed) with the accepts fraction (22.8%) reporting to the flotation plant after secondary crushing to -15mm. Using the testwork value of 45% total mass split of ROM ore to flotation plant feed, this would calculate to a flotation plant feed of approximately 122ktpa.

However, it should be noted that, in the schedule above, the mass split of ROM ore to the flotation plant has been increased from 45% to 66%. The higher mass split results in a flotation plant throughput of approximately 180ktpa, as per design, with the stage sorter mass recovery increasing from 22.8% to 52.1% (192ktpa). Approximately 92ktpa of waste will be rejected in the sorter and 100ktpa report, after secondary crushing, with the -15mm fraction (78ktpa) after primary crushing, as flotation plant feed.

In addition, the Zn recovery has been increased from the 69.7% achieved in the testwork to 99%, matching that for the Ag and Pb. The higher mass split to the flotation plant, i.e. less rejects, is conservative and implies higher metal recovery and, as the Ag and Pb recovery is already very high, the Zn recovery has been increased as stated. This is not unreasonable, although with no further testwork planned, there is a risk that actual Zn recoveries may be lower. The Ag and Pb recoveries seem very high but appear to be corroborated by the testwork results. The higher mass split also results in a reduced upgrade of the head assays compared to the testwork results.

16.4.3 Design and Construction

SBR propose to commission the new ore sorter by end-April 2020.

As opposed to the three size fractions tested, SBR plan to treat just two size fractions through the single ore sorter on a batch-basis, with different sorter programming and feed conveyor belt speed for each fraction. The two size fractions are -100mm+40mm and -40mm+15mm.

After primary crushing, the product is screened to remove the -15mm material that reports as flotation plant feed. The +15mm material is then screened into the two size fractions. These will be separately batch processed through the single ore sorter, adjusting the conveyor speed and sorter programming for each fraction.

According to the testwork results, the indicated ore sorter throughputs for the different size fractions tested were 18tph for the -30mm+15mm fraction, 31tph for the -60mm+30mm fraction and 63tph for the -100mm+60mmm fraction. Using conservative estimates for the two size fractions to be sorted and with the estimated ore sorter throughput of 192ktpa (24tph @ 91% availability or approximately 12tph for each size fraction), then this is well within the capacity of the ore sorter unit, allowing plenty of time for maintenance.

One concern is that SBR report only one loader (FEL) will be utilised for feeding the primary crusher, feeding the ore sorter and rehandling the sorter accepts and rejects stockpiles. The accepts stockpile, along with the -15mm stockpile from screening the primary crusher product, must also be transported to the plant. The rejects stockpile must also be transported to a waste stockpile. One loader is unlikely to be sufficient for this purpose without significant risk to production and so it is strongly recommended that a second FEL should be purchased.

The installed capital cost for the ore sorter and associated infrastructure is estimated by SBR at $2 million and the additional operating cost as $2.25/t of ore sorter feed. This is considered reasonable.

16.5 Conclusions

After producing first silver production in April 2018, silver recoveries have improved from approximately 55% in 2018 to 70% for September 2019 YTD, although still short of the design for oxide ore of 85%. The improvement in 2019 is likely mainly due to better washing of the leach tailings solids filter cake, where Benitex reported that up to 19% of the silver was previously being lost due to poor washing. There is also a significant impact on recovery and costs from primary ore being included in the feed blend, reportedly 5-15% according to SBR, but likely higher than this. Higher cyanide concentrations of 5,000ppm are therefore being utilised, compared to the design of 2,000ppm.

Therefore, WAI recommends that the design silver recovery of 85% for oxide ore is still appropriate to be used for pit optimisation studies. A recovery of 29% should be applied to primary ore processed through the current plant without the circuit changes recommended in the feasibility study.

Apart from the lower recovery, the additional impact of any primary ore in the oxide feed through the current plant will be higher operating costs, with the cost of $123.7/t used in the financial model for primary ore. The oxide operating cost used is $72.9/t, significantly higher than the design of $47.9/t, and reflects the inclusion of sulphide ore in the feed blend and actual current operating costs. Overall process unit costs are also higher due to the lower throughput compared to design.

Lime consumption is significantly higher than design, although this appears to be due to an incorrect design figure of 0.7kg/t used in the feasibility study, compared to the testwork data of 20-30kg/t. Further issues contributing to the actual lime consumption of 23.9kg/t are low activity and inefficient dosing.

For the proposed processing of primary sulphide ore, a new flotation circuit is required for the production of separate lead and zinc concentrates, with cyanide leaching of the lead flotation middlings as per the current circuit configuration. Most of the existing circuit can be utilised with the addition of the new flotation circuit and extra crushing and milling capacity for the proposed higher throughput of 180,000tpa, compared to the current design for oxide ore of 110,000tpa. The new plant is scheduled to be commissioned in June 2021.

The capital cost provided by SBR of approximately $17.3m is considered reasonable for an approximate 500tpd brand new plant, although this reduces to approximately $9.2m if the existing oxide circuit equipment is used and the additional equipment retrofitted, mainly the new flotation circuit and additional crushing and grinding capacity. SBR has assumed in their schedule that most of the new equipment can be constructed alongside the existing plant with minimal time required for the final tie-in.

The process operating cost for the new plant treating primary ore has been estimated by SBR as US$47.1/t and is considered reasonable for use in the pit optimisation studies.

The recoveries used for primary ore in the pit optimisation studies are based on the ESTAGeo testwork results, with silver, lead and zinc recoveries of 85.4%, 65.9% and 82.2% respectively.

The zinc concentrate at 42.4% Zn is saleable based on typical western smelter contracts, but the lead concentrate at only 17.1% Pb is very low grade, but high in silver value at 10,215g/t Ag. This is more likely to be saleable to an Asian smelter. The NSR terms for both concentrates have been provided by SBR for use in the pit optimisation studies (84% and 45% respectively for the lead and zinc concentrates respectively).

Contract quotations should be sourced from interested smelters and a full elemental analysis conducted to determine the effect of all the potential deleterious elements, as not all appear to have been analysed.

It should be noted that the testwork on primary ore appears to have been conducted solely on Vertikalny ore and the results are assumed for pit optimisation studies to apply equally to Mangazeisky ore. The metallurgical characteristics of the Mangazeisky deposit may not be the same and it is strongly recommended that further testwork be conducted on representative samples as soon as possible, including locked cycle flotation tests on all the major primary ore zones that form part of the LOM plan.

SBR has conducted ore sorter testwork on samples of oxide ore from current production. Based on these results, the current schedule assumes that approximately 270ktpa of ore will be mined with 180,000ktpa reporting to the flotation plant after crushing and ore sorting with 99% recovery of Ag, Pb and Zn to the flotation feed. This applies to both oxide and sulphide ore. The ore sorter is scheduled to be commissioned in April 2020.

If the actual overall mass split of 45% of ROM ore to flotation plant feed, obtained during the testwork, was used instead of the 66% in the schedule, this would result in a much smaller capacity plant (122ktpa) and therefore significant savings to capital costs.

The installed capital cost for the ore sorter and associated infrastructure is estimated by SBR at $2 million and the additional operating cost as $2.25/t of ore sorter feed.

16.5.1 Risks

Some of the risks to be evaluated are the following:

- Testwork should be conducted on Mangazeisky primary ore to confirm flotation response;

- Full elemental analysis should be conducted on samples of the final Pb and Zn concentrates to determine the effect of any penalty elements and to obtain an up-to-date NSR from suitable smelters;
- Ore sorter testwork was conducted on oxide ore only and should be conducted on primary ore to confirm response and the high metal recoveries, in particular for Zn;
- Another FEL is likely required for the ore sorting operation, the current plan is to use the same FEL as for feeding the primary crusher, otherwise there is risk to production from low FEL availability;
- Throughput for 2020/2021 through the existing plant may be lower than scheduled depending on the amount and hardness of the sulphide ore in the blend;
- The current schedule assumes minimal time for a final tie-in of the upgraded plant (flotation circuit, additional crushing and grinding capacity).

17 INFRASTRUCTURE (ITEM 18)

An investigation into actual on or off-site infrastructure does not form part of WAI's terms of reference for this report.

WAI has not had access to recent site plans or the construction 'zero' report and is not in a position to comment on actual site infrastructure or issues arising, thereof, with the current site layout.

18 MARKET STUDIES (ITEM 19)

18.1 Product Realisation

The main products from the Mangazeisky Project deposit are proposed to be silver bullion and two concentrates: silver bearing zinc concentrate and silver bearing lead concentrate.

Silver bullion as precious metals is always in demand among the Russian banks. WAI notes that SBR has currently got an established cooperation and signed agreement with a Russian bank for realisation of silver bullion.

Zinc concentrate is expected to be produced at 42.3% Zinc and average 1,133g/t Silver and is considered to be saleable based on typical western smelter contracts.

Lead concentrate brings 74% of the overall project NSR on the strength of its silver content. And according to the testwork results, is assumed to be produced at 17% lead and 10,215g/t of silver. WAI was advised that both lead and silver payable content is expected to be around 84% to allow for realisation of a lower grade concentrate and smelter costs.

In due course of this study, silver content (in lead concentrate) has been estimated at 2,929g/t vs 10,215g/t. The difference in concentrate grades is explained by variance in head grades of feed materials. Whilst the historical testwork sample contained 1.8% of lead and 702g/t of Ag, WAI production schedule provides 5.8% of lead and 723.9g/t of silver in the flotation plant feed. With the much higher lead head grade than what was tested, an estimated theoretical concentrate yield resulted in 20% vs 4.5% shown in testwork results. This mass pull and concentrate yield was considered too high in practice given that variation in head feed grade ranged <10% for concentrate yield so WAI decided to run a preliminary scenario with mass split being set at 5% and using all other parameters as per testwork results and original payment terms. This exercise resulted in improved lead concentrate quality of 66% of lead and 10,026g/t of silver, and improved project economics due to significantly reduced concentrate shipment costs.

WAI comment: Caution is urged in interpretation of this scenario given the high variability in feed grade and other variables, including a lack of definitive testwork and further testwork is recommended to confirm potential improvement of the lead concentrate. WAI has utilised 17% lead content assumption in order to derive financial results presented in this report (as the base case).

Zinc concentrate is expected to be produced with 42.3% content of zinc and average 1,133g/t Silver.

Although lead concentrate is expected to be of a lower grade than is typically accepted on the market, (60-70% Pb) it is assumed to be sold to a smelter in Kazakhstan on the strength of the Ag grades (10,215g/t Ag in the Pb concentrate). There is also a potential route of realisation to China. Considering that production of zinc and lead concentrates is scheduled to commence in the end of 2021 – beginning of 2022, there is currently no official agreements between SBR and potential off-takers. Concentrate realisation arrangements are planned to be set at the following stages of the project development. Provisional agreements will help to minimise risk of uncertainty in realisation terms for lead and zinc concentrates.

WAI notes that Mangazeisky project value is mostly formed by silver content, and therefore significantly less sensitive to change in lead prices. Therefore, an impact from the potential changes in payment terms for lead and zinc prices are considered moderate to low.

18.2 Commodity Market Outlook

All costs assumptions and commodity prices used in this study have been estimated as of the end of 2019.

Table 18.1 below provides a summary of commodity prices used in the preliminary economic assessment (PEA) and mine design. These assumptions have been based on the SP Angel Report dated 27 Aug 2019 and with consideration of the World Bank Commodity Market Outlook.

Although, the prices outlined below may look relatively optimistic given current market conditions, WAI notes that project break-even silver price has been estimated at US$14.48/toz, which is six percent below the current spot prices that is ranging between US$15.35/toz - US$15.55/toz (May 2020).

Latest World Bank's Commodity Market Outlook (published in April 2020) suggests that albeit Silver prices declined to levels unseen since the global financial crisis in March, precious metals prices are expected to average 13.2% higher in 2020, with silver prices being also anticipated to recover moderately later in 2020.

Table 18.1: Commodity Price Assumptions

Scenarios	Price Assumption (as of 2019)
Ag (US$ / oz)	17.76
Pb (US$ / t)	2,069
Zn (US$ / t)	2,252

19 ENVIRONMENTAL STUDIES, SOCIAL IMPACT AND PERMITTING (ITEM 20)

An investigation into environmental impact of emissions from operations, review of environmental management plan, current monitoring strategy or mine closure plan does not form part of WAI's terms of reference for this report.

20 CAPITAL AND OPERATING COST DEVELOPMENT (ITEM 21)

Capital and operating costs reported this section in US Dollars are shown in 2019 US Dollars. These costs assumptions have been used in the preliminary economic assessment with appropriate inflation rates being applied. Therefore, costs reported in this section appear different to the costs shown in the Financial Analysis Section.

20.1 Mining - Introduction

A mining cost model was developed to assess the open pit and underground mining capital and operating expenditures for the Mangazeisky Project. The combined open pit and underground production schedule was used as the basis for cost estimation. The cost estimates were developed by WAI based on data provided by SBR and WAI's internal cost database.

The calculated costs are estimated to have an accuracy equivalent to a Preliminary Economic Assessment (PEA) level of detail. The study offers a valuable view in determining the merits of pursuing further engineering studies but should not be the sole reference for the purposes of economic decision making.

20.2 Open Pit Costs

20.2.1 Capital Cost Estimates

Open pit capital costs were estimated based on WAI's cost database and project experience of similar operations.

No equipment capital costs were considered for the open pit operations. It is assumed that additional equipment for drill & blasting, load & haul will be leased as detailed in Table 20.1 below. This table assumes only the primary equipment used in earthmoving will be leased for D&B, L&H from major suppliers. It is not ordinarily cost effective for such suppliers to lease support and auxiliary equipment. Overhaul costs for the existing primary equipment (i.e., production drills, loaders and haul trucks) were scheduled at 50% of the equipment operating life and costed at 40% of the initial equipment purchase price. Overhaul costs are estimated to be in the region of US$1.23M.

Provision was made for the construction of various access routes for pit development and material transport. A summary of these routes is provided below:

- Vertikalny Pit 1 Cut & Fill road
- Mangazeisky North Cut & Fill road
- Vertikalny to Mangazeisky North connecting road - *Approximately 7.8km long dirt road with a planned width of 16m.*

Cut & Fill road costs were based on the anticipated average mining cost, less the costs of drilling and blasting. Costs to develop the connecting road are based on rates from similar projects. A summary of the capital costs required for the preparation of these access routes is provided in Table 20.2, below.

Table 20.1: Summary of Leasing Payments for main OP mining equipment (D&B, L&H)

	Cost of equipment (incl. VAT)	Interest on Leasing	Currency	Years	Months
Drill Rig Flexi Rock D60	56,776,534	8,023,273	RUB	2.00	24
Excavator CAT 374FL	730,000	105,876	USD	3.00	36
Dump Truck CAT740GC	586,400	85,049	USD	3.00	36
Dump Truck CAT740GC	586,400	85,049	USD	3.00	36
Dump Truck CAT740GC	586,400	85,049	USD	3.00	36
Dump Truck SCANIA G440	12,340,000	1,670,840	RUB	1.25	15
Dump Truck SCANIA G440	12,340,000	1,670,840	RUB	1.25	15
Dump Truck SCANIA G440	12,340,000	1,670,840	RUB	1.25	15
Dump Truck SCANIA G440	12,340,000	1,670,840	RUB	1.25	15
Dump Truck SCANIA G440	12,340,000	334,168	RUB	0.25	3
Dump Truck SCANIA G440	12,340,000	334,168	RUB	0.25	3
Dump Truck SCANIA G440	12,340,000	334,168	RUB	0.25	3
Dump Truck SCANIA G440	12,340,000	334,168	RUB	0.25	3
Total Rub	**157,323,618**	**16,231,814**	RUB		
Total USD	**2,501,554**	**362,815**	USD		
Total in USD	**4,699,680**	**590,195**	USD		

Table 20.2: Access Route Development Cost

ITEM	TOTAL COST (US$ 000's)
Vertikalny Pit 1 CAF road	575
Mangazeisky North CAF road	598
Vertikalny – Mangazeisky North connecting road	123
TOTAL	**1,300**

20.2.2 Operating Cost Estimates

Open pit operating costs were estimated by WAI based on the generated production schedule, equipment operating cost estimates, consumable price estimates and labour estimates.

The operating costs were estimated on a per tonne of rock mined basis and broken down by operational activity. A summary of the overall open pit operating costs by centre is provided in Table 20.3, below.

Table 20.3: Open Pit Operating Costs by Centre			
COST CENTRE	**UNIT**	**COST**	**SPLIT**
Hauling	US$/t	0.61	28%
Blasting (Contractor)	US$/t	0.46	21%
Drilling	US$/t	0.39	18%
Loading & Stockpiling	US$/t	0.34	16%
General Mine Maintenance	US$/t	0.12	6%
Dozing & Grading	US$/t	0.12	5%
Engineering/Geology	US$/t	0.05	2%
Supervision & Technical	US$/t	0.05	2%
Other	US$/t	0.04	2%
TOTALS	**US$/$t_{MOVED}$**	**2.17**	**100%**
	US$/$t_{ORE}$	**53.88**	
	US$/$t_{WASTE}$	**2.27**	

WAI notes that additional equipment required to carry out the production schedule (Section 15.9.3) are treated as leased. Operating costs for these additional items of equipment include a mark-up factor of 25% to account for leasing, resulting in approximately 1.4% of the total operating unit costs shown in the table above.

Estimated overall open pit costs are in the region of US$2.17/t rock mined. Any costs not associated with mining activities are included in the financial analysis.

20.3 Underground Costs

20.3.1 Capital Costs

Underground capital costs were estimated based on WAI's cost database and project experience of similar operations. Estimated capital costs include mine development and mine equipment.

Mine development capital is inclusive of any mine development that is capitalised. The cost estimates are based on the completed mine designs and WAI's cost database. A summary of the mine development categories, unit costs and cost allocation are provided in Table 20.4, below.

Table 20.4: Underground Development Costs		
ITEM	**UNIT COST (US$/m)**	**COST ALLOCATION**
Access Decline	472	CAPEX
Level Access Drive	432	CAPEX
Ventilation Drive	432	CAPEX
Remuck Bay	694	CAPEX
Ventilation Raise	26	CAPEX
On-Vein Drive	433	OPEX

Equipment capital costs include the purchase of new equipment, initial spare parts inventory and sustaining capital for equipment overhaul. Overhauls were scheduled at 50% of the equipment operating life and costed at 40% of the initial equipment purchase price. Given the relatively short life of the underground operations, equipment overhauls were favoured over new equipment purchases.

A breakdown of the total capital costs incurred over the life of the underground project is provided in Table 20.5, below.

Table 20.5: Capital Expenditure Summary

ITEM	PRE-PROD (US$ 000's)	LOM (US$ 000's)	TOTAL (US$ 000's)
Capitalised Development	0.844	3.47	**4.31**
Mine Equipment - Purchase	10.12	6.08	**19.02**
Mine Equipment – Sustaining (Overhaul)	-	2.62	
Mine Equipment – First Fill & Spares (2%)	0.20	-	
TOTAL	**11.16**	**12.17**	**23.33**

A breakdown of the pre-production capital equipment purchase for the project is provided in Table 20.6, below.

Table 20.6: Pre-Production Underground Equipment Capital Expenditure (2021)

ITEM	UNIT COST (US$ 000's)	QTY	TOTAL COST (US$ 000's)
Development Jumbo – Single Boom	563	2	1,126
Load Haul Dump – 1.5m3	373	2	745
Underground Haul Truck – 20t	720	2	1,440
Explosives Truck	576	1	576
Small Motor Grader	288	1	288
Fuel & Lube Truck	576	1	576
Water Truck (Dust Suppression)	576	1	576
Underground 4x4	48	6	286
Scissor Lift	350	1	350
Primary Fan	750	4	3,000
Auxiliary Equipment, including: Secondary Fans & Starters Compressors Main Pump Face Pump Jumbo Boxes	-	-	1,158
First Fill & Initial Spares	-	-	202
TOTAL PRE-PRODUCTION CAPEX	**-**	**-**	**10,323**

20.3.2 Operating Cost Estimate

Mining operating costs were estimated by WAI, based on the mine designs, equipment operating cost estimates, consumable price estimates and labour estimates. A summary of the overall underground operating costs is provided in Table 20.7, below.

Table 20.7: Underground Operating Cost Summary			
ITEM	**UNIT**	**TOTAL COST**	**SPLIT**
Operating Development	US$M	5.18	15%
Operating Expenditure	US$M	19.73	58%
Personnel Salaries	US$M	9.17	27%
TOTAL OPEX	**US$M**	**34.08**	**100%**
	US$/tORE	**40.56**	

WAI notes that raise boring equipment was treated as leased in this study due to the high purchase price, life of the operation and anticipated workload. Operating costs for raise-boring include a mark-up factor of 50% to account for leasing. Overall underground mining costs are estimated to be in the region of US$40.56/t *ore* mined. Any costs not associated with mining activities are included in the financial analysis.

20.4 Processing Costs

20.4.1 Capital Costs

SBR provided a capital cost estimate for the proposed primary sulphide flowsheet of RUB 1,156,061,000 (approximately **US$17.3m**). This is considered reasonable for an approximate 500tpd operation. However, this is based on a new plant, independent from the current oxide plant. This may be required if it is desired to process both oxide ore and sulphide ore simultaneously. If the sulphide ore is to be processed after exhaustion of the oxide ores, then the capital cost can be significantly reduced by utilising most of the current installed equipment. In this case, the capital cost is estimated at approximately **US$9m** with the requirement for the new flotation circuit and additional crushing and grinding capacity. An additional cost of **US$2m** has been estimated to install a new XRT system on site.

20.4.2 Operating Costs

Table 20.8 below provides summary of the Project processing costs:

Table 20.8: Project Processing Opex Summary		
Ore Sorting Cost	US$ /t	2.25
Leach Plant (Current Plant)		
Unit Processing Cost (Oxides)	US$ /t	72.95
Unit Processing Cost (Sulphides)	US$ /t	123.71
Flotation Plant (New Plant)		
Unit Processing Cost (Sulphides)	US$ /t	47.18

The process operating cost has been estimated by SBR as **US$47.18/t,** based on the flotation testwork results and reagent consumptions, and is considered reasonable for use in the pit optimisation studies. This compares with the Tetra Tech design operating cost of **US$121.8/t** based on using the existing oxide plant, but with the modifications for finer grinding and additional leach residence time, with US$85.4/t contributed by the increased reagent consumptions (lime and cyanide in particular).

21 FINANCIAL ANALYSIS (ITEM 22)

21.1 Overview

WAI has undertaken a preliminary economic assessment of the Mangazeisky Project, using Discounted Cash Flow (DCF) analysis, from which the Net Present Value (NPV), payback period and other measures of project viability have been determined.

The financial analysis has been performed to reflect valuation as of the end of 2019 and does not include any sunk costs that have already been invested in the project.

The Project Internal Rate of Return (IRR) cannot be estimated due to more than one occurrence of the negative cash flows during the project life: initially at the end of 2019 and secondly in 2021. Despite current production relative stability, occurrence of the negative cash flows in 2021 is explained by additional capital expenditures required for completion of the new flotation plant construction, and production shortfall caused by transition from oxide ore to the sulphides.

The Project Financial Model ("Model") has been developed using the production schedule developed by WAI, with all costs being estimated in 2019 US Dollars based on the actual production data and available databases.

Forecasted fluctuating US Dollar (US$) and Ruble inflation rates have been applied appropriately to both commodity prices and project costs to provide financial results in nominal values.

All costs and cash flows reported in this section are shown in nominal US Dollars after inflation has been incorporated (unless stated otherwise), therefore costs appear different to the costs reported in the engineering sections above.

Summary of key input assumptions is outlined below.

21.2 Metal Prices

The main products from the Mangazeisky Project are proposed to be silver bullion and two concentrates: silver bearing zinc concentrate and silver bearing lead concentrate.

Price forecast as of 2019 has been used as the basis for the project assessment, with an appropriate inflation rate being included in valuation.

Table 21.1: **Commodity Price Assumptions**

Scenarios	Price Assumption (as of 2019)
Ag (US$ / oz)	17.76
Pb (US$ / t)	2,069
Zn (US$ / t)	2,252

21.3 Macroeconomic Parameters

The financial model has been developed using the macroeconomic parameters shown in Table 21.2.

Table 21.2: Macroeconomic Assumptions

Period	Y1 Q4	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Year	2019	2020F	2021F	2022F	2023F	2024F	2025F	2026F
RUB/USD	64.7	72.1	70.0	70.0	70.0	71.4	72.8	74.2
Annual Inflation for RUB	0.00%	4.70%	4.00%	4.00%	4.00%	4.00%	4.00%	4.00%
Estimated Cummulative - RUB		4.78%	9.80%	15.05%	19.65%	24.44%	29.41%	34.59%
Long Term Inflation USD	0.50%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%
Estimated Cummulative Inflation USD		2.00%	4.04%	6.12%	8.24%	10.41%	12.62%	14.87%

Data on exchange rates and Ruble inflation is used as per the SBR's corporate forecasts. US Dollar inflation rate applied as per WAI assumption.

21.4 Payment & Realisation Terms

Realisation terms for silver have been provided by the Client based on the actual data and products assumed to be sold to a smelter located in Kazakhstan. A summary of assumptions on lead and zinc concentrates payment terms is presented in Table 21.3 below.

Due to the limited data on impurities contained in concentrates, no penalties have been included in this valuation and that low lead grade assumptions in the concentrates will be offset by high silver grades.

Table 21.3: Project Payment Terms

Assay Payable		
Silver Net Assay Payable	%	98.00%
Pb and Ag Payable in Lead Concentrate	%	84.00%
Zn and Ag Payable in Zinc Concentrate	%	45.00%
Selling and Realisation		
Ag Selling Cost	US$/oz	0.4
Concentrate delivery and transportation	US$/wmt	274.9
Moisture Content	%	8%
Pb in Pb Concentrate	%	17.1%
Zn in Zn Concentrate	%	42.3%

WAI notes that concentrate treatment charges are considered to be covered by the payment terms outlined in the table above.

21.5 Processing Recovery Rates and Production Summary

Summary of the overall processing recovery rates and recovered metals is shown in Table 21.4 below:

Table 21.4: Summary of the Project Processing Recovery and Metals Production

Metals	Total Processing Recovery	Units	Mined	Recovered
Silver	82.47%	oz '000	26,774	22,081
Lead	68.81%	t	44,948	30,929
Zinc	94.09%	t	17,969	16,908

21.6 Capital Costs

Overall capital cost for the project have been estimated at US$43m. Summary of the Project Capital Cost is shown in Table 21.5 below.

Table 21.5: Project Capital Costs Summary (US$m, nominal total for the LOM)

Total Project Capital Costs, including	**43**
Mining Capex for Open Pit	2.5
Mining Capex for Underground	24.6
Leasing of Mining Equipment – Principal Repayment	4.7
Processing Plant Cost: Upgraded XRT and Flotation Plant VS New Plant	11.2

No plant sustaining cost or TSF costs have been included at this stage of valuation. WAI has also considered that all general infrastructure is already in place.

21.7 Operating Costs

The overall operating cost has been estimated at US$242.7M (nominal values). Summary of the costs is provided in Table 21.6 below.

Table 21.6: Less Operating Costs (US$M, nominal values)

Mining Cost	82.3
Plant Processing Cost	68.3
G&A	46.7
Mining Royalty (Mineral Extraction Tax)	45.0
Total Operating Cost LOM	**242.7**

Payments to reclamation and closure fund, total of US$4.2m payable in the last project year have been included into the financial model as provided by the Client.

21.8 Tax Regime

WAI has developed a post cash flow model where the tax regime shown in Table 21.7 has been implemented.

Carried forward losses from previous periods in the amount of CAD6.9m (as per IFRS data) or US$5.3m have been incorporated in the model for tax purposes.

Table 21.7: Project Tax Summary

	Rate	Total (US$M, nominal)
MET: Silver	6.5%	33.31
MET: Lead	8.0%	8.12
MET: Zinc	8.0%	3.57
Corporate Income Tax	20%	8.2

No VAT rebate has been considered in the financial model.

21.9 Financial Summary

Project financial summary is presented in Table 21.8 and Table 21.9 below.

Table 21.8: Key Project Technical and Economic Indicators

Gross Revenue	449
Less Realisation Costs	81
Net Revenue	368
Less Operating Costs	
Less Mining Cost	82.3
Less Plant Processing Cost	68.7
Less G&A	46.7
Less Mining Roylty Tax	45.0
Total Operating Cost LOM	**242.7**
EBITDA	125.5
Less Interest Cost (Leasing)	0.6
Less Depreciation & Amortisation	100.4
Less Payments to Reclamation Fund	4.2
EBT	20.3
Less Income Tax	8.2
Net Income	12
Plus Depreciation & Amortisation	100
Less Increase in Net Working Capital	0
Cash Flow from Operations	112
Less Capital Costs, including	43.0
Mining Capex for Open Pit	2.5
Mining Capex for Underground	24.6
Equipment Leasing	4.7
Processing Plant Upgrade Capital Cost	11.2
Pre-Tax Cash Flow	78
Post Tax Free Cash Flow	69

Table 21.9: Financial Project Summary		
NPV @ Discount Rate of 8.64%	US$ M	**46.51**
Ag Break-even price	US$/oz	14.11
NPV @ Discount Rate of 10%	US$ M	43.87
NPV @ Discount Rate of 15%	US$ M	35.77
NPV @ Discount Rate of 20%	US$ M	29.60
IRR	%	**N/A**
Payback period of capital (Discounted, Cumulative)	date	**Q3 2021**

The results from preliminary economic assessment show positive NPVs at various discount rates. Break-even silver price was estimated at US$14.11/oz which is 21% lower than the base case price assumption.

Current financial results have been derived from the production schedule that considers oxide material from stockpile No 5, in the amount of approximately 50kt.

An additional upside scenario with revised lead concentrate yield at 5% and upgraded lead concentrate quality to 66% resulted in improved economics with NPV at $58.7M at 8.64%. Although greater definition of concentrate products and other variables will be required to accept these concepts.

21.10 Sensitivity Analysis

A sensitivity analysis was performed on the key parameters within the financial model to assess the impact of changes upon the Net Present Value of the project (at a base case 8.64% discount rate). These parameters are as follows: metal prices; operating costs and capital costs. Each factor was variated within a range of +/-40% (while other parameters remained unchanged) to examine the sensitivity of the model to changing economic and operational conditions.

Sensitivity analysis results show that the Project is mostly sensitive to change in Ag price, as it forms the major part of the project revenue and production costs (mining and processing), and less sensitive to changes in the lead and zinc prices.

The Project is also significantly sensitive to mining operating costs (both OP and UG), and relatively less sensitive to processing operating costs.

Considering relatively low proportion of the remaining capital costs, the Project is seen to be least sensitive to changes in capex. No sunk costs have been included in this analysis and major part of the capex is considered to be already invested.

The results are shown in Table 21.10 and presented in Charts below (Figure 21.1).

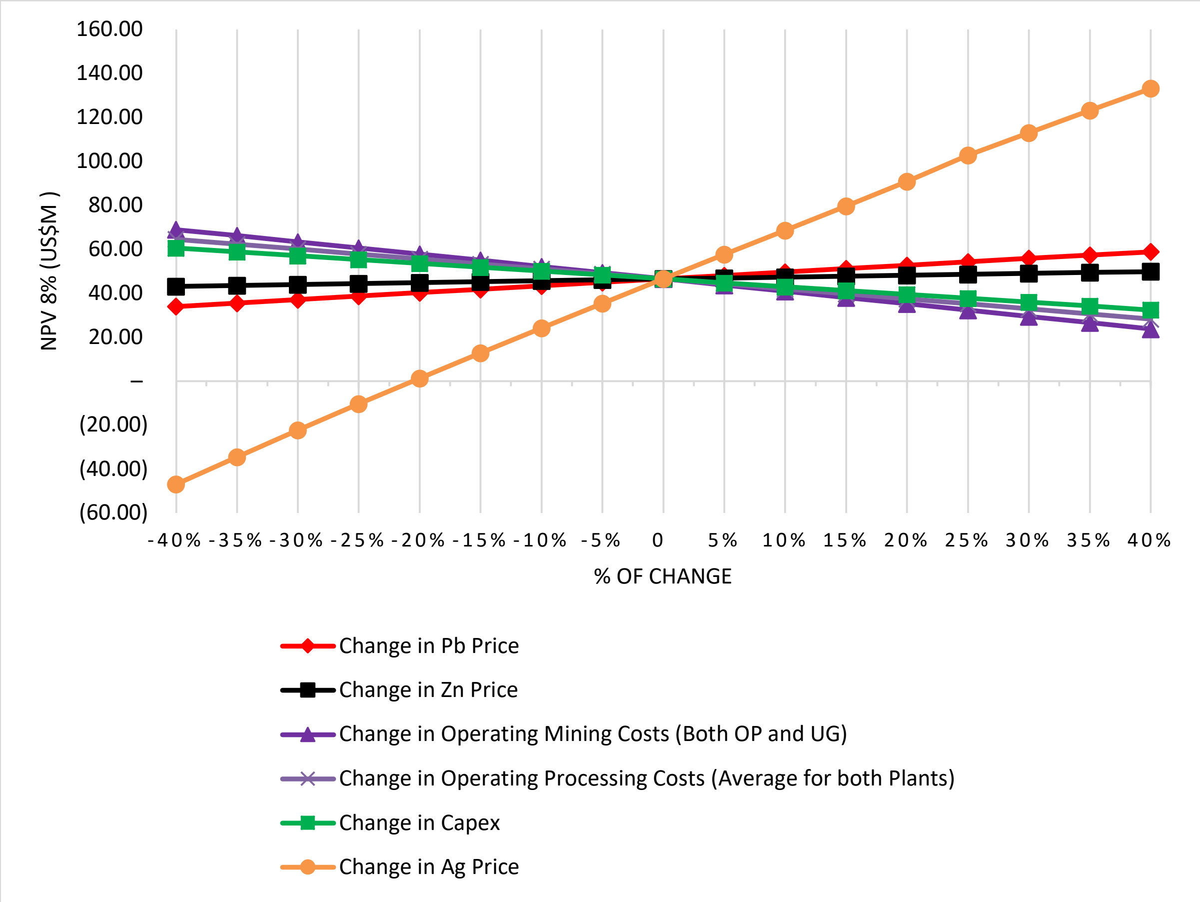


Figure 21.1: Project NPV (8.64%) Sensitivity Analysis Results

Table 21.10: Project NPV (8%) Sensitivity Analysis Results

	60%	75%	90%	100%	110%	125%	140%
Pb Price	1,241	1,552	1,862	2,069	2,276	2,586	2,897
NPV @ 8.64%	29.56	33.96	38.36	41.30	44.23	48.63	53.01
Zn Price	1,351	1,689	2,027	2,252	2,477	2,815	2,815
NPV @ 8.64%	43.12	44.39	45.66	46.51	47.35	48.62	49.89
Average Mining Opex	29.69	37.12	44.54	49.49	54.44	61.86	69.29
NPV @ 8.64%	68.98	60.58	52.14	46.51	40.86	32.31	23.73
Average Processing Opex	24.80	31.00	37.20	41.33	45.47	51.67	57.87
NPV @ 8.64%	64.58	57.80	51.02	46.51	41.98	35.15	28.30
Capex (US$ M, nominal)	25.80	32.25	38.71	43.01	47.31	53.76	60.21
NPV @ 8.64%	60.61	55.32	50.03	46.51	42.98	37.69	32.40
Ag Price	10.66	13.32	15.98	17.76	19.54	22.20	24.86
NPV @ 8.64%	-46.89	-10.30	24.10	46.51	68.60	102.84	133.14

22 ADJACENT PROPERTIES (ITEM 23)

WAI is not aware of any properties adjacent to the Mangazeisky EL.

23 OTHER RELEVANT DATA AND INFORMATION (ITEM 24)

24 RISKS AND OPPORTUNITIES (ITEM 25)

Areas of risk and opportunity material to the project are set out in Table 24.2 within the framework of the Strengths, Weaknesses, Opportunities and Threats (SWOT) analysis. The legend for the SWOT analysis is set out in Table 24.1.

Table 24.1: Legend for SWOT Analysis

	Element related to Data
	Element related to Geology and Mineral Resources
	Element related to Mining
	Element related to Processing and Infrastructure
	Element related to Financial
	Element related to Other Modifying Factors

Table 24.2: SWOT Analysis for the Vertikalny and North Mangazeisky Projects

Strengths	Adequate exploration SOPs and QA/QC procedures over 15 years since 2004 with good recovery of drill core. Low risk to provenance of data.
	Good reconciliation of grade control data over a nine-month period.
	Better definition of ore types and oxide/sulphide boundary since 2016 at Vertikalny.
	Density appears to be appropriately assigned to the model and is considered reasonable.
	Better confidence in Indicated resources as a result of metallurgical and infil drilling for Vertikalny.
	Issues in getting plant to early steady state much improved with installation of Merril Crowe circuit in parallel.
	Preliminary Economic Assessment of combined project has resulted in positive NPV at various discount rates.
	Assay results for blanks for silver show their possible contamination.
	Infil drilling as part of the 2017 campaign did not demonstrate continuity as modelled for the 2016 MRE in the Vertikalny Southern Pit area downdip nor across the gap with Central Pit area.
	A lack of Measured and Indicated Resources defined for North Mangazeisky.
	The mining schedule indicates a significant increase in ramp up of waste material to be moved during 2020/21 in order to expose enough ore will put pressure on existing haulage fleet and availability of equipment in order to strip the required volumes of material.
	The schedule runs short of oxide for direct haul from pit to crusher in Q3 2020. There is a gap in production until the flotation circuit comes on stream in mid-2021. Careful consideration needs to be given as to how this is managed through stockpile drawdown and blending, reducing throughput and bringing in oxide material from off-balance resources and additional sources. Planning to ensure such material is available to mine and of the necessary oxide content (>55% target and tested in advance) needs to be considered.
	Lack of detailed geotechnical data and analysis for the Mangazeisky North Pit.
	The mineable tonnage does not represent Ore Reserves.

Weaknesses	Insufficient geotechnical data and analysis to refine the underground geotechnical design criteria as derived for the Vertikalny deposit by SRK Consulting in 2014.
	Disconnect between steady state underground production rate of 110ktpa used in the Vertikalny underground mine design (based on the design parameters outlined in the Tetra Tech study dated 21-08-17) and the production rate target used by WAI in underground scheduling was 272ktpa.
	Geometallurgical uncertainties and a lack of representative testwork to support definition of ore types, particularly at N. Mangazeisky distinguishing oxide from primary ore.
	Lack of practical XRT ore sorter testwork conducted on bulk primary ore.
	Lack of mobile equipment to maintain schedule and manage different streams and throughputs feeding the ore sorter.
	The current schedule assumes minimal time for a final tie-in of the upgraded plant (flotation circuit, additional crushing and grinding capacity).
	Lack of testwork conducted on Mangazeisky primary ore to confirm flotation response.
	Lack of variability testwork conducted on Vertikalny primary ore for hardness/grindability.
	Lack of phase analytical testwork conducted to define ore types on Mangazeisky oxide ore.
	No penalties have been considered in the PEA valuation due to limited geological data and undefined payment terms.
Opportunities	Initiate representative phase analytical testwork on existing samples from N. Mangazeisky core to define the oxide/sulphide boundary. Subject to access, haulage and permitting, this would open up oxide resources amenable to fill the production gap as oxide runs out in Q3 2020.
	Low-grade stockpiled (stockpile no.5) oxide material may offer an opportunity to address the oxide feed gap indicated in the production schedule although further sampling and testing is recommended before being considered a viable source of feed to bridge the oxide production gap.
	XRT sorter presents an opportunity to increase recovery and reduce operating costs but has yet to be tested at a commercial scale on sulphide ore in particular
Threats	Downgrade of the previous MRE for Vertikalny at a 200g/t Ag cut-off grade for open pit by 3% on grade and 29% on tonnes if taking into account mined-out material. For UG resources at 300g/t Ag cut-off grade was decreased by 24% and tonnes by 56% due to re-interpretation of mineralisation.
	The downgrade has put pressure on the amenability of sulphide ore to be mined and increased the strip ratio.
	Should mining productivity or equipment capacity be lower than required to move waste during the pushback in Central Vertikalny, ore production may be adversely impacted and exacerbate the oxide feed gap.
	Should a smooth ramp-up period be required during construction of the flotation plant, actual metal production may be lower than that indicated in the production schedule; therefore, adversely impacting project economics.
	Underground development dimensions must be re-evaluated to accommodate the potentially larger equipment required to achieve the higher production rates.
	Greatest threat is understanding the processing characteristics of the sulphide ore scheduled for throughput for 2020/2021 through the existing plant. High risk that

	recoveries may be lower and more variable than scheduled depending on the amount of sulphide in the blend feeding the Merril/electrowinning circuit and hardness of the sulphide ore through the crusher feeding the concentrator. Testwork needs to be done on synthetic mixes of the expected blends of oxide:sulphide for this period.
	Effect of penalty elements in the final Pb and Zn concentrates and constraints on smelter contracts.
	Risk to sorter scheduling and ultimately production from low FEL availability. An additional FEL is recommended for the ore sorting area.
	Mining capital and operating cost estimates are based on a Preliminary Economic Assessment (PEA) level of confidence (±45%). The study offers a valuable view in determining the merits of pursuing further engineering studies but should not be the sole reference for the purposes of economic decision making.
	From the threat to understanding the processing characteristics of sulphide ore there is reliance on data for concentrates produced on sparse historical testwork data and subsequent risk to saleability of the final Pb concentrate products.

The following presents a synthesis of the major risks and recommendations for actions to mitigate. The matrix is presented in a tabular matrix format colour-coded so issues and high-risk areas can be readily flagged as follows:

Risk Category	Definition
	Critical (unquantifiable but warrants a halt to proceed pending critical decision)
	Significant (>= 30% negative impact on metal, costs or revenue)
	Moderate (>=10% and <=30% negative impact on metal, costs or revenue)
	Low (<10% negative impact on metal, costs or revenue)

ITEM	DESCRIPTION	STATUS	ISSUES	ACTIONS/MITIGATIONS	PRIORITY
1	**Licence Tenure**				
1.1	Security of Tenure	CSJC Prognoz is in possession of a mining licence YaKU 03626 BE for Vertikalniy. The license has an expiry date of 01.09.2033 and covers an area of 13.55 km^2 CSJC Prognoz is in possession of an exploration licence with the reference YaKU 12692 BP for North Mangazeiskiy. The license has an expiry date of 31.12.2023 and covers an area of 570 km^2.	None. Valid for silver extraction.	None	LOW
1.2	Compliance with Licence Agreement	Not considered	Assumed sub-soil licence compliant, no material violations in conditions to jeopardize terms of licence agreement.	None	
1.3	Project Permitting	Not considered	Assumed all necessary project and construction permits in place.	None	
2	**Resources and Reserves**				
2.1	Resource base Vertikalniy	As per Tables 13.22 and 13.23 effective 31.05.2019.	Downgrade of the previous MRE for Vertikalny at a 200g/t Ag cut-off grade for open pit by 3% on grade and 29% on tonnes if taking into account mined-out material. For UG resources at 300g/t Ag cut-off grade was decreased by 24% and tonnes by 56% due to re-interpretation of mineralisation.	No material change since effective date. Reasons for downgrade: • Re-interpretation of mineralized structures to incorporate new infill drilling. Lower global grade with more conservative search parameters but higher confidence with closer drill spacing;	LOW

				• A more conservative approach for Inferred resource definition; • Introduction of oxide/primary which was not distinguished in the TT resource. This has been important in drawing in a better-defined open pittable oxide resource and reclassified some of the TT indicated resource as inferred; • Using separate Net Smelter Return parameters for both oxide/primary and open pit/underground resource definition. • mineralisation boundary based on the recent testwork data	
2.2	Resource base Mangazeiskiy	As per Table 13.40 effective 31.05.2019.	Reclassification to inferred at 200g/t Ag cut-off grade due to a lack of definition of ore types on the deposit supported by testwork. Contained *in-situ* silver for Mangazeisky deposit reduced by 28%, average silver grade may be increased by 14%.	No material change since effective date. Reasons for change due to application of constraining wireframes and search parameters more appropriate to the style of mineralization but provides better consistency in distribution of silver grade.	MOD
2.3	Data Adequacy	Anomalous assay results from blank samples. Accuracy of Pb/Zn duplicates.	Potential contamination from high grade silver.		LOW
2.4	Reconciliation	Good reconciliation of grade control data over a nine-month period in 2019.	Short period and small population.	Study recommended to expand and include all long & short-term GC data.	LOW

3	**Mining Engineering**				
3.1	Mining Equipment	Current status of equipment deployed: • 1x CAT 336 DL Excavator; • 1x CAT 349 DL Excavator; • 1x Sunward SWDE-120Atl Blast rig; • 1x URB-2A2 truck mounted Blast rig; • 8x Scania G440 trucks; • 2x CAT D9R Dozers; • 2x CAT 950GC FELs; • 1x SEM-922 Grader	None. Fleet is adequately sized to meet future production in the conceptual schedule provided utilization, availability and maintenance is optimized.	May enhance and reduce risk through direct lease of replacement fleet from supplier(s) or contractor with own operators. Additional FEL recommended for sorting circuit to ensure availability In ore sorting area.	MOD

3.2	Production Scheduling	• Key stage in diverting equipment from Vertikalny South to Central pit to undertake pushback in 2021.
		
		• Production shortfall starting end Q3 2020 when oxide depletes to full commissioning of sulphide flotation plant in Q2 2021.
			As much attention needs to be given to waste haulage at this time as ore haulage at a time when several faces may need to be available to access/blend oxide ore.
			
			WAI accepts the shortfall can be addressed and the production gap narrowed but risk remains to production hiatus or lower recovery through the oxide plant as the result of blending sulphide material.
				• Ensure timely commissioning of sulphide plant.
				• Open up alternative sources of oxide as a back-up. This can be from;
				  - N Mangazeisky (reserves approved but not well defined with added transport costs and permitting)
				  - Vertikalny, extension to current open pits or near pit upside resources. (well defined but not necessarily approved).
					HIGH

4	**Geotechnical**				
4.1	Geotechnical	Basis of design at definition phase study level for Vertikalny underground. North Mangazeisky Open Pit	Study required to support underground design to establish rating of rock mass and stand-up for development, stopes and infrastructure. Needs greater definition and study for pit slope stability	Program of geotechnical drilling within next 2 years	LOW

5	**Metallurgy**	Processing characteristics of oxide and sulphide planned for transition period as oxide depletes and sulphide comes on stream. Penalty elements in Pb concentrate.	• Oxide ore well defined but process characteristics of transition/sulphide material not so well understood as scheduled for this period. Risk of variable and lower recoveries than estimated. • Lack of representative testwork to support definition of ore types, particularly at N. Mangazeisky distinguishing oxide from primary ore. Potential concentrates on smelter contract for Pb/Zn concentrate	Geometallurgical testwork incorporating bulk sampling required to inform the plant 1 month and eventually 1 week in advance.	HIGH
6	**Processing**				
6.1	Process Plant	• Merrill Crowe circuit installed in parallel with SXEW. • XRT sorter installed and undergoing commissioning. • Construction and schedule for sulphide flotation plant.	Demonstrable improvement in recovery and subsequent opcosts. Needs to be fully tested on a commercial basis with ore trialled through a separate line. Not assessed at time of writing.		MOD
6.2	Tailings Storage Facility (TSF)	Not Assessed as part of this exercise.			
7	**Infrastructure**	Not Assessed as part of this exercise.			
8	**Hydrology & Hydrogeology**	Level of definition of supporting studies	Current permafrost assumptions reasonable but requires verification and greater level of understanding of variability. Cannot assume zero flow in permafrost conditions.	As part of geotechnical study needs greater definition for surface water management and seasonal pit inflow and effect of Talikhs in the groundwater model across the site.	LOW
9	**Financial**				

9.1	Capital Costs	• Open Pit Capital Costs: **US$ 2.53M** • Underground Capital Costs: **US$ 23.33M.** • US$17.3M for 500 tpd new plant reducing to US$9M if the existing oxide circuit can be retrofitted.	Cost assumptions for financial modelling are reasonable at a PEA level of accuracy.		LOW
9.2	Operating Costs Mining	• Open Pit Operating Costs: **US$ 2.17 /tMINED** • Underground Operating Cost: **US$ 40.56/tORE**	Cost assumptions for financial modelling are reasonable at a PEA level of accuracy. These costs do not reflect cost parameters used in NPV optimisation which use actual operating cost numbers prior to November 2019. Financial model parameters are more optimistic than the NPV optimisation parameters used to constrain the open pit resources in the MRE.	See below.	MOD
9.3	Operating Costs Processing	Total **US$47.18/t** concentrate for financial analysis compared with Tetra Tech design opcost of **US$121.8/t**. Assumptions based on improvements in oxide plant, finer grind and optimal reagent consumptions. YTD opcost of **US$74/t** used in NPV optimization.	Cost assumptions for financial modelling are reasonable at a PEA level of accuracy. These costs do not reflect cost parameters used in NPV optimisation which use actual operating cost numbers prior to November 2019. Financial model parameters are more optimistic than the NPV optimisation parameters used to constrain the open pit resources in the MRE.	Financial Model needs greater definition and level of accuracy from 'steady state' G&A and process costs once data has been fed back from the expected improvements (oxide processing, sorting, sulphide flotation etc).	HIGH

25 CONCLUSIONS & RECOMMENDATIONS (ITEM 26)

25.1 Vertikalny - Mineral Resource Estimate

In WAI opinion, the established understanding of the geological and grade continuity is sufficient to support the classification of the Mineral Resources as Measured Indicated and Inferred.

At Vertikalny, 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. Underground Mineral Resources located below the base of the optimised pit shell and above the NSR cut-off value of US$162.0/t.

Mineral Resources are estimated as of 31 May 2019 based on an open pit mine survey of the same date.

25.2 Mangazeisky North – Mineral Resource Estimate

Since it is impossible to delineate and determine the geometry of oxide and primary mineralization at Northern Mangazeisky, WAI believes that the silver, lead, and zinc resources can only be classified as Inferred.

At Northern Mangazeisky, 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.

Mineral Resources are estimated as of 31 May 2019.

25.3 Hydrological & Hydrogeological Review

The following comments are made based on the work completed:

- The assumption that the underground mine will be dry with negligible ground water inflow ("Tetra Tech 2017 pp.16-74") needs to be confirmed. The assumption is based on limited mine data, extrapolation of permafrost base levels and a homogenous distribution of hydraulic property values and geometry. It is probable given the increased depth of the underground workings in Vertikalny Zones 1 and 4 that freeflowing groundwater will be encountered in lower levels.
- The occurrence of artesian conditions in boreholes below the permafrost in the Sirilendzhe River valley demonstrates the confining behaviour of the permafrost isolating the aquifer from surface waters across most of the catchment. We have not seen any comment however on the potential for elevated porewater pressures below the permafrost and whether this could be a modifying factor to mining.
- The overall conclusions about the permafrost are reasonable based on the data available for the open pit but require verification. More understanding of the

potential heterogeneity of hydraulic properties across the pit area is required. Modifiers that may affect groundwater in the pit include preferential flow zones, alteration and mineralisation, hydro-stratigraphy (layering) and subordinate structures and fracture zones. Permafrost behaviour may be substantially altered where there are conduits such as fault and fracture zones creating mechanisms for groundwater circulation or recharge. The permafrost distribution will likely change once the pit has been developed and new thermal equilibria are established.

- It is agreed that the placement of the proposed water supply borehole near borehole GS15-05 remains the most suitable location on the basis of yield and supply.

25.4 Geotechnical Review

WAI has carried out a review of the geotechnical information provided by Silver Bear Resources (SBR) for the Vertikalny and Mangazeisky North deposits. The review has aimed to summarise the geotechnical parameters for use in mine optimisation and design. Information was drawn from the findings of the geotechnical study carried out by SRK consulting in late 2014. WAI has not carried out a site visit, nor has it carried out an independent review of the geotechnical data used in the SRK study.

The geotechnical characteristics of the Vertikalny rock mass are considered to be suitably detailed and well defined. The open pit design parameters were defined by SRK based on kinematic and numerical slope stability analysis. The underground design parameters were taken from the Tetra Tech study; having originally been derived from the SRK study. The underground design parameters were defined by SRK using industry standard techniques; inclusive of Barton's Q system, Mathew's stability graph method and numerical modelling. The geotechnical work was underpinned by relatively robust geotechnical dataset collected by SRK in support of the study.

The geotechnical characteristics of the Mangazeisky North deposit are poorly defined. WAI were unable to gather any detailed structural or rock mas strength data. Consequently, the derived mine optimisation and design parameters were based on a standard WAI base case; not detailed geotechnical analysis. A geotechnical data collection exercise will be required to support further geotechnical analysis and substantiate any derived mine optimisation and design criteria.

25.5 NSR Model

A basic Net Smelter Return (NSR) calculation was performed which considered grade, metal price, metallurgical recovery, and metal payability. The payable metal includes the applicable concentrate and refining charges but does not include price participation or penalty element payments. The metal price assumptions were derived by WAI and approved by SBR. All metallurgical recoveries/costs used in the NSR calculation are based on data provided by SBR.

WAI notes that only the sulphide blocks have considered the value contributions of each payable element. This is based on the premise that most of the sulphide blocks will be processed through a flotation plant; following depletion of the oxide blocks which form a relatively contiguous volume within the current Vertikalny pit. Oxide blocks have only considered the value contribution of silver.

NSR factors were calculated and directly applied to each block within the Resource block models. This enabled the subsequent mine optimisation exercises to be carried out on the block NSR values. The NSR model forms a critical input into the development of the mining study and further detail regarding the NSR inputs must be understood to enhance the confidence of the study.

The key recommendations to improve the confidence of the NSR model are listed below:

- Marketability of concentrate products (especially lead concentrate due to low lead assay);
- Identifiy concentrate off-takers and generation of agreements in principle; and,
- NSR input parameters (i.e., concentrate moisture content, metal payability, metal deductions and penalties, transport costs, treatment, and refining charges, etc.).

25.6 Open Pit Mining

WAI has carried out an open pit mining study to define a mineable tonnage estimate for the Vertikalny and Mangazeisky North deposits.

Open pit optimisation was carried out using the Datamine NPV Scheduler v4 (NPVS) software package. Pit optimisations were carried out on the Resource block models generated for the two deposits and driven on the calculated block NSR values. The optimisations included *Measured, Indicated* and *Inferred* resources.

Detailed mine designs were generated from the selected optimal shells using the Datamine Studio OP V2.4 general mine planning package. The designs were used to derive the mineable tonnage estimates and formed the basis for subsequent production scheduling. It should be noted that 'minable tonnage estimates' are not Ore Reserves and are not demonstrative of technical and economic viability.

The key recommendations to improve the confidence of the open pit mining study are listed below:

- Further refine the access requirements for Vertikalny Pit1 and Mangazeisky North pit;
- Conduct dilution and loss study specific to the Mangazeisky North pit;
- Generate and implement new pit design criteria for the Mangazeisky North pit following geotechnical data collection, investigation, and analysis;
- Carry out waste dump design and positioning exercise to improve confidence in the waste disposal strategy; and,
- Carry out optimisation on *Measured* and *Indicated* Resources to determine influence of *Inferred* Resources and identify measures to improve geological confidence.

25.7 Underground Mining

WAI has carried out a mining study to define an underground mineable tonnage estimate for the Vertikalny deposit. The study has considered the volume of mineralised material below the generated Vertikalny pit designs.

Underground mineable tonnage estimates were prepared using the Vertikalny Resource block model. Stope optimisation was completed using the Mineable Shape Optimiser (MSO) module in the Datamine Studio 5D Planner software package. The optimisations included *Measured, Indicated* and *Inferred* resources.

A total of four underground mining zones were designed in line with generated stope zones. The designs were used to derive the mineable tonnage estimates and formed the basis for subsequent production scheduling. It should be noted that 'minable tonnage estimates' are not Ore Reserves and are not demonstrative of technical and economic viability.

The key recommendations to improve the confidence of the underground mining study are listed below:

- Further geotechnical studies are required to optimise the stope dimensions, identify the in-situ pillar requirements to ensure regional underground stability, identify stand-off distance of access declines from mineralised zones, etc.;
- Ventilation studies are required to understand airflow requirements, identify suitable primary/secondary fan sizes, generate more detailed ventilation costs, etc.; and,
- The original Tetra Tech design was carried out on the basis of resource estimates which have since been downgraded due to revised geological conditions. It will be necessary to carry out further stope optimisation on *Measured* and *Indicated* Resources to determine influence of *Inferred* Resources and identify measures to improve geological confidence.
- Underground development dimensions used in the Vertikalny underground mine design were based on the design parameters outlined in the Tetra Tech study (dated 21-08-17). The Tetra Tech study assumed a steady state underground production rate of 110ktpa. The production rate target used by WAI in underground scheduling was 272ktpa. This is due to the higher capacity of the new flotation plant (180ktpa) and the presence of an upstream ore sorter which rejects approximately 33% of ROM plant feed. Underground development dimensions must be re-evaluated to accommodate the potentially larger equipment required to achieve the higher production rates.

25.8 Mine Production Scheduling & Equipment Requirements

The generated mine designs were used as the basis for developing a combined open pit and underground production schedule. Effort was made to sequence the operations such that a steady flow of plant feed is maintained over the life-of-mine. Key points noted from the generated production schedule include:

- Overall mine life anticipated at just over 8 years,
- Depletion of oxide feed from Vertikalny pit anticipated at the end of Y2 (2020); indicating the point at which floatation plant would likely need to be established,

- Mining at Mangazeisky North anticipated to commence in Q3 of Y3 (2021) with production ceasing at the start of Y5(2023),
- Underground pre-production development anticipated to start at the end of Y3 (2021) with stope production commencing at the start of Y5 (2023).

The permitting requirements and minimum time required to commence mining at the Managzeisky North deposit must be understood.

Open pit and underground mining equipment requirements were estimated on first principles analysis to achieve the generated production schedule. No ventilation studies were carried out for the underground mining operations and it is recommended that such studies be considered in more detailed engineering studies.

25.9 Capital and Operating Costs – Mining

A mining cost model was developed to assess the open pit and underground mining capital and operating expenditures for the Mangazeisky Project. The cost estimates were developed by WAI based on data provided by SBR and WAI's internal cost database.

A summary of the costs is presented below:

Open Pit Capital Costs:	**US$2.53M**
Open Pit Operating Costs:	**US$2.17 /$t_{MINED}$**
Underground Capital Costs:	**US$23.33M**
Underground Operating Cost:	**US$40.56/$t_{ORE}$**

The calculated mining cost estimates are lower than those used in open pit and underground optimisation; implying a degree of margin within the generated mine designs. Given the level of study, WAI consider the differences in costs to be acceptable

The calculated costs are estimated to have an accuracy equivalent to a Preliminary Economic Assessment (PEA) level of detail. The study offers a valuable view in determining the merits of pursuing further engineering studies but should not be the sole reference for the purposes of economic decision making.

25.10 Processing

After producing first silver production in April 2018, silver recoveries have generally been in the range of 60-70%, compared to 85% design, but since April 2019 have been steadily increasing to >82% in July. This is thought to be due mainly to better washing of the leach tailings solids filter cake, where Benitex reported that up to 19% of the silver was previously being lost due to poor washing. There is also likely an impact due to primary ore being included in the oxide feed, reportedly 5-15% according to SBR. Higher cyanide concentrations of 5,000ppm are being utilised to allow for this, compared to the design of 2,000ppm.

Therefore, WAI recommends that the design silver recovery of 85% for oxide ore is appropriate to be used for pit optimisation studies.

The additional impact of any primary ore in the oxide feed will be higher reagent consumptions and moderate increases in cyanide and steel ball consumption are noted compared to design.

The lime consumption, however, is significantly higher than design, although this appears to be due to an incorrect design figure of 0.7kg/t used in the feasibility study, compared to the testwork data of 20-30kg/t, which translates to an expected field consumption rate of approximately 15kg/t. Further issues contributing to the actual lime consumption of 23.9kg/t are low activity and inefficient dosing, so there is scope to reduce the lime consumption. Overall process unit costs are also higher due to the lower throughput compared to design.

However, at this stage, WAI recommends using the actual YTD process operating cost of US$74.9/t for oxide ore for pit optimisation studies.

For the proposed processing of primary sulphide ore, the process design incorporates a new flotation circuit for the production of separate lead and zinc concentrates, with cyanide leaching of the lead flotation middlings as per the current circuit configuration. Most of the existing circuit can be utilised with the addition of the new flotation circuit and extra crushing and milling capacity.

The capital cost of approximately US$17.3M is considered reasonable for an approximate 500 tpd new operation, although this reduces to approximately US$9M if the existing oxide circuit can be used and the additional equipment retro fitted. Much will depend on whether there is a requirement to process both oxide and sulphide ores at the same time, or whether sulphide processing can start after oxide resources are depleted.

25.11 Financial Analysis

Preliminary Economic Assessment of Mangazeisky project has resulted in positive NPV at various discount rates. The project is mostly sensitive to change in Silver prices. Base Case NPV @ Discount Rate of 8.64% was estimated at US$46.51m (nominal values).

The Project is mostly sensitive to changes in Silver prices. Break-even price of the Project has been estimated at US$14.11/oz, which is 21% lower than the base case silver price assumption.

Current financial results have been derived from the production schedule that considers oxide material from stockpile No 5, in the amount of approximately 50kt.

WAI notes that no penalties have been considered in the PEA valuation and includes the approximate estimate of the payable metal content. This is due to limited geological data on penalty elements, concentrate characteristics based on limited historical testwork results and lack of potential off-take agreements with buyers given lead and zinc concentrates are not going to be produced earlier than Q4 2021. Hence there is a downside risk in the marketability of the lead concentrate.

Upside potential is seen as significantly improved concentrate quality and consequently improved project economics, should further testwork confirm better concentrate grade.

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APPENDIX 1: VERTIKALNY - QUANTILE ANALYSIS

Quantile Analysis of Silver Grades for Individual Zones								
Zone	Q%_from	Q%_to	Qty of samples	Ave	Min	Max	Accumulated metal	Accumulated metal (%)
1	0	10	107	39.66	4.20	62.00	4 243.44	0.40
1	10	20	108	78.10	62.00	96.23	8 434.98	0.79
1	20	30	108	119.32	96.80	143.00	12 886.53	1.21
1	30	40	107	199.23	143.43	264.00	21 317.16	2.00
1	40	50	108	335.33	264.08	415.02	36 215.42	3.39
1	50	60	108	525.45	416.35	638.80	56 748.70	5.32
1	60	70	107	832.90	641.50	1 024.10	89 120.68	8.35
1	70	80	108	1 286.68	1 025.00	1 551.00	138 961.10	13.02
1	80	90	108	1 980.38	1 567.00	2 627.41	213 880.87	20.05
1	90	100	108	4 491.80	2 650.10	11 832.50	485 114.01	45.47
1	90	91	10	2 722.68	2 650.10	2 865.34	27 226.77	2.55
1	91	92	11	2 993.57	2 934.70	3 060.77	32 929.24	3.09
1	92	93	11	3 203.13	3 085.00	3 340.00	35 234.45	3.30
1	93	94	11	3 424.88	3 366.50	3 481.50	37 673.67	3.53
1	94	95	11	3 591.89	3 495.63	3 808.00	39 510.75	3.70
1	95	96	10	3 976.01	3 816.00	4 235.00	39 760.12	3.73
1	96	97	11	4 598.66	4 257.00	4 860.00	50 585.23	4.74
1	97	98	11	5 126.21	4 861.25	5 546.00	56 388.36	5.29
1	98	99	11	6 229.79	5 765.16	6 804.63	68 527.66	6.42
1	99	100	11	8 843.43	6 844.76	11 832.50	97 277.76	9.12
1	0	100	1 077	990.64	4.20	11 832.50	1 066 922.89	100
2	0	10	52	11.04	-	28.10	574.31	0.25
2	10	20	52	58.10	28.83	73.87	3 021.00	1.32
2	20	30	52	85.57	74.25	98.20	4 449.51	1.94
2	30	40	52	112.64	98.60	130.50	5 857.09	2.56
2	40	50	52	155.62	131.00	185.00	8 092.00	3.54
2	50	60	52	216.50	185.00	246.75	11 258.21	4.92
2	60	70	52	330.04	248.15	409.00	17 162.26	7.50
2	70	80	52	515.53	409.95	634.00	26 807.39	11.72
2	80	90	52	856.54	659.00	1 179.50	44 540.26	19.46
2	90	100	52	2 058.90	1 194.00	5 185.00	107 062.85	46.79
2	90	91	5	1 208.20	1 194.00	1 224.50	6 041.01	2.64
2	91	92	5	1 309.86	1 259.20	1 335.50	6 549.32	2.86
2	92	93	5	1 386.64	1 350.50	1 424.00	6 933.22	3.03
2	93	94	5	1 494.90	1 432.25	1 549.50	7 474.50	3.27
2	94	95	6	1 665.01	1 587.95	1 712.00	9 990.09	4.37
2	95	96	5	1 776.93	1 720.50	1 825.70	8 884.63	3.88
2	96	97	5	2 105.07	2 053.00	2 173.20	10 525.36	4.60
2	97	98	5	2 311.11	2 219.89	2 475.00	11 555.55	5.05
2	98	99	5	2 850.57	2 574.04	3 525.80	14 252.84	6.23
2	99	100	6	4 142.72	3 691.00	5 185.00	24 856.34	10.86
2	0	100	520	440.05	-	5 185.00	228 824.88	100
3	0	10	5	24.15	5.99	54.65	120.73	0.47
3	10	20	6	85.00	67.35	111.20	510.01	1.97
3	20	30	5	140.82	112.00	155.75	704.11	2.72
3	30	40	6	198.53	183.00	209.00	1 191.20	4.60
3	40	50	5	221.70	213.00	226.00	1 108.50	4.28

3	50	60	6	306.91	271.67	330.00	1 841.47	7.11
3	60	70	5	408.78	341.00	570.50	2 043.88	7.89
3	70	80	6	668.65	585.20	796.08	4 011.90	15.49
3	80	90	5	938.26	801.60	1 090.00	4 691.30	18.12
3	90	100	6	1 612.12	1 275.00	2 229.18	9 672.69	37.35
3	91	92	1	1 275.00	1 275.00	1 275.00	1 275.00	4.92
3	93	94	1	1 444.50	1 444.50	1 444.50	1 444.50	5.58
3	94	95	1	1 490.00	1 490.00	1 490.00	1 490.00	5.75
3	96	97	1	1 549.01	1 549.01	1 549.01	1 549.01	5.98
3	98	99	1	1 685.00	1 685.00	1 685.00	1 685.00	6.51
3	99	100	1	2 229.18	2 229.18	2 229.18	2 229.18	8.61
3	0	100	55	470.83	5.99	2 229.18	25 895.78	100
4	0	10	9	39.14	4.55	80.00	352.22	0.69
4	10	20	9	97.43	87.68	110.67	876.83	1.73
4	20	30	9	128.63	114.91	139.90	1 157.69	2.28
4	30	40	9	185.80	140.00	244.00	1 672.17	3.30
4	40	50	9	292.94	267.92	339.06	2 636.46	5.20
4	50	60	9	391.43	341.18	434.28	3 522.91	6.94
4	60	70	9	512.73	443.00	624.00	4 614.58	9.09
4	70	80	9	799.88	645.78	901.54	7 198.95	14.19
4	80	90	9	1 087.54	940.94	1 342.50	9 787.88	19.29
4	90	100	9	2 102.18	1 429.61	2 839.94	18 919.59	37.29
4	91	92	1	1 429.61	1 429.61	1 429.61	1 429.61	2.82
4	92	93	1	1 442.16	1 442.16	1 442.16	1 442.16	2.84
4	93	94	1	1 643.89	1 643.89	1 643.89	1 643.89	3.24
4	94	95	1	1 980.89	1 980.89	1 980.89	1 980.89	3.90
4	95	96	1	1 987.00	1 987.00	1 987.00	1 987.00	3.92
4	96	97	1	2 309.00	2 309.00	2 309.00	2 309.00	4.55
4	97	98	1	2 559.98	2 559.98	2 559.98	2 559.98	5.05
4	98	99	1	2 727.12	2 727.12	2 727.12	2 727.12	5.37
4	99	100	1	2 839.94	2 839.94	2 839.94	2 839.94	5.60
4	0	100	90	563.77	4.55	2 839.94	50 739.28	100
5	0	10	1	80.74	80.74	80.74	80.74	2.42
5	10	20	1	108.00	108.00	108.00	108.00	3.23
5	20	30	1	118.99	118.99	118.99	118.99	3.56
5	30	40	2	195.00	171.00	219.00	390.00	11.67
5	40	50	1	234.00	234.00	234.00	234.00	7.00
5	50	60	1	235.50	235.50	235.50	235.50	7.05
5	60	70	2	244.98	241.00	248.96	489.96	14.66
5	70	80	1	248.96	248.96	248.96	248.96	7.45
5	80	90	1	376.20	376.20	376.20	376.20	11.26
5	90	100	2	530.00	530.00	530.00	1 060.00	31.71
5	94	95	1	530.00	530.00	530.00	530.00	15.86
5	99	100	1	530.00	530.00	530.00	530.00	15.86
5	0	100	13	257.10	80.74	530.00	3 342.35	100
6	0	10	3	54.97	52.00	57.91	164.91	1.31
6	10	20	3	66.10	64.00	67.29	198.29	1.57
6	20	30	4	88.21	68.83	102.00	352.83	2.79
6	30	40	3	113.09	102.00	125.00	339.26	2.69
6	40	50	4	155.41	128.65	170.00	621.65	4.92
6	50	60	3	186.73	176.20	192.00	560.20	4.44

6	60	70	3	281.49	268.12	305.36	844.48	6.69
6	70	80	4	384.60	314.58	477.01	1 538.41	12.19
6	80	90	3	681.87	490.52	802.40	2 045.62	16.20
6	90	100	4	1 489.95	881.00	2 450.20	5 959.80	47.20
6	92	93	1	881.00	881.00	881.00	881.00	6.98
6	94	95	1	1 154.70	1 154.70	1 154.70	1 154.70	9.15
6	97	98	1	1 473.90	1 473.90	1 473.90	1 473.90	11.67
6	99	100	1	2 450.20	2 450.20	2 450.20	2 450.20	19.41
6	0	100	34	371.34	52.00	2 450.20	12 625.45	100
7	10	20	1	3.00	3.00	3.00	3.00	0.09
7	30	40	1	243.80	243.80	243.80	243.80	7.55
7	50	60	1	304.00	304.00	304.00	304.00	9.41
7	70	80	1	1 090.00	1 090.00	1 090.00	1 090.00	33.74
7	90	100	1	1 590.00	1 590.00	1 590.00	1 590.00	49.21
7	99	100	1	1 590.00	1 590.00	1 590.00	1 590.00	49.21
7	0	100	5	646.16	3.00	1 590.00	3 230.80	100
8	10	20	1	106.77	106.77	106.77	106.77	7.72
8	30	40	1	122.00	122.00	122.00	122.00	8.82
8	50	60	1	198.80	198.80	198.80	198.80	14.37
8	70	80	1	366.40	366.40	366.40	366.40	26.49
8	90	100	1	589.00	589.00	589.00	589.00	42.59
8	99	100	1	589.00	589.00	589.00	589.00	42.59
8	0	100	5	276.59	106.77	589.00	1 382.97	100
9	10	20	1	87.00	87.00	87.00	87.00	4.80
9	20	30	1	89.80	89.80	89.80	89.80	4.95
9	30	40	1	96.50	96.50	96.50	96.50	5.32
9	40	50	1	143.00	143.00	143.00	143.00	7.88
9	60	70	1	169.00	169.00	169.00	169.00	9.32
9	70	80	1	215.50	215.50	215.50	215.50	11.88
9	80	90	1	243.60	243.60	243.60	243.60	13.43
9	90	100	1	769.50	769.50	769.50	769.50	42.42
9	99	100	1	769.50	769.50	769.50	769.50	42.42
9	0	100	8	226.74	87.00	769.50	1 813.90	100

Quantile Analysis of Lead Grades for Individual Zones

Zone	Q%_from	Q%_to	Qty of samples	Ave	Min	Max	Accumulated metal	Accumulated metal (%)
1	0	10	86	0.03	-	0.08	2.78	0.17
1	10	20	87	0.11	0.08	0.15	9.51	0.59
1	20	30	86	0.19	0.15	0.24	16.56	1.02
1	30	40	87	0.30	0.24	0.37	26.25	1.62
1	40	50	86	0.46	0.38	0.56	39.47	2.44
1	50	60	87	0.70	0.56	0.88	61.25	3.78
1	60	70	86	1.08	0.88	1.34	93.31	5.76
1	70	80	87	1.73	1.34	2.26	150.84	9.31
1	80	90	86	3.16	2.27	4.47	272.04	16.79
1	90	100	87	10.90	4.61	28.29	948.56	58.53
1	90	91	8	4.79	4.61	4.98	38.30	2.36
1	91	92	9	5.21	5.00	5.35	46.86	2.89
1	92	93	9	5.80	5.49	6.30	52.19	3.22
1	93	94	8	6.96	6.39	7.54	55.66	3.43
1	94	95	9	7.96	7.57	8.77	71.67	4.42
1	95	96	9	10.20	9.10	11.12	91.84	5.67
1	96	97	8	12.17	11.32	13.12	97.32	6.01
1	97	98	9	14.58	13.34	15.64	131.26	8.10
1	98	99	9	16.98	16.16	18.15	152.78	9.43
1	99	100	9	23.41	18.27	28.29	210.68	13.00
1	0	100	865	1.87	-	28.29	1 620.56	100
2	0	10	37	0.07	0.01	0.11	2.58	0.39
2	10	20	38	0.17	0.11	0.23	6.57	0.99
2	20	30	38	0.30	0.24	0.36	11.38	1.72
2	30	40	37	0.45	0.37	0.54	16.70	2.53
2	40	50	38	0.68	0.55	0.86	25.89	3.92
2	50	60	38	0.99	0.87	1.12	37.74	5.71
2	60	70	37	1.28	1.12	1.50	47.26	7.15
2	70	80	38	1.87	1.50	2.46	71.05	10.75
2	80	90	38	3.34	2.47	4.59	127.10	19.23
2	90	100	38	8.29	4.79	19.83	314.85	47.62
2	90	91	3	4.85	4.79	4.90	14.54	2.20
2	91	92	4	5.35	4.90	5.60	21.42	3.24
2	92	93	4	5.83	5.63	5.91	23.31	3.53
2	93	94	4	6.23	6.10	6.45	24.92	3.77
2	94	95	4	6.86	6.48	7.33	27.43	4.15
2	95	96	3	7.81	7.71	7.90	23.43	3.54
2	96	97	4	8.42	8.14	8.60	33.67	5.09
2	97	98	4	9.07	8.90	9.21	36.27	5.49
2	98	99	4	10.71	10.18	11.57	42.83	6.48
2	99	100	4	16.76	14.47	19.83	67.03	10.14
2	0	100	377	1.75	0.01	19.83	661.12	100
3	0	10	4	0.02	0.01	0.06	0.10	0.05
3	10	20	4	0.51	0.41	0.64	2.06	1.07
3	20	30	5	0.90	0.70	1.05	4.48	2.32
3	30	40	4	1.25	1.14	1.40	5.02	2.60
3	40	50	5	2.19	1.40	2.86	10.94	5.66
3	50	60	4	3.21	2.91	3.47	12.86	6.65

3	60	70	4	4.08	3.76	4.35	16.30	8.44
3	70	80	5	6.68	5.03	7.62	33.38	17.27
3	80	90	4	9.74	9.10	10.80	38.97	20.17
3	90	100	5	13.83	10.88	16.50	69.13	35.78
3	91	92	1	10.88	10.88	10.88	10.88	5.63
3	93	94	1	12.65	12.65	12.65	12.65	6.55
3	95	96	1	12.70	12.70	12.70	12.70	6.57
3	97	98	1	16.40	16.40	16.40	16.40	8.49
3	99	100	1	16.50	16.50	16.50	16.50	8.54
3	0	100	44	4.39	0.01	16.50	193.23	100
4	0	10	7	0.05	-	0.10	0.33	0.35
4	10	20	8	0.15	0.11	0.20	1.22	1.32
4	20	30	8	0.24	0.21	0.27	1.96	2.11
4	30	40	8	0.33	0.27	0.42	2.66	2.87
4	40	50	8	0.51	0.44	0.56	4.09	4.41
4	50	60	8	0.62	0.56	0.68	4.95	5.34
4	60	70	8	0.79	0.70	0.92	6.30	6.79
4	70	80	8	1.31	0.96	1.62	10.52	11.34
4	80	90	8	2.38	1.86	3.17	19.01	20.49
4	90	100	8	5.22	3.25	9.18	41.74	44.99
4	91	92	1	3.25	3.25	3.25	3.25	3.50
4	92	93	1	3.38	3.38	3.38	3.38	3.64
4	93	94	1	3.42	3.42	3.42	3.42	3.68
4	94	95	1	3.93	3.93	3.93	3.93	4.23
4	96	97	1	4.84	4.84	4.84	4.84	5.22
4	97	98	1	6.43	6.43	6.43	6.43	6.93
4	98	99	1	7.33	7.33	7.33	7.33	7.90
4	99	100	1	9.18	9.18	9.18	9.18	9.89
4	0	100	79	1.17	-	9.18	92.78	100
5	10	20	1	0.01	0.01	0.01	0.01	0.09
5	20	30	1	0.08	0.08	0.08	0.08	0.75
5	30	40	1	0.23	0.23	0.23	0.23	2.14
5	40	50	1	0.23	0.23	0.23	0.23	2.14
5	50	60	1	0.27	0.27	0.27	0.27	2.50
5	60	70	1	0.67	0.67	0.67	0.67	6.25
5	70	80	1	2.80	2.80	2.80	2.80	26.10
5	80	90	1	3.22	3.22	3.22	3.22	30.01
5	90	100	1	3.22	3.22	3.22	3.22	30.01
5	99	100	1	3.22	3.22	3.22	3.22	30.01
5	0	100	9	1.19	0.01	3.22	10.73	100
6	0	10	3	-	-	-	-	-
6	10	20	3	0.02	-	0.04	0.07	0.07
6	20	30	3	0.04	0.04	0.05	0.13	0.12
6	30	40	3	0.10	0.08	0.13	0.29	0.28
6	40	50	4	0.19	0.15	0.29	0.77	0.75
6	50	60	3	0.36	0.31	0.39	1.09	1.06
6	60	70	3	0.67	0.42	0.87	2.01	1.96
6	70	80	3	3.76	3.11	4.16	11.27	11.01
6	80	90	3	7.23	4.93	9.57	21.68	21.18
6	90	100	4	16.27	12.01	18.90	65.06	63.56
6	92	93	1	12.01	12.01	12.01	12.01	11.74

6	94	95	1	16.91	16.91	16.91	16.91	16.52
6	97	98	1	17.24	17.24	17.24	17.24	16.84
6	99	100	1	18.90	18.90	18.90	18.90	18.46
6	0	100	32	3.20	-	18.90	102.36	100
7	10	20	1	0.01	0.01	0.01	0.01	1.52
7	30	40	1	0.14	0.14	0.14	0.14	20.94
7	50	60	1	0.15	0.15	0.15	0.15	23.37
7	70	80	1	0.17	0.17	0.17	0.17	25.34
7	90	100	1	0.19	0.19	0.19	0.19	28.83
7	99	100	1	0.19	0.19	0.19	0.19	28.83
7	0	100	5	0.13	0.01	0.19	0.66	100
8	10	20	1	0.36	0.36	0.36	0.36	1.35
8	30	40	1	0.42	0.42	0.42	0.42	1.57
8	50	60	1	4.76	4.76	4.76	4.76	17.91
8	70	80	1	6.20	6.20	6.20	6.20	23.32
8	90	100	1	14.85	14.85	14.85	14.85	55.86
8	99	100	1	14.85	14.85	14.85	14.85	55.86
8	0	100	5	5.32	0.36	14.85	26.58	100
9	10	20	1	0.07	0.07	0.07	0.07	1.20
9	30	40	1	0.11	0.11	0.11	0.11	1.88
9	50	60	1	0.49	0.49	0.49	0.49	8.39
9	70	80	1	0.69	0.69	0.69	0.69	11.82
9	90	100	1	4.48	4.48	4.48	4.48	76.71
9	99	100	1	4.48	4.48	4.48	4.48	76.71
9	0	100	5	1.17	0.07	4.48	5.84	100

Quantile Analysis of Zinc Grades for Individual Zones								
Zone	Q%_from	Q%_to	Qty of samples	Ave	Min	Max	Accumulated metal	Accumulated metal (%)
1	0	10	86	0.11	-	0.26	9.15	0.59
1	10	20	87	0.40	0.26	0.54	34.64	2.24
1	20	30	86	0.70	0.54	0.83	59.78	3.86
1	30	40	87	0.96	0.84	1.09	83.88	5.42
1	40	50	86	1.21	1.10	1.35	104.25	6.74
1	50	60	87	1.50	1.36	1.65	130.59	8.44
1	60	70	86	1.81	1.65	1.96	155.43	10.04
1	70	80	87	2.19	1.96	2.44	190.32	12.30
1	80	90	86	2.86	2.44	3.58	246.11	15.90
1	90	100	87	6.13	3.60	13.26	533.71	34.48
1	90	91	8	3.69	3.60	3.74	29.48	1.90
1	91	92	9	4.08	3.76	4.26	36.71	2.37
1	92	93	9	4.37	4.30	4.48	39.37	2.54
1	93	94	8	4.78	4.49	4.98	38.27	2.47
1	94	95	9	5.30	5.08	5.60	47.71	3.08
1	95	96	9	5.83	5.62	6.09	52.46	3.39
1	96	97	8	6.48	6.21	7.02	51.86	3.35
1	97	98	9	7.38	7.06	7.61	66.38	4.29
1	98	99	9	8.42	7.65	9.10	75.81	4.90
1	99	100	9	10.63	9.35	13.26	95.67	6.18
1	0	100	865	1.79	-	13.26	1 547.86	100
2	0	10	37	0.12	0.03	0.20	4.43	0.57
2	10	20	38	0.27	0.20	0.33	10.18	1.32
2	20	30	38	0.41	0.35	0.46	15.56	2.02
2	30	40	37	0.54	0.47	0.61	19.89	2.58
2	40	50	38	0.71	0.62	0.81	27.16	3.52
2	50	60	38	1.05	0.82	1.26	39.93	5.17
2	60	70	37	1.51	1.31	1.80	55.91	7.24
2	70	80	38	2.31	1.83	2.89	87.83	11.38
2	80	90	38	4.09	2.95	5.35	155.51	20.14
2	90	100	38	9.36	5.37	21.18	355.72	46.07
2	90	91	3	5.53	5.37	5.71	16.58	2.15
2	91	92	4	5.86	5.76	6.00	23.44	3.04
2	92	93	4	6.43	6.13	6.66	25.71	3.33
2	93	94	4	6.93	6.69	7.08	27.73	3.59
2	94	95	4	7.25	7.20	7.31	29.00	3.76
2	95	96	3	7.60	7.35	7.74	22.79	2.95
2	96	97	4	8.25	7.75	8.82	33.01	4.27
2	97	98	4	9.55	9.19	10.54	38.20	4.95
2	98	99	4	14.87	12.99	18.40	59.47	7.70
2	99	100	4	19.95	18.79	21.18	79.80	10.33
2	0	100	377	2.05	0.03	21.18	772.12	100
3	0	10	4	0.09	0.03	0.15	0.38	0.36
3	10	20	4	0.29	0.16	0.37	1.14	1.10
3	20	30	5	0.41	0.37	0.49	2.06	1.98
3	30	40	4	0.60	0.55	0.67	2.42	2.32
3	40	50	5	0.77	0.68	0.83	3.83	3.68
3	50	60	4	0.87	0.83	0.95	3.48	3.34

3	60	70	4	1.16	0.98	1.28	4.63	4.45
3	70	80	5	1.47	1.30	1.70	7.33	7.03
3	80	90	4	2.02	1.75	2.29	8.07	7.74
3	90	100	5	14.16	2.37	18.10	70.82	68.00
3	91	92	1	2.37	2.37	2.37	2.37	2.28
3	93	94	1	16.50	16.50	16.50	16.50	15.84
3	95	96	1	16.85	16.85	16.85	16.85	16.18
3	97	98	1	17.00	17.00	17.00	17.00	16.32
3	99	100	1	18.10	18.10	18.10	18.10	17.38
3	0	100	44	2.37	0.03	18.10	104.15	100
4	0	10	7	0.04	-	0.09	0.25	0.11
4	10	20	8	0.27	0.13	0.43	2.18	0.95
4	20	30	8	0.72	0.60	0.84	5.76	2.50
4	30	40	8	0.95	0.84	1.03	7.60	3.29
4	40	50	8	1.20	1.04	1.29	9.63	4.17
4	50	60	8	1.59	1.37	1.95	12.72	5.51
4	60	70	8	2.33	1.97	2.57	18.61	8.06
4	70	80	8	3.21	2.69	3.92	25.69	11.13
4	80	90	8	6.59	4.21	8.19	52.74	22.85
4	90	100	8	11.95	8.40	17.70	95.61	41.43
4	91	92	1	8.40	8.40	8.40	8.40	3.64
4	92	93	1	8.44	8.44	8.44	8.44	3.66
4	93	94	1	9.27	9.27	9.27	9.27	4.02
4	94	95	1	11.01	11.01	11.01	11.01	4.77
4	96	97	1	11.14	11.14	11.14	11.14	4.83
4	97	98	1	12.78	12.78	12.78	12.78	5.54
4	98	99	1	16.88	16.88	16.88	16.88	7.31
4	99	100	1	17.70	17.70	17.70	17.70	7.67
4	0	100	79	2.92	-	17.70	230.79	100
5	10	20	1	0.42	0.42	0.42	0.42	3.62
5	20	30	1	0.78	0.78	0.78	0.78	6.78
5	30	40	1	1.01	1.01	1.01	1.01	8.79
5	40	50	1	1.01	1.01	1.01	1.01	8.79
5	50	60	1	1.04	1.04	1.04	1.04	9.05
5	60	70	1	1.16	1.16	1.16	1.16	10.09
5	70	80	1	1.80	1.80	1.80	1.80	15.66
5	80	90	1	1.80	1.80	1.80	1.80	15.66
5	90	100	1	2.48	2.48	2.48	2.48	21.57
5	99	100	1	2.48	2.48	2.48	2.48	21.57
5	0	100	9	1.28	0.42	2.48	11.50	100
6	0	10	3	-	-	-	-	-
6	10	20	3	0.37	-	0.72	1.12	2.15
6	20	30	3	1.02	0.72	1.17	3.05	5.85
6	30	40	3	1.24	1.22	1.27	3.73	7.16
6	40	50	4	1.37	1.28	1.43	5.47	10.50
6	50	60	3	1.55	1.48	1.59	4.64	8.90
6	60	70	3	1.68	1.59	1.81	5.04	9.68
6	70	80	3	1.87	1.87	1.87	5.61	10.77
6	80	90	3	2.92	2.49	3.43	8.76	16.81
6	90	100	4	3.67	3.44	3.89	14.69	28.19
6	92	93	1	3.44	3.44	3.44	3.44	6.60

6	94	95	1	3.66	3.66	3.66	3.66	7.02
6	97	98	1	3.70	3.70	3.70	3.70	7.10
6	99	100	1	3.89	3.89	3.89	3.89	7.46
6	0	100	32	1.63	-	3.89	52.12	100
7	10	20	1	0.01	0.01	0.01	0.01	0.58
7	30	40	1	0.27	0.27	0.27	0.27	19.32
7	50	60	1	0.30	0.30	0.30	0.30	21.98
7	70	80	1	0.33	0.33	0.33	0.33	24.00
7	90	100	1	0.47	0.47	0.47	0.47	34.12
7	99	100	1	0.47	0.47	0.47	0.47	34.12
7	0	100	5	0.28	0.01	0.47	1.38	100
8	10	20	1	0.38	0.38	0.38	0.38	2.59
8	30	40	1	1.54	1.54	1.54	1.54	10.46
8	50	60	1	4.03	4.03	4.03	4.03	27.47
8	70	80	1	4.20	4.20	4.20	4.20	28.60
8	90	100	1	4.53	4.53	4.53	4.53	30.87
8	99	100	1	4.53	4.53	4.53	4.53	30.87
8	0	100	5	2.93	0.38	4.53	14.67	100
9	10	20	1	0.19	0.19	0.19	0.19	3.97
9	30	40	1	0.19	0.19	0.19	0.19	3.97
9	50	60	1	0.55	0.55	0.55	0.55	11.50
9	70	80	1	0.71	0.71	0.71	0.71	14.85
9	90	100	1	3.14	3.14	3.14	3.14	65.70
9	99	100	1	3.14	3.14	3.14	3.14	65.70
9	0	100	5	0.96	0.19	3.14	4.78	100

APPENDIX 2: VERTIKALNY – JORC TABLE 1

Section 1 Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	**Exploration Campaign 2005-2018** • Sampling was carried out using a combination of diamond core drillholes and surface trench channel samples. • Diamond drilling was used to obtain predominantly 1.0m samples (minimum length 0.25m to a maximum of 3.00m) that were subsequently cut in half along its length to produce half core for sample preparation (crushing/pulverising) to produce a final sub-sample for laboratory analysis. • Trenching was used to obtain predominately 1.0m samples (minimum length 0.10m to a maximum of 2.00m). The entire sample was taken for sample preparation (crushing/pulverising) to produce a final sub-sample for laboratory analysis.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• Drilling at Vertikalny consists of diamond core drilling only. • In the majority of drillholes, the core was oriented at the commencement of every run to allow structural measurements to be made and all holes are subject to down-hole survey at generally 20.0m intervals. • Data from HQ (63.5mm) and NQ (47.6mm) wireline

Criteria	JORC Code explanation	Commentary
		diamond drillholes is used for interpretation and grade estimation. The predominate drilling diameter was of HQ size. • The main drill campaigns at Vertikalny have taken place in 2005-2015 with no drilling in 2010. • Metallurgical holes were drilled in 2017 • Grade control drilling was carried out in 2018. • A total of 304 diamond holes have been drilled for 44,060m.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• WAI is not aware of any specific measures taken to reduce losses through drilling or that any drilling campaign suffered from poor recovery. • Diamond drill recovery averages approximately 95%. • Due to good drilling practices followed at Vertikalny samples are considered homogenous and representative. • No apparent relationship is observed between sample recovery and grade.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• Core was logged on site by company geological personnel using a standardised logging convention, to a level sufficient to support geological interpretation, modelling, and subsequent mineral resource estimation. • Core was geologically logged including a description of lithology, alteration/weathering, major structures, mineralisation, and veining on a qualitative basis. • Core was logged manually before transfer to an electronic system using Excel spreadsheets. • Rock Quality Designation (RQD) measurements were also completed by the field geologists.

Criteria	JORC Code explanation	Commentary
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representativity of samples.* • *Measures taken to ensure that the sampling is representative of the in-situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• Sample preparation has followed standard industry practices: – Diamond drill core was cut lengthways along its long axis with half core used for primary analysis and the other half retained for reference purposes. – Trench channel samples was cut by portable diamond saw and collected using hammer and chisel. • Sample preparation for Vertikalny was carried out on site. The sample preparation flowsheet comprised: – Two stage crushing to 85% passing 1mm; – Split to 1kg sample; – Submit for futher analysis. • Prior 2011 final milling and pulverising to 85% passing 75µm was carried out in Chemical Laboratory of State Enterprise Aldangeologia in Aldan (Russia) and later in ALS Chemex in Chita, Russia. • Sub-sampling quality control has been maintained through use of company SOP's being adopted to ensure consistency by following a standard set of practices throughout the process. • The use of field duplicate sample (1/4 of core or parallel channel sample next to original trench sample) analysis has been used throughout the drill campaign at Vertikalny in order to monitor precision and reproducibility.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF*	• No geophysical or portable analysis tools were used to determine assay values stored in the final exploration database used for mineral resource estimation. • For the diamond drillhole and trench channel samples,

Criteria	JORC Code explanation	Commentary
	instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	QA/QC results (from duplicate and standard samples) were in line with expectations for precision and accuracy. Certified reference material (CRM) samples were obtained from Geostats Pty Ltd (Australia), ORE Research & ExplorationPty Ltd (Australia), OJSC Irgiredmet (Russia) and LLC “NTC Minstandart” (Russia). • Local non-mineralised rock used for blank samples. Approximately 10% of blank samples were found to be out of range. Approximately 1.5% of blank samples had significant grade, i.e. >50g/t Ag. • Prior 2011 samples sent for spectral assay for 36 elements. Samples with significant Ag grade determined by spectral assay were analysed for Ag, Cu, Pb and Zn using atomic absorption. In addition, all analysis was conducted for Ag using fire assay. • From 2011 onwards, analyses were completed using a four acid sample digestion of 0.25g, followed by ICP finish and reporting of 33 elements (laboratory code ME-ICP62). Where values of silver, lead and zinc exceed upper detection limits further four acid digestion analyses were carried out of 0.4g followed by ICP finish (lab code ME-OG62). Where values of silver exceeded the upper detection limit (1,500g/t), a 50g sample was taken for FA analysis with gravimetric finish (lab code Ag-GRA22). • The assays of Certified Reference Material, which cover a range of metal values for each of Ag, as well as field duplicate assays show no significant bias. • No systematic bias appears to be present in results. • The quality control and assurance data reviewed by the CP indicates the assays are generally within expected

Criteria	JORC Code explanation	Commentary
		limits. The CP is satisfied the quality assurance and control data is sufficient to support the Mineral Resource classification presented herein.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• All work has been supervised by senior technical staff. • No site visit was conducted by WAI Competent Person and no verification of the data was done. That includes review of collar locations in the field, review of core logging, review data from primary assay sheets. • Significant intersections have not been verified by either independent or alternate company personnel. • Logging data in the first instance was recorded by hand to form documentation for each hole that includes collar and down hole survey information and assay information once available. This information was subsequently transferred to an electronic database. • WAI completed a number of checks on the raw data and data entry process. Based on the verification work completed, WAI is confident that the compiled database is an accurate reflection of the available drilling data. • No adjustments to assay data have been made.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• All data was supplied in the World Geodetic System 1984, Zone 36J Northern Hemisphere (UTM). • Collar positions for all holes were laid out by the on-site surveyor using a differential GPS and then checked again once drilling was completed. • Downhole surveys were carried out for all of the diamond drillholes using Reflex Ez-Shot equipment. The measurement was taken every 20m in general. • A topographic survey was conducted in 2014. The survey

Criteria	JORC Code explanation	Commentary
		was carried out using Topcon 5GR satellite receiver. The field data was processed using TOPCONTOOLS software package. This survey is used for the current Mineral Resource Estimate. • The small differences between the GPS readings and the topographical survey data do not influence the interpreted mineralisation widths.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• Data spacing is down to 40m x 40m in the central part of deposit with some area of infill drilling to 25m x 25m. On the flanks the data spacing is more generally between 80m x 80m. The grade control trenches is developed every 10m on the each 5m bench. This spacing is sufficient to establish geological and mineralisation continuity appropriate for the reporting of Mineral Resources. • Mineral Resources are classified as Measured, Indicated and Inferred in accordance with the guidelines of the JORC Code (2012), and through geostatistical analysis considering the spatial distribution of sample data. • Sample compositing was carried out as part of the mineral resource estimation process. • The diamond drill and trench data spacing is deemed by the CP to be sufficient to imply/confirm geological and grade continuity, sufficient for the classification of Inferred resources only. • The average length of the samples is 0.91m therefore the composite length of 1.0m was chosen.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit*	• In general, drilling is carried out so that the intersections of holes with mineralised zones occurs at a high angle which results in limited sample bias.

Criteria	JORC Code explanation	Commentary
	type. • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• The general strike of mineralisation is to north-west at 310° with sub-vertical steeply dipping mineralisation zone hence drilling is generally inclined at –50-60° towards the strike of the zones. • Intercepts are reported as apparent thicknesses except where otherwise stated.
Sample security	• *The measures taken to ensure sample security.*	• Samples were transported to site sample preparation facilities. After initial crushing and splitting approximately 1kg material was prepared for further assay. • Crushed samples were transported regularly (typically monthly during the drilling campaigns) by commercial carrier to ALS lab in Chita in sealed bags. • After preparation in the field, samples were packed into bags and dispatched to the freight forwarders directly by the Company. All bags were transported by the Company directly to the sample preparation/assay laboratory. The assay laboratory audits the samples on arrival and reports any discrepancies back to the Company. • Sample security was managed by the Company. The CP was not able to inspect the sample dispatches and relies on the Company's representative to ensure that no discrepancies occurred, and the chain of custody is acceptable.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• No site visit has been conducted by CP due to international, regional and operational travel restrictions imposed as a result of Covid-19 pandemic, no review of sampling techniques and data.

Section 2 Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.*	• The Vertikalny license is located in the north of Kobyakskiy district in the central of Republic Sakha (Yakutia), Russia, some 400km to the north of Yakutsk city, the Republic capital, and centred on coordinates 65°40'N, 130°07'E. • CSJC Prognoz is in possession of a mining licence with the reference YaKU 03626 BE. The license has an expiry date of 01.09.2033 and covers an area of 13.55 km^2. • WAI is not aware of any known impediments to obtaining and maintaining a licence to operate the Vertikalny Property. • The CP has relied on the information provided by Silver Bear that the tenement is in good standing and all fees are paid.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• The first mention of the presence of silver-base metal mineralisation is related to 1764. Following that up until 1930s individuals were carried out prospecting and small-scale mining in the area. • Sporadic exploration was carried out during 1930s and 1940s. • Different scale geological mapping and soil-geochemistry sampling as well as different ground and airborne geophysical survey methods was carried out in 1950s to 1970s.More detailed prospecting works had been carried out on the areas with detected metal anomalies. • Form 1991 to 2003 JSC Yanageologia completed 151,452m3 of trenching and 1,303m of drilling focusing on the 15 principal veins systems.

Criteria	JORC Code explanation	Commentary
		• Prospecting/exploration activities include surface trenching, a restricted amount of drilling and underground developments (shallow shafts and adits with crosscuts). • CJSC Prognoz has carried out exploration at Vertikalny since 2004 up to present.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Vertikalny Property is part of Endybal area which occurs in the north-eastern wing of Kuranakh anticlinorium and being a part of Zapadno-Verkhoyanskiy mega-anticlinorium. The Endybal area is composited by terrigenious sediments of Carboniferous-Triassic age. The sediments intruded by Late Jurassic, Early and Late Cretaceous magmatic rock. • The mineralisation is associated with crestal plane of Endybal anticline. South-north striking Newktominskiy fault and transverse Severo-Tirekhtyaxskiy deep fault are associated with crestal of Endybal anticline. • Mineralisation of Vertikalny is related to the feather structures of this faults having north-west strike with steep dipping to north-east. • Vertikalny is a vein type deposit representing combination of conjugated faults and brecciated sections and associated mineralisation. • Mineralised zones are grouped into three domains – Central, North-East and North-West areas. • Mineralisation is being a epigenetic polymetallic silver-lead-zinc veins hosted by metasediment.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following*	• Exploration data held in the database and used in the mineral resource estimate can be summarised as

Criteria	JORC Code explanation	Commentary
	information for all Material drill holes: o *easting and northing of the drill hole collar* o *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* o *dip and azimuth of the hole* o *down hole length and interception depth* o *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	follows: – Number of drillholes – 304; – Number of exploration trenches – 74; – Number of grade control trenches – 210; – East collar ranges – 548,350m to 552,450m – North collar ranges – 7,286,050m to 7,282,820m – Collar elevation ranges – 529.3m to 1,247.6m – Azimuth ranges – 0° to 360° – Dip ranges –90° to +90° – Length of holes/trenches – 2.54m to 496m • Both diamond drillhole and trench information and assay results were used in the Mineral Resource Estimation.
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high-grade results and longer lengths of low-grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• Top cutting was used during the mineral resource estimation process to reduce the potential for outlier grades to have an overbearing effect on estimated block grades. Top cutting is based on decile analysis and log probability graphs for all zones and applied to Ag, Pb and Zn (detailed in the main body of the text). • No metal equivalent equations were used during the mineral resource estimation procedure or reporting. • Samples were composited to 1m lengths during the mineral resource estimation procedure to ensure a consistent level of support during the estimation process.
Relationship between mineralisation	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.*	• The nature of the main zones of mineralisation at Vertikalny is well recognised as being steeply dipping narrow vein structures. • In general, drilling is carried out so that the intersections

Criteria	JORC Code explanation	Commentary
widths and intercept lengths	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	of holes with mineralised zones occurs at a high angle to minimise sample bias. • Down hole length reflects drilled meters not the true width of the mineralised structures.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• Appropriate data tabulations, plans and sections showing the nature of the mineralisation, exploration and final mineral resource estimate are included in the main body of the report.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Individual exploration results are not being reported. This section is not considered relevant to the overall reporting of the mineral resource estimate. • A total of 304 diamond drillholes and 284 trenches (including grade control trenches) have been completed on the Vertikalny and used for the current mineral resource estimate.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• Metallurgical testwork was used to define recovery factors during pit optimisation used as a basis for limiting potential Mineral Resources based on the expectation of economic extraction. • Geotechnical data of Vertikalny deposit was used during pit optimisation used as a basis for limiting potential Mineral Resources based on the expectation of economic extraction. • Density measurement was done for both oxide and primary mineralisation as following: – 144 samples in 2004-2012; – 88 samples in 2012; – 53 samples in 2015.

Criteria	JORC Code explanation	Commentary
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• No planned exploration drilling is currently known about. • Mineralisation is closed along strike to north-west and south-east. • Mineralisation is not closed at depth.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	• The project database is held in MS Access and Excel format files. Data held includes; collar location, downhole surveys, assay information, lithological logging and oxidation logging. Also held in Microsoft Excel spreadsheets is information on duplicate samples certified reference materials and blanks. • Access to the Vertikalny drilling/trenching database used for resource estimation is restricted to geological and selected technical staff. • WAI completed a number of checks on the raw data supplied by CJSC Prognoz and is satisfied that the data does not contain significant errors nor has it been corrupted. • Validation of the database was carried out during import of the data in to Datamine Studio 3 for production of the mineral resource estimate, no major issues were found with duplicate or overlapping samples.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate*	• No site visit has been conducted by CP due to international, regional and operational travel restrictions imposed as a result of Covid-19 pandemic.

Criteria	JORC Code explanation	Commentary
	why this is the case.	
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• Grade estimation for Vertikalny uses diamond and trench sampling only. • The confidence in the geological interpretation is deemed good. Exploration drilling has been carried out on a grid down to 40m x 40m, with wider spacing on the flanks - between 80m and 100m, and geological logging is comprehensive. • Geological logging has been carried out from drill core samples and in trenches. • Geological logging was used to define mineralised domains within the overall resource model. • The wireframes used to constrain the block model and grade interpolation were constructed based on Prognoz's understanding of the geology and mineralisation of the Vertikalny deposit. • The resource model reflects the interpretation north-west orientated vein system (zones) reflecting areas of elevated mineralisation.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The mineralisation is split on three domains which have north-west. The overall mineralisation dimension is ~3.5km in north-west direction and ~50-80m across strike. • The current mineral resource is constrained by series of optimised open pit with a total strike length of 3.5km, a maximum width of ~200m at the crest, and a maximum depth of pit = 130m. • The unconstrained block model has a maximum depth of mineralisation up to 400m from the surface.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and*	• Three domains were created to represent each of the mineralised structures (zones). • DTM surfaces were created to represent the pre-mining topographical surface, pit contours as on 31st of May 2019,

Criteria	JORC Code explanation	Commentary
	maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	overburden material and base of oxide/primary material. • A block model was created using the geological and mineralised zone wireframes as boundaries. A parent block size of 10m (X) x 10m (Y) x 10m (Z) was used in the block model with key fields established for geological and mineralised domains. Additional key fields were established to denote oxide/fresh rock domains, mined out material and overburden rock. • Grade capping: Grade capping was carried out to stop local overestimation of grade from high-grade outlier samples. Grade capping was used for all variables on a zone-by-zone basis where outlier grades were identified using a combination of decile analysis and a review of log-probability plots. • Composites: A 1m composite length was chosen to ensure consistent sample support during estimation. Composites were limited to the boundaries of mineralised domains. • Variography: A variographic study by domain identified reasonably robust variogram models for Ag across two mineralised zones. • Estimation: Estimation was carried out using Ordinary Kriging as the primary method. An Inverse distance (squared) estimate was carried out for validation purposes. Only composite samples within an individual zone were used for estimation of that zone. Estimation parameters were based on models of grade continuity produced during geostatistical analysis. Dynamic anisotropy was used to change orientations of search ellipses based on local variations of dip and strike. Minimum and maximum sample criteria, an octant search restriction, and restrictions of number of composite samples from a single drillhole were employed during grade estimation to assist with declustering and to reduce local grade bias. A multiple pass estimation as carried out with expanding search ellipses and less restrictive estimation parameters for estimating blocks in more

Criteria	JORC Code explanation	Commentary
		poorly sampled areas. • Estimation was carried out in to parent cells only to reduce risk of conditional bias. Estimation was carried out using a discretisation of five points in each dimension. • The block model was verified first by comparing drillhole composite sample values with estimated block values on a sectional and plan basis. Grade comparison was also carried out statistically by zone to ensure the global grade estimate was unbiased. Grade profile (swath) plots were also constructed to compare modelled grades and input composite grades in slices or varying width. During this process a comparison was made between declustered and clustered data to identify any possible local bias introduced by irregular grade spacing. • No estimation of deleterious components was carried out. • The estimated block model was validated by visual inspection of block grades in comparison with drillhole data, and comparison of the block model statistics.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• All tonnages are reported as dry tonnages. • Moisture content has been measured using weighing waxed samples and dried ones.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• Mineralised zones are defined at a natural cut-off grade of 50g/t Ag. • The mineral resource estimate is restricted to material falling within an NPV Scheduler optimised pit shell as described below in “Mining factors or assumptions”, and above a cut-off grade representing breakeven cut-off grade derived from open pit optimisation parameters for each zone (Oxide and Fresh).
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining*	• The deposit is an operating open pit mine. Part of the deposit below pit is deemed to be mined by underground. • Reporting of mineral resources suitable for open pit extraction were

Criteria	JORC Code explanation	Commentary
	dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	limited by the creation of an optimised open pit shell in NPV Scheduler. The optimisation was carried out using Net Smelter Return data. The approach to NSR estimation is presented in the main text body. The pit shell was created with the following major parameters: – NSR (oxide) – US$/t 172.78; – NSR (primary) – US$/t – 139.06; – Mining cost (mineralisation/waste) of US$2.53/t; – Oxide processing cost of US$72.91/t; – Primary processing cost US$46.97/t – Processing recovery – 95%; – G&A cost of US$60.0/t – Slope angle - 56° at hanging wall, 48°at foot wall; – Mining dilution of 30% and mining losses of 0%. Reporting of mineral resources for underground mining is based on the following parameters: – NSR (primary only) – US$162.00/t; – Mining cost – US$55/t; – Processing cost – US$46.97/t; – G&A – US$60.00/t – Processing recovery – 95%.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral*	• Metallurgical recovery was utilised during the construction of an optimised pit shell used for limiting mineral resources based on an expectation of eventual economic extraction.

Criteria	JORC Code explanation	Commentary
	Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	• WAI is unaware of any environmental factors which would preclude the reporting of Mineral Resources.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates*	• Density measurements have been taken for oxide and primary material with respect to natural moisture. • A total of 285 density measurements have been taken for oxide and primary material. • Measurements were made using the Archimedes water immersion method, the results were recorded and imported into Excel spreadsheet. • Density was assigned to the block model during the Mineral Resource estimation by applying the 3.13 t/m^3 value for oxide material 3.56t/m^3 for primary material and 2.75 for waste.

Criteria	JORC Code explanation	Commentary
	used in the evaluation process of the different materials.	• Moisture content was measured for oxide and primary material. • The tonnage is reported on a dry basis.
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• Mineral Resource classification was carried out in accordance with the guidelines of the JORC Code (2012) to Measured, Indicated and Inferred. • The Vertikalny Silver Project is an operating mine. Classification is based on sample density, confidence in geological continuity and mineralisation continuity, and reliability of the exploration database used as basis of mineral resource estimation: – Measured classification was assigned to the areas drillhole spacing was 40m x 40m and lower; – Indicated classification was assigned to the areas where drillhole spacing was 80m x 80m or below; – Inferred classification was assigned to the areas where drillhole spacing was greater than 80m x 80m or if the mineralisation continuity was not established. • The mineral resource estimate classification reflects the Competent Person's view of the Vertikalny Project. • Mineral Resources for open pit mining were limited using an optimised pit shell using parameters as laid out in the main section of the report and as described in "Mining factors and assumptions" above. • Mineral Resources for underground operation was defined below open pit shell. • The mineral resource estimate has been limited to the surveyed pit surface as detailed in the main report.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• WAI is not aware of any audits or reviews of this Mineral Resource Estimate other than internal peer review.

Criteria	JORC Code explanation	Commentary
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• The relative accuracy and confidence in the mineral resource estimate is reflected in the reporting of the mineral resource as set out in the JORC Code (2012) • The statement relates to global estimates of tonnes and grade. • The classification applied to the mineral resource estimate is based upon; confidence of continuity of mineralisation, quality of data (QA/QC) and validation of the block model.

APPENDIX 3: MANGAZEISKY NORTH – JORC TABLE 1

Section 1 Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	**Exploration Campaign 2013-2015** • Sampling was carried out using a combination of diamond core drillholes and surface trench channel samples. • Diamond drilling was used to obtain predominantly 1.0m samples (minimum length 0.25m to a maximum of 3.00m) that were subsequently cut in half along its length to produce half core for sample preparation (crushing/pulverising) to produce a final sub-sample for laboratory analysis. • Trenching was used to obtain predominately 1.0m samples (minimum length 0.10m to a maximum of 2.00m). The entire sample was taken for sample preparation (crushing/pulverising) to produce a final sub-sample for laboratory analysis.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• Drilling at North Mangazeisky (NM) consists of diamond core drilling only. • In the majority of drillholes, the core was oriented at the commencement of every run to allow structural measurements to be made and all holes are subject to down-hole survey at generally 20.0m intervals. • Data from HQ (63.5mm) and NQ (47.6mm) wireline

Criteria	JORC Code explanation	Commentary
		diamond drillholes is used for interpretation and grade estimation. The predominate drilling diameter was of HQ size. • The main drill campaigns at NM have taken place in 2014-2016 including 29 holes to collect material for metallurgical testwork (2016). • A total of 160 diamond holes have been drilled for 7,214m.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• WAI is not aware of any specific measures taken to reduce losses through drilling or that any drilling campaign suffered from poor recovery. • Diamond drill recovery averages approximately 95%. • Due to good drilling practices followed at NM samples are considered homogenous and representative. • No apparent relationship is observed between sample recovery and grade.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• Core was logged on site by company geological personnel using a standardised logging convention, to a level sufficient to support geological interpretation, modelling, and subsequent mineral resource estimation. • Core was geologically logged including a description of lithology, alteration/weathering, major structures, mineralisation, and veining on a qualitative basis. • Core was logged manually before transfer to an electronic system using Excel spreadsheets. • Rock Quality Designation (RQD) measurements were also completed by the field geologists.

Criteria	JORC Code explanation	Commentary
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• Sample preparation has followed standard industry practices: – Diamond drill core was cut lengthways along its long axis with half core used for primary analysis and the other half retained for reference purposes. – Trench channel samples was cut by portable diamond saw and collected using hammer and chisel. • Sample preparation for Vertikalny was carried out on site. The sample preparation flowsheet comprised: – Two stage crushing to 85% passing 1mm; – Split to 1kg sample; – Submit for futher analysis. • Final milling and pulverising to 85% passing 75µm was carried out in ALS Chemex in Chita, Russia. • Sub-sampling quality control has been maintained through use of company SOP's being adopted to ensure consistency by following a standard set of practices throughout the process. • The use of field duplicate sample (1/4 of core or parallel channel sample next to original trench sample) analysis has been used throughout the drill campaign at NM in order to monitor precision and reproducibility.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model,*	• No geophysical or portable analysis tools were used to determine assay values stored in the final exploration database used for mineral resource estimation. • For the diamond drillhole and trench channel samples, QA/QC results (from duplicate and standard samples) were in line with expectations for precision and

Criteria	JORC Code explanation	Commentary
	reading times, calibrations factors applied and their derivation, etc. • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	accuracy. Certified reference material (CRM) samples were obtained from Geostats Pty Ltd (Australia), OJSC Irgiredmet (Russia) and LLC "NTC Minstandart" (Russia). • Local non-mineralised rock used for blank samples. Approximately 12% of blank samples were found to be out of range. Approximately 5% of blank samples had significant grade, i.e., >50g/t Ag. • Analyses were completed using a four-acid sample digestion of 0.25g, followed by ICP finish and reporting of 33 elements (laboratory code ME-ICP62). Where values of silver, lead and zinc exceed upper detection limits further four acid digestion analyses were carried out of 0.4g followed by ICP finish (lab code ME-OG62). Where values of silver exceeded the upper detection limit (1,500g/t), a 50g sample was taken for FA analysis with gravimetric finish (lab code Ag-GRA22). • The assays of Certified Reference Material, which cover a range of metal values for each of Ag, as well as field duplicate assays show no significant bias. • No systematic bias appears to be present in results. • The quality control and assurance data reviewed by the CP indicates the assays are generally within expected limits. The CP is satisfied the quality assurance and control data is sufficient to support the Mineral Resource classification presented herein.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry*	• All work has been supervised by senior technical staff. • No site visit was conducted by WAI Competent Person and no verification of the data was done. That includes review of collar locations in the field, review of core

Criteria	JORC Code explanation	Commentary
	procedures, data verification, data storage (physical and electronic) protocols. • *Discuss any adjustment to assay data.*	logging, review data from primary assay sheets. • Significant intersections have not been verified by either independent or alternate company personnel. • Logging data in the first instance was recorded by hand to form documentation for each hole that includes collar and down hole survey information and assay information once available. This information was subsequently transferred to an electronic database. • WAI completed a number of checks on the raw data and data entry process. Based on the verification work completed, WAI is confident that the compiled database is an accurate reflection of the available drilling data. • No adjustments to assay data have been made.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• All data was supplied in the World Geodetic System 1984, Zone 52 Northern Hemisphere (UTM). • Collar positions for all holes were laid out by the on-site surveyor using a differential GPS and then checked again once drilling was completed. • Downhole surveys were carried out for all of the diamond drillholes using Reflex Ez-Shot equipment. The measurement was taken every 20m in general. • A topographic survey was conducted in 2014. The survey was carried out using Topcon 5GR satellite receiver. The field data was processed using TOPCONTOOLS software package. This survey is used for the current Mineral Resource Estimate. • The small differences between the GPS readings and the topographical survey data do not influence the interpreted mineralisation widths.

Criteria	JORC Code explanation	Commentary
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• Data spacing is down to 25m x 25m in the central part of deposit. On the flanks the data spacing is more generally between 50m x 50m. This spacing is sufficient to establish geological and mineralisation continuity appropriate for the reporting of Mineral Resources. • Mineral Resources are classified as Inferred in accordance with the guidelines of the JORC Code (2012), and through geostatistical analysis considering the spatial distribution of sample data. • Sample compositing was carried out as part of the mineral resource estimation process. • The diamond drill and trench data spacing is deemed by the CP to be sufficient to imply/confirm geological and grade continuity, sufficient for the classification of Inferred resources only. • The average length of the samples is 0.85m therefore the composite length of 1.0m was chosen.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• In general, drilling is carried out so that the intersections of holes with mineralised zones occurs at a high angle which results in limited sample bias. • The general strike of mineralisation is to north-west at 330° with shallow dipping at 30-35° to north-east mineralisation hence drilling is generally inclined at –50-60° towards the strike of the zones. • Intercepts are reported as apparent thicknesses except where otherwise stated.
Sample security	• *The measures taken to ensure sample security.*	• Samples were transported to site sample preparation facilities. After initial crushing and splitting approximately 1kg material was prepared for further

Criteria	JORC Code explanation	Commentary
		assay. • Crushed samples were transported regularly (typically monthly during the drilling campaigns) by commercial carrier to ALS lab in Chita in sealed bags. • After preparation in the field, samples were packed into bags and dispatched to the freight forwarders directly by the Company. All bags were transported by the Company directly to the sample preparation/assay laboratory. The assay laboratory audits the samples on arrival and reports any discrepancies back to the Company. • Sample security was managed by the Company. The CP was not able to inspect the sample dispatches and relies on the Company's representative to ensure that no discrepancies occurred, and the chain of custody is acceptable.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• No site visit was carried out by CP, no review of sampling techniques and data

Section 2 Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting*	• The NM license is located in the north of Kobyakskiy district in the central of Republic Sakha (Yakutia), Russia, some 400km to the north of Yakutsk city, the Republic capital, and centred on coordinates 65°40'N, 130°07'E. • CSJC Prognoz is in possession of a exploration licence with the reference YaKU 12692 BP. The license has an expiry date of 31.12.2023 and covers an area of 570 km^2.

Criteria	JORC Code explanation	Commentary
	along with any known impediments to obtaining a license to operate in the area.	• WAI is not aware of any known impediments to obtaining and maintaining a licence to operate the NM Property. • The CP has relied on the information provided by Silver Bear that the tenement is in good standing and all fees are paid.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• The first mention of the presence of silver-base metal mineralisation is related to 1764. Following that up until 1930s individuals were carried out prospecting and small-scale mining in the area. • Sporadic exploration was carried out during 1930s and 1940s. • Different scale geological mapping and soil-geochemistry sampling as well as different ground and airborne geophysical survey methods was carried out in 1950s to 1970s.More detailed prospecting works had been carried out on the areas with detected metal anomalies. • Form 1991 to 2003 JSC Yanageologia completed 151,452m3 of trenching and 1,303m of drilling focusing on the 15 principal veins systems. • Prospecting/exploration activities include surface trenching, a restricted amount of drilling and underground developments (shallow shafts and adits with crosscuts). • CJSC Prognoz has carried out exploration at NM since 2013 up to present.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The NM Property is part of Endybal area which occurs in the north-eastern wing of Kuranakh anticlinorium and being a part of Zapadno-Verkhoyanskiy mega-

Criteria	JORC Code explanation	Commentary
		anticlinorium. The Endybal area is composited by terrigenious sediments of Carboniferous-Triassic age. The sediments intruded by Late Jurassic, Early and Late Cretaceous magmatic rock. • Mineralisation occurs within Mangazeiskiy sincline which is part of the eastern wing of Endubal anticline. The dip of the rocks of the Endybal anticline in the area of NM averages 20 to 45°. • Mineralisation of NM forms strata-bound veins within sandstone thickness. • Mineralised zones are grouped into two domains – Central and South areas. • Mineralisation is being a epigenetic polymetallic silver-lead-zinc veins hosted by metasediment.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* o *easting and northing of the drill hole collar* o *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* o *dip and azimuth of the hole* o *down hole length and interception depth* o *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	• Exploration data held in the database and used in the mineral resource estimate can be summarised as follows: – Number of drillholes – 157; – Number of exploration trenches – 50; – East collar ranges – 551,960m to 552,700m. – North collar ranges – 7,289,680m to 7,291,290m – Collar elevation ranges – 1,052.9m to 1,201.5m – Azimuth ranges – 0° to 300° – Dip ranges –37° to +90° – Length of holes/trenches – 2.0m to 122.0m • Both diamond drillhole and trench information and assay results were used in the Mineral Resource Estimation.

Criteria	JORC Code explanation	Commentary
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• Top cutting was used during the mineral resource estimation process to reduce the potential for outlier grades to have an overbearing effect on estimated block grades. Top cutting is based on decile analysis and log probability graphs for all zones and applied to Ag, Pb and Zn (detailed in the main body of the text). • No metal equivalent equations were used during the mineral resource estimation procedure or reporting. • Samples were composited to 1m lengths during the mineral resource estimation procedure to ensure a consistent level of support during the estimation process.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• The nature of the main zones of mineralisation at NM is well recognised as being gently dipping narrow strata-bound vein structures. • In general, drilling is carried out so that the intersections of holes with mineralised zones occurs at a high angle to minimise sample bias. • Down hole length reflects drilled meters not the true width of the mineralised structures.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• Appropriate data tabulations, plans and sections showing the nature of the mineralisation, exploration and final mineral resource estimate are included in the main body of the report.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration*	• Individual exploration results are not being reported. This section is not considered relevant to the overall reporting of the mineral resource estimate. • A total of 157 diamond drillholes and 50 trenches have

Criteria	JORC Code explanation	Commentary
	Results.	been completed on the NM and used for the current mineral resource estimate.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• Metallurgical testwork was used to define recovery factors during pit optimisation used as a basis for limiting potential Mineral Resources based on the expectation of economic extraction. • Geotechnical data of Vertikalny deposit was used during pit optimisation at NM as a basis for limiting potential Mineral Resources based on the expectation of economic extraction. • Density measurement was done for both mineralisation and waste for total 68 samples (40 samples for mineralisation and 28 for waste). • No oxide/primary boundary was defined at NM, the entire mineralisation is considered to be primary.
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• No planned exploration drilling is currently known about. • Mineralisation of Central domain is closed along strike at north-west and south-east. • Mineralisation of South domain not closed to the south-east. • Mineralisation is not closed at depth.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for*	• The project database is held in MS Access and Excel format files. Data held includes collar location, downhole surveys, assay information, lithological logging and

Criteria	JORC Code explanation	Commentary
	Mineral Resource estimation purposes. • *Data validation procedures used.*	oxidation logging. Also held in Microsoft Excel spreadsheets is information on duplicate samples certified reference materials and blanks. • Access to the NM drilling/trenching database used for resource estimation is restricted to geological and selected technical staff. • WAI completed a number of checks on the raw data supplied by CJSC Prognoz and is satisfied that the data does not contain significant errors, nor has it been corrupted. • Validation of the database was carried out during import of the data in to Datamine Studio 3 for production of the mineral resource estimate, no major issues were found with duplicate or overlapping samples.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• No site visit was conducted by CP.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• Grade estimation for NM uses diamond and trench sampling only. • The confidence in the geological interpretation is deemed good. Exploration drilling has been carried out on a grid down to 25m x 25m, with wider spacing on the flanks - between 50m and 50m, and geological logging is comprehensive. • There is no data for definition of oxide/primary boundary therefor the entire mineralisation is considered as primary. • Geological logging has been carried out from drill core

Criteria	JORC Code explanation	Commentary
		samples and in trenches. • Geological logging was used to define mineralised domains within the overall resource model. • The wireframes used to constrain the block model and grade interpolation were constructed based on Prognoz's understanding of the geology and mineralisation of the NM deposit. • The resource model reflects the interpretation north-west orientated vein system (zones) reflecting areas of elevated mineralisation.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The mineralisation is split on two domains which have north-west strike. The overall mineralisation dimension is 1,095m in north-west direction and up to 10m across strike. The depth of mineralisation is 130m from the surface. • The current mineral resource is constrained by two optimised open pit with a total strike length of 1,1km, a maximum width of 250m at the crest, and a maximum depth of pit 120m (measured from south-west highwall to pit bottom). • The unconstrained block model has a maximum depth of mineralisation up to 130m from the surface.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*	• Two domains were created to represent each of the mineralised structures (zones). • DTM surfaces were created to represent the pre-mining topographical surface. • A block model was created using the geological and mineralised zone wireframes as boundaries. A parent block size of 10m (X) x 10m (Y) x 10m (Z) was used in the

Criteria	JORC Code explanation	Commentary
	• *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	block model with key fields established for geological and mineralised domains. • Grade capping: Grade capping was carried out to stop local overestimation of grade from high-grade outlier samples. Grade capping was used for all variables on a zone-by-zone basis where outlier grades were identified using a combination of decile analysis and a review of log-probability plots. • Composites: A 1m composite length was chosen to ensure consistent sample support during estimation. Composites were limited to the boundaries of mineralised domains. • Variography: A variographic study by domain identified reasonably robust variogram models for Ag across main mineralised zone. • Estimation: Estimation was carried out using Ordinary Kriging as the primary method. An Inverse distance (squared) estimate was carried out for validation purposes. Only composite samples within an individual zone were used for estimation of that zone. Estimation parameters were based on models of grade continuity produced during geostatistical analysis. Dynamic anisotropy was used to change orientations of search ellipses based on local variations of dip and strike. Minimum and maximum sample criteria, an octant search restriction and restrictions of number of composite samples from a single drillhole were employed during grade estimation to assist with declustering and to reduce local grade bias. A multiple pass estimation as carried out with expanding search

Criteria	JORC Code explanation	Commentary
		ellipses and less restrictive estimation parameters for estimating blocks in more poorly sampled areas. • Estimation was carried out into parent cells only to reduce risk of conditional bias. Estimation was carried out using a discretisation of five points in each dimension. • The block model was verified first by comparing drillhole composite sample values with estimated block values on a sectional and plan basis. Grade comparison was also carried out statistically by zone to ensure the global grade estimate was unbiased. Grade profile (swath) plots were also constructed to compare modelled grades and input composite grades in slices or varying width. During this process a comparison was made between declustered and clustered data to identify any possible local bias introduced by irregular grade spacing. • No estimation of deleterious components was carried out. • The estimated block model was validated by visual inspection of block grades in comparison with drillhole data, and comparison of the block model statistics.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• All tonnages are reported as dry tonnages. • Moisture content has been measured using weighing waxed samples and dried ones.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• Mineralised zones are defined at a natural cut-off grade of 50g/t Ag. • The mineral resource estimate is restricted to material falling within an NPV Scheduler optimised pit shell as

Criteria	JORC Code explanation	Commentary
		described below in “Mining factors or assumptions”, and above a cut-off grade representing breakeven cut-off grade derived from open pit optimisation parameters for each zone.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• The deposit is deemed to be appropriate to being mined by standard open pit mining operation. • Reporting of mineral resources suitable for open pit extraction were limited by the creation of an optimised open pit shell in NPV Scheduler. The optimisation was carried out using Net Smelter Return data. The approach to NSR estimation is presented in the main text body. The pit shell was created with the following major parameters: – NSR (primary) – US$/t – 139.06; – Mining cost (mineralisation/waste) of US$2.53/t; – Oxide processing cost of US$72.91/t; – Primary processing cost US$46.97/t – Processing recovery – 95%; – G&A cost of US$60.0/t – Slope angle - 56° at hanging wall, 48°at foot wall; – Mining dilution of 30% and mining losses of 0%.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case,*	• Metallurgical recovery was utilised during the construction of an optimised pit shell used for limiting mineral resources based on an expectation of eventual economic extraction.

Criteria	JORC Code explanation	Commentary
	this should be reported with an explanation of the basis of the metallurgical assumptions made.	
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	• WAI is unaware of any environmental factors which would preclude the reporting of Mineral Resources.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	• Density measurements have been taken for primary material and waste with respect to natural moisture. • A total of 68 density measurements have been taken for primary and waste material. • Measurements were made using the Archimedes water immersion method, the results were recorded and imported into Excel spreadsheet. • Density was assigned to the block model during the Mineral Resource estimation by applying 3.56t/m^3 for primary material and 2.75 for waste. • Moisture content was measured for oxide and primary material. • The tonnage is reported on a dry basis.
Classification	• *The basis for the classification of the Mineral*	• Mineral Resource classification was carried out in

Criteria	JORC Code explanation	Commentary
	Resources into varying confidence categories. • *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	accordance with the guidelines of the JORC Code (2012). • The NM Silver Project is considered to be at an advance stage of development being explored on the tight drilling pattern of 25m x 25m. However, there is no robust definition of oxide/primary mineralisation based on the appropriative assay data and/or metallurgical testwork and as such the resources are reported of Inferred category only. • The mineral resource estimate classification reflects the Competent Person's view of the NM Project. • Mineral Resources for open pit mining were limited using an optimised pit shell using parameters as laid out in the main section of the report and as described in "Mining factors and assumptions" above. • The mineral resource estimate has been limited to the surveyed surface as detailed in the main report.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• WAI is not aware of any audits or reviews of this Mineral Resource Estimate other than internal peer review.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to*	• The relative accuracy and confidence in the mineral resource estimate is reflected in the reporting of the mineral resource as set out in the JORC Code (2012) • The statement relates to global estimates of tonnes and grade. • The classification applied to the mineral resource estimate is based upon; confidence of continuity of mineralisation, quality of data (QA/QC) and validation of the block model.

Criteria	JORC Code explanation	Commentary
	global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	

APPENDIX 4: FINANCIAL MODEL

Wardell Armstrong International
Financial Model

SBR

Russia

May-20

Assumptions

46.51 NPV @ 8.64%

Time Parameters | Source of data | Units | Average | Total

	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
Number of period	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
Beginning of period	01-Nov-19	01-Jan-20	01-Apr-20	01-Jul-20	01-Oct-20	01-Jan-20	01-Jan-21	01-Apr-21	01-Jul-21	01-Oct-21	01-Jan-21	01-Jan-22	01-Apr-22	01-Jul-22	01-Oct-22	01-Jan-22	01-Jan-23	01-Jan-24	01-Jan-25	01-Jan-26
End of period	31-Dec-19	31-Mar-20	30-Jun-20	30-Sep-20	31-Dec-20	31-Dec-20	31-Mar-21	30-Jun-21	30-Sep-21	31-Dec-21	31-Dec-21	31-Mar-22	30-Jun-22	30-Sep-22	31-Dec-22	31-Dec-22	31-Dec-23	31-Dec-24	31-Dec-25	31-Dec-26
Days in year	61	91	91	92	92	366	90	91	92	92	365	90	91	92	92	365	365	366	365	365

Metal Prices

	Source of data	Units	17.76 / Real 2019	Total	Nominal Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
Ag	SP ANGLE (27.08.19)	$/oz	17.76	17.76	17.76	17.85	17.94	18.03	18.12	18.12	18.21	18.30	18.39	18.48	18.48	18.57	18.66	18.75	18.85	18.85	19.22	19.61	20.00	20.40
Pb	SP ANGLE (27.08.19)	$/t	2,069	21%	2,069	2,079	2,090	2,100	2,110	2,110	2,121	2,131	2,142	2,153	2,153	2,163	2,174	2,185	2,196	2,196	2,240	2,284	2,330	2,377
Zn	SP ANGLE (27.08.19)	$/t	2,252		2,252	2,263	2,274	2,286	2,297	2,297	2,308	2,320	2,331	2,343	2,343	2,355	2,366	2,378	2,390	2,390	2,438	2,486	2,536	2,587

Macroeconomic Assumptions

	Source of data	Average	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8	
RUB/USD	SBR forecast	65	65	72	72	72	72	72	70	70	70	70	70	70	70	70	70	70	70	71	73	74	В этой строчке поправили лин
Annual Inflation for Capex (RUB)	SBR forecast		0.00%	1.18%	1.18%	1.18%	1.18%	4.70%	1.18%	1.18%	1.18%	1.18%	4.00%	1.18%	1.18%	1.18%	1.18%	4.00%	4.00%	4.00%	4.00%	4.00%	
Cummulative - capex (RUB)				1.17%	2.36%	3.57%	4.78%	4.78%	6.01%	7.26%	8.52%	9.80%	9.80%	11.09%	12.39%	13.71%	15.05%	15.05%	19.65%	24.44%	29.41%	34.59%	
Annual Inflation for Opex (RUB)			0.00%	1.18%	1.18%	1.18%	1.18%	4.70%	1.18%	1.18%	1.18%	1.18%	4.00%	1.18%	1.18%	1.18%	1.18%	4.00%	4.00%	4.00%	4.00%	4.00%	
Cummulative - Opex (RUB)				1.17%	2.36%	3.57%	4.78%	4.78%	6.01%	7.26%	8.52%	9.80%	9.80%	11.09%	12.39%	13.71%	15.05%	15.05%	19.65%	24.44%	29.41%	34.59%	
Long Term Inflation USD	WAI Assumption	2.00%	0.50%	0.50%	0.50%	0.50%	0.50%	2.00%	0.50%	0.50%	0.50%	0.50%	2.00%	0.50%	0.50%	0.50%	0.50%	2.00%	2.00%	2.00%	2.00%	2.00%	
Cummulative Inflation USD				0.50%	1.00%	1.50%	2.00%	2.00%	2.51%	3.01%	3.53%	4.04%	4.04%	4.56%	5.08%	5.60%	6.12%	6.12%	8.24%	10.41%	12.62%	14.87%	

Taxes

	Average	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
Corporate Income Tax	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%
VAT (not in use)	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%
Mineral Extraction Tax (Mining Royalty)																					
Base Metals	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%
Precious Metals	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%	6.5%

Depreciation Groups

		Units	Average
1	Buildings	%	10.00%
2	Machinery and equipment	%	25.00%
3	Vehicles	%	40.00%
4	Fixtures , facilities	%	15.00%
	Depreciation Rate (Weighted average)	%	9.50%

Working Capital

	Units	Average	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
No Days in Year	days	365.25																				
No Months in Year	months	12																				
Accounts payable	days	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45
Accounts receivable	days	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Inventory	days	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30

Payment Terms

Currency components

USD	RUB	Concentrate Charges	Units	Real 2019	Nominal Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
100%	0%	Transport	US$/tconc	274.9	275	276	278	279	280	280	282	283	285	286	286	287	289	290	292	292	298	304	310	316
100%	0%	Treatment	US$/tconc	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
100%	0%	Refining Zin	US$/kg	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
100%	0%	Refining Pb	US$/kg	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
100%	0%	Refining/Transport Silver	US$/tOz	0.4	0.40	0.40	0.40	0.41	0.41	0.41	0.41	0.41	0.41	0.42	0.42	0.42	0.42	0.42	0.42	0.42	0.43	0.44	0.45	0.46

Payment to Reclamation Fund

	Source of data	Units	Total	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
				0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	100.00%
Payment to Reclamation	*ARO 2017 -2028 v3.xlsx*	Rub'000	312,168	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4,207

Leasing Terms

Source: SBR data, based on the actual contracts

Leasing of OP mining equipment
Period between Jan 2020 to Feb 2023

N Yrs	N Months	Leasing Payments	Price (incl VAT)	Interest on Leasing	Currency	Total	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
2.00	24	Drill Rig Flexi Rock D60	56,776,534	8,023,273	Rub	56,776,534		7,097,067	7,097,067	7,097,067	7,097,067	28,388,267	7,097,067	7,097,067	7,097,067	7,097,067	28,388,267					0				
3.00	36	Excavator CAT 374FL	730,000	105,876	USD	730,000		60,833	60,833	60,833	60,833	243,333	60,833	60,833	60,833	60,833	243,333	60,833	60,833	60,833	60,833	243,333				
3.00	36	Dump Truck CAT740GC	586,400	85,049	USD	586,400		48,867	48,867	48,867	48,867	195,467	48,867	48,867	48,867	48,867	195,467	48,867	48,867	48,867	48,867	195,467				
3.00	36	Dump Truck CAT740GC	586,400	85,049	USD	586,400		48,867	48,867	48,867	48,867	195,467	48,867	48,867	48,867	48,867	195,467	48,867	48,867	48,867	48,867	195,467				
3.00	36	Dump Truck CAT740GC	586,400	85,049	USD	586,400		48,867	48,867	48,867	48,867	195,467	48,867	48,867	48,867	48,867	195,467	48,867	48,867	48,867	48,867	195,467				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	12,340,000		2,468,000	2,468,000	2,468,000	2,468,000	9,872,000	2,468,000				2,468,000					0				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	12,340,000		2,468,000	2,468,000	2,468,000	2,468,000	9,872,000	2,468,000				2,468,000					0				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	12,340,000		2,468,000	2,468,000	2,468,000	2,468,000	9,872,000	2,468,000				2,468,000					0				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	12,340,000		2,468,000	2,468,000	2,468,000	2,468,000	9,872,000	2,468,000				2,468,000					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	12,340,000		12,340,000				12,340,000					0					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	12,340,000		12,340,000				12,340,000					0					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	12,340,000		12,340,000				12,340,000					0					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	12,340,000		12,340,000				12,340,000					0					0				
		Total Rub	including inflation	Rub nominal	Rub	**157,323,618**		**67,108,433**	**17,168,453**	**17,168,453**	**17,168,453**	**118,613,793**	**17,168,453**	**7,180,457**	**7,180,457**	**7,180,457**	**38,709,825**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
		Total USD	including inflation	US$ nominal	USD	**2,501,554**		**208,463**	**208,463**	**208,463**	**208,463**	**833,851**	**208,463**	**208,463**	**208,463**	**208,463**	**833,851**	**208,463**	**208,463**	**208,463**	**208,463**	**833,851**	**0**	**0**	**0**	**0**
		Total in USD	including inflation	US$ nominal	USD	**4,699,680**		**1,139,232**	**446,583**	**446,583**	**446,583**	**2,478,980**	**453,726**	**311,041**	**311,041**	**311,041**	**1,386,849**	**208,463**	**208,463**	**208,463**	**208,463**	**833,851**	**0**	**0**	**0**	**0**
0.00		Interest on Leasing				0																				
2.00	24	Drill Rig Flexi Rock D60	56,776,534	8,023,273	Rub	8,023,273		1,002,909	1,002,909	1,002,909	1,002,909	4,011,637	1,002,909	1,002,909	1,002,909	1,002,909	4,011,637					0				
3.00	36	Excavator CAT 374FL	730,000	105,876	USD	105,876		8,823	8,823	8,823	8,823	35,292	8,823	8,823	8,823	8,823	35,292	8,823	8,823	8,823	8,823	35,292				
3.00	36	Dump Truck CAT740GC	586,400	85,049	USD	85,049		7,087	7,087	7,087	7,087	28,350	7,087	7,087	7,087	7,087	28,350	7,087	7,087	7,087	7,087	28,350				
3.00	36	Dump Truck CAT740GC	586,400	85,049	USD	85,049		7,087	7,087	7,087	7,087	28,350	7,087	7,087	7,087	7,087	28,350	7,087	7,087	7,087	7,087	28,350				
3.00	36	Dump Truck CAT740GC	586,400	85,049	USD	85,049		7,087	7,087	7,087	7,087	28,350	7,087	7,087	7,087	7,087	28,350	7,087	7,087	7,087	7,087	28,350				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	1,670,840		334,168	334,168	334,168	334,168	1,336,672	334,168				334,168					0				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	1,670,840		334,168	334,168	334,168	334,168	1,336,672	334,168				334,168					0				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	1,670,840		334,168	334,168	334,168	334,168	1,336,672	334,168				334,168					0				
1.25	15	Dump Truck SCANIA G440	12,340,000	1,670,840	Rub	1,670,840		334,168	334,168	334,168	334,168	1,336,672	334,168				334,168					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	334,168		334,168				334,168					0					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	334,168		334,168				334,168					0					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	334,168		334,168				334,168					0					0				
0.25	3	Dump Truck SCANIA G440	12,340,000	334,168	Rub	334,168		334,168				334,168					0					0				
		Total Rub	including inflation	Rub nominal	Rub	**16,231,814**		**3,719,449**	**2,367,071**	**2,367,071**	**2,367,071**	**10,820,663**	**2,367,071**	**1,014,693**	**1,014,693**	**1,014,693**	**5,411,151**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
		Total USD	including inflation	US$ nominal	USD	**362,815**		**30,235**	**30,235**	**30,235**	**30,235**	**120,938**	**30,235**	**30,235**	**30,235**	**30,235**	**120,938**	**30,235**	**30,235**	**30,235**	**30,235**	**120,938**	**0**	**0**	**0**	**0**
		Total in USD	including inflation	US$ nominal	USD	**590,195**		**81,822**	**63,065**	**63,065**	**63,065**	**271,017**	**64,050**	**44,730**	**44,730**	**44,730**	**198,240**	**30,235**	**30,235**	**30,235**	**30,235**	**120,938**	**0**	**0**	**0**	**0**

Source of data:

Прошу рассмотреть следующую информацию по стоимости карьерного оборудования, которое будет находиться в лизинге с января 2020 года по февраль 2023 года:

1. *Буровой станок FlexiRock D60 – покупная стоимость 56 776 534 руб.(вкл.НДС), сумма процентов по лизинговым платежам 8 023 273 руб. (вкл.НДС), 24 месяца;*
2. *Экскватор CAT 374FL – покупная стоимость 730 000 долларов США (вкл.НДС), сумма процентов по лизинговым платежам 105 876 долларов США (вкл.НДС), 36 месяцев;*
3. *Самосвал CAT740GC - покупная стоимость 586 400 долларов США (вкл.НДС), сумма процентов по лизинговым платежам 85 049 долларов США (вкл.НДС), 36 месяцев;*
4. *Самосвал CAT740GC - покупная стоимость 586 400 долларов США (вкл.НДС), сумма процентов по лизинговым платежам 85 049 долларов США (вкл.НДС), 36 месяцев;*
5. *Самосвал CAT740GC - покупная стоимость 586 400 долларов США (вкл.НДС), сумма процентов по лизинговым платежам 85 049 долларов США (вкл.НДС), 36 месяцев;*
6. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 1 670 840 руб. (вкл.НДС), 15 месяцев;*
7. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 1 670 840 руб. (вкл.НДС), 15 месяцев;*
8. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 1 670 840 руб. (вкл.НДС), 15 месяцев;*
9. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 1 670 840 руб. (вкл.НДС), 15 месяцев;*
10. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 334 168 руб. (вкл.НДС), 3 месяца;*
11. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 334 168 руб. (вкл.НДС), 3 месяца;*
12. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 334 168 руб. (вкл.НДС), 3 месяца;*
13. *Самосвал SCANIA G440 - покупная стоимость 12 340 000 руб.(вкл.НДС), сумма процентов по лизинговым платежам 334 168 руб. (вкл.НДС), 3 месяца;*

Depreciation of leased equipment – expense.
Principal payments – capex
Lease Interest payments – financial expenses

Supporting data

Units Convertion

			Name	3.7
	1000		thous	
	1000000		mln	
1 oz =	31.1035	g	oz to g	
1 kg =	32.15074326	oz	kg to oz	

TonnageUnitsList

Units Converter	Convert to (t)	TonnageCoefficientList Multiply by
kt	t	1,000
Mt	t	1,000,000
t	t	1

GradeUnitsList

Units Converter	Convert to (t)	GradeCoefficientList Multiply by
%	t/t	0.01
g/t	t/t	0.000001

ProductUnitList

From (t) to	ProductCoefficientList Multiply by	PriceUnitsList	Multiply by
g	1,000,000	US$/g	1,000,000
kg	1,000	US$/kg	1,000
kOz	32.15074326	US$/kOz	32.15074326
kt	0.001	US$/kt	0.001
lb	2,205	US$/lb	2,205
Oz	32,151	US$/oz	32,151
t	1	US$/t	1

List

No	1
Yes	2
to be updated	

Production Inputs 46.51 NPV @ 8.64%

Currency component	YEAR	Units	Assumption	Total	Y1 Q4	Q 1/2020	Q 2/2020	Q 3/2020	Q 4/2020	Y2	Q 1/2021	Q 2/2021	Q 3/2021	Q 4/2021	Y3	Q 1/2022	Q 2/2022	Q 3/2022	Q 4/2022	Y4	Y5	Y6	Y7	Y8
USD RUB	PERIOD END				01-Nov-19 31-Dec-19	01-Jan-20 31-Mar-20	01-Apr-20 30-Jun-20	01-Jul-20 30-Sep-20	01-Oct-20 31-Dec-20	01-Jan-20 31-Dec-20	01-Jan-21 31-Mar-21	01-Apr-21 30-Jun-21	01-Jul-21 30-Sep-21	01-Oct-21 31-Dec-21	01-Jan-21 31-Dec-21	01-Jan-22 31-Mar-22	01-Apr-22 30-Jun-22	01-Jul-22 30-Sep-22	01-Oct-22 31-Dec-22	01-Jan-22 31-Dec-22	01-Jan-23 31-Dec-23	01-Jan-24 31-Dec-24	01-Jan-25 31-Dec-25	01-Jan-26 31-Dec-26
Updat 23/04/2020	1. PRODUCTION SCHEDULE																							
	1.1 Mining Physicals: total mined			1,662,104	23,640	30,717	48,836	51,169	44,893	175,615	36,024	61,350	91,972	52,146	241,492	52,148	51,388	72,095	85,850	261,482	269,704	254,121	273,121	162,929
	Vertiklany Open Pit																							
	Mineralised Material	t		402,843	23,640	30,717	48,836	51,169	44,893	175,615	36,024	61,350	83,413	22,801	203,588	0	0	0	0	0	0	0	0	0
	Waste Material	t		10,995,762	382,943	703,343	1,610,736	1,603,752	1,611,106	5,528,938	1,988,975	1,986,149	846,328	262,430	5,083,882	0	0	0	0	0	0	0	0	0
	Mangazeisky North Open Pit																							
	Mineralised Material	t		418,996	-	0	0	0	0	0	0	0	8,559	29,345	37,904	52,148	50,147	68,340	73,335	243,970	137,121	0	0	0
	Waste Material	t		8,543,326	-	0	0	0	0	0	0	0	221,441	844,655	1,066,096	1,162,851	1,178,353	1,173,660	1,168,664	4,683,528	2,793,702	0	0	0
	Vertiklany Underground Mining																							
	Mineralised Material	t		840,265	-	0	0	0	0	0	0	0	0	0	0	0	1,241	3,756	12,515	17,512	132,583	254,121	273,121	162,929
										0					0					0				
	Vertiklany Underground Development																							
	Decline	m		7,411	-	0	0	0	0	0	0	0	0	0	0	0	269	638	580	1,487	2,192	2,343	1,389	0
	Level Access	m		9,982	-	0	0	0	0	0	0	0	0	0	0	0	153	190	576	919	3,650	3,532	1,784	97
	Vent Connection	m		1,061	-	0	0	0	0	0	0	0	0	0	0	0	13	91	72	175	261	450	175	0
	1.2 Ore Sorter Feed																							
	Total Ore feed to Sorter			1,707,264	20,039	29,894	45,500	45,500	46,251	167,145	45,500	45,500	45,500	68,181	204,681	68,180	68,180	68,180	68,180	272,720	272,720	272,720	272,720	224,519
	Leach Plant (Current)	t		344,525	20,039	29,894	45,500	45,500	46,251	167,145	45,500	45,500	45,500	20,841	157,341	-	-	-	-	0	0	0	0	0
	Oxide Feed	t		302,594	20,039	29,894	45,500	45,500	37,418	158,311	12,402	45,500	45,500	20,841	124,243	0	0	0	0	0	0	0	0	0
	Ag (Oxide)	g/t			588	581	783	412	177	495	393	803	766	716	734	0	0	0	0	0	0	0	0	0
	Ag (Oxide)	oz'000		5,831	379	558	1,146	603	213	2,520	157	1,175	1,121	480	2,933	0	0	0	0	0	0	0	0	0
	Sulphide Feed	t		41,931	-	0	0	0	8,833	8,833	33,098	0	0	0	33,098	0	0	0	0	0	0	0	0	0
	Ag (Sulphide)	g/t			-	0	0	0	762	762	671	0	0	0	671	0	0	0	0	0	0	0	0	0
	Ag (Sulphide)	oz'000		931	0	0	0	0	216	216	714	0	0	0	714	0	0	0	0	0	0	0	0	0
	Flotation Plant																							
	Feed	t		1,362,739	0.0	0	0	0	0	0	0	0	0	47,340	47,340	68,180	68,180	68,180	68,180	272,720	272,720	272,720	272,720	224,519
	Ag	g/t			0.0	0	0	0	0	0	0	0	0	641	641	564	538	502	475	520	451	394	428	460
	Ag	oz'000		25,544	0	0	0	0	0	0	0	0	0	975	975	1,236	1,180	1,100	1,042	4,557	3,952	3,457	3,749	3,322
	Pb	%			-	0	0	0	0	0	0	0	0	2	2	3	4	5	5	4	5	2	2	3
	Zn	%			-	0	0	0	0	0	0	0	0	1	1	1	1	1	1	1	2	2	1	1
Overall Recovery	**Total Mined Metals (Mining Royalty Basis)**		Recovered	Mined					Shortfall		Shortfall													
82.47%	Ag	oz'000	22,081	26,774	379	558	1,146	603	429	2,736	871	1,175	1,121	1,455	4,622	1,236	1,180	1,100	1,042	4,557	3,952	3,457	3,749	3,322
68.81%	Pb	t	30,929	44,948	0	0	0	0	0	0	0	0	0	1,095	1,095	2,141	2,934	3,362	3,297	11,734	14,494	6,469	4,340	6,815
94.09%	Zn	t	16,908	17,969	0	0	0	0	0	0	0	0	0	594	594	581	420	341	504	1,846	4,100	5,154	4,012	2,263
	1.3 Process Plant Feed			1,143,771	20,039	29,894	30,030	30,030	30,526	120,479	30,030	30,030	30,030	45,000	135,090	44,999	44,999	44,999	44,999	179,995	179,995	179,995	179,995	148,182
			Sorter Output						Shortfall		Shortfall													
	Leach Plant (Current)		71%	244,364	20,039	29,894	30,030	30,030	30,526	120,479	30,030	30,030	30,030	13,755	103,845	0	0	0	0	0	0	0	0	0
	Oxide Feed	t	72%	216,689	20,039	29,894	30,030	30,030	24,696	114,650	8,185	30,030	30,030	13,755	82,001	0	0	0	0	0	0	0	0	0
	Ag (Oxide)	g/t			588	581	1,175	618	266	678	589	1,205	1,150	1,074	1,101	0	0	0	0	0	0	0	0	0
	Ag (Oxide)	oz'000		5,782	379	558	1,134	597	211	2,500	155	1,163	1,110	475	2,903	0	0	0	0	0	0	0	0	0
	Sulphide Feed	t	66%	27,675	-	0	0	0	5,830	5,830	21,845	0	0	0	21,845	0	0	0	0	0	0	0	0	0
	Ag (Sulphide)	g/t			-	0	0	0	1,143	1,143	1,007	0	0	0	1,007	0	0	0	0	0	0	0	0	0
	Ag (Sulphide)	oz'000		921	0	0	0	0	214	214	707	0	0	0	707	0	0	0	0	0	0	0	0	0
	Flotation Plant (Available in mid 2021)																							
	Sulphide	t	66%	899,408	-	0	0	0	0	0	0	0	0	31,244	31,244	44,999	44,999	44,999	44,999	179,995	179,995	179,995	179,995	148,182
	Ag	g/t			-	0	0	0	0	0	0	0	0	961	961	846	807	752	713	780	676	591	641	690
	Pb	%			-	0	0	0	0	0	0	0	0	3	3	5	6	7	7	6	8	5	4	3
	Zn	%			-	0	0	0	0	0	0	0	0	2	2	1	1	1	1	1	2	3	3	3
	2. REVENUE																							
	Metal Prices																							
	Ag	US$/tOz			17.76	17.85	17.94	18.03	18.12	18.12	18.21	18.30	18.39	18.48	18.48	18.57	18.66	18.75	18.85	18.85	19.22	19.61	20.00	20.40
	Pb	US$/t			2,069	2,079	2,090	2,100	2,110	2,110	2,121	2,131	2,142	2,153	2,153	2,163	2,174	2,185	2,196	2,196	2,240	2,284	2,330	2,377
	Zn	US$/t			2,252	2,263	2,274	2,286	2,297	2,297	2,308	2,320	2,331	2,343	2,343	2,355	2,366	2,378	2,390	2,390	2,438	2,486	2,536	2,587
	g:tOz	31.1035																						
	2.1 Leach Plant (Current Plant)																							
	Mill Recovery (**Silver** Only)																							
	Oxides	%			85.00	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85	85
	Sulphides	%			28.90	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29
	Recovered Silver	g		161,154,798	10,020,283	14,755,517	29,986,205	15,781,909	7,499,856	68,023,488	10,454,097	30,758,723	29,342,118	12,556,090	83,111,028	0	0	0	0	0	0	0	0	0
	Recovered Silver	oz'000		5,181	322	474	964	507	241	2,187	336	989	943	404	2,672	0	0	0	0	0	0	0	0	0
	Payability	%			98.00	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98
Gross Revenue	**Gross Value**	US$ nominal		94,011,110	5,721,550	8,467,168	17,292,401	9,146,256	4,368,042	39,273,867	6,118,859	18,092,653	17,345,046	7,459,135	49,015,693	0	0	0	0	0	0	0	0	0
Sales Cost	Refining Cost	US$ nominal		2,117,367	128,864	190,702	389,468	205,997	98,379	884,547	137,812	407,492	390,654	167,999	1,103,957	0	0	0	0	0	0	0	0	0
	Value (Less: Refining Cost)	US$ nominal		91,893,742	5,592,686	8,276,466	16,902,932	8,940,260	4,269,662	38,389,321	5,981,047	17,685,161	16,954,392	7,291,136	47,911,736	0	0	0	0	0	0	0	0	0
	2.2 Flotation Plant (Proposed Plant)																							
	i) Zinc Concentrate																							
	Mill Recovery																							
	Zn	%			82.20	82	82	82	82	82	82	82	82	82	82	82	82	82	82	82	82	82	82	82
	Ag	%			4.70	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
	Contained Metal																							
	Zn	t		16,908	0	0	0	0	0	0	0	0	0	484	484	473	342	277	410	1,503	3,337	3,983	3,832	3,770
	Ag	g		28,961,909	0	0	0	0	0	0	0	0	0	1,411,036	1,411,036	1,788,718	1,707,075	1,591,434	1,507,683	6,594,910	5,719,329	5,003,594	5,425,445	4,807,594
	Recovered Silver	oz'000		931	0	0	0	0	0	0	0	0	0	45	45	58	55	51	48	212	184	161	174	155
	Concentrate																							
	Zn	%			42.3	42	42	42	42	42	42	42	42	42	42	42	42	42	42	42	42	42	42	42
	Ag	g/t			-	-	-	-	-	-	-	-	-	1,234	1,234	1,600	2,113	2,426	1,554	1,857	725	531	599	539
	Mass	t		39,972	-	0	0	0	0	0	0	0	0	1,144	1,144	1,118	808	656	970	3,552	7,888	9,416	9,059	8,912
	Zinc Component																							
	Deductions	%			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Payability	%			45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45
Gross Revenue	**Gross Value**	US$ nominal		18,991,918	-	0	0	0	0	0	0	0	0	510,026	510,026	501,183	363,812	296,896	441,458	1,603,349	3,660,184	4,456,508	4,373,377	4,388,473
	Transport Cost	US$ nominal	8%	13,153,643	0	0	0	0	0	0	0	0	0	353,240	353,240	347,115	251,973	205,628	305,750	1,110,466	2,535,013	3,086,540	3,028,964	3,039,420
	Treatment Cost	US$ nominal		-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Refining Cost	US$ nominal		-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sales Cost	**Total Costs**	US$ nominal		13,153,643	0	0	0	0	0	0	0	0	0	353,240	353,240	347,115	251,973	205,628	305,750	1,110,466	2,535,013	3,086,540	3,028,964	3,039,420
	Zn Value in Zn Concentrate (Less: Sales Costs	US$ nominal		5,838,275	-	0	0	0	0	0	0	0	0	156,786	156,786	154,068	111,839	91,268	135,708	492,883	1,125,171	1,369,968	1,344,413	1,349,053
	Silver Component																							
	Deductions	g		-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Payability	%			45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45
Gross Revenue	**Gross Value**	US$ nominal		8,160,673	0	0	0	0	0	0	0	0	0	377,211	377,211	480,550	460,892	431,803	411,109	1,784,354	1,590,714	1,419,480	1,569,938	1,418,977
Sales Cost	Refining Cost	US$ nominal		408,442	0	0	0	0	0	0	0	0	0	18,879	18,879	24,052	23,068	21,612	20,576	89,307	79,615	71,045	78,575	71,020

	Ag Value in Zn Concentrate (Less: Refining Co:	US$ nominal		7,752,231	0	0	0	0	0	0	0	0	0	358,332	358,332	456,498	437,824	410,191	390,533	1,695,047	1,511,098	1,348,435	1,491,363	1,347,957
	Zinc Concentrate NSR	US$ nominal		13,590,506	0	0	0	0	0	0	0	0	0	515,118	515,118	610,566	549,663	501,459	526,241	2,187,929	2,636,270	2,718,403	2,835,776	2,697,010
	ii) Lead Concentrate																							
	Mill Recovery																							
	Pb	%			66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66	66
	Ag	%			65	65	65	65	65	65	65	65	65	65	65	65	65	65	65	65	65	65	65	65
	Contained Metal																							
	Pb	t		30,929	0	-	-	-	-	0	-	-	-	715	715	1,397	1,914	2,194	2,151	7,655	9,456	5,760	4,346	2,997
	Ag	g		400,537,043	0	-	-	-	-	0	-	-	-	19,514,327	19,514,327	24,737,591	23,608,488	22,009,193	20,850,936	91,206,208	79,097,109	69,198,645	75,032,746	66,488,008
	Ag	oz'000		12,878	0	-	-	-	-	0	-	-	-	627	627	795	759	708	670	2,932	2,543	2,225	2,412	2,138
	Concentrate																							
	Pb	%			17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17
	Ag	g/t			-	-	-	-	-	-	-	-	-	4,669	4,669	3,028	2,109	1,716	1,658	2,037	1,430	2,054	2,952	3,794
	Mass	t		180,870	0	-	-	-	-	0	-	-	-	4,180	4,180	8,169	11,192	12,828	12,578	44,767	55,300	33,682	25,416	17,526
	Payment Terms																							
	Deductions for Lead	%			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Pb Payability	%			84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84
	Deductions for Ag	g			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Ag Payability	%			84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84	84
Gross Revenue	**Gross Value of Lead Concentrate**	US$ nominal		269,321,534	0	-	-	-	-	0	-	-	-	11,030,259	11,030,259	14,944,020	15,393,250	15,173,134	14,579,877	60,090,281	58,854,745	47,696,486	49,035,156	42,614,606
	Pb in Lead Concentrate Value	US$ nominal		58,648,706	0	-	-	-	-	0	-	-	-	1,292,316	1,292,316	2,538,326	3,495,034	4,025,881	3,966,849	14,026,089	17,789,487	11,051,741	8,506,227	5,982,846
	Ag in Lead Concentrate Value	US$ nominal		210,672,828	0	-	-	-	-	0	-	-	-	9,737,942	9,737,942	12,405,694	11,898,217	11,147,253	10,613,029	46,064,192	41,065,258	36,644,746	40,528,930	36,631,761
	Transport Cost	US$ nominal	8%	58,589,668	0	-	-	-	-	0	-	-	-	1,291,015	1,291,015	2,535,771	3,491,515	4,021,829	3,962,856	14,011,970	17,771,579	11,040,616	8,497,664	5,976,823
	Treatment Cost	US$ nominal		-	0	-	-	-	-	0	-	-	-	-	0	-	-	-	-	0	0	0	0	0
	Refining Cost Pb	US$ nominal		-	0	-	-	-	-	0	-	-	-	-	0	-	-	-	-	0	0	0	0	0
	Refining Cost Ag	US$ nominal		5,648,671	0	-	-	-	-	0	-	-	-	261,099	261,099	332,628	319,021	298,886	284,562	1,235,097	1,101,063	982,538	1,086,683	982,190
Sales Cost	Total Costs	US$ nominal		64,238,339	0	-	-	-	-	0	-	-	-	1,552,114	1,552,114	2,868,399	3,810,537	4,320,715	4,247,418	15,247,068	18,872,643	12,023,154	9,584,347	6,959,013
	Lead Concentrate NSR	US$ nominal		205,083,195	0	-	-	-	-	0	-	-	-	9,478,144	9,478,144	12,075,621	11,582,713	10,852,419	10,332,460	44,843,214	39,982,102	35,673,332	39,450,809	35,655,593
	iii) Lead/Silver Middlings																							
	Mill Recovery - Ag	%			15.6	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16
	Contained Metal - Ag	g		122,701,819	0	0	0	0	0	0	0	0	0	4,683,439	4,683,439	5,937,022	5,666,037	5,282,206	5,004,225	21,889,490	18,983,306	16,607,675	18,007,859	15,957,122
	Recovered Silver	oz'000		3,091	0	0	0	0	0	0	0	0	0	151	151	191	182	170	161	704	610	534	579	513
	Payability	%			98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98	98
Gross Revenue	**Gross Value of Lead/Silver Middlings**	US$ nominal		58,988,354	-	-	-	-	-	0	-	-	-	2,726,622	2,726,622	3,473,592	3,331,498	3,121,229	2,971,646	12,897,965	11,498,265	10,260,522	11,348,093	10,256,886
Sales Cost	Refining Cost	US$ nominal		1,355,680	-	-	-	-	-	0	-	-	-	62,664	62,664	79,831	76,565	71,733	68,295	296,423	264,255	235,809	260,804	235,725
	NSR Value (Less: Refining Cost)	US$ nominal		57,632,674	-	-	-	-	-	0	-	-	-	2,663,958	2,663,958	3,393,761	3,254,933	3,049,496	2,903,351	12,601,542	11,234,010	10,024,713	11,087,289	10,021,161
	Total Flotation Plant Net Smelter Return	US$ nominal		276,306,374	-	-	-	-	-	0	-	-	-	12,657,221	12,657,221	16,079,949	15,387,310	14,403,375	13,762,052	59,632,685	53,852,382	48,416,448	53,373,874	48,373,764
2.3	**Total Net Revenue**																							
	Leach Plant Revenue	US$ nominal		91,893,742	5,592,686	8,276,466	16,902,932	8,940,260	4,269,662	38,389,321	5,981,047	17,685,161	16,954,392	7,291,136	47,911,736	0	0	0	0	0	0	0	0	0
	Flotation Pant Revenue	US$ nominal		276,306,374	0	0	0	0	0	0	0	0	0	12,657,221	12,657,221	16,079,949	15,387,310	14,403,375	13,762,052	59,632,685	53,852,382	48,416,448	53,373,874	48,373,764
	Total Revenue	US$ nominal		368,200,117	5,592,686	8,276,466	16,902,932	8,940,260	4,269,662	38,389,321	5,981,047	17,685,161	16,954,392	19,948,357	60,568,957	16,079,949	15,387,310	14,403,375	13,762,052	59,632,685	53,852,382	48,416,448	53,373,874	48,373,764
	3. OPERATING COSTS																							
3.1	**MINING OPEX**																							
USD / RUB	**Open Pit Operating Costs**				5.24					1.91					2.04					2.47	2.14	-	-	-
0% / 100%	Model 1	US$ nominal		2.15	5.24	1.85	1.87	1.89	1.91	1.91	1.97	1.99	2.01	2.04	2.04	2.38	2.41	2.44	2.47	2.47	2.14	-	-	-
	Total Moved Tonnes	t		20,360,927	406,582	734,059	1,659,573	1,654,921	1,655,999	5,704,553	2,024,999	2,047,499	1,159,741	1,159,231	6,391,469	1,215,000	1,228,500	1,242,000	1,242,000	4,927,499	2,930,824	-	-	-
	Open Pit Operating Costs	US$ nominal		43,867,513	2,128,880	1,356,048	3,101,797	3,129,447	3,168,281	10,755,574	3,981,570	4,073,113	2,334,195	2,360,582	12,749,460	2,897,028	2,963,636	3,031,409	3,067,028	11,959,100	6,274,499	-	-	-
	Leasing Interest	US$ nominal		590,195	0	81,822	63,065	63,065	63,065	271,017	64,050	44,730	44,730	44,730	198,240	30,235	30,235	30,235	30,235	120,938	0	0	0	0
	Underground Operaitng Costs				-					-					-					264.55	66.07	39.10	31.80	39.25
0% / 100%	Model 1 (Fully owned/operated)	US$/$t_{ore}$ nominal		45.69	-	-	-	-	-	-	-	-	-	-	-	255.43	258.44	261.47	264.55	264.55	66.07	39.10	31.80	39.25
	Total Mineralised Tonnes	t		840,265	-	-	-	-	-	-	-	-	-	-	-	-	1,241	3,756	12,515	17,512	132,583	254,121	273,121	162,929
	Underground Operating Costs	US$ nominal		38,389,477	-	-	-	-	-	-	-	-	-	-	-	-	320,785	982,012	3,310,686	4,613,483	8,759,682	9,937,268	8,683,948	6,395,095
				82,256,990																				
	Total Mining Operating Costs	US$ nominal		82,256,990	2,128,880	1,356,048	3,101,797	3,129,447	3,168,281	10,755,574	3,981,570	4,073,113	2,334,195	2,360,582	12,749,460	2,897,028	3,284,421	4,013,421	6,377,714	16,572,583	15,034,181	9,937,268	8,683,948	6,395,095
3.2	**PROCESSING OPEX**																							
	Share of actual material sorted of total ROM		71%																					
0% / 100%	Ore Sorting Cost	US$nominal/t	2.25	2,889,817	-	-	2.07	2.09	2.12	2.12	2.21	2.23	2.26	2.28	2.28	2.31	2.34	2.37	2.39	2.39	2.49	2.54	2.59	2.64
	Ore Sorting Cost	US$ nominal		2,889,817	0	0	66,966	67,752	69,681	204,399	71,435	72,275	73,124	110,863	327,697	112,164	113,481	114,815	116,164	456,624	483,242	492,907	502,765	422,183
	Leach Plant (Current Plant)																							
0% / 100%	Unit Processing Cost (Oxides)	US$(2019)/t	72.95		72.95	66.24	67.02	67.81	68.60	68.60	71.49	72.33	73.18	74.04	74.04	74.91	75.79	76.68	77.58	77.58	80.69	82.30	83.95	85.63
0% / 100%	Unit Processing Cost (Sulphides)	US$(2019)/t	123.71		123.71	112.33	113.65	114.99	116.34	116.34	121.24	122.66	124.11	125.56	125.56	127.04	128.53	130.04	131.57	131.57	136.83	139.57	142.36	145.21
	Oxide Processing Cost	US$ nominal		15,158,759	1,461,851	1,980,270	2,012,609	2,036,254	1,694,247	7,723,380	585,190	2,172,167	2,197,690	1,018,481	5,973,529	0	0	0	0	-	0	0	0	0
	Sulphide Porcessing Cost	US$ nominal		3,326,705	0	0	0	0	678,258	678,258	2,648,447	0	0	0	2,648,447	0	0	0	0	-	0	0	0	0
	Leach Plant Processing Cost	US$ nominal		18,485,464	1,461,851	1,980,270	2,012,609	2,036,254	2,372,505	8,401,637	3,233,637	2,172,167	2,197,690	1,018,481	8,621,976	-	-	-	-	-	0	0	0	0
	Flotation Plant (New Plant)																							
0% / 100%	Unit Processing Cost (Sulphides)	US$(nominal)/t	47.18	52.62	47.18	42.84	43.35	43.86	44.37	44.37	46.24	46.78	47.33	47.89	47.89	48.45	49.02	49.60	50.18	50.18	52.19	53.23	54.30	55.38
	Flotatation Plan Processing Cost	US$ nominal		47,327,536	-	-	-	-	-	-	-	-	-	1,496,301.04	1,496,301	2,180,323.49	2,205,942.29	2,231,862.12	2,258,086.50	8,876,214	9,393,640	9,581,513	9,773,143	8,206,725
	Total Processing Operating Costs	US$ nominal		68,702,817	1,461,851	1,980,270	2,079,574	2,104,006	2,442,185	8,606,036	3,305,072	2,244,442	2,270,814	2,625,646	10,445,974	2,292,487	2,319,424	2,346,677	2,374,250	9,332,838	9,876,882	10,074,420	10,275,908	8,628,908
3.3.	**General and Administration Cost**																							
USD / RUB	*Proportion to the annual cost*			46,732,579	25%	25%	25%	25%	25%		25%	25%	25%	25%		25%	25%	25%	25%		100%	100%	100%	100%
10% / 90%	Total G&A (Both Infrastructure and Management)	US$ nominal	6,000,000	46,732,579	1,500,000	1,376,611	1,391,763	1,407,088	1,422,588	5,598,049	1,476,800	1,493,109	1,509,604	1,526,288	6,005,801	1,543,163	1,560,230	1,577,493	1,594,954	6,275,840	6,636,460	6,769,190	6,904,573	7,042,665
	4. PROJECT CAPITAL COSTS																							
4.1	**Mining Capital Expenditure**																							
	Open Pit Capital Cost Schedule																							
85% / 15%	*CAPEX Equipment (Overhaul)*	*US$ (2019)*		*1,233,831*	-	-	-	-	-	*0*	-	-	*616,915*	*616,915*	*1,233,831*	-	-	-	-	*0*	-	-	-	-
0% / 100%	*CAPEX Road Prepartion*	*US$ (2019)*		*1,296,272*	-	*324,068*	*324,068*	*324,068*	*324,068*	*1,296,272*	-	-	-	-	*0*	-	-	-	-	*0*	-	-	-	-
	CAPEX Equipment (Overhaul)	US$ nominal		1,275,189	-	-	-	-	-	0	-	-	635,701.89	639,486.90	1,275,189	-	-	-	-	0	-	-	-	-
	CAPEX Road Prepartion	US$ nominal		1,197,987	-	294,269.56	297,727.22	301,225.52	304,764.92	1,197,987	-	-	-	-	0	-	-	-	-	0	-	-	-	-
	Total for Open Pit	US$ nominal		2,473,176	0	294,270	297,727	301,226	304,765	1,197,987	0	0	635,702	639,487	1,275,189	0	0	0	0	0	0	0	0	0
	Underground Capital Cost Schedule																							
10% / 90%	CAPEX Development	US$ (2019)		4,311,545	-	-	-	-	-	0	-	-	-	-	0	211,110	211,110	211,110	211,110	844,439	1,253,400	1,373,929	839,776	-
10% / 90%	CAPEX Equipment	US$ (2019)		19,016,097	-	-	-	-	-	0	2,580,949	2,580,949	2,580,949	2,580,949	10,323,797	951,000	951,000	951,000	951,000	3,804,000	1,255,500	2,339,400	1,293,400	-
	CAPEX Development	US$ nominal		4,777,723	-	-	-	-	-	0	-	-	-	-	0	217,184.54	219,586.65	222,016.24	224,473.63	883,261	1,383,393.49	1,546,751.16	964,316.91	-
	CAPEX Equipment	US$ nominal		19,817,251	-	-	-	-	-	0	2,541,030.66	2,569,092.16	2,597,474.47	2,626,181.32	10,333,779	978,365.14	989,186.07	1,000,130.80	1,011,200.76	3,978,883	1,385,710.76	2,633,664.98	1,485,214.13	-
	Total for Underground	US$ nominal		24,594,974	0	0	0	0	0	0	2,541,031	2,569,092	2,597,474	2,626,181	10,333,779	1,195,550	1,208,773	1,222,147	1,235,674	4,862,144	2,769,104	4,180,416	2,449,531	0
0																								
	Mining Equipment Leasing Principal Payback Sche	US$ nominal		4,699,680		1,139,232	446,583	446,583	446,583	2,478,980	453,726	311,041	311,041	311,041	1,386,849	208,463	208,463	208,463	208,463	833,851	0	0	0	0
	Mining CAPEX Total	US$ nominal		31,767,830	-	1,433,501	744,310	747,808	751,348	3,676,967	2,994,757	2,880,133	3,544,217	3,576,709	12,995,816	1,404,012	1,417,236	1,430,610	1,444,137	5,695,995	2,769,104	4,180,416	2,449,531	-
4.2	**Processing Captial Costs**																							
	Processing Captial Costs (For new Plant)																							

	Flotation Plant Captial Cost. Choose option >>			9,000,000																					
90% 10%	1. New flotation processing plant; or,			17,865,257																					
	2. Retrofited and upgraded current plant			9,000,000																					
	XRT Component			2,000,000																					
	Processing Capital Cost	US$ nominal		11,238,257			2,001,655			2,001,655		9,236,602			9,236,602					0				
	TOTAL PROJECT CAPITAL COST	US$ nominal		**43,006,087**	**0**	**1,433,501**	**2,745,965**	**747,808**	**751,348**	**5,678,622**	**2,994,757**	**12,116,735**	**3,544,217**	**3,576,709**	**22,232,418**	**1,404,012**	**1,417,236**	**1,430,610**	**1,444,137**	**5,695,995**	**2,769,104**	**4,180,416**	**2,449,531**	**0**
	5. Taxes and Depreciation																								
	Depreciation																								
	Disposal of Assets	US$'000		0																					
1	Group (Average Weghted Standard Rate)		9.50%																						
	Initial Balance				–	98,201,541	90,305,896	84,472,800	77,195,693	**70,613,450**	70,613,450	66,899,929	72,661,171	69,302,576	**66,295,541**	66,295,541	61,401,477	56,985,572	53,002,553	**49,411,447**	49,411,447	47,486,464	47,155,666	45,125,409
	Capex			43,006,087	0	1,433,501	2,745,965	747,808	751,348	**5,678,622**	2,994,757	12,116,735	3,544,217	3,576,709	**22,232,418**	1,404,012	1,417,236	1,430,610	1,444,137	**5,695,995**	2,769,104	4,180,416	2,449,531	0
	Depreciation				0	9,329,146	8,579,060	8,024,916	7,333,591	**33,266,713**	6,708,278	6,355,493	6,902,811	6,583,745	**26,550,327**	6,298,076	5,833,140	5,413,629	5,035,242	**22,580,089**	4,694,087	4,511,214	4,479,788	4,286,914
	Final Balance				98,201,541	90,305,896	84,472,800	77,195,693	70,613,450	70,613,450	66,899,929	72,661,171	69,302,576	66,295,541	66,295,541	61,401,477	56,985,572	53,002,553	49,411,447	49,411,447	47,486,464	47,155,666	45,125,409	40,838,495
	Total Depreciation			**100,369,132**	**0**	**9,329,146**	**8,579,060**	**8,024,916**	**7,333,591**	**33,266,713**	**6,708,278**	**6,355,493**	**6,902,811**	**6,583,745**	**26,550,327**	**6,298,076**	**5,833,140**	**5,413,629**	**5,035,242**	**22,580,089**	**4,694,087**	**4,511,214**	**4,479,788**	**4,286,914**
	Corporate Income Tax																								
	Income subject to tax			**59,156**	64	–	352	–	–	**352**	0	2,077	2,552	4,760	**9,389**	1,047	339	0	0	**1,387**	**9,276**	**10,511**	**16,533**	**11,644**
	Estimated Income Tax for the period			**11,831**	13	–	70	–	–	**70**	–	415	510	952	**1,878**	209	68	–	–	**277**	1,855	2,102	3,307	2,329
	Allowable reduction of payable tax in the current period		50%																						
	Losses from previous periods as of 2019 in CAD	CAD (2019)	34,571,789 CAD																						
	in USD	USD (2019)	26,618,255 USD																						
	Exchanged rate applied		1.30																						
	Allowance for carried forward taxes @ 20%	CAD (2019)	6,914,358 CAD																						
		USD (2019)	5,323,651																						
	Allowance for carried forward losses O/B	US$'000			5,324	5,317	6,616	6,581	7,880	**7,880**	10,013	12,130	11,922	11,667	**11,667**	11,191	11,086	11,053	11,247	**11,247**	11,967	11,040	9,988	8,335
	Current period losses allowance (20% applied)	US$'000		**7,763**	–	1,299	–	1,299	2,133	**4,731**	2,117	–	–	–	**2,117**	–	–	194	720	**915**	–	–	–	–
	Reduction in tax accounted for carried losses	US$'000		**-3,587**	(6.4)	–	(35.2)	–	–	**- 35**	–	(207.7)	(255.2)	(476.0)	**- 939**	(104.7)	(33.9)	–	–	**- 139**	(927.6)	(1,051.1)	(1,653.3)	1,164.4
	Allowance for carried forward losses C/B	US$'000			5,317	6,616	6,581	7,880	10,013	**10,013**	12,130	11,922	11,667	11,191	**11,191**	11,086	11,053	11,247	11,967	**11,967**	11,040	9,988	8,335	9,500
	Payable Income Tax for the period	US$'000		**8,244**	6.4	–	35.2	–	–	**35**	–	207.7	255.2	476.0	**939**	104.7	33.9	–	–	**139**	927.6	1,051.1	1,653.3	3,493.2
	Savings on tax			*3,587*																					
	Mining Royalty (MET)																								
US$ M	**Mining Royalty (MET)**			**44,999**	438	647	1,336	706	506	**3,195**	1,030	1,398	1,340	2,047	**5,815**	1,972	2,020	1,993	1,952	**7,937**	**8,335**	**6,614**	**6,497**	**6,169**
33.31	Silver	US$'000 nominal	7%	**33,310**	438	647	1,336	706	506	**3,195**	1,030	1,398	1,340	1,747	**5,515**	1,492	1,431	1,340	1,276	**5,539**	**4,938**	**4,407**	**4,874**	**4,405**
8.12	Lead	US$'000 nominal	8%	**8,120**	0	0	0	0	0	**-**	0	0	0	189	**189**	371	510	588	579	**2,048**	**2,597**	**1,182**	**809**	**1,296**
3.57	Zinc	US$'000 nominal	8%	**3,569**	0	0	0	0	0	**-**	0	0	0	111	**111**	109	79	65	96	**350**	**800**	**1,025**	**814**	**468**
	6. Working Capital																								
	Inventories	US$'000 nominal			**1,765,933**	1,099,885	1,708,145	1,706,561	1,829,500	**6,344,091**	2,428,881	2,082,711	1,501,633	1,625,944	**7,639,169**	1,729,838	1,847,421	2,073,945	2,853,901	**8,505,106**	**2,047,485**	**1,640,302**	**1,558,344**	**1,234,850**
	A/R	US$'000 nominal			**2,750,501**	2,728,505	5,572,395	2,915,302	1,392,281	**12,608,484**	1,993,682	5,830,273	5,528,606	6,504,899	**19,857,460**	5,359,983	5,072,740	4,696,753	4,487,626	**19,617,101**	**4,426,223**	**3,968,561**	**4,386,894**	**3,975,926**
	A/P	US$'000 nominal			**4,078,225**	2,650,756	3,910,971	3,593,654	3,687,355	**13,842,737**	4,896,930	4,553,504	3,646,173	4,186,830	**17,283,437**	4,352,247	4,541,794	4,857,360	6,015,645	**19,767,046**	**4,916,973**	**4,105,904**	**3,989,723**	**3,481,102**
	Total Working Capital	US$'000 nominal			**438,209**	**1,177,634**	**3,369,569**	**1,028,209**	**-465,574**	**5,109,838**	**-474,367**	**3,359,480**	**3,384,067**	**3,944,013**	**10,213,192**	**2,737,574**	**2,378,367**	**1,913,337**	**1,325,881**	**8,355,160**	**1,556,735**	**1,502,960**	**1,955,515**	**-**
	Change in Working Capital	US$'000 nominal	0	OK	**438,209**	739,425	2,191,935	-2,341,360	-1,493,782	**- 903,783**	-8,793	3,833,847	24,587	559,946	**4,409,586**	-1,206,438	-359,207	-465,030	-587,456	**- 2,618,131**	230,853	**- 53,775**	**452,555**	**- 1,955,515**

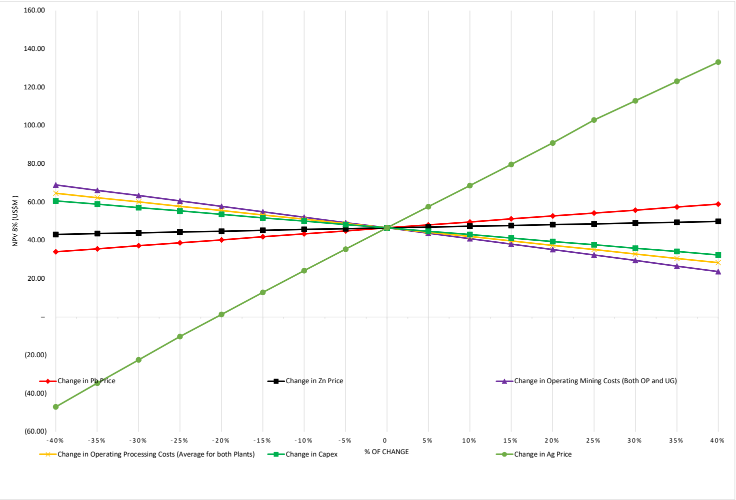
			46.51 NPV	@ 8.64%	01-Nov-19	01-Jan-20	01-Apr-20	01-Jul-20	01-Oct-20	01-Jan-20	01-Jan-21	01-Apr-21	01-Jul-21	01-Oct-21	01-Jan-21	01-Jan-22	01-Apr-22	01-Jul-22	01-Oct-22	01-Jan-22	01-Jan-23	01-Jan-24	01-Jan-25	01-Jan-26
		End of period			31-Dec-19	31-Mar-20	30-Jun-20	30-Sep-20	31-Dec-20	31-Dec-20	31-Mar-21	30-Jun-21	30-Sep-21	31-Dec-21	31-Dec-21	31-Mar-22	30-Jun-22	30-Sep-22	31-Dec-22	31-Dec-22	31-Dec-23	31-Dec-24	31-Dec-25	31-Dec-26
		Project Year	Unit	Total LOM	0.17	0.42	0.67	0.92	1.17	1.17	1.42	1.67	1.92	2.17	2.17	2.42	2.67	2.92	3.17	3.17	4.17	5.17	6.17	7.17
	CASH FLOW MODEL		US$m nominal	US$'000 nominal					Shortfall in feeding material and drop in grade															
0.0%		Gross Revenue	449	449,474	5,722	8,467	17,292	9,146	4,368	39,274	6,119	18,093	17,345	22,103	63,660	19,399	19,549	19,023	18,404	76,376	75,604	63,833	66,327	58,679
		Less Realisation Costs	81	(81,273)	(129)	(191)	(389)	(206)	(98)	(885)	(138)	(407)	(391)	(2,155)	(3,091)	(3,319)	(4,162)	(4,620)	(4,642)	(16,743)	(21,752)	(15,417)	(12,953)	(10,305)
		Net Revenue	368	368,200	5,593	8,276	16,903	8,940	4,270	38,389	5,981	17,685	16,954	19,948	60,569	16,080	15,387	14,403	13,762	59,633	53,852	48,416	53,374	48,374
		Less Operating Costs																						
0.0%		Less Mining Cost	82.3	(82,257)	(2,129)	(1,356)	(3,102)	(3,129)	(3,168)	(10,756)	(3,982)	(4,073)	(2,334)	(2,361)	(12,749)	(2,897)	(3,284)	(4,013)	(6,378)	(16,573)	(15,034)	(9,937)	(8,684)	(6,395)
0.0%		Less Plant Processing Cost	68.7	(68,703)	(1,462)	(1,980)	(2,080)	(2,104)	(2,442)	(8,606)	(3,305)	(2,244)	(2,271)	(2,626)	(10,446)	(2,292)	(2,319)	(2,347)	(2,374)	(9,333)	(9,877)	(10,074)	(10,276)	(8,629)
		Less G&A	46.7	(46,733)	(1,500)	(1,377)	(1,392)	(1,407)	(1,423)	(5,598)	(1,477)	(1,493)	(1,510)	(1,526)	(6,006)	(1,543)	(1,560)	(1,577)	(1,595)	(6,276)	(6,636)	(6,769)	(6,905)	(7,043)
		Less Mining Royalty Tax	45.0	(44,999)	(438)	(647)	(1,336)	(706)	(506)	(3,195)	(1,030)	(1,398)	(1,340)	(2,047)	(5,815)	(1,972)	(2,020)	(1,993)	(1,952)	(7,937)	(8,335)	(6,614)	(6,497)	(6,169)
		Total Operating Cost LOM	242.7	(242,691)	(5,528)	(5,360)	(7,909)	(7,347)	(7,539)	(28,155)	(9,794)	(9,208)	(7,454)	(8,560)	(35,016)	(8,704)	(9,185)	(9,931)	(12,299)	(40,118)	(39,882)	(33,395)	(32,361)	(28,236)
									Shortfall		Shortfall													
		EBITDA	125.5	125,509	64	2,916	8,994	1,593	(3,269)	10,234	(3,813)	8,477	9,500	11,389	25,553	7,375	6,203	4,473	1,463	19,514	13,970	15,022	21,013	20,138
		Less Interest Cost (Leasing)	0.6	(590)	–	(82)	(63)	(63)	(63)	(271)	(64)	(45)	(45)	(45)	(198)	(30)	(30)	(30)	(30)	(121)	–	–	–	–
		Less Depreciation & Amortisation	100.4	(100,369)	–	(9,329)	(8,579)	(8,025)	(7,334)	(33,267)	(6,708)	(6,355)	(6,903)	(6,584)	(26,550)	(6,298)	(5,833)	(5,414)	(5,035)	(22,580)	(4,694)	(4,511)	(4,480)	(4,287)
		Less Payments to Reclamation Fund	4.2	(4,207)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	(4,207)
		EBT	20.3	20,343	64	(6,495)	352	(6,495)	(10,666)	(23,303)	(10,585)	2,077	2,552	4,760	(1,196)	1,047	339	(971)	(3,602)	(3,187)	9,276	10,511	16,533	11,644
		Less Income Tax (carried forward losses considered)	8.2	(8,244)	(6)	–	(35)	–	–	(35)	–	(208)	(255)	(476)	(939)	(105)	(34)	–	–	(139)	(928)	(1,051)	(1,653)	(3,493)
		Net Income	12	12,098	58	(6,495)	317	(6,495)	(10,666)	(23,338)	(10,585)	1,869	2,297	4,284	(2,135)	942	305	(971)	(3,602)	(3,325)	8,349	9,459	14,880	8,151
		Plus Depreciation & Amortisation	100	100,369	–	9,329	8,579	8,025	7,334	33,267	6,708	6,355	6,903	6,584	26,550	6,298	5,833	5,414	5,035	22,580	4,694	4,511	4,480	4,287
		Less Increase in Net Working Capital	0	–	(438)	(739)	(2,192)	2,341	1,494	904	9	(3,834)	(25)	(560)	(4,410)	1,206	359	465	587	2,618	(231)	54	(453)	1,956
		Cash Flow from Operations	112	112,467	(380)	2,095	6,704	3,872	(1,838)	10,832	(3,868)	4,391	9,175	10,308	20,006	8,447	6,498	4,908	2,021	21,873	12,812	14,024	18,907	14,393
0.0%		Less Capital Costs, including	43.0	(43,006)	–	(1,434)	(2,746)	(748)	(751)	(5,679)	(2,995)	(12,117)	(3,544)	(3,577)	(22,232)	(1,404)	(1,417)	(1,431)	(1,444)	(5,696)	(2,769)	(4,180)	(2,450)	–
		Mining Capex for Open Pit	2.5	(2,473)	–	(294)	(298)	(301)	(305)	(1,198)	–	–	(636)	(639)	(1,275)	–	–	–	–	–	–	–	–	–
		Mining Capex for Underground	24.6	(24,595)	–	–	–	–	–	–	(2,541)	(2,569)	(2,597)	(2,626)	(10,334)	(1,196)	(1,209)	(1,222)	(1,236)	(4,862)	(2,769)	(4,180)	(2,450)	–
		Leasing Principal Repayment	4.7	(4,700)	–	(1,139)	(447)	(447)	(447)	(2,479)	(454)	(311)	(311)	(311)	(1,387)	(208)	(208)	(208)	(208)	(834)	–	–	–	–
		Processing Plant Updrade: XRT and Flotation Plant	11.2	(11,238)	–	–	(2,002)	–	–	(2,002)	–	(9,237)	–	–	(9,237)	–	–	–	–	–	–	–	–	–
		Pre Tax Cash Flow	78	77,706	(374)	661	3,993	3,124	(2,590)	5,189	(6,863)	(7,518)	5,886	7,207	(1,287)	7,148	5,115	3,477	576	16,316	10,970	10,895	18,111	17,886
		Post Tax Free Cash Flow	69	69,461	(380)	661	3,958	3,124	(2,590)	5,153	(6,863)	(7,726)	5,631	6,731	(2,226)	7,043	5,081	3,477	576	16,177	10,043	9,844	16,457	14,393
		Cumulative Project Cash Flow		281,813	(380)	281	4,239	7,363	4,773	4,773	(2,090)	(9,816)	(4,184)	2,547	2,547	9,590	14,670	18,147	18,724	18,724	28,766	38,610	55,068	69,461
					(380)					4,773					2,547					18,724	28,766	38,610	55,068	69,461
8.64%		Discount Factor			0.99	0.97	0.95	0.93	0.91	0.91	0.89	0.87	0.85	0.84	0.84	0.82	0.80	0.79	0.77	0.77	0.71	0.65	0.60	0.55
		Discounted Cash Flow	47	46,508	(375)	639	3,745	2,895	(2,351)	4,928	(6,103)	(6,729)	4,804	5,625	(2,403)	5,765	4,073	2,730	443	13,012	7,110	6,415	9,872	7,948
		Cumulative Discounted Cash Flow		211,884	(375)	264	4,009	6,904	4,553	4,553	(1,549)	(8,279)	(3,474)	2,150	2,150	7,915	11,988	14,719	15,162	15,162	22,273	28,688	38,561	46,508
					(375)					4,553					2,150					15,162	22,273	28,688	38,561	46,508
					-					0.1					1.9					0.17	2.13	3.47	2.91	4.85
10.00%		Discount Factor			0.98	0.96	0.94	0.92	0.89	0.89	0.87	0.85	0.83	0.81	0.81	0.79	0.78	0.76	0.74	0.74	0.67	0.61	0.56	0.51
		Discounted Cash Flow	44	43,874	(374)	636	3,714	2,862	(2,317)	4,895	(5,996)	(6,591)	4,691	5,475	(2,421)	5,594	3,940	2,633	426	12,594	6,751	6,016	9,143	7,270
15.00%		Discount Factor			0.98	0.94	0.91	0.88	0.85	0.85	0.82	0.79	0.77	0.74	0.74	0.71	0.69	0.67	0.64	0.64	0.56	0.49	0.42	0.37
		Discounted Cash Flow	36	35,772	(371)	624	3,606	2,748	(2,200)	4,778	(5,630)	(6,121)	4,308	4,973	(2,470)	5,024	3,500	2,313	370	11,207	5,610	4,782	6,951	5,286
20.0%		Discount Factor			0.97	0.93	0.89	0.85	0.81	0.81	0.77	0.74	0.71	0.67	0.67	0.64	0.61	0.59	0.56	0.56	0.47	0.39	0.32	0.27
		Discounted Cash Flow	30	29,605	(369)	613	3,505	2,643	(2,093)	4,667	(5,301)	(5,701)	3,970	4,535	(2,497)	4,533	3,124	2,043	324	10,024	4,698	3,838	5,347	3,897

Financial Project Summary			
NPV @ Discount Rate of 8.64%	US$ M		46.51
IRR	%		1186%
Payback period of capital (Discounted)	Years	1.00	Q3 2021
Max Cash Exposure	US$ M		0.38
NPV @ Discount Rate of 10%	US$ M		43.87
NPV @ Discount Rate of 15%	US$ M		35.77
NPV @ Discount Rate of 20%	US$ M		29.60
Ag Break-even price			14.11

(375)	4,928	(2,403)	13,012	7,110	6,415	9,872	7,948

0%	Pb Price
0%	Zn Price
0%	Operating Mining Costs (Both OP and UG)
0%	Operating Processing Costs (Average for both Plants)
0%	Capex
0%	Ag Price

Sensitivity Analysis



	Change in Pb Price																	
	Nominal Values	1,241	1,345	1,448	1,552	1,655	1,759	1,862	1,966	2,069	2,172	2,276	2,379	2,483	2,586	2,690	2,793	2,897
	Base Case	-40%	-35%	-30%	-25%	-20%	-15%	-10%	-5%	0	5%	10%	15%	20%	25%	30%	35%	40%
NPV @ 8%	46.5	34.01	35.57	37.14	38.71	40.27	41.84	43.40	44.95	46.51	48.06	49.62	51.17	52.72	54.28	55.83	57.37	58.91

	Change in Ag Price																	
g/t	minal Values	10.66	11.54	12.43	13.32	14.21	15.10	15.98	16.87	17.76	18.65	19.54	20.42	21.31	22.20	23.09	23.98	24.86
	Base Case	-40%	-35%	-30%	-25%	-20%	-15%	-10%	-5%	0	5%	10%	15%	20%	25%	30%	35%	40%
8%	46.5	(46.89)	(34.57)	(22.36)	(10.30)	1.33	12.78	24.10	35.35	46.51	57.57	68.60	79.61	90.87	102.84	112.96	123.09	133.14

	Change in Zn Price																	
	Nominal Values	1,351	1,464	1,576	1,689	1,802	1,914	2,027	2,139	2,252	2,365	2,477	2,590	2,702	2,815	2,815	2,815	2,815
	Base Case	-40%	-35%	-30%	-25%	-20%	-15%	-10%	-5%	0	5%	10%	15%	20%	25%	30%	35%	40%
8%	46.5	43.1	43.5	44.0	44.4	44.8	45.2	45.7	46.1	47	46.9	47.4	47.8	48.2	48.6	49.0	49.5	49.9

	Change in Operating Mining Costs (Both OP and UG)																	
Mining Opex ($/t ore mir		29.69	32.17	34.64	37.12	39.59	42.07	44.54	47.02	49.49	51.96	54.44	56.91	59.39	61.86	64.34	66.81	69.29
	Base Case	-40%	-35%	-30%	-25%	-20%	-15%	-10%	-5%	0	5%	10%	15%	20%	25%	30%	35%	40%
8%	46.5	69.0	66.2	63.4	60.6	57.8	55.0	52.1	49.3	47	43.7	40.9	38.0	35.2	32.3	29.5	26.6	23.7

	Change in Operating Processing Costs (Average for both Plants)																	
Proc Opex ($/t ore)		24.80	26.87	28.93	31.00	33.07	35.13	37.20	39.27	41.33	43.40	45.47	47.54	49.60	51.67	53.74	55.80	57.87
	Base Case	-40%	-35%	-30%	-25%	-20%	-15%	-10%	-5%	0	5%	10%	15%	20%	25%	30%	35%	40%
8%	46.5	64.6	62.3	60.1	57.8	55.5	53.3	51.0	48.8	47	44.2	42.0	39.7	37.4	35.2	32.9	30.6	28.3

	Change in Capex																	
Capex (US$M)		25.8	28.0	30.1	32.3	34.4	36.6	38.7	40.9	43.0	45.2	47.3	49.5	51.6	53.8	55.9	58.1	60.2
	Base Case	-40%	-35%	-30%	-25%	-20%	-15%	-10%	-5%	0	5%	10%	15%	20%	25%	30%	35%	40%
8%	46.5	60.6	58.8	57.1	55.3	53.6	51.8	50.0	48.3	47	44.7	43.0	41.2	39.5	37.7	35.9	34.2	32.4

	60%	75%	90%	100%	110%	125%	140%
Pb Price	1,241	1,552	1,862	2,069	2,276	2,586	2,897
NPV @ 8.64%	29.56	33.96	38.36	41.30	44.23	48.63	53.01
Zn Price	1,351	1,689	2,027	2,252	2,477	2,815	2,815
NPV @ 8.64%	43.12	44.39	45.66	46.51	47.35	48.62	49.89
Operating Mining Costs (Both OP and UG)	29.69	37.12	44.54	49.49	54.44	61.86	69.29
NPV @ 8.64%	68.98	60.58	52.14	46.51	40.86	32.31	23.73
Operating Processing Costs (Average for both Plants)	24.80	31.00	37.20	41.33	45.47	51.67	57.87
NPV @ 8.64%	64.58	57.80	51.02	46.51	41.98	35.15	28.30
Capex (US$ M, nominal)	25.80	32.25	38.71	43.01	47.31	53.76	60.21
NPV @ 8.64%	60.61	55.32	50.03	46.51	42.98	37.69	32.40
Ag Price	10.66	13.32	15.98	17.76	19.54	22.20	24.86
NPV @ 8.64%	- 46.89	- 10.30	24.10	46.51	68.60	102.84	133.14

YEAR			2019	2020					2021					2022					2023	2024	2025	2026
PERIOD START				01-Jan-20	01-Apr-20	01-Jul-20	01-Oct-20		01-Jan-21	01-Apr-21	01-Jul-21	01-Oct-21		01-Jan-22	01-Apr-22	01-Jul-22	01-Oct-22					
PROJECT PERIOD			19 Q4	20 Q1	20 Q2	20 Q3	20 Q4	Y20	21 Q1	21 Q2	21 Q3	21 Q4	Y21	22 Q1	22 Q2	22 Q3	22 Q4	Y22	Y23	Y24	Y25	Y26
1. PRODUCTION SCHEDULE																						
1.1 Mining Physicals																						
Vertiklany Open Pit																						
Mineralised Material	t	402,843	23,640	30,717	48,836	51,169	44,893	175,615	36,024	61,350	83,413	22,801	203,588	-	-	-	-	-	-	-	-	-
Waste Material	t	10,995,762	382,943	703,343	1,610,736	1,603,752	1,611,106	5,528,938	1,988,975	1,986,149	846,328	262,430	5,083,882	-	-	-	-	-	-	-	-	-
Mangazeisky North Open Pit																						
Mineralised Material	t	418,996	-	-	-	-	-	-	-	-	8,559	29,345	37,904	52,148	50,147	68,340	73,335	243,970	137,121	-	-	-
Waste Material	t	8,543,326	-	-	-	-	-	-	-	-	221,441	844,655	1,066,096	1,162,851	1,178,353	1,173,660	1,168,664	4,683,528	2,793,702	-	-	-
Vertiklany Underground Mining																						
Mineralised Material	t	840,265	-	-	-	-	-	-	-	-	-	-	-	-	1,241	3,756	12,515	17,512	132,583	254,121	273,121	162,929
Vertiklany Underground Development																						
Decline	m	7,411	-	-	-	-	-	-	-	-	-	-	-	-	269	638	580	1,487	2,192	2,343	1,389	-
Level Access	m	9,982	-	-	-	-	-	-	-	-	-	-	-	-	153	190	576	919	3,650	3,532	1,784	97
Vent Connection	m	1,061	-	-	-	-	-	-	-	-	-	-	-	-	13	91	72	175	261	450	175	-
1.2 Ore Sorter Feed																						
Leach Plant (Current)																						
Oxide	t	302,594	20,039	29,894	45,500	45,500	37,418	158,311	12,402	45,500	45,500	20,841	124,243	-	-	-	-	-	-	-	-	-
Ag	g/t		588	581	783	412	177	495	393	803	766	716	734	-	-	-	-	-	-	-	-	-
Sulphide	t	41,931	-	-	-	-	8,833	8,833	33,098	-	-	-	33,098	-	-	-	-	-	-	-	-	-
Ag	g/t		-	-	-	-	762	762	671	-	-	-	671	-	-	-	-	-	-	-	-	-
Oxide + Sulphide	t	344,525	20,039	29,894	45,500	45,500	46,251	167,145	45,500	45,500	45,500	20,841	157,341	-	-	-	-	-	-	-	-	-
Flotation Plant																						
Sulphide	t	1,362,739	-	-	-	-	-	-	-	-	-	47,340	47,340	68,180	68,180	68,180	68,180	272,720	272,720	272,720	272,720	224,519
Ag	g/t		-	-	-	-	-	-	-	-	-	641	641	564	538	502	475	520	451	394	428	460
Pb	%		-	-	-	-	-	-	-	-	-	2	2	3	4	5	5	4	5	2	2	3
Zn	%		-	-	-	-	-	-	-	-	-	1	1	1	1	1	1	1	2	2	1	1
1.3 Process Plant Feed																						
Leach Plant (Current)																						
Oxide	t	216,689	20,039	29,894	30,030	30,030	24,696	114,650	8,185	30,030	30,030	13,755	82,001	-	-	-	-	-	-	-	-	-
Ag	g/t		588	581	1,175	618	266	678	589	1,205	1,150	1,074	1,101	-	-	-	-	-	-	-	-	-
Sulphide	t	27,675	-	-	-	-	5,830	5,830	21,845	-	-	-	21,845	-	-	-	-	-	-	-	-	-
Ag	g/t		-	-	-	-	1,143	1,143	1,007	-	-	-	1,007	-	-	-	-	-	-	-	-	-
Oxide + Sulphide	t	244,364	20,039	29,894	30,030	30,030	30,526	120,479	30,030	30,030	30,030	13,755	103,845	-	-	-	-	-	-	-	-	-
Flotation Plant (Available in mid 2021)																						
Sulphide	t	899,408	-	-	-	-	-	-	-	-	-	31,244	31,244	44,999	44,999	44,999	44,999	179,995	179,995	179,995	179,995	148,182
Ag	g/t		-	-	-	-	-	-	-	-	-	961	961	846	807	752	713	780	676	591	641	690
Pb	%		-	-	-	-	-	-	-	-	-	3	3	5	6	7	7	6	8	5	4	3
Zn	%		-	-	-	-	-	-	-	-	-	2	2	1	1	1	1	1	2	3	3	3
3. MINING COSTS																						
3.1 Open Pit Operating Costs																						
Provided costs as WAI has incorpor Drilling	\$/t		0.50					0.40					0.40					0.37	0.36	-	-	-
Blasting	\$/t		0.45					0.46					0.46					0.46	0.46	-	-	-
Dozing & Grading	\$/t		0.51					0.12					0.11					0.14	0.07	-	-	-
Loading & Stockpiling	\$/t		1.22					0.32					0.31					0.36	0.28	-	-	-
Hauling	\$/t		0.81					0.51					0.54					0.74	0.72	-	-	-
Conveyor	\$/t		-					-					-					-	-	-	-	-
Engineering/Geology	\$/t		0.63					0.05					0.04					0.05	0.00	-	-	-
General Mine Maintenance	\$/t		0.52					0.13					0.11					0.14	0.04	-	-	-
Supervision & Technical	\$/t		0.56					0.04					0.04					0.05	0.02	-	-	-
Other	\$/t		0.10					0.04					0.04					0.05	0.02	-	-	-
Pumping	\$/t		-					-					-					-	-	-	-	-
99% **Model 1**	**US\$/t$_{moved}$**		**5.24**	2.03	2.03	2.03	2.03	**2.03**	2.01	2.01	2.01	2.01	**2.01**	2.32	2.32	2.32	2.32	**2.32**	**1.94**	**-**	**-**	**-**
Total	**\$/t ore**		**91.32**					**67.01**					**53.84**					**47.55**	**41.95**	**-**	**-**	**-**
Total	**\$/t Hard Rock Waste**		**5.64**					**2.13**					**2.11**					**2.48**	**2.06**	**-**	**-**	**-**
Total Moved Tonnes	t	20,360,927	406,582	734,059	1,659,573	1,654,921	1,655,999	5,704,553	2,024,999	2,047,499	1,159,741	1,159,231	6,391,469	1,215,000	1,228,500	1,242,000	1,242,000	4,927,499	2,930,824	-	-	-
Open Pit Operating Costs	**US\$**	43,671,143	2,128,880	1,493,364	3,376,221	3,366,758	3,368,951	11,605,294	4,062,702	4,107,843	2,326,758	2,325,734	12,823,037	2,821,113	2,852,458	2,883,804	2,883,804	11,441,180	5,672,752	-	-	-
3.2 Underground Operaitng Costs																						
Operating Development	\$/t ore		-					-					-	15.38	15.38	15.38	15.38	15.38	11.28	6.85	4.43	2.82
Operating Expenditure	\$/t ore		-					-					-	184.96	184.96	184.96	184.96	184.96	31.99	18.63	15.48	20.15
Personnel Salaries	\$/t ore		-					-					-	48.40	48.40	48.40	48.40	48.40	16.46	9.18	7.71	10.47
Model 1 (Fully owned/operated)	**US\$/t$_{ore}$**		-	-	-	-	-	-	-	-	-	-	-	248.74	248.74	248.74	248.74	248.74	59.73	34.66	27.63	33.44
Total Mineralised Tonnes	t	840,265	-	-	-	-	-	-	-	-	-	-	-	-	1,241	3,756	12,515	17,512	132,583	254,121	273,121	162,929
Underground Operating Costs	**US\$**	34,078,098	-	-	-	-	-	-	-	-	-	-	-	-	308,751	934,196	3,112,906	4,355,854	7,919,597	8,808,087	7,546,257	5,448,303
3.3 Total Operating Costs																						
Open Pit Operating Costs	US\$	43,671,143	2,128,880	1,493,364	3,376,221	3,366,758	3,368,951	11,605,294	4,062,702	4,107,843	2,326,758	2,325,734	12,823,037	2,821,113	2,852,458	2,883,804	2,883,804	11,441,180	5,672,752	-	-	-
		-																				
Underground Operating Costs	US\$	34,078,098	-	-	-	-	-	-	-	-	-	-	-	-	308,751	934,196	3,112,906	4,355,854	7,919,597	8,808,087	7,546,257	5,448,303
Total Operating Costs	**US\$**	77,749,241	2,128,880	1,493,364	3,376,221	3,366,758	3,368,951	11,605,294	4,062,702	4,107,843	2,326,758	2,325,734	12,823,037	2,821,113	3,161,210	3,818,000	5,996,711	15,797,034	13,592,349	8,808,087	7,546,257	5,448,303
3.5 Underground Capital Cost Schedule																						
3.6 Total Capital Expenditure																						
CAPEX Open Pit	US\$	2,530,102	-	324,068	324,068	324,068	324,068	1,296,272	-	-	616,915	616,915	1,233,831	-	-	-	-	-	-	-	-	-
CAPEX Underground	US\$	23,327,643	-	-	-	-	-	-	2,580,949	2,580,949	2,580,949	2,580,949	10,323,797	1,162,110	1,162,110	1,162,110	1,162,110	4,648,439	2,508,900	3,713,329	2,133,176	-
## CAPEX Total	US\$	25,857,745	-	324,068	324,068	324,068	324,068	1,296,272	2,580,949	2,580,949	3,197,865	3,197,865	11,557,628	1,162,110	1,162,110	1,162,110	1,162,110	4,648,439	2,508,900	3,713,329	2,133,176	-

OPEN PIT CAPITAL COST SCHEDULE

ROAD CONSTRUCTION																						
Vertikalny Cut & Fill Road	US$		-					575,356					-					-	-	-	-	-
Mangazeisky North Cut & Fill Road	US$		-					598,065					-					-	-	-	-	-
Mangazeisky North Connecting Road	US$		-					122,850					-					-	-	-	-	-
TOTAL	**US$**	**1,296,272**	**-**	**324,068**	**324,068**	**324,068**	**324,068**	**1,296,272**					**-**					**-**	**-**	**-**	**-**	**-**
EQUIPMENT OVERHAUL																						
Overhaul Schedule																						
Production Drill	units		-					-					2					-	-	-	-	-
Excavator Primary (Waste)	units		-					-					1					-	-	-	-	-
Excavator Secondary (Ore)	units		-					-					1					-	-	-	-	-
Haul Trucks	units		-					-					6					-	-	-	-	-
Overhaul Cost																						
Production Drill	US$		-					-					352,991					-	-	-	-	-
Excavator Primary (Waste)	US$		-					-					222,480					-	-	-	-	-
Excavator Secondary (Ore)	US$		-					-					157,960					-	-	-	-	-
Haul Trucks	US$		-					-					500,400					-	-	-	-	-
Total Overhaul Cost	**US$**	**1,233,831**	**-**	-	-	-	-	**-**	-	-	616,915	616,915	**1,233,831**					**-**	**-**	**-**	**-**	**-**

UNDERGROUND CAPITAL COST SCHEDULE

Advance Costs	
472	US$/m
26	US$/m
432	US$/m
432	US$/m
694	US$/m

UG CAPITAL DEVELOPMENT		**23,125,215**																				
Development Meterage																						
Decline	m		-					-					-					1,487	2,192	2,343	1,389	-
Ventilation Raise	m		-					-					-					175	261	450	175	-
Level Access	m		-					-					-					193	328	395	293	-
Ventilation Connection	m		-					-					-					69	75	79	51	-
Remuck Bay	m		-					-					-					36	55	74	44	-
Development Costs																						
Decline	US$		-					-					-					701,870	1,034,684	1,106,296	655,807	-
Ventilation Raise	US$		-					-					-					4,570	6,803	11,713	4,560	-
Level Access	US$		-					-					-					83,291	141,764	170,599	126,669	-
Ventilation Connection	US$		-					-					-					29,722	32,283	34,070	21,862	-
Remuck Bay	US$		-					-					-					24,986	37,867	51,251	30,878	-
Total Development Cost	**US$**	**4,311,545**	**-**					**-**					**-**	211,110	211,110	211,110	211,110	**844,439**	**1,253,400**	**1,373,929**	**839,776**	**-**
UG EQUIPMENT PURCHASE																						
Pruchase Schedule																						
Development Jumbo	units		-					-					2					2	-	-	-	-
Production Drill	units		-					-					-					1	-	1	-	-
Load Haul Dump	units		-					-					2					1	1	-	-	-
Underground Truck	units		-					-					2					1	1	-	-	-
Explosive truck	units		-					-					1					-	-	-	-	-
Motor grader	units		-					-					1					-	-	-	-	-
Fuel & lube truck	units		-					-					1					-	-	-	-	-
Scissor lift	units		-					-					1					-	-	-	-	-
Underground 4x4	units		-					-					6					-	-	-	-	-
Water truck (for dust suppression)	units		-					-					1					-	-	-	-	-
Primary Fan	units		-					-					4					-	-	-	-	-
Secondary Fans & Starters	units		-					-					16					-	-	-	-	-
Compressors	units		-					-					4					-	-	-	-	-
Main Pump	units		-					-					4					-	-	-	-	-
Face Pump	units		-					-					9					14	4	-	-	-
Jumbo Boxes	units		-					-					9.0					14	4	-	-	-
Purchase Cost																						
Development Jumbo	US$		-					-					1,126,000					1,126,000	-	-	-	-
Production Drill	US$		-					-					-					1,015,000	-	1,015,000	-	-
Load Haul Dump	US$		-					-					745,000					372,500	372,500	-	-	-
Underground Truck	US$		-					-					1,440,000					720,000	720,000	-	-	-
Explosive truck	US$		-					-					576,000					-	-	-	-	-
Motor grader	US$		-					-					287,500					-	-	-	-	-
Fuel & lube truck	US$		-					-					576,000					-	-	-	-	-
Scissor lift	US$		-					-					350,200					-	-	-	-	-
Underground 4x4	US$		-					-					286,320					-	-	-	-	-
Water truck (for dust suppression)	US$		-					-					576,000					-	-	-	-	-
Primary Fan	US$		-					-					3,000,000					-	-	-	-	-
Secondary Fans & Starters	US$		-					-					377,600					-	-	-	-	-
Compressors	US$		-					-					197,600					-	-	-	-	-
Main Pump	US$		-					-					216,400					-	-	-	-	-
Face Pump	US$		-					-					20,250					31,500	9,000	-	-	-
Jumbo Boxes	US$		-					-					346,500					539,000	154,000	-	-	-
Total Purchase Cost	**US$**	**16,195,870**	**-**					**-**	2,530,343	2,530,343	2,530,343	2,530,343	**10,121,370**	951,000	951,000	951,000	951,000	**3,804,000**	**1,255,500**	**1,015,000**	**-**	**-**
First Fill & Initial Spares (2% of Pre Prod CAPEX)		**202,427**	**-**	-	-	-	-	**-**	50,607	50,607	50,607	50,607	**202,427**	-	-	-	-	**-**	**-**	**-**	**-**	**-**
EQUIPMENT OVERHAUL																						
Overhaul Schedule																						
Development Jumbo	unit		-					-					-					-	-	2	2	-
Production Drill	unit		-					-					-					-	-	-	1	-
Load Haul Dump	unit		-					-					-					-	-	2	1	-
Underground Truck	unit		-					-					-					-	-	2	1	-
Overhaul Cost																						
Development Jumbo	US$		-					-					-					-	-	450,400	450,400	-
Production Drill	US$		-					-					-					-	-	-	406,000	-
Load Haul Dump	US$		-					-					-					-	-	298,000	149,000	-
Underground Truck	US$		-					-					-					-	-	576,000	288,000	-
Total Overhaul Cost	**US$**	**2,617,800**	**-**					**-**					**-**					**-**	**-**	**1,324,400**	**1,293,400**	**-**

YEAR		TOTAL	2019	2020					2021					2022	2023	2024	2025	2026
PERIOD START			01-Nov-19	01-Jan-20	01-Apr-20	01-Jul-20	01-Oct-20	01-Jan-20	01-Jan-21	01-Apr-21	01-Jul-21	01-Oct-21	01-Jan-21	01-Jan-22	01-Jan-23	01-Jan-24	01-Jan-25	01-Jan-26
PROJECT PERIOD			19 Q4	20 Q1	20 Q2	20 Q3	20 Q4	Y20	21 Q1	21 Q2	21 Q3	21 Q4	Y21	Y22	Y23	Y24	Y25	Y26

1. VERTIKALNY OP

Oxide (NSR>=117 US$/t)	t	212,438	15,939	23,213	38,184	20,092	6,100	87,589	8,203	45,352	50,961	4,395	108,910	-	-	-	-	-
Ag	g/t	800	716	714	913	778	331	789	541	912	811	527	821	-	-	-	-	-
Oxide (NSR<117 US$/t)	t	44,996	4,100	6,682	6,658	11,434	790	25,563	4,199	7,032	4,102	-	15,333	-	-	-	-	-
Ag	g/t	104	92	116	101	91	89	100	103	103	144	-	114	-	-	-	-	-
Sulphide (NSR>=113.06 US$/t)	t	116,362	3,451	822	2,845	14,495	29,017	47,179	16,824	7,298	24,333	17,276	65,732	-	-	-	-	-
Ag	g/t	846	814	802	2,328	1,758	430	959	586	413	1,105	617	767	-	-	-	-	-
Pb	%	1.70	0.95	0.64	2.34	1.91	1.48	1.65	1.55	1.94	1.67	2.12	1.79	-	-	-	-	-
Zn	%	1.66	2.37	2.03	0.92	1.46	1.49	1.46	1.21	2.80	1.60	2.08	1.76	-	-	-	-	-
Sulphide (NSR<113.06 US$/t)	t	29,047	150	-	1,150	5,148	8,987	15,285	6,797	1,668	4,018	1,130	13,613	-	-	-	-	-
Ag	g/t	131	136	-	63	154	153	147	119	126	107	93	114	-	-	-	-	-
Pb	%	0.98	0.32	-	0.21	0.65	0.97	0.81	0.85	3.04	0.83	1.82	1.19	-	-	-	-	-
Zn	%	1.36	0.34	-	1.18	1.69	0.66	1.04	1.27	3.25	2.02	1.10	1.72	-	-	-	-	-
Total Mineralised Material	t	402,843	23,640	30,717	48,836	51,169	44,893	175,615	36,024	61,350	83,413	22,801	203,588	-	-	-	-	-
Waste	t	10,995,762	382,943	703,343	1,610,736	1,603,752	1,611,106	5,528,938	1,988,975	1,986,149	846,328	262,430	5,083,882	-	-	-	-	-
		-																

2. MANGAZEISKY NORTH OP

Sulphide (NSR>=113.06 US$/t)	t	346,794	-	-	-	-	-	-	-	-	5,600	26,526	32,126	199,371	115,297	-	-	-
Ag	g/t	570	-	-	-	-	-	-	-	-	408	527	507	554	617	-	-	-
Pb	%	7.47	-	-	-	-	-	-	-	-	3.26	5.18	4.84	6.35	10.16	-	-	-
Zn	%	0.82	-	-	-	-	-	-	-	-	0.05	0.09	0.08	0.40	1.75	-	-	-
Sulphide (NSR<113.06 US$/t)	t	72,201.61	-	-	-	-	-	-	-	-	2,959	2,819	5,778	44,599	21,824	-	-	-
Ag	g/t	129	-	-	-	-	-	-	-	-	194	125	161	125	128	-	-	-
Pb	%	1.38	-	-	-	-	-	-	-	-	0.79	0.30	0.55	1.51	1.33	-	-	-
Zn	%	0.37	-	-	-	-	-	-	-	-	0.02	0.01	0.01	0.16	0.90	-	-	-
Total Mineralised Material	t	418,996	-	-	-	-	-	-	-	-	8,559	29,345	37,904	243,970	137,121	-	-	-
Waste	t	8,543,326	-	-	-	-	-	-	-	-	221,441	844,655	1,066,096	4,683,528	2,793,702	-	-	-

3. VERTIKALNY UG

Waste Development	t	284,155	-	-	-	-	-	-	-	-	-	-	-	55,390	81,247	92,781	54,738	-
Development Mineralised Material	t	231,658	-	-	-	-	-	-	-	-	-	-	-	17,512	89,320	82,009	40,223	2,594
Ag	g/t	263	-	-	-	-	-	-	-	-	-	-	-	281	269	231	306	239
Pb	%	1.37	-	-	-	-	-	-	-	-	-	-	-	1.34	1.17	1.35	1.88	1.13
Zn	%	1.26	-	-	-	-	-	-	-	-	-	-	-	2.35	1.53	0.84	1.07	0.72
Stope Mineralised Tonnes	t	608,607	-	-	-	-	-	-	-	-	-	-	-	-	43,263	172,111	232,897	160,335
Ag	g/t	462	-	-	-	-	-	-	-	-	-	-	-	-	457	452	466	468
Pb	%	2.16	-	-	-	-	-	-	-	-	-	-	-	-	2.39	1.65	1.51	3.60
Zn	%	1.68	-	-	-	-	-	-	-	-	-	-	-	-	2.95	2.50	1.35	0.92
Total Mineralised Tonnes	t	840,265	-	-	-	-	-	-	-	-	-	-	-	17,512	132,583	254,121	273,121	162,929
Ag	g/t	407	-	-	-	-	-	-	-	-	-	-	-	281	331	381	442	465
Pb	%	1.95	-	-	-	-	-	-	-	-	-	-	-	1.34	1.57	1.56	1.57	3.56
Zn	%	1.56	-	-	-	-	-	-	-	-	-	-	-	2.35	1.99	1.97	1.31	0.92
Inclined	m	7,411	-	-	-	-	-	-	-	-	-	-	-	1,487	2,192	2,343	1,389	-
Horizontal	m	9,982	-	-	-	-	-	-	-	-	-	-	-	919	3,650	3,532	1,784	97
Vertical	m	1,061	-	-	-	-	-	-	-	-	-	-	-	175	261	450	175	-

4. STOCKPILES

OFF-BALANCE OXIDE																		
Open Balance																		
Mass	t				45,160	44,502	30,528	45,160	-	-	-	-	-	-	-	-	-	-
Ag	g/t				149	149	149	149	-	-	-	-	-	-	-	-	-	-
Ag Contained	kg				6,709	6,612	4,536	6,709	-	-	-	-	-	-	-	-	-	-
Input																		
Mass	t				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag	g/t				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag Contained	kg				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Output (Ore Sorter Leach Plant Stream)																		
Mass	t				658	13,974	30,528	45,160.00	-	-	-	-	-	-	-	-	-	-
Ag	g/t				149	149	149	148.57	-	-	-	-	-	-	-	-	-	-
Ag Contained	kg				98	2,076	4,536	6,709.42	-	-	-	-	-	-	-	-	-	-
Closing Balance																		
Mass	t				44,502	30,528	-	-	-	-	-	-	-	-	-	-	-	-
Ag	g/t				149	149	-	-	-	-	-	-	-	-	-	-	-	-
Ag Contained	kg				6,612	4,536	-	-	-	-	-	-	-	-	-	-	-	-
ROM OXIDE																		
Open Balance																		
Mass	t		-	-	-	-	-	-	-	-	6,884	16,446	-	-	-	-	-	-
Ag	g/t		-	-	-	-	-	-	-	-	803	766	-	-	-	-	-	-
Ag Contained	kg		-	-	-	-	-	-	-	-	5,530	12,603	-	-	-	-	-	-
Input																		
Mass	t		20,039	29,894	44,842	31,526	6,890	113,151	12,402	52,384	55,062	4,395	124,243	-	-	-	-	-
Ag	g/t		588	581	792	529	303	633	393	803	762	527	734	-	-	-	-	-
Ag Contained	kg		11,789	17,359	35,536	16,678	2,089	71,663	4,871	42,082	41,942	2,318	91,213	-	-	-	-	-
Output (Ore Sorter Leach Plant Stream)																		
Mass	t		20,039	29,894	44,842	31,526	6,890	113,151	12,402	45,500	45,500	20,841	124,243	-	-	-	-	-
Ag	g/t		588	581	792	529	303	633.34	393	803	766	716	734.15	-	-	-	-	-
Ag Contained	kg		11,789	17,359	35,536	16,678	2,089	71,663	4,871	36,552	34,869	14,921	91,212.69	-	-	-	-	-
Closing Balance																		
Mass	t		-	-	-	-	-	-	-	6,884	16,446	-	-	-	-	-	-	-
Ag	g/t		-	-	-	-	-	-	-	803	766	-	-	-	-	-	-	-
Ag Contained	kg		-	-	-	-	-	-	-	5,530	12,603	-	-	-	-	-	-	-
ROM SULPHIDE																		
Open Balance																		
Mass	t		-	3,601	4,423	8,418	28,061	3,601	57,231	47,755	56,721	93,631	57,231	94,042	82,804	79,788	61,189	61,590
Ag	g/t		-	786.15	789	1,210	1,300	786.15	762	671	622	699	762	641	475	429	382	449
Ag Contained	kg		-	2,831	3,490	10,186	36,467	2,831	43,600	32,050	35,273	65,461	43,600	60,242	39,352	34,210	23,397	27,633
Pb	%		-	0.92	0.87	1.27	1.49	0.92	1.41	1.39	1.51	1.61	1.41	2.31	4.84	4.50	1.76	1.66
Pb Contained	kg		-	33,094	38,349	107,215	418,098	33,094	808,571	665,186	857,727	1,503,581	808,571	2,176,204	4,003,841	3,591,118	1,078,340	1,021,249
Zn	%		-	2.29	2.24	1.65	1.56	2.29	1.41	1.35	1.60	1.47	1.41	1.26	0.74	1.71	1.97	1.25
Zn Contained	kg		-	82,384	99,088	138,929	437,262	82,384	805,526	646,619	905,506	1,379,671	805,526	1,180,852	612,623	1,365,812	1,207,886	771,311
Input																		
Mass	t		3,601	822	3,995	19,643	38,003	62,464	23,622	8,966	36,910	47,751	117,249	261,482	269,704	254,121	273,121	162,929
Ag	g/t		786	802	1,676	1,338	365	760	451	359	818	526	590	462	437	381	442	465
Ag Contained	kg		2,831	659	6,696	26,281	13,862	47,499	10,664	3,223	30,188	25,106	69,181	120,845	117,775	96,722	120,836	75,690
Pb	%		0.92	0.64	1.72	1.58	1.36	1.44	1.34	2.15	1.75	3.70	2.49	5.19	5.22	1.56	1.57	3.56
Pb Contained	kg		33,094	5,255	68,866	310,883	515,270	900,274	317,647	192,541	645,854	1,768,112	2,924,154	13,561,411	14,081,747	3,956,118	4,283,368	5,793,664
Zn	%		2.29	2.03	1.00	1.52	1.30	1.36	1.22	2.89	1.28	0.83	1.21	0.49	1.80	1.97	1.31	0.92
Zn Contained	kg		82,384	16,704	39,841	298,333	492,590	847,469	289,258	258,886	474,165	395,616	1,417,925	1,278,219	4,853,477	4,995,623	3,575,360	1,491,303
Output (Ore Sorter Leach Plant Stream)																		
Mass	t		-	-	-	-	8,833	8,833	33,098	-	-	-	33,098	-	-	-	-	-
Ag	g/t		-	-	-	-	762	762	671	-	-	-	671	-	-	-	-	-
Ag Contained	kg		-	-	-	-	6,729	6,729	22,214	-	-	-	22,214	-	-	-	-	-
Pb	%		-	-	-	-	1.41	1.41	1.39	-	-	-	1.39	-	-	-	-	-
Pb Contained	kg		-	-	-	-	124,797	124,797	461,032	-	-	-	461,032	-	-	-	-	-
Zn	%		-	-	-	-	1.41	1.41	1.35	-	-	-	1.35	-	-	-	-	-
Zn Contained	kg		-	-	-	-	124,327	124,327	448,164	-	-	-	448,164	-	-	-	-	-
Output (Ore Sorter Flotation Plant Stream)																		
Mass	t		-	-	-	-	-	-	-	-	-	47,340	47,340	272,720	272,720	272,720	272,720	224,519
Ag	g/t		-	-	-	-	-	-	-	-	-	641	641	520	451	394	428	460
Ag Contained	kg		-	-	-	-	-	-	-	-	-	30,325	30,325	141,735	122,917	107,535	116,601	103,322
Pb	%		-	-	-	-	-	-	-	-	-	2.31	2.31	4.30	5.31	2.37	1.59	3.04

Pb Contained	kg		-	-	-	-	-	-	-	-	-	1,095,489	**1,095,489**	**11,733,774**	**14,494,470**	**6,468,896**	**4,340,458**	**6,814,913**
Zn	%		-	-	-	-	-	-	-	-	-	1.26	**1.26**	**0.68**	**1.50**	**1.89**	**1.47**	**1.01**
Zn Contained	kg		-	-	-	-	-	-	-	-	-	594,435	**594,435**	**1,846,449**	**4,100,288**	**5,153,549**	**4,011,934**	**2,262,614**
Closing Balance																		
Mass	t		**3,601**	4,423	8,418	28,061	57,231	**57,231**	47,755	56,721	93,631	94,042	**94,042**	**82,804**	**79,788**	**61,189**	**61,590**	-
Ag	g/t		**786**	789	1,210	1,300	762	**762**	671	622	699	641	**641**	**475**	**429**	**382**	**449**	-
Ag Contained	kg		**2,831**	3,490	10,186	36,467	43,600	**43,600**	32,050	35,273	65,461	60,242	**60,242**	**39,352**	**34,210**	**23,397**	**27,633**	-
Pb	%		**0.92**	0.87	1.27	1.49	1.41	**1.41**	1.39	1.51	1.61	2.31	**2.31**	**4.84**	**4.50**	**1.76**	**1.66**	-
Pb Contained	kg		**33,094**	38,349	107,215	418,098	808,571	**808,571**	665,186	857,727	1,503,581	2,176,204	**2,176,204**	**4,003,841**	**3,591,118**	**1,078,340**	**1,021,249**	-
Zn	%		**2.29**	2.24	1.65	1.56	1.41	**1.41**	1.35	1.60	1.47	1.26	**1.26**	**0.74**	**1.71**	**1.97**	**1.25**	-
Zn Contained	kg		**82,384**	99,088	138,929	437,262	805,526	**805,526**	646,619	905,506	1,379,671	1,180,852	**1,180,852**	**612,623**	**1,365,812**	**1,207,886**	**771,311**	-

5. ORE SORTER FEED

LEACH PLANT STREAM																		
Off Balance Oxide																		
Mass	t	**45,160**	-	-	658	13,974	30,528	**45,160**	-	-	-	-	-	-	-	-	-	-
Ag	g/t	**149**	-	-	149	149	149	**149**	-	-	-	-	-	-	-	-	-	-
Ag Contained	kg	**6,709**	-	-	98	2,076	4,536	**6,709**	-	-	-	-	-	-	-	-	-	-
ROM Oxide																		
Mass	t	**257,434**	**20,039**	29,894	44,842	31,526	6,890	**113,151**	12,402	45,500	45,500	20,841	**124,243**	-	-	-	-	-
Ag	g/t	**678**	**588**	581	792	529	303	**633**	393	803	766	716	**734**	-	-	-	-	-
Ag Contained	kg	**174,665**	**11,789**	17,359	35,536	16,678	2,089	**71,663**	4,871	36,552	34,869	14,921	**91,213**	-	-	-	-	-
Total Oxide																		
Mass	t	**302,594**	**20,039**	29,894	45,500	45,500	37,418	**158,311**	12,402	45,500	45,500	20,841	**124,243**	-	-	-	-	-
Ag	g/t	**599**	**588**	581	783	412	177	**495**	393	803	766	716	**734**	-	-	-	-	-
Ag Contained	kg	**181,374**	**11,789**	17,359	35,634	18,754	6,625	**78,373**	4,871	36,552	34,869	14,921	**91,213**	-	-	-	-	-
ROM Sulphide																		
Mass	t	**41,931**	-	-	-	-	8,833	**8,833**	33,098	-	-	-	**33,098**	-	-	-	-	-
Ag	g/t	**690**	-	-	-	-	762	**762**	671	-	-	-	**671**	-	-	-	-	-
Ag Contained	kg	**28,943**	-	-	-	-	6,729	**6,729**	22,214	-	-	-	**22,214**	-	-	-	-	-
Total Feed																		
Mass	t	**344,525**	**20,039**	29,894	45,500	45,500	46,251	**167,145**	45,500	45,500	45,500	20,841	**157,341**	-	-	-	-	-
Ag	g/t	**610**	**588**	581	783	412	289	**509**	595	803	766	716	**721**	-	-	-	-	-
Ag Contained	kg	**210,317**	**11,789**	17,359	35,634	18,754	13,354	**85,102**	27,084	36,552	34,869	14,921	**113,426**	-	-	-	-	-
Sulphides in Blend	%	**12.2%**	**0.0%**	0.0%	0.0%	0.0%	19.1%	**5.3%**	72.7%	0.0%	0.0%	0.0%	**21.0%**	0.0%	0.0%	0.0%	0.0%	0.0%
FLOTATION PLANT STREAM																		
ROM Sulphide																		
Mass	t	**1,362,739**	-	-	-	-	-	-	-	-	-	47,340	**47,340**	**272,720**	**272,720**	**272,720**	**272,720**	**224,519**
Ag	g/t	**457**	-	-	-	-	-	-	-	-	-	641	**641**	**520**	**451**	**394**	**428**	**460**
Ag Contained	kg	**622,435**	-	-	-	-	-	-	-	-	-	30,325	**30,325**	**141,735**	**122,917**	**107,535**	**116,601**	**103,322**
Pb	%	**72,213**	-	-	-	-	-	-	-	-	-	2.31	**2.31**	**4.30**	**5.31**	**2.37**	**1.59**	**3.04**
Pb Contained	kg	**44,948,000**	-	-	-	-	-	-	-	-	-	1,095,489	**1,095,489**	**11,733,774**	**14,494,470**	**6,468,896**	**4,340,458**	**6,814,913**
Zn	%	**400**	-	-	-	-	-	-	-	-	-	1.26	**1.26**	**0.68**	**1.50**	**1.89**	**1.47**	**1.01**
Zn Contained	kg	**17,969,268**	-	-	-	-	-	-	-	-	-	594,435	**594,435**	**1,846,449**	**4,100,288**	**5,153,549**	**4,011,934**	**2,262,614**

Mass Recovery	0.66	NO ORE SORTER	NO ORE SORTER	0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66	0.66
Ag Recovery	0.99			0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99
Pb Recovery	0.99			0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99
Zn Recovery	0.99			0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99

6. PROCESS PLANT FEED

LEACH PLANT STREAM																		
Off Balance Oxide																		
Mass	t	**29,806**	-	-	434	9,223	20,148	**29,806**	-	-	-	-	-	-	-	-	-	-
Ag	g/t	**223**	-	-	223	223	223	**223**	-	-	-	-	-	-	-	-	-	-
Ag Contained	kg	**6,642**	-	-	97	2,055	4,490	**6,642**	-	-	-	-	-	-	-	-	-	-
ROM Oxide																		
Mass	t	**186,884**	**20,039**	29,894	29,595	20,807	4,547	**84,844**	8,185	30,030	30,030	13,755	**82,001**	-	-	-	-	-
Ag	g/t	**927**	**588**	581	1,189	794	455	**838**	589	1,205	1,150	1,074	**1,101**	-	-	-	-	-
Ag Contained	kg	**173,209**	**11,789**	17,359	35,181	16,512	2,068	**71,120**	4,822	36,187	34,520	14,772	**90,301**	-	-	-	-	-
Total Oxide																		
Mass	t	**216,689**	**20,039**	29,894	30,030	30,030	24,696	**114,650**	8,185	30,030	30,030	13,755	**82,001**	-	-	-	-	-
Ag	g/t	**830**	**588**	581	1,175	618	266	**678**	589	1,205	1,150	1,074	**1,101**	-	-	-	-	-
Ag Contained	kg	**179,852**	**11,789**	17,359	35,278	18,567	6,558	**77,763**	4,822	36,187	34,520	14,772	**90,301**	-	-	-	-	-
ROM Sulphide																		
Mass	t	**27,675**	-	-	-	-	5,830	**5,830**	21,845	-	-	-	**21,845**	-	-	-	-	-
Ag	g/t	**1,035**	-	-	-	-	1,143	**1,143**	1,007	-	-	-	**1,007**	-	-	-	-	-
Ag Contained	kg	**28,654**	-	-	-	-	6,662	**6,662**	21,992	-	-	-	**21,992**	-	-	-	-	-
Total Feed																		
Mass	t	**244,364**	**20,039**	29,894	30,030	30,030	30,526	**120,479**	30,030	30,030	30,030	13,755	**103,845**	-	-	-	-	-
Ag	g/t	**853**	**588**	581	1,175	618	433	**701**	893	1,205	1,150	1,074	**1,081**	-	-	-	-	-
Ag Contained	kg	**208,505**	**11,789**	17,359	35,278	18,567	13,220	**84,425**	26,813	36,187	34,520	14,772	**112,292**	-	-	-	-	-
Sulphides in Blend	%	**11.3%**	**0.0%**	0.0%	0.0%	0.0%	19.1%	**4.8%**	72.7%	0.0%	0.0%	0.0%	**21.0%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**
FLOTATION PLANT STREAM																		
ROM Sulphide																		
Mass	t	**899,408**	-	-	-	-	-	-	-	-	-	31,244	**31,244**	**179,995**	**179,995**	**179,995**	**179,995**	**148,182**
Ag	g/t	**685**	-	-	-	-	-	-	-	-	-	961	**961**	**780**	**676**	**591**	**641**	**690**
Ag Contained	kg	**616,211**	-	-	-	-	-	-	-	-	-	30,022	**30,022**	**140,317**	**121,688**	**106,459**	**115,435**	**102,289**
Pb	%	**4.95**	-	-	-	-	-	-	-	-	-	3.47	**3.47**	**6.45**	**7.97**	**3.56**	**2.39**	**4.55**
Pb Contained	kg	**44,498,520**	-	-	-	-	-	-	-	-	-	1,084,534	**1,084,534**	**11,616,436**	**14,349,525**	**6,404,207**	**4,297,054**	**6,746,764**
Zn	%	**2.89**	-	-	-	-	-	-	-	-	-	1.88	**1.88**	**1.02**	**2.26**	**2.83**	**2.21**	**1.51**
Zn Contained	kg	**17,789,576**	-	-	-	-	-	-	-	-	-	588,490	**588,490**	**1,827,984**	**4,059,285**	**5,102,013**	**3,971,815**	**2,239,988**

			Бортовое содержание серебра, g/t	50			
			Production rate, t/year	180,000			
				Specific consumption		Annual consumption	
No.	Item	Units измерения (удельные / годовые)	Цена, RUB	normal rate	RUB/t	normal rate	thous.RUB
1	**auxiliary materials - Total, incl.:**				**136.23**		**24,522**
1.1	*crusher lining*	*кг / т*	*75*	*0.04*	*3.00*	*7.20*	*540*
1.2	*mill lining (rubber)*	*кг / т*	*300*	*0.10*	*30.00*	*18.0*	*5,400*
1.3	*balls 80 мм*	*кг / т*	*57*	*0.9*	*51.62*	*162*	*9,291*
1.4	*balls 40 мм*	*кг / т*	*57*	*0.9*	*51.62*	*162*	*9,291*
2	**Total for reagents, incl.:**				**455.44**		**81,980**
2.1	*hydrated lime*	*кг / т*	*11*	*6.87*	*75.52*	*1,236*	*13,594*
2.2	*zinc sulphate*	*кг / т*	*180*	*0.38*	*69.23*	*69.2*	*12,462*
2.3	*Aerophine 3418*	*кг / т*	*245*	*0.03*	*7.35*	*5.40*	*1,323*
2.4	*T-92*	*кг / т*	*25*	*0.05*	*1.25*	*9.00*	*225*
2.5	*butyl xanthate*	*кг / т*	*140*	*0.07*	*9.33*	*12.0*	*1,680*
2.6	*flotation pine oil*	*кг / т*	*275*	*0.02*	*5.78*	*3.78*	*1,040*
2.7	*liquid glass*	*кг / т*	*15*	*0.40*	*6.00*	*72.0*	*1,080*
2.8	*copper sulphate*	*кг / т*	*104*	*0.30*	*31.20*	*54.0*	*5,616*
2.10	*flocculant Magnafloc 10*	*кг / т*	*278*	*0.100*	*27.78*	*18.0*	*5,001*
2.11	*sodium cyanide*	*кг / т*	*178*	*0.50*	*89.00*	*90.0*	*16,020*
2.12	*calcium hypochlorite*	*кг / т*	*70*	*0.50*	*35.00*	*90*	*6,300*
2.13	*ferrous sulfate*	*кг / т*	*14*	*7.00*	*98.00*	*1,260*	*17,640*
3	**Total for energy resources, incl.:**				**537.83**		**96,809**
3.1	*Diesel fuel*	*л / тыс. л*					
	electrical energy	*кВт*ч / тыс. кВт*ч*	*4.69*	*114.68*	*537.83*	*20,642*	*96,809*
	transportation services	0			203.55		36,639
4	Technical staff salary				720.00		129,600
5	deductions to social insurance				217.44		39,139
6	Depreciation				418.68		75,363
7	spare-parts pool				314.01		56,522
8	shop's expenses				468.72		84,370
	Total				**3,471.91**		**624,943**
	same without depreciation				3,053.23		549,581
	same without depreciation , USD				**47.18**		

No.	Item	75				150				250			
	Production rate, t/year	180,000				180,000				180,000			
		Specific consumption		Annual consumption		Specific consumption		Annual consumption		Specific consumption		Annual consumption	
		normal rate	RUB/t	normal rate	thous.RUB	normal rate	RUB/t	normal rate	thous.RUB	normal rate	RUB/t	normal rate	thous.RUB
1	**auxiliary materials - Total, incl.:**		**136.23**		**24,522**		**136.23**		**24,522**		**136.23**		**24,522**
1.1	*crusher lining*	*0.04*	*3.00*	*7.20*	*540*	*0.04*	*3.00*	*7.20*	*540*	*0.04*	*3.00*	*7.20*	*540*
1.2	*mill lining (rubber)*	*0.10*	*30.00*	*18.0*	*5,400*	*0.10*	*30.00*	*18.0*	*5,400*	*0.10*	*30.00*	*18.0*	*5,400*
1.3	*balls 80 мм*	*0.9*	*51.62*	*162*	*9,291*	*0.9*	*51.62*	*162*	*9,291*	*0.9*	*51.62*	*162*	*9,291*
1.4	*balls 40 мм*	*0.9*	*51.62*	*162*	*9,291*	*0.9*	*51.62*	*162*	*9,291*	*0.9*	*51.62*	*162*	*9,291*
2	**Total for reagents, incl.:**		**455.44**		**81,980**		**455.44**		**81,980**		**455.44**		**81,980**
2.1	*hydrated lime*	*6.87*	*75.52*	*1,236*	*13,594*	*6.87*	*75.52*	*1,236*	*13,594*	*6.87*	*75.52*	*1,236*	*13,594*
2.2	*zinc sulphate*	*0.38*	*69.23*	*69.2*	*12,462*	*0.38*	*69.23*	*69.2*	*12,462*	*0.38*	*69.23*	*69.2*	*12,462*
2.3	*Aerophine 3418*	*0.03*	*7.35*	*5.40*	*1,323*	*0.03*	*7.35*	*5.40*	*1,323*	*0.03*	*7.35*	*5.40*	*1,323*
2.4	*T-92*	*0.05*	*1.25*	*9.00*	*225*	*0.05*	*1.25*	*9.00*	*225*	*0.05*	*1.25*	*9.00*	*225*
2.5	*butyl xanthate*	*0.07*	*9.33*	*12.0*	*1,680*	*0.07*	*9.33*	*12.0*	*1,680*	*0.07*	*9.33*	*12.0*	*1,680*
2.6	*flotation pine oil*	*0.02*	*5.78*	*3.78*	*1,040*	*0.02*	*5.78*	*3.78*	*1,040*	*0.02*	*5.78*	*3.78*	*1,040*
2.7	*liquid glass*	*0.40*	*6.00*	*72.0*	*1,080*	*0.40*	*6.00*	*72.0*	*1,080*	*0.40*	*6.00*	*72.0*	*1,080*
2.8	*copper sulphate*	*0.30*	*31.20*	*54.0*	*5,616*	*0.30*	*31.20*	*54.0*	*5,616*	*0.30*	*31.20*	*54.0*	*5,616*
2.10	*flocculant Magnafloc 10*	*0.100*	*27.78*	*18.0*	*5,001*	*0.100*	*27.78*	*18.0*	*5,001*	*0.100*	*27.78*	*18.0*	*5,001*
2.11	*sodium cyanide*	*0.50*	*89.00*	*90*	*16,020*	*0.50*	*89.00*	*90*	*16,020*	*0.50*	*89.00*	*90*	*16,020*
2.12	*calcium hypochlorite*	*0.50*	*35.00*	*90*	*6,300*	*0.50*	*35.00*	*90*	*6,300*	*0.50*	*35.00*	*90*	*6,300*
2.13	*ferrous sulfate*	*7.00*	*98.00*	*1,260*	*17,640*	*7.00*	*98.00*	*1,260*	*17,640*	*7.00*	*98.00*	*1,260*	*17,640*
3	**Total for energy resources, incl.:**		**537.83**		**96,809**		**537.83**		**96,809**		**537.83**		**96,809**
3.1	*Diesel fuel*												
	electrical energy	*114.68*	*537.83*	*20,642*	*96,809*	*114.68*	*537.83*	*20,642*	*96,809*	*114.68*	*537.83*	*20,642*	*96,809*
	transportation services		203.55		36,639		203.55		36,639		203.55		36,639
4	Technical staff salary		720.00		129,600		720.00		129,600		720.00		129,600
5	deductions to social insurance		217.44		39,139		217.44		39,139		217.44		39,139
6	Depreciation		418.68		75,363		418.68		75,363		418.68		75,363
7	spare-parts pool		314.01		56,522		314.01		56,522		314.01		56,522
8	shop's expenses		468.72		84,370		468.72		84,370		468.72		84,370
	Total		**3,471.91**		**624,943**		**3,471.91**		**624,943**		**3,471.91**		**624,943**
	same without depreciation		3,053.23		549,581		3,053.23		549,581		3,053.23		549,581
	same without depreciation , USD		**47.18**				**47.18**				**47.18**		

Ore sorting operating cost US$/t ore treated 2.25

1. Salary of sorting stuff (2+2 person) near 100 K$ per year
2. Electricity. SBR has checked design documentation, sorting complex designed for electicity consumption 250Kw per hour. Using YGK partial cost return for electricity in Yakutia, consider 1kw/h cost 10 cents. In total 76800$ per 3200 working hours of XRT complex.
3. Maintenance and repair. I had check spare and wear part for 1 year from steinert, part cost is 14,971Euro , but we do not know costs for maintenance work in case we will use steinert engineers. So, my suggestion make estimate something near 100K$ per year
4. Costs of diesel fuel and work time of loader. In practice can use the same loader working on the ore crushing now but it will work more intensively. Currently assume working at 30% of possible performance. So fuel consumption for loader and maintenance and repair costs will increase but SBR is not sure how to estimate it properly. But it is understood the loader fuel consumption approximates 20l of diesel per hour or 20$ per hour for fuel. Consider for estimation 3200 hours per year loader work just for xrt sorter and fuel consumption 64000$.
5. There is no additional costs per ore transportation from open pit, because now we also transport ore, but there will be some additional costs for tails transportation. 80 000 tons of tail is something near 20K$.

100K$+76800$+100K$+64K$+20K$=360 800. If we consider XRT performance 160 000 t of ore to the process plant 360 800/160 000= 2,25$

64.71

Processing Capex for Primary Ore

No.	Item	Model	Price, thous.Rub	Processing flowsheet		Power Consumption, kW		
				Qty	Cost, thous.Rub	Units	Total	Thous US$
1	Технологическое оборудование, в том числе:				**753,625**		**1200**	
1.1	*Base case crusher*	*ЩДС-1-5x9*	*14,858*	*2*	*29,717*	55	110	
1.2	*Base case cone crusher*	*СМД-120А-Р-200*	*11,413*	*2*	*22,825*	55	110	
1.3	*Ball mill*	*МШЦ 3,9x3,0*	*91,373*	*2*	*182,747*	500	1000	
1.5	*conditioning tank*	*КЧ-4*	*250*	*1*	*250*	18.5	18.5	
1.6	*Flotation cell*	*РИФ-1,5*	*3,500*	*14*	*49,000*	7	98	
1.7	*Filter press (Pb KT)*	*OUTOTEC Larox 800x800 (17*	*16,000*	*1*	*16,000*	18.5	18.5	
1.8	*Filter press (Zn KT)*	*OUTOTEC Larox 800x800 (33*	*20,000*	*1*	*20,000*	18.5	18.5	
1.9	*conditioning tank*	*КЧР-0,8А*	*130*	*1*	*130*	1.5	1.5	
1.10	*Cyanidation tank with me*	*70 м^3*	*1,150*	*4*	*4,600*	11	44	
1.11	*Radial thickener*	*СЦ-2,5А*	*1,000*	*1*	*1,000*	0.75	0.75	
1.12	*Filter press (кек)*	*OUTOTEC Larox 800x800 (33*	*20,000*	*1*	*20,000*	18.5	18.5	
1.16	*Electrowinning unit*	*emew PLANT*	*183,939*	*1*	*183,939*	375	32	
1.17	*Filter press (хвосты)*	*BILFINGER ME1500.3500 (35*	*28,938*	*3*	*86,813*	19.6	58.8	
1.18	*Radial thickener*	*СЦ-15*	*11,000*	*1*	*11,000*	4	4	
1.19	*unaccounted equipment and metal structures*				*125,604*			
2	plumping and electrical engineering				60,290			
3	Automation				75,363			
	Total for equipment				**889,278**			**13,743**
4	transporation costs				35,571			550
	Equipment + delivery				**924,849**			**14,292**
5	building and installation work				184,970			2,858
6	commissioning				46,242			715
	Total for capital investments				**1,156,061**			**17,865**
	Complementary to existing				224,541			3,470

Version per SRK and ERM:

Period	Nominal	Inflated	PV OB	Accretion	Payment		PV EB	LOM	Inflation	Discount rate
31/12/2017	87,622	148,301					54,212	11	4.90%	8.64%
31/12/2018	91,127	134,891	54,212	4,684	0		58,896	10	4.00%	8.64%
31/12/2019	94,772	134,891	58,896	5,089	0		63,984	9	4.00%	8.64%
31/12/2020	98,563	134,891	63,984	5,528	0		69,513	8	4.00%	8.64%
31/12/2021	102,506	134,891	69,513	6,006	0		75,518	7	4.00%	8.64%
31/12/2022	106,606	134,891	75,518	6,525	0		82,043	6	4.00%	8.64%
31/12/2023	110,870	134,891	82,043	7,089	0		89,132	5	4.00%	8.64%
31/12/2024	115,305	134,891	89,132	7,701	0		96,833	4	4.00%	8.64%
31/12/2025	119,917	134,891	96,833	8,366	0		105,199	3	4.00%	8.64%
31/12/2026	124,714	134,891	105,199	9,089	0		114,288	2	4.00%	8.64%
31/12/2027	129,702	134,891	114,288	9,875	0		124,163	1	4.00%	8.64%
31/12/2028	134,891	134,891	124,163	10,728	(134,891)		(0)	0	4.00%	8.64%

Version per EMS:

Period	Nominal	Inflated	PV OB	Accretion	Payment		PV EB	LOM	Inflation	Discount rate
31/12/2017	219,325	371,209					125,459	11	4.90%	8.64%
31/12/2018	228,098	337,641	125,459	10,840	0	0	136,298	10	4.00%	8.64%
31/12/2019	237,222	337,641	136,298	11,776	0		148,075	9	4.00%	8.64%
31/12/2020	246,711	337,641	148,075	12,794	0		160,868	8	4.00%	8.64%
31/12/2021	256,579	337,641	160,868	13,899	0		174,767	7	4.00%	8.64%
31/12/2022	266,842	337,641	174,767	15,100	0		189,867	6	4.00%	8.64%
31/12/2023	277,516	337,641	189,867	16,405	0		206,272	5	4.00%	8.64%
31/12/2024	288,617	337,641	206,272	17,822	0		224,093	4	4.00%	8.64%
31/12/2025	300,161	337,641	224,093	19,362	0		243,455	3	4.00%	8.64%
31/12/2026	312,168	337,641	243,455	21,035	0		264,490	2	4.00%	8.64%
31/12/2027	312,168	312,168	264,490	22,852	0		287,342	1		
31/12/2028	312,168	312,168	287,342	0	(312,168)		(24,826)	0		

ARO liability estimate:	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**
Discount rate	7.70%	8.64%	6.41%	6.41%	6.41%	6.41%	6.41%	6.41%	6.41%			
Inflation rate	4.90%	4.00%	4.00%	4.00%	4.00%	4.00%	4.00%	4.00%	4.00%			
LOM-end year	2028	2028	2028	2028	2028	2028	2028	2028	2028	2028	2028	2028

Reporting period	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
LOM-end year	2028	2028	2028	2028	2028	2028	2028	2028	2028	2028	2028	2028
	in kRUB	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*
Nominal value	219,325	228,098	237,222	246,711	256,579	266,842	277,516	288,617	300,161	300,161	300,161	300,161
Inflated value	371,209	337,641	337,641	337,641	337,641	337,641	337,641	337,641	337,641	300,161	300,161	300,161
Discounted value	164,152	147,420	193,025	205,398	218,564	232,573	247,481	263,345	280,225	300,161	300,161	300,161

	in kRUB	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*	*in kRUB*
OB	48,239	65,580	56,631	193,025	205,398	218,564	232,573	247,481	263,345	280,225	300,161	300,161
Unwinding of discount	4,027	5,050	4,893	12,373	13,166	14,010	14,908	15,864	16,880	17,962	0	0
Change in underlaying value	13,314		84,090									
Change in estimate		(13,999)	47,411	(0)	(0)	0	0	0	0	1,974	0	0
EB	65,580	56,631	193,025	205,398	218,564	232,573	247,481	263,345	280,225	300,161	300,161	300,161

Change in estimate effects:

Additions			84,090									
Change in assumptions		(13,999)	47,411	(0)	(0)	0	0	0	0	1,974	0	0
Change in estimate total		(13,999)	131,501	(0)	(0)	0	0	0	0	1,974	0	0

ARO asset estimate	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**
OB	48,239	61,554	47,554	179,056	179,056	179,056	179,056	179,056	179,056	179,056	181,029	181,029
Additions	13,314		84,090									
Change in estimate		(13,999)	47,411	(0)	(0)	0	-	-	-	1,974	-	-
EB	61,554	47,554	179,056	179,056	179,056	179,056	179,056	179,056	179,056	181,029	181,029	181,029
Accumulated depletion												
OB		(4,385)	(8,771)	(12,649)	(31,139)	(49,628)	(68,118)	(86,608)	(105,097)	(123,587)	(142,076)	(160,566)
Charge for the year	(4,385)	(4,385)	(3,878)	(18,490)	(18,490)	(18,490)	(18,490)	(18,490)	(18,490)	(18,490)	(18,490)	(18,490)
EB	(4,385)	(8,771)	(12,649)	(31,139)	(49,628)	(68,118)	(86,608)	(105,097)	(123,587)	(142,076)	(160,566)	(179,056)
Net book value												
OB	**48,239**	**57,168**	**38,784**	**166,406**	**147,917**	**129,427**	**110,938**	**92,448**	**73,958**	**55,469**	**38,953**	**20,463**
EB	**57,168**	**38,784**	**166,406**	**147,917**	**129,427**	**110,938**	**92,448**	**73,958**	**55,469**	**38,953**	**20,463**	**1,974**

Prognoz ARO
Provision for decommissioning and restoration liability

Activity	Source	Nominal, kRUB	Nominal, $k	$m
			60.00	
PY				
Explosive storage	ERM PY	287	5	
Main fuel farm	ERM PY	18,522	309	
Temp fuel storage	ERM PY	1,406	23	
Endybal airstrip	ERM PY	489	8	
Fleet demobilisation	ERM PY	3,957	66	
Hogin mancamp	ERM PY	15,690	262	
Endybal decommissioning	ERM PY	4,957	83	
Hogin sawmill	ERM PY	606	10	
Boreholes, pump stations	ERM PY	4,174	70	
Waste disposal	ERM PY	2,485	41	
				0.88
ADJs:				
Less: airstrip (no need)	Estimate	(489)	(8)	
Less: fleet demobilisation (accounted in sale surplus)	Estimate	(3,957)	(66)	
Less: fuel tanks freight (no need)	Estimate	(18,150)	(303)	
Less: mancamp freight (no need)	Estimate	(14,025)	(234)	
				(0.61)
PIT				
Fencing & re-seeding pit rims	SRK adjusted	2,000	33	
Channel excavation, control and engineering works, allow	SRK adjusted	3,240	54	
Design and site supervision	SRK adjusted	1,200	20	
Waste rock dumps/stockpiles re-contouring, soil replacement	SRK adjusted	9,771	163	
				0.27
TMF				
Re-contouring, capping, re-seeding	SRK adjusted	4,320	72	
Operation, maintenance and removal of pumps and other items	SRK adjusted	600	10	
Design and supervision, allow	SRK adjusted	1,200	20	
				0.10
PLANT				
Cleaning of process equipment	SRK adjusted	3,000	50	
Treatment of effluent	SRK adjusted	3,000	50	
Dismantling equipment, salvage or disposal of equipment	SRK adjusted	9,180	153	
Dismantling drainage system and hard standing	SRK adjusted	3,960	66	
Testing for contamination	SRK adjusted	3,000	50	
Preparation of surface	SRK adjusted	3,000	50	
				0.42
OTHER				
Post closure fund for sustainable development	SRK adjusted	2,000	33	
Monitoring of TMF, allow $10,000 for 5 years	SRK adjusted	3,000	50	
Staff redundancies etc	Estimate	15,000	250	
Insurance	SRK adjusted	1,200	20	
Contingency	SRK adjusted	3,000	50	
				0.40
Total		**87,622**	**1,460**	**1.46**

Макрокроэкономический проноз (апдейт) на 2020, 2021-2030 гг.

№	Параметр	2019	2020F	2021F	2022F	2023F	2024F	2025F	2026F	2027F	2028F	2029F	2030F
1	**Сценарий 1. БАЗОВЫЙ**												
2	Цена на нефть Brent	$65	$36	$50	$55	$60	$61	$62	$64	$65	$66	$68	$69
3	USD/RUB (среднее за год)	64.7	72.1	70.0	70.0	70.0	71.4	72.8	74.2	75.7	77.1	78.6	80.2
4	USD/RUB (конец года)	62.3	70.0	70.0	70.0	70.0	71.4	72.8	74.2	75.7	77.1	78.6	80.2
5	Потребительская инфляция в РФ, среднее за год, %	4.5%	4.7%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%
6	EUR/USD (средний)	1.12	1.10	1.10	1.12	1.15	1.18	1.20	1.22	1.22	1.22	1.23	1.23
7	EUR/USD (на конец года)	1.11	1.10	1.10	1.12	1.15	1.18	1.20	1.22	1.22	1.22	1.23	1.23
8	EUR/RUB (средний)	72.5	79.3	77.0	78.6	80.5	84.2	87.3	90.5	92.5	94.5	96.5	98.6
9	EUR/RUB (на конец года)	69.1	77.0	77.0	78.6	80.5	84.2	87.3	90.5	92.5	94.5	96.5	98.6
10	**Справочно: предыдущий прогноз для СБП**												
11	Цена на нефть Brent	$65	$62	$64	$64	$65	$66	$67	$69	$70	$71	$73	$74
12	USD/RUB (средний за год)	64.7	63.1	66.1	66.5	66.9	67.4	68.7	70.1	71.4	72.8	74.3	75.7
13	Потребительская инфляция в РФ, среднее за год, %	4.5%	3.0%	3.7%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%
14	EUR/USD	1.11	1.12	1.15	1.18	1.20	1.22	1.22	1.22	1.23	1.23	1.23	1.23
15	**Сценарий 2. Низкие цены на нефть**												
16	Цена на нефть Brent	$65	$30	$32	$35	$40	$40	$40	$41	$42	$42	$43	$44
17	USD/RUB (средний за год)	64.7	81.0	78.0	77.0	75.0	76.5	78.0	79.5	81.1	82.6	84.3	85.9
18	Потребительская инфляция в РФ, среднее за год, %	4.5%	6.8%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%

Источник: оценки управления корпоративной стратегии на основе прогнозов аналитиков

Ag	$/oz	17.76	SP Angel Report, published in Aug 2019
Pb	$/t	2,069	
Zn	$/t	2,252	

World Bank CMO October 2019		2017	2018	2019	2020	2021	2022	2023	2025	2030
Lead	$/mt	2,315	2,240	1,970	1,950	1,965	1,979	1,979	2,024	2,100
Zinc	$/mt	2,891	2,922	2,570	2,450	2,455	2,460	2,460	2,475	2,500
Silver	$/toz	17.10	15.70	16.20	17.00	17.00	17.00	17.00	17.00	17.00

World Bank CMO April 2020		2017	2018	2019	2020	2021	2022	2023	2025	2030
Silver	$/toz	17.10	15.70	16.20	16.80	17.00	17.10	17.20	17.40	18.00
Lead	$/mt	2,315	2,240	1,997	1,700	1,800	1,831	1,863	1,928	2,100
Zinc	$/mt	2,891	2,922	2,550	1,900	2,000	2,050	2,102	2,209	2,500

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