wardell-armstrong.com

ENERGY AND CLIMATE CHANGE
ENVIRONMENT AND SUSTAINABILITY
INFRASTRUCTURE AND UTILITIES
LAND AND PROPERTY
MINING AND MINERAL PROCESSING
MINERAL ESTATES
WASTE RESOURCE MANAGEMENT

LLC DALTSVETMET
TECHNICAL REPORT

NI 43-101 COMPLIANT MINERAL RESOURCE AND ORE RESERVE ESTIMATE FOR THE NASEDKINO GOLD PROJECT, RUSSIAN FEDERATION

August 2017

your earth our world

wardell armstrong

Wardell Armstrong International
Baldhu House, Wheal Jane Earth Science Park, Baldhu, Truro, Cornwall, TR3 6EH, United Kingdom
Telephone: +44 (0)1872 560738 www.wardell-armstrong.com

wardell armstrong
your earth our world

DATE ISSUED:	**10 August 2017**
JOB NUMBER:	**RU10051**
VERSION:	**V5.0**
REPORT NUMBER:	**MM1119**
STATUS:	**Final**

LLC DALTSVETMET
TECHNICAL REPORT

NI 43-101 COMPLIANT MINERAL RESOURCE AND ORE RESERVE ESTIMATE FOR THE NASEDKINO GOLD PROJECT, RUSSIAN FEDERATION

August 2017

PREPARED BY:

Mark Mounde	Technical Director of Mining
Richard Ellis	Principal Resource Geologist
Narina Shorland	Senior Mining Engineer
Philip King	Technical Director of Process Engineering
Stephen Holley	Senior Mining and Geotechnical Engineer
Stuart Richardson	Senior Mining Engineer
Veronika Luneva	Senior Financial Analyst
Alison Allen	Associate Director of Environment and Social aspects
Lauren Ballarini	Associate Director of Hydrology and Hydrogeology

APPROVED BY:

Dr. P S Newall	Managing Director of WAI

CONTENTS

TABLES

FIGURES

PHOTOS

APPENDICES

SUMMARY

Wardell Armstrong International Limited ("WAI") was commissioned by LLC Daltsetmet ("Daltsvetmet") in June 2016, to prepare an updated estimate of Mineral Resources in accordance with the guidelines of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2012)] for the Nasedkino Gold Project, East Siberia Russian Federation. WAI considers that the Nasedkino mineral resources and reserves are reported following the industry best practices and to be compliant as outlined in the National Instrument 43-101.

The Nasedkino Gold Project is located in the Mogocha district, Zabaikalsky Krai, Russian Federation, 56km north west of Mogocha, the nearest town and accessed by a gravel (all-season) road, and 650km northeast of Chita, the regional capital. The nearest village is Chaldonka and is located 15.5km from the Nasedkino Gold Project and is accessed by an earth road. The Trans-Baikal railway runs 16km south of the site.

Daltsvetmet holds two mining and exploration licences at the project. The first comprises mining and exploration licence ЧИТ 01663 БЭ and is valid until December 1, 2025. The licence covers an area of 10.9km^2 and includes Pridolinny and Gora 5 deposits. The second comprises mining and exploration licence ЧИТ 02652 БР and is valid until December 31, 2036. The licence covers an area of 19.6km^2 and includes Zhelanny and Pravoberezhny deposits.

The Nasedkino Gold Project is located along the southeast of the Siberian Craton and is hosted by a middle Palaeozoic accretionary subduction complex. The host rocks comprise Archaen ultra-metamorphic formations including granulites with skarn mineral assemblages and gneisses that have been folded into isoclinal northwest trending folds. Typical lithologies include pyroxene-garnetiferous granulites (skarnoid) and quartz-biotite-plagioclase gneisses often containing silliminite, hypersthene, graphite, high-aluminium crystalline schists and amphibolites. Of the two host lithologies the most significant mineralisation is hosted by the skarnoids. The host lithologies have been intruded by Late Jurassic intrusions of medium acidic composition which are the source of the mineralisation. Fault zones acted as feeders for the mineralised hydrothermal fluids while higher permeability metamorphic strata comprised suitable fluid migration pathways. The mineralised zones occur as lenticular zones of hydrothermal alteration and lie conformably with the metamorphic strata which strike north - west (330-345°) and dip steeply (70-80°) to the northeast. A series of cross cutting late stage faults off-set the mineralised zones.

The Nasedkino Gold Project consists of four deposits and four advanced exploration prospects. The four deposits consist of Pridolinny, Gora 5, Zhelanny and Pravoberezhny and are located on two parallel mineralised corridors which extend for more than 2.5km in a northwest direction. Zhelanny and Pravoberezhny are located on the western most of the mineralised corridors. Pridolinny and Gora 5 are located on a mineralised corridor located 700m to the east of this. These two main mineralised corridors exhibit similar mineralisation, geology and structure. The four advanced exploration prospects consist of Severo Zapadny, Shtokverkovy, Medny (Gora 4) and Glukharinny.

Gold bearing mineralisation mostly includes pyrite and to a lesser degree, chalcopyrite, and more rarely, galena, sphalerite, fahlite, arsenopyrite, molybdenite, magnetite, chalcocite and other sulphides. The average sulphide content is 5-10%. Gold grades are not directly dependent on the sulphide content, while major quantities of gold (>60%) are coupled with sulphides – mostly in intergrowth with pyrite. The remaining gold is dispersed as free gold in quartz and limonite.

Three drilling campaigns have been undertaken at the Nasedkino Gold Project. During 2006-2008, 352 drill holes totalling 37,939m were drilled. During 2010-2012, 431 drill holes totalling 52,634m were drilled. During 2015, 27 drill holes totalling 2,255m were drilled. All of the drilling was conducted by diamond core drilling.

Mineral Resource estimation involved the use of drill hole data to construct three dimensional wireframes to define individual vein structures. Samples were selected inside these wireframes, coded and grade caps applied where applicable before compositing. Boundaries were treated as hard with statistical and geostatistical analysis conducted on composites identified in individual veins. Gold grades were estimated into a geological block model representing each vein. Grade estimation was carried out by inverse power of distance. Estimated grades were validated globally, locally, and visually prior to tabulation of the Mineral Resources.

The effective date of the Mineral Resource estimate is February 10, 2017. A summary of the Mineral Resource statement is shown in table below.

Nasedkino Gold Project Mineral Resource Estimate
(WAI, February 10, 2017)

	Cut-Off (g/t Au)	Resource Classification	Tonnes (Kt)	Au Grade (g/t)	Au Metal (kg)	Au Metal (Koz)
Pridolinny	0.70	Indicated	5,410	1.79	9,696	312
		Inferred	895	1.84	1,644	53
Gora 5	0.70	Indicated	1,926	2.26	4,354	140
		Inferred	375	1.87	702	23
Zhelanny	0.70	Indicated	1,748	3.77	6,592	212
		Inferred	402	1.83	737	24
Pravoberezhny	0.70	Indicated	1,128	2.11	2,383	77
		Inferred	1,669	1.80	3,002	97
Total	**0.70**	**Indicated**	**10,212**	**2.25**	**23,025**	**740**
		Inferred	**3,341**	**1.82**	**6,085**	**196**

Notes:
1. Mineral Resources limited by an optimised pit shell using a gold price of $1500/oz and reasonable economic and technical parameters;
2. Mineral Resources are not reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study;
3. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery and;
4. Numbers may not add due to rounding.

Mineral Reserve estimates have considered only Measured and Indicated Mineral Resources as only these categories have sufficient geological confidence to be considered Mineral Reserves. Subject to the application of certain economic and mining-related qualifying factors, Measured Resources may become Proven Reserves and Indicated Resources may become Probable Reserves.

Ore Reserves Estimate

The Nasedkino project contains four main deposits: Zhelanny, Gora 5, Pridolinny and Pravoberezhny. The Ore Reserve Estimate was conducted for each deposit separately, and was carried out following the guidelines of the CIM Definition Standards for Ore Reserves. Estimated Ore Reserve tonnages were based on a long term price forecast of US$1,100/oz Au.

The Ore Reserve estimate was completed using industry standard practices for open pit using Studio 5D Planner and NPV Scheduler (NPVS) Datamine softwares. The software was used to generate optimal pit envelopes using NPVS and subsequent mine design using 5D Planner, before final scheduling was completed on the mine design in NPVS.

Mineral Resource Base

Open pit optimisation and mine design for the purpose of Ore Reserve Estimation has been carried out based upon the WAI Mineral Resource block models for the four deposits:

- Gora-5: gora5_opt.dm
- Pridolinny: prid_opt.dm
- Pravoberezhny: parvo_opt.dm
- Zhelanny: zhel_opt.dm

Optimisation Parameters

WAI has based the optimisation parameters on data presented by Daltsvetmet, considering market studies, quotations from suppliers, as well as reasonable projections or estimates; supplemented by WAI estimates where appropriate. The tables below present the data used for open pit optimisation.

Pit Optimisation Parameters						
Parameter	**Unit**	**Value**				**Source**
Site		**Gora 5**	**Pridolinny**	**Pravober.**	**Zhelanny**	
Exchange Rate	RUB:US$	65				Mangazeya Financial Model
Metal Price	RUB/g	2299				
	US$/oz	1,100				
Cut-off grade for Au	g/t	0.7				TEO 2016
Discount Rate	%	10				
Ore specific gravity	t/m³	2.79		2.8		
Waste specific gravity	t/m³					
Waste Mining Cost	RUB/t	54.0				Mangazeya financial model
	US$/t	0.83				
Ore Mining Cost	RUB/t	51.9				
	US$/t	0.80				
Ore Treatment Cost (inc G&A)	RUB/t	980				
	US$/t	15.08				
Rehabilitation Costs	RUB/m³	4.11				TEO 2016
	US$/m³	0.06				
Royalties	%	6				
Refining	RUB/g	5.5				
	US$/g	0.08				
Metallurgical Recovery	%	91.09	88.84	95.06	93.75	
Mining Dilution	%	2.4	5.62	10	10	WAI estimate/TEO2016
Mining Losses	%	1.81	3.44	2.5	2.5	
Overall Slope Angle	°	45				TEO 2016
Pit base	m	21				TEO 2016

Pit Design Parameters			
Parameter	**Unit**	**Value**	**Source**
Face Angle of non-working Slope	°	53	Technical Project 2016
Safety berm width	m	8	TEO 2016
Haul Road Berm Width	m	21	
Haul Road Berm Width at lower horizons	m	15	
Bench Height	m	5	
Bench Height of non-working bench	m	20	
Face Angle of non-working bench	°	70	
Haul Road Gradient	%	8	

Optimisation Results

WAI has used NPVS to produce the open pit optimisation for the Nasedkino deposits. A series of pit shells was generated at varying metal price factors ranging from 1% to 120% of the base data. A notional discounted revenue was reported for each pit shell, enabling comparison of the economic value of the pit shell and determination of the optimum economic return mining scenario. In accordance with Daltsvetmet strategy, WAI has considered the 100% LG Phase shells for delineation of the open pit design and estimation of Ore Reserves.

Mine Designs

WAI has utilised the derived optimised pit shells as a basis for the mine design, in conjunction with the pit design parameters and geotechnical analysis.

The in-situ tonnage of the material in designed pits, exclusive of losses and dilution is as following:

Nasedkino Pit Design In-situ Mineable Content			
Deposit	**Ore (Mt)**	**Grade (Au g/t)**	**Gold (t)**
Gora 5	1.75	2.27	3.98
Pridolinny	4.48	1.77	7.91
Pravoberezhny	0.95	2.15	2.05
Zhelanny	1.64	3.86	6.34
TOTAL	**8.82**	**2.30**	**20.28**
Note: Contains only Indicated category Mineral Resources.			

- The Nasedkino ore reserve statement was based upon the Gold price forecast at the time of carrying out the Nasedkino mine design optimisation of US$1100. If the mine design was carried out using current spot prices of over US$1200, and revised gold forecasting, there would be a significant uplift in the total gold at Nasedkino.

The designed pit content, inclusive of losses and dilution, is presented in the table below:

Nasedkino Ore Reserves Estimate (WAI, February 10, 2017) to JORC 2012			
Reserves Classification	**Ore**	**Grade**	**Metal**
	tonnes	**g/t Au**	**Tonnes Au**
Gora 5			
Proved	-	-	-
Probable	1,760,565	2.22	3.91
Total	1,760,565	2.22	3.91
Pridolinny			
Proved	-	-	-
Probable	4,567,010	1.67	7.63
Total	4,567,010	1.67	7.63
Pravoberezhny			
Proved	-	-	-
Probable	1,021,413	1.96	2.00
Total	1,021,413	1.96	2.00
Zhelanny			
Proved	-	-	-
Probable	1,760,242	3.51	6.18
Total	1,760,242	3.51	6.18
TOTAL			
Proved	**-**	**-**	**-**
Probable	***9,109,230***	***2.17***	***19.72***
Total	**9,109,230**	**2.17**	**19.72**

Mining Schedule

Following derivation of the mine designs, a combined mining schedule was derived using NPV Scheduler software to produce annual pit shells and year-by-year mineable tonnages and grades for ore and waste.

The mining schedule, presented in the table below, is predicated upon a total ROM ore tonnage of 1.1Mtpa, after losses and dilution have been considered.

Nasedkino Life-of-Mine Schedule											
Deposit	**Year**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**TOTAL**
Zhelanny ROM Schedule	*Ore (kt)*	550	1,101	109							**1,760**
	Grade (g/t Au)	2.71	4.02	2.41							**3.51**
	Gold (t)	*1.49*	*4.43*	*0.26*							***6.18***
	Waste (kt)	7,291	12,883	308							**20,482**
Gora 5 ROM Schedule	*Ore (kt)*			990	770						**1,761**
	Grade (g/t Au)			2.33	2.08						**2.22**
	Gold (t)			*2.31*	*1.6*						***3.91***
	Waste (kt)			13,666	4,962						**18,627**
Pravoberezhny ROM Schedule	*Ore (kt)*	63*	95*	168*	260	436					**1,021**
	Grade (g/t Au)	1.51	1.51	1.69	1.23	2.66					**1.96**
	Gold (kg)	*0.09*	*0.14*	*0.28*	*0.32*	*1.16*					***2.00***
	Waste (kt)	2,825	3,756	3,094	5,191	5,096					**19,961**
Pridolinny ROM Schedule	*Ore (kt)*					682	1,101	1,099	1,100	585	**4,567**
	Grade (g/t Au)					1.81	1.7	1.68	1.56	1.64	**1.67**
	Gold (t)					*1.24*	*1.88*	*1.85*	*1.71*	*0.96*	***7.63***
	Waste (kt)					7,530	8,255	8,071	8,073	3,181	**35,110**
TOTAL Process Schedule	***Ore (kt)***	**550**	**1,101**	**1,099**	**1,356***	**1,118**	**1,101**	**1,099**	**1,100**	**585**	**9,109**
	Grade (g/t Au)	**2.71**	**4.02**	**2.34**	**1.8**	**2.14**	**1.7**	**1.68**	**1.56**	**1.64**	**2.17**
	Gold (t)	**1.49**	**4.43**	**2.57**	**2.45**	**2.39**	***1.88***	***1.85***	***1.71***	***0.96***	***19.72***
	Waste (kt)	**10,116**	**16,639**	**17,067**	**10,153**	**12,625**	**8,255**	**8,071**	**8,073**	**3,181**	**94,181**

* Note: Ore mined from Pravoberezhny in Years 2018-2020 will be stockpiled and processed in Year 2021

Processing

The results of the extensive pilot scale and bulk-scale testwork on the four Nasedkino areas using three processing flowsheets - gravity-flotation, gravity-cyanidation and head cyanide leach, established that the following gold recoveries were achievable:

- Pridolinny: with the gravity-cyanide flowsheet recovery was 88.8% versus 82.3% using the gravity-flotation and 83.04% using the cyanide head leach flowsheet with differences of 6.57 and 5.80%, respectively;
- Gora 5 area: - recovery was 91.4% using the head cyanide leach flowsheet versus 80.2% using the gravity-flotation and 91.1% using the gravity-cyanide leach flowsheet, with recovery differences of 11.2 and 0.29%, respectively;
- Zhelanny: the recovery was 93.8% using the gravity-cyanide leach flowsheet versus 87.4% using the gravity-flotation and 91.7% using the head cyanide leach flowsheet, with differences of 6.4 and 2.4%, respectively; and
- Pravoberezhny: the recovery was 95.1% with the gravity-cyanide leach flowsheet followed by 90.6% using gravity-flotation and with 94.5% using the head cyanide leach flowsheet, with a differences of 4.5% and 0.58%, respectively.

According to the Russian system, the dressability of the Gora 5, Zhelanny and Pravoberezhny ores were defined as “readily processed”, and the Pridolinny ores are defined as “moderately easily processed”. It was concluded that the aforementioned classifications does not exclude the possibility of processing them using a single processing flowsheet and therefore they can be referred to as one metallurgical type.

A process design was developed for the treatment of the Nasedkino ores based on treating approximately 1Mtpa as part of the TEO. The processing flowsheet involved:

- Primary crushing to 250mm;
- SAG-Ball mill grinding;
- Gravity processing with the gravity concentrate and tailings leached separately;
- Carbon adsorption and desorption’; and
- Smelting the cathode product.

After a detox stage, leach tailings will be thickened, filtered and disposed of as a filter cake. Process plant production mass balances were produced based on the mining schedule and calculated for five gold cut-off grades. The plant uses conventional and well established technologies including gravity and cyanidation which have been thoroughly tested on all four ore sources through extensive programmes of laboratory and pilot plant testwork. A combination of Chinese and Western equipment has been selected in the design study.

The process operating consumable levels were determined from testwork data and experience with similar ores and are typical for the treatment of a moderately hard, non-refractory gold ore.

Hydrological and Hydrogeological

The River Uryum and its tributaries flows in a southerly direction across the Nasedkino deposit. Alluvial deposits associated with these water bodies are reportedly water bearing. The majority of the flow occurs in these rivers during the summer months following snow melt. Permafrost is present across the region and leads to water being perched above the permafrost layer.

The bedrock across the region is fractured and groundwater is present within these fractures. The hydraulic conductivity of the bedrock is controlled by the degree of fracturing and the inter connectivity of these fractures. The fractures are reportedly fed by direct precipitation.

Hydrogeological studies have been undertaken at Gora 5, Pridolinny and Zhelanny. These studies indicate that the water table in the fractured bedrock is generally encountered in the upper 20m. Yields are generally low although some localised, higher flows have been encountered.

Market Review and Studies

The final product is going to be sold to Russian commercial banks.

The gold price had a strong performance in 2016. 2017 is expected to be an eventful and unpredictable year given the high degree of geopolitical uncertainty.

The Competent Authority for the world Bullion Market, LBMA, forecast contributors are predicting that the gold price will average $1,244/oz in 2017, 5.3% higher than the first half of January 2017, but broadly in line with the actual average price in 2016.

In contrast, the World Bank expects gold prices to fall 8 percent in 2017 (resulting in forecasted US$1,150/oz in 2017), mainly due to weak investment demand, prospects of a stronger dollar, and rising real interest rates.

WAI suggested to follow the more conservative approach and, therefore, the long-term gold price of US$1,100/oz was selected for reserve estimation of the Nasedkino project.

Capital and Operating Costs

The Nasedkino Project will require total pre-production capital cost of US$84.2M (including contingency) with overall LOM capital requirements of US$99M. A summary of the Project capital costs is presented in the table below.

Overall Project Capital Costs Summary		
Project design, Engineering Study, Permitting	US$ M	1.5
Mining Equipment	US$ M	18.6
Gold Processing Plant	US$ M	37.9
Power Infrastructure	US$ M	5.5
General Infrastructure	US$ M	15.7
Mining Equipment Maintenance	US$ M	5.6
Plant Maintenance Cost	US$ M	3.9
Contingency (10%)	US$ M	7.9
Mine Closure and Reclamation	US$ M	2.22
Total Capital Costs	**US$ M**	**99**

The LOM cost of production was estimated at US$244M or total cash cost of US$420/oz and includes operating mining and processing costs, auxiliary works and general and administration, royalty (mineral tax) and property tax. A summary table is shown below.

Project Operating Costs per t of Ore Mined and Au oz Sold		
	US$/t ore mined	**US$/oz sold**
Mining Opex	6.21	98
Beneficiation Opex	11.10	174
Auxil. Works and G&A	4.31	68
Royalty	4.19	66
Property Tax	0.95	15
TOTAL	**26.75**	**420**

Financial Results

The Nasedkino Project has been examined from a financial perspective using Discounted Cash Flow (DCF) analysis, using a conservative fixed long-term price of US$1,100/oz and exchange rate of 65 Rub/US$. The model is presented in real cash flows (as of January 2017) with no escalation or inflation considered in the current valuation.

The results of the economic analysis presented in this Study indicate that within the given economic environment and based on technical and economic parameters used, the Nasedkino Project has 3 years of payback period (based on debt-free discounted cash flows) and NPV of US$127M at 10% base case discount rate, and project internal rate of return of 82%, using Au price of US$1,100/oz. An upside scenario resulted in US$154M (at 10% discount rate) using US$1,200/oz price.

A financed scenario resulted in base case NPV (at 10% discount rate) of US$126M and US$153M, using US$1,100/oz and US$1,200/oz respectively.

An all-in sustaining cost was estimated at US$443/oz (using Au price of US$1,100/oz).

1 INTRODUCTION

This Technical Report has been prepared by WAI in accordance with the disclosure requirements of Canadian National Instrument 43-101 (NI 43-101) to disclose recent information about the Nasedkino Gold Project. This information has resulted from additional exploration drilling and updated Mineral Resource and Ore Reserve estimates.

1.1 Scope of Work

The scope of work comprised the development of a Mineral Resource model and an estimate of Mineral Resources and Ore Reserves in accordance with the guidelines of the JORC Code (2012) and the preparation of a Technical Report in accordance with the guidelines of NI 43-101 and Form 43-101F1. There are no material differences between the Mineral Resources and Ore Reserve categories under the guidelines of the JORC Code (2012) used in this report and the categories of Mineral Resources under NI 43-101.

The general technical areas that were reviewed are detailed below:

- History and current status;
- Location, topography and climate;
- Licence details;
- Ownership, tenure, royalties and permits;
- Geology, structure and mineralogy of the orebody(s);
- Quality of the historic and more recent exploration data;
- Audit of any available existing resources;
- Identification of the main geological (and other) risks to the project;
- A consideration of the likely mining operations, mine plan, mining methods, plant and associated infrastructure;
- Processing operations, technologies that may be employed and suitability, including product mix, quality and yields;
- Process design;
- Tailings management;
- Project logistics including road, rail, communications, power and water;
- Markets and contracts;
- Environmental and social impacts, and
- Financial assessment of the Project including capital and operating costs for all aspects of the operations.

1.2 Independent Consultants

WAI has provided the mineral industry with specialised geological, mining and processing expertise since 1987, initially as an independent company, but from 1999 as part of the Wardell Armstrong

Group (WA). WAI's experience is worldwide and has been developed in the coal and metalliferous mining sector.

Our parent company is a mining engineering/environmental consultancy that services the industrial minerals sector from nine regional offices in the UK and international offices in Almaty, Kazakhstan and Moscow. Total worldwide staff compliment is now in excess of 400.

WAI, its directors, employees and associates neither has nor holds:

- Any rights to subscribe for shares in Daltsvetmet either now or in the future;
- Any vested interests in any licences held by Daltsvetmet;
- Any rights to subscribe to any interests in any of the licences held by Daltsvetmet either now or in the future;
- Any vested interests in either any licences held by Daltsvetmet or any adjacent licences;
- Any right to subscribe to any interests or licences adjacent to those held by Daltsvetmet, either now or in the future.

WAI's only financial interest is the right to charge professional fees at normal commercial rates, plus normal overhead costs, for work carried out in connection with the investigations reported here. Payment of professional fees is not dependent either on project success or project financing.

WAI has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, MERs and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide.

1.3 Qualified Persons and Site Visit

Qualified Persons (QPs) from WAI who have supervised the production of this report are as follows:

- Richard Ellis, BSc, MSc (MCSM), CGeol, EurGeol, FGS, Principal Resource Geologist
- Mark Mounde, BEng, CEng, MIMMM, Technical Director of Mining

These consultants have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. They are considered to be independent Qualified Persons according to the definitions given in NI 43-101. The responsibilities of the QP's for the preparation of the different sections of this Technical Report are shown in Table 1.1.

Table 1.1: Author's Responsibilities

Author	Responsible for Section/s
Richard Ellis	1. Summary; 2. Introduction; 3. Reliance on Other Experts; 4. Property Description and Location; 5. Accessibility, Climate, Local Resources, Infrastructure and Physiography; 6. History; 7. Geological Setting and Mineralisation; 8. Deposit Types; 9. Exploration; 10. Drilling; 11. Sample Preparation, Analyses and Security; 12. Data Verification; 14. Mineral Resource Estimates; 23. Adjacent Properties; 24. Other Relevant Data and Information; 25. Interpretation and Conclusions; 26. Recommendations; 27. References
Mark Mounde	1. Summary; 13. Mineral Processing and Metallurgical Testing; 15. Mineral Reserve Estimates; 16. Mining Methods; 17. Recovery Methods; 18. Project Infrastructure; 19. Market Studies and Contracts; 20. Environmental Studies, Permitting and Social or Community Impact; 21. Capital and Operating Costs; 22. Economic Analysis; 24. Other Relevant Data and Information; 25. Interpretation and Conclusions; 26. Recommendations; 27. References

A site visit of the Nasedkino Project was undertaken by Richard Ellis and Mark Mounde between August 9, 2016 to August 11, 2016, covering aspects related to access and infrastructure, geology, exploration, QAQC, mineralogy, mining, laboratory testwork, processing and environmental and social issues.

Other members of the WAI site visit team included:

- Rina Zolutukhina, Senior Translator

Other WAI consultants who contributed to this report from the UK included:

- Stephen Holley, BSc MSc ACSM MCSM CEng, Senior Mining and Geotechnical Engineer; *Optimisation, Mine design and Scheduling*;
- Stuart Richardson, BSc MSc IEng ACSM MCSM, Senior Mining Engineer; *Mine design;*
- Narina Shorland, BSc MSc MCSM ProfGradIMMM, Senior Mining Engineer; *Mining methods and Infrastructure;*
- Veronica Luneva, Dip Economist, IMC (CFA UK member), Senior Financial Analyst; *Market studies and contracts, Capital and Operating costs and Economic alanysis;* and
- Alison Allen, BSc MSc CEng MIEMA MIEEM, Associate Director; *Environmental and Social review*.

1.4 Units and Currency

All units of measurement used in this report are metric unless otherwise stated. Tonnages are reported as metric tonnes ("t"), precious metal values in grams per tonne ("g/t") or parts per million ("ppm").

Unless otherwise stated, all references to currency or "US$" are to United States Dollars (US$).

2 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by WAI on behalf of Daltsvetmet for which WAI has wholly relied upon the data presented by Daltsvetmet in formulating its opinion. The information, conclusions, opinions, and estimates contained herein are based on:

- Information made available to WAI by Daltsvetmet at the time of preparing this Technical Report including previous internal and external reports (on the varied disciplines) prepared by or for Daltsvetmet on these assets; and
- Assumptions, conditions, and qualifications as set forth in this Technical Report.

In the preparation of this report WAI has relied on the opinion and content of several reports provided to WAI and include:

- The Technical Report prepared by SRK in October 2012
- The Scoping Study prepared by IMC Montan in October 2014

The qualified persons have not carried out any independent exploration work, drilled any holes or carried out any sampling and assaying at the various project areas.

The authors have not reviewed the land tenure situation and have not independently verified the legal status or ownership of the properties or any agreements that pertain to the licence areas. The results and opinions expressed in this report are based on the authors' field observations and assessment of the technical data supplied by Daltsvetmet.

The metallurgical, geological, mineralisation, exploration techniques and certain procedural descriptions, figures and tables used in this report are taken from reports prepared by others and provided to WAI by Daltsvetmet.

The observations, comments and results of this resource estimation represent the opinion of WAI as of February 10, 2017 and are based on the work as stated in the report. Though WAI is confident that the opinions presented are reasonable, a substantial amount of data has been accepted in good faith. Whilst WAI has endeavoured to validate as much of the information as possible, WAI cannot be held responsible for any omissions, errors or inadequacies of the data received. WAI has not conducted any independent verification or quality control sampling, or drilling.

WAI has not undertaken any accounting, financial or legal due diligence of the asset or the associated company structures and the comments and opinions contained in this report are restricted to technical and economic aspects associated with the asset.

WAI has not undertaken any independent testing, analyses or calculations beyond limited high level checks intended to give WAI comfort in the material accuracy of the data provided. WAI cannot accept any liability, either direct or consequential for the validity of information that has been accepted in good faith.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party are at that party's sole risk.

3 PROPERTY DESCRIPTION AND LOCATION

The Nasedkino Gold Project is located in the Mogocha district, Zabaikalsky Krai, Russian Federation. The region was created in March 2008 by the merger of the Chita district ('Oblast') and the Agin-Buryat Autonomous area ('Okrug') following a referendum held on March 11, 2007. The region has a 998km international border with China to the east and an 868km border with Mongolia to the south and internally borders the Irkutsk and Amur regions to the west and east, respectively. The location of the Nasedkino Gold Project within the Russian Federation is shown in Figure 3.1.

Figure 3.1: Location of the Nasedkino Gold Project within the Russian Federation

The Nasedkino Gold Project is located 650km northeast of Chita, the regional capital and 56km northwest of Mogocha, the nearest town. The location of the Nasedkino Gold Project mining licence areas within the Mogocha District are shown in Figure 3.2.

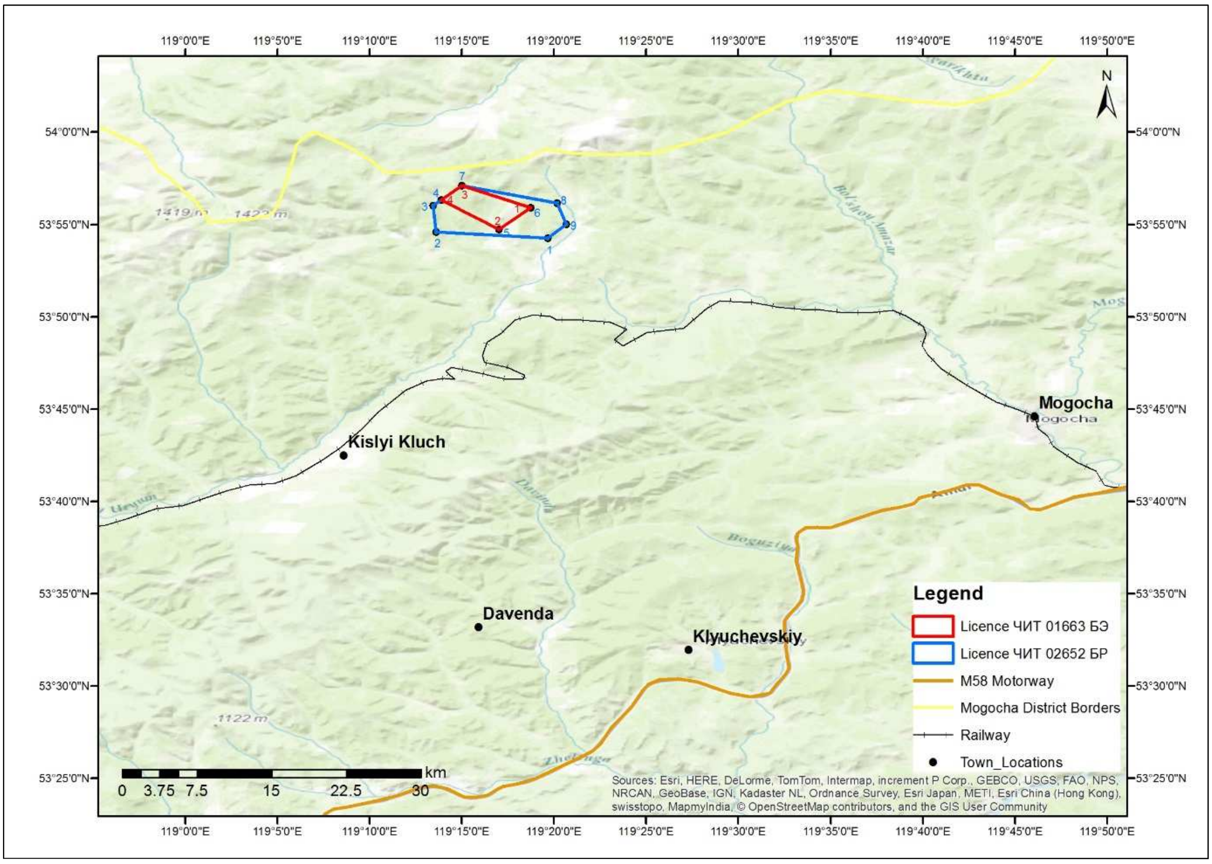


Figure 3.2: Location of the Nasedkino Gold Project

The Nasedkino Gold Project comprises two licences:

- Mining and exploration licence ЧИТ 01663 БЭ (dated December 26, 2005). The licence is valid until December 1, 2025 and covers an area of 10.9km^2 and includes the Pridolinny and Gora 5 deposits.
- Mining and exploration licence ЧИТ 02652 БР (dated April 1, 2016). The licence is valid until December 31, 2036 and covers an area of 19.6km^2 and includes the Zhelanny and Pravoberezhny deposits.

The extent of the licence areas is shown in Figure 3.3.

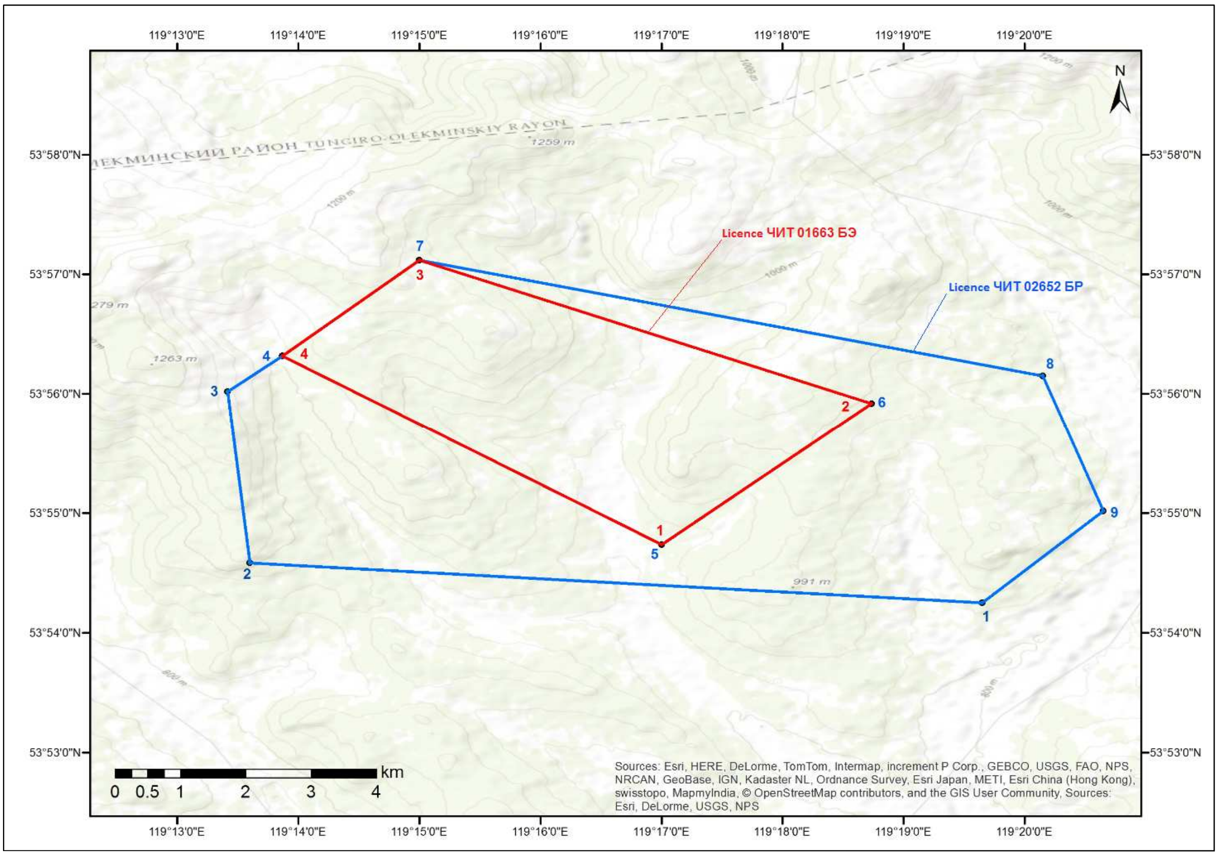


Figure 3.3: Location of Licences at Nasedkino

3.1 Licence ЧИТ 01663 БЭ

Daltsvetmet LLC holds licence ЧИТ 01663 БЭ with the right to use subsoil resources for geological survey and mining of gold, copper, silver and associated components. The licence was issued by the Ministry of Natural Resources of the Russian Federation at an auction on December 26, 2005 and is valid until December 1, 2025. The licence covers area of 10.9km^2 and includes the Pridolinny and Gora 5 deposits. Mining operations are limited to a depth of 250m. A summary of the licence coordinate locations is given in Table 3.1.

Table 3.1: ЧИТ 01663 БЭ Mining and Exploration Licence Coordinates

Coordinate Point	Easting	Northing
1	53°54’44”	119°17’00”
2	53°56’19”	119°13’52”
3	53°57’07”	119°15’00”
4	53°55’55”	119°18’44”

3.2 Licence ЧИТ 02652 БР

Daltsvetmet LLC holds licence ЧИТ 02652 БР with the right to use subsoil resources for geological survey and mining of gold, copper, silver and associated components. The licence was issued by the Ministry of Natural Resources of the Russian Federation at an auction on April 1, 2016 and is valid until

December 31, 2036. The licence covers area of 19.6km^2 and includes the Zhelanny, Zhelanny 3, Pravoberezhny and Glukharinny deposits. The licence supersedes part of an earlier released exploration licence. A summary of the licence coordinate locations is given in Table 3.2.

Table 3.2: ЧИТ 02652 БР Mining and Exploration Licence Coordinates		
Coordinate Point	**Easting**	**Northing**
1	53°54’15”	119°19’39”
2	53°54’35”	119°13’36”
3	53°56’01”	119°13’25”
4	53°56’19”	119°13’52”
5	53°54’44”	119°17’00”
6	53°55’55”	119°18’44”
7	53°57’07”	119°15’00”
8	53°56’09”	119°20’09”
9	53°55’01”	119°20’39”

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Accessibility

The regional transport infrastructure includes the Trans-Baikal and BAM rail roads which provide links to the Far East and Central Russia. The region also has direct rail connections with China and Mongolia. All principal cities and towns are connected by paved and all-season gravel roads. There is an airport in Chita with regular flights to Moscow and other major Russian cities.

The Nasedkino Gold Project is accessed via the recently built M58 road from Chita (650km), the administrative centre of the Zabaikalsky region, or from the town of Mogocha (56km), the administrative centre for the district, which has regular train services to Chita, Moscow and other major Russian cities. From Mogocha the site is accessed via a gravel (all-season) road. Alternatively, the station of Arteushka is located 16km south of the site on the Trans-Baikal railway from here the site can be accessed by an earth road. The small towns and villages surrounding the site are interconnected by earth roads of which the nearest village, Chaldonka is located 15.5km to the south.

4.2 Climate

The climatic conditions in the Zabaikalsky region are extreme continental, exhibiting long cold winters and short cool summers. Historical weather observation records are available from a weather station situated in Mogocha.

The coldest recorded month is January with an average temperature of -28°C; the warmest month is July with an average temperature of +17°C. The overall average annual temperature of the region is -4°C.

The recorded data indicates that the region is subject to relatively low snowfall during the winter period, with the maximum annual precipitation occurring in July (430mm), with variations ranging between 269.4mm (1954) and 691.5mm (1934). Precipitation in the form of snowfall occurs on average between the end of October and March, giving an average snow-cover thickness between 0.15m and 0.20m. The snow-cover typically melts between March and April.

The Nasedkino Gold Project is located within a transitional zone of permafrost and seasonal ground-frost. The maximum depth of ground/permafrost in the region is recorded to be between 30m and 90m below ground level. The depth of seasonal freezing is generally 2.3 – 3.6m below surface.

4.3 Local Resources & Infrastructure

The main industries in the region are alluvial and hard rock gold mining, with forestry enterprises and railway maintenance. The nearest high voltage power line is located in the town of Mogocha, 56km east of the site. Aggregate, dimension stone, construction sand and refractory clays are all available locally. There is no spare labour force in the area, therefore manpower will need to be brought in.

4.4 Physiography

The relief around the site consists of gently undulating terrain, with wide valleys and generally low hills. Absolute elevations in the region vary from 600-1600 m ASL. The terrain at the project site shows moderate relief, however, this does not present any immediate problems to either exploration or mining activities.

5 HISTORY

5.1 Ownership History

Daltsvetmet currently hold two exploration and mining licences at the Nasedkino Gold Project. Exploration and mining licence (ЧИТ 01663 БЭ) was issued to Daltsvetmet on December 26, 2005 whereas exploration and mining licence (ЧИТ 02652 БР) was issued to Daltsvetmet on April 1, 2016. There were no previous owners as historic exploration was conducted by government agencies on State land.

5.2 Exploration History

In the second half of the 19th Century alluvial gold was discovered in the Mogocha region, within the Zheltuga River and its tributaries. In 1860 alluvial gold was mined from the Bolshaya Kudech River, to the south of the project area. Further exploration continued in the early 20th Century. From 1901 to 1910 a joint Anglo-Russian company, the Nerchinskoye Gold Production Society commenced hardrock mining of the Kluchevskoye gold deposit, located 45km to the south of the Nasedkino Gold Project. Following this initial phase, open pit production at Kluchevskoye continued from 1954 to 1965.

The Nasedkino Gold Project was discovered by the Uryumskaya Expedition between 1948 and 1950 in which the western flanks of the Malouryumskoe zone were identified. Additional exploration during 1950 and 1960 discovered the Zhelanny and Pridolinny zones. From 1976 to 1979 further exploration was undertaken on the Pridolinny zone to define the geometry of the mineralisation. The Nasedkino Gold Project was explored by Daltsvetmet during 2006 to 2009 and included an exploration drilling campaign. In this time metallurgical test work was conducted based on 7 laboratory samples and 2 bulk samples collected from exploration trenches and a trial pit opened on the Pridlinny zone. The results of this phase of exploration resulted in the approval of the “TEO of Temporary Conditions”. During 2011 to 2012 verification and infill drilling were undertaken and 3 metallurgical test-work samples were collected and sent for analysis. The work was conducted for the purposes of attaining the “TEO of Permanent Conditions” with the results compiled and submitted to the State Committee of Reserves (GKZ) in 2012. During 2015 verification drilling was undertaken at Zhelanny, Pravoberezhny and Gluckhariny.

5.3 Historical Resources and Reserves

In September 2010, a NI 43-101 compliant Technical Report was filed summarising a Mineral Resource estimate signed off by Micon International. The Mineral Resource estimates were prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards. Mineral Resources are shown in Table 5.1 and were reported within an optimised pit shell based on a $1,200/oz gold price and using a cut-off grade of 0.6g/t Au.

Table 5.1: Nasedkino Gold Project Mineral Resource Estimate (MICON, September 25, 2010)					
Area	**Resource Classification**	**Tonnage (Mt)**	**Grade (g/t Au)**	**Contained Au (t)**	**Contained Au (Koz)**
Pridolinny	Indicated	2.72	1.52	4.13	133
	Inferred	0.32	1.43	0.45	14
Gora 5	Inferred	1.05	2.19	2.31	74
Zhelanny	Inferred	0.97	1.99	1.93	62
Total	**Indicated**	**2.72**	**1.52**	**4.13**	**133**
	Inferred	**2.34**	**2.01**	**4.70**	**151**

Between 2010 and 2012 an additional 431 drill holes (52,634m) were drilled at the property. In October 2012, an updated NI 43-101 compliant Technical Report was filed summarising an updated Mineral Resource estimate and a maiden Mineral Reserve estimate for the Nasedkino Gold Project.

The Mineral Resource estimates were prepared by SRK in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards. Mineral Resources reported by SRK are shown in Table 5.2 and were reported within an optimised pit shell based on a $1,600/oz gold price and using a cut-off grade of 0.55g/t Au.

Table 5.2: Nasedkino Gold Project Mineral Resource Estimate (SRK, May 1, 2012)						
Area	**Resource Classification**	**Tonnage (Mt)**	**Au Grade (g/t)**	**Cu Grade (%)**	**Contained Au (Koz)**	**Contained Cu (Kt)**
Pridolinny	Indicated	10.0	1.1	0.19	360	19.0
	Inferred	1.0	1.1	0.12	36	1.3
Gora 5	Indicated	3.1	1.8	0.10	174	3.0
	Inferred	0.2	0.9	0.14	6	0.3
Zhelanny	Indicated	1.2	3.4	0.05	134	0.6
	Inferred	0.2	1.1	0.31	8	0.7
Pravoberezhny	Inferred	2.5	1.8	0.05	145	1.3
Glukharinny	Inferred	3.1	1.7	0.06	170	1.8
Zhelanny 3	Inferred	0.3	1.5	0.05	15	0.2
Total	**Indicated**	**14.3**	**1.5**	**0.16**	**668**	**22.6**
	Inferred	**7.3**	**1.6**	**0.08**	**380**	**5.6**

A maiden Mineral Reserve estimate based on open pit mining at Nasedkino was also prepared and signed off by SRK and reported within the October 2012, NI 43-101 Technical Report. Mineral Reserves reported by SRK are shown in Table 5.3 and were reported within designed open pits based on a gold price of $1,241/oz and using a cut-off grade of 0.8g/t Au.

Table 5.3: Nasedkino Gold Project Mineral Reserve Estimate (SRK, March 7, 2012)					
Area	**Ore Tonnes (Mt)**	**Au Grade (g/t)**	**Cu Grade (%)**	**Contained Au (Koz)**	**Contained Cu (Kt)**
Pridolinny	7.0	1.18	0.21	267	14.8
Gora 5	2.4	1.83	0.09	144	2.2
Zhelanny	0.9	3.97	0.05	115	0.5
TOTAL	**10.4**	**1.58**	**0.17**	**526**	**17.5**

5.4 Production

No production has been undertaken on the Nasedkino Gold Project to date.

6 GEOLOGICAL SETTING AND MINERALISATION

6.1 Regional Geology

The Nasedkino Gold Project is located along the south - eastern portion of the Siberian Craton, within the Mogocha Mineral Field, within a major tectonic middle Palaeozoic accretionary subduction complex of the south-western part of the regional Aldan-Vitim Shelf. The region comprises metamorphosed, sedimentary-volcanic, sedimentary and magmatic rock formations of various ages. Stratifed rocks occur in abundance in the region and are typically Archean metamorphosed formations. Various granites intrude these metamorphosed formations. Plutons from the Middle Palaeozoic (Olyekminskiy Complex), Permian (Bichurskiy Complex) and Jurassic (Amudzhikano-Sretenskiy and Amanansky Complexes) intrude the fold belt and host a number of gold and molybdenum deposits. The location of the Nasedkino Gold Project within the regional geology is shown in Figure 6.1.

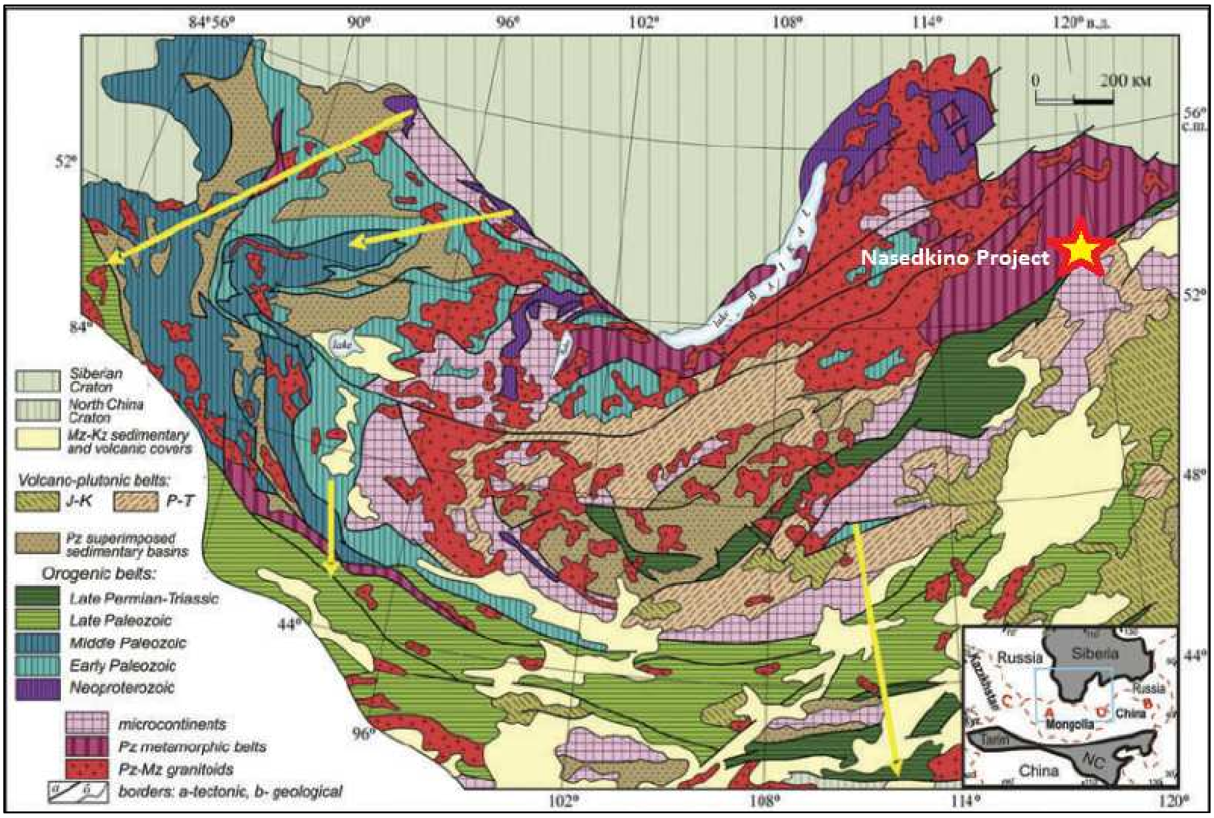


Figure 6.1: Regional Geology of Southern Siberia

6.2 Property Geology

Host rocks comprise Archaen ultra-metamorphic formations including granulites with skarn mineral assemblages and gneisses that have been folded into isoclinal northwest trending folds. Typical lithologies include pyroxene-garnetiferous granulites (skarnoid) and quartz-biotite-plagioclase gneisses often containing silliminite, hypersthene, graphite, high-aluminium crystalline schists and amphibolites. Of the two host lithologies the most significant mineralisation is hosted by the skarnoids.

The Archaean rocks are folded into isoclinal folds along a northwest strike and steeply dip to the northeast and are intruded by granitoid and porphyry complexes of Middle and Late Jurassic age. The Late Jurassic intrusions are of medium-acidic composition and are the source of mineralisation bearing hydrothermal fluids.

The earliest metasomatic alteration phase involved high-temperature propylitisation of the granulites. Lower temperature alteration is present in peripheral areas consisting of propylitic alteration assemblages of epidote-chlorite-carbonate. A final low temperature argillic alteration phase, comprising kaolinite-quartz-calcite is also evident.

The major fault zones observed in the area strike northwest (330-345°) and dip steeply (70-80°) to the northeast, concordant with the metamorphic strata. The fault zones acted as feeders for the mineralised hydrothermal fluids while the higher permeability metamorphic strata comprised suitable fluid migration pathways. The mineralised zones therefore lie conformably with the metamorphic strata and occur as lenticular zones of hydrothermal alteration up to 1.2km in length and between 1-10m in thickness. A series of late stage faults are present and strike northeast (60-70°) and dip sub-vertically (80-90°). The late stage faults off-set the mineralised zones.

Gold mineralisation is confined only to the zones of hydrothermal alteration. The mineralised zones are not visually distinct from the host lithology and can only be identified by sampling data. Gold mineralisation is predominantly associated with pyrite and less frequently with chalcopyrite and infrequently with galena, sphalerite, grey copper ores, arsenopyrite, molybdenite, magnetite, ilmonite, chalcocite and other sulphides. Gangue minerals include quartz, plagioclase, and potassium feldspar with accessory minerals including biotite, pyroxene, sericite, chlorite and garnet. Sulphide content varies from 1-2% to 20-30% and typically averaging 5-10%. No direct correlation between sulphide content and gold grade is observed, although most of the gold (over 60%) is associated with sulphides, mainly in intergrowth with pyrite and to a much lesser extent with chalcopyrite, grey copper ores and with bismuth minerals. The remaining gold is distributed as free gold in quartz with limonite. Gold grain size varies from 0.002mm - 1.2mm and typically averaging 0.02mm – 0.08mm.

A surface oxidation zone is present and typically only to a depth of 5m, increasing to a depth of 10-20m where faults are present. The limited depth of oxidation results in no significant oxide resources.

The location of the Nasedkino Gold Project and the property geology is shown in Figure 6.2.

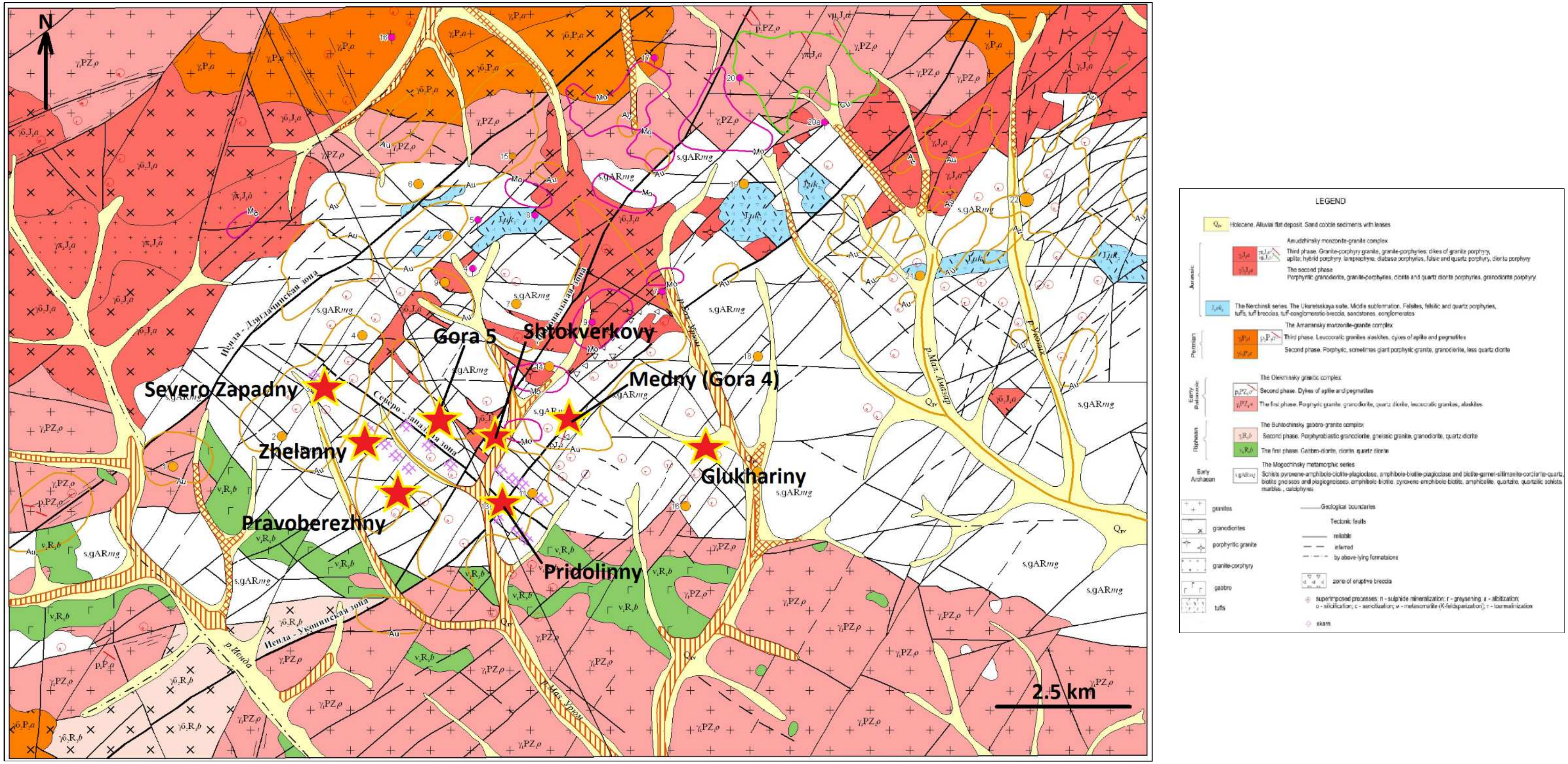


Figure 6.2: Property Geology of the Nasedkino Gold Project Showing Locations of Deposits (Pridolinny, Gora 5, Zhelanny and Pravoberezhny) and Advanced Exploration Prospects (Glukharinny, Severo Zapadny, Shtokverkovy and Medny (Gora 4)

6.3 Description of Mineralised Zones

The Nasedkino Gold Project consists of four deposits and four advanced exploration prospects. The four deposits consist of Pridolinny, Gora 5, Zhelanny and Pravoberezhny and are located on two parallel mineralised corridors which extend for more than 2.5km in a northwest direction. Zhelanny and Pravoberezhny are located on the western most of the mineralised corridors. Pridolinny and Gora 5 are located on a mineralised corridor located 700m to the east of this. These two main mineralised corridors exhibit similar mineralisation, geology and structure. The four advanced exploration prospects consist of include Severo Zapadny, Shtokverkovy, Medny (Gora 4) and Glukharinny.

7 DEPOSIT TYPES

7.1 Mineral Deposit Type

The Nasedkino Gold Project has the characteristics of a low-sulphidation epithermal deposit based on the classification of Corbett (2002). The deposit is interpreted (although not confirmed) to be associated with a deeper buried Cu-Au porphyry intrusion which is likely the ultimate metal source for epithermal mineralisation. Low sulphidation epithermal Au deposits characteristically develop from dilute circulating meteoric waters in dilatant structural settings and are distinguished as the group with a closer relationship to intrusive source rocks. Low sulphidation epithermal Au deposits are sub-classified by decreasing crustal level as: quartz-sulphide Au ± Cu, carbonate-base metal Au (including polymetallic Au-Ag), and epithermal quartz Au-Ag. Based on the main minerals, gangue minerals and alteration exhibited, the Nasedkino Deposit is interpreted as a quartz-sulphide Au ± Cu deposit and formed at deeper crustal levels. The model for mineral deposition is shown in Figure 7.1.

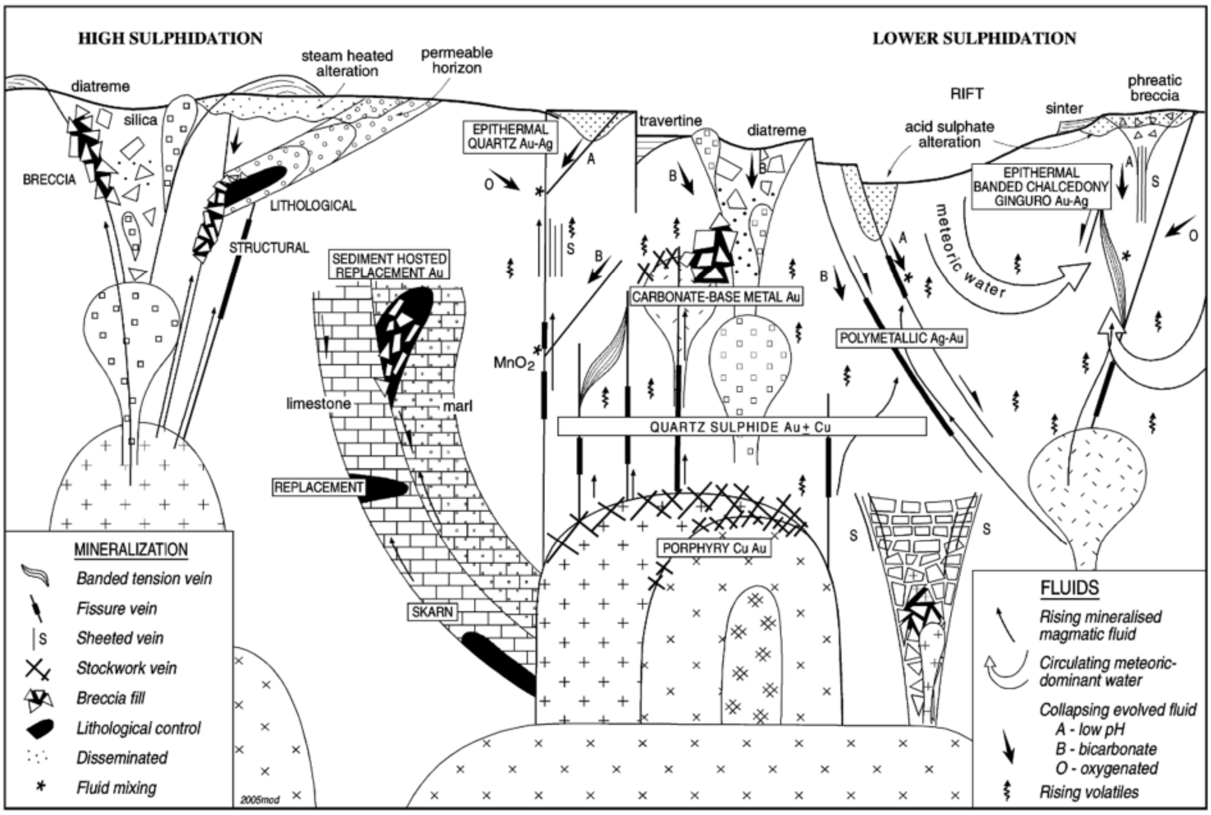


Figure 7.1: Classification of Epithermal and Base Metal Deposits by Corbett (2002)

7.2 Exploration Model

Based on the deposit type the exploration model comprises additional quartz sulphide Au ± Cu deposits and skarns, intrusion related gold systems (IRGS) and Cu-Au porphyry deposits.

8 EXPLORATION

Exploration work conducted on the Nasedkino licence area focused on evaluating the historical mineral resources and conducting additional exploration.

During 2006-2008, exploration work was conducted by Vostokgeologiya on behalf of Daltsvetmet and comprised the initial prospecting and evaluation stage of the Project. The programmes involved a variety of exploration activities that included geological mapping, topographic-geodetic surveying, grid-based soil sampling, diamond drilling and down-hole logging, trenching, and ground and airborne geophysics. Ground geophysical surveys were subcontracted to Tellur SV of Saint Petersburg, while the airborne geophysical survey was undertaken by Noril´sky Branch of the Saint Petersburg All Russian Geological Institute. In addition, a shallow open pit was also excavated on the Pridolinny deposit with mining of some 65,390m^3. The pit comprised a total of 4 benches, with the height of the first three benches being 4m and the fourth bench being 2m. Channel sampling within the pit was undertaken using a grid of 4 x 5m for a total of 3,952m. Seven bench-scale test samples and two bulk samples were collected from the trial pit and were used for metallurgical test work and mineralogical and petrological studies. A total of 32,102m of trenching (including Pridolinny trial pit channel samples) from 131 trenches were driven using excavators to depths of around 1-2m depending on thickness of diluvium. The trenching grid was typically around 100-200m. A total of 352 diamond drill holes for 36,612m were drilled during this period and were focussed predominantly on Pridolinny, Gora 5 and Zhelanny deposits. The drilling grid was typically 100 x 50m with infill drilling in the central part of the Pridolinny deposit at 50 x 50m and 25 x 25m.

During 2010-2012, exploration work was conducted by TOMS-Engineering on behalf of Daltsvetmet and comprised the exploration stage of the Project. Topographic surveying at 1:2,000 scale was undertaken along with additional metallurgical testwork, trenching and drilling. A total of 11,438m of trenching from 83 trenches were excavated. Trenching was undertaken to provide a trench grid of 50-100m. A total of 424 diamond drill holes for 51,822m were drilled during this period. The programme predominantly included verification and infill drilling at Pridolinny, Gora 5 and Zhelanny deposits and additional exploration drilling at Zhelanny, Pravoberezhny and Glukharinny deposits. The drilling grid at this stage was typically 50 x 25m.

During 2015, additional exploration work was undertaken by Daltsvetmet and comprised infill drilling at Zhelanny, Pravoberezhny and Glukharinny. A total of 27 diamond drill holes for a total of 2,200m were drilled.

A summary of exploration undertaken at the Nasedkino Gold Project is shown in Table 8.1.

Table 8.1: Summary of Exploration at the Nasedkino Gold Project					
Description	**Unit**	**2006-2008**	**2010-2012**	**2015**	**Total**
Topographic -Geodetic Survey	km^2	10.9	-	-	10.9
Topographic survey, scale 1:2000 source: ZabAGP 2011 tech report, Zamkov 2015 tech report	km^2	-	16.3	1.4	17.7
Ground Magnetics	km	113.9	-	-	113.9
Ground IP/Resistivity Gradient Array Survey	km	113.9	-	-	113.9
Airborne Magnetics and Radiometrics	km	936	-	-	936
Geochemical Sampling	km^2	8.8	-	-	8.8
Geochemical Soil Samples	number	1,148	-	-	1,148
Metallurgical and Bulk Samples	number	9	34	-	43
Trenching	m	32,102	11,438	-	43,540
	number	131	83	-	214
Trial Pit	m^2	65,390	-	-	65,390
Drill Holes	m	37,939	52,634	2,255	92,829
	number	352	431	27	810

The locations of all drill holes and trenches at the Nasedkino Gold Project are shown in Figure 8.1 along with licence areas.

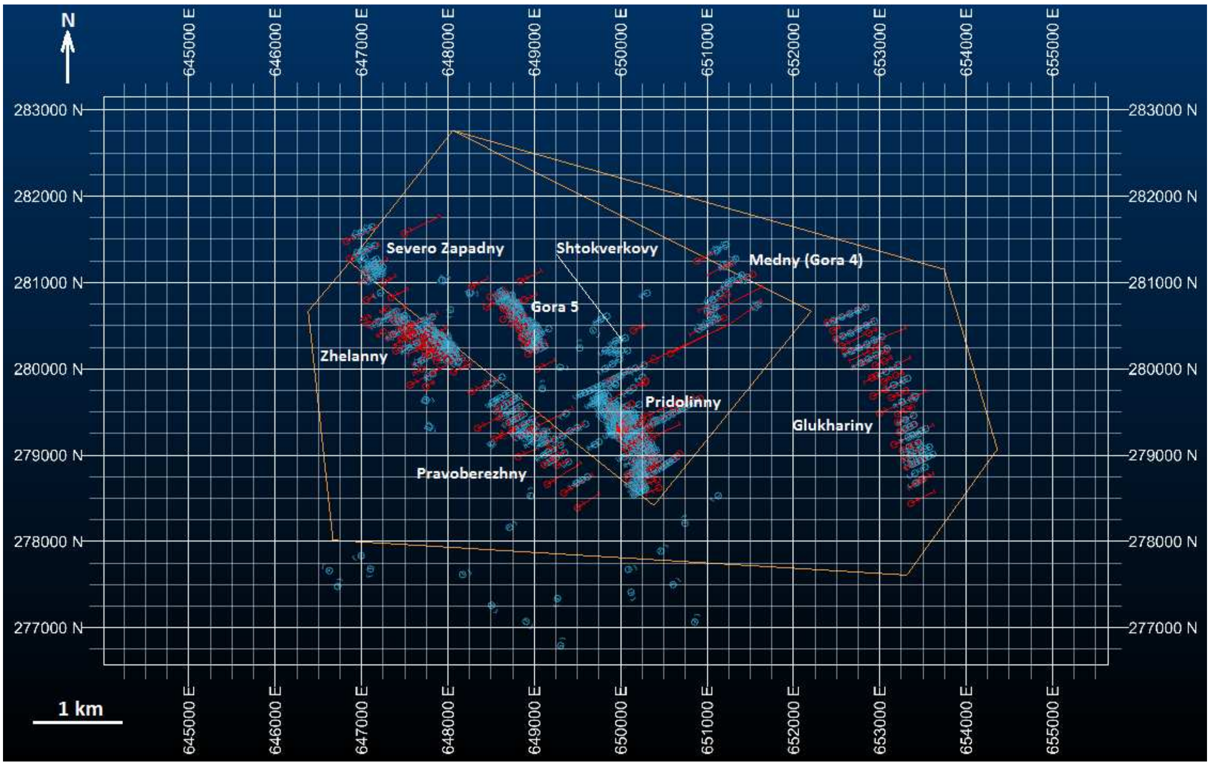


Figure 8.1: Plan View Showing Location of Drill Holes and Trenches at the Nasedkino Gold Project (Trenches in Red, Drill Holes in Blue) and Extent of Mining Licences (Orange Perimeters)

9 DRILLING

As of February 10, 2017, a total of 810 drill holes totalling 92,825m have been completed at the Nasedkino Gold Project. All drilling was conducted by diamond core drilling. Three drilling campaigns have been undertaken and comprise a 2006 to 2008 campaign, a 2011 to 2012 campaign and a 2015 campaign. During the 2006-2008 campaign diamond drilling typically commenced with HQ (63.5mm) diameter core in the oxidised material before continuing with NQ (47.6mm) diameter core once competent rock was encountered. During the 2010-2012 campaign and the 2015 campaign all drilling was conducted using HQ diameter core. All of the drilling completed at the Nasedkino Gold Project has been carried out by contract drilling service companies using Boart Longyear (LM-55, LM-75 and LF-90) drill rigs. A summary of the drilling campaigns is given below and includes information contained in the NI 43-101 Technical Report prepared by SRK in 2012.

9.1 2006-2008 Drilling Campaign

During the 2006-2008 drilling campaign a total of 352 drill holes totalling 37,939m were drilled at the Nasedkino Gold Project. All drilling and exploration work was carried out by Vostokgeologiya on behalf of Daltsvetmet. The drilling campaign was designed to test and delineate mineralised material, to obtain metallurgical samples and to gather geotechnical and environmental information.

9.2 2011-2012 Drilling Campaign

During the 2010-2012 drilling campaign a total of 431 drill holes totalling 52,634m were drilled at the Nasedkino Gold Project. All drilling and exploration work was carried out by Urangeo and supervised by TOMS-Engineering (TOMS) on behalf of Daltsvetmet.

9.3 2015 Drilling Campaign

During the 2015 drilling campaign a total of 27 drill holes totalling 2,255m were drilled at the Nasedkino Gold Project. Drilling was carried out by Urangeo and supervised by Daltsvetmet. Verification drilling was undertaken at Zhelanny high grade zone while additional exploration drilling was undertaken at Pravoberezhny and Glukharinny.

9.4 Summary of Drilling

A summary of the drilling to date is shown in Table 9.1.

Table 9.1: Summary of Drilling at the Nasedkino Gold Project

Type of Drilling	Deposit																	
	Pridolinny		Gora 5		Zhelanny		Pravoberezhny		Glukharinny		Medny/Gora4		Shtokverkovy		Severo Zapadny		Total	
	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)	No. of Drill Holes	Length (m)
2006 to 2008 Drilling Campaign																		
Exploration Drilling	111	8,287	16	2,201	18	2,927	14	2,176	-	-	11	1,755	15	1,779	25	2,576	210	21,701
Confirmation Drilling	87	13,606	-	-	-	-	-	-	-	-	-	-	-	-	-	-	87	13,606
Verification Drilling	11	322	-	-	-	-	-	-	-	-	-	-	-	-	-	-	11	322
Twin holes	36	1,397	-	-	-	-	-	-	-	-	-	-	-	-	-	-	36	1,397
Hydrogeological Drilling	3	402	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3	402
Metallurgical Drilling	5	511	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5	511
Sub-Total	**253**	**24,525**	**16**	**2,201**	**18**	**2,927**	**14**	**2,176**	**-**	**-**	**11**	**1,755**	**15**	**1,779**	**25**	**2,576**	**352**	**37,939**
2010 to 2012 Drilling Campaign																		
Exploration Drilling	126	12,163	64	6,063	72	8,533	49	7,831	48	8,213	6	1,208	-	-	4	809	369	44,820
Hydrogeological Drilling	3	400	2	200	2	200	-	-	-	-	-	-	-	-	-	-	7	800
Metallurgical Drilling	5	270	9	640	11	728	-	-	-	-	-	-	-	-	-	-	25	1,638
Sterilisation Drilling	28	4,773	-	-	-	-	-	-	-	-	-	-	-	-	-	-	28	4,773
Structural Drilling	-	-	1	304	1	300	-	-	-	-	-	-	-	-	-	-	2	604
Sub-Total	**162**	**17,606**	**76**	**7,207**	**86**	**9,761**	**49**	**7,831**	**48**	**8,213**	**6**	**1,208**	**-**	**-**	**4**	**809**	**431**	**52,634**
2015 Drilling Campaign																		
Confirmation Drilling	-	-	-	-	11	774	6	643	10	838	-	-	-	-	-	-	27	2,255
Sub-Total	**-**	**-**	**-**	**-**	**11**	**774**	**6**	**643**	**10**	**838**	**-**	**-**	-	-	-	-	**27**	**2,255**
Grand Total	**415**	**42,131**	**92**	**9,408**	**115**	**13,462**	**69**	**10,650**	**58**	**9,051**	**17**	**2,963**	**15**	**1,779**	**29**	**3,385**	**810**	**92,829**

9.5 Drill Core Recovery

Core recovery is measured using the standard Russian methodology of calculating the volumetric mass of core in the interval for a given lithology. The core recovery by this gravimetric method can then be compared with the manually measured linear core recovery of the same core interval. Typically core recovery of the gravimetric measurement and the linear measured intervals should differ by only a very small percentage.

During 2006-2008 core recovery was measured for 1,627 intervals of 1m core. Gravimetric measurements returned a core recovery of 97.53% while the measured linear core recovery returned a core recovery of 97.49%. During 2010-2012 core recovery was measured for 2,651 samples. Gravimetric measurements returned a core recovery of 90% while the linear core recovery returned a core recovery of 98%.

WAI consider that there are no material issues resulting from the reported drill core recovery and that the reported core recovery is acceptable use in resource estimation.

9.6 Extent of Drilling

To date, drilling has defined two main zones of mineralisation over a strike length of 3,000m and to depths of 220m from surface. The most westerly of these mineralised zones includes the deposits of Zhelanny and Pravoberezhny. The deposits of Gora 5 and Pridolinny are located 700m to the east of this zone.

9.7 Drill Hole Collar Surveys

Surface drill hole collars were surveyed using a Sokkia SET 510 total station. Elevation was determined using three point intersections and combined intersections and theodolite traverses. The local coordinate system and the Baltic elevation system were used for all drill hole collar surveys.

9.8 Downhole Surveys

Down hole surveys were undertaken on all drill holes exceeding 100m in depth. Drill holes were surveyed using a Tropari-like survey instrument (Gyromagnetic Inclinometer AEM-36) with readings taken every 20m. All downhole surveys were carried out by drilling contractor personnel.

9.9 Drill Sections

Drill sections are orientated at 240° and are orientated perpendicular to the general strike of the deposits. The deposits dip sub-vertically to the northeast and drill holes are inclined at -60° to the southwest along the drill sections. Drill sections at Zhelanny, Gora 5 and Pridolinny are spaced generally at 50m spacings (reducing to 25m and 10m in the central part of Pridolinny). Along section spacing at these deposits is generally 25m (reducing to 10m in the central part of Pridolinny). Drill sections at Pravoberezhny are 50m in the centre of the deposit with 100m spacing beyond this.

10 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The sampling methodology, preparation and analyses differ depending on whether the sample is drill core or a trench sample. During the different drilling and sampling campaigns various sampling procedures were used. The 2006 -2008 campaign was carried out by a subcontractor (Vostokgeology). The 2010-2012 campaign was supervised by TOMS-Engineering (TOMS) and undertaken by Daltsvetmet geologists. The 2015 campaign was undertaken by Daltsvetmet geologists.

10.1 Trench Sampling

After levelling and cleaning the trenches, samples are taken along the base of the trench in sequence from hangingwall to footwall perpendicular to the mineralised zone. The depth of the trench samples is generally 5cm with a width of 10cm. The trench sample length is variable (generally from 0.3m to 1.0m) depending on lithology, sulphidation or alteration. Samples, comprised of fragments, chips and mineral dust, are extracted using a chisel, hammer, pick or saw blade. The obtained sample is deposited into a heavy duty sample bag and labelled with the trench ID, from and to interval and sample number. Trenches are photographed and then backfilled on completion of the sampling. Samples are then dispatched to the assay laboratory.

10.2 Core Sampling

Drill core is removed from the core barrel and placed into wooden core boxes. Sample intervals are recorded on the core box and on wooden separators used to define the sample interval. Core boxes are transported from the drill sites to the on-site logging facilities.

The drill core is photographed and core recovery measurements are taken for each sample of core. During the 2010-2012 campaign a second photograph of the wet drill core was also taken after cutting of the drill core. The drill core is geologically logged. A geologist is responsible for determining and marking the intervals to be sampled, selecting them based on geological, sulphidation, alteration or structural logging. Sampling is undertaken from top to bottom of the drill hole. During the 2006-2008 campaign core sampling was undertaken only for the identified mineralised intervals. Non-mineralised intervals (based on the geological logging) were sampled by chip sampling over a 3–4m interval. Continuous sampling for the entire length of the drill core was undertaken during the 2010-2012 campaign and the 2015 campaign. Sampling lengths range between 20cm to 1.5m (mostly of 1m). Splitting of the core is performed by diamond saw in such a way that two equal halves of core are produced.

Once the core has been split, half the sample is placed in a heavy duty sample bag. A sampling card with the appropriate information is inserted with the core. Samples are then dispatched to the assay laboratory. For each drill hole a hard copy geology logbook and drilling book are kept. The details of the logging, sampling and drilling are recorded into Microsoft® Excel spreadsheets.

10.3 Bulk Density Determination

Samples submitted for density measurements were based on lithology. Density measurements were taken by measuring the sample weight in air then suspending the sample in water and measuring the weight again. A total of 1,683 density measurements are contained within the database.

10.4 Sample Preparation

A summary of the sample preparation facilities used during the different drilling and sampling campaigns is given in Table 10.1.

Table 10.1: Summary of Sample Preparation Facilities used for Nasedkino Gold Project

2006 to 2008 Campaign	
2006	LITsIMS (Chita) and Irgiredmet
2007-2008	Vostokgeology and ALS Chemex (Chita)
2010 to 2012 Campaign	
Sample preparation by Daltsvetmet using Rocklabs on site mobile sample preparation facility	
ALS Chemex (Chita)	
2015 Campaign	
ALS Chemex (Chita)	

10.4.1 2006 to 2008 Campaign

In 2006 sample preparation was carried out by LITsIMS laboratory (Laboratory Research Centre of Mineral Studies) located in Chita, and by Irgiredmet (Irkutsk Research Institute of Precious and Rare Metals and Diamonds), located in Irkutsk. From 2007 to 2008 sample preparation was carried out by Vostokgeology and ALS Chemex laboratory located in Chita. A summary flow sheet for the sample preparation of trench samples and drill core samples during the 2006 to 2008 campaign is shown in Figure 10.1 and Figure 10.2, respectively.

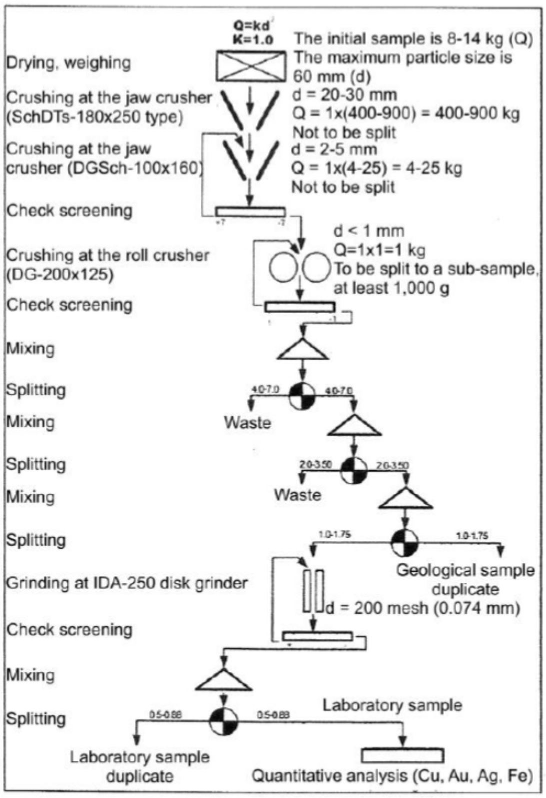


Figure 10.1: Flow Sheet for Sample Preparation of Trench Samples During 2006 to 2008 Campaign

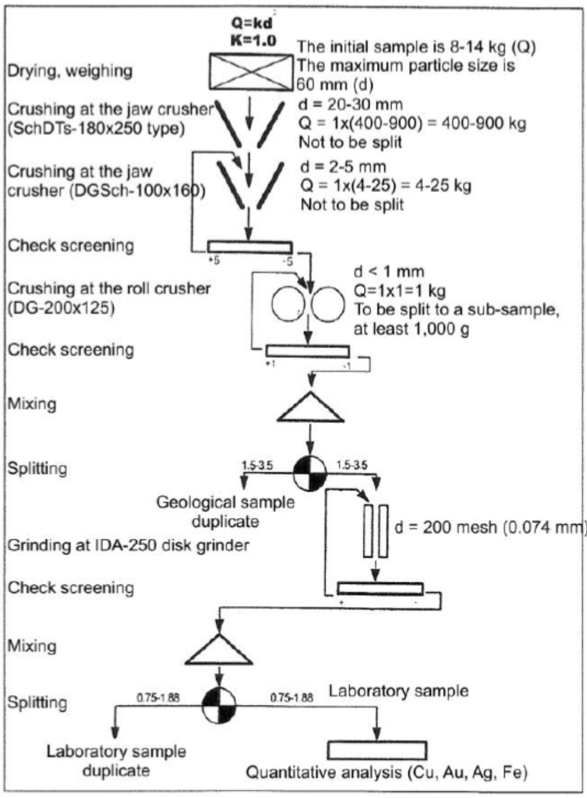


Figure 10.2: Flow Sheet for Sample Preparation of Drill Core Samples During 2006 to 2008 Campaign

10.4.2 2010 to 2012 Campaign

During 2010-2012 sample preparation was carried out on-site at Nasedkino by Daltsvetmet in a mobile sample preparation facility supplied by Rocklabs. A summary flow sheet for the sample preparation of trench samples and dill core samples at the on-site facility during the 2010-2012 campaign is shown in Figure 10.3 and Figure 10.4, respectively. Due to an excess of samples produced during this campaign, some whole core samples were sent for sample preparation at ALS Chemex, located in Chita. The sample preparation undertaken by ALS Chemex involved crushing of the entire sample to 1-2mm, splitting using Jones splitter to 1kg and pulverising to 0.074mm using a LM2 pulveriser.

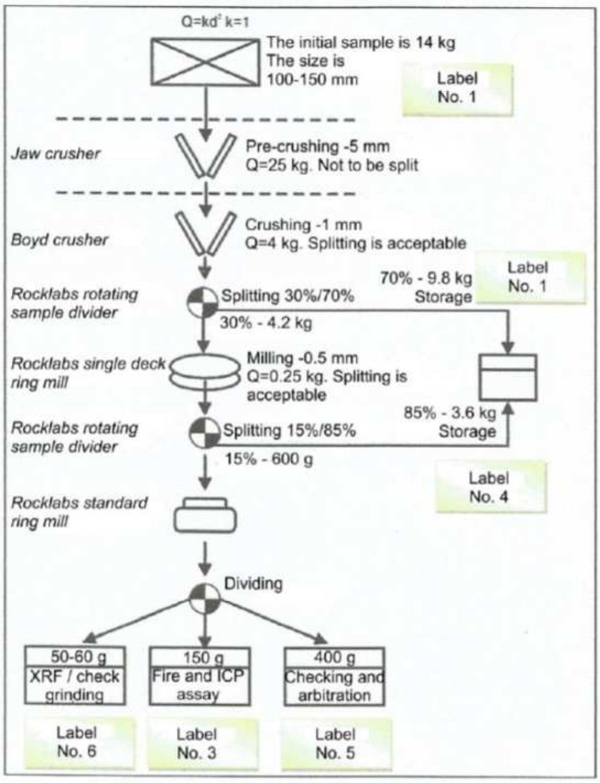


Figure 10.3: Flow Sheet for Sample Preparation of Trench Samples at the On Site Facility at Nasedkino during 2010-2012 Campaign

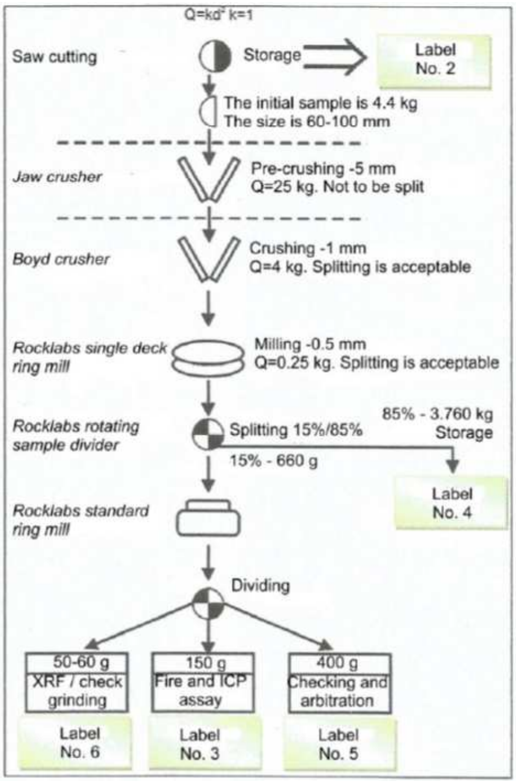


Figure 10.4: Flow Sheet for Sample Preparation of Trench Samples at the On Site Facility at Nasedkino during 2010-2012 Campaign

10.4.3 2015 Drilling Campaign

During 2015, all sample preparation was carried out by ALS Chemex, located in Chita using whole core samples. The sample preparation undertaken by ALS Chemex again involved crushing of the entire sample to 1-2mm, splitting using Jones splitter to 1kg and pulverising to 0.074mm using a LM2 pulveriser.

10.5 Analyses

A summary of the analytical laboratories used during the different drilling and sampling campaigns is given in Table 10.2. Assaying was predominantly undertaken by fire assay with atomic absorption (AA) finish, however some check samples on unmineralised core were also undertaken using chip samples taken from the core which were subsequently analysed by XRF. This provided a means of checking the core sampling, however the quality of assays produced by XRF are not considered appropriate for use in resource estimation. The samples assayed by XRF, however are clearly labelled in the assay database and can be separated from those samples assayed by AA.

Table 10.2: Summary of Sample Preparation Facilities used for the Nasedkino Gold Project

Routine Assaying	Check Assaying	Number Routine Samples Analysed
	2006 to 2008 Campaign	
Irgiredmet	LITsIMS	21,017
Daltsvetmet	LITsIMS	12,897
	ZabNII	
	ALS Chemex (Chita)	
ALS Chemex (Chita)	ZabNII	8,905
LITsIMS	ZabNII	3,442
ZabNII	LITsIMS	5,087
	Sub-Total	**51,348**
	2011 to 2012 Campaign	
ALS Chemex (Chita)	Alex Stewart (Moscow)	35,093
	2015 Campaign	
ALS Chemex (Chita)	ALS Chemex (Chita)	2,398
	Grand Total	88,839

10.6 Sample Security and Chain of Custardy

Sample collection and transportation of drill core and trench samples is undertaken by Daltsvetmet geology department.

Exploration core boxes are transported to the core logging facilities located at the on-site compound where there is sufficient room to layout and examine several drill holes at a time. Once logging and sampling have been performed, the core is transferred to the permanent storage facility located in the same compound. The drill core boxes are covered. The on-site storage facility is dry with internal lighting and metal shelving for core storage. Coarse reject material from exploration drill core and trench sampling along with pulp duplicate material are stored in the same warehouses as the exploration drill core.

Where whole core sampling has been undertaken for additional metallurgical test work or whole core samples have been sent to the ALS Chemex laboratory (Chita) for sample preparation and analyses, no remaining core is available. Pulp duplicates of these samples, drill core photographs taken during core logging and the ALS Chemex laboratory assay certificates of these samples are however available.

10.7 Quality Assurance and Quality Control Programmes

The implementation of a quality assurance / quality control (QAQC) programme is current industry best practice and involves establishing appropriate procedures and the routine insertion of certified reference material, blanks and duplicates to monitor the sampling, sample preparation and analytical process. Analysis of QAQC data is made to assess the reliability of sample assay data and the confidence in the data used for the estimation.

10.7.1 2006-2008 Campaign

QAQC procedures during the 2006–2008 campaign were carried out in line with Russian industry standards and included internal and external pulp duplicate checks for drill core samples. Approximately 5% of the pulps were re-assayed in the same laboratory and then sent to an external laboratory as an independent check. No quality control procedures were undertaken on the trench sampling during 2006 to 2008. No blanks or CRM's were used in the 2006-2008 campaign and as such QAQC procedures, although in line with Russian standards, were not considered to be in line with western industry standard procedures.

To address this, in 2010, 1% of the database was subsequently resubmitted by Daltsvetmet for re-analysis and included 358 pulp duplicates, 11 certified reference materials and 7 blanks for a total of 376 samples. Re-analysis samples were submitted to ALS Chemex laboratory (Chita).

The duplicate re-analysis was reviewed by WAI and a summary of which is shown in Table 10.3.

Table 10.3: Summary of WAI Review of Duplicate Re-Analysis of 2006-2008 Drilling Campaign by Assay Laboratory

Laboratory	Number Samples Submitted for Re-Analysis	Number of Samples >0.7g/t Au	WAI QAQC Review Comments
Irgiredmet	253	97	Acceptable.
Daltsvetmet	45	10	Limited number of samples for analysis. Issues with assays above 1g/t Au.
LITsIMS	34	12	Limited number of samples for analysis. Issues with assays above 1g/t Au.
ZabNII	23	5	Insufficient number of samples for analysis.
ALS Chemex (Chita)	5	0	Insufficient number of samples for analysis.

Based on this review WAI consider the Irgiredmet laboratory assays to be acceptable. However, assays from Daltsvetmet, LITsIMS, ZabNII and ALS Chemex (Chita) laboratories produced during the 2006-2008 campaign have issues associated either with the assaying or suffer from a lack of duplicate sample submissions. As such, WAI does not consider that assays from these laboratories from the 2006-2008 campaign are sufficiently supported for use in Mineral Resource estimation. It is recommended that future additional duplicate sample submissions (including standard and blank samples) should be undertaken to attempt to support the assaying of these laboratories.

The results of the internal duplicates, external duplicates, CRM's and blanks analysis by the Irgiredmet laboratory are shown in Figure 10.5, Figure 10.6, Figure 10.7 and Figure 10.8. The results, while acceptable, do however indicate a variable level of precision which has been accounted for during Mineral Resource classification.

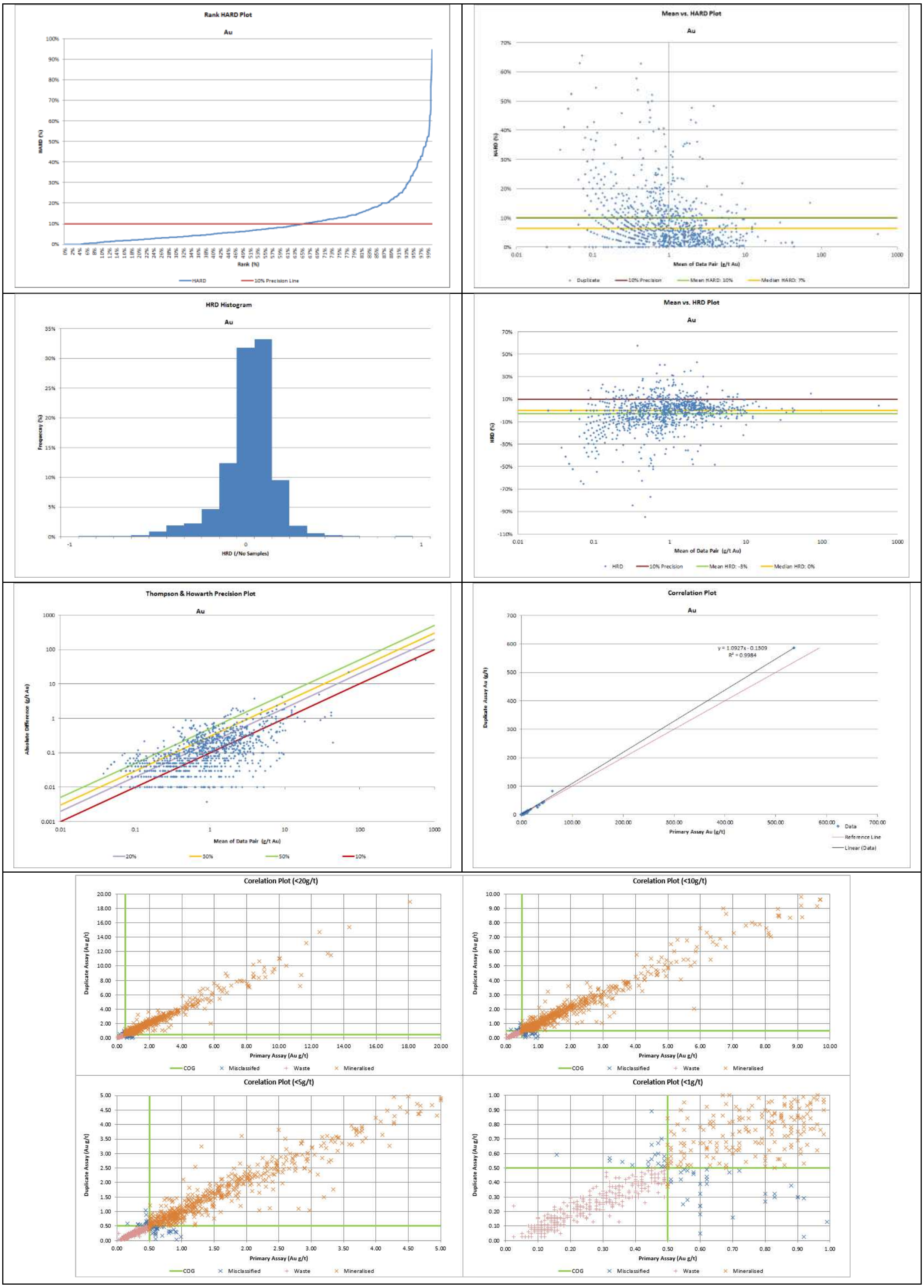


Figure 10.5: 2006-2008 Drilling Campaign Internal Pulp Duplicate Analysis – Irgiredmet

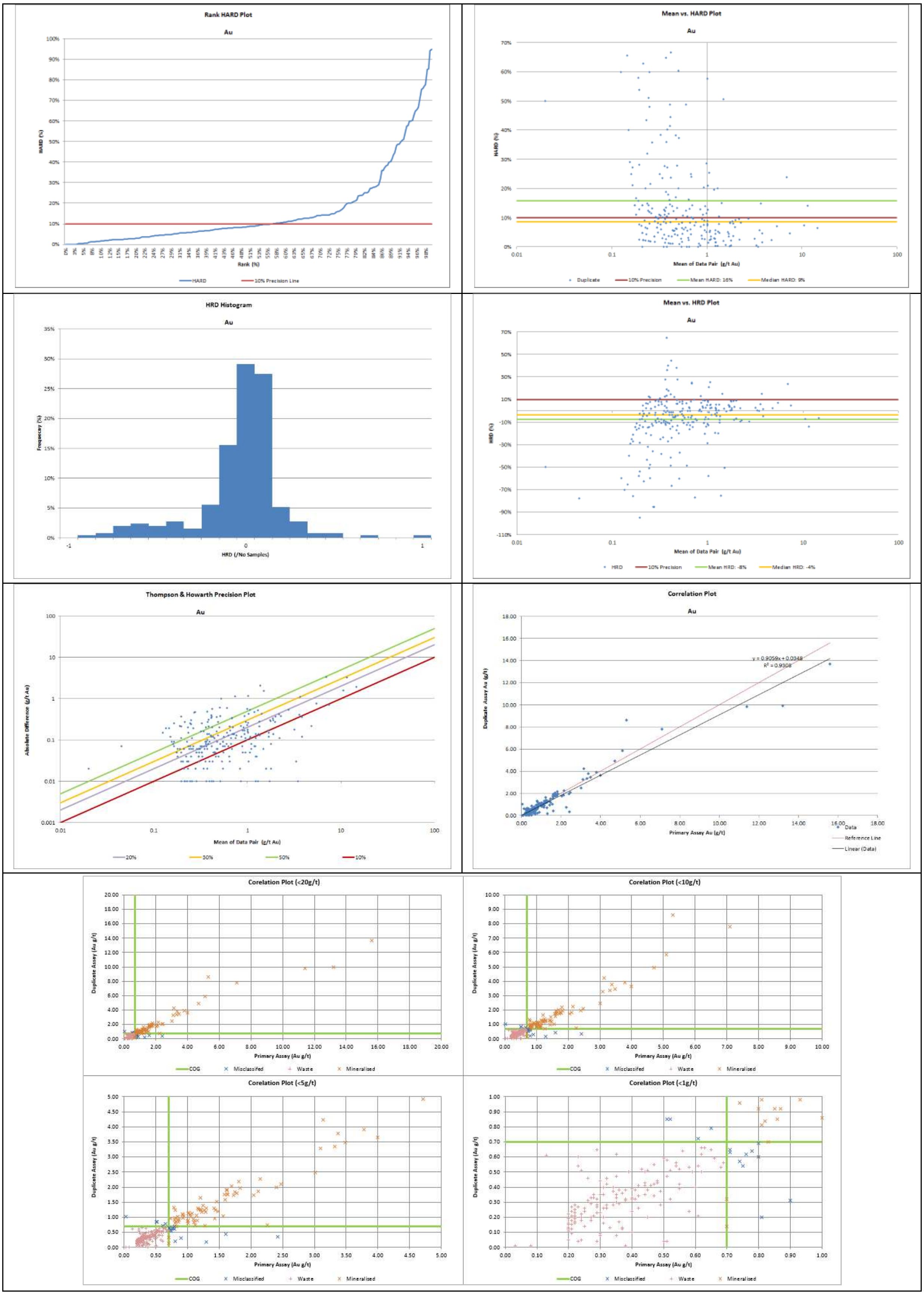


Figure 10.6: 2006-2008 Drilling Campaign External Pulp Duplicate Analysis - Irgiredmet vs ALS Chemex (Chita)

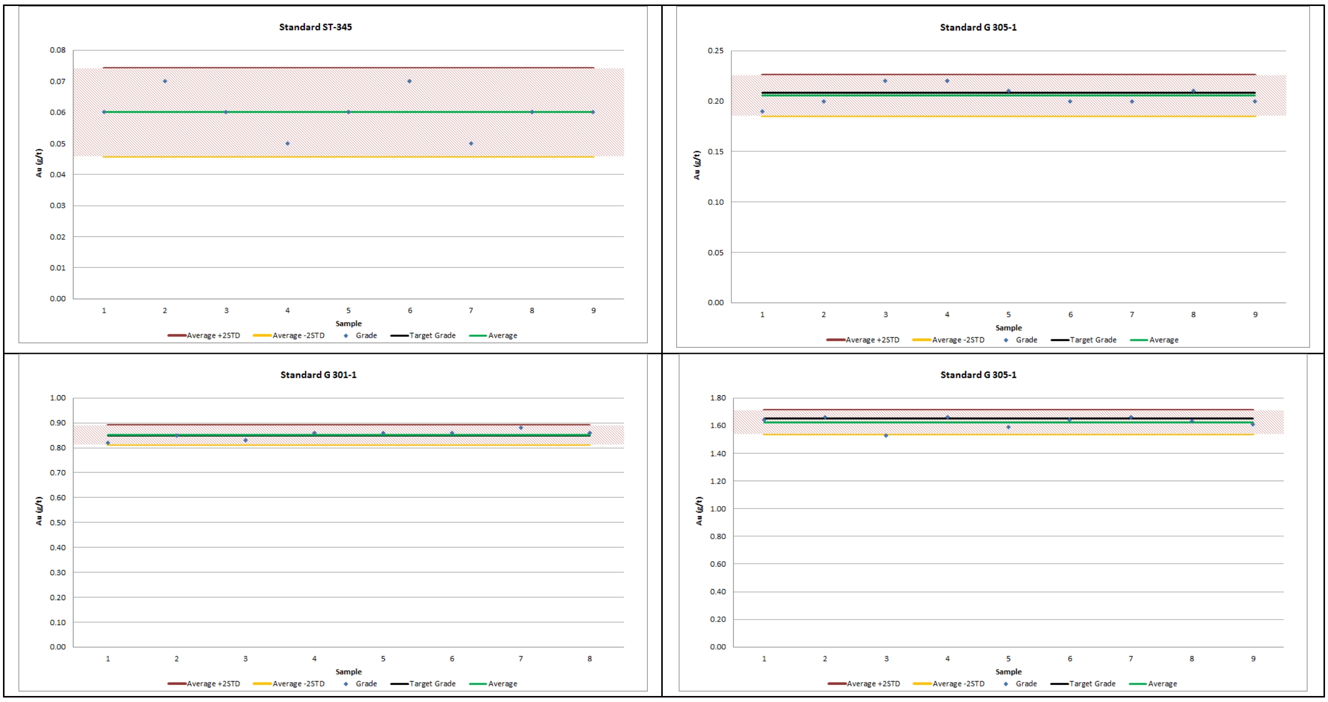


Figure 10.7: 2006-2008 Drilling Campaign CRM Analysis by ALS Chemex (Chita)

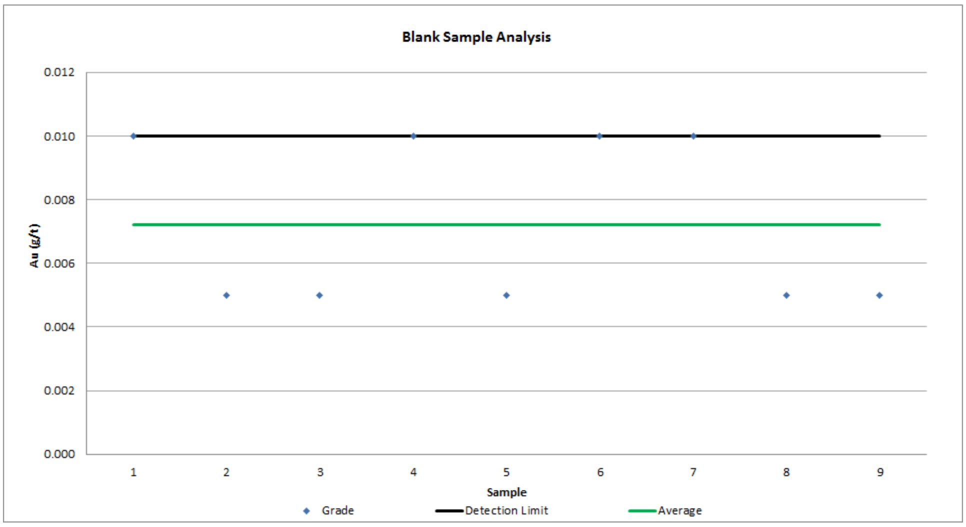


Figure 10.8: 2006-2008 Drilling Campaign Blank Sample Analysis by ALS Chemex (Chita)

10.7.2 2010-2012 Campaign

During the 2010 to 2012 drilling campaign, full QAQC procedures were implemented by Daltsvetmet including routine insertion of CRM, blanks, field preparation (coarse reject) and pulp duplicates to the ALS Chemex laboratory (Chita) and regularly sent, coarse reject, pulp duplicates, standards and blanks to the umpire Alex Stewart laboratory (Moscow).

QAQC procedures carried out during the 2010-2012 campaign included a CRM for every 20 routine assays, a coarse blank for every 100 routine assays, a fine blank for every 100 routine assays and a pulp duplicate for every 20-30 routine assays. Quality control samples were also inserted in the trench sampling during 2010 to 2012.

10.7.2.1 Duplicates

The precision of sampling and analytical results can be measured by re-analysing the same sample using the same methodology. The variance between the measured results is a measure of their precision. A total of 1,449 pulp duplicates were inserted into the 2010-2012 sample stream and represents 4.8% of the total samples submitted during this campaign. Three types of duplicate analysis were undertaken including internal duplicates, external duplicates and screen fire assay duplicates.

Internal Duplicates

Internal pulp duplicates were undertaken by ALS Chemex laboratory (Chita) which is the same laboratory as where the routine analysis was performed. A summary of the internal duplicate analysis carried out by WAI is shown in Figure 10.9.

External Duplicates

External pulp duplicates were undertaken by Alex Stewart laboratory (Moscow). A total of 735 samples were sent for analyses. A summary of the external duplicate analysis carried out by WAI is shown in Figure 10.10.

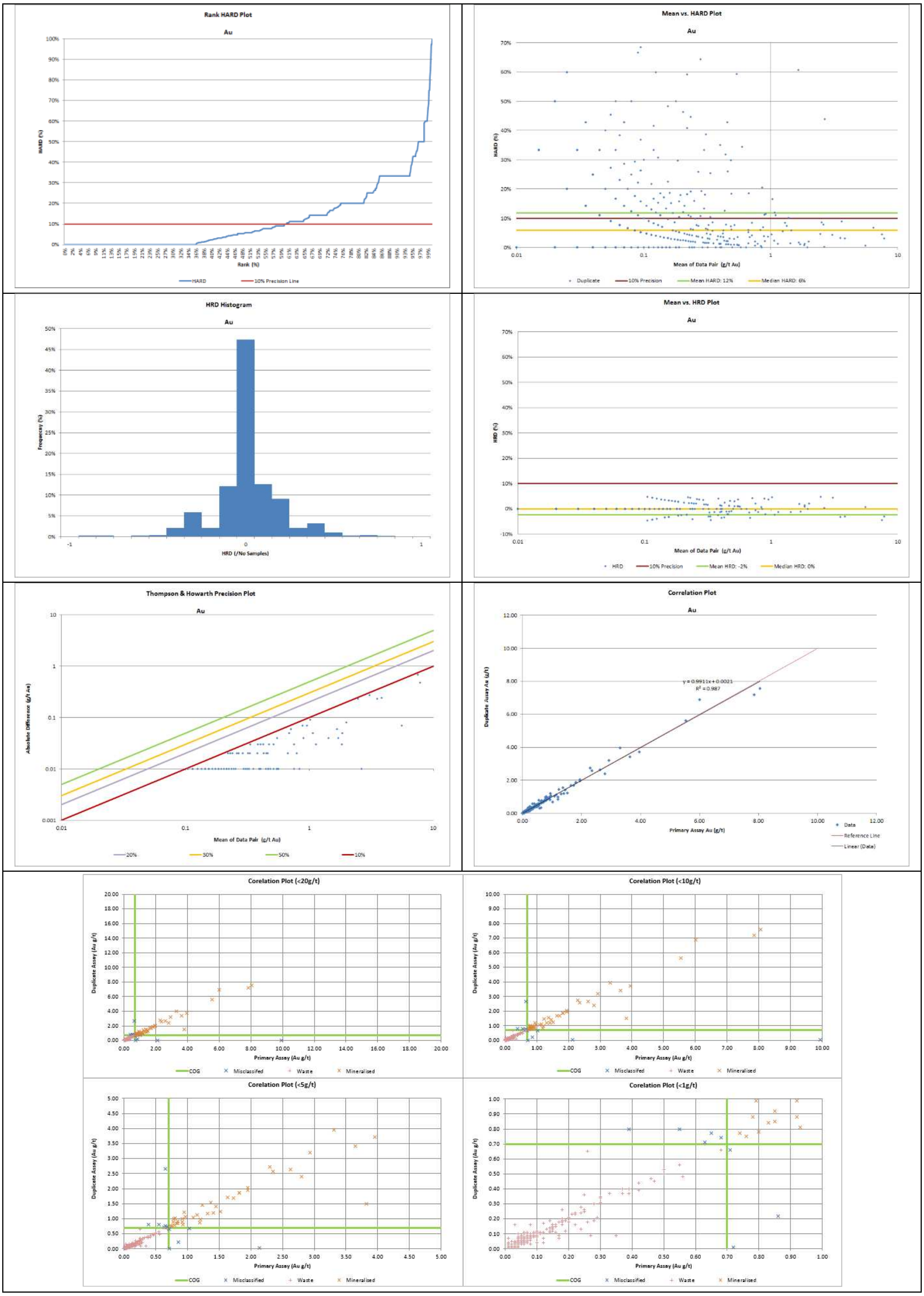


Figure 10.9: 2010-2012 Drilling Campaign Internal Pulp Duplicate Analysis by ALS Chemex (Chita)

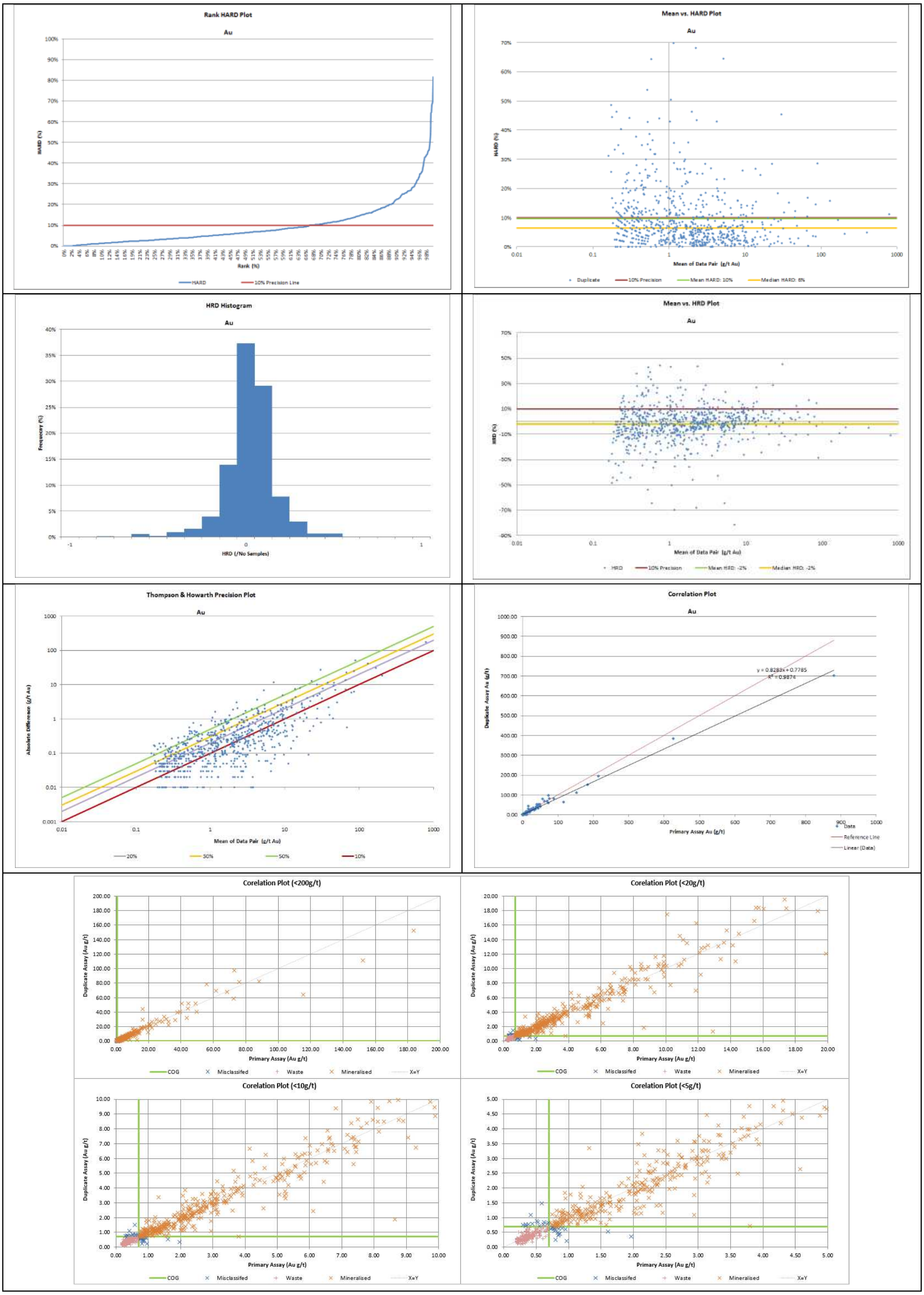


Figure 10.10: 2010-2012 Drilling Campaign External Pulp Duplicate Analysis ALS Chemex (Chita) vs Alex Stewart (Moscow)

10.7.2.2 Certified Reference Material

CRM are samples that are used to measure the accuracy of the analytical process and are composed of material thoroughly analysed to determine its grade within known error limits. The accuracy of a laboratory's analysis is assessed by comparing the laboratory's analysis of a CRM to the CRM's certified value. CRM's should cover a range of grades that are reflective of the grades encountered at a deposit. A summary of the CRM's used are shown in Table 10.4.

Table 10.4: Summary of Au Certified Reference Material Used in 2010-2012 Campaign

Supplier	Standard	Grade Au (g/t)	Standard Deviation
Geostats	G303-2	4.154	0.174
Geostats	G901-7	1.520	0.060
OREAS AU	OR-50c	0.836	0.028
OREAS AU	OR-52c	0.346	0.017
OREAS AU	OR-502	0.491	0.020
OREAS AU	OR-66a	1.237	0.054
Geostats	G301-1	0.85	0.05
Geostats	G302-2	2.5	0.14
Geostats	G302-5	1.65	0.09
Geostats	G305-1	0.21	0.01
Geostats	G305-9	4.28	0.18
OREAS	OxE74	0.615	0.017
OREAS	OxI67	1.817	0.062
OREAS	OxN62	7.706	0.117
Gannet Holding	ST-289	1.26	0.06
Gannet Holding	ST-345	0.055	0.005

CRM results are monitored by Daltsvetmet geological department on a routine basis as each batch is reported from the laboratory. The following guidelines are used by Daltsvetmet to monitor CRM analsysis performance.

- A single CRM greater than three times the standard deviation is considered unacceptable and means the subsequent samples are rejected;
- A single CRM greater than two times the standard deviation, but less than three standard deviations, is considered acceptable and no action is taken; and
- Two consecutive CRM's greater than two times the standard deviation, but less than three standard deviations, is considered unacceptable, the laboratory is notified and samples falling between the two CRM's are re-assayed.

The results of the CRM assaying have been reviewed by WAI and are shown in Figure 10.11 below.

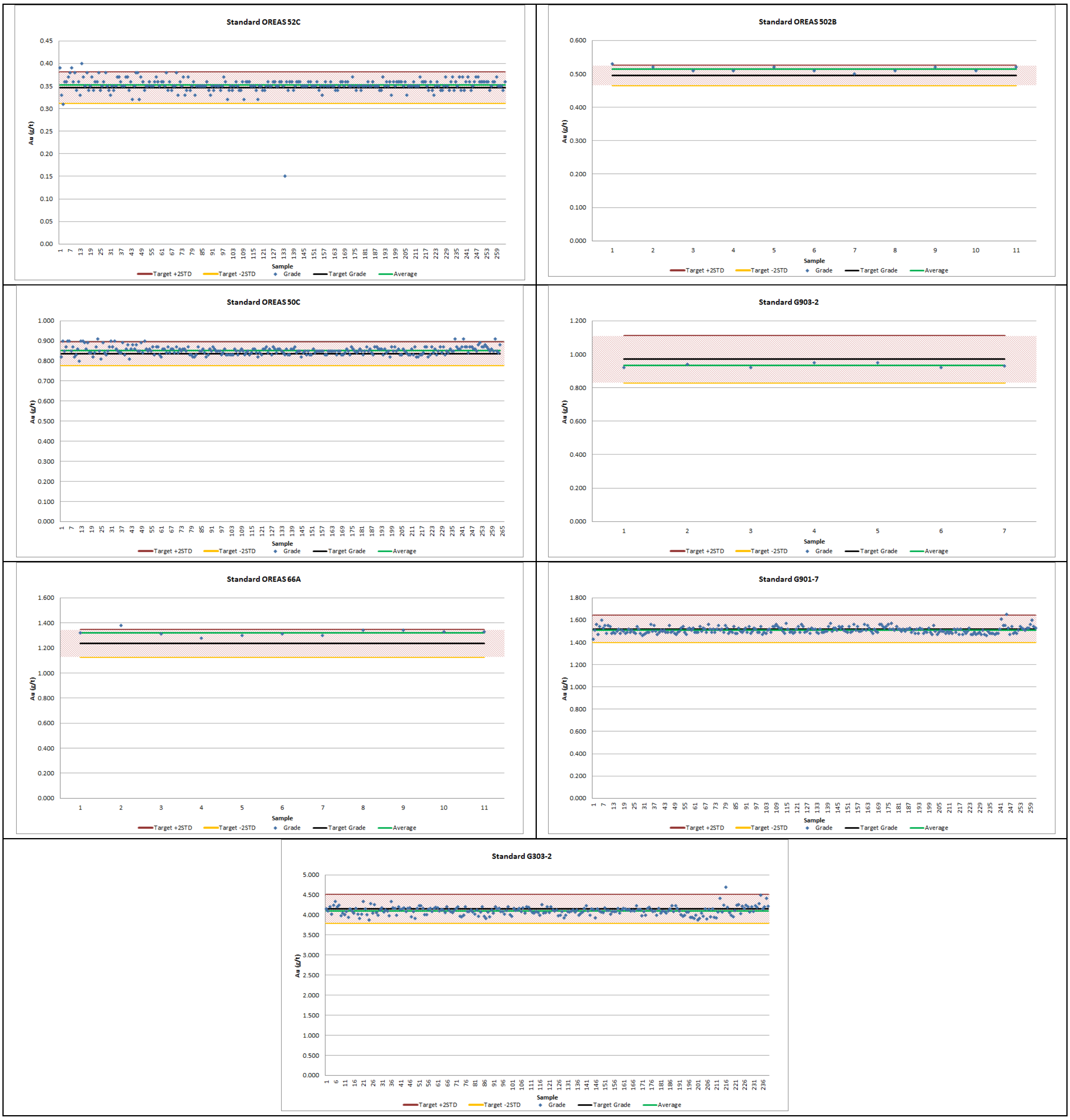


Figure 10.11: 2010-2012 Drilling Campaign CRM Analysis by ALS Chemex (Chita)

10.7.2.3 Blanks

Blank sample analysis is a method of determining sample switching and cross-contamination of samples during the sample preparation or analysis processes. Coarse blank and fine blank samples composed of the barren Upper Jurassic rocks (acid lava, felsite, quartz porphyry, tuffs and sandstone) were inserted into the sample stream by the geologist. This material is known to contain grades that are less than the detection limit of the analytical method in use. The coarse blank material was used to assess any potential contamination associated with the sample preparation stage while both the coarse and fine blank samples were used to assess any potential contamination associated with the assaying. A total of 1,362 blank samples were inserted into the sample stream at regular intervals of which 795 samples comprised coarse blank material and 567 samples comprised fine blank material.

The results of the blank assaying have been reviewed by WAI and are shown in Figure 10.12 below.

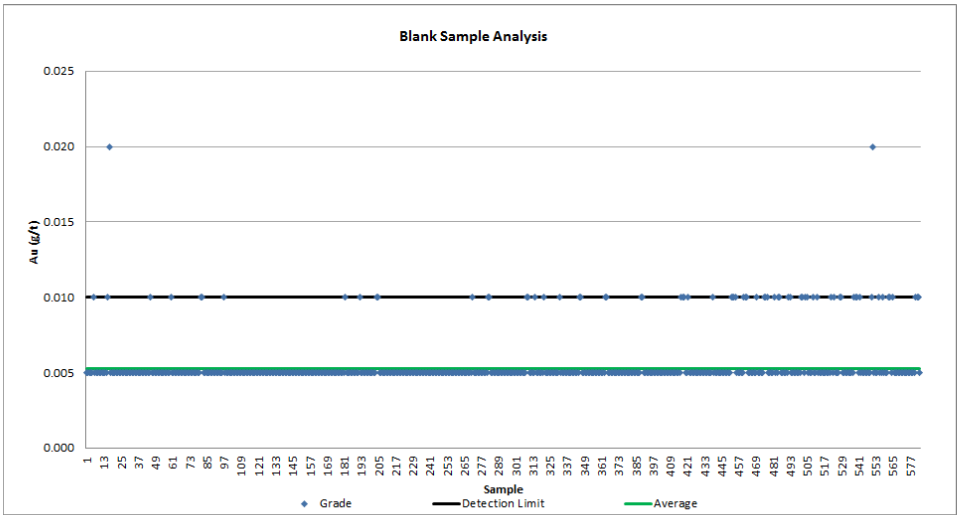


Figure 10.12: 2010-2012 Drilling Campaign Blank Sample Analysis by ALS Chemex (Chita)

10.7.3 2015 Campaign

The same procedures as carried out during the 2010-2012 campaign were implemented during the 2015 drilling campaign. ALS Chemex laboratory (Chita) was used as the primary assay laboratory and undertook both sample preparation (on whole core) and assaying. Internal pulp duplicates, CRMS's and blank assays were submitted into the sample stream, however during the 2015 drilling campaign no external duplicates were undertaken. Although this is not considered by WAI to be a significant issue, it is however recommended that for future drilling campaigns the use of an umpire laboratory be used. A summary of the QAQC analysis undertaken by WAI for the 2015 campaign internal duplicates, CRM's and blanks samples is shown in Figure 10.13, Figure 10.14 and Figure 10.15, respectively.

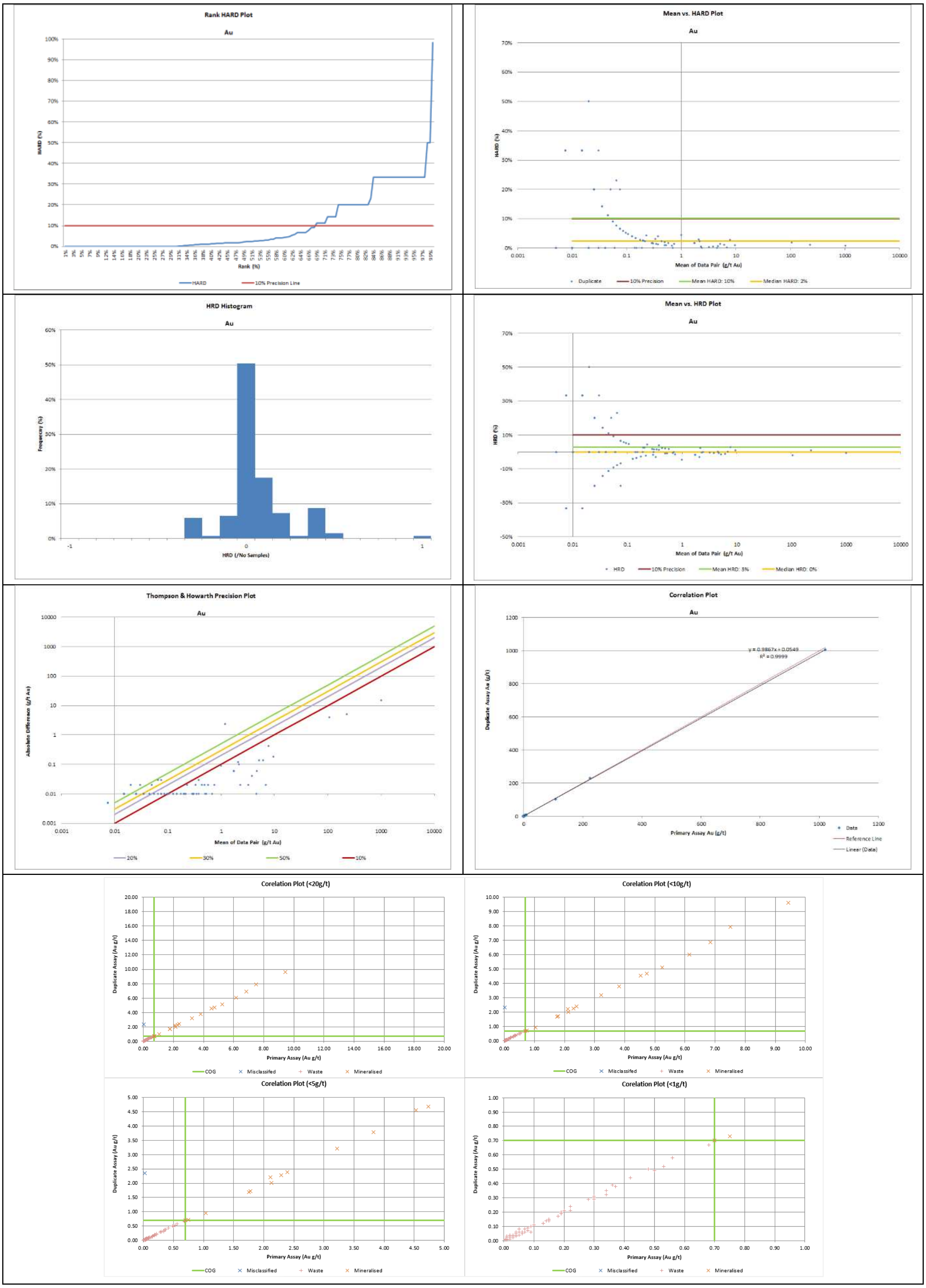


Figure 10.13: 2015 Drilling Campaign Pulp Duplicate Analysis by ALS Chemex (Chita)

Figure 10.14: 2015 Drilling Campaign CRM Analysis by ALS Chemex (Chita)

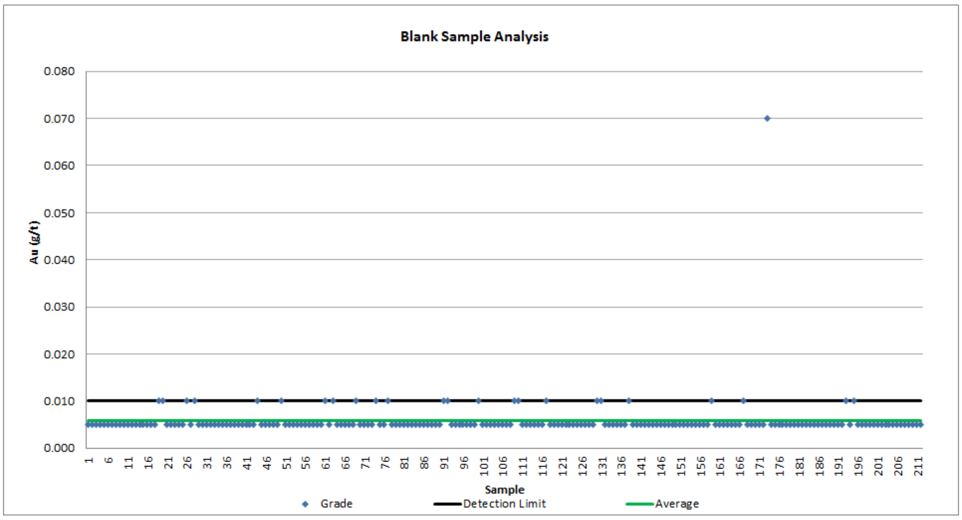


Figure 10.15: 2015 Drilling Campaign Blank Sample Analysis by ALS Chemex (Chita)

10.7.4 Zhelanny High Grade Zones

The presence of coarse gold within the Zhelanny high grade zones is problematic in terms of assaying precision and accuracy. A review of the duplicate data for samples located within the Zhelanny high grade zones was undertaken by WAI. The purpose of the review was to evaluate whether any systematic bias has been introduced into the assay database due to: 1) the small sample sizes used for assaying and; 2) conventional AA analysis of samples containing coarse gold. All pulp duplicate data available for the Zhelanny high grade zones were subsequently selected and included internal pulp duplicates at ALS Chemex (Chita) (38 samples) and external pulp duplicates at Alex Stewart (Moscow) (32 samples) from the 2010-2012 and 2015 drilling campaigns.

A programme of screen fire assaying was also undertaken by TOMS in 2012 using half core (field duplicates) from 23 samples located within the Zhelanny high grade zones. The procedure is equivalent to assaying a large sample and is used to mitigate the extreme nugget associated with coarse gold samples. Samples were weighed, jaw crushed to 1mm and then split. 100g was then sent for pulverising and fire assay, the rejects were collected for gravity separation. The gravity concentrate was then sent for fire assay and the gravity tails sent for pulverising and fire assay.

A comparison between the various duplicates located within the Zhelanny high grade zones is shown in Figure 10.16. Overall the comparison between the ALS Chemex (Chita) and Alex Stewart (Moscow) pulp duplicates is relatively good. Some variability is evident in the screen fire assay and the regular fire assay of the field duplicates assayed by TOMS compared with the ALS Chemex (Chita) primary assay, however this does not indicate any systematic bias resulting from the different assay methods and is considered by WAI to be typical for field duplicates containing coarse gold. WAI considers that there does not appear to be any systematic bias introduced into the assay database resulting from assaying of coarse gold in the Zhelanny high grade zones.

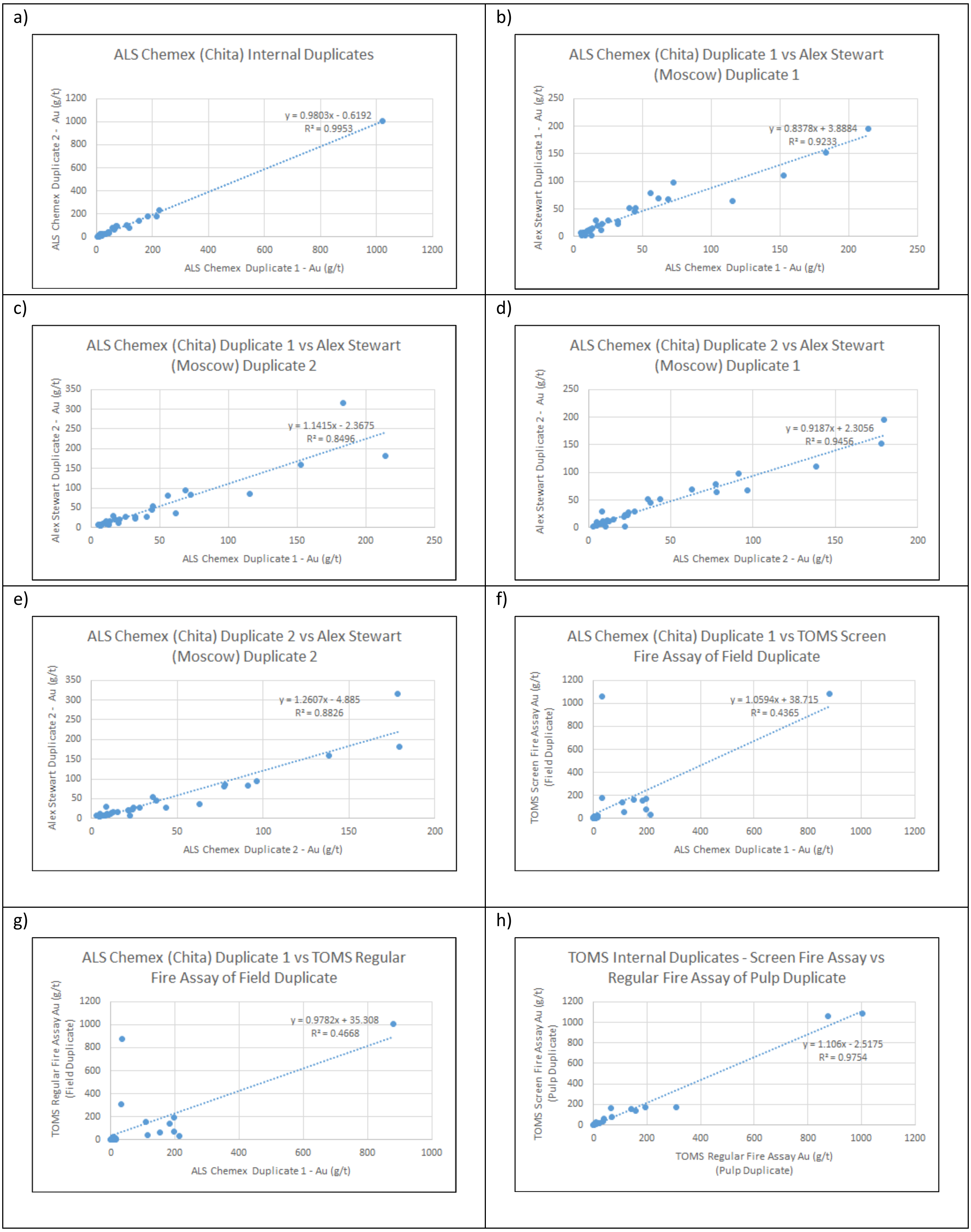


Figure 10.16: Duplicate Analysis of Samples from Zhelanny High Grade Zones (a to e - pulp duplicates ALS Chemex (Chita) and Alex Stewart (Moscow), f and g - field duplicates ALS Chemex (Chita) and TOMS, h - TOMS internal pulp duplicate

10.7.5 Adequacy of Procedures

WAI consider that the assays derived from the 2006-2008 drilling campaign that were assayed at Irgiredmet laboratory and the assays derived from the 2010-2012 drilling campaign (all assayed at ALS Chemex (Chita)) and the assays derived from the 2015 drilling campaign (all assayed at ALS Chemex (Chita)) have been generally conducted in accordance with acceptable industry standards and the assay results generated following these procedures are suitable for use in Mineral Resource estimation.

11 DATA VERIFICATION

Data entry, verification and database maintenance is carried out by Daltsvetmet staff. All data are input into a central Microsoft® Excel database located at the Daltsvetmet Company offices located in Chita.

The drill hole database was received by WAI in Microsoft® Excel format. A summary of the data verification procedures carried out by WAI are detailed below:

- Inspection of selected drill core to assess the nature of the mineralisation and to confirm geological descriptions. In the case where whole core sampling has been undertaken WAI relied on core photos and ALS Chemex (Chita) laboratory certificates;
- Verification that collar coordinates coincide with topographical surfaces;
- Verification that downhole survey azimuth and inclination values display consistency;
- Randomly selecting assay data from the drill hole database and comparing stored grades to ALS Chemex assay certificates;
- 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;
- Assessing for sample gaps or overlaps;
- Assays present in the database that were check samples for unmineralised core are recorded as chip samples (CHIP) and were subsequently assayed by XRF. These assays, although do provide a check on the core sampling, are not considered suitable for purposes of resource estimation and were subsequently assigned a half trace assay tolerance value of 0.005g/t Au; and
- Rejection of any drill holes in which data reliability issues were identified or were not considered appropriate for inclusion in the Mineral Resource estimate (see below for further description).

Only inclined exploration drill holes, inclined confirmation drill holes and the following inclined metallurgical drill holes: 811 and 812 at Gora 5 and 807, 808, 809 and 832 at Zhellany were used by WAI in the Mineral Resource estimate. These drill holes included the 2015 drilling, 2010-2012 drilling, but only included the the 2006-2008 drill holes that were assayed at Irgiredmet laboratory. WAI consider these drill holes to be sufficiently supported by QAQC and are orientated to intercept perpendicular to the mineralisation. All trenches, twinned drill holes, hydrogeological drill holes, verification drill holes, sterilisation drill holes, vertical drill holes and all other remaining drill holes from the 2006-2008 drilling not assayed at Irgiredmet laboratory were subsequently removed from the database and were not used in the Mineral Resource estimate.

The majority of the verification procedures carried out by WAI on the electronic databases confirmed the integrity of the data in these databases and used for the purposes of deriving the Mineral Resource estimate presented in this document. Minor validation errors were discovered in terms of overlapping intervals and duplicate levels, however these were subsequently corrected.

WAI has not undertaken any independent check analysis of any drill core and therefore cannot independently verify the data. However, WAI can independently state that the current database provided and used in the Mineral Resource estimate appears to be complete and is supported by the available information.

WAI have reviewed the current chain of custody procedures in place and conclude that there are no issues in terms of security of samples.

12 MINERAL PROCESSING AND METALLURGICAL TESTING

12.1 Laboratory and Pilot Plant Testwork

12.1.1 Introduction

The study of the processing characteristics of the Nasedkino gold ores was conducted in two phases between 2006-2008 and 2011-2015. Five areas of the Nasedkino deposit were studied:

- Pridolinny;
- Gora 5;
- Zhelanny;
- Pravoberezhny; and
- Glukharinny.

However, the ore reserves estimate in the feasibility study considered only four areas, namely Pridolinny, Gora 5, Zhelanny and Pravoberezhny, with Glukharinny ore regarded as a future mineral resource base.

The samples tested ranged from small-scale (mapping samples) to bulk-scale and pilot tests:

- Thirty mapping samples in 2011-2012 (TOMS);
- Three lab-scale process samples in 2006-2008 (no sample weight data available) "Center-NVP ESTAgeo";
- Bulk laboratory sample in 2010 (TOMS) 706kg;
- Two pilot samples in 2007 (OJSC "Udokan Copper" 64t and 53t);
- Three bulk-scale samples in 2011-2012 (TOMS), 1521.1, 1209.7 and 1242.0kg); and
- Four bulk-scale samples in 2014-2015 (Center-NVP ESTAgeo), 1707, 1503, 1579 and 1314.5kg.

12.1.2 Testwork Sample Head Grades

The head grades of the samples tested are summarised in Table 12.1 and Table 12.2.

Table 12.1: Chemical Composition of Samples Analyzed in 2006-2012

Element	CJSC DEKA		NVP Center-ESTAgeo LLC	OJSC Udokan Copper		NIiPI TOMS LLC							
	2006		2006	2007		2010		2011			2012		
	LTP-5	ON-1	LTP-3	PP-6-1	PP-6-2	ULTP-1	Zhelanny	Gora 5	Zhelanny	Pridolinny	1PP	2PP	4PP
	Grade, %												
SiO_2	49,50	53.60	50.00	53.00	71.80	54,10	49,20	55,70	46,00	54,40	53,20	54,10	52,10
Al_2O_3	6.50	8.90	8.90	9.00	9.30	7.16	7.09	8.10	4.80	7.90	6.54	8.19	7.41
K_2O	1.97	2.49	2.25	6.00	5.30	2.17	1.75	2.04	0.62	2.70	2.39	2.24	2.29
CaO	14.20	5.38	7.40	5.00	3.75	14.29	15.80	15.80	22.80	11.90	11.40	12.80	14.30
MgO	3.12	1.54	3.70	3.18	0.85	3.10	3.90	3.02	3.60	3.60	4.73	3.47	4.36
MnO	-	-	0.30	0.25	0.20	0.11	0.23	0.17	0.34	0.12	0.13	0.21	0.14
P_2O_5	-	-	0.17	-	-	0.07	0.07	0.10	0.08	0.09	0.07	0.09	0.04
TiO_2	0.28	0.35	0.60	0.20	0.30	0.21	-	0.32	0.28	0.32	0.27	0.33	0.30
Na_2O	0.64	0.58	2.47	0.17	0.15	<0.01	0.51	1.17	0.24	1.20	0.74	0.85	0.73
Ctotal	-	-	-	-	-	-	1.29	0.15	1.30	0.63	0.80	1.10	0.89
$CO_{2\,carb.}$	-	-	0.50	6.00	1.60	2.97	4.64	0.53	4.60	2.10	2.77	3.98	3.20
Corg	-	-	-	-	-	0.17	0.09	<0.05	<0.05	0.05	0.16	0.05	0.06
Fe_{total}	10.40	10.80	11.20	8.20	2.70	9.42	9.00	7.09	9.20	8.20	8.20	7.07	8.57
Fe_{oxide}	-	-	FeO 3.90	-	-	6.88	-	5.01	8.10	4.30	4.40	5.27	6.35
$Fe_{sulfide}$	-	-	-	-	-	2.54	-	2.08	1.10	4.10	3.80	1.80	2.22
S_{total}	2.38	1.41	1.92	3.50	1.85	2.90	-	2.16	1.00	4.20	4.25	2.00	2.53
$S_{sulfate}$	-	-	SO_3 <0.05	-	-	<0.1	-	<0.20	<0.10	<0.10	<0.10	<0.10	0.021
Zn	-	-	-	0.02	0.004	0.007	0.013	0.006	0.278	0.011	0.011	0.043	0.017
Pb	-	-	-	0.04	0.035	0.005	0.007	0.004	0.027	0.018	0.016	0.029	0.017
Cu	0.17	0.10	-	1.05	0.17	0.179	0.127	0.078	0.064	0.32	0.27	0.059	0.139
As	-	-	0.05	0.04	0.014	0.036	0.132	0.027	0.155	0.039	0.022	0.069	0.045
Sb	-	-	<0.004	-	-	0.014	<0.005	<0.005	<0.005	<0.005	<0.005	0.006	<0.005
Au. g/t	**1.20**	**1.40**	**2.70**	**2.74**	**0.69**	**1.50**	**10.84**	**2.49**	**14.02**	**1.80**	**2.97**	**1.72**	**1.93**
Ag. g/t	**5.40**	**2.80**	**15.80**	**16.40**	**3.52**	**3.80**	**5.50**	**1.49**	**4.58**	**8.79**	**5.79**	**4.41**	**3.99**

Table 12.2: Chemical Composition of Samples Analyzed in 2014-2015

Component	Area				
	Zhelanny	Gora 5	Pridolinny	Pravoberezhny	Glukharinny
	Grade %				
SiO_2	46.87	54.02	59.87	51.60	53.70
TiO_2	0.27	0.55	0.42	0.39	0.67
Al_2O_3	5.14	10.02	12.34	11.80	11.30
Fe_2O_3	17.07	10.80	9.00	11.70	12.80
- Fe_2O_3	*12.28*	*7.31*	*5.00*	*6.07*	*1.12*
- FeO	*4.31*	*2.04*	*2.15*	*3.04*	*4.54*
- $Fe_{sulfide}$	-	*0.85*	*1.12*	*1.58*	*4.65*
MnO	0.30	0.10	0.09	0.16	0.17
MgO	2.72	4.65	2.26	2.61	2.88
CaO	23.35	13.30	5.68	8.42	2.78
K_2O	0.20	2.78	3.01	2.96	2.09
Na_2O	0.38	1.60	2.00	1.68	0.45
P_2O_5	0.21	0.14	0.19	0.14	0.42
S_{TOT}	<0.20	0.97	1.28	1.80	4.67
- SO_3	-	<0.10	<0.10	<0.10	<0.10
Other impurities	3.20	2.05	3.50	7.25	10.80
H_2O	0.26	0.30	0.54	0.51	0.21
CO_2	2.62	0.55	1.12	3.85	5.08
Au. g/t*	1.42	2.03	1.17	2.68	2.00
Ag. g/t*	3.50	1.37	2.45	2.49	7.16
Total	99.23	99.89	98.27	99.14	99.66

Silver grades range from 1.37 to 16.40g/t Ag (PP-6-1). The FeO total grades ranged from 7.70 to 17.07%. except for sample PP-6-2 (2.7%). The total sulphur content in the samples ranged from 0.97 to 4.67% S.

Based on copper grades, it was decided that the process samples could be divided into three groups:

- "Low grade" 0.016-0.078% Cu (mapping samples from the Zhelanny area, Gora 5; bulk samples from the Zhelanny, Gora 5, Pridolinny, Pravoberezhny and Glukharinny areas);
- "Average grade" 0.10-0,18% Cu (ON-1 sample, LTP-5, PP-6-2, ULTP-1, 4PP); and
- "Higher grade" 0.27-0.32% Cu (Sample 1PP, mapping samples from Pridolinny) and 1.05% in sample PP-6-1.

The determined copper grades were not considered commercial (except for sample PP-6-1), but could influence the effectiveness of cyanide leaching,

The grades of base metals such as zinc and lead and other minor elements were generally low.

12.1.3 Diagnostic Leach Tests

Diagnostic gold leaching tests were performed on all Mapping and Process samples. The results are given in Table 12.3, Table 12.4 and Table 12.5.

Table 12.3: Results of Diagnostic Gold Leach of Mapping Samples

Form	Area					
	Zhelanny		Pridolinny		Gora 5	
	Gold grade, g/t	Gold Dist. %	Gold grade, g/t	Gold Dist, %	Gold grade, g/t	Gold distribution, %
Freely cyanidable gold (clean surface)	0.33-54.4	22.15-89.3	0.11-1.76	7.5-43.8	0.41-1.82	27.8-71.9
Intergrown gold	0.42-6.07	7.57-51.8	0.29-1.89	32.6-48.3	0.52-1.67	20.6-69.6
Total: cyanidable gold	**1.0-60.45**	**67.12-99.3**	**0.45-3.65**	**52.4-83.7**	**1.03-3.32**	**85.2-96.2**
Gold locked in films	0.05-0.38	0.62-8.72	0.01-0.2	1.8-13.6	0.01-0.08	0.49-3.23
Sulphide locked gold	0.03-0.31	0.06-18.8	0.07-0.46	6.3-24.5	0.07-0.17	2.25-7.65
Quartz encapsulated gold	0.01-0.36	0.03-5.37	0.03-0.23	4.4-9.5	0.03-0.08	0.83-3.47

Table 12.4: Results of Diagnostic Gold Leach in 2006-2012 (CJSC DEKA. NIiPI TOMS LLC)

Form	Grade g/t							Distribution %						
	Sample													
	LTP-5	ON-1	ULTP-1	Zhelanny	1PP	2PP	4PP	LTP-5	ON-1	ULTP-1	Zhelanny	1PP	2PP	4PP
Free cyanidable gold (clean surface),2mm	0.37	0.13	0.27	2.58	1.17	0.51	0.67	19.37	7.32	18.20	22.15	36.56	30.18	35.45
Free cyanidable gold 60% -0.071 mm	0.23	0.18	0.43	3.74	0.16	0,42	0.47	12.06	10.30	28.80	32.19	5.01	24.85	24.87
Free cyanidable gold (clean) 90% -0.071m	0.32	0.08	0.11	1.55	0.26	0.13	0.10	16.98	4.80	7.40	13.33	8.12	7.69	5.29
Total: Free CN gold (clean).	0.92	0.39	0.81	7.87	1.59	1.06	1.24	48.41	22.42	54.40	67.67	49.69	62.72	65.61
Open intergrowths	0.75	1.00	0.47	3.23	1.33	0.34	0.46	39.0	56.89	31.50	27.77	41.56	20.12	24.34
Total cyanidable gold	**1.67**	**1.39**	**1.28**	**11.10**	**2.92**	**1.40**	**1.70**	**87.41**	**79.31**	**85.90**	**95.44**	**91.25**	**82.84**	**89.95**
Locked in films	н/о	н/о	0.05	0.09	0.05	0.06	0.02	н/о	н/о	3.60	0.74	1.56	3.55	1.06
Sulfide locked	0.11	0.32	0.05	0.36	0.20	0.20	0.12	5.80	18.66	3.30	3.08	6.25	11.83	6.35
Locked in silicates	0.13	0.04	0.11	0.09	0.03	0.03	0.05	6.79	2.03	7.20	0.74	0.94	1.78	2.64
Total: non-cyanidable gold	**0.24**	**0.36**	**0.21**	**0.54**	**0.28**	**0.29**	**0.19**	**12.59**	**20.69**	**14.10**	**4.56**	**8.75**	**17.16**	**10.05**
Head ore	**1.91**	**1.75**	**1.49**	**11.64**	**3.20**	**1.69**	**1.89**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**

Table 12.5: Results of Diagnostic Gold Leach in 2014-2015 (NVP Center-ESTAgeo LLC)

Form	Area									
	Zhelanny	Gora 5	Pridolinny	Pravoberezhny	Glukharinny	Zhelanny	Gora 5	Pridolinny	Pravobere-zhny	Glukharny
	Gold grade, g/t					Gold distribution, %				
Free (amalgamable)	1.02	1.17	0.50	2.64	0.44	60.79	55.00	45.45	74.58	22.11
Open intergrowths (cyanidable)	0.52	0.82	0.44	0.64	0.44	30.89	38.44	40.00	18.08	22.11
Total: cyanidable gold	**1.54**	**1.99**	**0.94**	**3.28**	**0.88**	**91.68**	**93.44**	**85.45**	**92.66**	**44.22**
locked in acid-soluble minerals	0.06	0.01	0.02	0.07	0.25	3.74	0.29	2.16	1.86	12.71
Sulfide locked	0.03	0.13	0.12	0.12	0.78	1.51	6.27	10.98	3.52	39.11
Locked in acid-insoluble rock-forming minerals	0.05	0	0.02	0.07	0.08	3.07	0	1.41	1.96	3.96
Total: non-cyanidable gold	**0.14**	**0.14**	**0.16**	**0.26**	**1.11**	**8.32**	**6.56**	**14.55**	**7.34**	**55.78**
Head ore	**1.68**	**2.13**	**1.10**	**3.54**	**1.99**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**
Direct determination	**1.42**	**2.03**	**1.17**	**2.68**	**2.00**					

The mapping samples gave the following results:

- Zhelanny samples contained 67.1-99.3% cyanidable gold (including free gold ranging from 22.2-89.3%);
- Pridolinny samples contained – 52.4-83.7% cyanidable gold (including free gold ranging from 7.5-43.8%); and
- Gora 5 – 85.2-96.2% cyanidable gold (including free gold 27.8-71.9%).

The bulk of non-cyanidable gold was associated with sulphides - up to 18.8% (Zhelanny), up to 24.5% (Pridolinny) and up to 7.6% (Gora 5) and, to a lesser extent, in "films" (8.72, 13.6% and 3.23, respectively), and in quartz (5.3, 9.5 and 3.5%, respectively).

The results of the 2006-2012 diagnostic gold leach tests indicated that even at a -2mm grind size free the gold contents ranged from 7.3% to 36.6%. Free gold contents, at a size of 60% passing -0.071mm, ranged from 5.0% to 32.2%.

The overall cyanidable gold content ranged from 79.3% to 95.4% and overall non-cyanidable gold locked in sulphides from 3.1% to 18.6%.

The results of the 2014-2015 diagnostic gold leach tests for samples from Zhelanny, Gora 5, Pridolinny, Pravoberezhny and Glukharinny showed that 85.5-93.4% of gold was cyanidable, i.e. free and in the form of open intergrowths, except for the Glukharinny area where cyanidable gold amounted to only 44.2%. The proportion of refractory gold ranged from 6.6 to 14.6%, and amounted to 55.8% in the sample from Glukharinny. This gold is generally sulphide locked and less silicate encapsulated.

It was concluded that the Zhelanny, Gora 5 and Pravoberezhny ores were "easy-processing" (readily cyanidable), the Pridolinny ores were "refractory" (difficult to cyanide), and the Glukharinny ores are "highly refractory".

12.1.4 Beneficiation Tests

12.1.4.1 Introduction

A considerable range of research studies were undertaken on processing of the Nasedkino ores in 2006-2015. The studies were carried out in the following areas:

- Processing testing of the gravity, gravity-flotation, flotation, gravity-cyanide leach, cyanide leach flowsheets;
- Assessment of the feasibility of joint processing the ores;
- Determination of physical properties of the ores, substantiation and development of a rational ore preparation flowsheet; and
- Development, substantiation and selection of a rational processing technology of the ores and metallurgical processing of the products.

Processing studies were performed by DEKA, Udokan Copper, "Vostokgeologiya", "LITsIMS", RATs, Irgiredmet, TOMS, and Center-ESTAgeo.

12.1.4.2 Samples LTP-5 and ON-1 (DEKA, 2007)

Sample LTP-5 was a Primary ore sample and Sample ON-1 represented Oxide ore. Processing studies were carried out on these samples as a composite at a ratio of 90 - 10%.

Gravity testing of sample LTP-5 ground to 85% passing -0.074mm produced a gravity concentrate with a mass yield of 0.85%, a gold grade of 100g/t Au and a recovery of 47.4%. The oxide ore sample, ground to 96% passing -0.071mm gave lower yields and grades at 0.97%, 16.52g/t Au and a gold recovery of 19.7%.

Flotation of the primary ore gave recoveries of gold and silver of 88.1% and 85.6%. Flotation of the oxide ore gave a gold recovery of 58.3%.

Cyanide leaching of the gravity concentrate was carried out in two stages: the first stage included intensive cyanidation and the second conventional cyanidation after grinding to 98% passing -0.071mm. Gold recovery was 99.9%.

Cyanidation of the flotation concentrate was conducted after regrinding the material to 85% passing -0.020 mm and oxidation. Gold recovery was 90.6% and silver recovery was 31.5% (by stage).

Based on the results, a gravity–flotation flowsheet was recommended with leaching of the concentrates to produce dore.

12.1.4.3 Sample LTP-3 (Center-ESTAgeo, 2006)

Processing testing was carried out on sample LTP-3 (Pridolinny) weighing 252.5kg and containing 2.7g/t Au, 15.8g/t Ag and 0.54% Cu. Testing comprised multi-stage gravity concentration, giving a concentrate grading 25.3g/t Au at a recovery of 40.8%.

Additional tests using a Falcon concentrator gave a gravity concentrate grading 107g/t Au at a recovery of 33.9%.

12.1.4.4 Pilot Samples PP-6-1 and PP-6-2 (Udokan Copper, 2007)

These samples, weighing 64t and 53t respectively, were taken from Pridolinny. Sample PP-6-1 assayed 2.74g/t Au, 16.4g/t Ag and 0.95% Cu. Sample PP-6-2 represented off-balance ore (0.67Au g/t, Ag 4.0g/t and 0.09% copper). A gravity-flotation flowsheet was used after grinding to 80-85% passing -0.074mm. Rougher and scavenging was followed by two stages of cleaning and regrinding of the 1st cleaner scavenger tails to 98-100% passing -0.044mm.

Testing of sample PP-6-1 produced a gravity gold concentrate grading 722.8g/t Au, 105.6g/t Ag at recoveries of 51.0% for gold and 1.13% for silver. The copper concentrate assayed 18.4% Cu, 20.6g/t Au and 302.4g/t Ag at recoveries of 67.9%, 28.6% and 63.5% respectively. The overall gold recovery was 79.6%.

Sample PP-6-2 with the head gold grade of 0.68g/t Au, 4.43g/t Ag and 0.078% Cu gave a gravity concentrate grading 429.0g/t Au, 220.9g/t Ag at gold and silver recoveries of 46.9% and 3.71%. The copper concentrate graded 2.22% Cu, 6.7g/t Au and 92.5g/t Ag at recoveries of 29.4%, 10.2% and 21.6%. The overall gold recovery was 57.1%.

12.1.4.5 Results of Processing Testing on Sample ULTP-1 (NIiPI TOMS LLC, 2010)

A bulk sample of 706kg was made from samples from three areas:

- Pridolinny (542kg), with a grade of 1.15g/t Au and 2.9g/t Ag;
- Gora 5 (167kg), with a gold grade of 1.51g/t Au and 1.0g/t Ag; and
- Zhelanny (68kg), with a gold grade of 11.6g/t Au and 6.0g/t Ag.

Gravity testing indicated that two-stage concentration (60-70% passing -0.074mm at the first stage and 90% passing -0.074mm at the second stage) could yield a concentrate grading 353g/t Au at 60.0% recovery.

Cyanidation of the head ore after grinding to 90% passing -0.074mm gave a gold recovery of 95.4%.

Gold recovery from the gravity concentrate by intensive cyanidation was 92.8% (13-15 hours). Cyanide consumption was 24.6kg/t. Additional cyanidation of the residue gave a recovery of 88.9% (stage) at a rate of cyanide consumption 3.3kg/t. Sorption cyanidation of gravity tailings, ground to 90% passing -0.074mm gave a gold recovery of 89.6% and a cyanide consumption of 1.5kg/t.

12.1.4.6 Pilot Sample 1PP (TOMS, 2011-2012)

Gravity, flotation and hydrometallurgical flowsheets were used on this sample of Pridolinny and Gora 5 ores blended at a ratio of 85-15%. The gravity gold recovery was 60.5% to a concentrate grading 140.6g/t Au. Further gravity upgrading gave an overall recovery of 39.1% to a gold concentrate grading 3,240g/t Au.

A gravity-flotation flowsheet produced a gravity concentrate grading 610g/t Au at 54.4% recovery. Flotation gave a gold concentrate grading 98g/t Au at a recovery of 26.9%.

Cyanide leach tests were conducted on both the feed and concentrates. Cyanide leaching on the head gave a gold recovery of 92.3% (cyanide consumption 3.0kg/t). Intensive cyanidation of the gravity concentrate gave a stage recovery of 99.6%.

12.1.4.7 Pilot Sample 2PP (TOMS, 2011-2012)

A gravity (GRG test) on a sample with a head grade of 2.10g/t Au and 4.12g/t Ag gave a gold recovery of 41.7% to a concentrate grade of 51.1g/t Au. Upgrading the concentrate gave a gold recovery to the final concentrate of 8.34% at a grade of 1,025g/t Au.

A gravity-flotation flowsheet (locked cycle) gave a gravity concentrate assaying 148.8g/t Au at 33.3% recovery. The flotation concentrate had a gold grade of 20.2g/t Au at a recovery of 45.8% and a silver grade of 74.7g/t Ag at 69.3% recovery. The overall gold and silver recoveries were 79.1% and 80.9%.

Intensive cyanidation of the gravity concentrate gave a gold recovery of 97.2%. Cyanide consumption was 14.42kg/t. Cyanidation of the gravity tailings gave a stage gold recovery of 81.2%.

12.1.4.8 Pilot Sample PP4 (TOMS 2011-2012)

The sample was made from samples PP1, PP2 and PP3, which were formed in different ratios from Gora 5 and Pridolinny ores (Sample PP1) and Pridolinny, Gora 5 and Zhelanny (Samples PP2 and PP3). The gold grade of sample 4PP was 2.05g/t Au and 5.53g/t Ag.

Testing for samples PP4, PP1 and PP2 was the same. As a result of GRG-test the gold recovery to the overall concentrate amounted to 45-52% at a yield 0.16-0.57%, the gold grade in the gravity tailings was 0.96-1.02g/t Au.

12.1.4.9 Pilot Samples - NVP Center-ESTAgeo LLC (2014-2015)

Pilot scale processing testing on the Nasedkino ores (Pridolinny, Gora 5, Zhelanny, Pravoberezhny and Glukharinny) was undertaken and used to develop the process design criteria.

Laboratory studies and tests were also conducted on laboratory samples weighing about 300kg quartered off the pilot samples. The tests were carried out using the following processing flowsheets:

- gravity-flotation followed by cyanidation of gravity and flotation concentrates,
- flotation flowsheet followed by cyanidation of flotation concentrates,
- gravity-cyanide leach flowsheet, with cyanidation of gravity concentrates and gravity tailings, and
- direct cyanidation of the head ore.

The better results were obtained using either the gravity-cyanidation route or direct ore cyanidation, the results of which are given in Table 12.6 and Table 12.7.

Table 12.6: Summary Results for the Gravity-cyanide Leach Flowsheet

Products	Mass %	Au g/t	Recovery Au %
Pridolinny			
Gravity concentrate	0.50	97.00	33.41
Cyanide leach of gravity concentrate		94.28	32.48
Gravity concentrate residue	0.50	2.72	0.94
Loaded carbon from leach of the gravity tailings		0.82	56.36
Cyanide leach cake to the waste dump	99.50	0.15	10.22
Head	100.00	1.44	100.00
Overall Gold Recovery %			**88.84**
Gora 5			
Gravity concentrate	0.49	132.00	34.89
Cyanide leach of gravity concentrate		*129.64*	*34.26*
Gravity concentrate residue	0.49	2.36	0.62
Loaded carbon from leach of the gravity tailings		*1.06*	*56.83*
Cyanide leach cake to the waste dump	99.51	0.15	8.29
Head	100.00	1.86	100.00
Overall Gold Recovery %			**91.09**
Zhelanny			
Gravity concentrate	0.47	239.50	58.54
Cyanide leach of gravity concentrate		*235.62*	*57.59*
Gravity concentrate residue	0.47	3.88	0.95
Loaded carbon from leach of the gravity tailings		*0.70*	*36.16*
Cyanide leach cake to the waste dump	99.53	0.10	5.30
Head ore	100.00	1.93	100.00
Overall Gold Recovery %			**93.75**
Pravoberezhny			
Gravity concentrate	0.60	296.50	61.34
Cyanide leach of gravity concentrate		293.03	60.63
Gravity concentrate residue	0.60	3.47	0.72
Loaded carbon from leach of the gravity tailings		1.01	*34.43*
Cyanide leach cake to the waste dump	99.40	0.12	4.22
Head	100.00	2.90	100.00
Overall Gold Recovery %			**95.06**

Table 12.7: Results for the Head Leach Flowsheet			
Product	**Mass %**	**Gold grade g/t**	**Gold recovery %**
Pridolinny			
Gold loaded carbon		0.93	**83.04**
Cyanide leach cake	100.00	0.19	16.96
Head ore	100.00	1.12	100.00
Gora 5			
Gold loaded carbon		1.59	**91.38**
Cyanide leach cake	100.00	0.15	8.62
Head ore	100.00	1.74	100.00
Zhelanny			
Gold loaded carbon		1.66	**91.71**
Cyanide leach cake	100.00	0.15	8.29
Head ore	100.00	1.81	100.00
Pravoberezhny			
Gold loaded carbon		3.42	**94.48**
Cyanide leach cake	100.00	0.20	5.52
	100.00	3.62	100.00

12.2 Conclusions on the Processing Testwork Programmes

The results of the extensive pilot scale and bulk-scale testwork on the four Nasedkino areas using three processing flowsheets - gravity-flotation, gravity-cyanidation and head cyanide leach, established that the following recoveries were obtained:

- Pridolinny: with the gravity-cyanide flowsheet recovery was 88.8% versus 82.3% using the gravity-flotation and 83.04% using the cyanide heap leach flowsheet with differences of 6.57 and 5.80%, respectively;
- Gora 5 area: - recovery was 91.4% using the head cyanide leach flowsheet versus 80.2% using the gravity-flotation and 91.1% using the gravity-cyanide leach flowsheet, with recovery differences of 11.2 and 0.29%, respectively;
- Zhelanny: the recovery was 93.8% using the gravity-cyanide leach flowsheet versus 87.4% using the gravity-flotation and 91.7% using the head cyanide leach flowsheet, with differences of 6.4 and 2.4%, respectively; and
- Pravoberezhny: the recovery was 95.1% with the gravity-cyanide leach flowsheet followed by 90.6% using gravity-flotation and with 94.5% using the head cyanide leach flowsheet, with a differences of 4.5% and 0.58%, respectively.

According to the Russian system, the dressability of the Gora 5, Zhelanny and Pravoberezhny ores were defined as “readily processed”, and the Pridolinny ores are defined as “moderately easily processed”. It was concluded that the aforementioned classifications do not exclude the possibility of processing them using a single processing flowsheet and therefore they can be referred to as one metallurgical type.

13 MINERAL RESOURCE ESTIMATES

13.1 Introduction

The following sections describe in detail the methodology used by WAI to produce the Mineral Resource estimates for the Nasedkino Gold Project.

13.2 Database Compilation

Database validation undertaken by WAI included: an evaluation of minimum and maximum grade values and sample lengths, assessing for inconsistencies in spelling or coding (typographic or case sensitive errors), ensuring full data entry and that a specific data type (collar, survey, lithology and assay) is not missing), assessing for sample gaps and overlaps and a review of assay detection limits and identification of problematic assay records. A spatial on-screen review of the grade and lithology distributions of all drill holes was undertaken to identify any drill holes exhibiting data reliability issues. Overall the database was considered to be robust with no significant errors identified. A check on collar locations relative to topography also found only minor errors.

All drill holes in the database were imported into Datamine software and desurveyed using the HOLES3D process. Trench samples were also imported and constructed as drill holes using strings from sample points. The overall drill hole and trench sample database was then used as a guide to geological and structural interpretation only.

For resource estimation purposes, only inclined exploration drill holes, inclined confirmation drill holes and the following inclined metallurgical drill holes: 811 and 812 at Gora 5 and 807, 808, 809 and 832 at Zhelanny were used. These drill holes included the 2015 drilling, 2010-2012 drilling but only included the 2006-2008 drill holes that were assayed at Irgiredmet laboratory.

WAI considers the drill holes included in the Mineral Resource estimate to be sufficiently supported by QAQC and are orientated to intercept perpendicular to the mineralisation. All trenches, twinned drill holes, hydrogeological drill holes, verification drill holes, sterilisation drill holes, vertical drill holes and all other remaining drill holes from the 2006-2008 drilling that were not assayed at Irgiredmet laboratory were subsequently removed from the database

Assays present in the database that were check samples for unmineralised core are recorded as chip samples (CHIP) and were subsequently assayed by XRF. These assays, although do provide a check on the core sampling, are not considered suitable for purposes of resource estimation and were subsequently assigned a half trace assay tolerance value of 0.005g/t Au.

A summary of the drill hole database after data verification and database validation is shown in Table 13.1. All drill holes are from diamond core drilling.

Table 13.1: Drill Hole Data Used in Mineral Resource Estimate				
Deposit	**Drilling Campaign**	**Number of Drill Holes**	**Meterage (m)**	**Number of Au Assays**
Pridolinny	2006-2008	176	17,701	16,875
	2010-2012	127	12,010	13,239
	2015	-	-	-
	Sub Total	**303**	**29,711**	**30,114**
Gora5	2006-2008	-	-	-
	2010-2012	66	6,235	6,907
	2015	-	-	-
	Sub Total	**66**	**6,235**	**6,907**
Zhelanny	2006-2008	-	-	-
	2010-2012	76	8,759	9,571
	2015	11	744	818
	Sub Total	**87**	**9,503**	**10,389**
Pravoberezhny	2006-2008	-	-	-
	2010-2012	49	7,709	8,472
	2015	6	634	694
	Sub Total	**55**	**8,343**	**9,166**
Grand Total		**511**	**53,792**	**56,576**

13.3 Geological Interpretation and Domaining

13.3.1 General

The geology and mineralisation of the Nasedkino area is provided in Section 6.

Thin overburden and oxide interfaces are present and wireframes for these were constructed by WAI based on the lithological logging.

A summary of the geological interpretation for each deposit is given below. It is noted, however that the drill hole database for the Nasedkino Gold Project contains lithology data only for the 2010-2012 drilling campaign. Although the geological logs for the 2006-2008 campaigns are available they have not been digitised. To improve lithological domaining of the project it is recommended that all lithological data are digitised and incorporated into the drill hole database.

13.3.2 Pridolinny

Pridolinny consists of 132 mineralised zones which strike approximately 320° and dip sub-vertically to the northeast. The major mineralisation is hosted within the skarn package of the Main Zone, however a significant area of gneiss hosted mineralisation is located to the southwest of the deposit (Pridolinny Extension). Minor hangingwall gneiss hosted mineralisation is also present on the northeast flank of the deposit. Mineralisation wireframes were produced based on a 0.7g/t Au cut-off grade. A plan view showing the Pridolinny mineralised zones is shown in Figure 13.1.

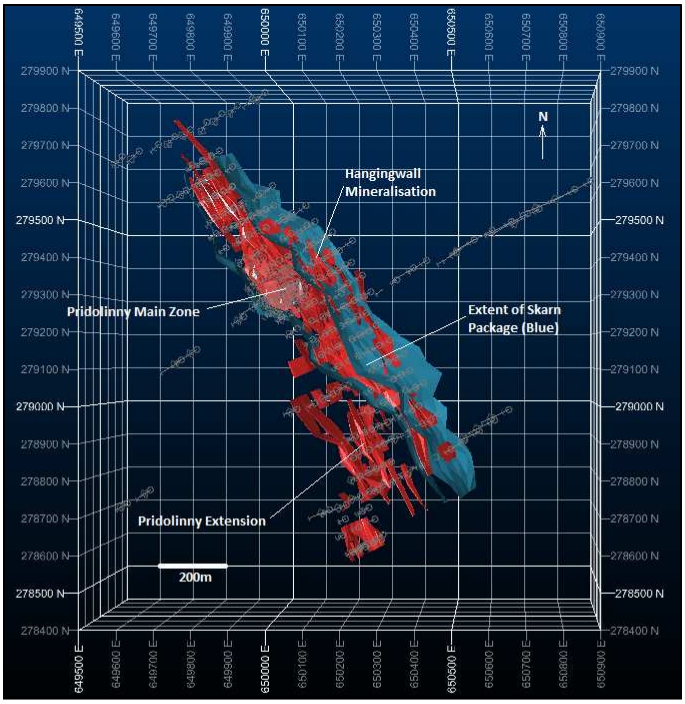


Figure 13.1: Plan View of Pridolinny Deposit Showing Mineralisation Wireframes (Red) and Extent of Skarn Package (Blue)

13.3.3 Gora5

Gora5 consists of 29 mineralised zones which strike approximately 330° and dip sub-vertically to the northeast. The majority of mineralisation is hosted within the skarn package. Mineralisation wireframes were produced based on a 0.7g/t Au cut-off grade. A plan view showing the Gora 5 mineralised zones is shown in Figure 13.2.

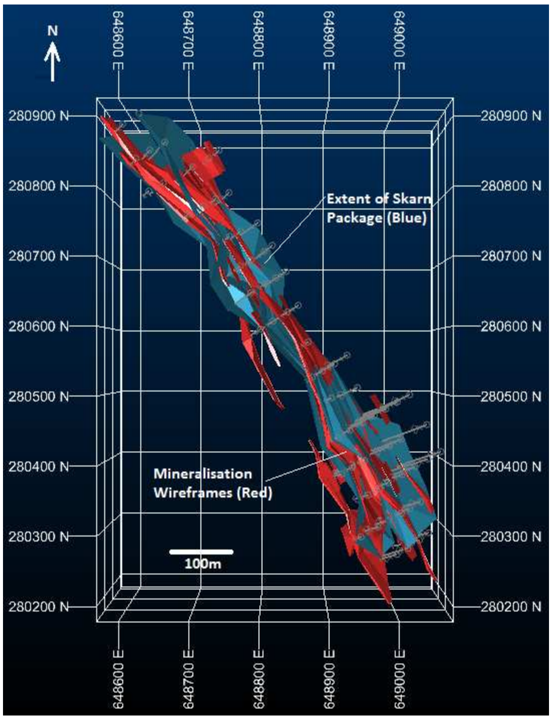


Figure 13.2: Plan View of Gora 5 Deposit Showing Mineralisation Wireframes (Red) and Extent of Skarn Package (Blue)

13.3.4 Zhelanny

Zhelanny comprises two separate fault bounded zones with 14 mineralised zones which strike approximately 320° and dip sub-vertically to the northeast. Three additional zones of high grade mineralisation containing significant coarse gold are present in the central part of the deposit and result from a separate later stage of mineralisation. The majority of mineralisation is again hosted within the skarn package. Mineralisation wireframes were produced based on a 0.7g/t Au cut-off grade for the normal grade zones while a 7g/t Au cut-off was used for the high grade zones. A plan view showing the Zhelanny normal grade mineralised zone wireframes is shown in Figure 13.3 while the Zhelanny high grade mineralised zone wireframes (located within the normal grade structures) are shown in Figure 13.4.

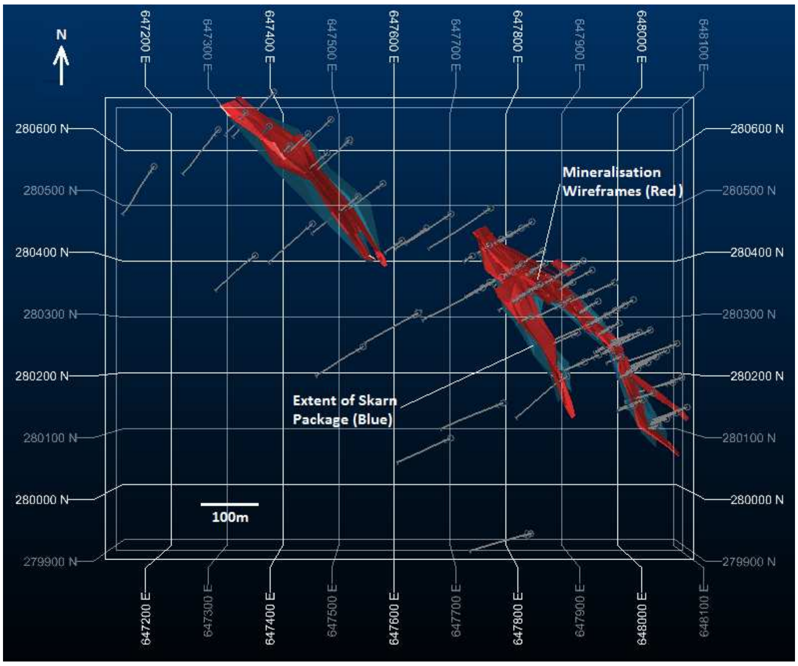


Figure 13.3: Plan View of Zhelanny Deposit Showing Mineralisation Wireframes (Red) and Extent of Skarn Package (Blue)

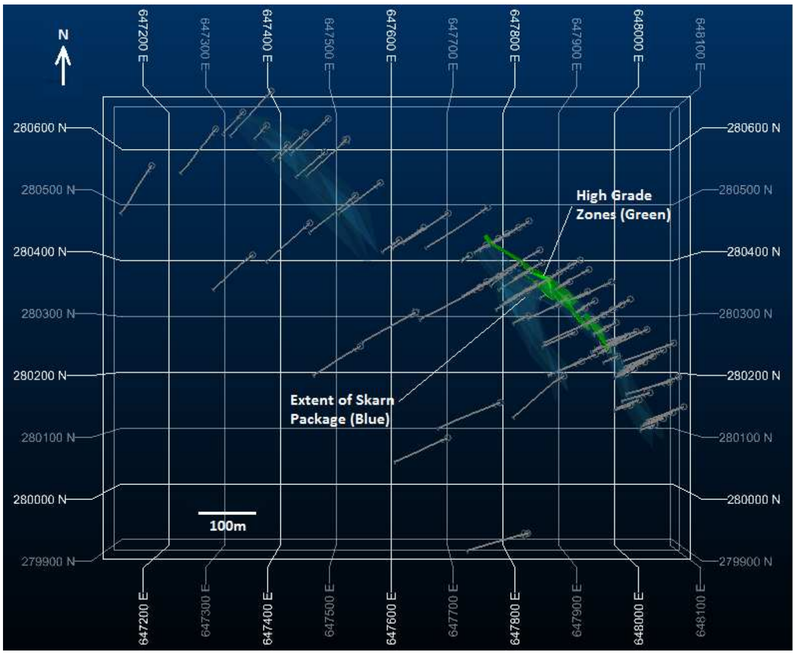


Figure 13.4: Plan View of Zhelanny Deposit Showing High Grade Zone Wireframes (Green) and Extent of Skarn Package (Blue)

13.3.5 Pravoberezhny

Pravoberezhny comprises 32 mineralised zones which strike approximately 320° and dip sub-vertically to the northeast. Skarn lithologies are less extensive at Pravoberezhny and a more significant proportion of the mineralisation is hosted within gneiss. Mineralisation wireframes were again produced based on a 0.7g/t Au cut-off grade. A plan view showing the Pravoberezhny mineralised zones is shown in Figure 13.5.

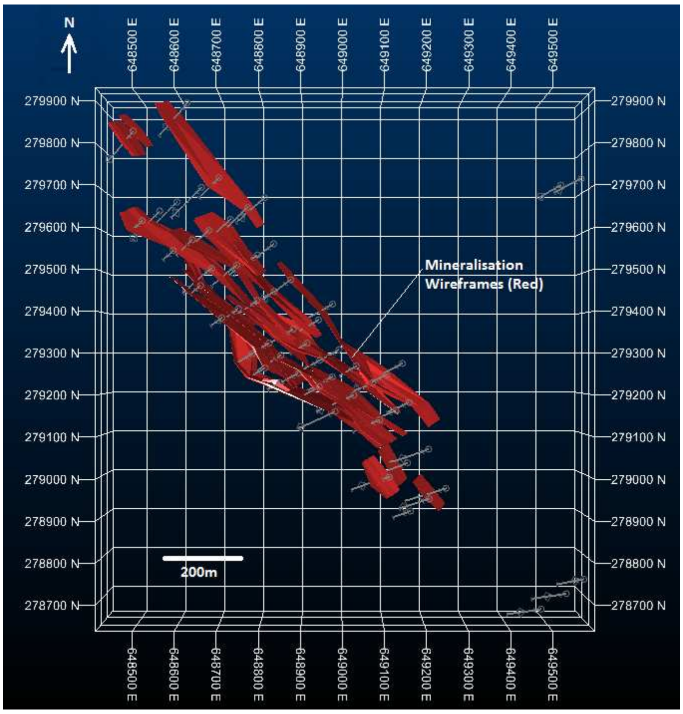


Figure 13.5: Plan View of Pravoberezhny Deposit Mineralisation Wireframes (Red)

13.4 Drill Hole Data Processing

Drillholes remaining following data verification were used to select samples located within the domain wireframes. To preserve the integrity of the assay sample lengths, the drill hole file containing only assay data was used as the basis for all further data processing. The selected samples were coded by the principal domains and formed the basis of the Mineral Resource estimate. Statistical analysis of the raw selected samples is contained in Appendix 2.

13.5 Grade Capping

Grade caps were applied to ensure that anomalously high grade samples did not bias the grade estimation of the domain. Grade caps were reviewed on a domain by domain basis using log probability plots. Not all domains warranted grade caps and a summary of the grade caps applied to the relevant domains for Au are shown in Table 13.2. The spatial location of the high grade samples at Zhelanny located within the high grade domains (101, 102 and 103) were reviewed after grade capping. During grade estimation, limits to the search ellipses were also applied to drill hole 607 in Domain 101 at Zhelanny and drill hole 1033 in Domain 201 at Zhelanny to further limit the influence of localised extreme grades in these drill holes. This is further discussed in the grade estimation section.

Table 13.2: Grade Cap Levels

Pridolinny		Gora5		Zhelanny		Pravoberezhny	
Domain	**Grade Cap Au (g/t)**	**Domain**	**Grade Cap Au (g/t)**	**Domain**	**Grade Cap Au (g/t)**	**Domain**	**Grade Cap Au (g/t)**
Domain 2	14.20	Domain 1	14.50	Domain 3	16.25	Domain 2	15.75
Domain 3	12.20	Domain 2	19.18	Domain 101	195.80	Domain 4	7.96
Domain 5	19.30	Domain 23	3.00	Domain 102	195.80	Domain 9	3.00
Domain 6	20.90			Domain 103	195.80	Domain 14	3.00
Domain 28	9.35						
Domain 44	5.90						
Domain 47	3.00						
Domain 53	7.56						
Domain 65	3.00						
Domain 79	3.00						
Domain 97	3.00						
Domain 135	3.00						

Statistical analysis of the grade capped samples is contained in Appendix 3.

13.6 Compositing

Sample length analysis showed a predominant sample length of 1m with only minor populations above and below this length. Based on this it was decided to composite the data on 1m intervals with a minimum allowable composite interval of 0.5m. Compositing was carried out within each domain and composites coded by these domains. The composite lengths were averaged over the length of the domain to prevent individual small composites occurring at the end of each drillhole intercept. The averaging was carried out to produce composites as close as possible to the 1m interval specified. Statistical analysis of the composites by domain is contained in Appendix 4.

13.7 Continuity Analysis

Continuity analysis was undertaken prior to variography and was based on a Normal Score transformation of the 1m composite data. Continuity analysis refers to the analysis of the spatial correlation between sample pairs to determine the major axis of spatial continuity. Horizontal, across strike and down dip continuity maps were examined (and their underlying variograms) to determine the direction of greatest and least continuity. Continuity analysis was undertaken for the following domains: Domain 2 at Pridolinny Main Zone, Domain 2 at Gora5 and Domain 1 at Zhelanny. At Pridolinny Extension continuity analysis and subsequent variography was undertaken with all domains combined to provide sufficient sample pairs for analysis. Similarly, at Pravoberezhny continuity analysis and subsequent variography was also undertaken with all domains combined to provide sufficient sample pairs for analysis. An example continuity analysis for Pridolinny Main Zone is shown in Figure 13.6.

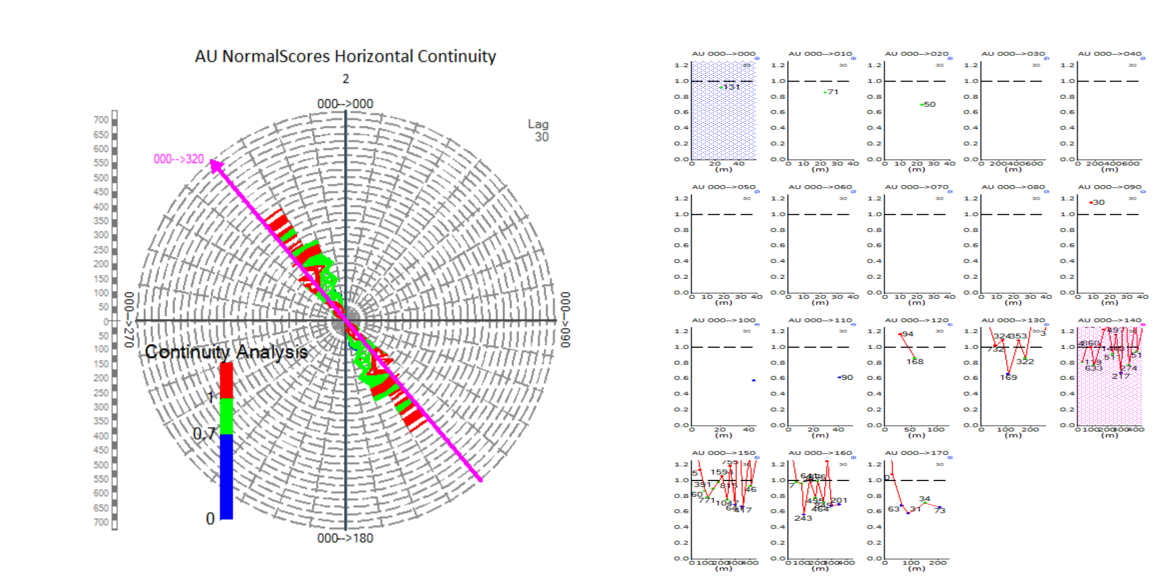


Figure 13.6: Continuity Map of Normal Score Au Values and Underlying Variograms for Domain 2 at Pridolinny

13.8 Variography

Based on the continuity analysis, variogram modelling was subsequently undertaken. Directional and down hole variograms were calculated for the 1m composites. In keeping with the general trend of mineralisation, the variograms were created in the orientation of the defined mineralisation as described by the continuity analysis. Along strike variograms were orientated at between 310° and 330°; down dip variograms were orientated at azimuths of between 40° and 60° dip and dipping sub-vertically; across strike variograms were defined based on the down hole variograms, but subsequently orientated to be orthogonal. The nugget variances were modelled from the down hole variograms based on a 1m lag reflecting the down hole composite spacing. Variograms were modelled using 3 structure spherical models. No variograms could be defined for Zhelanny high grade zone due to the limited number of sample pairs.

An example of the modelled variograms for Pridolinny Domain 2 are shown in Figure 13.7. Following variogram modelling the resultant variogram model (based on the normal score transformation) was subsequently back transformed.

All other continuity analysis and variography plots are contained in Appendix 5.

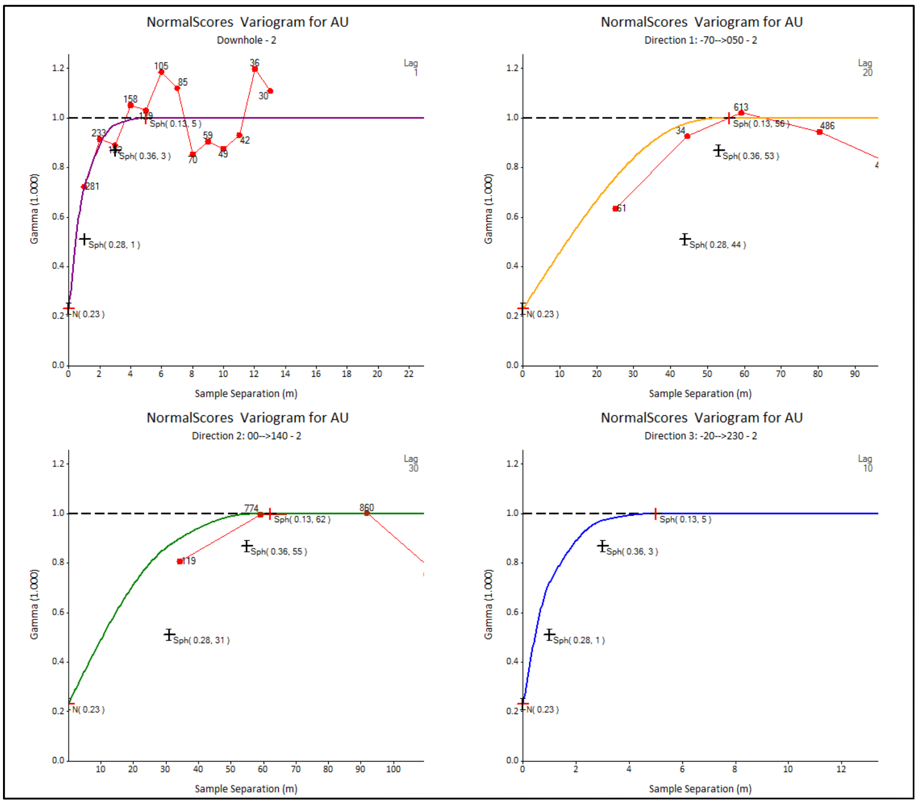


Figure 13.7: Modelled Variograms for Normal Score Au Grades at Pridolinny Domain 2

A summary of the modelled variogram parameters for all areas is shown in Table 13.3.

Table 13.3: Modelled Variogram Parameters

Area	Nugget	Structure 1 (m)			C1	Structure 2 (m)			C2	Structure 3 (m)			C3
		Down Dip	Along Strike	Across Strike		Down Dip	Along Strike	Across Strike		Down Dip	Along Strike	Across Strike	
Pridolinny Main Zone	0.33	44	31	1	0.38	53	55	3	0.22	56	62	5	0.08
Pridolinny Extension	0.35	32	49	1	0.28	52	83	2	0.22	62	108	3	0.15
Gora5	0.31	23	25	2	0.44	41	44	3	0.17	44	58	5	0.08
Zhellany	0.31	24	30	1	0.29	40	49	2	0.24	53	58	3	0.16
Pravoberezhny	0.28	35	42	1	0.27	72	88	2	0.29	100	114	3	0.17

WAI considers that the overall quality of the experimental variograms for the Nasedkino deposits are relatively poor and are affected by the relatively limited number of sample pairs available within each domain. In addition, the experimental variograms are somewhat dominated by the drill grid spacing. At Pridolinny Main Zone, Gora5 and Zhelanny the generated variograms potentially suggest ranges of between 40m and 60m in the along strike and down dip directions with moderately high nugget values of around 30%. At Pridolinny Extension and Pravoberezhny the generated variograms potentially suggest ranges of between 60m and 100m in the along strike and down dip directions with moderately high nugget values of around 30%.

13.9 Block Modelling

A block model defining the mineralised zones was constructed using the domain wireframes and subsequently coded by these principal domains. A model prototype comprising a parent cell size of 5m x 5m x 5m (X, Y, Z) was used for the mineralised zones as this is comparable to the selective mining unit. The impact of using a block size significantly smaller than the drill hole spacing was assessed during grade estimation with comparative estimates at larger block sizes undertaken. No significant impact to the resource estimate was found between the different parent block sizes. For the majority of domains sub-cell splitting was enabled down to a minimum cell size of 1.25m x 1.25m x 1m (X,Y,Z). Additional cell splits were used for thin zones or zones with complex geometry down to a minimum cell size of 0.5m x 0.5m x 1m (X,Y,Z).

A full block model comprising waste blocks outside of the mineralised zone and below the topographical survey was also constructed. The model prototype for the waste model comprised a parent cell size of 10m x 10m x 10m. The mineralised zone model and the full waste model were then combined and all blocks below the topographical surface selected. The oxide and overburden interfaces were then coded into the block model.

Directional control strings were used to define the local variation in the strike and dip of each mineralised structure. The orientation strings were used to define vectors from which the true dip and dip direction of each block was coded into the model. True dip and dip directions were then converted into SANG1, SANG2 and SANG3 (based on Datamine ZXY orientation). The SANG1, SANG2 and SANG3 fields were used to control the search ellipse orientation used in the estimation of grade for each domain.

13.10 Density

Density values were grouped based on lithology and mineralisation. It was noted that issues are present within the density database in which recurring density values for consecutive samples were observed and particularly for the 2006-2008 database. It is likely that either composite density samples were taken for these data, or the same density values were input for multiple samples. Care was taken to make sure that these samples were removed prior to density analysis. It is recommended that the reason for these recurring density samples be clarified and the database corrected. Density samples were selected based on location inside or outside of the mineralised zones wireframes.

Statistical analysis of the selected samples was then undertaken for Pridolinny, Gora 5, Zhelanny and Pravoberezhny as shown respectively in Figure 13.8, Figure 13.9, Figure 13.10 and Figure 13.11. Only samples less than 3m were used in the analysis. To prevent an over influence of anomalous values a maximum density of 3.56t/m^3 was used for Zhelanny mineralised zones and 3.30t/m^3 was used for Zhelanny and Gora5 waste.

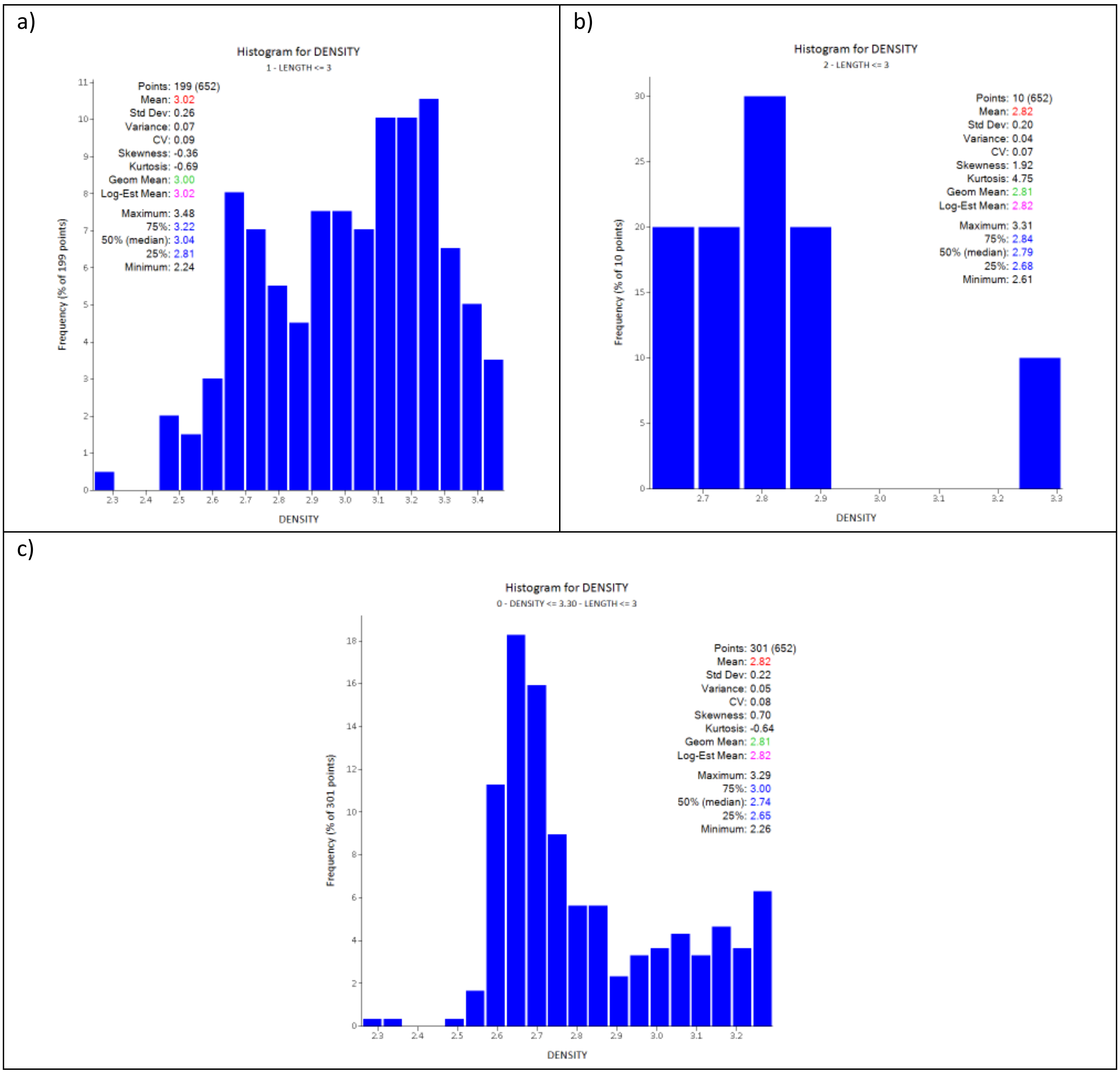


Figure 13.8: Histograms of Density Measurements for Pridolinny a) Mineralised Skarn Package b) Mineralised Gneiss Package c) Waste

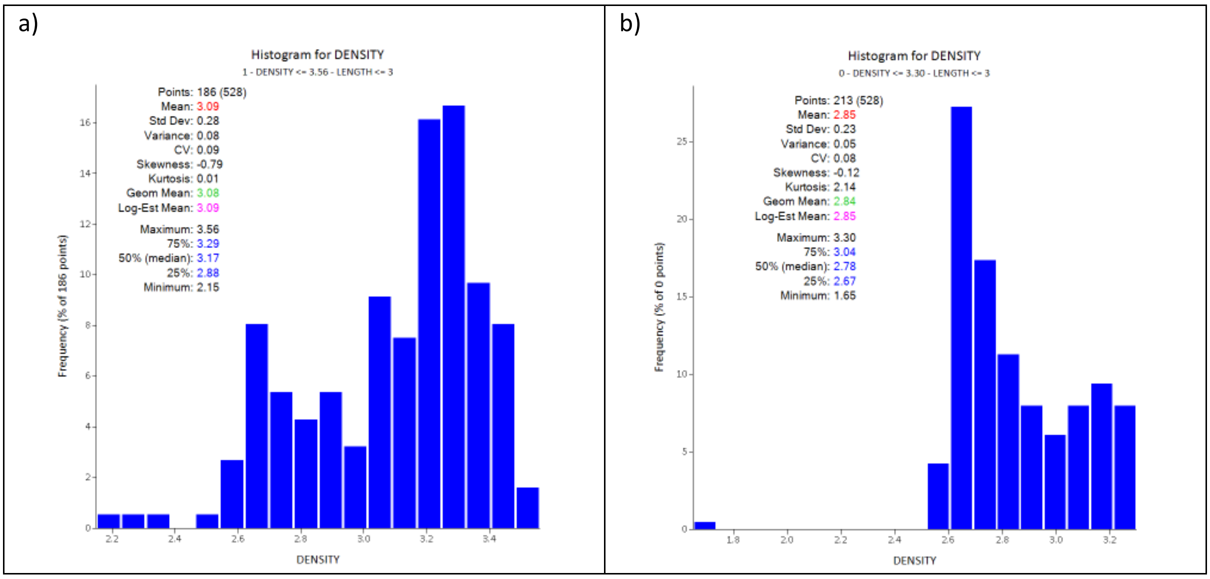


Figure 13.9: Histograms of Density Measurements for Gora5 a) Mineralised Skarn Package b) Waste

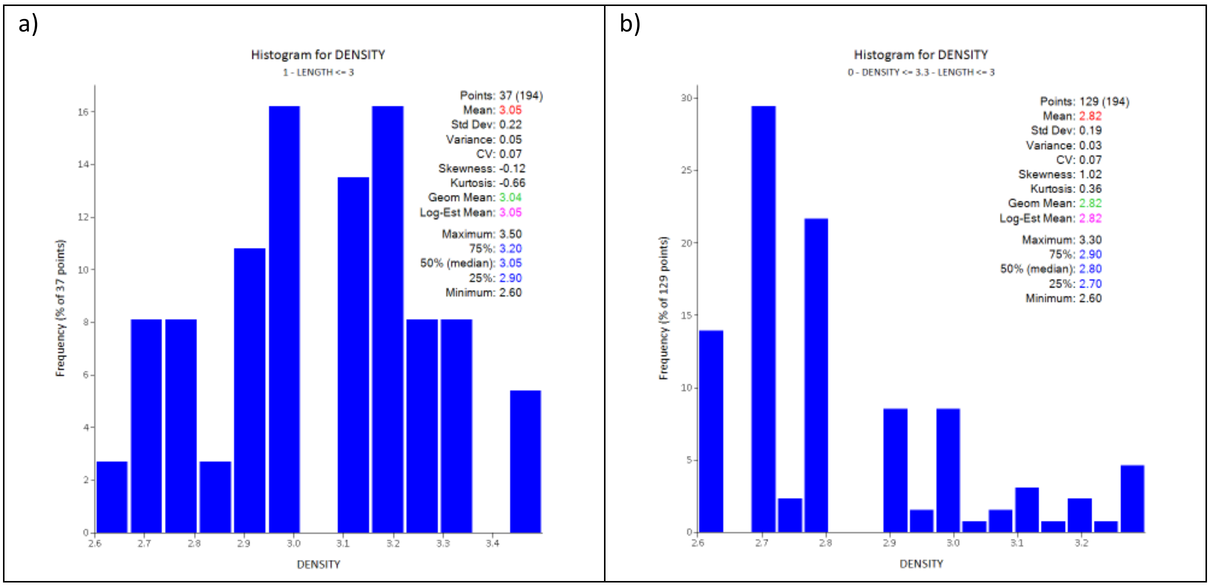


Figure 13.10: Histograms of Density Measurements for Zhelanny a) Mineralised Skarn Package b) Waste

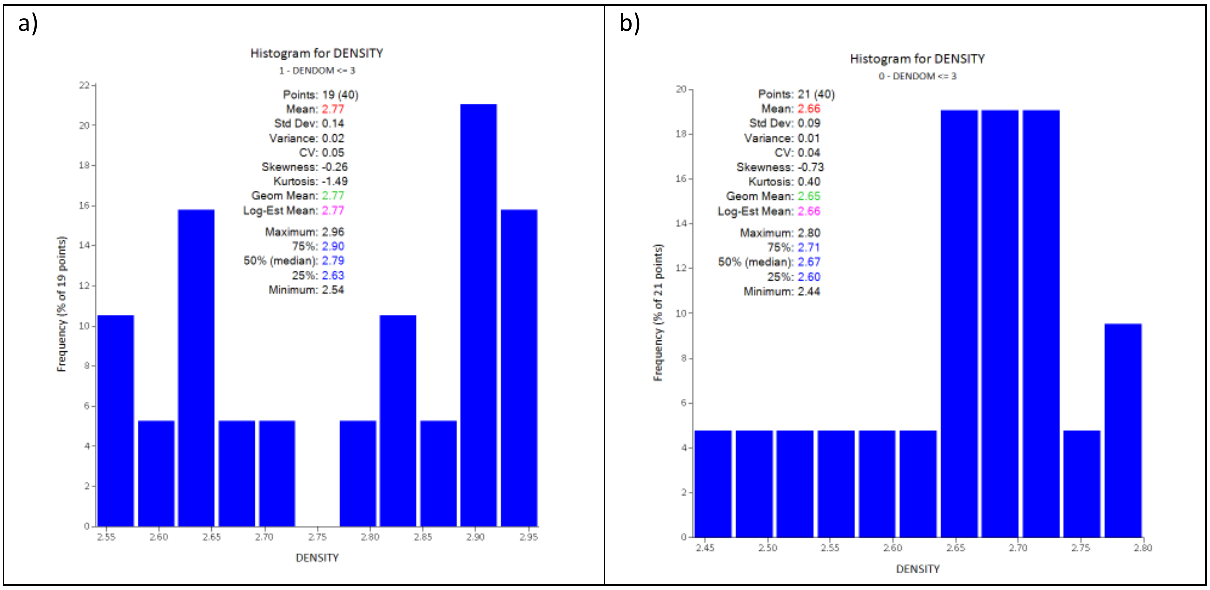


Figure 13.11: Histograms of Density Measurements for Pravoberezhny a) Mineralised Gneiss Package b) Waste

A summary of the density measurements used in the Mineral Resource estimate are shown in Table 13.4. No density measurements for oxide or overburden are available therefore assumed densities of 2.20t/m^3 for oxide and 2.00t/m^3 for overburden were used. The thickness of these units is not considered significant to the Mineral Resource estimate.

Table 13.4: Grouped Density Measurements

Deposit	Description	Number Samples	Density (t/m^3)		
			Minimum	Maximum	Mean
Pridolinny	Main Zone Mineralised Zones	199	2.24	3.48	3.02
	Pridolinny Extension Mineralised Zones	10	2.61	3.31	2.82
	Waste	301	2.26	3.29	2.82
Gora5	Mineralised Zones	213	2.15	3.56	3.09
	Waste	186	1.65	3.30	2.85
Zhelanny	Mineralised Zones	37	2.60	3.50	3.05
	Waste	129	2.60	3.30	2.82
Pravoberezhny	Mineralised Zones	19	2.54	2.96	2.77
	Waste	21	2.44	2.80	2.66

To improve lithological domaining of the project and assignment of density, it is again recommended that all lithological data are digitised and incorporated into the drill hole database. In addition, it is recommended that additional density measurements be undertaken, particularly for Pravoberezhny where there is limited data.

13.11 Grade Estimation

13.11.1 Introduction

Grade estimation was performed only on mineralised material defined within each domain. The domains were treated as hard boundaries and as such composites from an adjacent domain could not be used in the grade estimation of another domain. Ordinary kriging (OK), inverse distance weighting to power 3 (IDW^3) and nearest neighbour (NN) estimations were undertaken.

13.11.2 Grade Estimation Plan

Grade estimation was run in a three pass plan, the second and third passes using progressively larger search radii to enable the estimation of blocks unestimated on the previous pass. The search parameters were derived from the variography, with the first search distances corresponding to the variogram range. Sample weighting during grade estimation was determined by semi-variogram model parameters for the OK method for the low grade zones. The high grade zones at Zhelanny were estimated using IDW^3 and NN estimation only as no suitable variograms for these zones could be derived. Search distances for these zones were based on the general drill hole spacing.

Grade estimation was carried out using a parent block size of 5m x 5m x 5m as this is comparable to the selective mining unit. To assess the impact of using a block size significantly smaller than the drill hole spacing comparative estimates using a 10m x 10m x 5m parent cell size were also undertaken by WAI. No significant impact to the resource estimate was found between the different parent block sizes.

Sub-cells received the same grade as the parent cell. Block discretisation was set to 3 x 3 x 5 to estimate block grades. Search ellipse orientations were controlled by dynamic anisotropy. Any remaining unestimated zones were assigned the average grade for the domain and assigned a search volume of 4. A summary of the grade estimation plan is shown in Table 13.5.

Table 13.5: Nasedkino Grade Estimation Plan

Pridolinny Main Zone						
	Search Distance (m)			**Composites**		**Drillholes**
Search	**Down Dip**	**Along Strike**	**Across Strike**	**Minimum**	**Maximum**	**Minimum**
1st	56	62	10	6	15	2
2nd	112	124	20	6	15	2
3rd	168	186	30	6	15	1
Pridolinny Extension						
	Search Distance (m)			**Composites**		**Drillholes**
Search	**Down Dip**	**Along Strike**	**Across Strike**	**Minimum**	**Maximum**	**Minimum**
1st	62	108	10	4	15	2
2nd	124	216	20	4	15	2
3rd	186	324	30	4	15	1
Gora 5						
	Search Distance (m)			**Composites**		**Drillholes**
Search	**Down Dip**	**Along Strike**	**Across Strike**	**Minimum**	**Maximum**	**Minimum**
1st	44	58	10	6	15	2
2nd	88	116	20	6	15	2
3rd	132	174	30	6	15	1
Zhelanny (Normal Grade Zones)						
	Search Distance (m)			**Composites**		**Drillholes**
Search	**Down Dip**	**Along Strike**	**Across Strike**	**Minimum**	**Maximum**	**Minimum**
1st	53	58	10	6	15	2
2nd	106	116	20	6	15	2
3rd	159	174	30	6	15	1
Zhelanny (High Grade Zones)						
	Search Distance (m)			**Composites**		**Drillholes**
Search	**Down Dip**	**Along Strike**	**Across Strike**	**Minimum**	**Maximum**	**Minimum**
1st	25	25	10	6	15	2
2nd	50	50	20	6	15	2
3rd	75	75	30	6	15	1
Note: A maximum search radius of 10m x 10m x 10m applied to drill hole 607 in DOMAIN 101 and drill hole 1033 in DOMAIN 201 (no expansions to this search) to limit influence of localised high grades.						
Pravoberezhny						
	Search Distance (m)			**Composites**		**Drillholes**
Search	**Down Dip**	**Along Strike**	**Across Strike**	**Minimum**	**Maximum**	**Minimum**
1st	100	114	10	4	15	2
2nd	200	228	20	4	15	2
3rd	300	342	30	1	15	1

13.11.3 Grade Estimation Validation

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;
- Mean grade comparison (Appendix 6), and
- SWATH analysis (Figure 13.12).

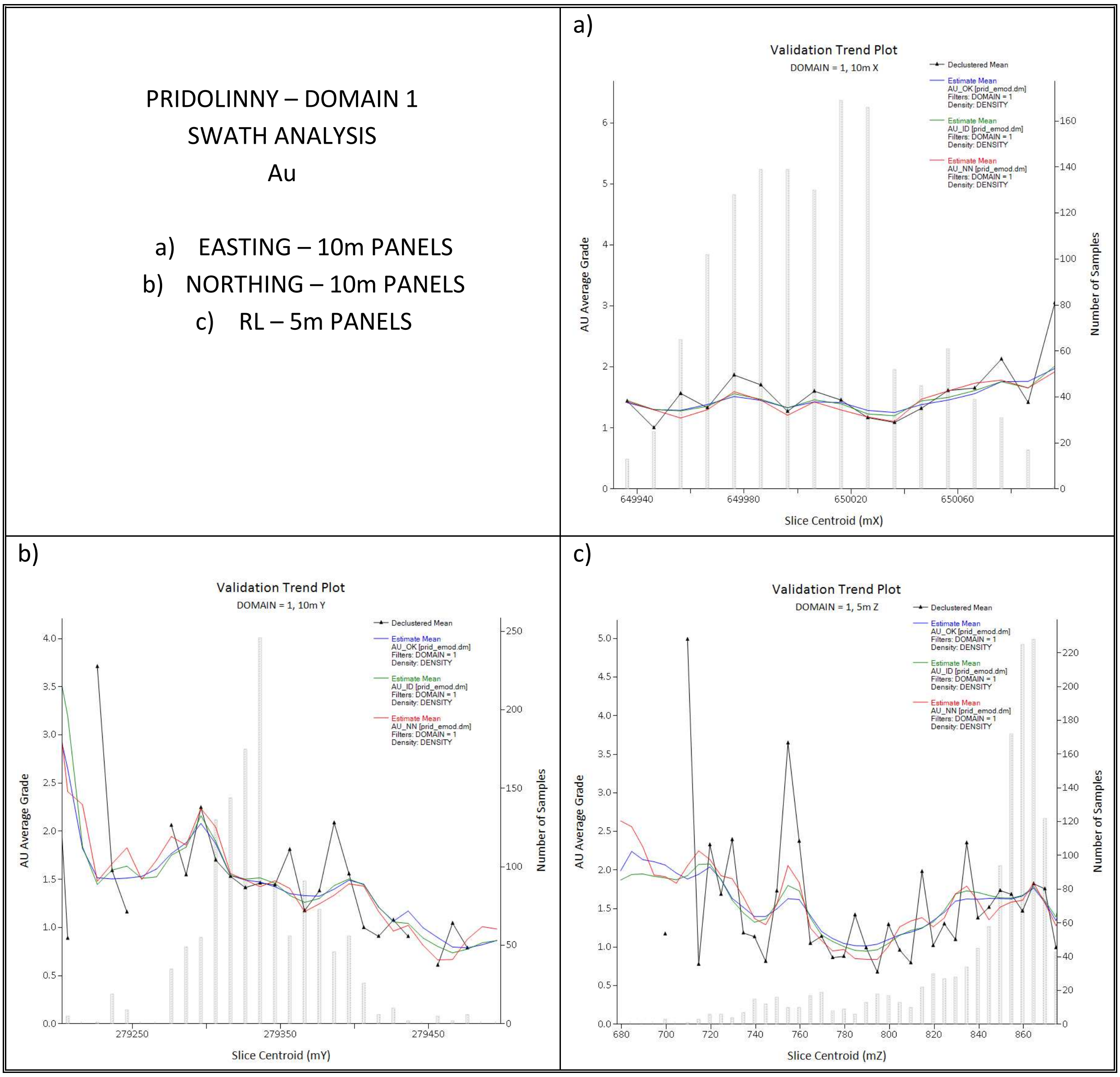


Figure 13.12: Example SWATH Analysis of Domain 1 at Pridolinny

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 IDW^3 estimation to best represent the sample data used. The Mineral Resource estimate is there based upon the IDW^3 estimation. The OK grade estimation and NN grade estimation were used for comparative purposes only.

13.12 Mineral Resource Reconciliation

No reconciliation has been carried out as no recorded production is available for the property.

13.13 Mineral Resource Classification

13.13.1 General

The Mineral Resource classification of the Nasedkino Gold Project incorporated the confidence in the drillhole data, the geological interpretation, geological continuity, data density and orientation, spatial grade continuity and confidence in the Mineral Resource estimation. The Mineral Resource classification for the Nasedkino Gold Deposit is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2012)].

13.13.2 Measured Mineral Resources

No Measured Mineral Resources were classified at the Nasedkino Gold Project. Although some of the drilling at Pridolinny is based on a 10m x 10m grid spacing, this drilling has predominantly been derived from the 2006-2008 drilling campaign which was subject to check assaying in 2012. The results, while acceptable, do however indicate a relatively high level of grade variability and as such no Measured Mineral Resources were assigned to this drilling. This drilling was subsequently assigned a maximum classification of Indicated Mineral Resources only.

13.13.3 Indicated Mineral Resources

Indicated Mineral Resources were defined on a zone by zone basis and were classified using limits around contiguous areas of blocks that were generally encountered during the first search of the grade estimation up to a 50m x 50m drill grid spacing. A minimum number of three drill holes were required. The spatial location and continuity of the mineralised zone was also considered when assigning Indicated Mineral Resources. Areas covered by a drill grid spacing of 10m x 10m were also classified as Indicated Mineral Resources due to the issues addressed above.

13.13.4 Inferred Mineral Resources

Inferred Mineral Resources were classified based on the remaining blocks defined within the mineralised zone wireframes that were generally encountered during the second or third search of the grade estimation. Typically Inferred Mineral Resources correspond to areas of 100m drilling grid spacing or isolated zones with limited drilling and/or limited continuity of mineralisation.

The Mineral Resource classification at Pridolinny, Gora 5, Zhelanny and Pravoberezhny are shown in Figure 13.13, Figure 13.14, Figure 13.15 and Figure 13.16, respectively.

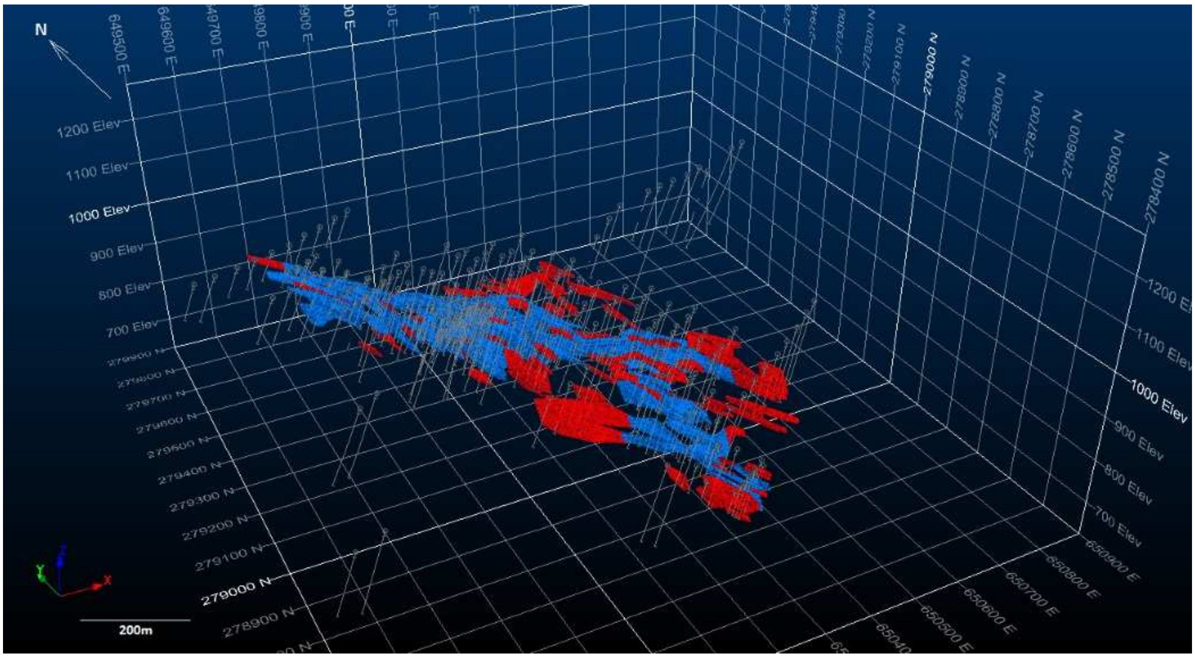


Figure 13.13: Mineral Resource Classification and Drill Holes at Pridolinny (Indicated Resources coloured blue, Inferred Resources coloured red)

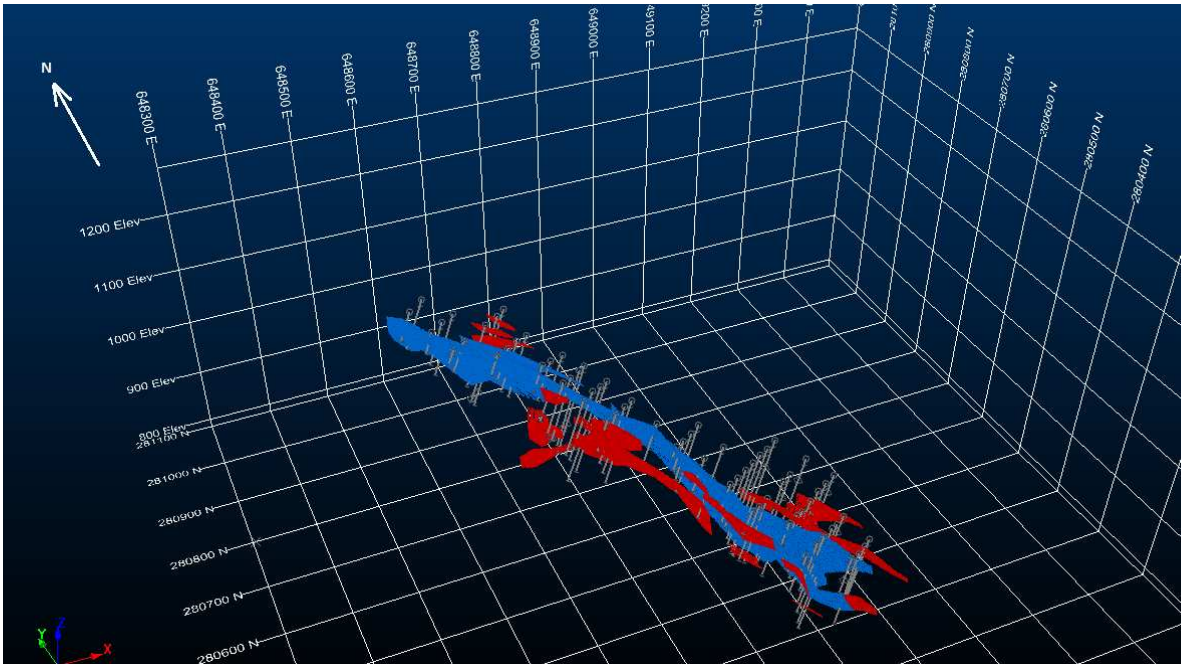


Figure 13.14: Mineral Resource Classification and Drill Holes at Gora 5 (Indicated Resources Coloured Blue, Inferred Resources Coloured Red)

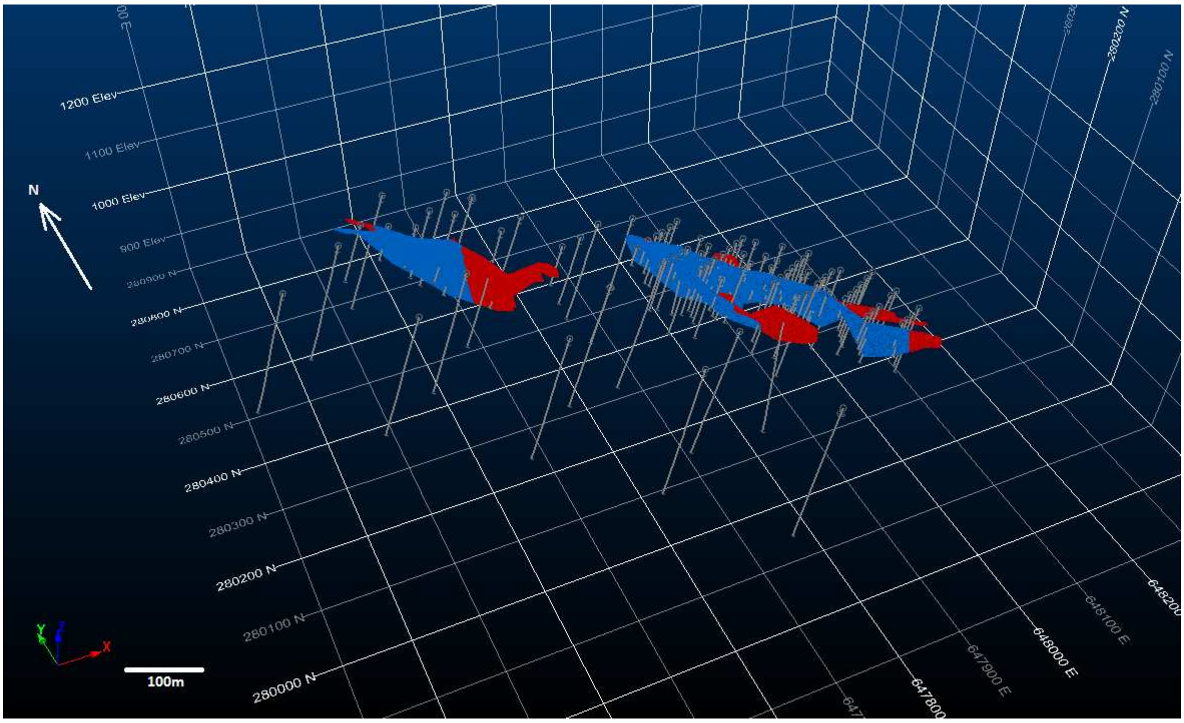


Figure 13.15: Mineral Resource Classification and Drill Holes at Zhelanny
(Indicated Resources Coloured Blue, Inferred Resources Coloured Red)

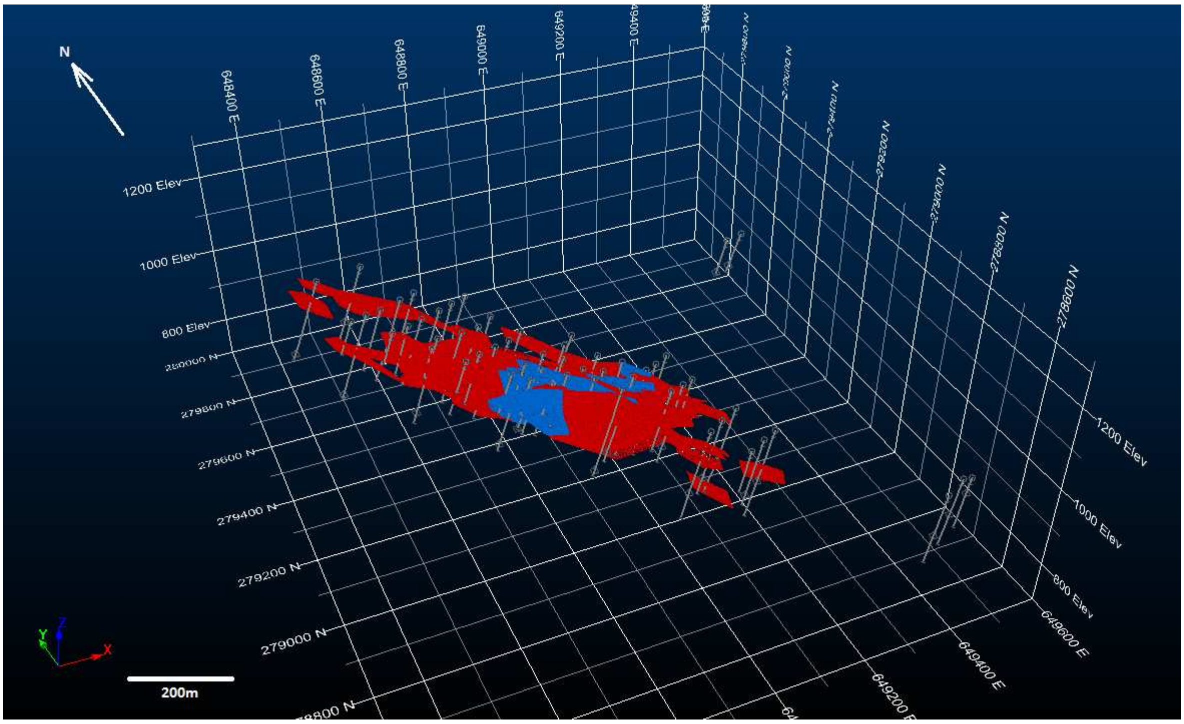


Figure 13.16: Mineral Resource Classification and Drill Holes at Pravoberezhny
(Indicated Resources Coloured Blue, Inferred Resources Coloured Red)

13.14 Pit Optimisation for Resource Delineation

An open pit optimisation was carried out to delineate the open pit Mineral Resources. Optimisation of the block model was carried out by WAI using NPV Scheduler™. A summary of the optimisation parameters used in the delineation of the open pit Mineral Resources are shown in Table 13.6.

Table 13.6: Pit Optimisation Parameters for Delineation of Mineral Resources

Description	Unit	Pridolinny	Gora 5	Zhelanny	Pravoberezhny
Gold Price	US$/oz	1,500	1,500	1,500	1,500
	US$/g	48.23	48.23	48.23	48.23
Selling Cost	% of revenue	6.00	6.00	6.00	6.00
	US$/g	2.89	2.89	2.89	2.89
Effective Gold Price	US$/oz	1,410	1,410	1,410	1,410
	US$/g	45.33	45.33	45.33	45.33
Mining Cost	US$/t	0.83	0.83	0.83	0.83
Plant Cost	US$/t	15.08	15.08	15.08	15.08
Mining Dilution	%	10.0	10.0	10.0	10.0
Mining Recovery	%	97.5	97.5	97.5	97.5
Process Recovery	%	88.84	91.09	93.75	95.06
Overall Slope Angle	Degrees	45	45	45	45

13.15 Mineral Resource Statement

13.15.1 Mineral Resources

The Mineral Resource estimate for the Nasedkino Gold Project is classified in accordance with the Australian 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, and legal, title, taxation, and 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, at this time.

The effective date of the Mineral Resource estimate is February 10, 2017. A summary of the Mineral Resource statement is shown in Table 13.7.

Table 13.7: Nasedkino Gold Project Mineral Resource Estimate (WAI, February 10, 2017)

	Cut-Off (g/t Au)	Resource Classification	Tonnes (Kt)	Au Grade (g/t)	Au Metal (kg)	Au Metal (Koz)
Pridolinny	0.70	Indicated	5,410	1.79	9,696	312
		Inferred	895	1.84	1,644	53
Gora 5	0.70	Indicated	1,926	2.26	4,354	140
		Inferred	375	1.87	702	23
Zhellany	0.70	Indicated	1,748	3.77	6,592	212
		Inferred	402	1.83	737	24
Pravoberezhny	0.70	Indicated	1,128	2.11	2,383	77
		Inferred	1,669	1.80	3,002	97
Total	**0.70**	**Indicated**	**10,212**	**2.25**	**23,025**	**740**
		Inferred	**3,341**	**1.82**	**6,085**	**196**

Notes:
1. Mineral Resources limited by an optimised pit shell using a gold price of $1500/oz and reasonable economic and technical parameters;
2. Mineral Resources are not reserves until they have demonstrated economic viability based on a feasibility study or pre-feasibility study;
3. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery and;
4. Numbers may not add due to rounding.

13.15.2 Grade Tonnage Curves

Grade tonnage curves for Pridolinny, Gora 5, Zhelanny and Pravoberezhny deposits are shown in Figure 13.17, Figure 13.18, Figure 13.19 and Figure 13.20, respectively.

Au Cut-Off (g/t)	Tonnes (Kt)	Au (g/t)	Metal (kg)
0.70	5,410	1.79	9,696
0.80	5,366	1.80	9,662
0.90	5,205	1.83	9,524
1.00	4,899	1.88	9,232
1.10	4,477	1.96	8,788
1.20	3,995	2.06	8,234
1.30	3,536	2.17	7,661
1.40	3,114	2.28	7,091
1.50	2,749	2.39	6,563

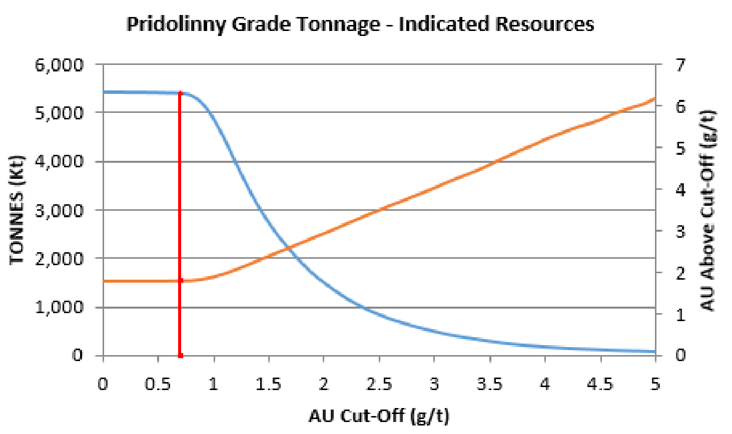


Figure 13.17: Grade Tonnage Curve for Pridolinny Indicated Resources

Au Cut-Off (g/t)	Tonnes (Kt)	Au (g/t)	Metal (kg)
0.70	1,926	2.26	4,354
0.80	1,916	2.27	4,347
0.90	1,895	2.28	4,329
1.00	1,829	2.33	4,266
1.10	1,756	2.38	4,189
1.20	1,677	2.44	4,098
1.30	1,585	2.51	3,982
1.40	1,492	2.58	3,857
1.50	1,393	2.67	3,713

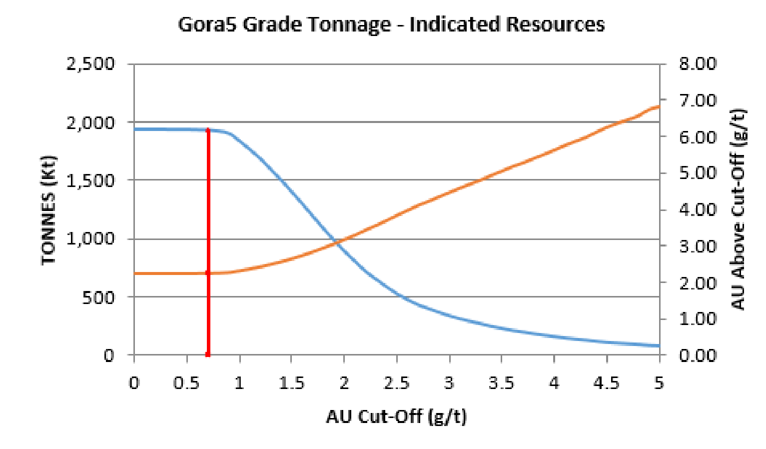


Figure 13.18: Grade Tonnage Curve for Gora5 Indicated Resources

Au Cut-Off (g/t)	Tonnes (Kt)	Au (g/t)	Metal (kg)
0.70	1,748	3.77	6,592
0.80	1,748	3.77	6,592
0.90	1,740	3.78	6,585
1.00	1,697	3.86	6,544
1.10	1,623	3.98	6,467
1.20	1,545	4.13	6,376
1.30	1,456	4.30	6,266
1.40	1,354	4.52	6,128
1.50	1,265	4.74	5,998

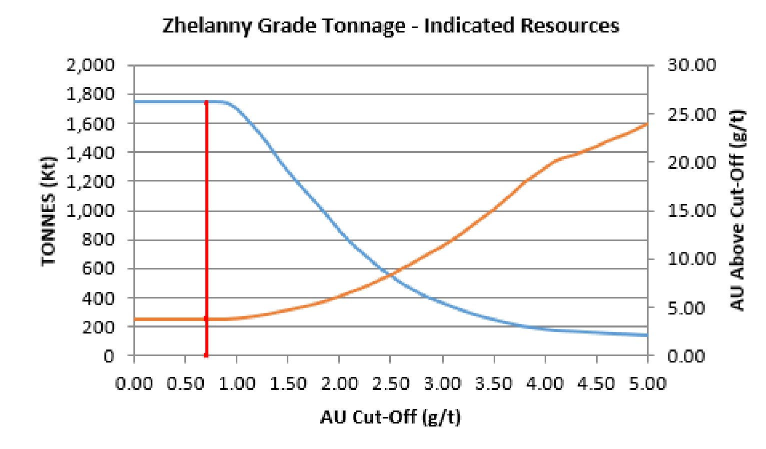


Figure 13.19: Grade Tonnage Curve for Zhelanny Indicated Resources

Au Cut-Off (g/t)	Tonnes (Kt)	Au (g/t)	Metal (kg)
0.70	1,128	2.11	2,383
0.80	1,111	2.13	2,371
0.90	1,056	2.20	2,324
1.00	978	2.30	2,249
1.10	883	2.43	2,150
1.20	784	2.60	2,036
1.30	715	2.73	1,949
1.40	644	2.88	1,854
1.50	586	3.02	1,769

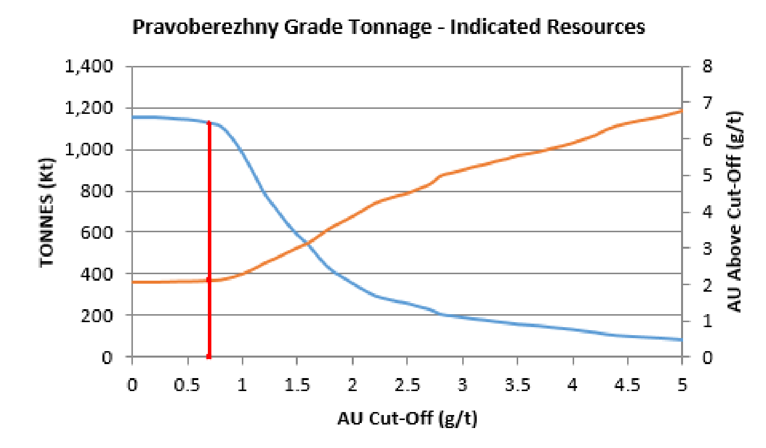


Figure 13.20: Grade Tonnage Curve for Pravoberezhny Indicated Resources

13.15.3 Comparison to Previous Mineral Resource Estimates

A previous Mineral Resource estimate was undertaken by SRK in 2012. The Mineral Resource estimate was prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

Mineral Resources reported by SRK are shown in Table 13.8 and were reported within an optimised pit shell based on a $1,600/oz gold price and using a cut-off grade of 0.55g/t Au.

Table 13.8: Previous Nasedkino Gold Project Mineral Resource Estimate (SRK, May 1, 2012)

	Resource Classification	Tonnage (Mt)	Au Grade (g/t)	Cu Grade (%)	Contained Au (Koz)	Contained Cu (Kt)
Pridolinny	Indicated	10.0	1.1	0.19	360	19.0
	Inferred	1.0	1.1	0.12	36	1.3
Gora 5	Indicated	3.1	1.8	0.10	174	3.0
	Inferred	0.2	0.9	0.14	6	0.3
Zhelanny	Indicated	1.2	3.4	0.05	134	0.6
	Inferred	0.2	1.1	0.31	8	0.7
Pravoberezhny	Inferred	2.5	1.8	0.05	145	1.3
Glukharinny	Inferred	3.1	1.7	0.06	170	1.8
Zhelanny 3	Inferred	0.3	1.5	0.05	15	0.2
Total	**Indicated**	**14.3**	**1.5**	**0.16**	**668**	**22.6**
	Inferred	**7.3**	**1.6**	**0.08**	**380**	**5.6**

The main differences between the Mineral Resource estimates are:

- Re-interpretation from a 0.3g/t Au wireframe cut-off to a 0.7g/t Au wireframe cut-off has resulted in a reduction in tonnes and an increase in grade;
- Infill drilling undertaken at Zhelanny, Pravoberezhny and Glukharinny during 2015. Glukharinny is no longer included in the Mineral Resource estimate as it is considered to not have economic potential due to issues associated with metallurgical processing;
- Zhelanny 3 reported in the SRK Mineral Resource estimate has been included as part of Zhelanny in the WAI Mineral Resource estimate; and
- Copper is not reported in the WAI Mineral Resource estimate as the updated process route for the project does not include copper production.

14 ORE RESERVE

14.1 Introduction

This section of the report describes the conversion of the WAI Mineral Resource estimation into an Ore Reserve Estimate for the Nasedkino deposit. The effective date of the Ore Reserve estimate is 10 February 2017.

The Nasedkino project contains four main deposits: Zhelanny, Gora 5, Pridolinny and Pravoberezhny. The Ore Reserve Estimate was conducted for each deposit separately, and was carried out following the guidelines of the CIM Definition Standards for Ore Reserves. Estimated Ore Reserve tonnages were based on a long term price forecast of US$1,100/oz Au.

Ore Reserves are reported in order of decreasing confidence as either Proved Ore Reserves (based upon *Measured* Mineral Resources) or Probable Ore Reserves (based upon *Indicated* Mineral Resources). As such, Ore Reserves for the Nasedkino deposit are classified as Probable, based upon the *Indicated* Mineral Resources for each orebody. *Inferred* material has been considered as waste with zero grade.

The Ore Reserve estimation was completed using industry standard practices for open pit using Studio 5D Planner and NPV Scheduler (NPVS) Datamine softwares. The software was used to generate optimal pit envelopes using NPVS and subsequent mine design using 5D Planner, before final scheduling was completed on the mine design in NPVS.

14.2 Open Pit Planning

The methodology for mine design and scheduling comprised the following steps:

- Preparation of Block model for use in Open pit optimisation;
- Pit optimisation including the production of series of optimal pit shells;
- Selection of the most appropriate ultimate pit shell for subsequent pit design;
- Pit design, followed by evaluation to produce an Ore Reserve estimate for the Open pits; and
- Final scheduling based on the final pit designs.

14.3 Open Pit Optimisation & Mine Design

14.3.1 Introduction

WAI undertook open pit optimisation for the Nasedkino deposits, prior to the development of detailed pit designs.

Pit optimisation is a recognised technique by which different open pit shells may be generated, based on a supplied geological resource block model, and a number of user-defined economic and operating parameters. WAI carried out optimisation using CAE Mining's NPV Scheduler (NPVS) software which offers various facilities for mine scheduling and optimisation of the pit shell extents. A general overview of the way NPVS is structured and operated is shown in the flowsheet in Figure 14.1. The main input data and parameters for pit optimisation include:

- Resource block model;
- Economic parameters;
- Processing parameters;
- Slope parameters; and
- Any existing pit limits.

The principal output data from pit optimisation includes:

- Block models and surface wireframes of optimal pit shells;
- Evaluation and economic data for optimal pits;
- Wireframes of suggested pushbacks; and
- Evaluation and economic data for long term mining schedules.

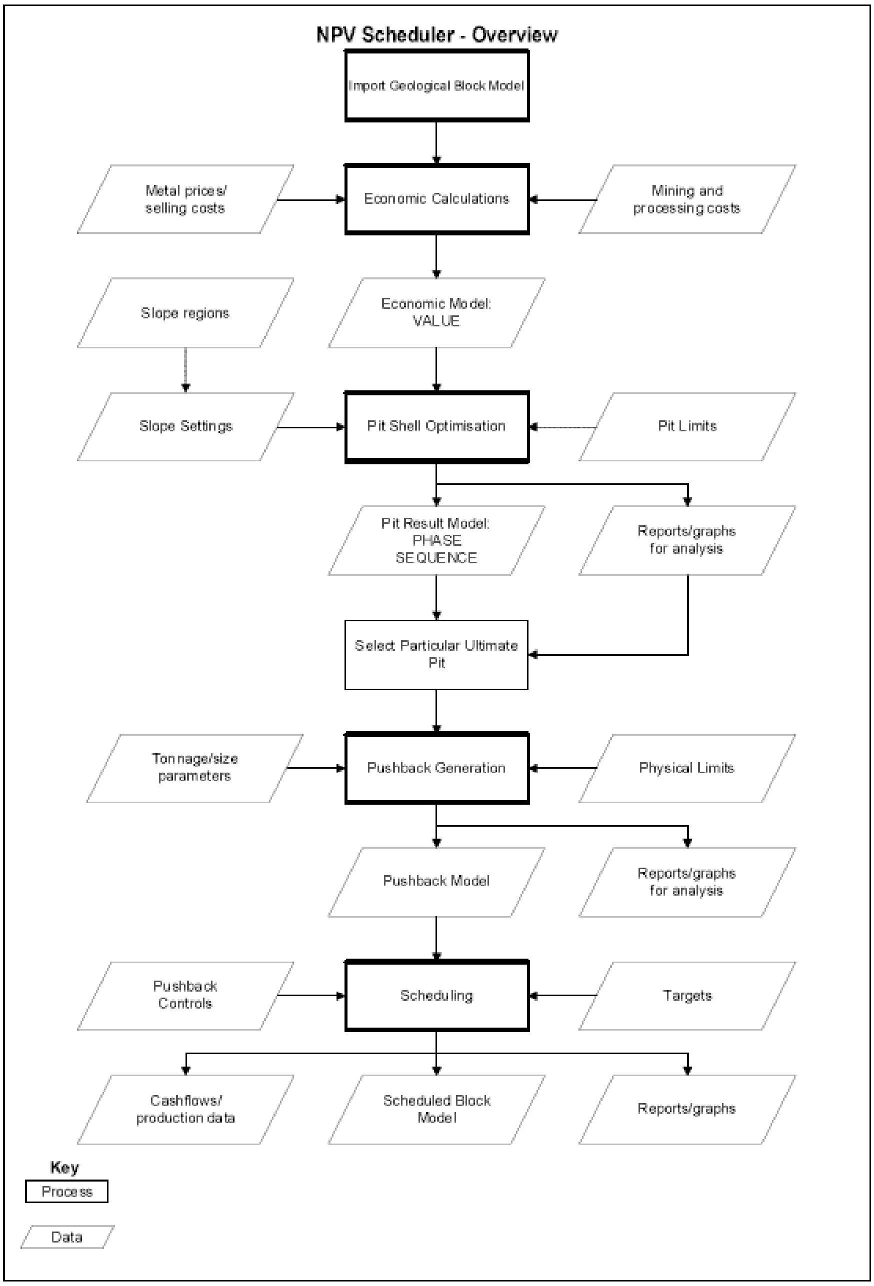


Figure 14.1: NPV Scheduler Process Flow

14.3.2 Mineral Resource Base

Open pit optimisation and mine design for the purpose of Ore Reserve Estimation has been carried out based upon the WAI Mineral Resource block models for the four deposits:

- Gora-5: *gora5_opt.dm*
- Pridolinny: *prid_opt.dm*
- Pravoberezhny: *parvo_opt.dm*
- Zhelanny: *zhel_opt.dm*

14.3.3 Optimisation Parameters

Only *Measured* and *Indicated* Mineral Resources may be considered for conversion to Ore Reserves under guidelines of the CIM Definition Standards for Ore Reserves. As such WAI has only considered these categories of Mineral Resources when evaluating the deposit.

WAI has based the optimisation parameters on data presented by Daltsvetmet, considering market studies, quotations from suppliers, as well as reasonable projections or estimates; supplemented by WAI estimates where appropriate. Table 14.1 and Table 14.2 below present the data used for open pit optimisation.

<table>
<tr><th colspan="7">Table 14.1: Pit Optimisation Parameters</th></tr>
<tr><th>Parameter</th><th>Unit</th><th colspan="4">Value</th><th>Source</th></tr>
<tr><td colspan="2">Site</td><td>Gora 5</td><td>Pridolinny</td><td>Pravober.</td><td>Zhelanny</td><td></td></tr>
<tr><td>Exchange Rate</td><td>RUB:US$</td><td colspan="4">65</td><td rowspan="3">Mangazeya Financial Model</td></tr>
<tr><td rowspan="2">Metal Price</td><td>RUB/g</td><td colspan="4">2299</td></tr>
<tr><td>US$/oz</td><td colspan="4">1,100</td></tr>
<tr><td>Cut-off grade for Au</td><td>g/t</td><td colspan="4">0.7</td><td rowspan="4">TEO 2016</td></tr>
<tr><td>Discount Rate</td><td>%</td><td colspan="4">10</td></tr>
<tr><td>Ore specific gravity</td><td>t/m^3</td><td colspan="2" rowspan="2">2.79</td><td colspan="2" rowspan="2">2.8</td></tr>
<tr><td>Waste specific gravity</td><td>t/m^3</td></tr>
<tr><td rowspan="2">Waste Mining Cost</td><td>RUB/t</td><td colspan="4">54.0</td><td rowspan="6">Mangazeya financial model</td></tr>
<tr><td>US$/t</td><td colspan="4">0.83</td></tr>
<tr><td rowspan="2">Ore Mining Cost</td><td>RUB/t</td><td colspan="4">51.9</td></tr>
<tr><td>US$/t</td><td colspan="4">0.80</td></tr>
<tr><td rowspan="2">Ore Treatment Cost (inc G&A)</td><td>RUB/t</td><td colspan="4">980</td></tr>
<tr><td>US$/t</td><td colspan="4">15.08</td></tr>
<tr><td rowspan="2">Rehabilitation Costs</td><td>RUB/m^3</td><td colspan="4">4.11</td><td rowspan="6">TEO 2016</td></tr>
<tr><td>US$/m^3</td><td colspan="4">0.06</td></tr>
<tr><td>Royalties</td><td>%</td><td colspan="4">6</td></tr>
<tr><td rowspan="2">Refining</td><td>RUB/g</td><td colspan="4">5.5</td></tr>
<tr><td>US$/g</td><td colspan="4">0.08</td></tr>
<tr><td>Metallurgical Recovery</td><td>%</td><td>91.09</td><td>88.84</td><td>95.06</td><td>93.75</td></tr>
<tr><td>Mining Dilution</td><td>%</td><td>2.4</td><td>5.62</td><td>10</td><td>10</td><td rowspan="2">WAI estimate/TEO 2016</td></tr>
<tr><td>Mining Losses</td><td>%</td><td>1.81</td><td>3.44</td><td>2.5</td><td>2.5</td></tr>
<tr><td>Overall Slope Angle</td><td>°</td><td colspan="4">45</td><td>TEO 2016</td></tr>
<tr><td>Pit base</td><td>m</td><td colspan="4">21</td><td>TEO 2016</td></tr>
</table>

Table 14.2: Pit Design Parameters

Parameter	Unit	Value	Source
Face Angle of non-working Slope	°	53	Technical Project 2016
Safety berm width	m	8	TEO 2016
Haul Road Berm Width	m	21	
Haul Road Berm Width at lower horizons	m	15	
Bench Height	m	5	
Bench Height of non-working bench	m	20	
Face Angle of non-working bench	°	70	
Haul Road Gradient	%	8	

14.3.4 Optimisation Results

WAI has used NPVS to produce the open pit optimisation for the Nasedkino deposits. A series of pit shells were generated at varying metal price factors ranging from 1% to 120% of the base data. A notional discounted revenue was reported for each pit shell, enabling comparison of the economic value of the pit shell and determination of the optimum economic return mining scenario.

The LG Phase analyses is shown in Figure 14.2 to Figure 14.5 below.

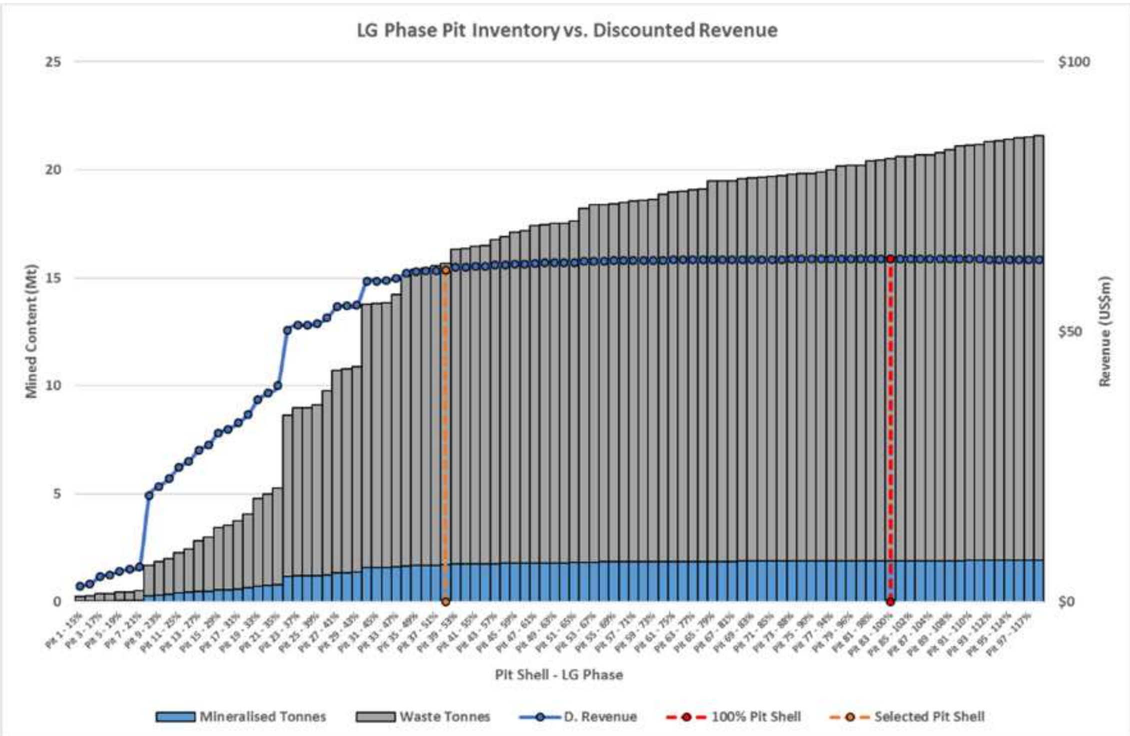


Figure 14.2: Gora 5 LG Phase Graph

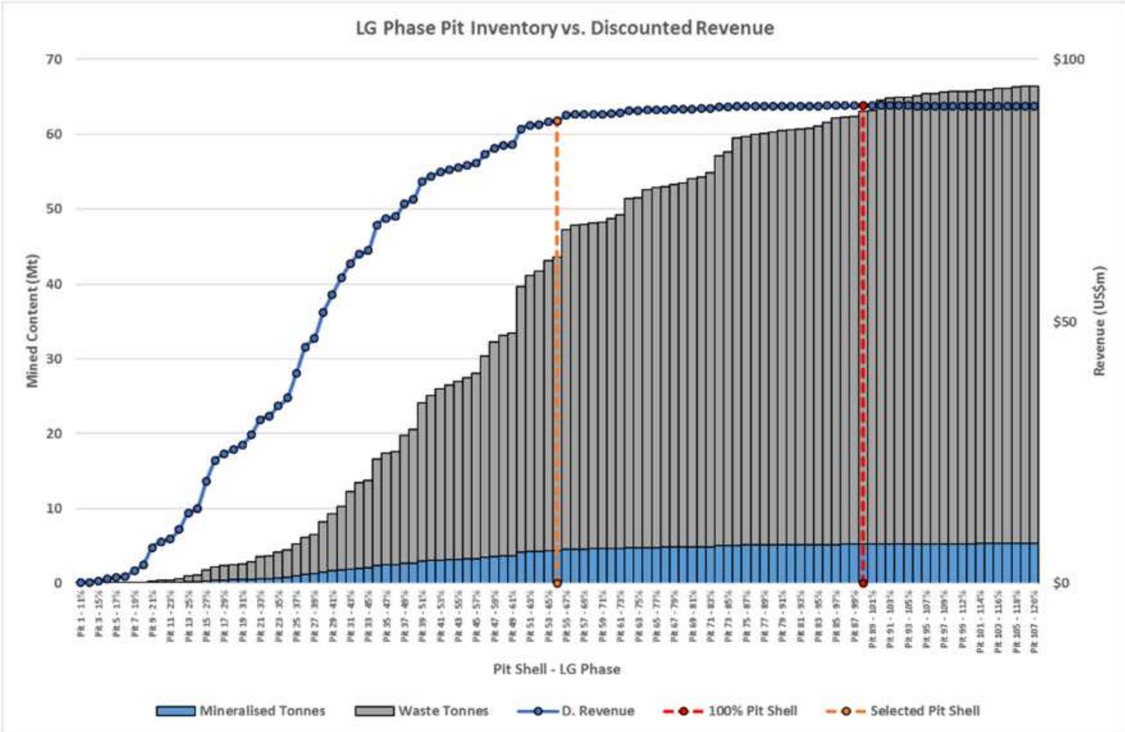


Figure 14.3: Pridolinny LG Phase Graph

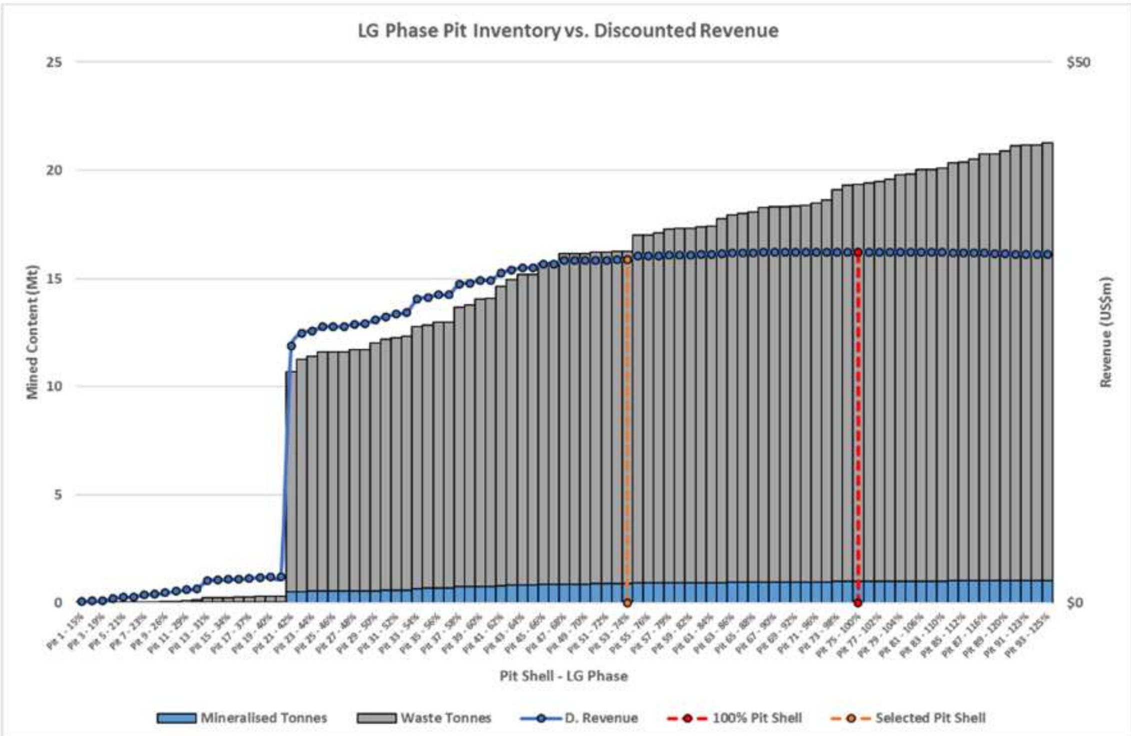


Figure 14.4: Pravoberezhny LG Phase Graph

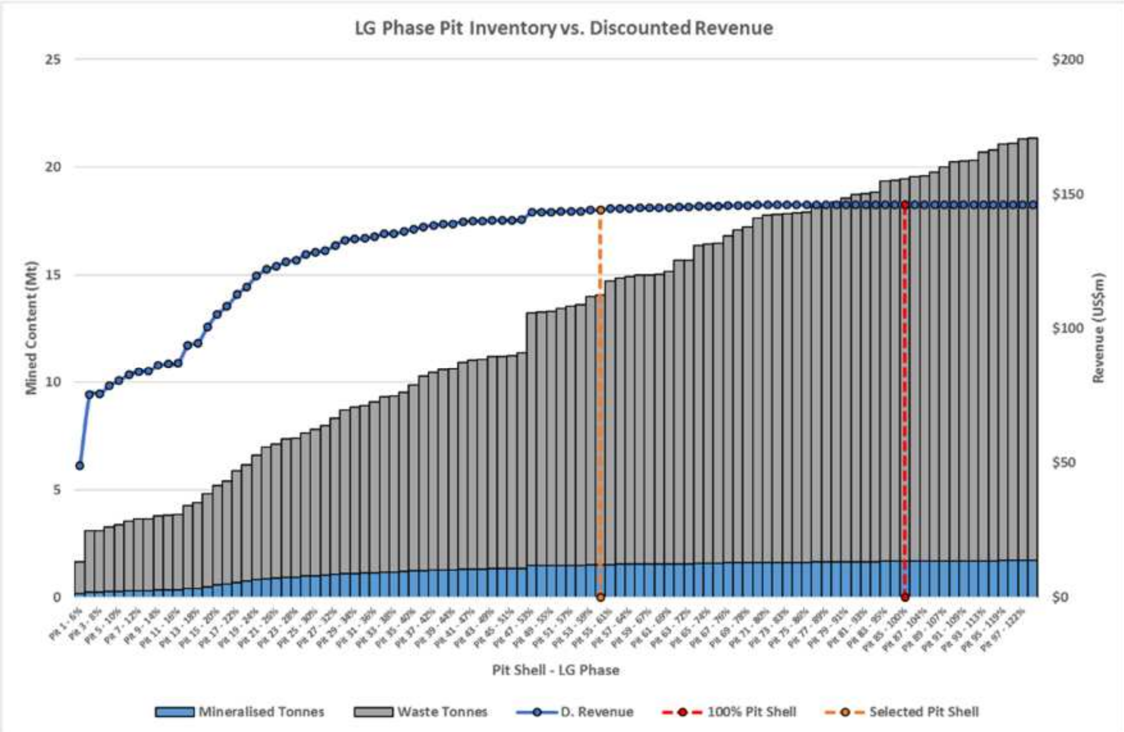


Figure 14.5: Zhelanny LG Phase Graph

14.3.5 Analysis of Optimisation Results & Pit Shell Selection

Selection of the optimised pit shell is generally made in order to consider maximising the notional NPV whilst minimising the quantity of material extracted and thus minimise the stripping ratio due to the sensitivity of NPV to early waste extraction.

In accordance with Daltsvetmet strategy, WAI has considered the 100% LG Phase shells for delineation of the open pit design and estimation of Ore Reserves.

The selected 100% LG Phase pit shells are presented in Figure 14.6 to Figure 14.9 below:

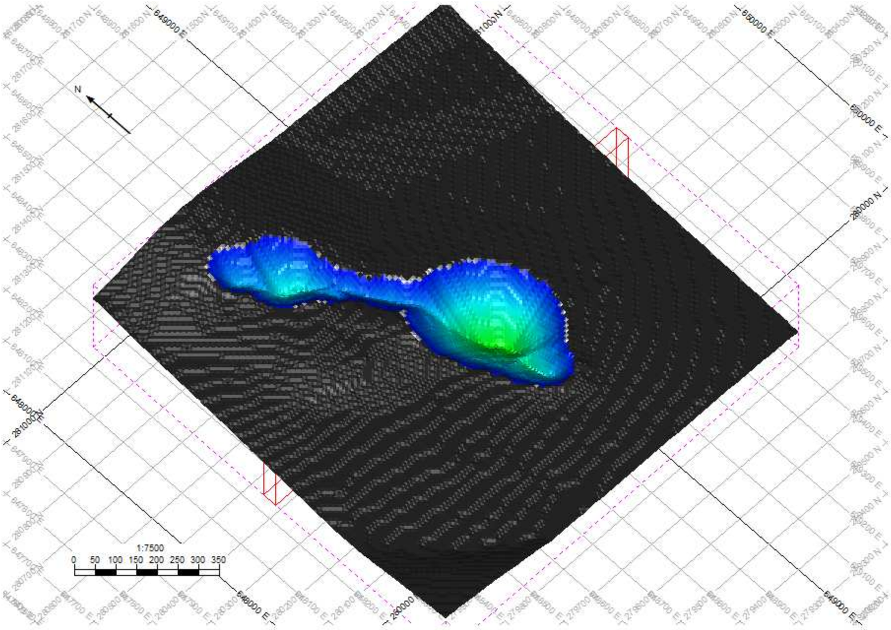


Figure 14.6: Gora 5 100% Optimised Pit 3D Section

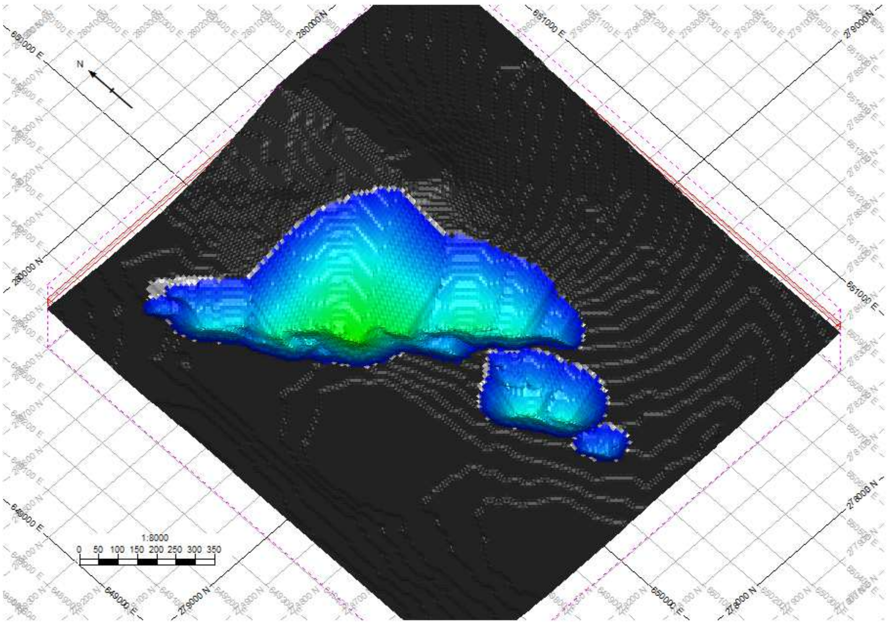


Figure 14.7: Pridolinny 100% Optimised Pit 3D Section

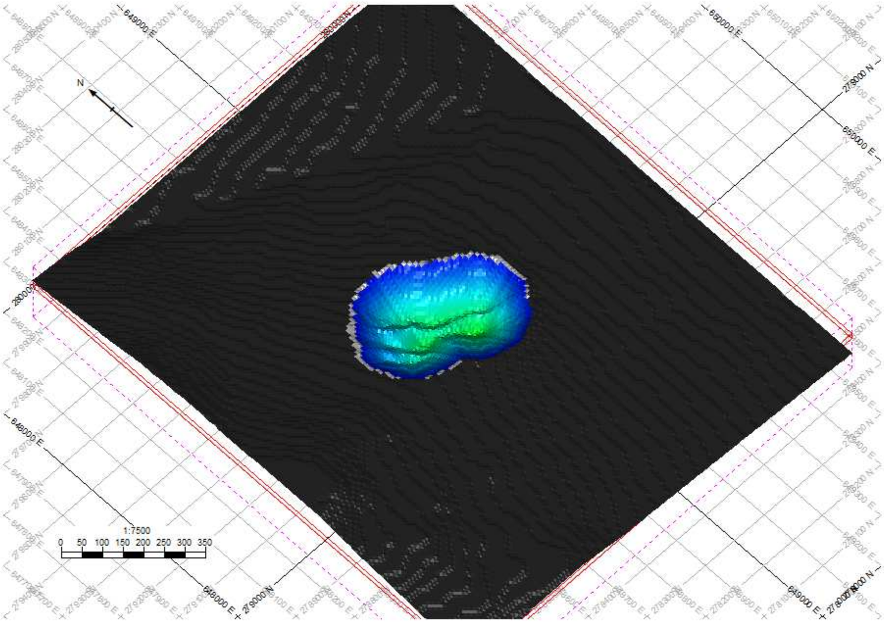


Figure 14.8: Pravoberezhny 100% Optimised Pit 3D Section

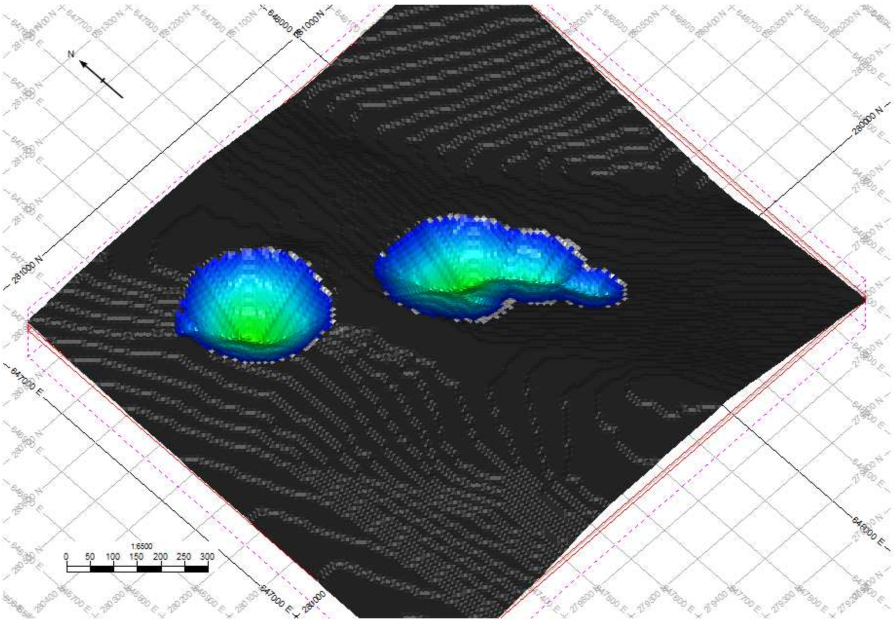


Figure 14.9: Zhelanny 100% Optimised Pit 3D Section

14.3.6 Mine Designs

WAI has utilised the derived optimised pit shells as a basis for the mine design, in conjunction with the pit design parameters in Section 14.3.3 and geotechnical analysis described in Section 15.3. Mine design has been carried out using Datamine Studio5-D Planner software, initially as strings, before linking to form a mine design wireframe.

Using the optimised pit shell, contours have been generated on the main mining levels and used to generate strings representing the toe and crest of the operating bench faces. Modifications to the toe and crest strings have then been made in order to align the pit design with the geotechnical and operational constraints, whilst attempting to minimise deviation from the optimised shell.

The mine designs also take due cognisance of constraints derived from the mining process and equipment to be employed. The smallest area available for mining within the pit is defined by the size of equipment to be employed in the mine. The minimum size of the pit base must be large enough to accommodate an excavator and truck with sufficient space to load and turn the truck. Minimum mining width will be approximately 40m, with minimum pit base size of 40m by 120m to allow sufficient working faces.

The designed pit content, inclusive of losses and dilution, is presented in Table 14.3 below:

Table 14.3: Pit Design Inventory

Deposit	Ore (Mt)	Grade (Au g/t)	Gold (t)	Waste (Mt)
Gora 5	1.76	2.22	3.91	18.6
Pridolinny	4.57	1.67	7.63	35.1
Pravoberezhny	1.02	1.96	2.00	20.0
Zhelanny	1.79	3.51	6.18	20.5
TOTAL	**9.11**	**2.17**	**19.72**	**94.2**

Figure 14.10 to Figure 14.21 present the open pit designs for each of the four orebodies.

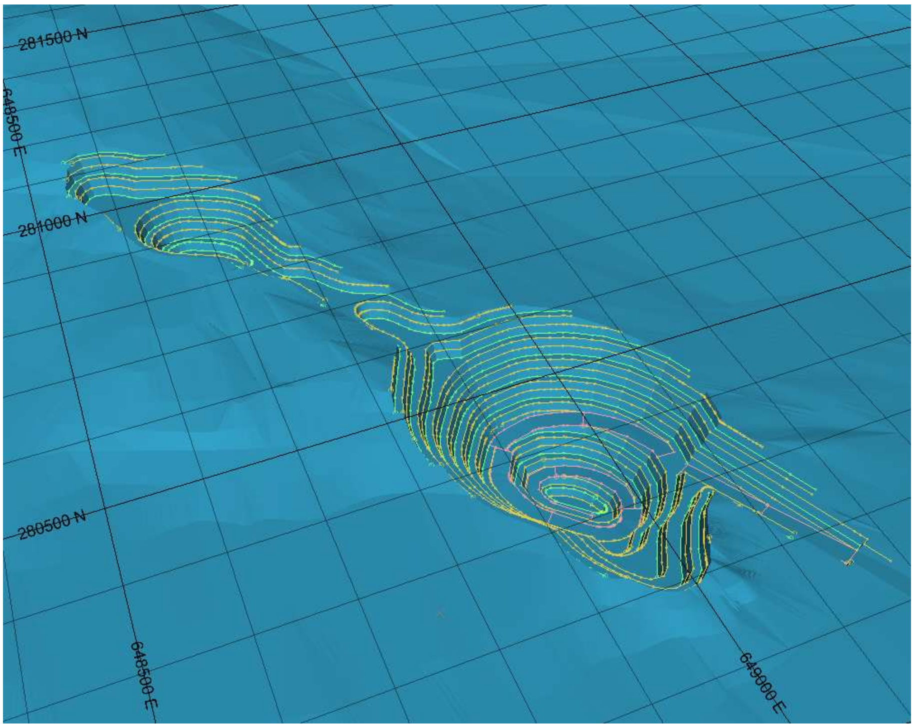


Figure 14.10: Isometric Projection of Gora 5 Open Pit (100m grid)

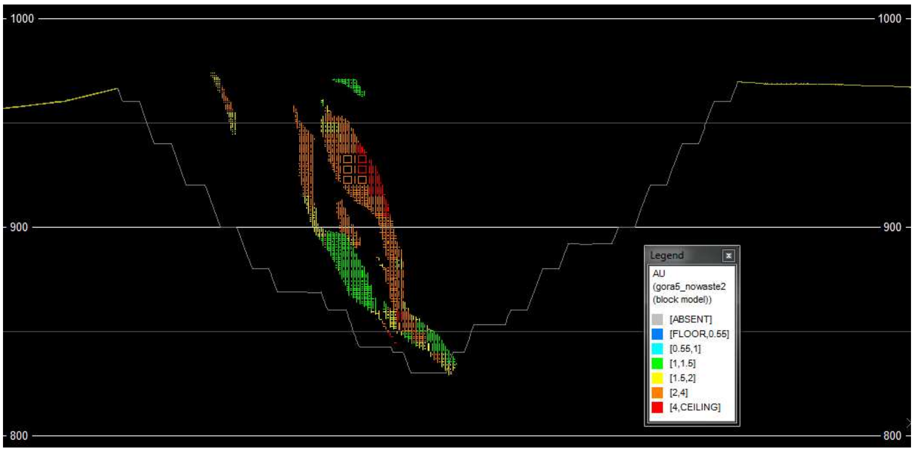


Figure 14.11: Cross Section with Au mineralisation through Gora 5 Main Pit (looking SE)

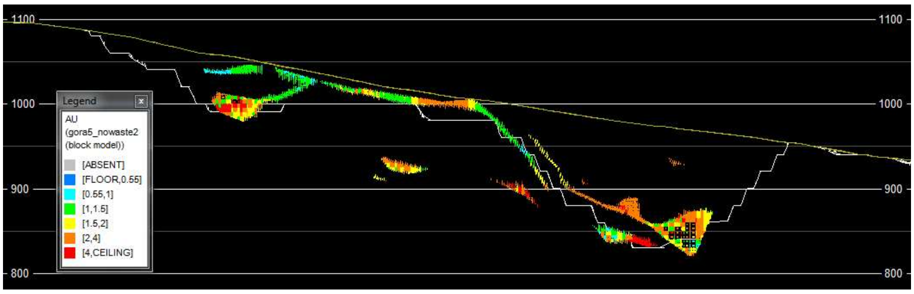


Figure 14.12: Long Section with Au mineralisation through Gora 5 Main Pit (looking NE)

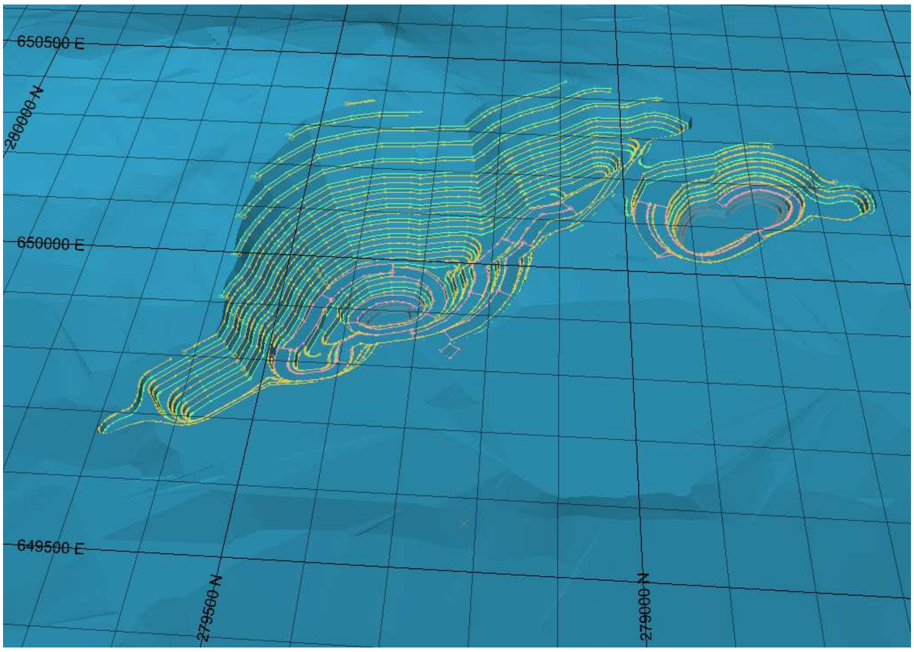


Figure 14.13: Isometric Projection of Pridolinny Open Pit (100m grid)

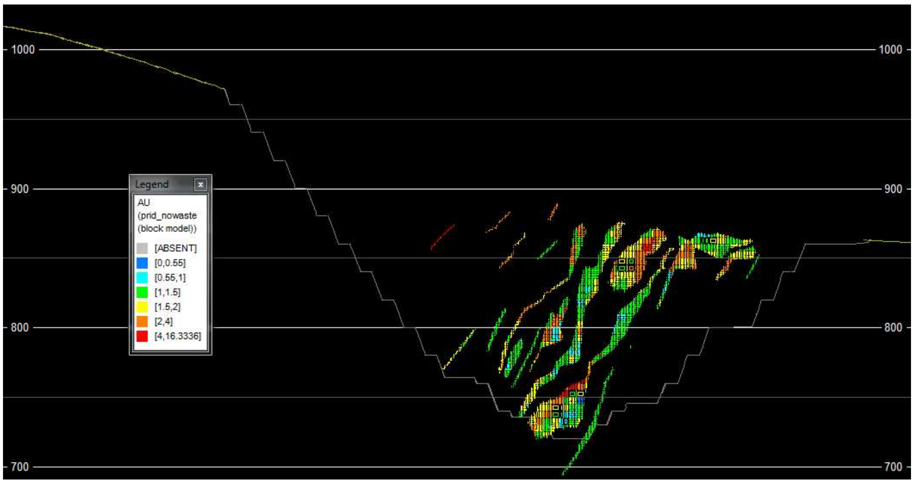


Figure 14.14: Cross Section with Au mineralisation through Pridolinny Main Pit (looking SE)

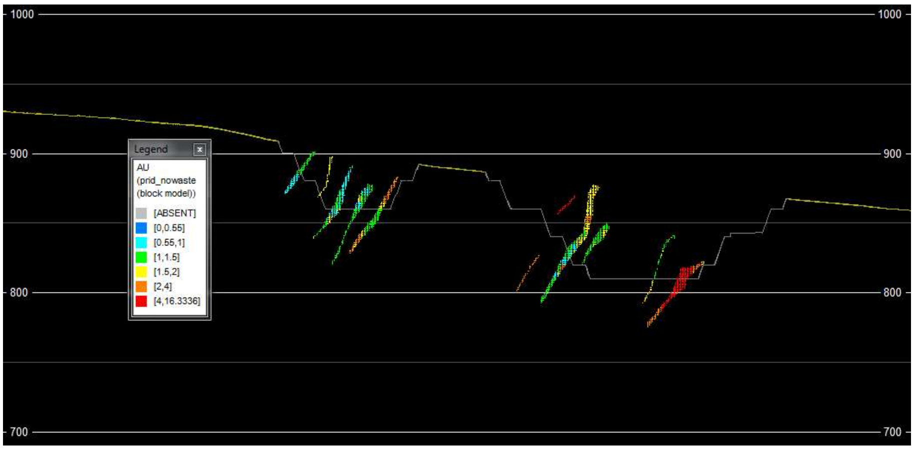


Figure 14.15: Cross Section with Au mineralisation through Pridolinny Satelite Pit (looking SE)

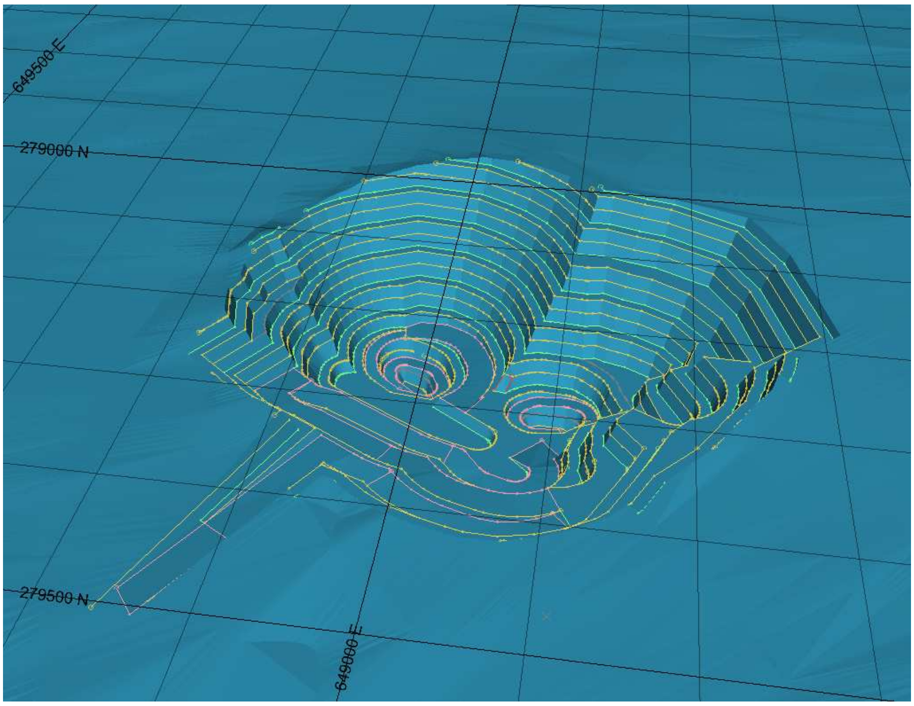


Figure 14.16: Isometric Projection of Pravoberezhny Pit (100m Grid)

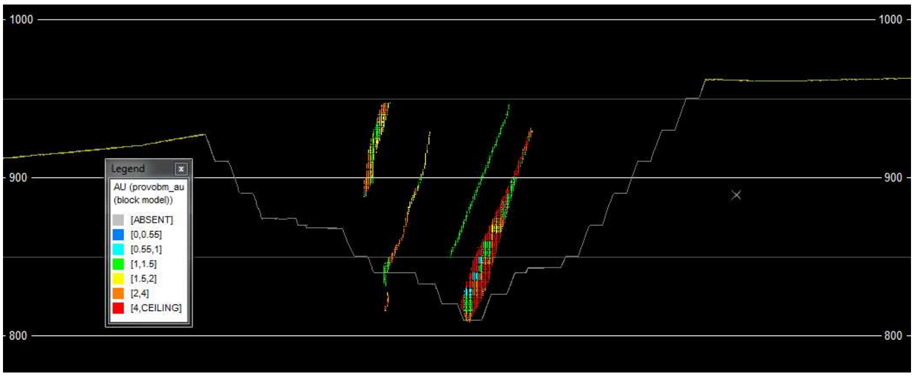


Figure 14.17: Cross Section with Au mineralisation through Pravoberezhny Pit (looking SE)

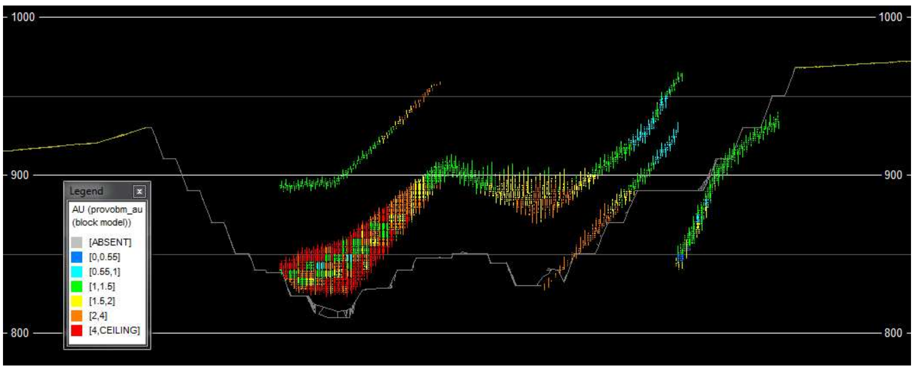


Figure 14.18: Long Section with Au mineralisation through Pravoberezhny Pit (looking SW)

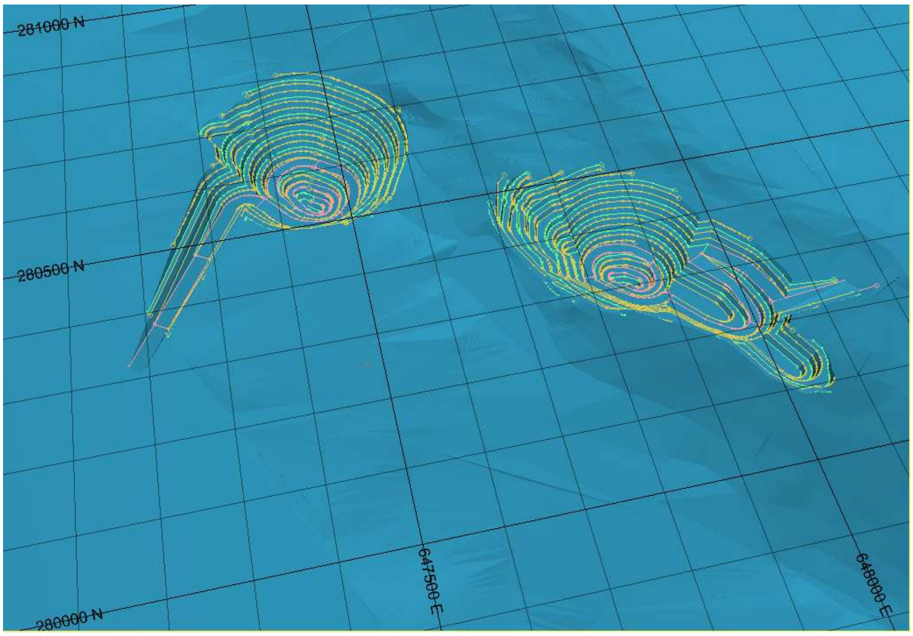


Figure 14.19: Isometric Projection of Zhelanny Pits (100m Grid)

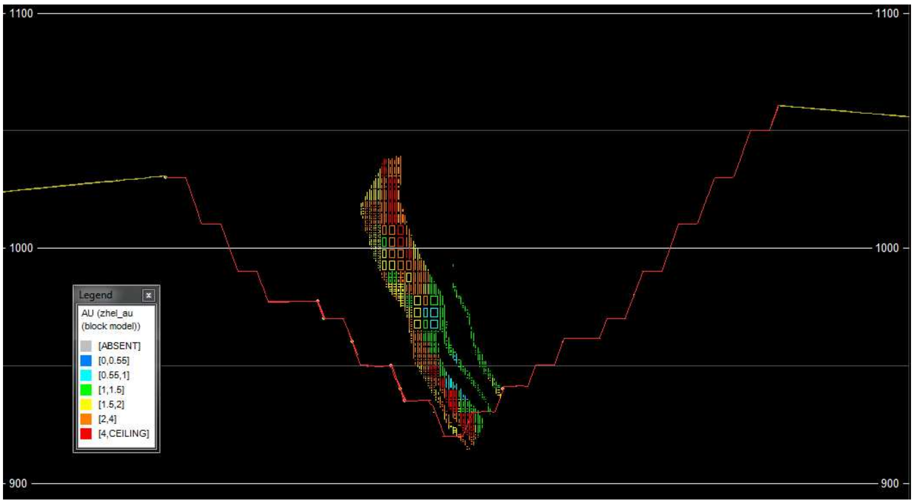


Figure 14.20: Cross Section of Zhelanny (West) Pit with Au Mineralisation (viewed looking SE)

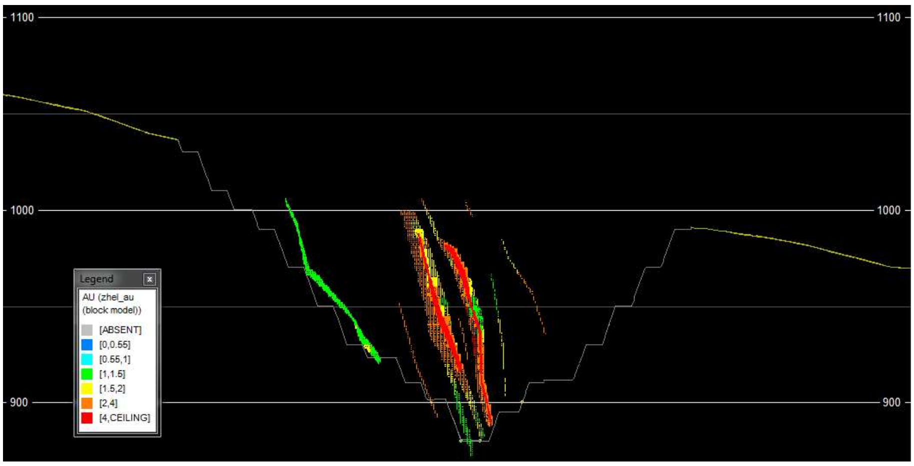


Figure 14.21: Cross Section of Zhelanny (East) Pit with Au Mineralisation (viewed looking NW)

14.3.7 Mining Schedule

Following derivation of the mine designs, a combined mining schedule was derived using NPV Scheduler software to produce annual pit shells and year-by-year mineable tonnages and grades for ore and waste.

The mining schedule, presented below in Table 14.4, is predicated upon a total ROM ore tonnage of 1.1Mtpa, after losses and dilution have been considered.

Table 14.4: Nasedkino Life-of-Mine Schedule

Deposit	Year	2018	2019	2020	2021	2022	2023	2024	2025	2026	TOTAL
Zhelanny ROM Schedule	*Ore (kt)*	550	1,101	109							**1,760**
	Grade (g/t Au)	2.71	4.02	2.41							**3.51**
	Gold (t)	*1.49*	*4.43*	*0.26*							***6.18***
	Waste (kt)	7,291	12,883	308							**20,482**
Gora 5 ROM Schedule	*Ore (kt)*			990	770						**1,761**
	Grade (g/t Au)			2.33	2.08						**2.22**
	Gold (t)			*2.31*	*1.6*						***3.91***
	Waste (kt)			13,666	4,962						**18,627**
Pravoberezhny ROM Schedule	*Ore (kt)*	63*	95*	168*	260	436					**1,021**
	Grade (g/t Au)	1.51	1.51	1.69	1.23	2.66					**1.96**
	Gold (kg)	*0.09*	*0.14*	*0.28*	*0.32*	*1.16*					***2.00***
	Waste (kt)	2,825	3,756	3,094	5,191	5,096					**19,961**
Pridolinny ROM Schedule	*Ore (kt)*					682	1,101	1,099	1,100	585	**4,567**
	Grade (g/t Au)					1.81	1.7	1.68	1.56	1.64	**1.67**
	Gold (t)					*1.24*	*1.88*	*1.85*	*1.71*	*0.96*	***7.63***
	Waste (kt)					7,530	8,255	8,071	8,073	3,181	**35,110**
TOTAL Process Schedule	***Ore (kt)***	**550**	**1,101**	**1,099**	**1,356***	**1,118**	**1,101**	**1,099**	**1,100**	**585**	**9,109**
	Grade (g/t Au)	**2.71**	**4.02**	**2.34**	**1.8**	**2.14**	**1.7**	**1.68**	**1.56**	**1.64**	**2.17**
	Gold (t)	**1.49**	**4.43**	**2.57**	**2.45**	**2.39**	***1.88***	***1.85***	***1.71***	***0.96***	***19.72***
	Waste (kt)	**10,116**	**16,639**	**17,067**	**10,153**	**12,625**	**8,255**	**8,071**	**8,073**	**3,181**	**94,181**

*Note: Ore mined from Pravoberezhny in Years 2018-2020 will be stockpiled and processed in Year 2021

The in-situ tonnage of material in designed pits is shown in Table 14.5 below.

Table 14.5: Nasedkino Pit Design In-situ Mineable Content

Deposit	Ore (Mt)	Grade (Au g/t)	Gold (t)
Gora 5	1.75	2.27	3.98
Pridolinny	4.48	1.77	7.91
Pravoberezhny	0.95	2.15	2.05
Zhelanny	1.64	3.86	6.34
TOTAL	**8.82**	**2.30**	**20.28**
Note: Contains only Indicated category Mineral Resources.			

- The Nasedkino ore reserve statement was based upon the Gold price forecast at the time of carrying out the Nasedkino mine design optimisation of US$1100. If the mine design was carried out using current spot prices of over US$1200, and revised gold forecasting, there would be a significant uplift in the total gold at Nasedkino.

14.4 Reserve Statement

Table 14.6, below, presents the Ore Reserve Estimate for the Nasedkino Deposits:

Table 14.6: Nasedkino Ore Reserves Estimate (WAI, February 10, 2017) to JORC 2012

Reserves Classification	Ore	Grade	Metal
	tonnes	g/t Au	Tonnes Au
Gora 5			
Proved	-	-	-
Probable	1,760,565	2.22	3.91
Total	1,760,565	2.22	3.91
Pridolinny			
Proved	-	-	-
Probable	4,567,010	1.67	7.63
Total	4,567,010	1.67	7.63
Pravoberezhny			
Proved	-	-	-
Probable	1,021,413	1.96	2.00
Total	1,021,413	1.96	2.00
Zhelanny			
Proved	-	-	-
Probable	1,760,242	3.51	6.18
Total	1,760,242	3.51	6.18
TOTAL			
Proved	**-**	**-**	**-**
Probable	***9,109,230***	***2.17***	***19.72***
Total	**9,109,230**	**2.17**	**19.72**

Notes:

1. *The Ore Reserve Statement has been compiled under the supervision of Mr. Mark Mounde C.Eng. who a Technical Director at Wardell Armstrong International Limited and is a Member of the Institute of Materials, Minerals and Mining. Mr. Mounde has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration to qualify as a Competent Person as defined in the JORC Code.*
2. *Tonnages are metric tonnes*
3. *All figures are rounded which may result in small tabulation errors.*
4. *A US$1,100/oz Au price has been used in the estimation.*

15 MINING METHODS

15.1 Overview

All mining inputs which has been used for design of the Nasedkino open pits are presented below.

A conventional open pit shovel-truck method is planned for mining of all Nasedkino pits. The accepted milling rate was 1.1Mtpa. At the time of reporting it is expected that a first few years of material hauling will be undertaken by a sub-contractor. All drilling and blasting operations will also be carried out by a specialist contractor.

15.2 Mining Factors

15.2.1 Losses and Dilution

Exploitation losses during mining will consist of losses of ore at the contacts with adjacent rocks, losses of ore from its transportation, and losses appearing during blasting.

Dilution will occur as a result of incorporation of adjacent rocks during blasting, as well as mixing of waste and non-conditional ore during excavation.

The accepted parameters for dilution and recovery by deposit are illustrated in Table 15.1 below.

Table 15.1: Dilution and Recovery Parameters

	Gora 5	Pridolinny	Zhelanny	Pravoberezhny
Losses, %	1.81	3.44	2.5	2.5
Dilution, %	2.4	5.62	10	10

15.2.2 Overburden Removal

In accordance with the TSKR-TPI Rosnedra Protocol, dated 24/05/2016 and the current mining licence ЧИТ 01663 БЭ, the first overburden removal, together with the first ore production should commence in 2017, where recovered ore will be stored at the ore stockpile.

As well as overburden removal for 2017, the licence allowed for commencement of construction of the water flow re-redirection channel.

15.2.3 Drill and Blast

Drill and blast operations will be conducted by a specialist sub-contractor.

Blasting operation will be carried out by three main methods:

- Blasting of overburden and ore using blastholes;

- Pre-split blasting of working benches; and
- Crushing of oversized rock by cumulative charges.

All blasting will be performed by vertically orientated blastholes using Grammonite 79/21 explosive. Initiation of charges will be provided by patronised explosives Ammonite No 6ZHV using Iskra type initiation system with low-energy waveway.

Atlas Copco FlexiROC D65 down-hole hammer drill rigs will be used at the Nasedkino open pits. These rigs are also equipped with pneumatic impact tools or similar.

It has been determined that in order to blast overburden, four Atlas Copco FlexiROC D65 drill rigs will be required.

Table 15.2 below presents drill and blast specifications.

Table 15.2: Drill and Blast Specifications, including Ore and Waste

Parameter	Ore	Waste
Drill Specifications		
Hole Diameter (mm)	108	170
Final Bench Height (m)	20	20
Flitch Height (m)	5	10
Hole Angle (from Hz) (°)	90	90
Hole Length (m)	6.5	13
Sub-Drill (m)	1.5	3
Stemming (m)	2	4
Charge Length (m)	4.5	9
Row Spacing (m)	3.3	5.6
Explosives		
Type	Grammonite 79/21	Grammonite 79/21
Density (t/m^3)	0.9	0.9
Explosive/hole (kg)	36.7	183.4
Detonators per hole (pcs)	2	2
Primers per hole (pcs)	1 or 2	1 or 2
Connectors per hole (pcs)	1	1

Note: numbers are based on SRK NI-43-101 2012 geological findings
Note: The actual consumption of the explosive (kg/m^3) must be confirmed experimentally, with a minimum of three test blasts.

The loading and charging of boreholes is undertaken using charging machine MSZU-14-NPB and loading machine 3S-1B.

Pre-split blastholes will be drilled at angles matching the final slope angles. Ammonite 6ZHV (d=32mm), will be used, with initiation is carried out using detonating cord.

It is planned to conduct pre-split drilling for the entire height of the bench (20m) to the point until it approaches a final contour but no further than 30m.

The maximum allowed size of blasted material (ore and waste) is 800mm.

The hazardous areas for blasting operations are:

- Shock wave – 550m;
- Seismic impact – 300m; and
- Fly rock – 450m at ore extraction operations and 250m at overburden removal.

15.3 Geotechnical Review

The geotechnical review has been conducted on previous investigations and data supplied by the client. The following sources of data have been used in the review:

- NI 43-101 Report by SRK Consulting (Oct 2012);
- TEO (2016); and
- WAI site visit August 2016.

The bulk of data for review is available from previous investigations conducted as part of the SRK NI 43 101 study in 2012. The previous investigations conducted core logging, visually reviewed rock mass, and conducted laboratory analysis of rock samples. Geotechnical logging was conducted on Gora 5, Zhelanny, and Pridolinny. Analytical and numerical analysis was completed on 2D sections of Gora 5, Zhelanny, and Pridolinny using Rocscience Phase2 software. SRK highlight the rock mass consists of numerous faults which intersect open pit boundaries.

As part of this review, WAI has not conducted any geotechnical logging or independent testing of rock samples. The following table (Table 15.3) shows data and analysis completed by deposit:

Table 15.3: Geotechnical Data Available by Deposit

Deposit	Geotechnical Logging	Laboratory Test work	Analysis
Gora 5	3	3	3
Zhelanny	3	3	3
Pridolinny	3	3	3
Pravoberezhny	5	5*	5

Notes: *The most recent investigation (TEO, 2016) has drilled eight boreholes through projected open pits, but test work, certificates, logging has not been available to review.

During the investigations and report by SRK in 2012, Pravoberezhny deposit was not geotechnically investigated as at the time of study, there was not sufficient drilling to allow a Mineral Resource estimate to be completed, therefore it was excluded from the analysis.

Although geotechnical investigations have not been completed specifically for Pravoberezhny by SRK, the geology is deemed similar in petrography, mineralogy, tectonic structure, and morphostrucutral peculiarities. The 2016 TEO is detailed to have drilled through projected open pits, however testwork has not been available to review. Therefore, the previous study results by lithology have been applied and used to consider Pravoberezhny. WAI recommends geotechnical testing be conducted to confirm and give confidence for the combined projects and Pravoberezhny.

SRK and the TEO conducted laboratory test work of rock samples at St. Petersburg State Mining University, testing certificates and details of testing standards were not available for review. The following table (Table 15.4) details rock strength properties and core logging parameters collected for the investigation:

Table 15.4: Rock Strength and Geotechnical Core Logging Parameters

Rock Type	Area	Density	Poisson's Ratio	UCS (MPa)	GSI	m_i	D
Gniess	Pridolinny	2.75	0.18	76	47	28	0.85
	Zhelanny	2.70	0.18	119	59	28	0.85
	Gora 5	2.70	0.16	93	55	28	0.85
Skarn	Pridolinny	2.82	0.13	58	46	19	0.85
	Zhelanny	2.80	0.13	78	50	19	0.85
	Gora 5	2.80	0.13	78	60	19	0.85
Diorite Dykes	Pridolinny	2.70	0.24	65	46	19	0.85
	Zhelanny	2.70	0.24	65	55	25	0.85
	Gora 5	2.70	0.24	65	55	25	0.85
Breccia	Gora 5	3.20	0.12	140	60	19	0.85

Rock Strength and Geotechnical Core Logging Parameters - Summary

Rock Type	Density	Poisson's Ratio	UCS (MPa)	GSI	m_i	D
Gniess	2.72	0.17	96	54	28	0.85
Skarn	2.81	0.13	71	52	19	0.85
Diorite Dykes	2.71	0.24	65	52	23	0.85
Breccia	3.20	0.12	140	60	19	0.85

Note: Summary of test results conducted by St. Petersburg State Mining University

In reviewing the testing data, results of UCS strength are within typical strength ranges, albeit ranging at the lower range of expected strength. Positions and details of tested core samples were not available for review, however reduced strength would be expected for weathered or oxidised material. The test results from the TEO, 2016 indicated an increase in average rock strength, but similar densities were recorded. The 2016 TEO testing certificates, testing standard, and sample locations have not been available to review.

Indicated GSI ranges between 52-60 indicates a blocky rock mass with fair (smooth/planar/infilled) joint surfaces. WAI has reviewed photographs of core from the 2016 site visit and concurs with the general ranges of GSI. The drill core also displays frequent rubble zones which are typical exposures of brittle faults, see below Photo 15.1.

Photo 15.1: Drillhole NS1030 (16.00-20.00)

The intact rock parameter (mi) meets typical parameters for the lithology types, except for the Diorite Dykes, which has a lower mi then quoted examples. However, being emplaced as dykes is reasonable to assume a lower intact rock parameter. The disturbance factor D (0.85), is equivalent to moderate disturbance to the rock mass, either from blast damage or stress relief from overburden removal. Blasting damage should only be applied to the actual zone of damaged rock, near benches, rather than applying to the whole rock mass, (Hoek, 2012).

In the previous geotechnical assessment, the report notes that there is a minor risk of kinematic wedge failures attributed to faults. In addition to the faults, primary joint sets will influence the stability of the bench faces. Results of geotechnical logging, investigations, or stereonets were not available for review by WAI, in any case bench faces should be designed to minimise undercutting or 'daylighting' of primary joint sets causing block fall out.

Faults have been highlighted as intersecting the open pit designs, although wireframe plans, descriptions, and orientations have not been available for review. Reviewing photographs of drill core, frequent rubble zones are observed indicating brittle faults are intersected. The rubble zones are typically narrow (50-300mm) in observed drill core, but are highly fractured. A representative plan of Pridolinny from the SRK 2012 review is shown below, Figure 15.1.

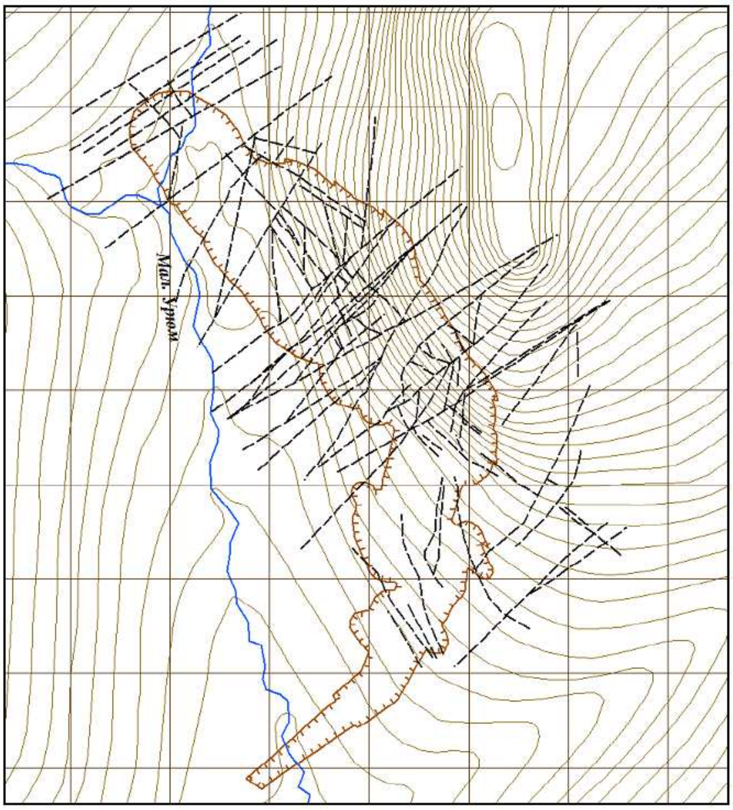


Figure 15.1: Plan of Pridolinny Area with Pit Outline and Known Tectonic Faults

The following bench design criteria, specified by the client has been used for design, (Table 15.5), Mining parameters from the current pit design are shown below, Table 15.6.

Table 15.5: **Bench Design Parameters**			
Parameter	**Value**	**Parameter**	**Value**
Face Angle of Non-working slope (°)	53	Bench Height (m)	5
Safety Berm Width (m)	8	Bench Height of non-working bench (m)	20
Haul road Width -upper (m)	21	Face angle of non-working bench (°)	70
Railroad width - lower (m)	15	Haul road Gradient (%)	8

Table 15.6: **Mining Parameters by Pit**			
Deposit	**Pit Bottom (m)**	**Max. Pit Depth from Surface (m)**	**Overall Slope Angle (°)**
Gora 5	830	140	44
Zhelanny	920	140	46
Pridolinny	730	145	45
Pravoberezhny	830	140	46
Notes: Inter-ramp angle (IRA) measured at 52.2°			

WAI conducted a cross-check of slope design based on the geotechnical parameter from the previous study and current mine designs. Numerical modelling using Rocscience Phase2 was used on a section through the steepest section of Pridolinny (Figure 15.2), a groundwater level has been added in the analysis. The results show that the highwalls have a factor of safety (FOS) greater than 1.2, however owing to the inferred blast damage near benches, some minor areas of reduced FOS are indicated, typically expected in open pits.

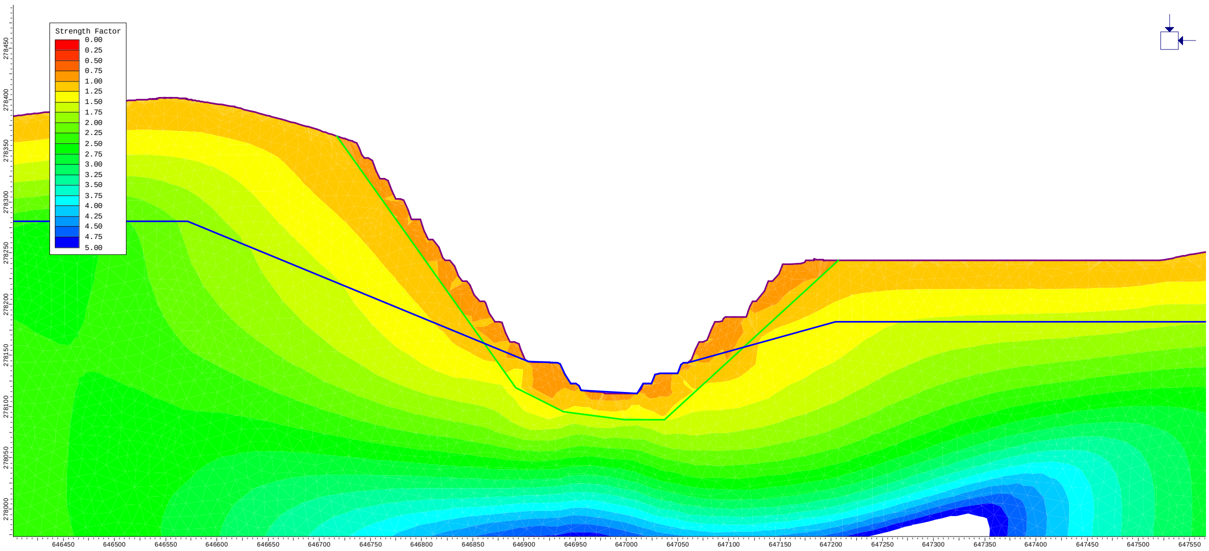


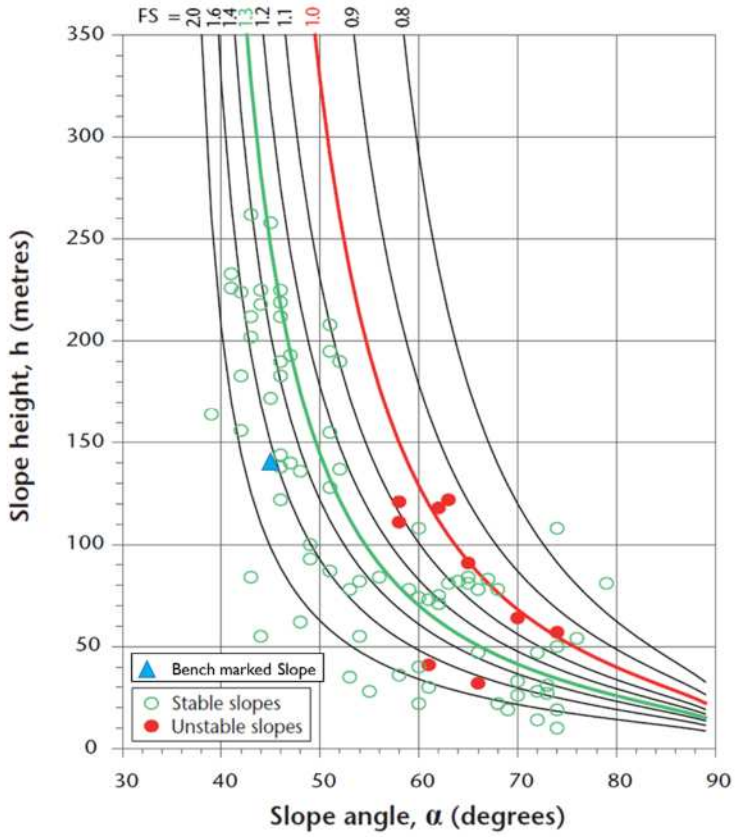
Figure 15.2: Plan of Pridolinny area with pit outline and known tectonic faults

Bench Marking

Pit slope stability is dependent on local, site specific features (rock mass structure, alteration, rock strength, groundwater, discontinuity characteristics/orientation, pit geometry, blasting, stress conditions, climatic conditions, and length of operation), therefore direct comparison with other operations is difficult. However, it is still useful to review the outcome of other open pit operations, Read and Stacey (2009), provide bench marked examples of other pit slopes for comparison, Figure 15.3.

The position of the steepest Pridolinny slope against other bench marked operations suggest that the design is within common limits of overall slope angles and slope height, however direct comparison owing to differences in geotechnical parameters must be considered when reviewing the bench marking.

Most large open pits have a significant likelihood that some areas of the pit slope may require flattening during operations in response to slope movement. Therefore, mine plans should be flexible so that extra step outs/buttresses can be maintained in critical areas until the end of the mine life at which point lower factors of safety can be tolerated. The comparison also highlights the importance of developing and maintaining good controlled blasting practices, effective groundwater management, and geotechnical data collection.



Notes:

1. DATA FROM READ AND STACEY (2009).

Figure 15.3: Slope Height versus Slope Angle – Bench Marking

15.4 Hydrological and Hydrogeological Review

15.4.1 Hydrological Conditions

The Nasedkino deposit is dominated by the River Uryum and its tributaries. Surface water measurements have been undertaken in 2006 – 2008 and 2011 – 2012. Additionally, results are presented in the 2016 TEO for flow characterisations for the River Uryum and its tributaries however it is not clear from the TEO which period of monitoring this relates to. Flow is reported to vary between 0.22 – 0.78m/s with water depths of between 0.15 – 0.24m. The majority of the flow occurs in the summer when flooding occurs, presumably following snow melt. Low or no flow is reported to occur in winter due to the freezing conditions. One of the tributaries of the River Uryum is shown as being fed by a spring. This would indicate that the surface water system receives a contribution from rainfall as well as groundwater. This interpretation is further strengthened by regional groundwater contours showing discharge of groundwater into these surface water bodies.

The River Uryum is marked as being located at the northwestern end of Pridolinny with the pit intersecting the alluvial deposits associated with this river. Stream diversions have been considered and initial engineering designs have been produced.

During mine development it is proposed to site a water supply reservoir and a Tailings Storage Facility in a tributary of the River Uryum to the south of the four pits. This is discussed further in section 17.

The main hydrogeological investigations for the deposit were conducted in 2011 by FGBR Chitinsky CGMS R.

15.4.2 Hydrogeological Conditions

The hydrogeological conditions of the Nasedkino deposit are summarised below:

- Holocene aged deposits associated with the river valleys: these deposits comprise sands, loams, gravels and boulder deposits. The deposits are reported to be up to 2.0m in thickness with groundwater present from approximately 0.2m below ground level in an unconfined aquifer. Studies undertaken at Pridolinny indicates that the groundwater within these deposits has limited connectivity with the underlying fractured bedrock. It is reasonable to assume that this groundwater is in continuity with the surface waters in the rivers; and
- Fracture flow dominated bedrock: these deposits are either dominated by schists and gneisses associated with metamorphic activity or the igneous granites and porphyries themselves. Groundwater within these deposits is mainly present within fractures. Recharge to these fractures is reportedly directly from rainfall. Groundwater is reported to be either unconfined or partially confined with low groundwater pressures measured.

Permafrost has not been reported in studies undertaken in the Gora 5 or Zhelanny pits. No studies are available regarding Pravoberezhny pit. Permafrost is reported as present at the Pridolinny pit with shallow groundwater reported to be present above the permafrost layer. An audit of the deposit in 2011 by SRK indicated that permafrost is present between 30 – 90m depth in the region with the layer extending to 40m in the vicinity of the proposed pits.

The main hydrogeological investigations for the deposit were conducted in 2012 by Zapadnoe.

15.4.3 Hydrogeology of Pridolinny

During 2011 – 2012, three hydrogeological boreholes (1G, 2G and 3G) were advanced at Pridolinny. Additionally, hydrogeological information was collected from 18 geological boreholes (134, 173, 175, 178, 196, 200, 201, 207, 209, 211, 215, 222, 228, 239, 246, 250, 256 and 764) in the proposed pit footprint and close to its edges. Groundwater level information was also recorded in two geological boreholes, 189 and 204, installed in the alluvial deposits associated with the River Uryum.

Hydrogeological borehole 1G was advanced to 100m below ground level (approximately 783m elevation) and above the proposed base of the pit. Hydrogeological borehole 2G was advanced to 200m below ground level (approximately 688m elevation) and through the proposed base of the open pit. A number of the geological boreholes advanced were also deep enough to intersect the base of the proposed pit.

The water level data collected indicates a continuous groundwater table is present within the fractured schist bedrock at between 1m to 35m below ground level (elevations between 867m to 853m). The regional groundwater contours indicate that groundwater flow is across the pit towards the southwest and southeast towards the River Uryum and its tributary.

Groundwater levels recorded in the alluvial deposits varied between 3.5m – 4.33m below ground level (elevations of 869m – 863m). These are spot values are likely to be dependent upon the flow conditions in the river at the given time however the elevations indicate that the groundwater in the alluviums may be in continuity with the groundwater in the fractured bedrock. No information is available regarding confining conditions or groundwater pressures.

Hydraulic testing was undertaken in the three hydrogeological boreholes and three (175, 222, 239) geological boreholes. Shallow hydraulic conductivity values (less than 70m) ranged between 2.4m/d – 9.3m/d. Hydraulic conductivities recorded including deeper horizons were between 0.05m/d to 4.2m/d. This would indicate that the shallower fracture horizon present at Pridolinny is better developed or connected with the surrounding fracture network. Limited drawdown was recorded at shallow depths during the hydraulic testing indicating that yields were more sustainable. At depth, some significant flow horizons were encountered which provided relatively high hydraulic gradients but with associated short term yields.

The Pridolinny pit may intercept the alluvial deposits, or approach close to their lateral extent, however no hydraulic testing has been undertaken in these deposits to provide information on their hydrogeological characteristics or potential connection with the underlying fractured bedrock. Reportedly tests undertaken in 3G close to the alluvial deposits indicated no connection between the fractured bedrock and the alluvial deposits, but no quantitative data to support this has been seen.

Groundwater quality data collected at the Pridolinny site indicates that the groundwater is sodium-bicarbonate in type.

15.4.4 Hydrogeology of Gora 5

During 2011-2012, two hydrogeological boreholes were advanced at the proposed Gora 5 pit location. Additionally, seven geological boreholes (374, 376, 389, 390, 391, 392 and 394) advanced at the Gora 5 pit had hydrogeological information recorded covering the period 2011 – 2012. The Gora pit will be advanced to approximately 140m below ground level to an approximate elevation of 840m. Both the hydrogeological boreholes were advanced to approximately 100m (5G to an elevation of 872m and 4G to an elevation of approximately 925m); neither of these boreholes will intercept the base of the pit. The geological boreholes were deep enough to intercept the proposed base of the pit and only water level data was collected from these holes.

The water level data collected indicates that groundwater is present at between 6m to 19m below ground level. A continuous water table is marked on plans showing a groundwater flow direction towards the southeast and the River Uryum with a gradient of approximately 0.19. No information is available regarding confining conditions or groundwater pressures.

Hydraulic testing undertaken in 4G and 5G within the fractured schists indicates hydraulic conductivities vary between 0.0004m/d to 0.2m/d. The upper range was recorded between 0 – 50m depth with values below 50m one or two orders of magnitude lower. Drawdowns of up to 42m were recorded with low abstraction rates (approximately $3m^3/hr$). This would indicate that these fractured rocks are low permeability and yields are unlikely to be sustainable long term although if significant shallow fracturing, or connection to the wider fracturing system to the northwest, is encountered then shorter duration, higher inflows may occur.

No water quality information is available for Gora 5.

15.4.5 Hydrogeological Characteristics of Zhelanny

During 2011 – 2012, two hydrogeological boreholes (6G and 7G) were advanced at Zhelanny. Additionally, hydrogeological information was collected from geological boreholes (366, 379, 380 and 383). Both hydrogeological boreholes were advanced to 100m below ground level (between 911m and 896m elevation). Neither hydrogeological borehole was advanced deep enough to intersect the proposed pit base. The geological borehole, 380, within the proposed pit outline is advanced to below the proposed base of the pit.

The water level data collected indicates that the groundwater table in the fractured bedrock is present at approximately elevations of 975 – 1,000m. However, it is noted that some geological boreholes did not intercept groundwater at shallow depths which may indicate that the groundwater table is partially confined to the northwest moving to unconfined conditions in the southeast. No information is available regarding groundwater pressures. The regional groundwater flow direction is towards the northeast towards a tributary of the River Uryum. The pit design places the northern extent of the pit close to the alluvial aquifer associated with this tributary, but no information is available regarding the connectivity of this aquifer with the underlying fractured bedrock.

Hydraulic testing in the fractured bedrock indicated that hydraulic conductivity was very low (0.0001 – 0.003m/d) with low yields. No groundwater quality samples were collected from the deposit.

15.4.6 Hydrogeological Characteristics of Pravoberezhny

No detailed hydrogeological information is available regarding this location. The groundwater is reported to be present in shallow aquifer horizons and at depth within fractures. The fractured groundwater is reported to be partially confined and under low pressure.

15.5 Seismicity

Data from the United Schmidt Institute of Physics of the Earth, Russian Academy of Sciences, Moscow and the Working Group of the GSHAP Region 7 (GSHAP Moscow Regional Centre 7) was reviewed for seismicity data. Figure 15.4 below shows 10% probability of exceedance in 50 years for Northern Eurasia, it is based on average soil conditions and should be confirmed based on specific ground type of the Nasedkino project.

In accordance with the map the expected peak ground acceleration is in range of 0.4 - 0.8m/s².

WAI is of the opinion that the seismic risk to the Project Nasedkino is low/moderate, however it is recommended to obtain more accurate data of the actual mine site prior to design of infrastructure, buildings and estimation of the final pit slope angles.

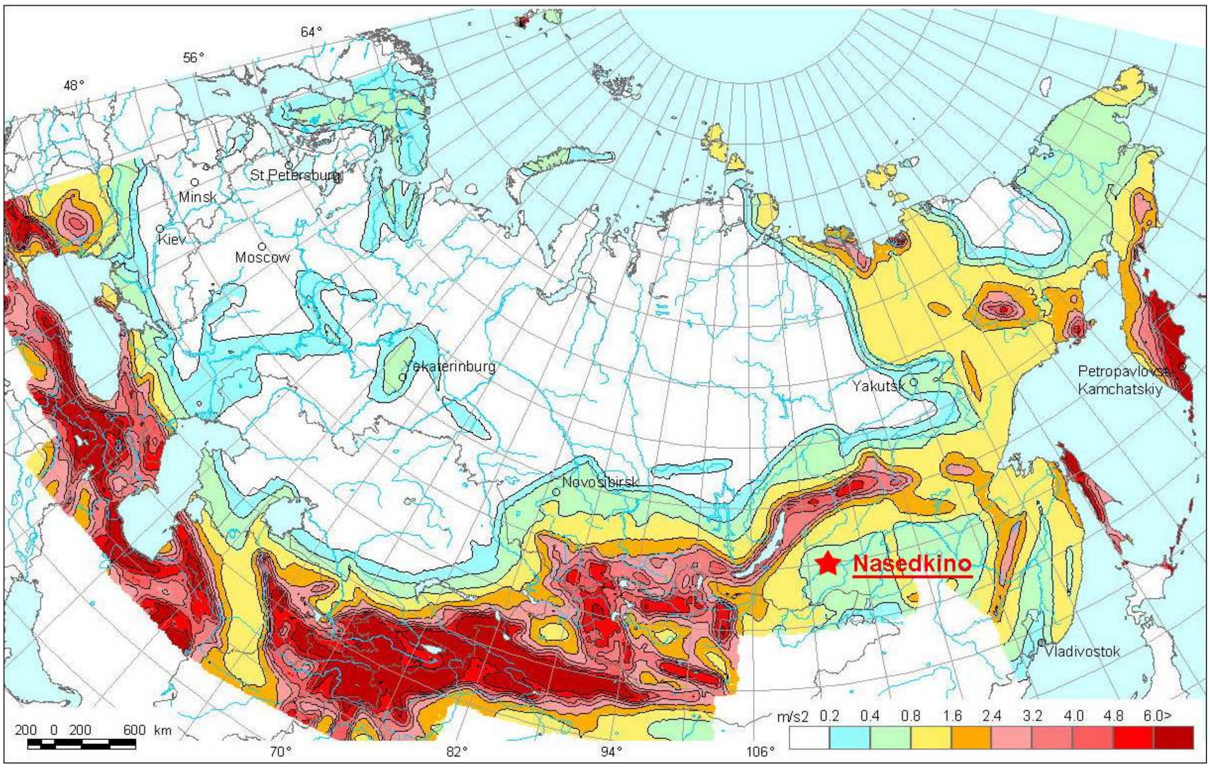


Figure 15.4: Seismic Hazard of Northern Eurasia

15.6 In-pit Haul Roads

The design parameters of the in-pit haul roads taken from the VNTP 35-86 Russian standard, are based on the volume of vehicle movements, the size of vehicles used and the operating conditions.

Two types of haul roads have been reviewed for the transportation: for the waste haulage and ore haulage. Due to the lower intensity of traffic at the deeper horizons the haulage of rockmass from lower levels is planned to proceed on single lane roads.

The haul road parameters for all categories are as following:

- Haul road incline – 8%;
- Minimum turning radius in plan - 20m;
- Roadway incline – 3.5%;
- Roadside incline - 6%;
- Haul road berm width (double lane) – 21m;
- Haul road berm width (single lane) – 15m;
- Ditch width – 1m; and
- Trench width – 1.2m.

The construction of in-pit haul road is as following in Figure 15.5 below:

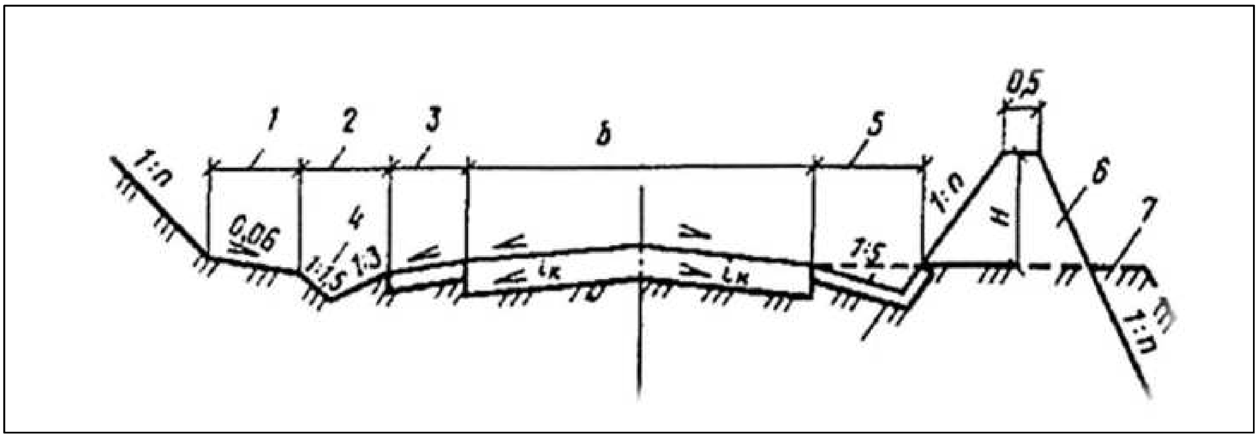


Figure 15.5: In-pit Haul Road Construction

1 – Out-of-cuvete bench; 2 – duct; 3 – road shoulder; 4 – trench; 5 – strengthen duct; 6 – earth bank; 7 – wedge of failure; ix – haul road incline; H – earth bank height; b – roadway width.

The roadway of the in-pit haul roads is protected from mined out areas by a safety berm. The vertical axis, laid through the top of the safety berm must be located outside of the zone of possible rock failure. Temporary roads are proposed at the working bench and waste dumps. It was suggested to use a DZ 98 grader in order to maintain haul roads. Scatter of sand during winter months and irrigation during summer months will be conducted by a special vehicle KO 829B, which is based on a KAMAZ 65115 chassis.

15.7 Mining Equipment Fleet Summary

The list of main and auxiliary equipment for the Project Nasedkino has been presented by Daltsvetmet and is shown in Table 15.7.

Table 15.7: Nasedkino Mining Equipment

	Units
Main technological equipment	
Hydraulic excavator Komatsu PC1250SP-7 (front shovel for waste, backhoe for ore). Bucket capacity 5.9m³	*4*
Belaz Dump 7555 B (waste)	*13*
Belaz Dump 75473 (ore)	*2*
Bulldozer Komatsu D65EX	*5*
Drill rig Atlas Copco FlexiROC D65	*4*
Hydro – Hammer Delta F7, based of Komatsu excavator	*1*
Auxiliary equipment	
Grader DZ-98V.00010	*2*
Combined road vehicle KO-829B1	*1*
Truck crane KC 45717	*1*
Mobile repair station (Parm) Ural 4320-41	*1*
Towing vehicle Belaz-7447	*1*
Bus Ural-32551-3171-59	*2*
Personal transporter Ural-4320	*1*
Minibus UAZ 2206	*1*
Fuelling tanker ATZ-10, based on Ural 4320-1951-60	*2*

The required fleet of open pit trucks is determined based on maximum possible use of the loading and trucking equipment, which is used for transportation of ore and accounting for the distance of transportation. Based on that, the Belaz dump truck 7555 B of 55t capacity has been accepted for transportation of waste, and Belaz dump truck 75473 of 45t capacity for ore transportation.

A Belaz-7447 will be used for vehicle recovery.

All preparation works will be conducted using drill and blast operations. Atlas Copco FlexiROC D65 drill rigs will be used for drilling. Such drill rigs work on diesel, and meet all required specifications.

The loading of blastholes is done by mechanised machines MSZU-14-NPB and charging machine ZS-1B.

At the extraction and loading operation, it is proposed to use hydraulic backhoe excavators Komatsu PC1250SP-7 with a bucket capacity of 5.9m³.

For cleaning and preparation of blocks for drilling, development of in-pit roads and working areas, and transportation of rock mass at the waste dumps, Komatsu D65EX bulldozers will be used.

During loading and off-loading operations, technical maintenance and repair works, a mobile repair station URAL 4320-41 will be used. The machine is based on a crane C-45717 of 25t capacity.

It is planned to use a special vehicle KO829B, based on a Kamaz 65115 chassis for irrigation of working faces and roads in the summer months and placement of sand on roads in the winter months.

Road maintenance will be conducted by autograder DZ 98.

At the time of reporting, there are a number of vehicles present on site already, Table 15.8.

Table 15.8: Existing Daltsvetmet Equipment List

	Equipment type	Units
1	UAZ wagon	2
2	UAZ-220695 passenger carrier	2
3	UAZ materials and passengers carrier	1
4	UAZ 39049	1
5	Land Cruiser Prado No245	1
6	Kamaz Bort K 582 RU 75	1
7	MAZ TZ No E 555 BE	1
8	Kamaz-Nefaz 4208-24	2
9	Crane KamazKS-45712	1
10	Kamaz KNU XDF650610G0001586	1
11	Kamaz ATS water XIF 66065AF000545	1
12	Kamaz tractor 44108	1
13	Semi-trailer for car SZAP 9327	1
14	Semi-trailer forestry materials 9047L	1
15	Kamaz tractor 780646	1
16	CAT 428 F2 loader	1
17	Kamaz 58146T concrete-mix	1
18	Kamaz 65116-A4	1
19	Semi-trailer Nefaz-9334 for Komatsu (65116-A4)	1
20	Mitsubihi L-200	1
21	Kamaz 65117-L4	1
22	Loader DIECI ICARUS 40.14	1
23	Skidding machine MTCH-4	1
24	Komatsu excavator PC-750	1
25	Komatsu excavator PC-250	1
26	Bulldozer CAT-D9R	1
27	Bulldozer Komatsu D155 A-5 659/66684	2
28	Swamp and snow-going machine	1

15.8 Repair and Storage Facility

All daily and planned repairs of mining equipment will be carried out by the open pit personnel and will only include mechanical repairs. Those repairs will be carried out in accordance with scheduled maintenance where broken parts are replaced by new or previously refurbished parts. This will require a fleet reserve and suitable spare part stocks.

The repair and storage facility requires a garage with an area for repairs as well as a heated parking area for the auxiliary equipment and workshops.

15.9 Mine Manpower Requirement

Shift work is allowed at the Nasedkino project, which provides for the replacement of all related staff working in the open pit every 30 - 45 days.

The terms of employment when servicing mining equipment are a double-shift in a continuous working week with a rotating schedule of alternation of day and night shifts. The terms for workers of drill and blast and geological and surveying services are a single-shift (day shift). The duration of one working shift is 12 hours.

The manpower of the open pit has been estimated taking into account the designed mining equipment and accepted terms of work.

In accordance with Daltsvetmet the list of manpower to be used at the Nasedkino Project is presented in Table 15.9 below.

Table 15.9: Total Nasedkino Project Manpower

Department and Job Description	Shift 1	Shift 2	Daily	List
Mining				
Production Area				
Engineering Staff				
Pit Manager	1		1	1
Deputy Pit Manager	1		1	1
Pit Supervisor	1	1	2	2
D&B Engineer	1	1	2	2
Power Engineer	1		1	1
Pit Geologist	2		2	2
Pit Surveyor	2		2	2
Pit Mechanic	1		1	1
Product and Haulage Staff				
Excavator Drive	4	4	8	22
Truck Driver	15	15	30	82
Bulldozer Driver	4	4	8	22
Drill Rig Operator	4	4	8	22
Hydro Hummer Operator	1	1	2	6
Loading Machine Operator	2	2	4	11
Charging Machine Operator	1	1	2	6
Blaster	4	4	8	22
Auxiliary Operations				
Autograder DZ 98 Driver	2	2	4	11
Crane KS 45717 Driver	1		1	3
Irrigation Machine Driver	1		1	3
Fuel Truck Driver	2	2	4	11
Welder	2	1	3	8
Miner	6	2	8	22
Mobile Repair Station Driver	1	1	2	6
Bus Driver	2	2	4	11
Material Truck Driver	1		1	3
Electrical Technician for Equipment Repair	2	1	3	8
Repair Technician	3	1	4	11
Lighting Equipment Operator		1	1	3
Pump Station Operator	1	1	2	6
Pit Total	**69**	**51**	**120**	**311**

Processing				
Specialists				
Plant Manager	1		1	1
Chief Engineer	1		1	1
Chied Mechanic	1		1	1
Chief Power Engineer	1		1	1
Head of the Technical Control Department	1		1	1
Crushing Unit Supervisor	3		3	6
Concentrator Supervisor	3		3	4
Hydromet Unit Supervisor	3		3	4
Detox Unit Supervisor	2		2	3
Smelting Unit Supervisor	3		3	4
TMF Supervisor	2		2	3
Mechanic	2		2	3
Power Engineer	2		2	2
Technical Control Department	2		2	3
Dispatcher	6		6	8
ROM Ore Stockpile, Crushed Ore Stockpile				
Excavator Operator	4		4	6
Loader (Bulldozer) Operator	4		4	6
Dump Truck Driver	9		9	14
Crushing Unit				
Crushing Unit Operator	3		3	5
Conveyor Operator	3		3	5
Crusher Operator	3		3	5
Main Building – Concentrator				
Operator	3		3	4
Mills Operator	4		4	6
Concentrators Operator	3		3	4
Pumps Operator	4		4	6
Main Building - Hydrometallurgy				
Sorption Leaching Plant Operator	2		2	3
Desorption Plant Operator	2		2	3
Doser	2		2	3
Main Building - Detoxification				
Detox Plant Operator	2		2	3
Sodium Cyanide Preparation Room				
Reagents Preparator	2		2	3
Smelting Unit. Electrowinning				
Electrowinner	3		3	4
Smelter	3		3	4
TMF				
TMF operator	3		3	4
TMF controller	2		2	3
On-Duty Repair and Maintenance Personnel				
Technician on Duty	2		2	3
Electrician on Duty	2		2	3
Repair Technician	3		3	5
Welder	2		2	3
Electric Repairman	3		3	4
Technical Control Department				
Commodity Quality Control	2		2	3

Senior Chemist	2		2	3
Technical Control Department Chemist	3		3	5
Samples Acceptor-Preparator	2		2	3
Total Processing Plant	**116**	**-**	**116**	**168**
General Personnel				
Engineers				
Machinery Repair Shop and Transportation Department	2	-	2	2
Water and Heating Supply	2	-	2	2
Electrotechnical Service	2	-	2	2
Storage Facilities	2	-	2	2
Mine Rescue Team	2	-	2	2
Security	2	-	2	2
Superintendent Service	2	-	2	2
Auxiliary Staff				
Machinery Repair Shop and Transportation Department	3	3	6	16
Water and Heating Supply	2	2	4	11
Electrotechnical Service	3	3	6	16
Storage Facilities	4	4	8	22
Mine Rescue Team	2	2	4	11
Security	4	4	8	22
Superintendent Service	4	4	8	22
Total General Personnel	**36**	**22**	**58**	**134**
Management				
Mine Managers				
General Director	1	-	1	1
Deputy General Director	1	-	1	1
Chief Engineer	1	-	1	1
Head of the Security Service	1	-	1	1
Chief Mechanic Service				
Chief Mechanic	1	-	1	1
Chief Power Engineer Service				
Chief Power Engineer	1	-	1	1
Chief Geologist Service				
Chief Geologist	1	-	1	1
Leading Geologist	2	-	2	2
Chief Surveyor Service				
Chief Surveyor	1	-	1	1
Map Maker	1	-	1	1
Industrial Safety Department				
Head of the Department	1	-	1	1
Engineer	1	-	1	1
Production and Technical Department				
Head of the Department	1	-	1	1
Leading Engineer	2	-	2	2
FP&A Department				
Deputy Director on Financials and Economics	1	-	1	1
Chief accountant	1	-	1	1
Economist	2	-	2	2
Accountant	5	-	5	5
Legal Department				

Head of the Legal Department	1	-	1	1
Lawyer	1	-	1	1
Human Resources Department				
Head of the Department	1	-	1	1
HR Specialist	2	-	2	2
Instruments and Controls and IT Department				
Head of The Department	1	-	1	1
Specialist	3	-	3	3
Procurement Department				
Head of the Department	1	-	1	1
Procurement Specialist	3	-	3	3
Dispatcher Service				
Senior Dispatcher	1	-	1	1
Dispatcher	1	1	2	6
Total Management Personnel	**40**	**1**	**41**	**45**
Total Personnel	**261**	**74**	**335**	**658**

15.10 Open Pit Pumping Requirement

The 2016 TEO provides details of water inflow estimation into open pits Gora 5 and Pridolinny only, 1180m^3/day and 6510m^3/day respectively. In TEO 2016 no data was provided for Zhelanny and Pravoberezhny pits.

These estimates are based on State regulations and recommendations. The calculation includes meltwater flow, flow during intense rainfall period and underground waters.

Pumps based on the anticipated flow rates have been identified as TSNS 60-132 and TSNS 180-170.

15.11 Waste Rock Dumps

Taking into consideration all mining conditions and parameters of design pits, the technical design allows for dozer managed waste dumping and transportation of overburden by trucks to the external dumps.

The total volume of overburden, which needs to be stored is 94,181Kt (in-situ), this includes off-balance resources. The coefficient of residual loosening is 1.1 (Standards, VNTP 35-86). The overburden rocks are predominantly rocky. Slope angles of dumps are 35-40°.

Four waste dumps have been designed for the Nasedkino Project:

- dump No1 to the west of the proposed open pit Zhelanny;
- dump No2 to the east of the open pits Zhelanny and Pravoberezhny and to the west of open pit Gora 5;
- dump No3 to the east of open pit Gora 5; and
- dump No4 is located east of the open pit Pridolinny.

The location of the proposed waste dumps is shown on Figure 17.1.

End dumping will be used, the material will then be levelled and spread using tracked dozers.

During operations if vertical displacement of the dump exceeds 0.2m/day, then operations are required to stop, barricaded, and reported to management.

Waste dump construction is established by an initial 30m embankment. Subsequent dumps are developed parallel to the initial dump.

The formation of waste dumps starts from the lower layer of the dump, construction of the road to the dump and development of pioneering embankment with width of no less than 25-30m. The further dump development is carried out by strips, in parallel to pioneering embankment.

15.11.1 Waste Dump Parameters

The location of the waste dumps have been checked for sub-surface mineralisation waste dump capacity as shown in Table 15.10 below.

Table 15.10: Total Estimated Waste Material

Name	Parameter
Zhelanny	
Waste rock volume, Kt	*20,482*
Swell factor	*1,1*
Required waste dump volume, Kt	*22,530*
Gora 5	
Waste rock volume, Kt	*18,627*
Swell factor	*1,1*
Required waste dump volume, Kt	*20,490*
Pravoberezhny	
Waste rock volume, Kt	*19,961*
Swell factor	*1,1*
Required waste dump volume, Kt	*21,957*
Pridolinny	
Waste rock volume, Kt	*35,110*
Swell factor	*1,1*
Required waste dump volume, Kt	*38,621*
Total	***103,598***

The drainage collection areas should be developed for collection of precipitations. Drainage trenches depth is 0.7-1.0m and 1.0-1.5m wide. Trenches will be filled with rocky material to ensure stability of the structure.

Diversion channel's are developed around the waste dumps to divert rain water/snow from operations.

15.11.2 Waste Dump Equipment

The overburden will be transported to the waste dump using Belaz 7555 B trucks, Komatsu D65EX dozers will be used to level and spread material. The required productivity calculation is shown in Table 15.11.

Table 15.11: Komatsu D65EX Productivity Calculation

Name	Value
Bulldozer blade height, m	*1.425*
Bulldozer blade width, m	*3.41*
Block Volume, m^3	*4.12*
Average cycle distance to the edge, m	*13.0*
Coefficient of material loss during transportation, m	*0.92*
Loosening coefficient	*2.9*
Coefficient of transportation in 1 cycle (in hard rock), m^3	*39*
Duration of one cycle, sec.	*26*
Productivity at extraction and transportation, m^3/hr	*270.7*
Filling up coefficient	*0.89*
Productivity at waste dump development, m^3/hr	*239.9*
Utilisation	*0.76*
Shift duration, hr	*12*
Shift productivity at waste dump development	*2638.5*
Working days per year, days/year	*340*
Downtime due to maintenance and repair work, working days/year	*47.6*
Working days per year, day/year	*292.4*
Amount of shifts per day, shift/day	*2*
Availability	*0.95*
Annual bulldozer production	*1465.9*

The total calculated required waste dump fleet is five when all open pits are in full production.

16 RECOVERY METHODS

16.1 Process Design

16.1.1 Introduction

A process design was developed for the treatment of the Nasedkino ores based on treating approximately 1Mtpa as part of the TEO. The production rate for the crushing department was based on 21h/day operation for 247 days per year with coarse ore storage for a capacity of two days ore supply for the main processing facility. The production rate of the processing facility is based on 336 days per year. Regular preventive and major maintenance (replacement of mill linings etc.) was planned to take 29 days a year.

The processing flowsheet involved:

- Primary crushing to 250mm,
- SAG-Ball mill grinding,
- Gravity processing with the gravity concentrate and tailings leached separately,
- Carbon adsorption and desorption',
- Smelting the cathode product.

Process plant production mass balances were produced based on the mining schedule and calculated for five gold cut-off grades.

16.1.2 Flowsheet Description

16.1.2.1 Crushing

ROM ore will be screened at 500mm and lumps over 500mm will be crushed with a rock breaker. The ore will be fed to a jaw crusher operating in open circuit and set at 160-190mm. The maximum size of pieces of crushed material is 260-300mm. Preliminary screening to 250mm was recommended prior to crushing to increase throughput.

Based on a production rate of 257t/h, a Chinese PEW1100 jaw crusher was recommended.

16.1.2.2 Grinding

SAG mill calculations were based on the Vasilyevskoye gold plant and ball mill sizings were based on the Bond ball mill Work Index calculations. The following equipment was selected:

- SAG mill MZS-75 × 28 (China) with a circulating load of 500%
- Two central discharge ball mills sized 3.5 x 5m

16.1.2.3 Grinding Circuit Classification

SAG mill discharge will pass to a double-spiral classifier 2KSN-30 (high-weir type), 3.0m in diameter and with a rotation speed of 2rpm.

The spiral overflow will be pumped to four hydrocyclones (with three on stand-by). Cyclone underflow gravitates to the ball mill. The overflow, at 85% passing 75 microns, passes to thickening.

16.1.2.4 Gravity Circuit

After pre-screening to remove oversize, a Falcon SB5200 will be used to recover free gold. The gravity concentrate, at 50% solids, will be directed to the cyanide leach area where it is partially thickened and then washed with fresh water and subjected to grinding. The ground concentrate is discharged into the leach reactor and the leach solution passes to the electrowinning unit. The accumulated cathode deposit is washed off, dewatered and smelted. The barren solution after electrowinning is pumped to cyanidation. The cyanide leach cake is washed with water and pumped to grinding.

16.1.2.5 Thickening

The cyclone overflow will pass to a 50m diameter thickener where water will be recovered and returned to the process.

16.1.2.6 Gravity Tailings Leach

The thickened product passes to preliminary cyanidation pachucas equipped with airlift pumps to mix and transport the pulp. The leach time will be 18 hours.

The pulp after preliminary cyanidation will pass to 14 sorption pachucas by gravity, where more gold is dissolved and adsorbed by activated carbon. Regenerated or fresh carbon is added to the system and loaded carbon is discharged from the first sorption Pachuca to an accumulation tank.

Slurry tailings pass to a carbon safety screen and then go to a detox stage using calcium hypochlorite and semi-dry storage. Any recovered carbon is returned to the sorption pachucas.

Loaded carbon is collected into storage tanks and washed and screened to remove the residual pulp. The undersize returns by gravity into the sorption leach tank and the oversize - loaded carbon – passes to elution and regeneration.

16.1.2.7 Elution, Regeneration and Electrowinning Area

After sorption leaching the loaded carbon is washed and subjected to an acid wash and then washed with fresh water to remove the residual acid. All acidic solutions are subjected to neutralisation.

The capacity of the carbon stripping vessel will be 2.0 tonnes per day. Loaded carbon is stripped after pre-treatment with 5.0% NaOH solution at a pressure of 350-400kPa and a temperature of 90-100°C. The resultant gold-rich solution is fed to electrowinning with the gold-bearing solutions produced by cyanide leaching the gravity concentrates. Gold is formed on the cathode of the electrowinning cell and is periodically removed, dried and smelted.

Barren carbon is pumped into a receiving hopper of a regeneration furnace. The hopper is equipped with a bar screen and screw feeder. The screen has a 0.6mm mesh and the undersize is removed from the process. The +0.6mm oversize material is fed by a screw feeder into the furnace for regeneration where organic contaminants are removed by heating the carbon to 550-650°C in an oxidising atmosphere.

The regenerated carbon is discharged from the furnace to a cooling tank and then pumped to a screen to remove any - 0.6mm material.

16.1.2.8 Pyrometallurgical Processing of Cathode Deposits

Cathode sludge resulting from the gold electrowinning is dried in the oven, and then calcined in a roasting furnace. Smelting is performed in an induction furnace at 1200-1250° C for 1.5-2.0 hours.

16.2 Process Consumables

The process consumables are shown in Table 16.1.

Table 16.1 : Nasedkino Plant Consumables

Item	Kg/t
Crusher lining	0.10
Mill lining	0.20
Mill balls	1.50
Flocculant	0.02
NaCN	0.97
NaOH	14.2
Lime	2.39
HCl	0.041

The consumable levels are typical for the treatment of a moderately hard, non-refractory gold ore.

17 PROJECT INFRASTRUCTURE

17.1 General Plan

The following facilities are required in order to mine the Nasedkino:

- Open Pits of deposits Gora 5, Pridolinny, Zhelanny and Pravoberezhny;
- Waste dumps;
- Mine water treatment facility;
- Production site;
- Processing plant;
- Storage for ore blending;
- Tailings dam of semi-dry type;
- Living camp;
- Fuel/lubricants storage facility with fuel station;
- Garage with workshop for heavy trucks maintenance;
- Process water storage facility with pumping station;
- Drinking water storage facility with pumping station;
- Shift camp;
- Treatment facilities for domestic effluents;
- Solid waste landfill;
- Ditches;
- Stream diversion channel;
- Power sub-station 110/6kW; and
- Roads.

The layout of the major facilities is shown in Figure 17.1 below:

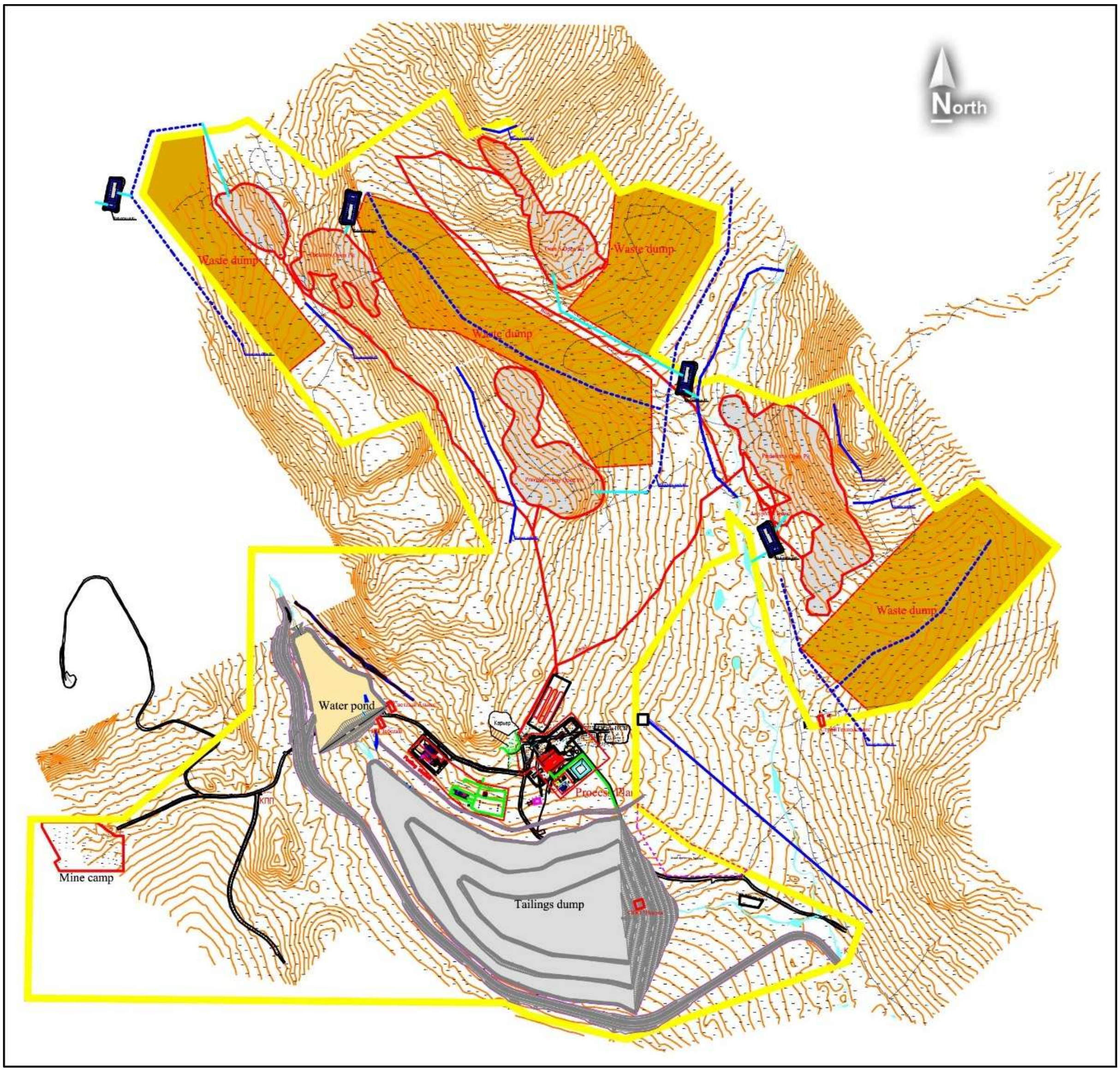


Figure 17.1: Nasedkino Infrastructure

Giving the fact that the area is located in a low/moderately seismic area, the production buildings are planned to be constructed with a steel frame clad with light insulation panels and sheeting to retain heat.

The sanitary protection zone of the Nasedkino project is: 1000m for open pits, 500m for the Process Plant and 500m for the solid waste polygon and treatment plants for the domestic wastewater.

The shift camp will be located outside of the sanitary protection zone, some 1.5km to the west of the production site and the Process Plant.

The shift camp facility includes:

- Hostel for technical engineering personnel for 100 people;
- Accommodation for management – 16 flats;
- Bathing facility for management;
- Hostel for other personnel for 110 people – 3 units;
- Canteen for 136 people;
- Food storage;
- Administration building;
- Sanitary-amenity block;
- Sport complex with laundry facility and feldsher's point;
- Closed parking with fire station;
- Sports ground.

Solid waste will be stored 500m southeast from the shift camp, the domestic waste water treatment plants will be positioned 500m east from the shift camp.

The drinking water storage facility together with a pumping station will be located at the river Ienda, 2km south from the shift camp.

17.2 Power Supply

To date Daltsvetmet has signed an agreement with Chitaenergo to conduct the technical connection of power at Nasedkino to the local grid.

In order to conduct the technical connection, the following will need to be constructed: HV line 110kV (double circuit), loading station 110kV and HV line 6kV. All construction located within the Nasedkino mining license boundary are the responsibility of Daltsvetmet, all other construction will be the liability of Chitaenergo. All works conducted up to the boundary of the Nasedkino site (by Chitaenergo) will be performed under an Investment program of IDG Company of Siberia PJSC.

Chitaenergo is currently advancing the project, and is carrying out the field studies to determine the path of the power line and are working on the project document.

The duration of the agreement is two years from signing of the contract until 05 Oct 2018. The date is in line with production commencement at Nasedkino.

The Diesel Power Plant (DPP) will provide 0.4kV voltage to the site. This option will be used on a temporary basis until the construction of the HV power lines is completed in 2018.

DPP has been specified as a container type "Ermak Energy", which is based on a Cummins diesel generator, which will be placed in a separate modular mobile type building.

The connection between the low-voltage consumers at the open pits and the DPP is conducted using KG-KHL-0.66 cables.

The metering of electricity is provided by Mercury-234 with GSM module. The installation will be located at the input point of the 0.4kV DPP. The monitoring of diesel consumption will be conducted by a DMF-100 flow rate meter.

17.3 Distribution Systems

The feed of power to the open pit consumers will be carried out using 0.4kW cables, which will be installed on mobile brackets. The type of cables used is KG-KHL-0.66.

At the surface, all cables will be laid open to avoid damage, freezing, blockage by rocks etc. Pumping equipment will be installed using special hangers, attached to mobile wooden ramps.

17.4 Electric Lighting

The lighting of mines and waste dumps will be provided by mobile lighting masts (up to 9.4m high) with Atlas Copco diesel generators QAS 14. The power capacity of QAS 14 is 13kVa with 220V load.

The tank capacity of the diesel generator is 45 liters, the mast time battery life 13.5 hours. Each lighting mast is equipped with 6 halogen lamps of 1.5kV, with total capacity 9kW.

Lighting control of mining operations will be performed automatically depending on the level of natural lighting.

17.5 Earthing

Earthing of the diesel power plant, pumping station, drainage and lighting installations will be performed using local earthing switches. Earthing of electric consumers of the open pit mining operations will be provided from vertical and horizontal earthing switches.

17.6 Water Supply

Drinking and technical water supply will be sourced from surface waters. Water will be collected in the dam, which will be constructed on the river Uryum 6, located 500m southwest from the site and the Process Plant. Water is required to be supplied for mining and production purposes (technical water) and to supply the mine camp (potable water).

Studies of the river Uryum 6 found that the annual flow is 2.37Mm^3. A water balance for the reservoir including the river flow rate, direct precipitation to and evaporation from the reservoir as well as drainage losses, indicates the total amount of water collected in the reservoir will reach 3.0Mm^3 in the third year.

In order to keep sanitary protection zone within its borders, and to deal with years where above average precipitation or flows occur, a spillway and a drainage channel will be constructed in the downstream dam. Excess water will be discharged into the old river channel, away from the tailings dam.

The accepted reservoir location is optimal for the organisation of water supply for the camp and the mine site. In addition, this solution eliminates water ingress from the river Uryum 6 into the tailings dam area.

The following water supply systems are provided:

- Process water supply;
- Drinking water supply;
- Hot water from the boiler house; and
- Recycled water supply.

Automated monitoring and control systems will be used to control the pumping stations equipment.

The total domestic drinking water demand will be approximately 100m³/day. Water demand for domestic water supply is determined based on the number of employees on site, as well as the dining room, laundry room, boiler houses consumption and unaccounted losses. The water will be purified to the quality that meets the Russian drinking water standards. To achieve this water quality, the sorption filter series "Rosa Super 4000" will be used which has a performance of 4.0m³/h.

The Process Plant will be the main consumer of process water. Required volumes of process water supplied from the reservoir will be about 40m³/h. The water intake from the reservoir will be 0.55Mm³/year. Water will be transferred from the reservoir, via a pump station, into two 500m³ storage tanks. Water from these tanks will also be used for firefighting purposes.

The gold recovery plant will have an internal water circulation and turnover of purified water from the water dam of about 58m³/h.

Estimated water consumption of the entire enterprise is presented in Table 17.1.

Table 17.1: Estimated Water Consumption

<table>
<tr><th>No</th><th>Consumer</th><th colspan="2">Drinking water supply</th><th colspan="2">Process water supply</th></tr>
<tr><td></td><td></td><td>m³/day</td><td>Thou m³/year</td><td>m³/day</td><td>Thou m³/year</td></tr>
<tr><td>1</td><td>Shift camp</td><td rowspan="2">100</td><td rowspan="2">36.5</td><td>-</td><td>-</td></tr>
<tr><td>2</td><td>Production site</td><td>960.0</td><td>326</td></tr>
<tr><td></td><td>Total</td><td>100</td><td>36.5</td><td>960.0</td><td>326</td></tr>
</table>

Water conservation measures and the water recycling system is proposed for the site and further water conservation measures are taken in the summer by using pit water for household consumption and dust suppression.

The water extracted from the Open Pits will be used to irrigate site roads, waste dumps and as technical water for the Process Plant. Mobile water pumps will be moved from upper levels to lower levels of the pits as required during drier seasons. Water will be pumped to the surface and then to the allocated water settlement pond, where it would undergo treatment for suspended solids and oil in special modular USV-20 plants.

To review various options for drinking water supply for the project Nasedkino, in October 2016 the water supply well No7 was drilled to assess reserves of existing underground waters. The borehole was drilled in the valley of the river Maliy Uryum to a depth of 70m. At the time of reporting, Daltsvetmet is reviewing options to develop a water supply well system at the valley of river Maliy Uryum, which is located above of the mouth of the spring Nasedkino. For 2017, Daltsvetmet plans to conduct geological explorational works to evaluate and test underground reserves of water.

17.7 Selection of the Water Drainage Installation

Water control at the pits will take the form of interception of precipitation from the catchment area of the pit and dewatering of groundwater and direct precipitation encountered in the pit.

In pit water including precipitation and groundwater will be controlled by a series of ditches which conduct water to a sump at the base of the pit.

From the sump the collected water will be transferred by gravity, if topography allows, into water settlement ponds or will be pumped to a higher elevation.

The selection of pumping equipment is based on the maximum expected daily water production with a duration of no more than 20 hours with an additional 25% error margin.

Water removed from the pits will have suspended solids and oil products removed in two stages. The pond capacity will be based on the maximum daily inflow of water from the pits. Each pit will have their own collection ponds.

Structurally, the collection pond is made of two cascade settlers, separated by a filtering dam-bridge.

Sediment, formed on the bottom of the pond, will be disposed of at to the waste dumps.

17.8 Protection from the Surface Water

Surface water ditches will be constructed around the open pits and waste dumps to protect them. All water, collected by these ditches, will not contain any specific impurities, therefore will not require any special treatment.

17.9 River Flow Re-Direction Channel

The River Maliy Uryum is present on the western boundary of the Pridolinny deposit. A river diversion is proposed to ensure the river is diverted outside the water protection zone associated with the mine. In accordance with State Guidelines the water protection zone must be a minimum of 50m.

The river diversion channel (Figure 17.2) has been designed based on parameters from the river itself: bottom width 2.0m and depth is 1.5m with a 1:1 side slope.

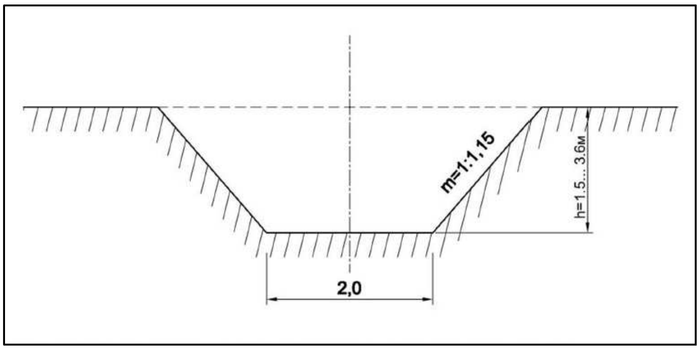


Figure 17.2: River Flow Re-Direction Channel

Note: At beginning and end of the channel the height of the section reduces to 1.5m.

Taking into account the actual speed of the water during the maximum flow rate (1.7m/sec), erosion protection will be required on the base and sides. To fulfill the requirement, the construction of a minimum 15cm thick stone wall will be required.

17.10 Heat Supply

Heat for all mine facilities, process plant and camp is planned to be provided from two separate boiler houses. One will be connected to all industrial infrastructure including the Process plant and another to the main camp. Both boilers will be fueled by brown coal taken from the Kharanorsky surface mine.

17.11 Access Road

Nasedkino is located some 56km away from Mogocha town.

Shipping of material for the period of construction and exploitation is provided by railway. The existing loading yard with a material storage area is located at Kisliy Kluch and belongs to Mangazeya Gold.

The main site access road runs from M58 motorway to Kisliy Kluch by gravel road and from Kisliy Kluch to the site by existing 32km gravel road (Figure 17.3).

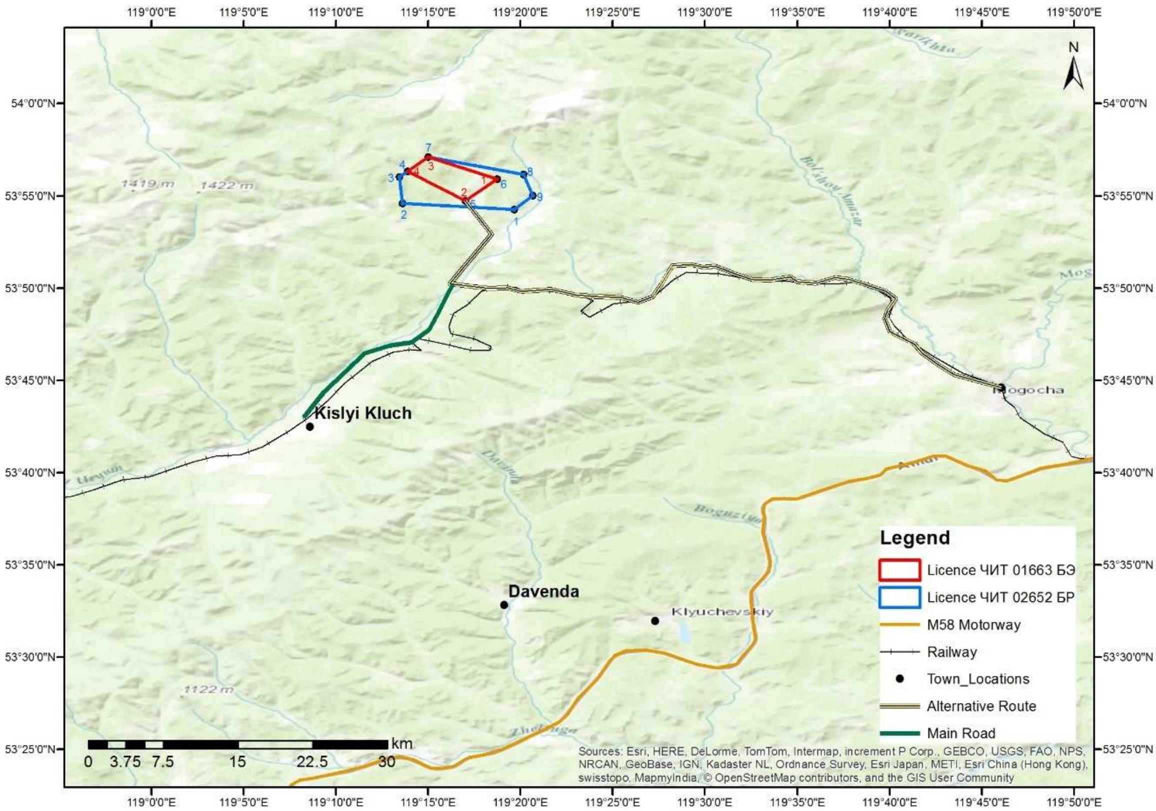


Figure 17.3: Site Access Road

The alternative site access road, which can be used for the haulage of goods passes from Mogocha to the Nasedkino site. The quality of the road is low.

17.12 Mine Camp

The mine camp design includes a number of residential and municipal buildings;

- Hostel for technical engineering personnel for 100 people;
- Accommodation for management – 16 flats;
- Bathing facility for management;
- Hostel for other personnel for 110 people – 3 units;
- Canteen for 136 people;
- Food storage;
- Administration building;
- Sanitary-amenity block;
- Sport complex with laundry facility and feldsher's point;
- Closed parking with fire station;
- Sports ground.

Engineering support infrastructure includes:

- Pump house for utility and drinking water system;
- Transformer substation;
- Boiler house; and
- Waste treatment facility.

17.13 Site Internal Roads

The site internal roads will be used for transportation purposes and to connect internal areas of the site. The total length of all internal roads is 17.8km, width 6.5m with 1.5m roadside.

The structure of the internal site road is show on Figure 17.4 below.

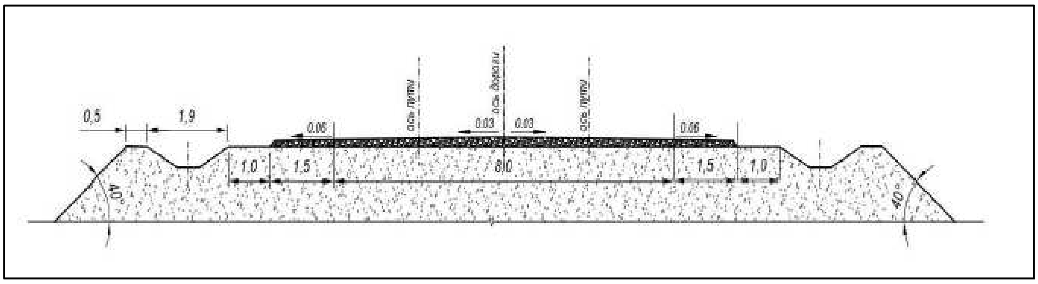


Figure 17.4: The Structure of the Site Internal Road

17.14 Tailings Storage Facility

During the processing of the Nasedkino ore using gravity-cyanide process route, the neutralised tailings from the absorption leaching are formed. Thickened and neutralised tailings will be stored at the provided specified tailings storage facility (TSF).

The location of the TSF has been selected based on comparison of various technical and economical parameters and in accordance with the State standards.

In order to store tailing it is recommended to neutralised pulp in thickeners. The thickeners design is based on Russian methodology depending on needed capacity.

- Feed rate 124t/hr;
- Feed rate 248m^3/hr;
- Content of solid material in feed 38%;
- Content of solid material in sand 55%;
- Productivity 0.204t/(m^2*day/hr);
- Required area of thickeners 607m^2;
- Required dia of the thickener 27.8m;
- Accepted dia of the thickener 30m;

- Actual area of thickening 706m^2;
- Amount of thickeners – 1;
- Flocculant consumption 0-20g/t of the feed; and
- Required thickener type – radical.

After the sorption leaching it is recommended to thicken pulp in the special structure which allow to proceed with the technology of paste storage, which is more ecological friendly and economical.

17.14.1 Tailings Storage Facility Design

In case of accumulation of drainage water, it is recommended to use them in the water recycling system of the Process plant.

For protection of surface and underground waters, it is recommended to equip the TSF with a solid polymer screen, such as membrane liner.

The membrane must be made of a clay base or from polymeric materials. For this reason, the base of the TSF must be levelled and compressed. The most optimal material for the construction of this compressed screen base is a fine sandy fraction. The maximum particle size should not exceed 5mm.

Based on chemical analysis data of the liquid and solid tailings the neutralised tailings from cyanidation are almost harmless to the environment.

18 MARKET STUDIES AND CONTRACTS

18.1 Market Studies

A Market Review has been carried out to provide an estimation and forecast of the world consumption of gold and to identify the long-term Au price for the Nasedkino project. The market analysis and review has been prepared as of February 2017.

A refinery loss was assumed at 99.94%.

As per current agreement with Krastsvetmet, the refinery cost is specified at the following rates:

- 4.95 Rub/g for 85,01% Au content; and
- 6.00 Rub/g for 65,01% - 85,00% Au content.

As instructed by the Client, WAI has used the second option (6 Rub/g) to follow a more conservative approach. WAI, notes that there is no reason to believe that a quality over 85% is not going to be achievable and, consequently, lower cost refinery could be utilised.

No by-products were considered in this valuation.

18.2 Market Review and Price

According to the World Gold Council (WGC), in 2016, investors around the world returned in large numbers to the gold market, as a combination of macroeconomic drivers and pent up demand kept interest in gold high. In the beginning of 2017, there are some concerns that US dollar strength may limit gold's appeal. WGC believes that, on the contrary, not only will gold remain highly relevant as a strategic portfolio component, but also six major trends will support demand for gold throughout 2017: heightened political and geopolitical risks, currency depreciation, rising inflation expectations, inflated stock market valuations, long-term Asian growth, and opening of new markets.

The gold price had a strong performance in 2016, rising close to 10% in US dollar terms (higher in most other currencies) and amassing multi-year record inflows through physically-backed gold ETFs (an exchange-traded fund) - making it one of the best performing assets last year, despite a post-US election pullback. The price has gained more than 5% since the Federal Reserve (Fed) increased rates in mid-December.

The Competent Authority for the world Bullion Market, LBMA, forecast contributors are predicting that the gold price will average $1,244/oz in 2017, 5.3% higher than the first half of January 2017, but broadly in line with the actual average price in 2016.

In contrast, the World Bank expects gold prices to fall 8% in 2017 (resulting in forecasted US$1,150/oz in 2017), mainly due to weak investment demand, prospects of a stronger dollar, and rising real interest rates. Downside risks to the forecast are stronger economic growth and faster than expected increases in U.S. interest rates. Upside risks include geopolitical tensions, stronger demand in China and India, delayed rates hikes, and mine supply shortfall.

LBMA expects 2017 to be an eventful and unpredictable year given the high degree of geopolitical uncertainty, with a more nationalistic US President in residence and the indications that the UK will pursue a hard Brexit in its negotiations with the EU. There is also the prospect of further uncertainty with elections to be held later in 2017 in both France and Germany, as well as the potential for tensions between the US and China.

Greater uncertainty should prove positive for gold as could higher inflation if the new US administration adopts reflationary policies. On the downside are the anticipated three US rate hikes in 2017, a stronger US currency and rising stock prices as well as continued weak demand from both China and India.

Analysts are predicting that the gold price will trade in an average range of $1,101/oz to $1,379/oz (source: LMBA).

WAI suggested to follow the more conservative approach and, therefore, the long-term gold price of US$1,100/oz was selected for reserve estimation of the Nasedkino project.

An upside scenario with Au price of US$1,200/oz, as well as sensitivity analysis demonstrating project behaviour depending on the variable gold prices was also performed. Sensitivity analysis results could be found in the relevant chapter of this report.

19 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

19.1 Introduction

19.1.1 Scope of Study

The scope of this chapter is to review and assess the information available regarding environmental and social performance of the Nasedkino Gold Mine and associated processing.

Whilst WAI believes it has gained sufficient insight into the key issues, there may be additional information that was not seen, or variations in interpretation of the available data that could not be explored further. The development is still primarily a green field site and therefore some aspects of staffing and environmental management have not yet been determined.

19.1.2 Method of Study and Information Sources

A site visit of the Nasedkino Project was undertaken by Richard Ellis and Mark Mounde between August 9, 2016 to August 11, 2016, covering aspects related to access and infrastructure, geology, exploration, QA/QC, mineralogy, mining, laboratory testwork, processing and environmental and social issues.

Primarily the environmental and social data reviewed has been sourced from documents translated from:

- Development of Gora Pyataya and Pridolinniyi sites of Nasedkino gold deposit using open pit mining, Volume 8, List of environmental protection measures, SGZ.14 / 06-CAB, Precious Metals Institute, July 2016.
- Audit Report for Nasedkino Deposit and Uryum Exploration Area - SRK Consulting (Russia) Ltd, December 2011
- Process Plant Tailings based on the gold deposit for "Nasedkino", Precious Metals Institute, July 2016.
- Industrial Safety Declaration for Tailings Storage Facility, Precious Metals Institute, July 2016.The Data on the technical equipment, utility networks, list of engineering and technical measures, technical solutions, Precious Metals Institute, July 2016.
- Explanatory Note for Tailings Storage Facility, Precious Metals Institute, July 2016

19.2 Environmental & Social Setting and Context

19.2.1 Landscape, Topography

The deposit is located in the north Nasedkino Mogochinsky area in the Trans-Baikal Territory. This area is known as the Olekminsky Stanovik Vitim-Olekma Mountains, and the deposit is located along the western part of the mountainous ridge called Cheremnyh. The surrounding mountains range in height from 800 to 1252m. The ridges are effectively flattened peaks, sometimes complicated by denudation outliers. The slopes of the ridges are steep and in places reach 40º.

The town of Mogocha, which is the administrative centre of the Rayon, is the closest town to the deposit; located 37km northwest from the deposit in a straight line. The distance from the deposit using the winter roads is 62km. The other closest settlements to the deposit include the village of Chaldonka (15.5km) and the mining village of Klyuch (45.5km). The closest railway station on the Zabaikalskaya Railway is located at Arteushka and is approximately 16km south of the Nasedkino deposit.

19.2.2 Climate

Climate information has been derived from data collected by the Mogocha meteorological station for the observation period from 1961 to 2004 [YF Kharitonov, 2008]. The climate is sharply continental characterised by long winters and short summers with sudden changes in air temperature and atmospheric pressure during the day regardless of the season.

Winter is typically calm and very cold. Spring is short and dry, spring frosts diminish towards the end of June. Summer is short and warm, hot in some years. Autumn is characterised by early frosts, clear and mostly dry weather.

The average annual temperature is -4°C. The coldest month is January, where the average temperature is -28.7°C and the absolute minimum air temperature is -53°C. The warmest month is July where the average temperature is +17.2°C and the absolute maximum is +37°C. The maximum and minimum temperatures give a significant temperature range of 90°C.

The Wind regime depends largely on the season. Winter months are calm, but in the spring (April-May), in connection with the arrival of low pressure systems, wind speed varies considerably. The average monthly wind speed for spring is 2.8m/s. whereas the maximum speeds can reach 24m/s with gusts - 28m/s. The average annual relative humidity is 72%. The lowest humidity occurs in April-May (56-59%), the highest (76-79%) in November and January. Total annual precipitation is 430mm, 93% of which occurs in summer. The highest daily amount of rainfall occurs in July-August. Snow cover on average appears on the 9th October and has all melted by 5th May. The number of days with snow cover is therefore in the region of 163 days. The snow cover is observed in the period from 28th October to 26th March where the average snow depth is 16cm.

Fog occurs on average for 46 days a year with the most frequent occurrence in August where 13 days of fog can usually be expected.

19.2.3 Land Use and Land Cover

There are different landscape terrains associated with the mine concession area, namely, Uplands Cap, Upland Lower Slope, Lowland Cap, Lowland Slope, and Lowland valley flat in addition to the historic tailings landscape.

Almost all slopes, valleys and peaks are covered by Nasedkin forests which are dissected by common marsh and meadow vegetation. Forests are both deciduous and coniferous, which consist predominantly of Birch and Pine. Understory species include Alder, Dahurian Rhododendron, Dwarf Birch, and Labrador Tea. Some areas have become overgrown with Red Bilberry. Cedar Elfine wood dominates the areas of highest water shed.

Much of the land is of natural habitat and has been used only for historic mining practises. A potential endangered species has been identified as the Lynx. Confirmation of whether this is the Russian or Eurasian Lynx is required to determine vulnerability and potential impacts on the project.

19.2.4 Water Resources

The main surface water feature within the Nasedkino resource area is the Small Uryum River. The watershed for the river's catchment area is located on the border between Nercha-Shilkinskim mountain-taiga regions (Muroysko-Uryumsky subdistrict) and Amazar Oldoyskim (northern mountain subdistrict).

The Small Uryum River originates in the mountain range Cheremnyh (Olekminsk Stanovik) and has a catchment area of 38.8km^2. Within the Nasedkino resource area, there are six small streams which range in length from 0.6 to 3.5km (with a total length of 11.3km) before joining the Small Uryum River. The Small Uryum runs for about 13km before joining the Black Uryum River as one of its tributaries. The river mouth is located a further 115km away. In accordance with the National Water Body Register, the Nasedkino resource area lies within the water use area known as Shilka Amur Basin District.

Permafrost can be found in all parts of the catchment area with areas of unfrozen ground occurring under rivers, on slope aspects which catch the sun and along fault zones. The width of the river in the Nasedkino resource area is on average 5-7m which can increase to 15m during times of flooding. The average annual flow of the river is 7.59l/s/km^2 and the maximum flow during a flood events is 595l/s/km^2. Between May and September, the river will generally flood two to three times, but in some years much drier conditions have prevailed and the river has not flooded. The river is not thought to have dried out during these periods as no such event has been recorded. Rivers and streams freeze in the winter and can remain frozen for up to 170 days, often up until the end of April.

During the thaw, interaction between surface waters and ground water occurs and therefore the chemical composition of both, is similar containing sulfate-bicarbonate-calcium-sodium and sodium bicarbonate. The corrosive nature of these chemicals in the surface and ground water needs to be carefully considered to ensure protection of structures such as reinforced concrete and cabling.

To reduce the risk of pollution and to ensure stability of the waste dumps, a side water drain is planned to prevent run off from entering the waste dump from the surrounding higher ground.

19.2.5 Wastewater

The company proposes to use three separate sewerage systems for domestic, rain and quarry waters. Sewerage treatment schemes are the same for all variants of gold cut-off grade.

Domestic wastewater from the shift camps and welfare facilities in the industrial areas are received by the wastewater treatment plant which has been designed for the complete biological treatment of domestic wastewater which includes organic and suspended solids, nitrogen and phosphorus. The concentration of contaminants in the discharged effluent are reduced to levels considered safe for aquatic life. The sewage treatment plant will have a capacity to treat $400m^3$ of sewage per day. Treated water is then discharged into the Small Uryum River. Discharge points require agreement with the regions department of Federal Agency for Fishery.

19.2.6 Communities and Livelihoods

The closest town to the Nasedkino deposit is Mogocha with a population of 13,697 and is also the administrative centre of the Rayon. Distance from the deposit to the closest village of Chaldonka is 15.5km and the mining village Klyuch is 45.5km away.

According to the TEO document, the unemployment rate is very low and there is limited available work force in the Rayon. This may be a challenge for the prospective project.

There is an official letter which confirms that there are no indigenous people or their homes contained within the Nasedkino deposit area.

19.2.7 Infrastructure & Communications

The distance from Mogocha to the Nasedkino deposit using winter roads is 62km. The closest railway station is at Arteushka approximately 16km south of the Nasedkino deposit and is part of the Zabaikalskaya Railway. A railway siding at Kisliy Kluch has been purchased for offloading of material for the construction of the site.

19.3 Project Status, Activities, Effects, Releases & Controls

19.3.1 Project Description & Activities

19.3.1.1 Past Activities

Geological prospecting within the area was started after gold-bearing placers were discovered in the basin of the river Cherny Uryum in the latter half of the 19th century. In the early 20 century, exploration works renewed due to the construction of Trans-Siberian railway.

The geological works were carried out with varied intensity, and based on the results a complex anomalous zone was discovered. The zone strikes to the northeast, and is 10km by 2km.

19.3.1.2 Current Operations

Woodland has been cleared from the mine areas and the processing plant area. There has also been some ground work preparation, but as yet there has been no construction.

Infrastructure required by the project includes; five open pits and associated waste rock dumps, water and sedimentation ponds, tailings management facility, water storage dam and treatment plant, process plant, explosives store and workers accommodation.

19.3.2 Mine Wastes – Rock

Four waste rock dumps have been designed as part of the mine planning and are located adjacent and down slope of each of the open pits.

In accordance with current regulatory documentation and preliminary research, the overburden that will be taken from the Nasedkino open pits is not considered to be contaminated and therefore classified as a category 5 hazard. The waste to ore ratio is expected to be 99%. To reduce the volume of waste dumps, the overburden will be utilised fully in the mine construction including roads, tailings and reservoir dams.

SRK Consulting (Dec 2012) produced a geochemical report for metal leaching and acid rock drainage (ML/ARD). The report considers waste rock samples from all of the open pits with the exception of Pravoberezhny.

Due to the presence of sulphides, SRK considers that there is potential for acid rock drainage (ARD) which needs to be taken into account for the planning of waste dumps, tailings facilities and mine closure. SRK recommends that the Company schedules some quick static tests to determine the potential for ARD so that this risk can be mitigated. Should ARD be identified then longer kinetic tests will be required.

19.3.3 Mine Wastes – Tailings

19.3.3.1 Tailings Properties & Treatment

The tailing dam facility will form part of the first stage of construction and will comprise of a Tailings dam, pumping stations, observation and control system, piped movement of slurry, road, ditches and a diesel power station.

The tailings dam will be constructed in line with the standards for a class II hydraulic structure. The maximum height of the dam will be set at 42m with upper and lower slope gradients set at 1:3 and 1:2 respectively. The width of the dam across the crest will be no less than 12m which broadens to 251m at the base. The capacity is expected to be sufficient to last 12 years.

There are no locally sourced impermeable soils and as such the tailings dam will be constructed primarily of waste rock with an impervious screen in the form of a three dimensional polymer cell with a geotextile membrane and outer covering.

19.3.4 Emissions to Air

The project has defined an 'Air Protection' plan in accordance with Russian requirements. This plan incorporates pollutants produced during the combustion of fuel for electrical generation and transportation, blasting and crushing. Charges for the emission of pollutants into the atmosphere have been calculated and a process for monitoring defined.

There is a permit for emissions of pollutants into the air for the period of 10.04.2012 to 09.04.2017. The permit specifies approved and permissible emissions.

The following mitigation measures will be taken to minimise air pollution:

- Investment in modern mining equipment with high levels of efficiency and dust suppression Rules for the operation of diesel engines;
- Regular maintenance of vehicles;
- Regular monitoring; and
- Watering to reduce availability of dust.

19.3.5 Waste Management – General

A waste management plan has been developed for the site in accordance with requirements of the State authorities and provides hazard classification for each type of waste produced.

Over burden has not been classified as waste and can therefore be used for construction across the mine site thereby reducing the size of the waste rock dumps.

Industrial waste will where possible be returned to the supplier for recycling. Metal wastes will be sent for recycling at steel plants. A landfill will be constructed to accommodate all other waste.

Woodland cleared from the areas of the open pit and tailings will be used for construction and heating around the mine site.

19.3.6 Hazardous Materials Storage & Handling

Consideration is given to the potential for accidents which might result in the release of pollutants into the environment. The primary risks have been identified as a dam breakage or leak and fire from waste oil accumulations.

To mitigate, the dam will be regularly inspected. Repairs will be made when necessary. To prevent accidents involving fire and waste oil combustible lubricants and fuel, these will be stored in designated areas in metal containers, mounted on pallets, in compliance with fire safety regulations set by the fire service.

19.3.7 Fire Safety

A Fire response plan will be developed in accordance with Russian regulation. Early risk assessment has been undertaken with regards storage of combustibles/oils and waste.

19.3.8 Security

There is currently a security post on the access road entrance. Fencing of the site will be conducted with commencement of construction of the mine infrastructure.

19.4 Permitting

19.4.1 ESIA/OVOS

A list of environmental protection measures for the Gora 5 and Pridolinny sites which form part of the Nasedkino mine development, has been produced. A check should be made to ensure that mitigation measures apply to all planned mine development areas.

An OVOS has been completed and is awaiting approval from the local expert in Chita. The submitted OVOS caters for all parts of the mine construction and operation.

19.4.2 Environmental Permits and Licenses

The Project will be subject to laws, regulations, guidelines and standards of the Russian Federation.

The project holds licenses covering the use of; subsoil for prospecting; exploration and extraction of underground water for portable and domestic use. The expiry date for the license is 01.08.2041 this agreement has been settled between the Ministry of Natural Resources, Industrial Policy of Zabaikalsky Krai and Daltsvetmet LLC.

There is a permit for the emission of pollutants into the air covering the period of 10.04.2012 to 09.04.2017. This will need to be extended to cover the life of the mine and the levels of pollution reviewed to include mining construction and operation.

There is a permit for approved levels of Domestic Waste which covers the period from 20.02.2012 to 19.02.2017. This will need to be updated based on current waste generated and extended to cover the life of the mine.

There is an agreement in place between the Government of Zabaikalsky Krai and Mangazeya Gold JSC, which regulates the social and economic cooperation between the State and the Company.

In addition to the licenses discussed above, the mine will need to obtain licenses and permits for:

- The abstraction of water for technical use at the process plant and general industrial use across the whole mine site; and
- Discharge of water into rivers.

19.4.3 Stakeholder Dialogue and Grievance Mechanisms

LLP Daltsvetmet has an agreement with the local authority regarding the implementation of the Nasedkino project. The agreement has allowed involvement of local specialists in the production operations of Nasedkino.

19.4.4 Social Initiatives and Community Development

LLC Daltsvetmet has in cooperation with the Regional Government paid approximately $50,000 for the repair and improvement of the village club used primarily by the people from Chashin-Ildikan and Nerchinsk-Zavadski districts. Villages are reported to be very happy with the assistance and community support given by Daltsvetmet. It is recommended that any social initiatives and community development opportunities are combined into a community development plan to comply with international best practice.

19.5 Mine Closure & Rehabilitation

Rehabilitation will be conducted in accordance with GOST 17.4.3.02-85, GOST 17.5.3.04-83, and GOST 17.5.1.01-83. There are plans for both the technical and biological reclamation of the site.

19.6 Conclusions

The OVOS will be final following review by the local Chita authorities.

From the documentation provided for review it is understood the project is currently in compliance with in country requirements for the exploration phase, and with the OVOS being finalised, the mine will be in compliance for the operational phase.

20 CAPITAL AND OPERATING COSTS

20.1 Summary

The Nasedkino Project will require total pre-production capital cost of US$84.2M (including contingency) with overall LOM capital requirements of US$99M.

The LOM cost of production was estimated at US$244M or total cash cost of US$420/oz and includes operating mining and processing costs, auxiliary works and general and administration, royalty (mineral tax) and property tax.

All-in sustaining cost was estimated at US$443/oz. Figure 20.1 below shows all-in sustaining cash cost distribution.

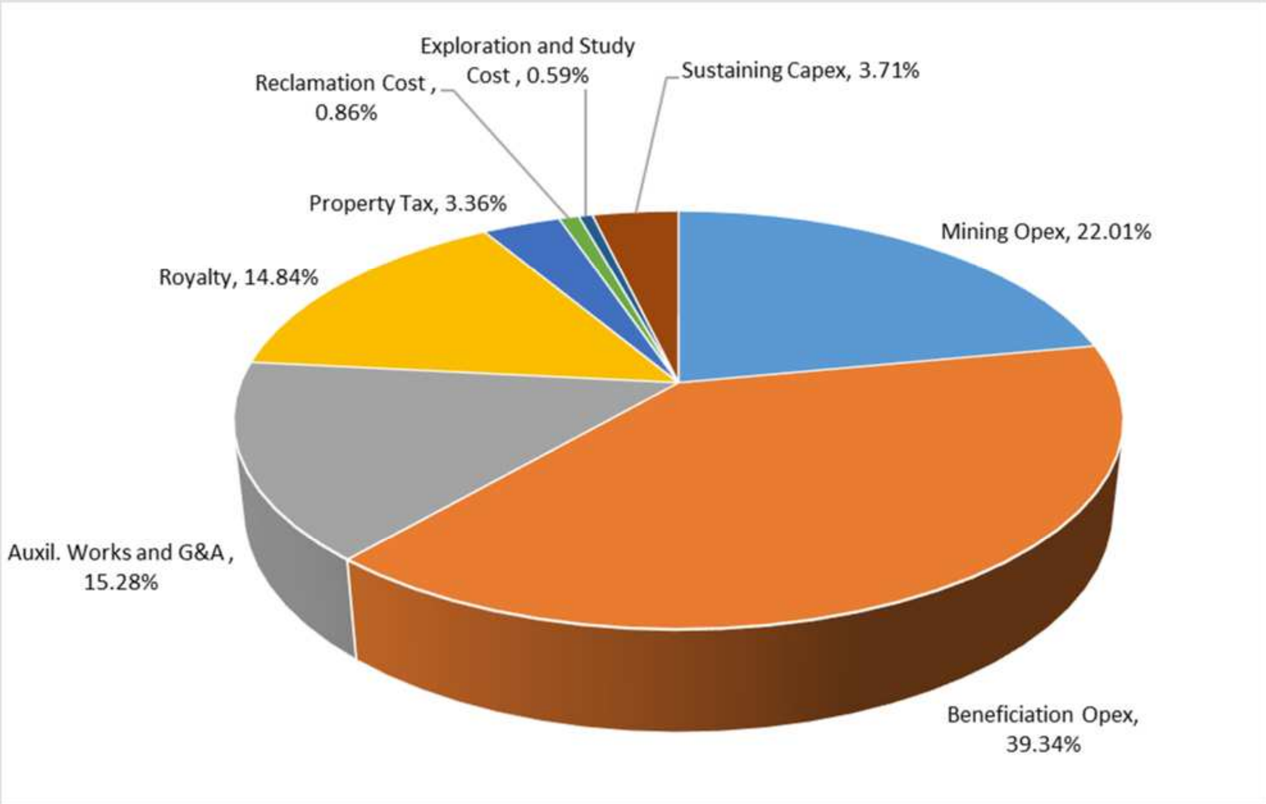


Figure 20.1: All-In Sustaining Cash Costs Breakdown

The project operating costs have been estimated by the client and reviewed by WAI engineers. Operating costs were derived from first principles based on development of the open pit mine.

The WAI financial model was built as of Q1 2017.

20.2 CAPEX

Capital costs were derived from first principles based on the minable volumes and actual cost of equipment received as of end of 2016, exclusive of VAT.

Direct capital costs consider the following:

- Preparatory and development works on site;
- Key construction objects;
- Service and maintenance facilities;
- Communications and transport facilities;
- Power, water, sewage and heating facilities; and
- General infrastructure.

Indirect capital costs are formed by the following:

- Temporary buildings and facilities;
- Engineering studies and survey; and
- Other costs.

The overall capital costs requirement for the Nasedkino Project is **US$99M.** A summary of the project capital costs is given in Table 20.1 and presented in Figure 20.2.

Table 20.1: Overall Project Capital Costs Summary

Project design, Engineering Study, Permitting	US$ M	1.5
Mining Equipment	US$ M	18.6
Gold Processing Plant	US$ M	37.9
Power Infrastructure	US$ M	5.5
General Infrastructure	US$ M	15.7
Mining Equipment Maintenance	US$ M	5.6
Plant Maintenance Cost	US$ M	3.9
Contingency (10%)	US$ M	7.9
Mine Closure and Reclamation	US$ M	2.22
Total Capital Costs	**US$ M**	**99**

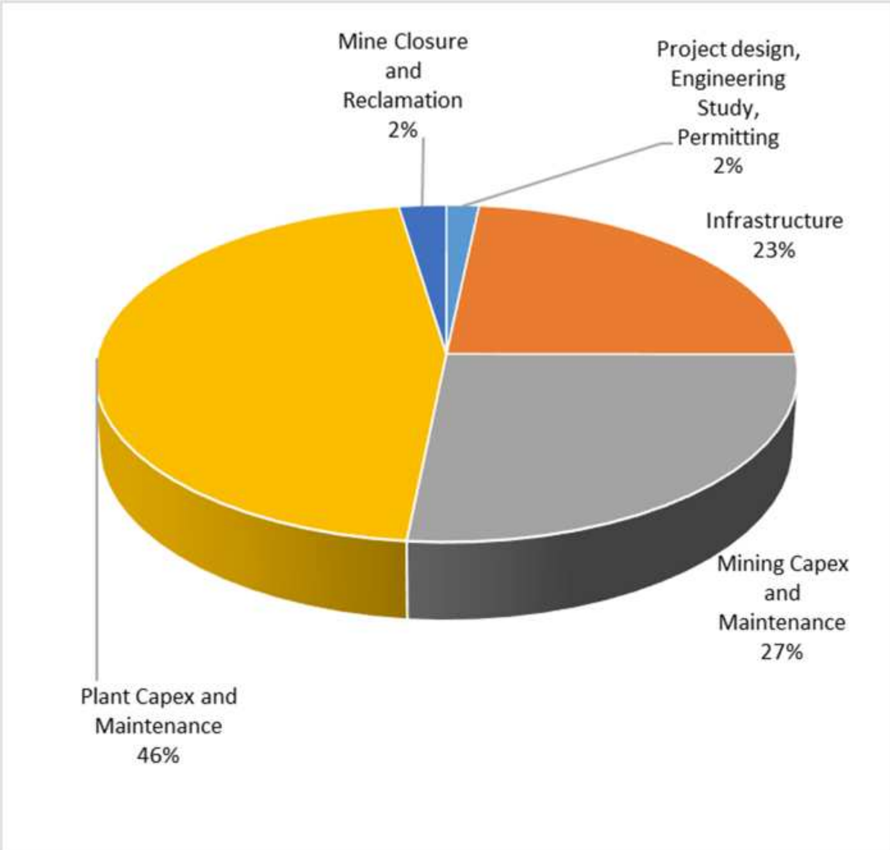


Figure 20.2: Project Capital Costs Distribution

The contingency is defined as a provision to cover unforeseen items of work (the unknowns) which will have to be performed or elements of cost which will be incurred within the defined Scope of Work of the estimate, but cannot be explicitly foreseen or described at the time the estimate is being prepared because of lack of complete, accurate and detailed information. For example: the impact of undetected latent conditions, risk of contractor insolvency, technology risk, methods of construction, industrial effects, and unexpected geotechnical conditions.

The contingency estimate for the project is broken down into the risk drivers identified as follows:

- project definition;
- engineering design efforts;
- site data and test work;
- known omissions of minor cost elements.

The determination of the project contingency is dependent on the risk profile adopted for the project. A contingency rate of 10% of the construction and installation costs was assumed in this valuation.

Life of mine contingency cost was estimated at US$7.9M.

20.2.1 Mining Capital Costs

The life of mine mining capital cost was estimated at US$24.2M, including initial fleet purchase cost of US$18.6M and sustaining capital required for equipment renewal of US$5.6M.

A summary of the mining capital expenditures required for Nasedkino Project development is given in Table 20.2 below.

Table 20.2: Project Mining Equipment Costs

Main Equipment	Number of Items	Cost per 1 item	Total RUB'000	Total US$'000
Hydraulic Excavator Komatsu PC1250SP-7 (front shovel for waste, backhoe for ore)	4	52,000	208,000	3,200
Belaz Dump 7555 B (waste)	13	32,500	422,500	6,500
Belaz Dump 75473 (ore)	2	32,500	65,000	1,000
Bulldozer Komatsu D65EX	5	45,500	227,500	3,500
Drill Rig ATLAS COPCO FlexiROC D65	4	52,000	208,000	3,200
Hydro - Hammer Delta F7 on the basis of Komatsu excavator	1	350	350	5
Auxiliary Equipment				
Grader DZ-98V.00010	2	7,000	14,000	215
Combined Road Machine KO-829B1	1	5,000	5,000	77
Truck crane KC 45717	1	5,000	5,000	77
Mobile repair Shops (PARM) URAL 4320-41	1	3,500	3,500	54
Towing Belaz-7447	1	32,500	32,500	500
Bus Ural-32551-3171-59	2	3,500	7,000	108
On board vehicle Ural -4320	1	3,000	3,000	46
Minibus УАЗ 2206	1	500	500	8
Fuel Tanker ATZ - 10 on the basis of the Urals 4320-1951-60	2	4,000	8,000	123
TOTAL	**41**		**1,209,850**	**18,613**

20.2.2 Processing Capital Costs

Total processing capital cost was estimated at US$37.90M, with additional US$3.9M sustaining capital cost (Table 20.3).

Table 20.3: Project Processing Capital Costs				
Cost Allocation Schedule	**RUB M**	**US$ M**	**Year 1**	**Year 2**
Gold Processing Plant (Building, construction, installation, equipment)	1,828	28.13	25%	75%
Tailings Dam	454	6.98	*	*
Water Reservoir	182	2.79	40%	60%
Plant	2,464	**37.90**		
Plant Maintenance Cost	265.57	**3.9**		

** Note: Tailings dam facilities construction works are scheduled for 2017 (60%), 2020 (20%) and 2023 (20%).*

20.2.3 Infrastructure Capital Costs

Nasedkino Project infrastructure capital costs estimate was performed by the Client based on the several tender quotations. A summary of these costs is given in Table 20.4.

Table 20.4: Project Infrastructure Capital Costs (US$M)		
	RUB M	**U$D M**
Power Infrastructure	**360**	**5.54**
Other Infrastructure	**1,255**	**19.31**
Road "Kisliy Kluch - Nasedkino"	200	3.08
Boiler	96	1.47
Camp	551	8.47
IT	10	0.15
Other	167	2.56
Checkpoint	1.50	0.02
Blasting materials storage	13.00	0.20
Lubricants storage	30.00	0.46
Materials and machinery storage	20.00	0.31
Reagents storage	27.00	0.42
Sewage	60.00	0.92
Water Reservoir facilities	15.00	0.23
Reclamation	144	2.22
TOTAL	**1,527**	**23.50**

20.3 OPEX

Nasedkino operating costs (Table 20.5) have been derived from first principles, using actual consumable rates and technical requirements identified in the TEO 2016 report.

Operating costs were estimated in local currency (RUB) and converted to US$ using a fixed 65 RUB/US$ exchange rate.

Table 20.5: Project Operating Costs Unit Rates

Stages	Units	Materials	Lubricants	Power	Total
Ore Mining	**US$/cu.m.**	**0.64**	**0.47**	**0.00**	**1.11**
Drill and Blast (ore)	US$/cu.m.	0.51	0.08	0.00	0.59
Haulage (ore)	US$/cu.m.	0.11	0.19	0.00	0.30
Mining (ore)	US$/cu.m.	0.01	0.20	0.00	0.22
Waste Mining	**US$/cu.m.**	**0.65**	**0.46**	**0.00**	**1.12**
Drill and Blast (waste)	US$/cu.m.	0.56	0.07	0.00	0.63
Haulage (waste)	US$/cu.m.	0.07	0.18	0.00	0.26
Mining (waste)	US$/cu.m.	0.02	0.21	0.00	0.23
Ore Processing	**US$/t**	**6.35**	**0.00**	**3.18**	**9.53**
Ore preparation and thickening	US$/t	1.63	0.00	3.18	4.81
Hydromet section Gravity Conc	US$/t (conc)	0.05	0.00	0.00	0.05
Sorption	US$/t	1.75	0.00	0.00	1.75
Sorption section	US$/t	0.35	0.00	0.00	0.35
Decontamination	US$/t	2.56	0.00	0.00	2.56
Pyrometallurgical Section	US$/t	0.00	0.00	0.00	0.00
Auxil. Production	**US$/cu.m.**	**0.12**	**0.35**	**0.00**	**0.47**
Auxil. Production	US$/cu.m.	0.12	0.35	0.00	0.47
G&A	**US$/t**	**0.00**	**0.00**	**0.17**	**0.17**
G&A	US$/t	0.00	0.00	0.17	0.17

A summary of overall project operating costs breakdown is presented in Table 20.6 and demonstrated graphically in Figure 20.3.

Table 20.6: Project Operating Costs per t of Ore Mined and Au oz Sold

	US$/t ore mined	US$/oz sold
Mining Opex	6.21	98
Beneficiation Opex	11.10	174
Auxil. Works and G&A	4.31	68
Royalty	4.19	66
Property Tax	0.95	15
TOTAL	**26.75**	**420**

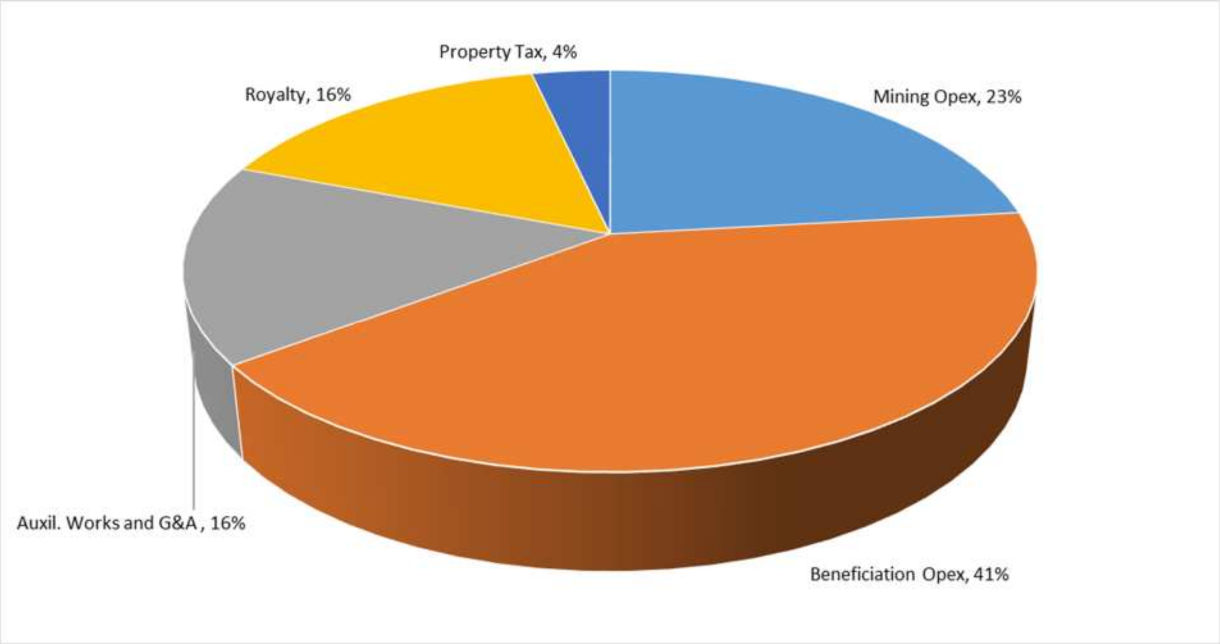


Figure 20.3: Project Operating Cost Breakdown

20.4 Escalation

No escalation or inflation have been included in the current valuation. There is also no allowance made for any fluctuations in currency exchange throughout life-of-mine.

20.5 Forex

The base currency for the project is US Dollars (US$). The exchange rate for this valuation was fixed at 65 RUB/US$.

21 ECONOMIC ANALYSIS

21.1 Summary

The Nasedkino Project has been examined from a financial perspective using Discounted Cash Flow (DCF) analysis, from which Post Tax Net Present Value (NPV), Internal Rate of Return (IRR), payback period and other measures of project viability have been determined.

Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The post-tax financial model was developed as of a project start date of 01 January 2017 based on the 10 years project life. The life of mine resulted in 8.5 years, commencing in third quarter 2018, with the annual processing capacity targeting 1.1Mt of ore.

The results of the economic analysis presented in this Study indicate that within the given economic environment and based on technical and economic parameters used, the Nasedkino Project has 3 years of payback period (based on debt-free discounted cash flows) and NPV of US$127M at 10% base case discount rate, and project internal rate of return of 82%, using Au price of US$1,100/oz. An upside scenario resulted in US$154M (at 10% discount rate) using US$1,200/oz price.

A financed scenario resulted in base case NPV (at 10% discount rate) of US$126M and US$153M, using US$1,100/oz and US$1,200/oz respectively.

An all-in sustaining cost was estimated at US$443/oz (using Au price of US$1,100/oz). A chart showing distribution of the AISC is given below (Figure 21.1).

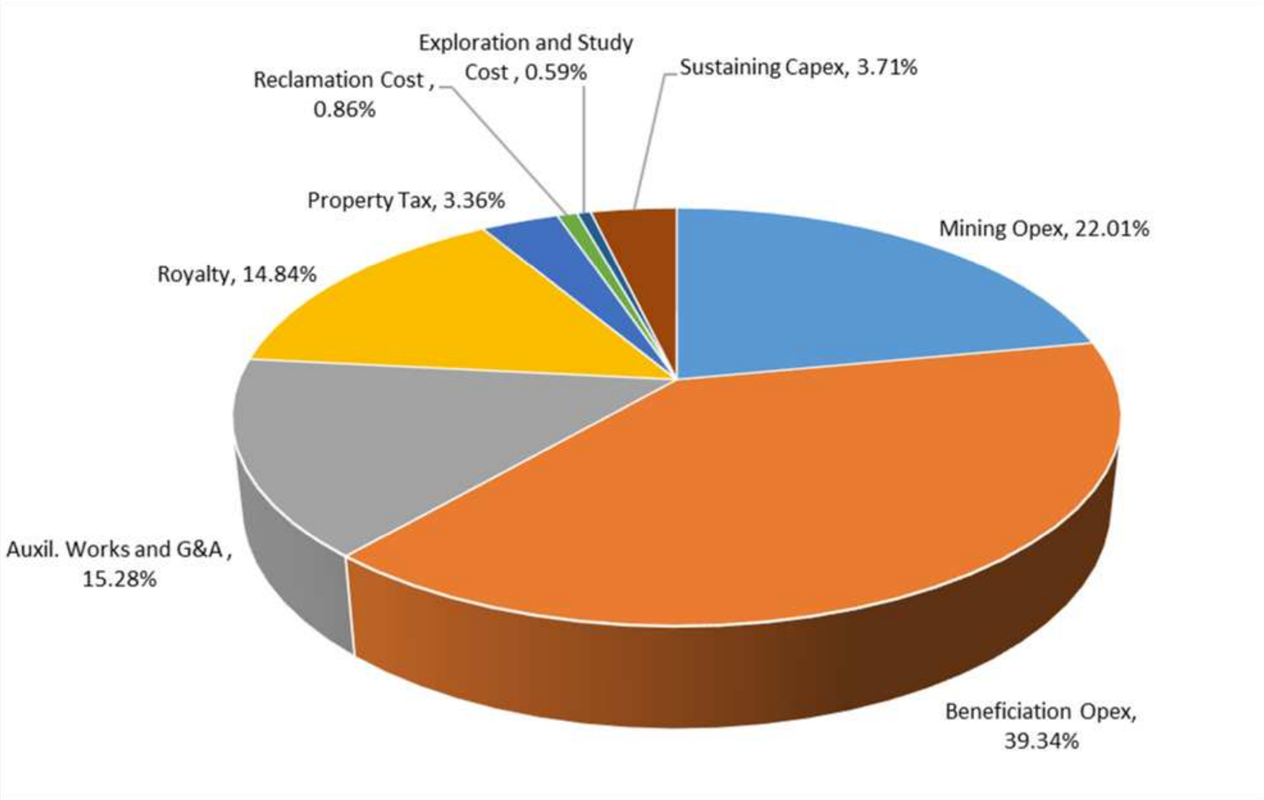


Figure 21.1: Project All-In Sustaining Costs

21.2 Capital Costs

The overall capital costs requirement for the Nasedkino Project were estimated at $99M.

The Project capital costs structure is presented in Table 21.1.

Table 21.1: Overall Project Capital Costs Summary

Project design, Engineering Study, Permitting	US$ M	1.5
Mining Equipment	US$ M	18.6
Gold Processing Plant	US$ M	37.9
Power Infrastructure	US$ M	5.5
General Infrastructure	US$ M	15.7
Mining Equipment Maintenance	US$ M	5.6
Plant Maintenance Cost	US$ M	3.9
Contingency (10%)	US$ M	7.9
Mine Closure and Reclamation	US$ M	2.22
Total Capital Costs	**US$ M**	**99**

21.2.1 Depreciation

Tax depreciation was calculated using 8 years of useful life for mining machinery and equipment; and 10 years for buildings, processing equipment, facilities and other assets.

No salvage value has been included in the cash flow analysis.

21.3 Operating Costs

Project total cash costs were estimated at $244M or US$420/oz of gold sold.

A summary of overall project operating costs breakdown is presented in Table 21.2.

Table 21.2: Project Operating Costs		
Opex, including	US$ M	157.63
Open pit cost	US$ M	56.54
Ore processing cost	US$ M	101.08
Auxiliary Works and G&A	US$ M	39.26
Royalty	US$ M	38
Property Tax	US$ M	9
Total Cash Costs	**US$ M**	**244**
TCC per oz	***$ / oz***	**420**

21.4 Key Project Inputs and Assumptions

A post tax cash flow model has been constructed to ascertain the economics of the project, based on the WAI production schedule at 1.1Mtpa production rate.

The model provides results for two scenarios: a 100% equity basis, and a financed scenario with US$93M debt.

The financial model inputs and results are expressed in current US Dollars. A fixed currency exchange rate was used at 65 RUB / US$.

As the economic analysis is based on the capital cost and operating cost estimates, the same qualifications, assumptions, and exclusions apply. Thus, no allowance was made for currency exchange rate fluctuations or price escalations, as for the capital cost and operating cost sections.

21.4.1 Revenue

The revenue estimation was based on realisation of 580koz at the fixed long-term price of US$1,100/oz. Refinery recovery used in the valuation was estimated at 99.94% and US$2.9/oz in Dore, as provided by the Client on the basis of an existing agreement with Krastsvetmet. Actual transportation and insurance cost resulted in 3.07 Rub/g (or US$0.05/g) in 2016 being used for this valuation.

21.4.2 Taxes

The following taxes were considered in the current valuation:

- Corporate Income Tax at 20% of the Project taxable income;
- Mineral Extraction Tax at 6% of the gold value contained in Dore bars, after sales costs deducted;
- Property tax of 2.2% of the Project fixed assets; and
- 0% VAT rate.

21.4.3 Working Capital

Working capital estimates have been included in the model using 15% (approximately 2 months' worth of costs) of the annual operating costs.

21.4.4 Project Financing

The base case financial model has been developed on the debt-free scenario basis. However, additionally a financed scenario was added on the back of the existing project model.

As instructed by the Client, a US$93M debt was assumed to be borrowed in year 1 and year 2 of the project life, prior any mining activities are commenced.

An interest rate was assumed at 12% and first year of the debt payback is scheduled in year 3 of the project life or in the first year of mining.

21.5 Discounted Cash Flow Model

The purpose of the discount rate is to reflect both the time value of money and the investment risk of the project. Traditionally mining type projects use higher than average discount rates, in the 10-15% range. According to the Note for mining and oil and gas companies listed on London Stock Exchange (AIM), 10% discount rate should be applied to the cash flow to estimate net present value of the reserves.

Therefore, WAI has used 10% discount rate for reserve estimation and consequently as a base case for financial results.

Additionally, a sensitivity analysis of NPVs was performed on various discount rates ranging between 8% and 15%.

21.5.1 Cash Flow Model Summary and Results

A post-tax cash flow model has been constructed based on the WAI production schedule developed as of Q1 2017.

All operating costs were derived from first principles, and capital costs being estimated by the Client using actual cost quotations dated end of 2016.
The financial model is presented in real cash flows (as of 2017) with no inflation or cost escalation being taken into account.

The results of the economic analysis presented in this Study indicate that within the given economic environment and based on technical and economic parameters used, the Nasedkino Project has 3 years of payback period (based on debt-free discounted cash flows) and NPV of US$127M at 10% base case discount rate, and project internal rate of return of 82%, using Au price of US$1,100/oz. An upside scenario resulted in US$154M (at 10% discount rate) using US$1,200/oz price.

A summary of the Project Discounted Cash Flow model is given in Table 21.3.

Table 21.3: Project Key Performance Indicators

NPV (100% Equity Based Scenario)		**US$1,100**	**US$1,200**
8%	US$ M	$143	$172
10%	**US$ M**	**$127**	**$154**
12%	US$ M	$114	$138
15%	US$ M	$96	$118
IRR project	%	82%	98%
Payback Period	years	3.00	3.00
NPV (Financed Scenario)			
8%	US$ M	$138	$167
10%	**US$ M**	**$126**	**$153**
12%	US$ M	$116	$140
15%	US$ M	$102	$124

21.6 Project Sensitivity Analysis

A sensitivity analysis was performed on key parameters within the financial model to assess the impact of changes upon Net Present Value of the project (debt-free). These parameters are as follows:

- Mining and Processing Operating Costs;
- Mining and Processing Initial Capital Costs;
- Au Price and RUB/US$ exchange rate; and
- Head Grade and Processing recovery.

In order to examine the sensitivity of the Project Base Case NPV to changing economic and operational conditions, each cost factor was variated within a range of +/- 25%; and head grade and recovery parameters within +/- 12.5%.

The results of the sensitivity analysis for NPV at 10% discount factor (Base case) are shown in Figure 21.2.

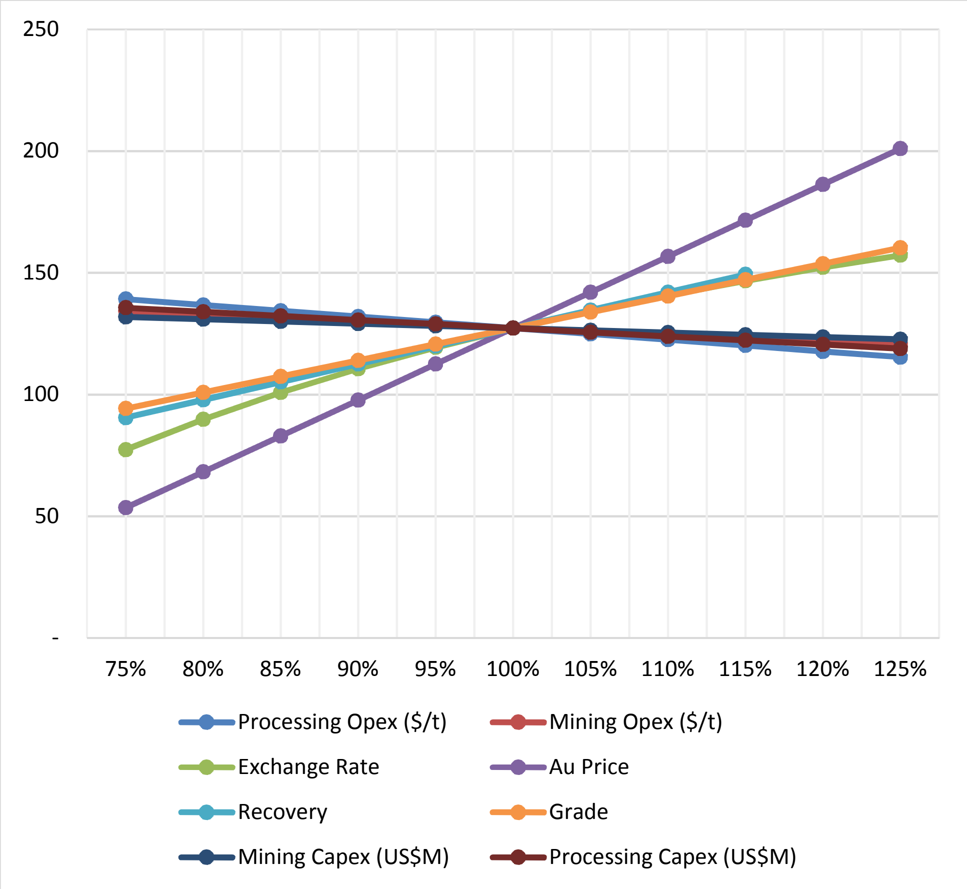


Figure 21.2: Project Sensitivity Analysis (debt-free NPV at 10% discount rate, US$1,100)

Project sensitivity analysis results demonstrates that the Nasedkino Project is mostly sensitive to change in Au price, followed by exchange rate and recovery, Au minable grade, processing operating and capital costs, mining operating and capital costs.

22 ADJACENT PROPERTIES

There are a number of deposits both alluvial and hard rock, located within proximity to the Nasedkino Project. These projects are both operational and undeveloped green field sites, with the main gold production in the Nasedkino neighborhood coming from alluvial mining.

The alluvial deposits within 50km of Nasedkino, operated historically and currently in production are on the rivers Uryum, Ienda, Itaka, Zhelktuga, Gorbiza, Amazar and Amazarkan.

Alexandrovskoe is a hard rock mine that is developed and in operation, Klyuchvskoe was previously operational but is currently not working. Trial mining of oxide ore has been previously carried out at the undeveloped hard rock deposits of Itaka.

Due to the existence of these neighboring alluvial and hard rock mines, there is well established regional mining operations infrastructure, which includes road, railway and HV power lines.

23 OTHER RELEVANT DATA AND INFORMATION

No other relevant data is available for the Nasedkino project.

24 INTERPRETATION AND CONCLUSIONS

24.1 Mineral Resources

An updated Mineral Resource estimate has been prepared for the Nasedkino Gold Project, East Siberia Russian Federation. The Mineral Resources estimate is based on drilling data of acceptable quality from three drilling and sampling programs conducted between 2006 and 2015. All of the drilling was conducted by diamond core drilling.

Mineral Resource estimation involved the usage of drill hole data to construct three dimensional wireframes to define individual vein structures. Gold grades were estimated into a geological block model representing each vein. Grade estimation was carried out by inverse power of distance. Estimated grades were validated globally, locally, and visually prior to tabulation of the Mineral Resources.

As of February 10, 2017 and at a cut-off grade of 0.7g/t Au the total Indicated Mineral Resources for the Nasedkino Gold Project are 10,212Kt with an average grade of 2.25g/t Au and total Inferred Mineral Resources are 3,341Kt with an average grade of 1.82g/t Au.

24.2 Geotechnical

In comparison to the previous studies completed, the WAI assessment is in general agreement to previous pit slope angle recommendations.

Geotechnical laboratory testing was conducted on samples of drill core; however, the testing standards used and locations of samples are not known. In comparison to referenced examples, the test results are at the lower end of expected results indicating weathering or oxidisation of the samples.

Photographs of drill core taken from a site visit conducted by WAI were reviewed and reflect the rock mass quality ratings used in the previous investigation.

Reviewing the rock mass in the core photographs, the drill core is shown to be jointed as well as having numerous rubble sections, indicated brittle faulting. Persuasive jointing indicates that joints are likely to be a dominate structural control on the bench design.

WAI conducted a slope stability review using numerical modelling based on the collected data from the previous investigation and current mine design. The numerical modelling indicates the slope highwalls are stable, but does show, owing to anticipated blasting damage, the immediate rock mass behind the bench faces is likely to be damaged. Owing to multiple interceptions from joint sets noted from the core photos, the bench faces risk unravelling from small blocks formation, and less likely from large block development on whole joint length. It is therefore critical to conduct careful blasting practice at final bench boundaries as well as conduct appropriate scaling which does not damage the integrity of the bench rock mass.

24.3 Project Infrastructure

At the time of reporting no permanent infrastructure has been developed at the site. Currently, there are deforest operations underway. Electric power will be supplied on a temporary basis by a container type Diesel Power Plant until High voltage power lines are constructed. The agreement between Daltsvetmet and Chitaenergo was signed to conduct the technical connection of power at Nasedkino to the local grid, with a anticipated completion date in October 2018. The date is in line with expected production commencement at Nasedkino.

The areas have already been allocated for all infrastructure facilities and at the time of reporting design documentation is being developed.

The sources of water which are available to supply the mine appear to be well understood. A water balance is provided detailing the potential water demands as well as the potential supplies for water.

Technical water will be supplied mainly from a reservoir to be constructed upstream of the tailing storage dam. Details are provided regarding the flow of the stream in which the reservoir will be constructed, the input from precipitation and losses due to evaporation and seepages. The water abstracted from the open pits will be used as technical water if required.

Daltsvetmet has develop a water supply well system in Maliy Uryum river valley, which is located above the mouth of the spring Nasedkino. For 2017, Daltsvetmet plans to conduct further geological explorational works to evaluate and test underground reserves of water.

24.4 Hydrogeology

Both the 2011/2014 and 2016 reports present values for the volumes of water derived from rainwater and groundwater along with predicted pump rates which could deal with this volume of water ingress. The 2016 report goes further and presents additional values of inflow from extreme climatic events including snowmelt and intense rainfall events. Neither report considers high inflows from large fractures or a better connected fracture network and neither report provides contingency pumping measures to deal with increased inflows.

24.5 Processing

The results of the extensive pilot scale and bulk-scale testwork on the four Nasedkino areas using three processing flowsheets - gravity-flotation, gravity-cyanidation and head cyanide leach, established that the following recoveries were obtained:

- Pridolinny: with the gravity-cyanide flowsheet recovery was 88.8% versus 82.3% using the gravity-flotation and 83.04% using the cyanide heap leach flowsheet with differences of 6.57 and 5.80%, respectively;
- Gora 5 area: - recovery was 91.4% using the head cyanide leach flowsheet versus 80.2% using the gravity-flotation and 91.1% using the gravity-cyanide leach flowsheet, with recovery differences of 11.2 and 0.29%, respectively;
- Zhelanny: the recovery was 93.8% using the gravity-cyanide leach flowsheet versus 87.4% using the gravity-flotation and 91.7% using the head cyanide leach flowsheet, with differences of 6.4 and 2.4%, respectively; and
- Pravoberezhny: the recovery was 95.1% with the gravity-cyanide leach flowsheet followed by 90.6% using gravity-flotation and with 94.5% using the head cyanide leach flowsheet, with a differences of 4.5% and 0.58%, respectively.

According to the Russian system, the dressability of the Gora 5, Zhelanny and Pravoberezhny ores were defined as "readily processed", and the Pridolinny ores are defined as "moderately easily processed". It was concluded that the aforementioned classifications do not exclude the possibility of processing them using a single processing flowsheet and therefore they can be referred to as one metallurgical type.

A process design was developed for the treatment of the Nasedkino ores based on treating approximately 1Mtpa as part of the TEO. Although the Nasedkino ore types contain elevated levels of copper, the levels are not considered to be economic and only gold dore will be produced. The gold process recoveries from each ore type are given below.

Summary of Nasedkino Process Route Recoveries (%)			
	Process Route		
Deposit	Head Leach	Gravity - Leach	Gravity - Flotation
Pridolinny	83.0	88.8	82.3
Gora 5	91.4	91.1	80.2
Zhelanny	91.7	93.8	87.4
Pravoberezhny	94.5	95.1	90.6

The proposed processing flowsheet involves:

- Primary crushing to 250mm;
- SAG-Ball mill grinding;
- Gravity processing with the gravity concentrate and tailings leached separately;
- Carbon adsorption and desorption; and

- Smelting the cathode product.

After a detox stage, leach tailings will be thickened, filtered and disposed of as a filter cake. Process plant production mass balances were produced based on the mining schedule and calculated for five gold cut-off grades. The plant uses conventional and well established technologies including gravity and cyanidation which have been thoroughly tested on all four ore sources through extensive programmes of laboratory and pilot plant testwork. A combination of Chinese and Western equipment has been selected in the design study.

The process operating consumable levels were determined from testwork data and experience with similar ores and are typical for the treatment of a moderately hard, non-refractory gold ore.

24.6 Environmental and Social Liabilities & Risks

The submitted OVOS catering for all parts of the mine construction and operation, is currently under final review.

24.7 Financial Analysis

The results of the economic analysis indicate the Nasedkino project has a positive NPV of $127M at 10% base case discount rate (100% equity based scenario), and project internal rate of return of 82%, using a conservative Au price of U$1,100/oz and all-in sustaining cost of US$443/oz.

Project sensitivity analysis results demonstrate that the Nasedkino Project is mostly sensitive to change in Au price, followed by exchange rate and recovery, Au minable grade, processing operating and capital costs, mining operating and capital costs.

25 RECOMMENDATIONS

25.1 Mineral Resources

- Additional duplicate re-assaying including full QAQC insertions for samples from the 2006-2008 drilling campaign primarily assayed at Daltsvetmet, LITsIMS, ZabNII and ALS Chemex (Chita) laboratories should be undertaken. These assays were not included in the current Mineral Resource estimate;
- The use of an umpire laboratory for all future drilling or sampling campaigns should be re-continued; and
- Lithology information from the 2006-2008 drilling campaign should be digitised and included in the drill hole database to further aid geological interpretation;
- Recurring density samples within the density database should be identified and corrected. These density values were not included in the current Mineral Resource estimate. Additional density measurements should be undertaken at Pravoberezhny; and
- The Nasedkino ore reserve statement was based upon the Gold price forecast at the time of carrying out the Nasedkino mine design optimisation of US$1100. If the mine design was carried out using current spot prices of over US$1200, and revised gold forecasting, there would be a significant uplift in the total gold at Nasedkino.

25.2 Geotechnical

In light of the findings, WAI recommend the following be completed for further analysis:

- **Geotechnical laboratory testing programme** – Further testing to expand current strength estimates of intact rock samples from domains identified. A programme with sample position and lithology is recommended.
- **Major Structures** – Data/information on major structures has not been reviewed nor were previous collected structures available in the analysis. A campaign to build a database of major structures in the rock mass is recommended for further analysis and use for open pit operation and design.
- **Detailed Geotechnical Mapping** – Where exposures of rock mass are available geotechnical mapping or remote techniques such as stereographic photos and subsequent analysis is recommended to increase the number of structures recorded, that may otherwise be unable to be mapped.
- **Further Geotechnical Drilling** – Results from any further drilling should be incorporated into a mine site geotechnical model. Further drilling locations within the central western areas should be considered to increase data density.

Any additional geotechnical information, such as mapping, logging, or testing should be used to reduce data gaps, enhance the geotechnical database, update the rock mass structural model, and refine the hydrogeological model. The pit design should be optimised when additional geotechnical information becomes available.

25.3 Hydrogeological and Hydrological

The following recommendations for additional hydrological and hydrogeological studies are made:

- Additional studies confirming the connectivity of the fracture network at Pridolinny with the more developed fracture network to the northwest of the pit;
- Hydrogeological studies should be undertaken at Pravoberezhny to provide information on the groundwater level, hydraulic properties and groundwater chemistry; and
- Studies regarding the connectivity of the fractured bedrock at Pridolinny, and potentially Zhelanny depending on how close the final pit design is to the alluvial deposits, with the alluvial deposits and rivers.

26 REFERENCES

- Hoek, E., 2012. Blast damage factor D. Technical note for RocNews.
- World Gold Council, Outlook 2017 Global economic trends and their impact on gold;
- World Bank Commodity Markets Outlook, January 2017;
- LMBA Forecast 2017;
- Financial model reviewed by WAI: *Nasedkino_N_v9.xlsx;*
- LLC NAEN Consult, 2016. Technical and economic assessment (TEO);
- 2011 hydrogeology report by FGBR Chitinsky CGMS R;
- Audit Report for Nasedkino Deposit and Uryum Exploration Area, SRK Consulting (Russia) Ltd, December 2011;
- NI 43-101 Technical Report for Nasedkino Project, Mogocha, Transbaikal Region, Russian Federation. SRK Consulting. October 2012;
- Resource Estimation and Scoping Study of Nasedkino Gold Deposit Development, Zabaikalsky Krai, Russia. LLC IMC Montan. October 2014; and
- Corbett, G., 2002. Epithermal Gold for Explorationists. *AIG Journal-Applied geoscientific practice and research in Australia*, 26p.

27 CERTIFICATES

CERTIFICATE OF AUTHOR

I, Mark Mounde C.Eng., as an author of this report entitled “NI 43-101 Compliant Mineral Resource and Ore Reserve Estimate for The Nasedkino Project, Russian Federation”, prepared for OOO Dalsvetmet, and dated 10 August 2017, do hereby certify that:

1. I am a Technical Director with Wardell Armstrong International Limited of Wheal Jane, Baldhu, Truro, TR3 6EH, United Kingdom.
2. I am a graduate of the Camborne School of Mines of the United Kingdom in 1993 with a B.Eng. (Hons) in Mining Engineering.
3. I am registered as a Chartered Engineer with Engineering Council UK through the Institute of Material, Minerals and Mining (Reg.# 569902). I have worked as a mining engineer for more than fifteen years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Practice as a mining engineer, production superintendent, mine planning engineer and a consultant in the design, operation and review of mining operations.
    - Review and report, as an employee and as a consultant, on numerous mining operations and projects around the world for due diligence and operational review related to project acquisition and technical report preparation, including:
    - Competent Person’s Report on the reopening of a gold mine in Botswana;
    - Mineral Resource and Ore Reserve estimate of an open pit gold mine in Iran;
    - Mineral Resource and Ore Reserve estimate of an open pit gold mine in Saudi Arabia;
    - Pre-Feasibility Study on an open pit gold deposit in Russia
    - Mineral Resource and Ore Reserve estimate of numerous open pit gold mines in Russia;
    - Technical Assistance on a gold deposit in Armenia.

I have read the definition of "qualified person" set out in National Instrument 43- 101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

1. I visited the Nasedkino Project from 9 August to 11 August 2016.
2. I am responsible for preparation of Sections 28 of the Technical Report.
3. I am independent of the Issuer applying the test set out in Section 1.4 of NI 43- 101.
4. I have had no prior involvement with the property that is the subject of the Technical Report.
5. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
6. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 10th day of August, 2017

(Signed & Sealed)
Mark Mounde, C.Eng.

CERTIFICATE OF AUTHOR

I, Richard John Ellis, BSc, MSc, MCSM, FGS, CGeol, EurGeol, do hereby certify that:

1. I am a Principle Resource Geologist of: Wardell Armstrong International Ltd Wheal Jane, Baldhu, Truro, TR3 6EH, United Kingdom;
2. I graduated with a Bachelor of Science Degree in Geology from the University of Bristol (UK) in 2001 and a Master of Science Degree in Mining Geology from the Camborne School of Mines (UK) in 2003;
3. I am a Fellow and Chartered Geologist of the Geological Society of London (Membership No. 1013201) and member of the European Federation of Geologists;
4. I have practiced my profession continuously for the last 13 years in a variety of countries and geological environments and have prepared Mineral Resource estimates for epithermal gold deposits for over 5 years;
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that I am a “qualified person” for the purposes of NI 43-101;
6. I last visited the property from August 9 to August 11, 2016;
7. I am responsible for the preparation of sections 1. Summary; 2. Introduction; 3. Reliance on Other Experts; 4. Property Description and Location; 5. Accessibility, Climate, Local Resources, Infrastructure and Physiography; 6. History; 7. Geological Setting and Mineralisation; 8. Deposit Type; 9. Exploration; 10. Drilling; 11. Sample Preparation, Analyses and Security; 12. Data Verification; 14. Mineral Resource Estimates; 23. Adjacent Properties; 24. Other Relevant Data and Information; 25. Interpretation and Conclusions; 26. Recommendations; 27. References
8. I am independent of the issuer, LLC Daltsvetmet as defined by NI 43-101;
9. I have read the Instrument NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and;
10. As of the date of this certificate and to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 10th day of August, 2017

[signed]
Name **R J Ellis BSc, MSc, MCSM, FGS, CGeol, EurGeol**

APPENDIX 1: JORC Code, 2012 Edition – 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.*	• Sampling was carried out using diamond core drilling from surface obtained during three exploration programmes carried out from 2006-2008, 2010-2012 and 2015. • Additional sampling by surface trench sampling was also undertaken but this data was not included in the Mineral Resource estimate. • In 2006 sample preparation was carried out at LITsIMS laboratory located in Chita and Irgiredmet laboratory located in Irkutsk. From 2007 to 2008 sample preparation was carried out by Vostokgeology and ALS Chemex laboratory located in Chita. From 2010 to 2012 sample preparation was carried out by Daltsvetmet using a Rocklabs preparation facility located on site at Nasedkino. During 2010 to 2012 additional sample preparation was also undertaken at ALS Chemex (Chita). During 2015 all sample preparation was undertaken by ALS Chemex (Chita). • Drill core is sampled on 1m intervals from which half core sub-samples were pulverized to produce a final 10g sub-sample for AAS analysis. • During 2006-2008 various laboratories were used for assaying and included Irgiredmet, Daltsvetmet, ALS Chemex, LITsIMS and ZabNII. Only the assays derived from Irgiredemt laboratory were used in the Mineral Resource Estimate. During 2010-2012 and 2015 all assaying was undertaken at ALS Chemex (Chita).
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).*	• Currently surface diamond drilling is carried out producing HQ (63.5 mm) and NQ (47.6 mm) diameter drill core • Core is not orientated.
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.*	• Drill core sample recovery is assessed by gravimetric and linear measurements. • Core recovery is generally good and is typically greater than 90 %. • Areas of reduced core recovery does not correlate with reduced gold grade.

Criteria	JORC Code explanation	Commentary
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 is logged on site by company geological personnel. • Core is geologically logged including a description of lithology, structure, texture, colour, secondary alteration, mineralisation type, vein infill and contact type. • All core is currently logged to the above criteria. • Core is photographed during the logging procedure. • Core is logged manually before transfer to an electronic system using Excel spreadsheets.
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.*	• Core samples are sawn with diamond saws with half core being sent for sample preparation and half retained. Where whole core samples were sent for sample preparation (as during the 2015 drilling campaign) no half core is available. Photographic evidence and laboratory assay certificates are however available. • Core samples pass through a multi-stage crush process to 1mm to provide a 600 g sub sample which is then pulverized to 0.074 mm. From this 150 g is sent for primary analysis and 450 g is retained as a duplicate by the geology department. • High grade zones at Zhelanny contain coarse gold. The appropriateness of the procedures with respect to sample size on coarse gold was evaluated by WAI using analysis of duplicates. No systematic bias was identified.
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, 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.*	• During the 2006-2008 campaign assaying was carried out at Irgiredmet, LITsIMS, Daltsvetmet, ZabNII and ALS Chemex (Chita) laboratories. Only pulp duplicate samples were submitted for QAQC during this time. However a re-assaying programme was undertaken in 2010 using pulp duplicates from the 2006-2008 campaign with analysis undertaken at at ALS Chemex (Chita). The re-assaying programme included pulp duplicate, CRM's and blank QAQC insertions. Based on the results of the re-assaying programme only assays derived from the Irgiredmet laboratory during the 2006-2008 campaign have been used in the Mineral Resource estimate. During the 2010-2012 and 2015 campaigns all primary assaying was carried out by ALS Chemex (Chita) with acceptable QAQC. • Assay data quality is determined in the first instance through a series of

Criteria	JORC Code explanation	Commentary
		standards blanks and duplicate samples submitted in to the main sample stream by the geological department. Duplicate samples are pulp duplicates. Standard samples are commercially sourced certified reference materials from Geostats and OREAS at a range of Au grades. Blank samples are sourced from locally occurring barren acid lava, felsite, quartz porphyry, tuffs and sandstone. • Current control sample protocol for the geological department is for a CRM sample for every 20 routine assays, a coarse blank and a fine blank for every 100 routine assays and a pulp duplicate for every 20-30 routine assays. Additional external pulp duplicate analysis was undertaken at Alex Stewart (Moscow) during the 2010-2012 campaign. • Analysis of pulp duplicate QAQC results show a moderate level of precision and accuracy and reflect the nugget style of mineralisation. Analysis of CRM and blanks showed relatively few failures.
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.*	• WAI inspected core at the logging facilities at Nasedkino. Where whole core samples were sent for sample preparation (as during the 2015 drilling campaign) no half core is available. Photographic evidence and laboratory assay certificates are however available are were relied on in these instances. • Logging data in the first instance is recorded by hand to form documentation for each hole that includes collar and down hole survey information and assay information once available. This information is transferred to an electronic database. • 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.*	• It is reported that the topographic survey is based on the local coordinate system and using the Baltic elevation system. • Drill hole collars were surveyed using a Sokkia SET 510 total station. A comparison of collar surveys with topographic survey shows a good match with only minor errors identified. • Down hole surveys were undertaken on all drill holes exceeding 100 m depth. Drill holes were surveyed using a Tropari-like survey instrument (Gyromagnetic Inclinometer AEM-36) with readings taken every 20m. All

Criteria	JORC Code explanation	Commentary
		down hole surveys were carried out by drilling contractor personnel. • Resource estimation has been carried out in a local grid system where “north” coincides with true north.
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 sufficient to establish geological and mineralisation continuity appropriate for the reporting of Mineral Resources. Mineral Resources are classified as Indicated and Inferred following the guidelines of the JORC Code (2012). • Sample compositing was carried out as part of the Mineral resource estimation process.
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.*	• Drill sections are orientated at 240° and are orientated perpendicular to the general strike of the deposits. The deposits dip sub-vertically to the northeast. Only inclined drill holes were used in the Mineral Resource estimate and dip at -60° to intersect perpendicular to the mineralisation. • All vertical drill holes were removed from the database prior to resource estimation to prevent sampling bias resulting from drilling orientation.
Sample security	• *The measures taken to ensure sample security.*	• Drill core is transported from the drill site to logging facilities in sealed core boxes by the supervising driller and samples are transported from here to sample preparation facilities and the laboratory by company geological personnel. • At each stage appropriate documentation accompanies sample batches identifying sample numbers and weights. Sample tickets are included with sub-samples of core. Duplicate samples are stored in appropriately labelled and sealed envelopes. • Core storage areas are lockable and are located at the Nasedkino exploration site.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Comprehensive quality control sample programmes are implemented by the geological and laboratory staff. • WAI has reviewed documentation summarising exploration works and has reviewed internal and external QAQC results.

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 Nasedkino Gold Project comprises two licences: • Mining and exploration licence ЧИТ 01663 БЭ (dated December 26, 2005). The licence is valid until December 1, 2015 and covers an area of 10.9 km^2 and includes the Pridolinny and Gora 5 deposits. • Mining and exploration licence ЧИТ 02652 БР (dated April 1, 2016). The licence is valid until December 31, 2036 and covers an area of 19.6 km^2 and includes the Zhelanny and Pravoberezhny deposits.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• Exploration drilling has been carried out by Daltsvetmet at Nasedkino since 2006. A total of 352 drill holes for 37,939 m were drilled during the 2006-2008 drilling campaign. A total of 431 drill holes for 52,634 m were drilled during the 2010 – 2012 drilling campaign. A total of 27 drill holes for 2,255 m were drilled during the 2015 drilling campaign. A
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Nasedkino Gold Project has the characteristics of a low-sulphidation epithermal deposit • The Project is located along the south-east of the Siberian Craton and is hosted by a middle Palaeozoic accretionary subduction complex. The host rocks comprise Archaen ultra-metamorphic formations including granulites with skarn mineral assemblages and gneisses that have been folded into isoclinal north-west trending folds. • Typical lithologys include pyroxene-garnetiferous granulites (skarnoid) and quartz-biotite-plagioclase gneisses often containing sillimanite, hypersthene, graphite, high-aluminimum crystalline schists and amphibolites. Of the two host lithologys the most significant mineralisation is hosted by the skarnoids. • The host lithologys have been intruded by Late Jurassic intrusions of medium acidic composition and are the source of the mineralisation. • Fault zones acted as feeders for the mineralised hydrothermal fluids while higher permeability metamorphic strata comprised suitable fluid migration pathways. • The mineralised zones occur as lenticular zones of hydrothermal alteration and lie in conformity with the metamorphic strata which strike north-west

Criteria	JORC Code explanation	Commentary
		(330-345°) and dip steeply (70-80°) to the north-east. • A series of cross cutting late stage faults off-set the mineralised zones.
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.*	• Number of drill holes used – 511 • East collar ranges – 647,199.6 m to 650,886.1 m • North collar ranges – 278,579.8 m to 280,906.1 m • Collar elevation ranges – 866.5 m to 1,093.7 m • Azimuth ranges – 228° to 261° • Dip ranges – -45° to -65° • Length of holes – 23 m to 339 m
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 procedures and levels are outlined in the main body of the report. • During grade estimation limits to the search ellipses were also applied to drill hole 607 in Domain 101 at Zhelanny and drill hole 1033 in Domain 201 at Zhellany to further limit the influence of localised extreme grades in these drill holes. • 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	• *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.*	• Drill sections are orientated at 240° and are orientated perpendicular to the general strike of the deposits. The deposits dip sub-vertically to the northeast. Only inclined drill holes were used in the Mineral Resource estimate and dip at -60° to intersect perpendicular to the mineralisation.

Criteria	JORC Code explanation	Commentary
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').*	• All vertical drill holes were removed from the database prior to resource estimation to prevent sampling bias resulting from drilling orientation.
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 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.*	• Statistics of drill hole grades used during the Mineral Resource estimate are contained in the main body of the report
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. • Samples submitted for density measurements were based on lithology. Density measurements were taken by measuring the sample weight in air then suspending the sample in water and measuring the weight again. A total of 1,683 density measurements are contained within the database.
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.*	• WAI is unaware of any additional planned exploration activities at Nasedkino.

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 Excel spreadsheets. • Data held includes; collar location, downhole surveys, assay information, duplicate sample, standards and blank sample results and geological logging. • Geological logging is initially completed on paper but a standard logging

Criteria	JORC Code explanation	Commentary
		template is used and this data is transcribed to the electronic database. • Data verification and database validation was undertaken by WAI the details of which are contained in the main body of the report.
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.*	• The Competent Person visited the site between 9th and 11th August 2016. • The site visit included inspection of the exploration camp, logging and core storage facilities, site walkover and discussions with geological staff.
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.*	• The confidence in the geological interpretation is deemed reasonable. • The geological setting is a low-sulphidation epithermal deposit. • Geological logging has been carried out from drill core observations. • Typical lithologys include pyroxene-garnetiferous granulites (skarnoid) and quartz-biotite-plagioclase gneisses often containing sillimanite, hypersthene, graphite, high-aluminimum crystalline schists and amphibolites. Of the two host lithologys the most significant mineralisation is hosted by the skarnoids. • Geological logging was used to define the main geological controlling structures (skarn package or gneiss hosted) within the overall resource models.
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 deposits are located on two parallel mineralised corridors which extend for more then 2.5 km in a northwest direction. Zhelanny and Pravoberezhny are located on the western most of the mineralised corridors. Pridolinnny and Gora 5 are located on a mineralised corridor located 700 m to the east of this. • Pridolinny mineralisation has been modelled with a strike length of 1,280 m, a plan width of 340 m and to a depth of 260 m from surface. • Gora 5 mineralisation has been modelled with a strike length of 850 m, a plan width of 110 m and to a depth of 230 m from surface. • Zhelanny mineralisation has been modelled with a strike length of 920 m, a plan width of 120 m and to a depth of 150 m from surface. • Pravoberezhny mineralisation has been modelled with a strike length of 1,230 m, a plan width of 260 m and to a depth of 220 m from surface.

Criteria	JORC Code explanation	Commentary
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.* • *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.*	• Inverse Distance Weighting (to power 3) was used as the main estimation technique. • Domains: Multiple domains were created. Mineralised zones were defined based on a cut-off grade of 0.7g/t Au. High grade zones at Zhelanny were defined based on a cut-off grade of 7g/t Au. • Grade capping: Grade capping was used for all variables on a domain by domain basis where outlier grades were identified. • Composites: A 1m composite length was chosen to ensure consistent sample support during estimation. • Variography: A variographic study was undertaken and is detailed in the main body of the report. • Estimation: Estimation was carried out using Inverse Distance Weighting (to power 3) as the primary method. Ordinary Kriging and Nearest Neighbour estimates were carried out for validation purposes. • A block size of 5m (X) x 5m (Y) x 5m (Z) was used in this model. Estimation was carried out in to parent cells only. • The block model was verified by comparing drill hole assays with modelled values visually and statistically by zone. Domain mean grade checks and grade profile plots were also constructed to compare modelled grades and input composite grades. • No estimation of deleterious components was carried out.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• Tonnage is measured on a dry basis using bulk in-situ density • No moisture content has been measured.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• The Mineral Resource estimate is restricted to either material falling within an NPV Scheduler optimised pit shell and based on a cut-off grade of 0.7 g/t Au.
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*	• The project is deemed to be appropriate to being mined by standard open pit mining operations. • Reporting of Mineral Resources suitable for open pit extraction were limited by the creation of an optimised open pit shell in NPV Scheduler. The pit optimisation parameters are contained in the main body of the report.

Criteria	JORC Code explanation	Commentary
	rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	
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, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• 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.
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.*	• Appropriate environmental studies need to be completed as part of any open pit mining Feasibility Study to determine the impact of mining operations. • The area close to the Nasedkino Gold Project will provide sufficient space for waste and process residue.
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.*	• Samples submitted for density measurements were based on lithology. Density measurements were taken by measuring the sample weight in air then suspending the sample in water and measuring the weight again. A total of 1,683 density measurements are contained within the database. • Density was assigned to the block model during the Mineral Resource estimation by applying average values for the major lithology type. • A description of the density values applied in the Mineral Resource estimation is contained in the main body of the report.
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*	• Mineral Resource classification was made following the guidelines of the JORC Code (2012) to Indicated and Inferred status. • Classification was based on sample density, confidence in the geological and mineralisation continuity and reliability of the exploration database used as the basis of Mineral Resource estimation.

Criteria	JORC Code explanation	Commentary
	distribution of the data). • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• No Measured Mineral Resources were classified at the Nasedkino Gold Project. Although some of the drilling at Pridolinny is based on a 10m x 10m grid spacing this drilling has predominantly been derived from the 2006-2008 drilling campaign which was subject to check assaying in 2012. The results, while acceptable, do however indicate a relatively high level of grade variability and as such no Measured Mineral Resources were assigned to this drilling. This drilling was subsequently assigned a maximum classification of Indicated Mineral Resources only. • Indicated Mineral Resources were defined on a zone by zone basis and were classified using limits around contiguous areas of blocks that were generally encountered during the first search of the grade estimation up to a 50m x 50m drill grid spacing. A minimum number of three drill holes were required. The spatial location and continuity of the mineralised zone was also considered when assigning Indicated Mineral Resources. Areas covered by a drill grid spacing of 10m x 10m were also classified as Indicated Mineral Resources due to the issues addressed above. • Inferred Mineral Resources were classified based on the remaining blocks defined within the mineralised zone wireframes that were generally encountered during the second or third search of the grade estimation. Typically Inferred Mineral Resources correspond to areas of 100m drilling grid spacing or isolated zones with limited drilling and/or limited continuity of mineralisation. • The Mineral Resource estimate classification reflects the Competent Person's view of the Nasedkino Gold Project. • Mineral Resources 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.
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 or any previous Mineral Resource estimates.
Discussion of relative	• *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*	• 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).

Criteria	JORC Code explanation	Commentary
accuracy/ confidence	*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.*	• Validation procedures carried out on the final block models against input sample data show good correlation. • The statement relates to global estimates of tonnes and grade. • The Nasedkino Gold Project is a greenfield site and as such no comparison with production data is possible.

Section 4 – Estimation and Reporting of Ore Reserves

Criteria	JORC Code explanation	Commentary
Mineral Resource Estimate for Conversion to Ore Reserves	• *Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.* • *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	• The Ore Reserve Estimate is based upon the Mineral Resource Estimate carried out by WAI with an effective date of 16 December 2016 • The Mineral Resources are reported inclusive of Ore Reserves
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.*	• As part of the Pre-Feasibility Study works a personal inspection of the project was carried out by WAI between 9th-11th August 2016, covering aspects related to access and infrastructure, geology, exploration, QA/QC, mineralogy, sample preparation, laboratory testwork, and tailings and waste disposal. • The following personnel from WAI carried out the site visit: Mr. Mark Moude (Technical Director) Competent Person for Ore Reserves; Mr Richard Ellis (Principal Geologist) Competent Person for Mineral Resources
Study Status	• *The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.* • *The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such*	• Pre-Feasibility Study level

Criteria	JORC Code explanation	Commentary
	studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	
Cut-off Parameters	• *The basis of the cut-off grade(s) or quality parameters applied.*	• For the open pit operation, known costs were entered into NPV Scheduler software and the economic CoG derived.
Mining Factors and Assumptions	• *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).* • *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.* • *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.* • *The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).* • *The mining dilution factors used.* • *The mining recovery factors used.* • *Any minimum mining widths used.* • *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.* • *The infrastructure requirements of the selected mining methods.*	• Ore reserves have been calculated by evaluating mining designs for the proposed open pit. • The open pits is currently greenfield, and thus a ramp up to production of 1.1Mtpa (total across the four deposits) has been considered. • Inferred material has not been included with the Ore Reserve Estimate and has been treated as waste. • Mining Losses have been applied as follows: Gora 5 (1.81%), Pridolinny (3.44%), Zhelanny and Pravoberazhny (2.5%) • Dilution has been applied as follows: Gora 5 (2.4%), Pridolinny (5.62%), Zhelanny and Pravoberazhny (10%) • A minimum mining width of 40m has been assumed. • Standardised haul roads, declines, mine drainage, ventilation, electrical power and fuel supply systems have been incorporated in the study.
Metallurgical factors/assumptions	• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* • *Whether the metallurgical process is well-tested technology or novel in nature.* • *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.* • *Any assumptions or allowances made for deleterious elements.* • *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.*	• The results of extensive pilot scale and bulk-scale testwork using three processing flowsheets - gravity-flotation, gravity-cyanidation and head cyanide leach – are as follows: • Pridolinny: with the gravity-cyanide flowsheet recovery was 88.8% versus 82.3% using the gravity-flotation and 83.04% using the cyanide head leach; • Gora 5 area: - recovery was 91.4% using the head cyanide leach flowsheet versus 80.2% using the gravity-flotation and 91.1% using the gravity-cyanide leach;

Criteria	JORC Code explanation	Commentary
	• *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	• Zhelanny: the recovery was 93.8% using the gravity-cyanide leach flowsheet versus 87.4% using the gravity-flotation and 91.7% using the head cyanide leach; • Pravoberezhny: the recovery was 95.1% with the gravity-cyanide leach flowsheet followed by 90.6% using gravity-flotation and with 94.5% using the head cyanide leach.
Environmental	• *The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	• An OVOS will need to be completed which caters for all parts of the mine construction and operation. • The Nasedkino project has licenses in place for certain operations involving pre mine development. However, some of the licenses it holds will need to be extended for the life of the mine and levels applicable to the construction and operation of the mine.
Infrastructure	• *The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.*	• The project is currently greenfield and as such all operational and other necessary infrastructure required for the processing and mining of the Open Pits will require installation. Infrastructure costs and requirements have been estimated and included within the study.
Costs	• *The derivation of, or assumptions made, regarding projected capital The derivation of, or assumptions made, regarding projected capital costs in the study.* • *The methodology used to estimate operating costs.* • *Allowances made for the content of deleterious elements.* • *The source of exchange rates used in the study.* • *Derivation of transportation charges.* • *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.* • *The allowances made for royalties payable, both Government and private.*	• The project operating costs have been estimated by the client and reviewed by WAI engineers. Operating costs were derived from the first principles based on development of the open pit mine with annual production capacity of 1.1Mtpa. • Capital costs were derived from first principles based on the minable volumes and actual cost of equipment dated November 2015, exclusive of VAT • No contingency has been provided for within these estimates. • Taxes and royalties have been applied as appropriate at the levels outlined by the state.
Revenue Factors	• *The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* • *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	• Financial analysis; and mine optimisation works use a gold price of US$1,100/oz and are constant for the life of the project. • All other revenue factors and inputs have been derived from previous studies for the deposit.

Criteria	JORC Code explanation	Commentary
Market Assessment	• *The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* • *A customer and competitor analysis along with the identification of likely market windows for the product.* • *Price and volume forecasts and the basis for these forecasts.* • *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	• The supply and demand situation for gold is affected by a wide range of factors, and consumption changes with economic development and circumstances. Dore bars are securely shipped and delivered to a smelter in accordance with demand and market conditions on an ongoing basis.
Economic	• *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* • *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	• Discount rate has been applied at 10% for both the determination of NPV in the Financial Analysis, and as an input to open pit shell optimisations. • Price inflation has not been considered • A sensitivity analysis has been carried out considering variations to Process Capex and Opex, Exchange Rate, Process Recovery, Mining Capex and Opex, Au Price and Grade.
Social	• *The status of agreements with key stakeholders and matters leading to social licence to operate.*	• Mangazeya has an agreement with the local authority regarding the implementation of the Nasedkino project. The agreement has allowed involvement of local specialists in the production operations of Nasedkino. • No information regarding stakeholder dialogue and community development was provided to WAI for review. To comply with international requirements a stakeholder engagement plan and grievance mechanism should be developed for the project. Further it is recommended that any social initiatives and community development opportunities are combined into a community development plan to comply with international best practice.
Classification	• *The basis for the classification of the Ore Reserves into varying confidence categories.* • *Whether the result appropriately reflects the Competent Person's view of the deposit.* • *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	• Ore Reserves have been presented at the Proved and Probable categories, based upon Measured and Indicated Mineral Resources. • The Proven Ore Reserve has been converted from Measured Mineral Resources • The Probable Ore Reserve has been converted from Indicated Mineral Resources • The Competent Person believes the classification of the Mineral Resource, and conversion to Ore Reserves to be appropriate.

Criteria	JORC Code explanation	Commentary
Audits & Reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	• The Ore Reserve has been Peer Reviewed internally and is in line with current industry standards for a PFS.
Discussion of Accuracy/Confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve 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 reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which 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.* • *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• The mine designs, schedule and financial model on which the Ore Reserve is based has been completed to PFS standard with a corresponding level of confidence • All modifying factors have been applied to design the open pit mining blocks on a project-wide scale.

APPENDIX 2: Raw Drill Hole Data Statistical Analysis

Pridolinny - Au Statistical Analysis for Raw Drill Holes by Domain

Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	380	0.01	20.65	1.88	4.23	2.06	1.09
2	355	0.01	49.80	1.93	12.92	3.59	1.86
3	332	0.09	74.90	2.54	25.71	5.07	2.00
4	266	0.03	14.35	1.58	2.37	1.54	0.98
5	285	0.10	37.70	2.20	8.49	2.91	1.32
6	273	0.03	75.80	2.32	25.68	5.07	2.18
7	203	0.05	13.48	1.55	2.13	1.46	0.94
8	117	0.12	16.60	2.03	6.50	2.55	1.25
9	61	0.03	17.75	1.50	5.01	2.24	1.49
10	59	0.22	9.10	1.67	2.01	1.42	0.85
11	49	0.02	5.86	1.49	1.63	1.28	0.86
12	48	0.28	10.30	1.43	2.19	1.48	1.04
13	48	0.12	12.85	2.02	4.31	2.08	1.03
14	44	0.43	14.70	1.98	5.84	2.42	1.22
15	39	0.05	5.00	1.80	1.32	1.15	0.64
16	39	0.56	6.00	1.54	1.22	1.10	0.72
17	29	0.10	22.00	2.90	23.63	4.86	1.68
18	27	0.08	4.83	1.26	0.83	0.91	0.72
19	25	0.30	3.71	1.21	0.58	0.76	0.63
20	23	0.28	2.51	1.22	0.37	0.61	0.50
21	22	0.37	8.84	1.44	3.08	1.76	1.22
22	22	0.12	4.96	1.28	1.17	1.08	0.85
23	20	0.26	4.80	1.88	1.44	1.20	0.64
24	18	0.28	6.65	1.56	2.01	1.42	0.91
25	17	0.73	11.67	1.91	6.18	2.49	1.30
26	17	0.56	11.05	3.11	9.35	3.06	0.98
27	16	0.60	3.91	1.48	0.90	0.95	0.64
28	16	0.75	203.50	14.72	2379.86	48.78	3.31
29	16	0.15	3.73	1.38	0.84	0.92	0.67
30	15	0.15	2.17	1.05	0.39	0.62	0.59
31	14	0.63	3.14	1.23	0.44	0.66	0.54
32	14	0.76	3.14	1.54	0.49	0.70	0.45
33	13	0.14	2.98	1.23	0.70	0.84	0.68
34	12	0.12	1.72	0.96	0.21	0.46	0.48
35	12	0.08	9.70	1.78	6.22	2.49	1.40
36	11	0.41	2.58	1.19	0.40	0.63	0.53
37	11	0.22	2.47	1.40	0.39	0.62	0.44
38	10	0.92	1.44	1.14	0.03	0.18	0.16
39	9	0.43	2.48	1.16	0.46	0.68	0.59
40	9	0.54	3.33	1.26	0.64	0.80	0.63
41	8	0.52	1.86	1.03	0.16	0.40	0.39
42	8	0.76	2.00	1.32	0.13	0.36	0.27
43	8	0.74	2.58	1.34	0.34	0.58	0.44
44	7	0.70	1.83	1.20	0.17	0.41	0.34
45	7	0.81	4.10	1.89	1.68	1.30	0.69
46	6	0.93	2.39	1.39	0.24	0.49	0.35
47	6	0.20	1.92	1.11	0.28	0.53	0.48
48	5	0.75	4.40	2.55	1.37	1.17	0.46
49	5	0.88	2.31	1.70	0.28	0.53	0.31
50	5	0.62	1.78	1.10	0.19	0.44	0.40
51	5	1.23	2.00	1.58	0.06	0.25	0.16
52	5	0.07	2.16	0.85	0.52	0.72	0.84
53	5	0.65	2.46	1.45	0.43	0.65	0.45
54	5	0.66	7.20	2.71	6.35	2.52	0.93
55	5	0.64	1.26	0.85	0.05	0.23	0.27
56	4	0.97	1.85	1.42	0.10	0.32	0.22
57	4	0.46	1.38	0.92	0.13	0.36	0.39
58	4	0.72	1.46	1.19	0.09	0.30	0.25
59	4	1.03	1.34	1.17	0.01	0.12	0.10
60	4	0.71	1.04	0.88	0.01	0.12	0.14
61	4	1.60	2.33	2.00	0.11	0.32	0.16
62	4	1.74	2.40	2.09	0.08	0.29	0.14
63	4	0.84	9.90	3.68	13.29	3.65	0.99

64	4	0.69	1.40	1.10	0.07	0.27	0.24
65	4	1.40	26.80	8.15	116.15	10.78	1.32
66	4	0.78	3.34	1.46	1.18	1.09	0.75
67	4	0.68	1.22	1.01	0.04	0.21	0.21
68	4	0.46	1.21	0.89	0.07	0.27	0.31
69	4	0.93	2.24	1.42	0.25	0.50	0.35
70	4	1.04	1.82	1.29	0.10	0.32	0.24
71	4	0.77	1.58	1.18	0.10	0.31	0.27
72	4	0.69	6.60	3.12	5.76	2.40	0.77
73	3	0.74	1.39	1.10	0.07	0.27	0.25
74	3	0.83	3.53	1.79	1.51	1.23	0.69
75	3	0.73	0.83	0.79	0.00	0.04	0.05
76	3	1.46	6.06	3.87	3.55	1.88	0.49
77	3	1.33	3.26	2.04	0.75	0.87	0.43
78	3	1.45	1.55	1.50	0.00	0.04	0.03
79	3	0.90	18.50	7.01	66.13	8.13	1.16
80	3	0.60	1.62	1.25	0.21	0.46	0.37
81	3	3.09	9.86	5.42	9.88	3.14	0.58
82	3	0.72	3.80	2.12	1.62	1.27	0.60
83	3	1.12	1.52	1.35	0.03	0.17	0.12
84	2	1.22	1.61	1.42	0.04	0.20	0.14
85	2	1.07	1.17	1.12	0.00	0.05	0.04
86	2	1.58	2.11	1.85	0.07	0.27	0.14
87	2	0.73	0.77	0.75	0.00	0.02	0.03
88	2	0.78	0.82	0.80	0.00	0.02	0.03
89	2	2.25	2.81	2.53	0.08	0.28	0.11
90	2	0.72	0.95	0.84	0.01	0.12	0.14
91	2	0.80	1.77	1.29	0.24	0.49	0.38
92	2	1.63	2.36	2.00	0.13	0.37	0.18
93	2	2.64	2.89	2.77	0.02	0.13	0.05
94	2	0.91	2.56	1.74	0.68	0.83	0.48
95	2	1.27	1.51	1.39	0.01	0.12	0.09
96	2	1.92	2.54	2.23	0.10	0.31	0.14
97	2	5.90	17.80	11.85	35.40	5.95	0.50
98	2	0.81	1.46	1.14	0.11	0.33	0.29
99	2	0.74	1.60	1.17	0.18	0.43	0.37
100	2	0.83	0.86	0.85	0.00	0.02	0.02
101	2	0.79	1.75	1.27	0.23	0.48	0.38
102	2	0.81	0.81	0.81	-	-	-
103	2	0.73	1.93	1.33	0.36	0.60	0.45
104	2	1.06	1.69	1.38	0.10	0.32	0.23
105	2	2.74	2.88	2.81	0.00	0.07	0.02
106	2	3.04	3.22	3.13	0.01	0.09	0.03
107	2	0.76	0.93	0.85	0.01	0.09	0.10
108	2	0.76	0.85	0.81	0.00	0.05	0.06
109	2	1.11	3.28	2.20	1.18	1.09	0.49
110	2	1.57	4.05	2.81	1.54	1.24	0.44
111	2	0.81	1.60	1.21	0.16	0.40	0.33
112	2	0.87	2.05	1.46	0.35	0.59	0.40
113	2	0.70	0.93	0.82	0.01	0.12	0.14
114	2	0.77	0.82	0.80	0.00	0.03	0.03
115	2	0.79	0.82	0.81	0.00	0.02	0.02
116	2	1.10	1.12	1.11	0.00	0.01	0.01
117	2	0.70	4.22	2.46	3.10	1.76	0.72
118	2	0.75	1.02	0.89	0.02	0.14	0.15
119	2	1.36	1.70	1.53	0.03	0.17	0.11
120	2	0.83	0.86	0.85	0.00	0.02	0.02
121	1	4.64	4.64	4.64	-	-	-
122	1	1.82	1.82	1.82	-	-	-
123	1	6.51	6.51	6.51	-	-	-
124	1	1.52	1.52	1.52	-	-	-
125	1	1.87	1.87	1.87	-	-	-
126	1	2.18	2.18	2.18	-	-	-
127	1	1.85	1.85	1.85	-	-	-
128	1	1.64	1.64	1.64	-	-	-
129	1	4.04	4.04	4.04	-	-	-
130	1	1.58	1.58	1.58	-	-	-
131	1	2.30	2.30	2.30	-	-	-

132	1	1.57	1.57	1.57	-	-	-
133	1	1.59	1.59	1.59	-	-	-
134	1	3.42	3.42	3.42	-	-	-
135	1	17.30	17.30	17.30	-	-	-
136	1	4.29	4.29	4.29	-	-	-
137	1	1.50	1.50	1.50	-	-	-
138	1	1.66	1.66	1.66	-	-	-
139	1	1.55	1.55	1.55	-	-	-
140	1	1.31	1.31	1.31	-	-	-
141	1	4.25	4.25	4.25	-	-	-
142	36	0.47	4.08	1.27	0.53	0.73	0.57

Gora5 - Au Statistical Analysis for Raw Drill Holes by Domain

Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	297	0.16	48.90	2.68	18.02	4.25	1.58
2	274	0.06	88.20	2.71	55.72	7.46	2.76
3	54	0.51	6.98	1.87	1.34	1.16	0.62
4	31	0.08	9.05	1.94	2.68	1.64	0.84
5	26	0.16	3.28	1.21	0.52	0.72	0.60
6	26	0.01	4.86	1.36	1.25	1.12	0.82
7	24	0.24	20.40	3.22	21.46	4.63	1.44
8	21	0.30	14.13	3.17	10.78	3.28	1.03
9	21	0.36	2.38	1.14	0.27	0.52	0.45
10	12	0.73	3.54	1.65	0.59	0.77	0.47
11	10	0.79	1.85	1.28	0.15	0.38	0.30
12	5	0.13	2.65	1.19	0.84	0.92	0.77
13	4	0.72	1.44	0.99	0.08	0.29	0.29
14	3	0.82	1.85	1.22	0.20	0.45	0.37
15	3	1.49	4.61	2.72	1.84	1.36	0.50
16	2	0.70	1.66	1.18	0.23	0.48	0.41
17	2	1.22	1.29	1.26	0.00	0.04	0.03
18	2	1.55	2.27	1.91	0.13	0.36	0.19
19	2	1.73	4.00	2.87	1.29	1.14	0.40
20	2	0.70	1.37	1.04	0.11	0.34	0.32
21	2	0.71	0.97	0.84	0.02	0.13	0.15
22	2	0.74	1.17	0.96	0.05	0.22	0.23
23	2	0.96	19.37	10.17	84.73	9.21	0.91
24	2	0.94	1.52	1.23	0.08	0.29	0.24
25	2	0.84	3.09	1.97	1.27	1.13	0.57
26	1	2.28	2.28	2.28	-	-	-
27	1	2.36	2.36	2.36	-	-	-
28	1	1.50	1.50	1.50	-	-	-
29	1	2.08	2.08	2.08	-	-	-

Zhelanny - Au Statistical Analysis for Raw Drill Holes by Domain

Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	137	0.04	16.50	2.42	5.42	2.33	0.96
2	212	0.02	7.51	1.98	2.05	1.43	0.72
3	146	0.08	46.40	2.67	23.94	4.89	1.83
4	31	0.30	2.85	1.22	0.37	0.61	0.50
5	9	0.71	2.11	1.27	0.27	0.52	0.41
6	8	0.84	3.94	1.82	1.08	1.04	0.57
7	7	0.17	4.95	2.18	2.21	1.49	0.68
8	6	0.77	4.61	1.94	1.70	1.30	0.67
9	2	1.79	2.54	2.17	0.14	0.38	0.17
10	2	1.42	3.82	2.62	1.44	1.20	0.46
11	2	1.92	2.54	2.23	0.10	0.31	0.14
12	1	2.29	2.29	2.29	-	-	-
13	1	3.75	3.75	3.75	-	-	-
14	1	2.56	2.56	2.56	-	-	-

101	31	0.31	880.33	69.14	25604.90	160.02	2.31
102	29	0.01	225.00	31.80	2902.18	53.87	1.69
201	13	2.39	1020.00	131.04	73217.40	270.59	2.06

Pravoberezhny - Au Statistical Analysis for Raw Drill Holes by Domain

Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	76	0.09	9.95	1.52	1.73	1.32	0.86
2	55	0.28	100.00	4.54	180.71	13.44	2.96
3	46	0.06	10.08	2.23	5.43	2.33	1.04
4	39	0.24	16.05	2.31	7.35	2.71	1.17
5	24	0.05	6.85	2.01	2.65	1.63	0.81
6	15	0.12	1.89	1.00	0.26	0.51	0.51
7	15	0.63	3.92	1.76	1.17	1.08	0.61
8	15	0.38	6.54	1.98	3.01	1.74	0.88
9	14	0.71	24.15	2.97	34.73	5.89	1.98
10	7	0.28	2.23	1.15	0.41	0.64	0.55
11	7	0.65	4.93	2.26	1.77	1.33	0.59
12	7	0.71	4.77	1.96	1.81	1.34	0.69
13	7	0.12	1.29	0.93	0.17	0.42	0.45
14	5	0.67	32.85	7.59	159.91	12.65	1.67
15	5	0.78	2.48	1.40	0.35	0.59	0.42
16	4	0.55	1.96	1.27	0.28	0.53	0.42
17	3	1.37	3.99	2.54	1.18	1.09	0.43
18	3	0.73	2.48	1.54	0.52	0.72	0.47
19	3	0.26	1.27	0.89	0.20	0.45	0.50
20	2	1.81	8.15	4.98	10.05	3.17	0.64
21	2	1.55	1.57	1.56	0.00	0.01	0.01
22	2	0.74	4.21	2.48	3.01	1.74	0.70
23	1	2.33	2.33	2.33	-	-	-
24	1	1.97	1.97	1.97	-	-	-
25	1	1.67	1.67	1.67	-	-	-
26	1	1.55	1.55	1.55	-	-	-
27	1	1.90	1.90	1.90	-	-	-
28	1	4.66	4.66	4.66	-	-	-
29	1	4.51	4.51	4.51	-	-	-
30	1	3.14	3.14	3.14	-	-	-
31	1	2.07	2.07	2.07	-	-	-
32	1	2.53	2.53	2.53	-	-	-

APPENDIX 3: Grade Capped Drill Hole Data Statistical Analysis

Pridolinny - Au Statistical Analysis for Grade Cap Drill Holes by Domain

Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	380	0.01	20.65	1.88	4.23	2.06	1.09
2	355	0.01	14.20	1.77	3.82	1.96	1.11
3	332	0.09	12.20	2.24	5.13	2.27	1.01
4	266	0.03	14.35	1.58	2.37	1.54	0.98
5	285	0.10	19.30	2.14	5.09	2.26	1.06
6	273	0.03	20.90	2.12	7.13	2.67	1.26
7	203	0.05	13.48	1.55	2.13	1.46	0.94
8	117	0.12	16.60	2.03	6.50	2.55	1.25
9	61	0.03	17.75	1.50	5.01	2.24	1.49
10	59	0.22	9.10	1.67	2.01	1.42	0.85
11	49	0.02	5.86	1.49	1.63	1.28	0.86
12	48	0.28	10.30	1.43	2.19	1.48	1.04
13	48	0.12	12.85	2.02	4.31	2.08	1.03
14	44	0.43	14.70	1.98	5.84	2.42	1.22
15	39	0.05	5.00	1.80	1.32	1.15	0.64
16	39	0.56	6.00	1.54	1.22	1.10	0.72
17	29	0.10	22.00	2.90	23.63	4.86	1.68
18	27	0.08	4.83	1.26	0.83	0.91	0.72
19	25	0.30	3.71	1.21	0.58	0.76	0.63
20	23	0.28	2.51	1.22	0.37	0.61	0.50
21	22	0.37	8.84	1.44	3.08	1.76	1.22
22	22	0.12	4.96	1.28	1.17	1.08	0.85
23	20	0.26	4.80	1.88	1.44	1.20	0.64
24	18	0.28	6.65	1.56	2.01	1.42	0.91
25	17	0.73	11.67	1.91	6.18	2.49	1.30
26	17	0.56	11.05	3.11	9.35	3.06	0.98
27	16	0.60	3.91	1.48	0.90	0.95	0.64
28	16	0.75	9.35	2.58	6.98	2.64	1.02
29	16	0.15	3.73	1.38	0.84	0.92	0.67
30	15	0.15	2.17	1.05	0.39	0.62	0.59
31	14	0.63	3.14	1.23	0.44	0.66	0.54
32	14	0.76	3.14	1.54	0.49	0.70	0.45
33	13	0.14	2.98	1.23	0.70	0.84	0.68
34	12	0.12	1.72	0.96	0.21	0.46	0.48
35	12	0.08	9.70	1.78	6.22	2.49	1.40
36	11	0.41	2.58	1.19	0.40	0.63	0.53
37	11	0.22	2.47	1.40	0.39	0.62	0.44
38	10	0.92	1.44	1.14	0.03	0.18	0.16
39	9	0.43	2.48	1.16	0.46	0.68	0.59
40	9	0.54	3.33	1.26	0.64	0.80	0.63
41	8	0.52	1.86	1.03	0.16	0.40	0.39
42	8	0.76	2.00	1.32	0.13	0.36	0.27
43	8	0.74	2.58	1.34	0.34	0.58	0.44
44	7	0.70	1.83	1.20	0.17	0.41	0.34
45	7	0.81	4.10	1.89	1.68	1.30	0.69
46	6	0.93	2.39	1.39	0.24	0.49	0.35
47	6	0.20	1.92	1.11	0.28	0.53	0.48
48	5	0.75	4.40	2.55	1.37	1.17	0.46
49	5	0.88	2.31	1.70	0.28	0.53	0.31
50	5	0.62	1.78	1.10	0.19	0.44	0.40
51	5	1.23	2.00	1.58	0.06	0.25	0.16
52	5	0.07	2.16	0.85	0.52	0.72	0.84
53	5	0.65	2.46	1.45	0.43	0.65	0.45
54	5	0.66	7.20	2.71	6.35	2.52	0.93
55	5	0.64	1.26	0.85	0.05	0.23	0.27
56	4	0.97	1.85	1.42	0.10	0.32	0.22
57	4	0.46	1.38	0.92	0.13	0.36	0.39
58	4	0.72	1.46	1.19	0.09	0.30	0.25
59	4	1.03	1.34	1.17	0.01	0.12	0.10
60	4	0.71	1.04	0.88	0.01	0.12	0.14
61	4	1.60	2.33	2.00	0.11	0.32	0.16
62	4	1.74	2.40	2.09	0.08	0.29	0.14
63	4	0.84	9.90	3.68	13.29	3.65	0.99

64	4	0.69	1.40	1.10	0.07	0.27	0.24
65	4	1.40	3.00	2.20	0.39	0.63	0.29
66	4	0.78	3.34	1.46	1.18	1.09	0.75
67	4	0.68	1.22	1.01	0.04	0.21	0.21
68	4	0.46	1.21	0.89	0.07	0.27	0.31
69	4	0.93	2.24	1.42	0.25	0.50	0.35
70	4	1.04	1.82	1.29	0.10	0.32	0.24
71	4	0.77	1.58	1.18	0.10	0.31	0.27
72	4	0.69	6.60	3.12	5.76	2.40	0.77
73	3	0.74	1.39	1.10	0.07	0.27	0.25
74	3	0.83	3.53	1.79	1.51	1.23	0.69
75	3	0.73	0.83	0.79	0.00	0.04	0.05
76	3	1.46	6.06	3.87	3.55	1.88	0.49
77	3	1.33	3.26	2.04	0.75	0.87	0.43
78	3	1.45	1.55	1.50	0.00	0.04	0.03
79	3	0.90	3.00	1.84	0.76	0.87	0.47
80	3	0.60	1.62	1.25	0.21	0.46	0.37
81	3	3.09	9.86	5.42	9.88	3.14	0.58
82	3	0.72	3.80	2.12	1.62	1.27	0.60
83	3	1.12	1.52	1.35	0.03	0.17	0.12
84	2	1.22	1.61	1.42	0.04	0.20	0.14
85	2	1.07	1.17	1.12	0.00	0.05	0.04
86	2	1.58	2.11	1.85	0.07	0.27	0.14
87	2	0.73	0.77	0.75	0.00	0.02	0.03
88	2	0.78	0.82	0.80	0.00	0.02	0.03
89	2	2.25	2.81	2.53	0.08	0.28	0.11
90	2	0.72	0.95	0.84	0.01	0.12	0.14
91	2	0.80	1.77	1.29	0.24	0.49	0.38
92	2	1.63	2.36	2.00	0.13	0.37	0.18
93	2	2.64	2.89	2.77	0.02	0.13	0.05
94	2	0.91	2.56	1.74	0.68	0.83	0.48
95	2	1.27	1.51	1.39	0.01	0.12	0.09
96	2	1.92	2.54	2.23	0.10	0.31	0.14
97	2	3.00	3.00	3.00	-	-	#VALUE!
98	2	0.81	1.46	1.14	0.11	0.33	0.29
99	2	0.74	1.60	1.17	0.18	0.43	0.37
100	2	0.83	0.86	0.85	0.00	0.02	0.02
101	2	0.79	1.75	1.27	0.23	0.48	0.38
102	2	0.81	0.81	0.81	-	-	-
103	2	0.73	1.93	1.33	0.36	0.60	0.45
104	2	1.06	1.69	1.38	0.10	0.32	0.23
105	2	2.74	2.88	2.81	0.00	0.07	0.02
106	2	3.04	3.22	3.13	0.01	0.09	0.03
107	2	0.76	0.93	0.85	0.01	0.09	0.10
108	2	0.76	0.85	0.81	0.00	0.05	0.06
109	2	1.11	3.28	2.20	1.18	1.09	0.49
110	2	1.57	4.05	2.81	1.54	1.24	0.44
111	2	0.81	1.60	1.21	0.16	0.40	0.33
112	2	0.87	2.05	1.46	0.35	0.59	0.40
113	2	0.70	0.93	0.82	0.01	0.12	0.14
114	2	0.77	0.82	0.80	0.00	0.03	0.03
115	2	0.79	0.82	0.81	0.00	0.02	0.02
116	2	1.10	1.12	1.11	0.00	0.01	0.01
117	2	0.70	4.22	2.46	3.10	1.76	0.72
118	2	0.75	1.02	0.89	0.02	0.14	0.15
119	2	1.36	1.70	1.53	0.03	0.17	0.11
120	2	0.83	0.86	0.85	0.00	0.02	0.02
121	1	4.64	4.64	4.64	-	-	-
122	1	1.82	1.82	1.82	-	-	-
123	1	6.51	6.51	6.51	-	-	-
124	1	1.52	1.52	1.52	-	-	-
125	1	1.87	1.87	1.87	-	-	-
126	1	2.18	2.18	2.18	-	-	-
127	1	1.85	1.85	1.85	-	-	-
128	1	1.64	1.64	1.64	-	-	-
129	1	4.04	4.04	4.04	-	-	-
130	1	1.58	1.58	1.58	-	-	-
131	1	2.30	2.30	2.30	-	-	-

132	1	1.57	1.57	1.57	-	-	-
133	1	1.59	1.59	1.59	-	-	-
134	1	3.42	3.42	3.42	-	-	-
135	1	3.00	3.00	3.00	-	-	-
136	1	4.29	4.29	4.29	-	-	-
137	1	1.50	1.50	1.50	-	-	-
138	1	1.66	1.66	1.66	-	-	-
139	1	1.55	1.55	1.55	-	-	-
140	1	1.31	1.31	1.31	-	-	-
141	1	4.25	4.25	4.25	-	-	-
142	36	0.47	4.08	1.27	0.53	0.73	0.57

Gora5 - Au Statistical Analysis for Grade Cap Drill Holes by Domain

Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	297	0.16	14.50	2.44	6.11	2.47	1.01
2	274	0.06	19.18	2.18	9.30	3.05	1.40
3	54	0.51	6.98	1.87	1.34	1.16	0.62
4	31	0.08	9.05	1.94	2.68	1.64	0.84
5	26	0.16	3.28	1.21	0.52	0.72	0.60
6	26	0.01	4.86	1.36	1.25	1.12	0.82
7	24	0.24	20.40	3.22	21.46	4.63	1.44
8	21	0.30	14.13	3.17	10.78	3.28	1.03
9	21	0.36	2.38	1.14	0.27	0.52	0.45
10	12	0.73	3.54	1.65	0.59	0.77	0.47
11	10	0.79	1.85	1.28	0.15	0.38	0.30
12	5	0.13	2.65	1.19	0.84	0.92	0.77
13	4	0.72	1.44	0.99	0.08	0.29	0.29
14	3	0.82	1.85	1.22	0.20	0.45	0.37
15	3	1.49	4.61	2.72	1.84	1.36	0.50
16	2	0.70	1.66	1.18	0.23	0.48	0.41
17	2	1.22	1.29	1.26	0.00	0.04	0.03
18	2	1.55	2.27	1.91	0.13	0.36	0.19
19	2	1.73	4.00	2.87	1.29	1.14	0.40
20	2	0.70	1.37	1.04	0.11	0.34	0.32
21	2	0.71	0.97	0.84	0.02	0.13	0.15
22	2	0.74	1.17	0.96	0.05	0.22	0.23
23	2	0.96	3.00	1.98	1.04	1.02	0.52
24	2	0.94	1.52	1.23	0.08	0.29	0.24
25	2	0.84	3.09	1.97	1.27	1.13	0.57
26	1	2.28	2.28	2.28	-	-	-
27	1	2.36	2.36	2.36	-	-	-
28	1	1.50	1.50	1.50	-	-	-
29	1	2.08	2.08	2.08	-	-	-

Zhelanny - Au Statistical Analysis for Grade Cap Drill Holes by Domain							
Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	137	0.04	16.50	2.42	5.42	2.33	0.96
2	212	0.02	7.51	1.98	2.05	1.43	0.72
3	146	0.08	16.25	2.36	7.90	2.81	1.19
4	31	0.30	2.85	1.22	0.37	0.61	0.50
5	9	0.71	2.11	1.27	0.27	0.52	0.41
6	8	0.84	3.94	1.82	1.08	1.04	0.57
7	7	0.17	4.95	2.18	2.21	1.49	0.68
8	6	0.77	4.61	1.94	1.70	1.30	0.67
9	2	1.79	2.54	2.17	0.14	0.38	0.17
10	2	1.42	3.82	2.62	1.44	1.20	0.46
11	2	1.92	2.54	2.23	0.10	0.31	0.14
12	1	2.29	2.29	2.29	-	-	-
13	1	3.75	3.75	3.75	-	-	-
14	1	2.56	2.56	2.56	-	-	-
101	31	0.31	195.80	46.47	4222.31	64.98	1.40
102	29	0.01	195.80	30.79	2541.51	50.41	1.64
201	13	2.39	195.80	60.30	6230.54	78.93	1.31

Pravoberezhny - Au Statistical Analysis for Grade Cap Drill Holes by Domain							
Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	76	0.09	9.95	1.52	1.73	1.32	0.86
2	55	0.28	15.75	3.00	14.94	3.87	1.29
3	46	0.06	10.08	2.23	5.43	2.33	1.04
4	39	0.24	7.96	2.10	3.28	1.81	0.86
5	24	0.05	6.85	2.01	2.65	1.63	0.81
6	15	0.12	1.89	1.00	0.26	0.51	0.51
7	15	0.63	3.92	1.76	1.17	1.08	0.61
8	15	0.38	6.54	1.98	3.01	1.74	0.88
9	14	0.71	3.00	1.46	0.41	0.64	0.44
10	7	0.28	2.23	1.15	0.41	0.64	0.55
11	7	0.65	4.93	2.26	1.77	1.33	0.59
12	7	0.71	4.77	1.96	1.81	1.34	0.69
13	7	0.12	1.29	0.93	0.17	0.42	0.45
14	5	0.67	3.00	1.62	0.90	0.95	0.58
15	5	0.78	2.48	1.40	0.35	0.59	0.42
16	4	0.55	1.96	1.27	0.28	0.53	0.42
17	3	1.37	3.99	2.54	1.18	1.09	0.43
18	3	0.73	2.48	1.54	0.52	0.72	0.47
19	3	0.26	1.27	0.89	0.20	0.45	0.50
20	2	1.81	8.15	4.98	10.05	3.17	0.64
21	2	1.55	1.57	1.56	0.00	0.01	0.01
22	2	0.74	4.21	2.48	3.01	1.74	0.70
23	1	2.33	2.33	2.33	-	-	-
24	1	1.97	1.97	1.97	-	-	-
25	1	1.67	1.67	1.67	-	-	-
26	1	1.55	1.55	1.55	-	-	-
27	1	1.90	1.90	1.90	-	-	-
28	1	4.66	4.66	4.66	-	-	-
29	1	4.51	4.51	4.51	-	-	-
30	1	3.14	3.14	3.14	-	-	-
31	1	2.07	2.07	2.07	-	-	-
32	1	2.53	2.53	2.53	-	-	-

APPENDIX 4: Composite Data Statistical Analysis

Pridolinny - Au Statistical Analysis for Composites by Domain

Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	365	0.05	20.65	1.89	4.15	2.04	1.07
2	337	0.03	14.20	1.76	3.72	1.93	1.09
3	320	0.09	12.20	2.19	4.42	2.10	0.96
4	259	0.03	14.35	1.57	2.27	1.51	0.96
5	261	0.18	19.30	2.12	4.25	2.06	0.97
6	260	0.03	20.36	2.05	5.33	2.31	1.13
7	199	0.05	13.48	1.53	2.03	1.43	0.93
8	102	0.12	13.50	1.93	4.41	2.10	1.09
9	56	0.03	17.75	1.48	5.30	2.30	1.56
10	58	0.23	9.10	1.67	1.89	1.37	0.82
11	43	0.02	4.84	1.33	0.78	0.89	0.67
12	46	0.28	8.28	1.41	1.51	1.23	0.87
13	42	0.54	12.85	2.02	3.91	1.98	0.98
14	40	0.48	10.69	1.93	4.14	2.03	1.06
15	38	0.05	5.00	1.76	1.23	1.11	0.63
16	36	0.58	6.00	1.53	1.22	1.10	0.72
17	27	0.10	10.24	2.23	6.42	2.53	1.14
18	24	0.20	4.83	1.31	0.87	0.93	0.71
19	21	0.38	2.17	1.20	0.27	0.52	0.44
20	19	0.49	2.51	1.28	0.35	0.59	0.46
21	22	0.37	8.84	1.46	2.93	1.71	1.18
22	20	0.33	4.37	1.34	0.93	0.97	0.72
23	21	0.26	4.80	1.83	1.41	1.19	0.65
24	15	0.41	5.91	1.64	1.78	1.33	0.81
25	17	0.81	11.67	1.93	6.10	2.47	1.28
26	17	0.76	10.26	3.17	8.11	2.85	0.90
27	14	0.60	3.06	1.49	0.59	0.77	0.51
28	14	0.75	9.22	2.65	7.07	2.66	1.00
29	15	0.32	3.65	1.30	0.69	0.83	0.64
30	15	0.15	2.17	1.05	0.38	0.62	0.59
31	13	0.63	2.61	1.19	0.33	0.57	0.48
32	14	0.76	3.14	1.54	0.49	0.70	0.45
33	12	0.23	2.98	1.21	0.62	0.79	0.65
34	12	0.12	1.72	0.96	0.21	0.46	0.48
35	11	0.08	9.70	1.86	6.70	2.59	1.39
36	11	0.41	2.58	1.21	0.39	0.62	0.51
37	9	0.22	2.47	1.42	0.35	0.59	0.42
38	9	0.92	1.35	1.14	0.03	0.16	0.14
39	9	0.46	2.11	1.17	0.31	0.56	0.48
40	9	0.54	2.85	1.20	0.44	0.66	0.55
41	8	0.52	1.86	1.03	0.16	0.40	0.39
42	8	0.76	2.00	1.32	0.13	0.36	0.27
43	8	0.74	2.58	1.34	0.34	0.58	0.44
44	7	0.70	1.83	1.16	0.14	0.37	0.32
45	7	0.81	4.10	1.89	1.67	1.29	0.68
46	5	1.07	2.39	1.47	0.23	0.48	0.33
47	5	0.75	1.92	1.22	0.19	0.44	0.36
48	5	0.75	4.40	2.55	1.37	1.17	0.46
49	5	0.88	2.31	1.70	0.28	0.53	0.31
50	5	0.62	1.78	1.10	0.19	0.44	0.40
51	5	1.24	2.00	1.59	0.06	0.25	0.16
52	5	0.07	2.16	0.85	0.52	0.72	0.84
53	5	0.65	2.46	1.45	0.43	0.65	0.45
54	4	0.80	7.20	2.91	6.57	2.56	0.88
55	5	0.64	1.26	0.85	0.05	0.23	0.27
56	4	1.11	1.80	1.44	0.06	0.25	0.17
57	4	0.54	1.38	0.94	0.11	0.34	0.36
58	3	0.72	1.46	1.13	0.09	0.31	0.27
59	4	1.03	1.34	1.17	0.01	0.12	0.10
60	4	0.71	1.04	0.88	0.01	0.12	0.14
61	4	1.60	2.33	2.00	0.11	0.32	0.16
62	4	1.74	2.40	2.09	0.08	0.29	0.14
63	4	0.84	9.90	3.68	13.29	3.65	0.99

64	4	0.69	1.40	1.10	0.07	0.27	0.24
65	3	1.81	3.00	2.32	0.25	0.50	0.22
66	4	0.78	3.34	1.46	1.18	1.09	0.75
67	4	0.68	1.22	1.01	0.04	0.21	0.21
68	4	0.46	1.21	0.89	0.07	0.27	0.31
69	4	0.93	2.24	1.42	0.25	0.50	0.35
70	4	1.04	1.82	1.29	0.10	0.32	0.24
71	4	0.77	1.58	1.18	0.10	0.31	0.27
72	4	0.69	6.60	3.12	5.76	2.40	0.77
73	3	0.74	1.39	1.10	0.07	0.27	0.25
74	2	0.93	3.53	2.23	1.70	1.30	0.58
75	2	0.74	0.82	0.78	0.00	0.04	0.05
76	3	1.46	6.06	3.87	3.55	1.88	0.49
77	3	1.35	3.05	1.97	0.58	0.76	0.39
78	3	1.45	1.55	1.50	0.00	0.04	0.03
79	4	0.90	3.00	2.13	0.82	0.91	0.43
80	2	1.32	1.62	1.47	0.02	0.15	0.10
81	2	3.30	5.12	4.21	0.83	0.91	0.22
82	3	0.83	3.80	2.16	1.52	1.23	0.57
83	3	1.12	1.52	1.35	0.03	0.17	0.12
84	2	1.26	1.61	1.43	0.03	0.18	0.12
85	2	1.07	1.14	1.10	0.00	0.03	0.03
86	2	1.58	2.11	1.85	0.07	0.27	0.14
87	2	0.73	0.77	0.75	0.00	0.02	0.03
88	2	0.78	0.82	0.80	0.00	0.02	0.03
89	1	2.53	2.53	2.53	-	-	-
90	2	0.73	0.95	0.84	0.01	0.11	0.13
91	2	0.80	1.77	1.29	0.24	0.49	0.38
92	2	1.63	2.36	2.00	0.13	0.37	0.18
93	1	2.72	2.72	2.72	-	-	-
94	2	0.91	2.56	1.74	0.68	0.83	0.48
95	2	1.27	1.51	1.39	0.01	0.12	0.09
96	2	1.92	2.54	2.23	0.10	0.31	0.14
97	2	3.00	3.00	3.00	-	-	-
98	2	0.96	1.46	1.21	0.06	0.25	0.20
99	2	0.74	1.60	1.17	0.18	0.43	0.37
100	2	0.83	0.86	0.85	0.00	0.02	0.02
101	2	0.79	1.75	1.27	0.23	0.48	0.38
102	2	0.81	0.81	0.81	-	-	-
103	2	0.73	1.93	1.33	0.36	0.60	0.45
104	2	1.06	1.50	1.28	0.05	0.22	0.17
105	2	2.74	2.88	2.81	0.00	0.07	0.02
106	1	3.12	3.12	3.12	-	-	-
107	2	0.76	0.93	0.85	0.01	0.09	0.10
108	2	0.76	0.84	0.80	0.00	0.04	0.05
109	2	1.11	3.28	2.20	1.18	1.09	0.49
110	2	1.57	4.05	2.81	1.54	1.24	0.44
111	2	0.81	1.60	1.21	0.16	0.40	0.33
112	2	0.87	2.05	1.46	0.35	0.59	0.40
113	2	0.70	0.93	0.82	0.01	0.12	0.14
114	2	0.77	0.82	0.80	0.00	0.03	0.03
115	2	0.79	0.82	0.81	0.00	0.02	0.02
116	2	1.10	1.12	1.11	0.00	0.01	0.01
117	2	0.70	3.05	1.87	1.38	1.17	0.63
118	2	0.75	1.02	0.89	0.02	0.14	0.15
119	2	1.36	1.70	1.53	0.03	0.17	0.11
120	2	0.83	0.86	0.85	0.00	0.02	0.02
121	1	4.64	4.64	4.64	-	-	-
122	1	1.82	1.82	1.82	-	-	-
123	1	6.51	6.51	6.51	-	-	-
124	1	1.52	1.52	1.52	-	-	-
125	1	1.87	1.87	1.87	-	-	-
126	1	2.18	2.18	2.18	-	-	-
127	1	1.85	1.85	1.85	-	-	-
128	1	1.64	1.64	1.64	-	-	-
129	1	4.04	4.04	4.04	-	-	-
130	1	1.58	1.58	1.58	-	-	-
131	1	2.30	2.30	2.30	-	-	-

132	1	1.57	1.57	1.57	-	-	-
133	1	1.59	1.59	1.59	-	-	-
134	1	3.42	3.42	3.42	-	-	-
135	1	3.00	3.00	3.00	-	-	-
136	1	4.29	4.29	4.29	-	-	-
137	1	1.50	1.50	1.50	-	-	-
138	1	1.66	1.66	1.66	-	-	-
139	1	1.55	1.55	1.55	-	-	-
140	2	1.31	1.31	1.31	-	-	-
141	1	4.25	4.25	4.25	-	-	-
142	36	0.47	4.08	1.28	0.53	0.73	0.57

Gora5 - Au Statistical Analysis for Composites by Domain

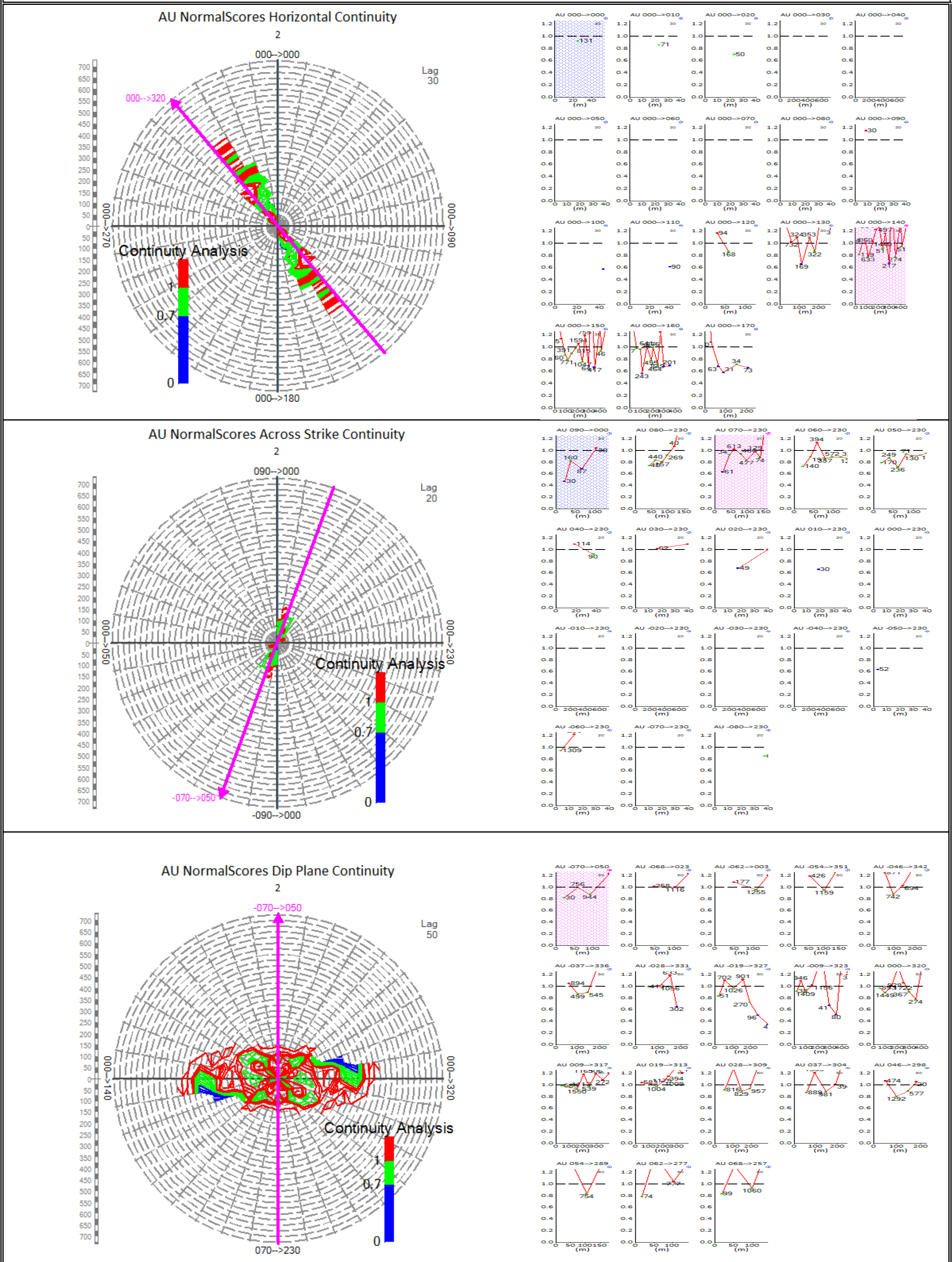
Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	269	0.20	14.50	2.40	5.30	2.30	0.96
2	247	0.09	19.18	2.15	6.71	2.59	1.20
3	50	0.55	4.69	1.86	0.93	0.96	0.52
4	28	0.10	8.34	1.91	2.31	1.52	0.80
5	24	0.16	3.28	1.17	0.47	0.69	0.59
6	25	0.17	3.90	1.38	0.93	0.97	0.70
7	22	0.74	20.40	3.19	20.54	4.53	1.42
8	19	0.35	14.13	3.35	11.50	3.39	1.01
9	19	0.39	2.38	1.15	0.23	0.48	0.42
10	10	0.82	3.07	1.64	0.47	0.68	0.42
11	10	0.79	1.85	1.31	0.13	0.35	0.27
12	4	0.18	1.83	1.13	0.46	0.68	0.60
13	4	0.72	1.44	0.99	0.08	0.29	0.29
14	3	0.85	1.85	1.23	0.19	0.44	0.36
15	3	1.49	4.61	2.72	1.84	1.36	0.50
16	2	0.70	1.66	1.18	0.23	0.48	0.41
17	2	1.22	1.29	1.26	0.00	0.04	0.03
18	1	2.10	2.10	2.10	-	-	-
19	1	2.87	2.87	2.87	-	-	-
20	2	0.74	1.37	1.05	0.10	0.32	0.30
21	2	0.71	0.97	0.84	0.02	0.13	0.15
22	2	0.74	1.17	0.96	0.05	0.22	0.23
23	2	1.47	3.00	2.24	0.59	0.77	0.34
24	2	0.94	1.52	1.23	0.08	0.29	0.24
25	2	0.84	2.34	1.59	0.56	0.75	0.47
26	1	2.28	2.28	2.28	-	-	-
27	1	2.36	2.36	2.36	-	-	-
28	1	1.50	1.50	1.50	-	-	-
29	1	2.08	2.08	2.08	-	-	-

Zhelanny - Au Statistical Analysis for Composites by Domain							
Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	123	0.04	16.50	2.39	4.67	2.16	0.90
2	193	0.13	7.51	1.99	1.82	1.35	0.68
3	141	0.08	16.25	2.36	7.41	2.72	1.15
4	31	0.30	2.85	1.22	0.37	0.61	0.50
5	8	0.71	2.11	1.28	0.28	0.53	0.41
6	8	0.84	3.94	1.84	1.07	1.04	0.56
7	6	0.28	3.60	1.94	1.17	1.08	0.56
8	4	0.84	2.61	1.75	0.44	0.67	0.38
9	2	1.79	2.54	2.17	0.14	0.38	0.17
10	2	1.55	3.82	2.68	1.29	1.14	0.42
11	2	1.92	2.54	2.23	0.10	0.31	0.14
12	1	2.29	2.29	2.29	-	-	-
13	1	3.75	3.75	3.75	-	-	-
14	1	2.56	2.56	2.56	-	-	-
101	30	0.72	195.80	45.69	4167.38	64.56	1.41
102	23	5.22	122.63	26.63	1033.78	32.15	1.21
201	11	2.71	195.80	61.08	5158.44	71.82	1.18

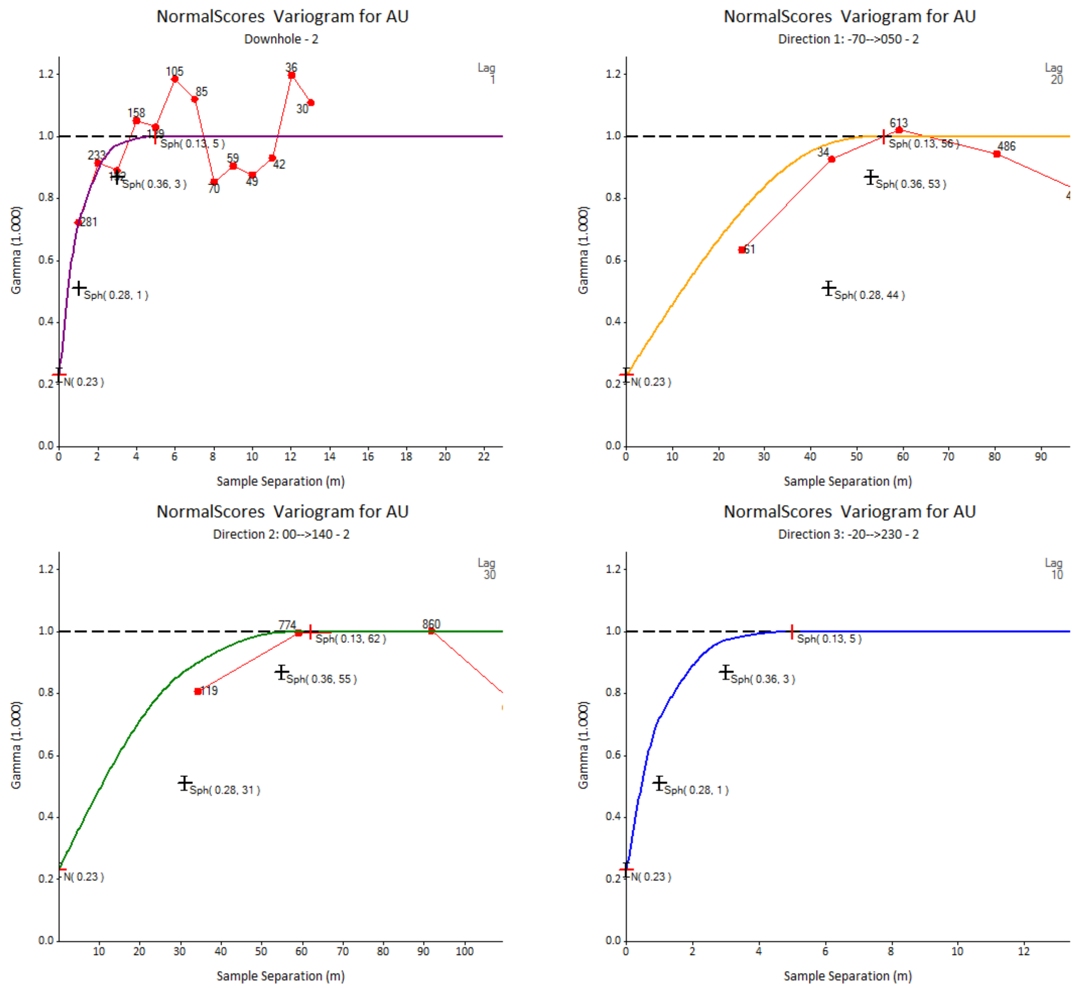
Pravoberezhny - Au Statistical Analysis for Composites by Domain							
Domain	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
1	69	0.09	9.95	1.47	1.59	1.26	0.86
2	51	0.60	15.75	2.97	12.92	3.59	1.21
3	44	0.20	10.08	2.32	5.51	2.35	1.01
4	36	0.24	7.96	2.05	2.45	1.56	0.76
5	22	0.10	6.85	2.12	2.45	1.57	0.74
6	13	0.33	1.89	1.01	0.20	0.45	0.44
7	14	0.70	3.89	1.78	1.04	1.02	0.57
8	14	0.41	5.32	1.95	2.01	1.42	0.73
9	13	0.80	3.00	1.46	0.37	0.61	0.42
10	7	0.28	2.23	1.15	0.41	0.64	0.55
11	7	0.65	4.93	2.26	1.77	1.33	0.59
12	7	0.71	4.77	1.96	1.81	1.34	0.69
13	6	0.26	1.29	1.03	0.12	0.35	0.34
14	5	0.74	3.00	1.63	0.88	0.94	0.57
15	4	0.99	2.48	1.50	0.34	0.58	0.39
16	4	0.59	1.96	1.22	0.26	0.51	0.42
17	3	1.37	3.99	2.54	1.18	1.09	0.43
18	3	0.86	2.48	1.55	0.47	0.69	0.44
19	2	0.79	1.27	1.03	0.06	0.24	0.23
20	1	6.34	6.34	6.34	-	-	-
21	2	1.55	1.57	1.56	0.00	0.01	0.01
22	2	0.74	4.21	2.48	3.01	1.74	0.70
23	1	2.33	2.33	2.33	-	-	-
24	1	1.97	1.97	1.97	-	-	-
25	1	1.67	1.67	1.67	-	-	-
26	1	1.55	1.55	1.55	-	-	-
27	1	1.90	1.90	1.90	-	-	-
28	1	4.66	4.66	4.66	-	-	-
29	1	4.51	4.51	4.51	-	-	-
30	1	3.14	3.14	3.14	-	-	-
31	1	2.07	2.07	2.07	-	-	-
32	1	2.53	2.53	2.53	-	-	-

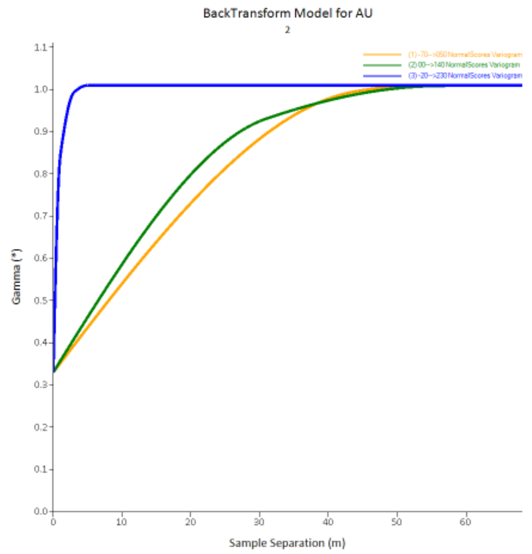
APPENDIX 5: Continuity Analysis and Variography

Pridolinny - Domain 2 Continuity Analysis for Au (g/t) 00°→320° (Strike) and -070°→050° (Dip)

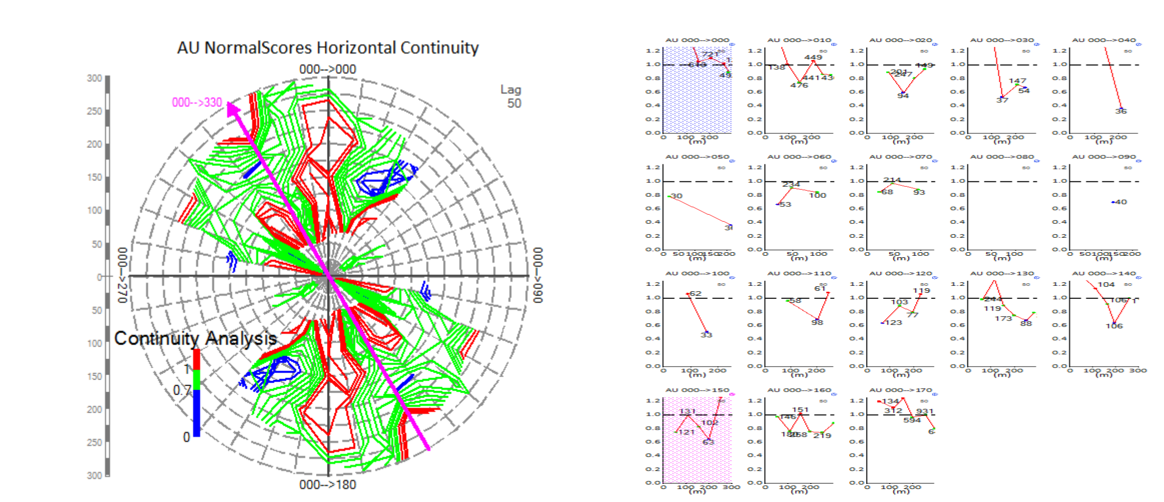


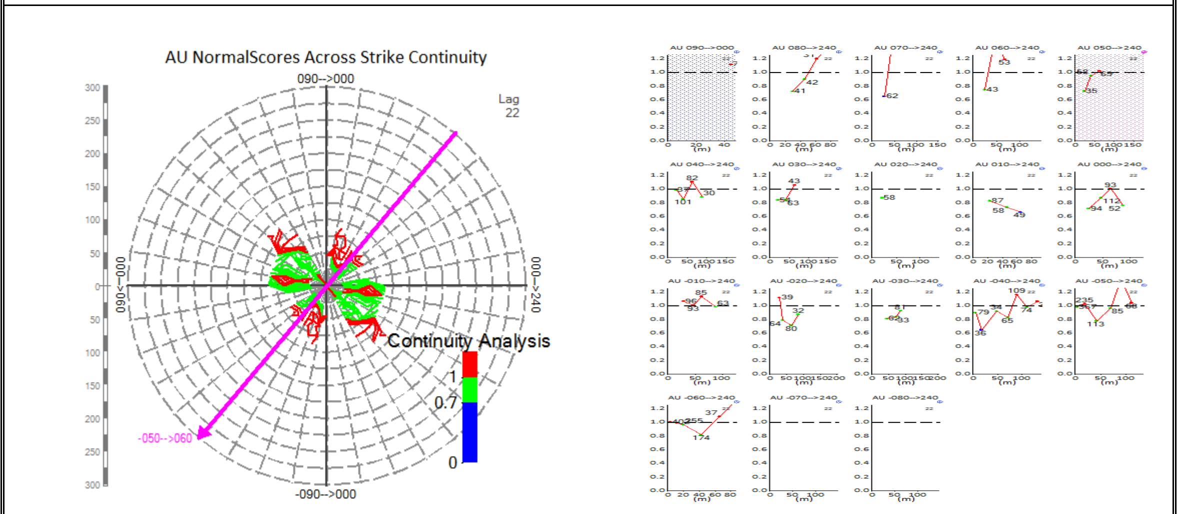
Pridolinny Domain 2 Variography for Au (g/t) 00°→140° (Strike) and -070°→050° (Dip)

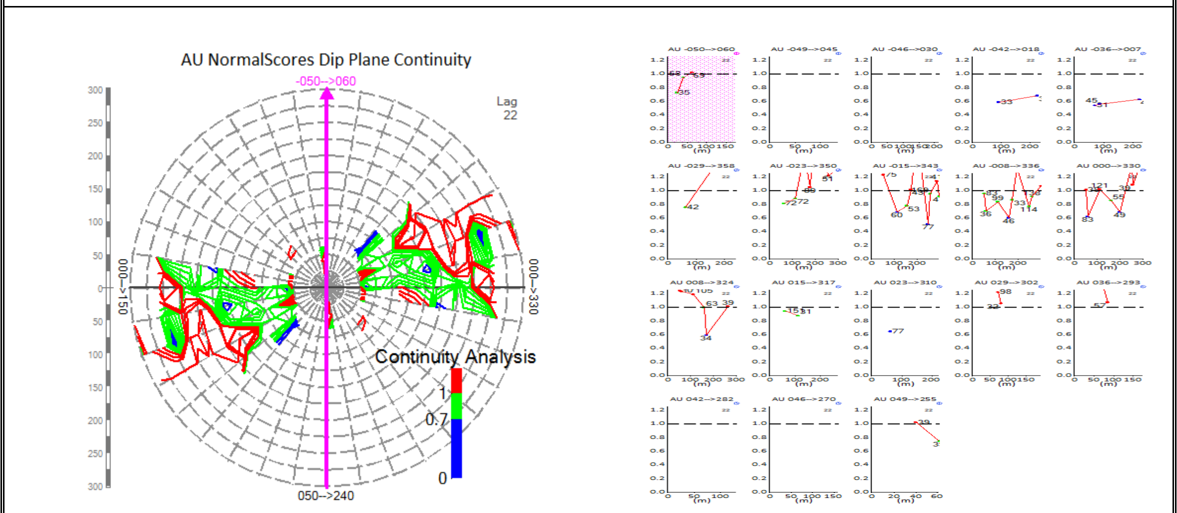




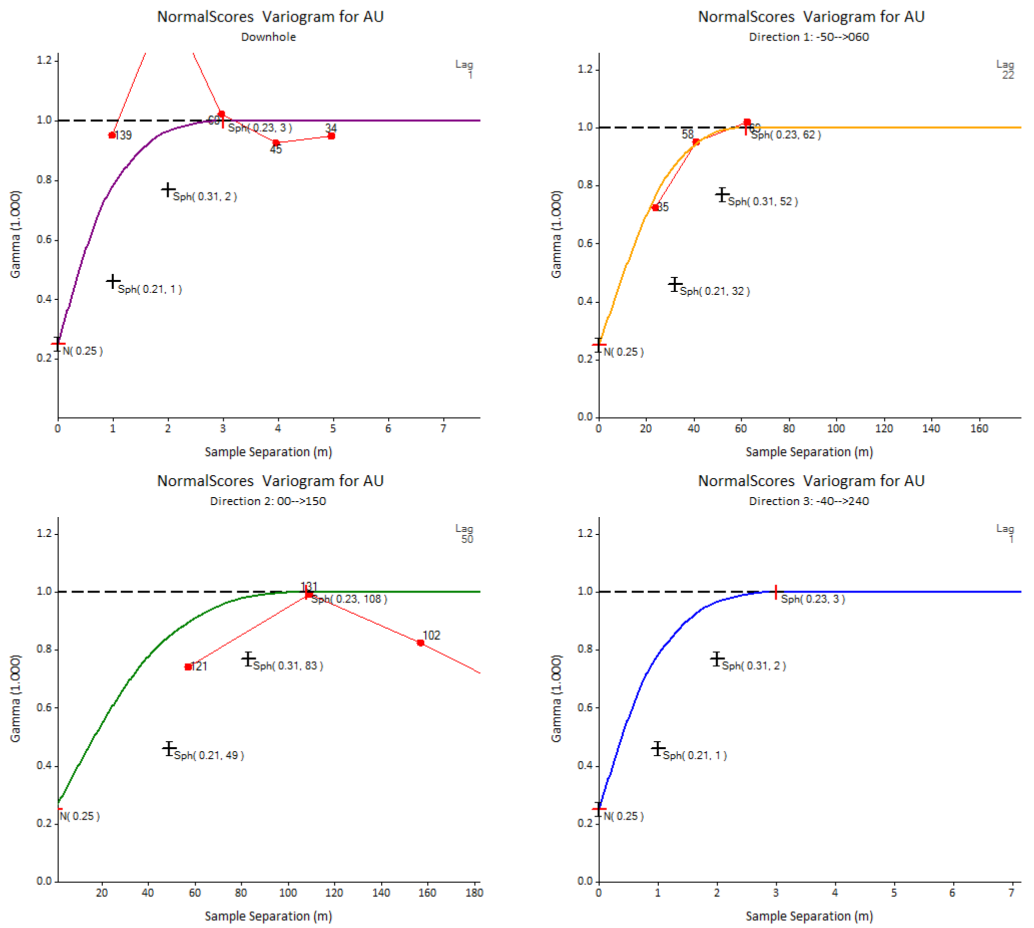
Pridolinny Extension Continuity Analysis for Au (g/t) 00°→330° (Strike) and -050°→060° (Dip)

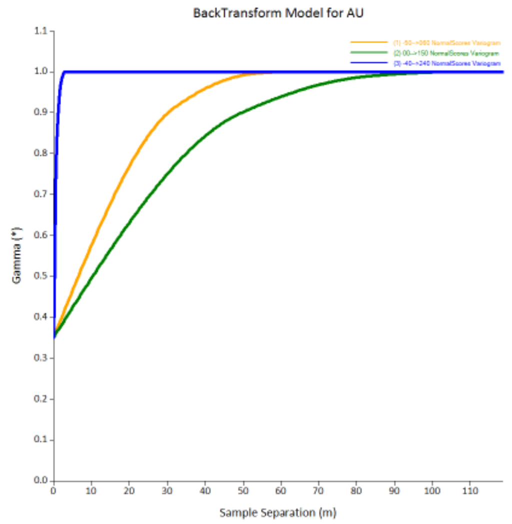




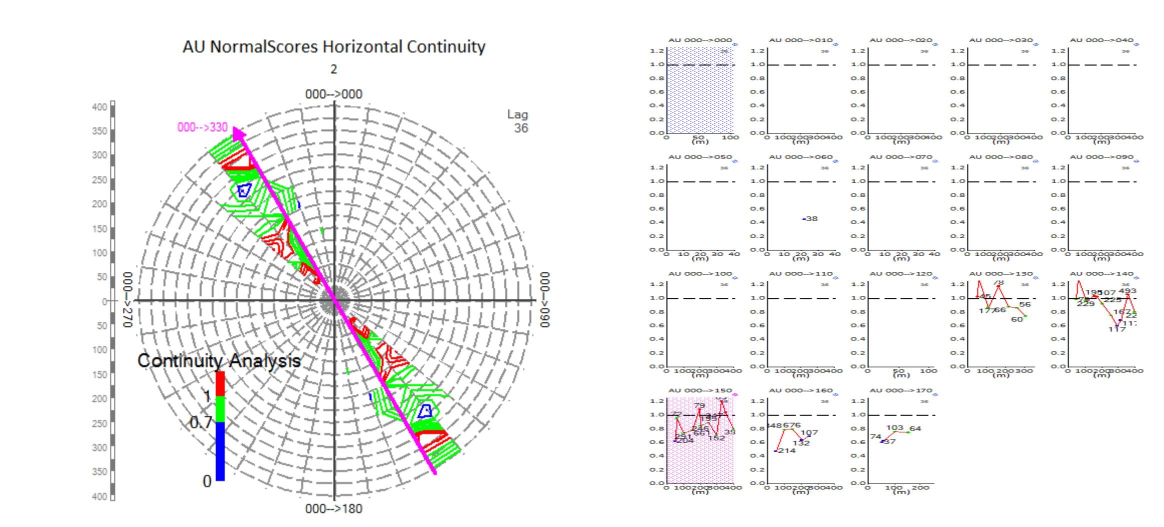


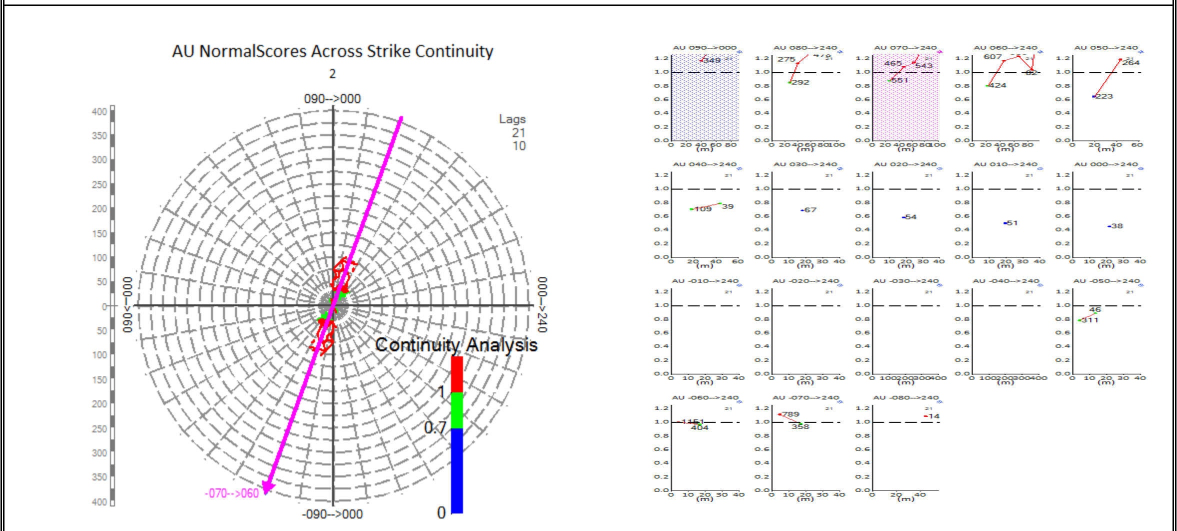
Pridolinny Extension Variography for Au (g/t) 00°→330° (Strike) and -050°→060° (Dip)

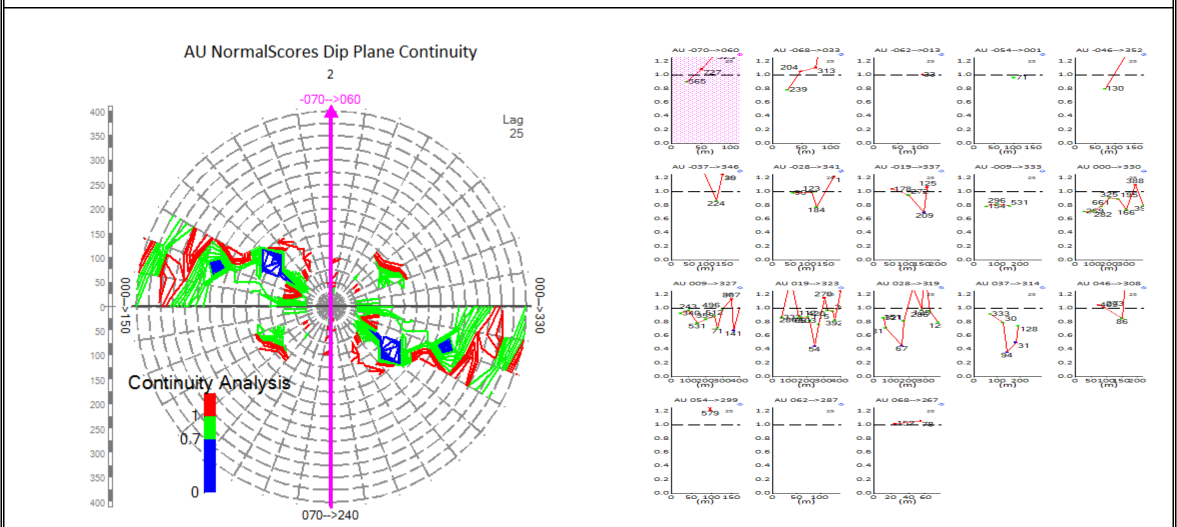




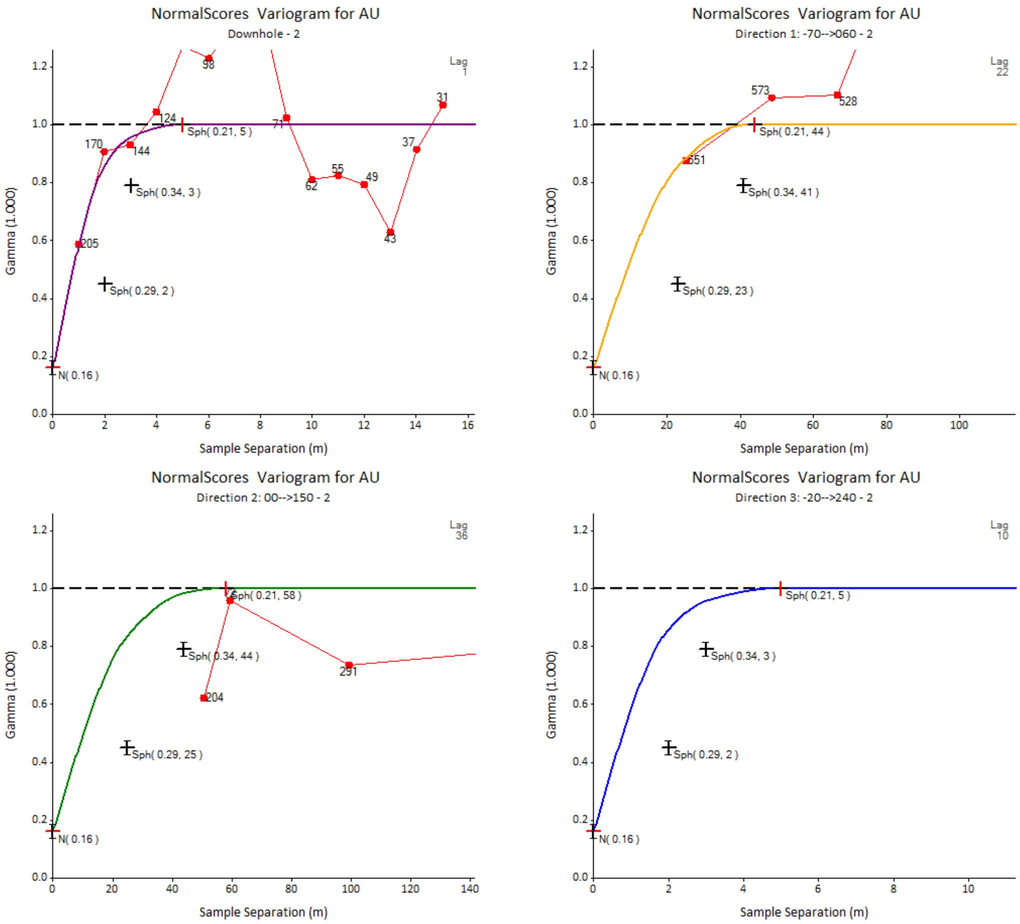
Gora5 Domain 2 Continuity Analysis for Au (g/t) 00°→330° (Strike) and -070°→060° (Dip)

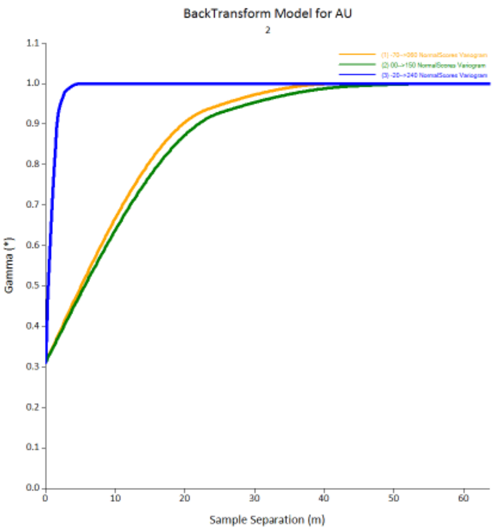




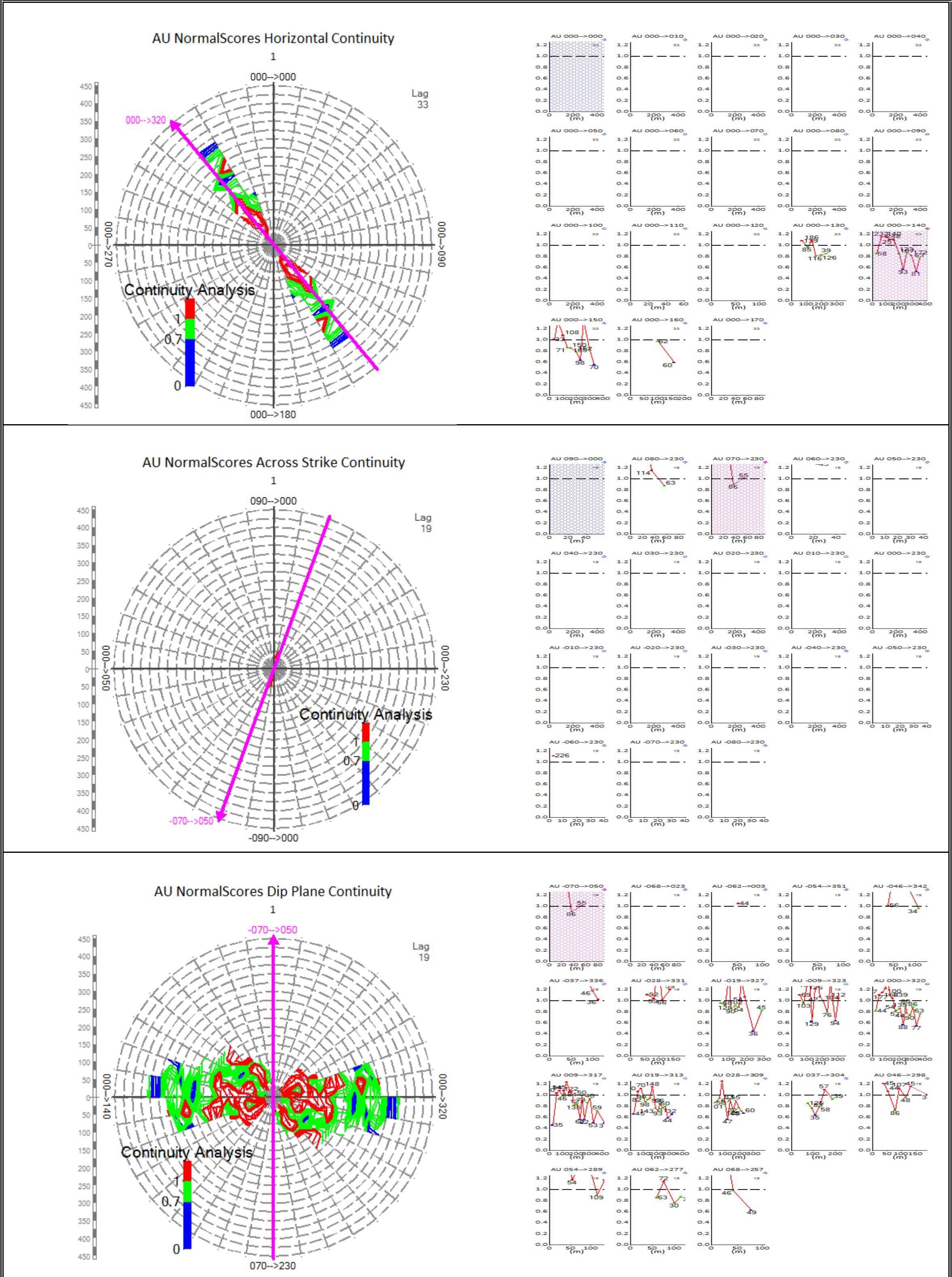


Gora5 Doman 2 Variography for Au (g/t) 00°→150° (Strike) and -070°→060° (Dip)

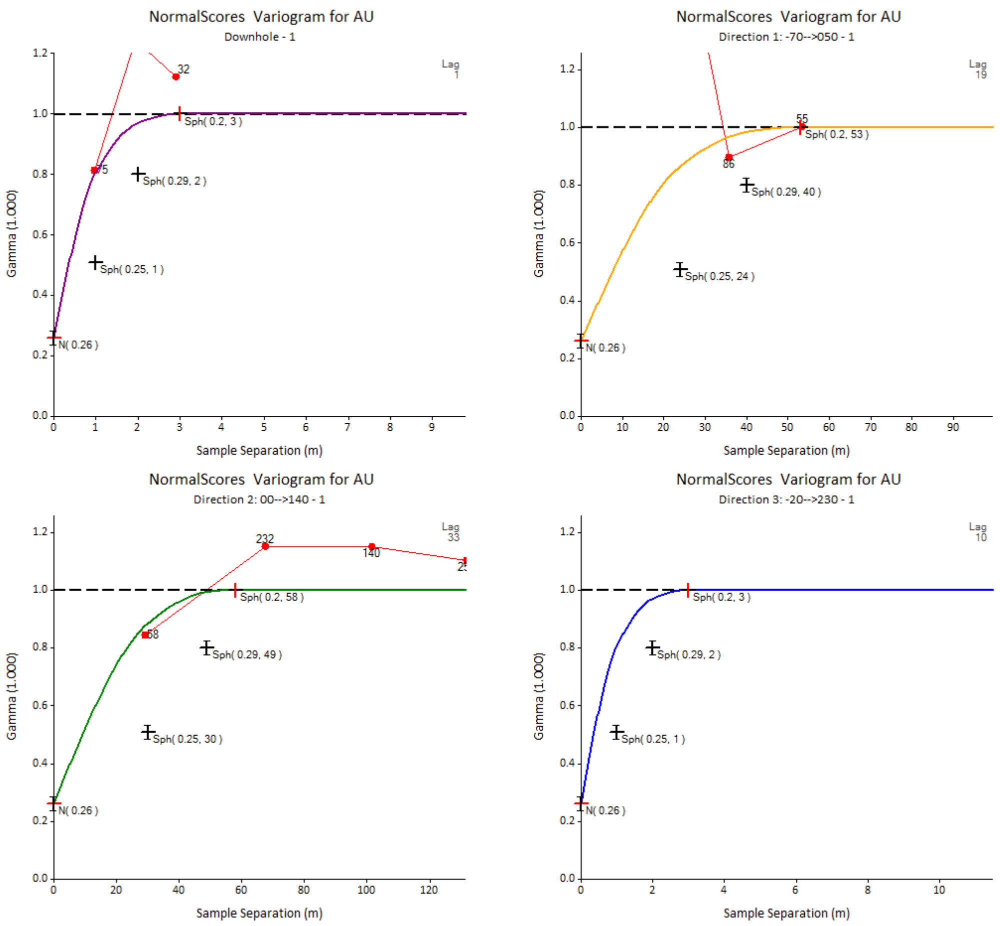


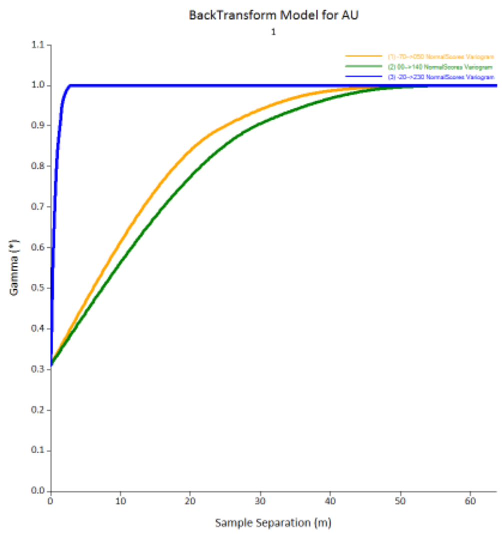


Zhelanny Domain 1 Continuity Analysis for Au (g/t) 00°→320° (Strike) and -070°→050° (Dip)

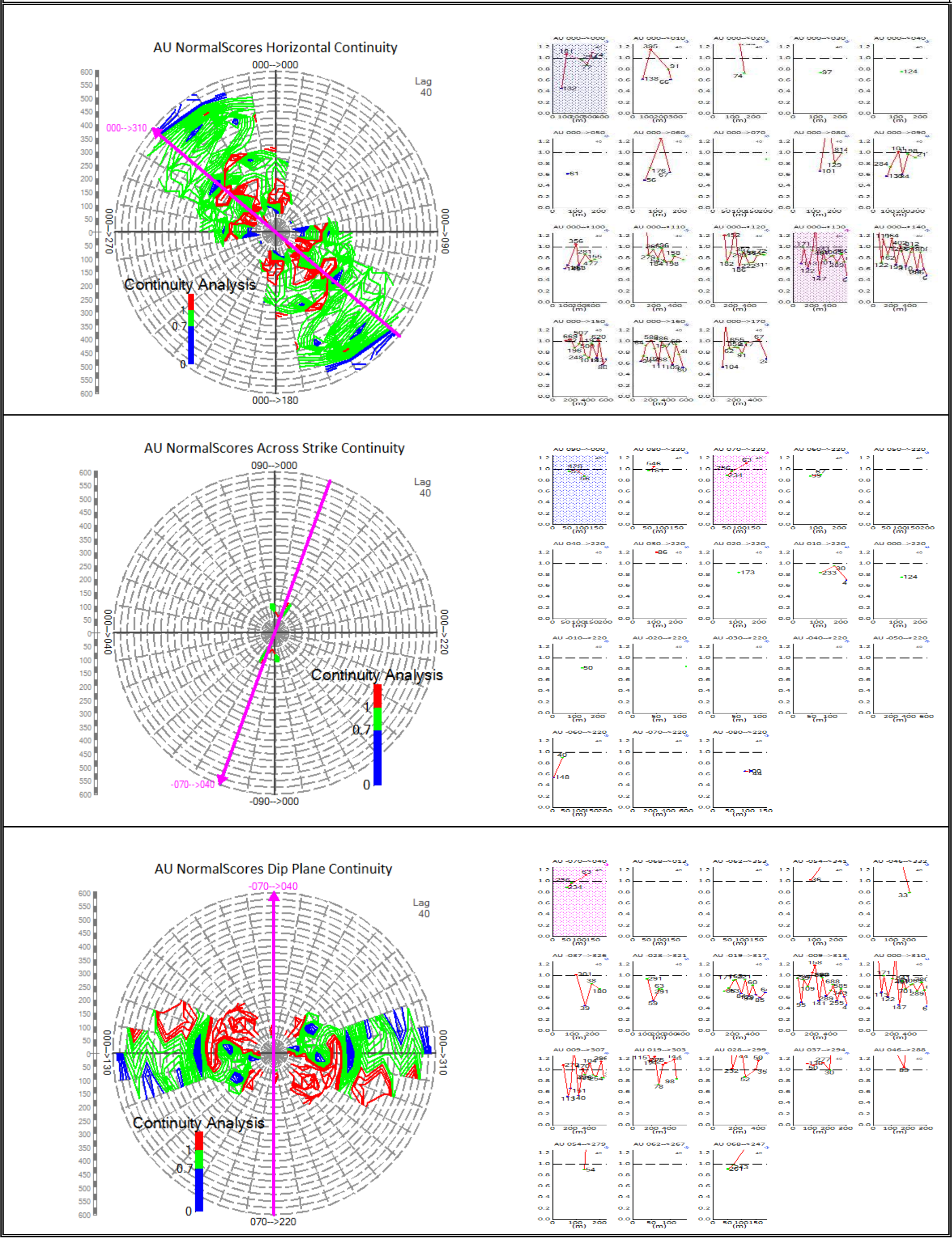


Zhelanny Doman 1 Variography for Au (g/t) 00°→140° (Strike) and -070°→050° (Dip)

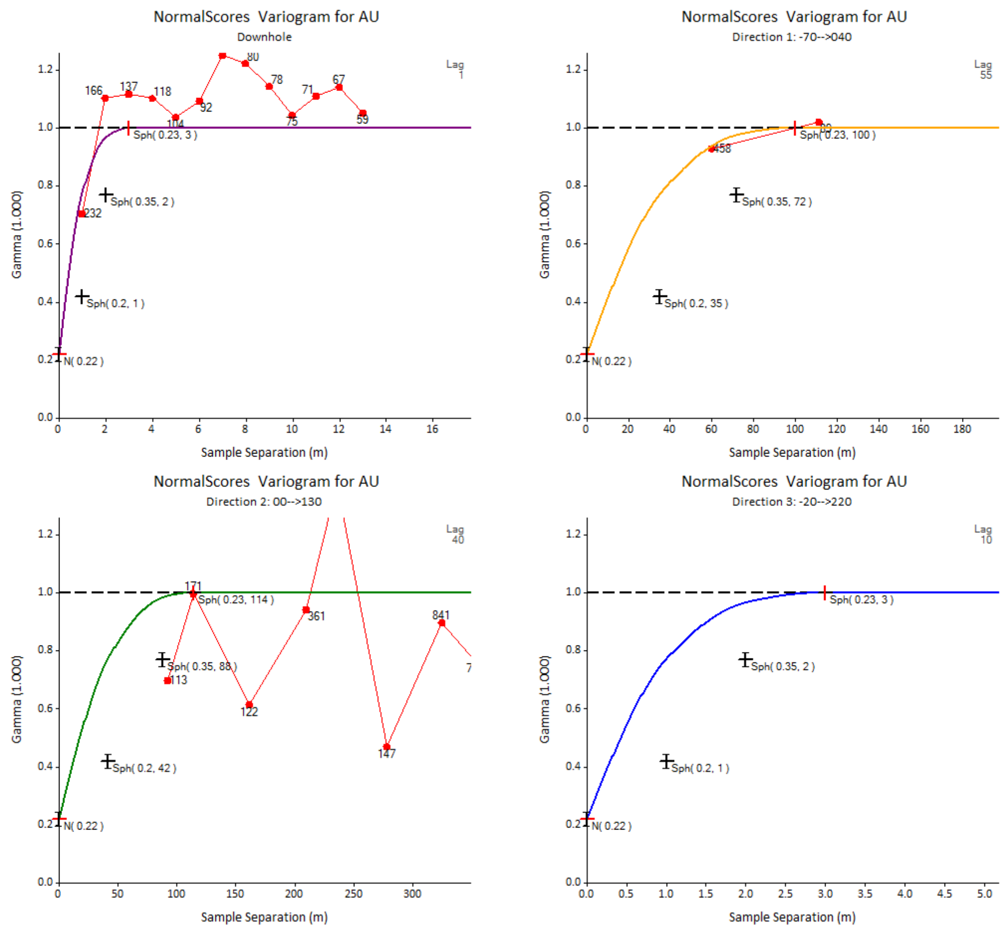


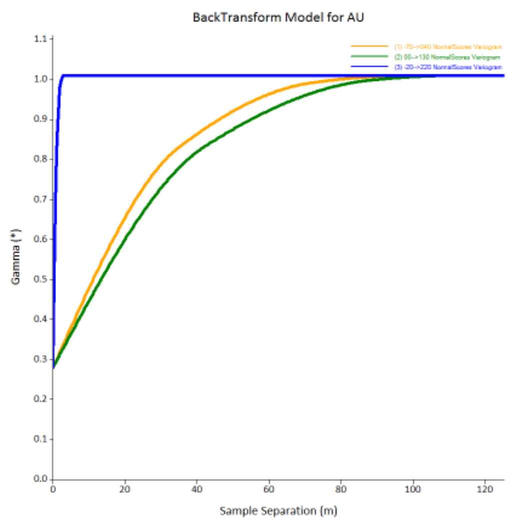


Pravoberezhny Continuity Analysis for Au (g/t) 00°→310° (Strike) and -070°→040° (Dip)



Pravoberezhny Variography for Au (g/t) 00°→130° (Strike) and -070°→040° (Dip)





APPENDIX 6: Mean Grade Analysis

Pridolinny - Au Mean Grade Analysis by Domain

Domain	Composites		Block Model		
	No. of Samples	Mean	IDW3	OK	NN
1	365	1.89	1.84	1.82	1.85
2	337	1.76	1.60	1.59	1.62
3	320	2.19	2.05	2.03	2.08
4	259	1.57	1.51	1.52	1.50
5	261	2.12	2.21	2.25	2.26
6	260	2.05	2.26	2.34	2.15
7	199	1.53	1.57	1.58	1.59
8	102	1.93	2.01	2.04	2.37
9	56	1.48	1.43	1.40	1.60
10	58	1.67	1.91	1.73	2.04
11	43	1.33	1.33	1.38	1.32
12	46	1.41	1.31	1.33	1.22
13	42	2.02	1.95	2.01	2.23
14	40	1.93	2.25	2.17	2.62
15	38	1.76	1.80	1.84	1.97
16	36	1.53	1.50	1.50	1.65
17	27	2.23	1.96	1.97	2.07
18	24	1.31	1.25	1.27	1.15
19	21	1.20	1.18	1.16	1.27
20	19	1.28	1.26	1.23	1.26
21	22	1.46	1.25	1.26	1.52
22	20	1.34	1.27	1.37	1.30
23	21	1.83	1.77	1.79	1.95
24	15	1.64	1.66	1.75	1.55
25	17	1.93	2.16	2.22	2.60
26	17	3.17	3.53	3.22	3.95
27	14	1.49	1.49	1.47	1.49
28	14	2.65	1.84	1.89	1.66
29	15	1.30	1.25	1.23	1.45
30	15	1.05	0.95	0.96	0.95
31	13	1.19	1.22	1.24	1.34
32	14	1.54	1.55	1.43	1.67
33	12	1.21	1.27	1.24	1.27
34	12	0.96	0.98	0.99	1.05
35	11	1.86	1.45	1.50	1.48
36	11	1.21	1.16	1.15	1.30
37	9	1.42	1.44	1.42	1.55
38	9	1.14	1.17	1.15	1.18
39	9	1.17	1.07	1.01	0.94
40	9	1.20	1.18	1.21	1.11
41	8	1.03	1.13	1.10	1.30
42	8	1.32	1.33	1.33	1.34
43	8	1.34	1.35	1.35	1.42
44	7	1.16	1.21	1.19	1.25
45	7	1.89	1.80	1.89	1.93
46	5	1.47	1.44	1.49	1.47
47	5	1.22	1.20	1.15	1.21
48	5	2.55	2.48	2.55	2.49
49	5	1.70	1.75	1.73	1.84
50	5	1.10	1.07	1.10	0.97
51	5	1.59	1.61	1.60	1.66
52	5	0.85	0.81	0.82	0.62
53	5	1.45	1.50	1.47	1.75
54	4	2.91	3.30	2.97	3.43
55	5	0.85	0.83	0.84	0.78
56	4	1.44	1.44	1.43	1.47
57	4	0.94	0.96	0.98	1.16
58	3	1.13	1.11	1.13	1.02
59	4	1.17	1.16	1.16	1.16
60	4	0.88	0.88	0.88	0.89
61	4	2.00	1.97	1.96	1.97
62	4	2.09	2.14	2.10	2.15
63	4	3.68	4.99	4.17	5.61

64	4	1.10	1.06	1.05	1.22
65	3	2.32	2.42	2.34	2.44
66	4	1.46	1.57	1.63	1.21
67	4	1.01	0.99	1.01	1.04
68	4	0.89	0.89	0.89	0.94
69	4	1.42	1.59	1.47	2.01
70	4	1.29	1.26	1.28	1.27
71	4	1.18	1.20	1.18	1.26
72	4	3.12	3.15	3.19	3.73
73	3	1.10	1.12	1.10	1.18
74	2	2.23	2.30	2.24	2.76
75	2	0.78	0.78	0.78	0.79
76	3	3.87	4.37	4.04	5.14
77	3	1.97	2.09	2.02	2.34
78	3	1.50	1.50	1.50	1.50
79	4	2.13	1.95	2.11	1.86
80	2	1.47	1.47	1.47	1.48
81	2	4.21	4.45	4.23	4.46
82	3	2.16	2.33	2.25	2.42
83	3	1.35	1.36	1.35	1.39
84	2	1.43	1.43	1.43	1.39
85	2	1.10	1.10	1.10	1.08
86	2	1.85	1.79	1.84	1.74
87	2	0.75	0.75	0.75	0.75
88	2	0.80	0.80	0.80	0.80
89	1	2.53	2.53	2.53	2.53
90	2	0.84	0.85	0.84	0.90
91	2	1.29	1.31	1.29	1.39
92	2	2.00	2.09	2.00	2.11
93	1	2.72	2.72	2.72	2.72
94	2	1.74	1.75	1.73	1.77
95	2	1.39	1.39	1.39	1.40
96	2	2.23	2.18	2.22	2.16
97	2	3.00	3.00	3.00	3.00
98	2	1.21	1.21	1.21	1.18
99	2	1.17	1.22	1.18	1.44
100	2	0.85	0.84	0.84	0.84
101	2	1.27	1.34	1.28	1.63
102	2	0.81	0.81	0.81	0.81
103	2	1.33	1.31	1.33	1.23
104	2	1.28	1.29	1.28	1.33
105	2	2.81	2.79	2.81	2.79
106	1	3.12	3.12	3.12	3.12
107	2	0.85	0.85	0.85	0.92
108	2	0.80	0.80	0.80	0.78
109	2	2.20	2.14	2.19	1.79
110	2	2.81	2.58	2.80	2.39
111	2	1.21	1.18	1.20	0.97
112	2	1.46	1.49	1.46	1.58
113	2	0.82	0.81	0.81	0.80
114	2	0.80	0.80	0.80	0.80
115	2	0.81	0.81	0.81	0.81
116	2	1.11	1.11	1.11	1.11
117	2	1.87	1.89	1.88	1.88
118	2	0.89	0.89	0.89	0.89
119	2	1.53	1.52	1.53	1.48
120	2	0.85	0.85	0.85	0.85
121	1	4.64	4.64	4.64	4.64
122	1	1.82	1.82	1.82	1.82
123	1	6.51	6.51	6.51	6.51
124	1	1.52	1.52	1.52	1.52
125	1	1.87	1.87	1.87	1.87
126	1	2.18	2.18	2.18	2.18
127	1	1.85	1.85	1.85	1.85
128	1	1.64	1.64	1.64	1.64
129	1	4.04	4.04	4.04	4.04
130	1	1.58	1.58	1.58	1.58
131	1	2.30	2.30	2.30	2.30

132	1	1.57	1.57	1.57	1.57
133	1	1.59	1.59	1.59	1.59
134	1	3.42	3.42	3.42	3.42
135	1	3.00	3.00	3.00	3.00
136	1	4.29	4.29	4.29	4.29
137	1	1.50	1.50	1.50	1.50
138	1	1.66	1.66	1.66	1.66
139	1	1.55	1.55	1.55	1.55
140	2	1.31	1.31	1.31	1.31
141	1	4.25	4.25	4.25	4.25
142	36	1.28	1.25	1.22	1.29

Gora5 - Au Mean Grade Analysis by Domain

Domain	Composites		Block Model		
	No. of Samples	Mean	IDW3	OK	NN
1	269	2.40	2.38	2.40	2.47
2	247	2.15	2.06	1.97	2.17
3	50	1.86	2.03	2.00	1.90
4	28	1.91	1.92	1.87	1.96
5	24	1.17	1.08	1.06	1.08
6	25	1.38	1.39	1.38	1.47
7	22	3.19	3.14	3.13	2.50
8	19	3.35	3.65	3.66	3.60
9	19	1.15	1.14	1.19	1.18
10	10	1.64	1.62	1.63	1.54
11	10	1.31	1.29	1.27	1.36
12	4	1.13	1.16	1.28	1.34
13	4	0.99	1.00	1.01	1.04
14	3	1.23	1.19	1.29	1.19
15	3	2.72	2.12	2.49	2.10
16	2	1.18	1.17	1.18	1.00
17	2	1.26	1.25	1.25	1.25
18	1	2.10	2.10	2.10	2.10
19	1	2.87	2.87	2.87	2.87
20	2	1.05	1.03	1.05	0.91
21	2	0.84	0.85	0.84	0.95
22	2	0.96	0.95	0.95	0.93
23	2	2.24	2.18	2.22	1.74
24	2	1.23	1.21	1.23	1.09
25	2	1.59	1.58	1.59	1.53
26	1	2.28	2.28	2.28	2.28
27	1	2.36	2.36	2.36	2.36
28	1	1.50	1.50	1.50	1.50
29	1	2.08	2.08	2.08	2.08

Zhelanny - Au Mean Grade Analysis by Domain					
Domain	Composites		Block Model		
	No. of Samples	Mean	IDW3	OK	NN
1	123	2.39	2.44	2.53	2.49
2	193	1.99	1.96	1.99	1.94
3	141	2.36	2.14	2.21	2.21
4	31	1.22	1.13	1.22	1.27
5	8	1.28	1.27	1.31	1.32
6	8	1.84	2.01	1.97	2.13
7	6	1.94	1.97	2.01	2.10
8	4	1.75	1.78	1.76	1.70
9	2	2.17	2.16	2.16	2.16
10	2	2.68	2.75	2.70	3.31
11	2	2.23	2.24	2.23	2.32
12	1	2.29	2.29	2.29	2.29
13	1	3.75	3.75	3.75	3.75
14	1	2.56	2.56	2.56	2.56
101	30	45.69	30.20	-	27.70
102	23	26.63	32.65	-	26.43
201	11	61.08	24.73	-	28.47

Pravoberezhny - Au Mean Grade Analysis by Domain					
Domain	Composites		Block Model		
	No. of Samples	Mean	IDW3	OK	NN
1	69	1.47	1.68	1.73	1.74
2	51	2.97	2.32	2.29	2.42
3	44	2.32	1.95	1.87	1.83
4	36	2.05	2.28	2.31	2.15
5	22	2.12	1.88	1.91	2.08
6	13	1.01	1.01	1.04	1.15
7	14	1.78	1.67	1.69	1.65
8	14	1.95	1.65	1.66	1.70
9	13	1.46	1.49	1.40	1.54
10	7	1.15	1.13	1.15	1.12
11	7	2.26	2.33	2.02	2.41
12	7	1.96	1.81	1.95	1.98
13	6	1.03	0.98	1.00	1.07
14	5	1.63	1.69	1.68	1.71
15	4	1.50	1.80	1.58	1.89
16	4	1.22	1.10	1.09	1.42
17	3	2.54	2.74	2.68	2.91
18	3	1.55	1.49	1.59	1.42
19	2	1.03	1.01	1.03	0.90
20	1	6.34	6.34	6.34	6.34
21	2	1.56	1.56	1.56	1.56
22	2	2.48	2.59	2.50	3.46
23	1	2.33	2.33	2.33	2.33
24	1	1.97	1.97	1.97	1.97
25	1	1.67	1.67	1.67	1.67
26	1	1.55	1.55	1.55	1.55
27	1	1.90	1.90	1.90	1.90
28	1	4.66	4.66	4.66	4.66
29	1	4.51	4.51	4.51	4.51
30	1	3.14	3.14	3.14	3.14
31	1	2.07	2.07	2.07	2.07
32	1	2.53	2.53	2.53	2.53

wardell-armstrong.com

STOKE-ON-TRENT
Sir Henry Doulton House
Forge Lane
Etruria
Stoke-on-Trent
ST1 5BD
Tel: +44 (0)845 111 7777

CARDIFF
22 Windsor Place
Cardiff
CF10 3BY
Tel: +44 (0)292 072 9191

EDINBURGH
Suite 2/3
Great Michael House
14 Links Place
Edinburgh
EH6 7EZ
Tel: +44 (0)131 555 3311

GREATER MANCHESTER
2 The Avenue
Leigh
Greater Manchester
WN7 1ES
Tel: +44 (0)194 226 0101

LONDON
Third Floor
46 Chancery Lane
London
WC2A 1JE
Tel: +44 (0)207 242 3243

NEWCASTLE UPON TYNE
City Quadrant
11 Waterloo Square
Newcastle upon Tyne
NE1 4DP
Tel: +44 (0)191 232 0943

SHEFFIELD
Unit 5
Newton Business Centre
Newton Chambers Road
Thorncliffe Park
Chapeltown
Sheffield
S35 2PH
Tel: +44 (0)114 245 6244

TAUNTON
Suite E1
Victoria House
Victoria Street
Taunton
Somerset
TA1 3JA
Tel: +44 (0)182 370 3100

TRURO
Baldhu House
Wheal Jane Earth Science Park
Baldhu
Truro
TR3 6EH
Tel: +44 (0)187 256 0738

WEST BROMWICH
Thynne Court
Thynne Street
West Bromwich
West Midlands
B70 6PH
Tel: +44 (0)121 580 0909

International offices:

ALMATY
29/6 Satpaev Avenue
Hyatt Regency Hotel
Office Tower, 7th Floor
Almaty
Kazakhstan
050040
Tel : +7(727) 334 1310

MOSCOW
Office 4014
Entrance 2
21/5 Kuznetskiy Most St.
Moscow
Russia
Tel: (495)626-07-67

Wardell Armstrong Archaeology:

CUMBRIA
Cocklakes Yard
Carlisle
Cumbria
CA4 0BQ
Tel: +44 (0)122 856 4820

your earth our world

wardell armstrong