SECTION D: RUSSIAN FEDERATION ASSETS

Wardell Armstrong International (WAI) has been commissioned by Nord Gold (Nord Gold) to undertake a review of the current status of eight properties, situated in south eastern Siberia, Russian Federation, comprising:

Berezitovy – Amurskaya Region
Irokinda - Buryatia Republic
Zun-Holba - Buryatia Republic
Tabornoe – Yakutia Region
Aprelkovo – Zabaikalsky Region
Gross – Yakutia Region
Prognoz – Yakutia Region
Nercha – Zabaikalsky Region
Uryakh – Irkutsk Region, Bodaybinskiy District (on the border of the Buryatia Republic)

As part of this on-going work WAI has undertaken site visits in order to review the current exploration, mining and processing practices at all operations including the exploration programme, the laboratory analysis and test work, through to methods and procedures of recording of information and resource estimation, in order to assess the data quality and possibility to classify the Mineral Resources and Ore Reserves of the Project in accordance with the guidelines of the JORC Code (2004).

1 BEREZITOVY

1.1 Location, Access & Infrastructure

1.1.1 Location

The Berezitovy property is located in the Amurskaya Region of the Russian Federation; 700km north of Blagoveshensk (see Figure 1.1 and Figure 1.2 below).

Figure 1.1: Location of Berezitovy in Far East Russia

Figure 1.2: Location of Berezitovy in Amurskaya Region Far East Russia

1.1.2 Access

The main road access to the site is from the railway station in the town of Scovorodino (a distance of approximately 128km), and then north-westwards from the village of Madalan on the Trans Siberian railway along the mine road, a distance of some 65km.

The route allows access for trucks and vehicular traffic to service the mine and has required construction of a 100m span bridge over the Khaikta River.

> ***WAI Comment:*** *although somewhat remote, the mine is served by excellent road communications which at the time of the visit, were in good condition.*

1.1.3 Topography and Climate

Berezitovy is located in a mountainous region with elevations ranging from 500-1,300masl. Outcrops are rare and overburden thicknesses range from 2-10m.

The average mean temperature for the region is -3.6°C. Summer temperatures can reach 36°C (average 25.5°C). The warmest month of the year is July. Winter temperatures can be as low as -50°C with the coldest month being January. The average winter temperature is -30°C, whilst the January average temperature is -34.7°C.
Average annual precipitation is 600mm with approximately 80% falling in July and August and minimum amounts in January to February. The average snow cover is 30 to 50cm in forested areas and up to 100cm on windward slopes. Snow cover lasts for 165 to 180 days per year.

Average annual wind direction is from the north and north-east, with wind speeds averaging 7m/sec (25km/hr).

1.1.4 Seismicity

Berezitovy is in a seismically active area where earthquakes of magnitude 7 on the Richter scale have been recorded. Buildings in this area are therefore designed and built to Russian Federation requirements, which the project civil engineering consultants indicate equates to international building code UBC.

1.1.5 Infrastructure

Power is provided to the site from the main substation at Skovorodino through construction of a 101km, 110kW power line. Backup power is provided at site to maintain essential power requirements in the event of a power failure.

A water reservoir and potable water wells are located on the Oroghzan river; other elements of infrastructure include sanitary landfill and a septic system for sewage treatment

A location plan showing the main infrastructure is given in Figure 1.3 below.

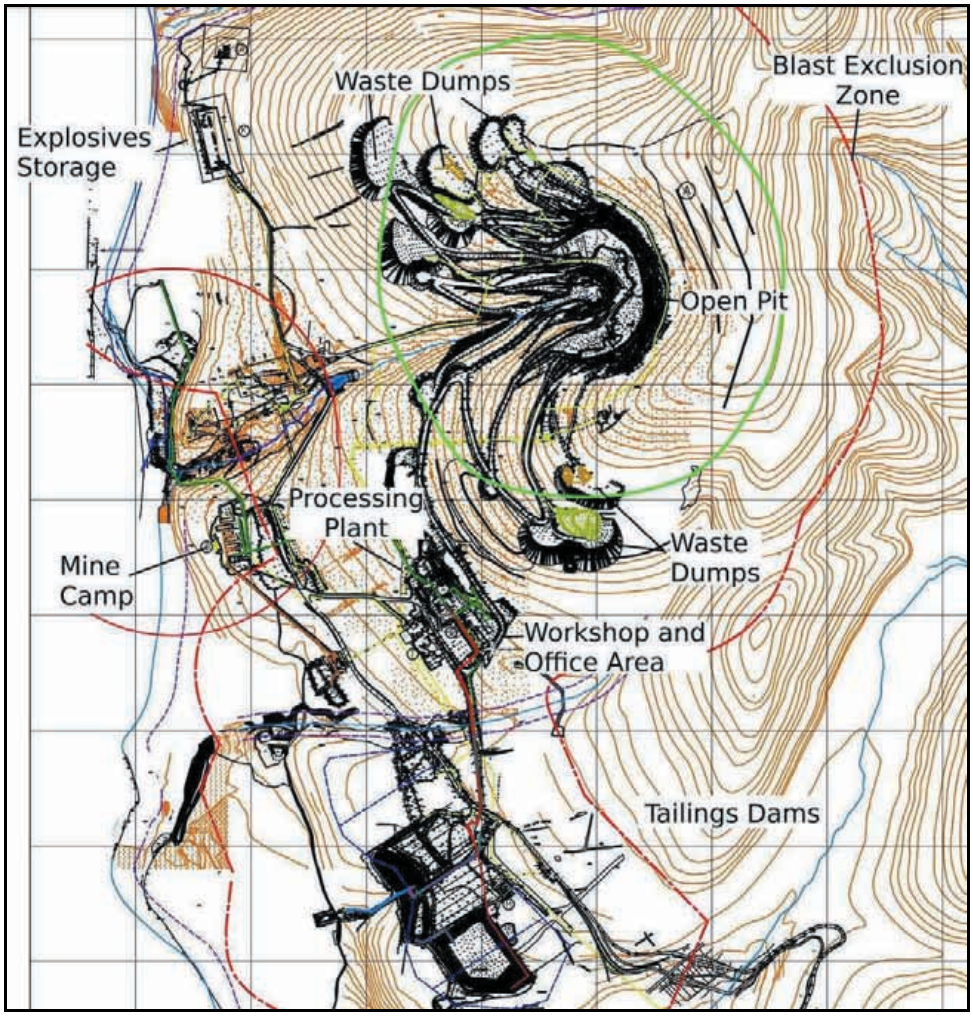


Figure 1.3: Plan showing the Location of the Main Elements of Infrastructure

1.1.6 Mineral Rights and Permitting

In November 2008, Nord Gold took a controlling stake in High River Gold Mines Ltd., a Canadian company who had a 99% ownership (Amur regional government has a 1% stake) of the Berezitovy Project.

The Berezitovy property comprises one Mineral Licence (BLG 11787-BR) covering a total area of 17km^2. The Mineral Licence (in accordance with Russian Mining Law) is in good standing, allowing exploration and mining to be carried out until 2017. The Mineral Licence area for the Berezitovy property is shown in Figure 1.4 below.

Mining Licence (Contract) No.1813 from 09.10.03 covers mining of Berezitovy deposit only, whilst the adjacent exploration areas such as Trubnaya, Yuzhnaya and Beregovaya are in the process of land allotment approval.

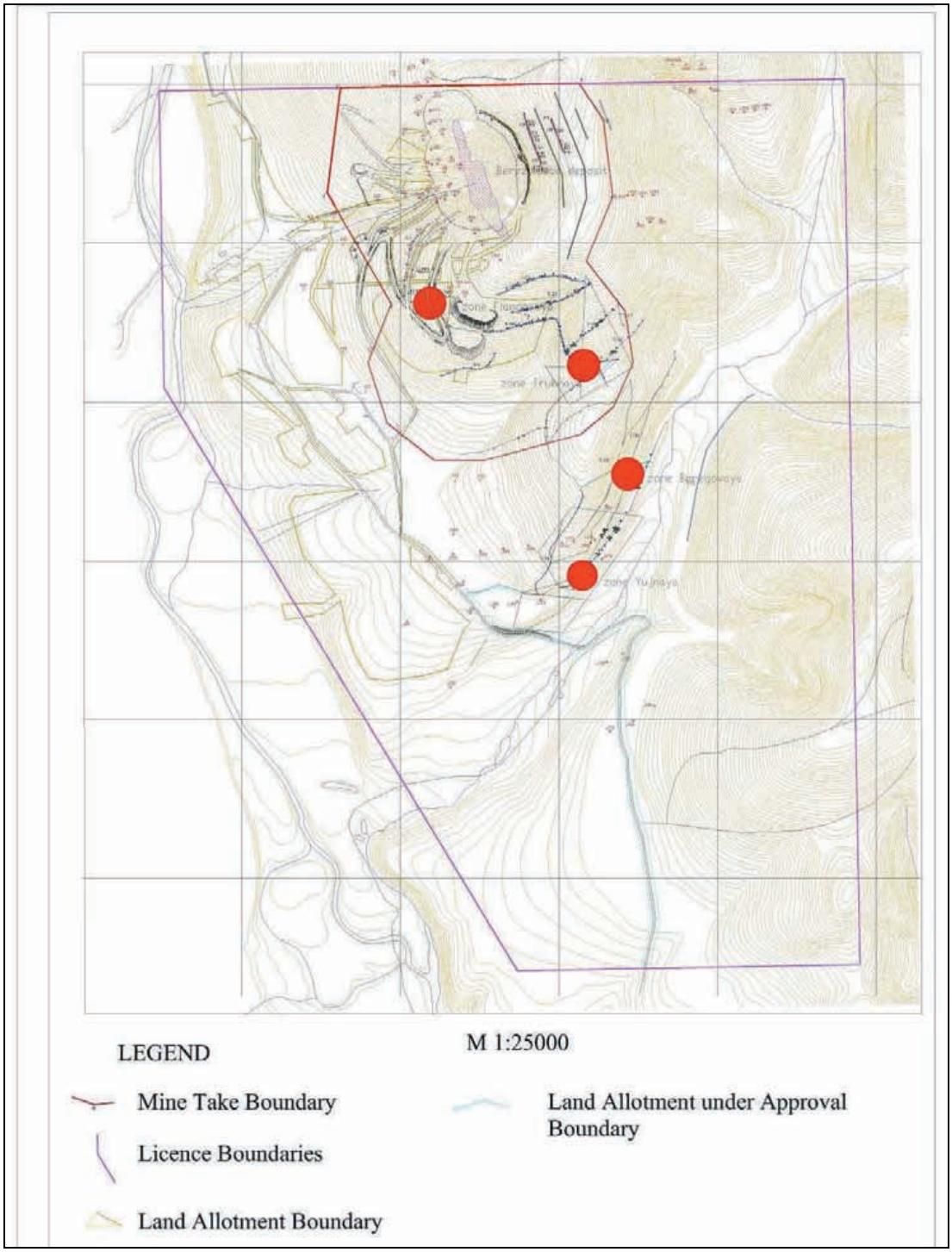


Figure 1.4: Position of the Licence Boundaries

Co-ordinates of the Berezitovy licence are given in Table 1.1 below.

Table 1.1: Licence Co-ordinates		
11787-BR Licence		
Point	**Northing**	**Easting**
1	54°29'00"	122°57'30"
2	54°29'00"	123°01'30"
3	54°26'00"	123°01'30"
4	54°26'00"	122°59'30"
5	54°28'00"	122°57'30"

The Sergachinskaya area, where some 6 identified exploration targets are located, is covered by licence BLG 14149-BR. The licence, allocated to 6 separate areas (Videnovskiy, Koloktikan, Perevalniy, Pravoberezhniy, Lazarevskiy and Kolbachi), covers an area of 162.7km^2, is valid until 2032 and allows both exploration and subsequent mining activities at the properties.

1.2 Project History

1.2.1 Background

The Berezitovy Deposit was discovered in 1932 during the development of gold placer deposits in the nearby Konstantinovsky Stream. In the 1950's the deposit was again studied, but as a polymetallic deposit to primarily exploit its base metal content. From 1960 to 1962 and from 1974 to 1980, Amurskaya Geological Expedition (AGE) undertook geological mapping, surface and underground exploration and evaluation of the Berezitovy Deposit.

1.2.2 Surface Exploration

1.2.2.1 Geophysical Surveys

In 1980, AGE carried out a combined airborne magnetic and radiometric survey and a ground induced polarization (IP) geophysical survey over the entire property. A magnetic low coincided with a strong IP anomaly detected over the Berezitovy deposit area but other magnetic anomalies are poorly correlated with the regional geology.

1.2.2.2 Geochemical Surveys

In 1980, AGE carried out regional geochemical soil sampling over the whole property with survey lines spaced 400m apart. Anomalous concentrations of up to 30ppb Au were detected in several areas the most prominent of which was 250m by 500m, coinciding with the main Berezitovy deposit.

1.2.2.3 Surface Drilling

From 1975 to 1980, AGE carried out diamond drilling to outline the gold and silver mineralised zones of the Berezitovy. In total, some 18,700m were completed in 59 inclined holes oriented due east.

1.2.2.4 Surface Trenching

Some 40 trenches totalling 2,750m and spaced 15-40m apart were excavated across the deposit.

1.2.2.5 Underground Exploration

From 1975 to 1980, AGE carried out detailed chip and channel sampling along cross-cuts. A total of 7,194m of underground development, in 61 cross-cuts was completed. To explore the orebodies across their true thickness, cross-cuts were driven from the adits and drifts 3 to 8m into the mineralised zone. In areas of complex morphology and uneven spacial distribution of mineralisation, the cross-cuts spacing was reduced to 20m. Up dip continuity of mineralisation was checked with 4 raises driven half way between adit levels (40m plane of lode).

AGE also carried out underground diamond drilling from many of these cross-cuts with sampling of the drill holes at intervals varying from 1-2m. Underground diamond drilling totalled 8,655m in 115 holes and mineralised zones were explored to the 516m level.

1.2.2.6 Further Work

Buryatzoloto completed underground sampling in cross-cuts off Adit No. 5 for metallurgical tests and a 25-hole infill surface diamond drilling programme totalling 4,644m.

In 2002-2003, Buryatzoloto carried out the surface angled infill diamond drilling programme on a 20 x 20m grid in the southern and central parts of the deposit (Zone 1), and on a 40 x 40m grid in the northern part of the deposit (Zone 2). Buryatzoloto also continued to collect channel samples from trenches and underground workings along with core samples from the infill drilling programme.

1.3 Geology and Mineralisation

1.3.1 Regional Geology

Berezitovy, located in the Prishulkino Structural-Metallogenic Zone, is situated within the Magochinskaya thrusted block of Archean age rocks and is near the intersection of Sergachinski and Khaiktinski regional faults.

Structurally, the region is characterised by Precambrian, Mesozoic and Quaternary events. The Precambrian and Mesozoic rocks have been intruded by several Jurassic age plutons, ranging from alaskites to lamprophyres. The folded volcanic and sedimentary rocks have also been affected by block faulting.

A regional map of the Sergachinsk ore field showing the position of the main gold occurrences is shown in Figure 1.5 below.

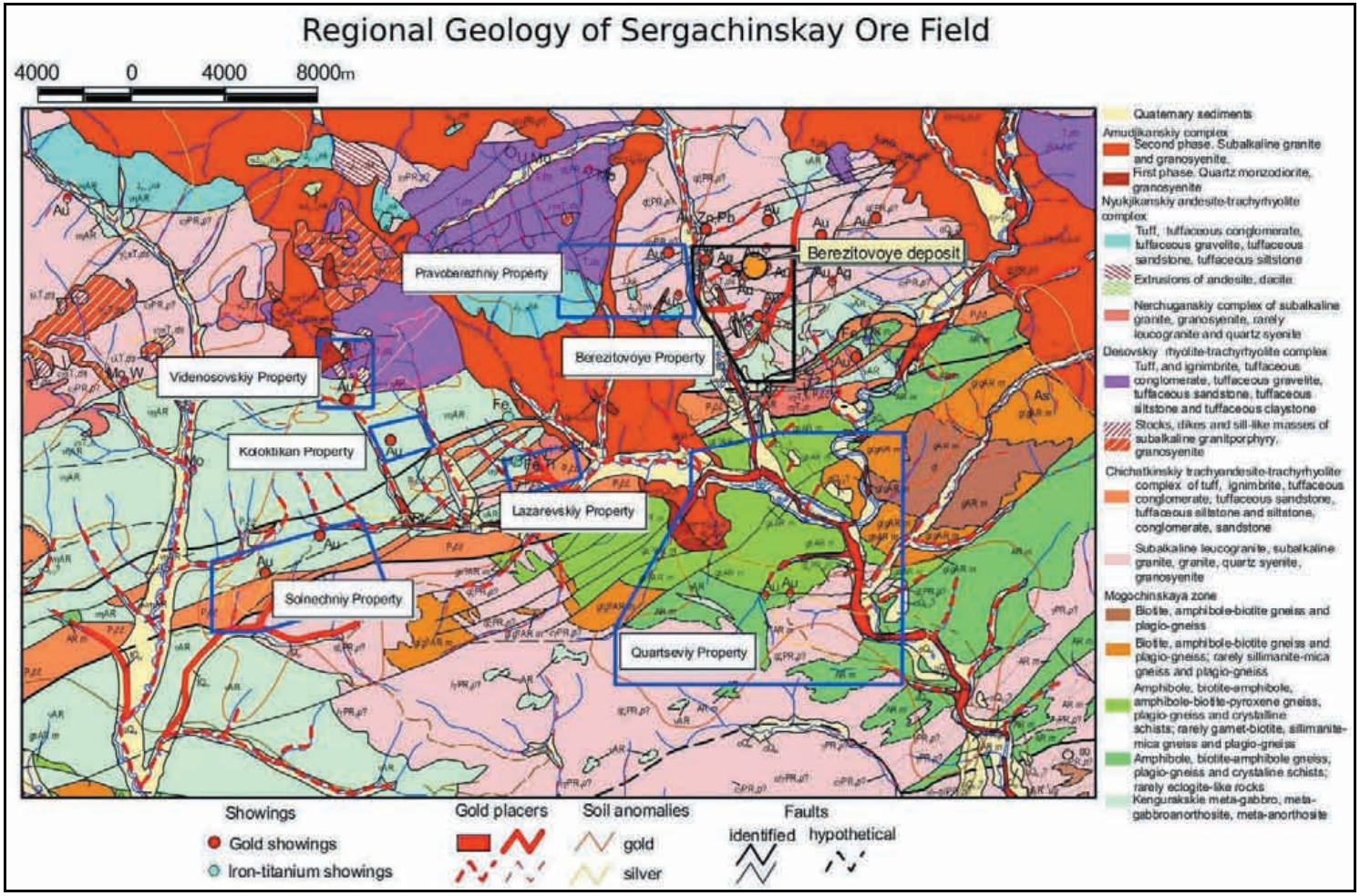


Figure 1.5: Regional Geology and Position of the Main Gold Deposits

1.3.2 Mine Geology & Mineralisation

1.3.2.1 Mine Geology

The property is fault bounded (see Figure 1.6 below), the east-northeast trending North and South Sergachinski faults defining a graben, whereas the north trending Khaiktinski and Bolshe Ilichinski faults define an upthrust block.

The Berezitovy deposit is underlain predominantly by early Proterozoic biotite-feldspar gneissic granites and granodiorites.

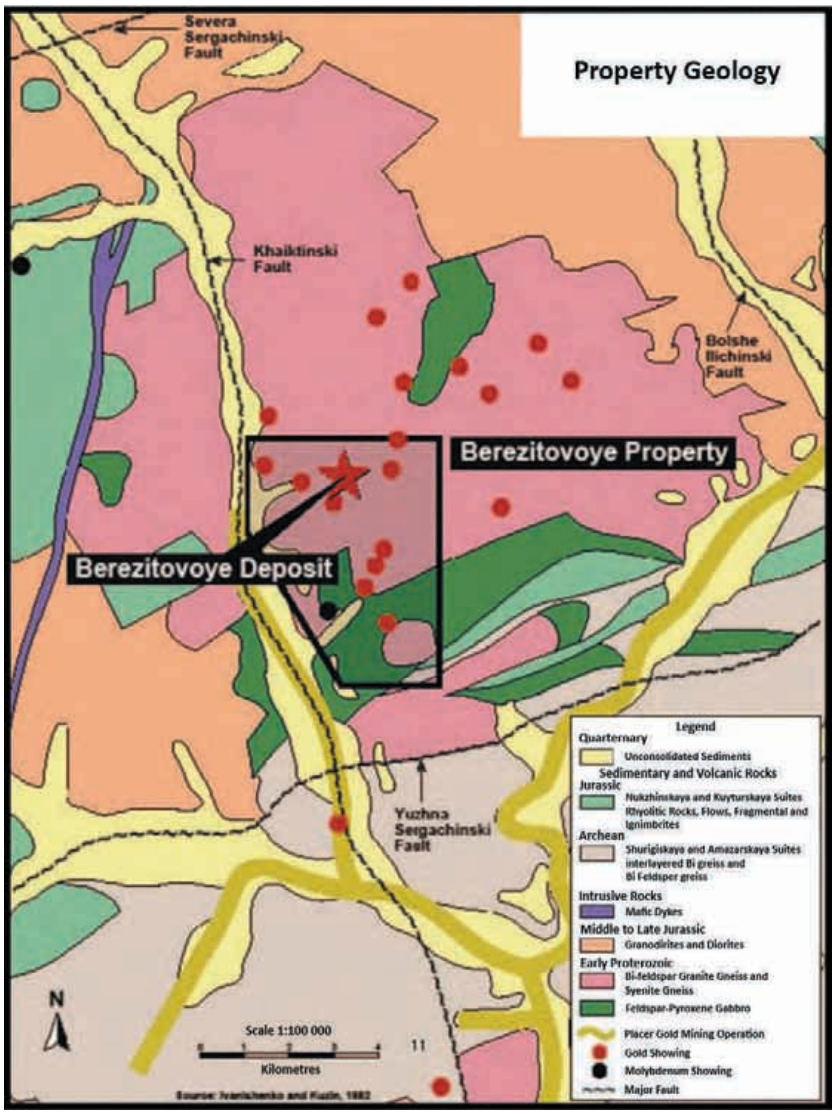


Figure 1.6: Property Geology

The main gold occurrences on the property are shown in Figure 1.7 below:

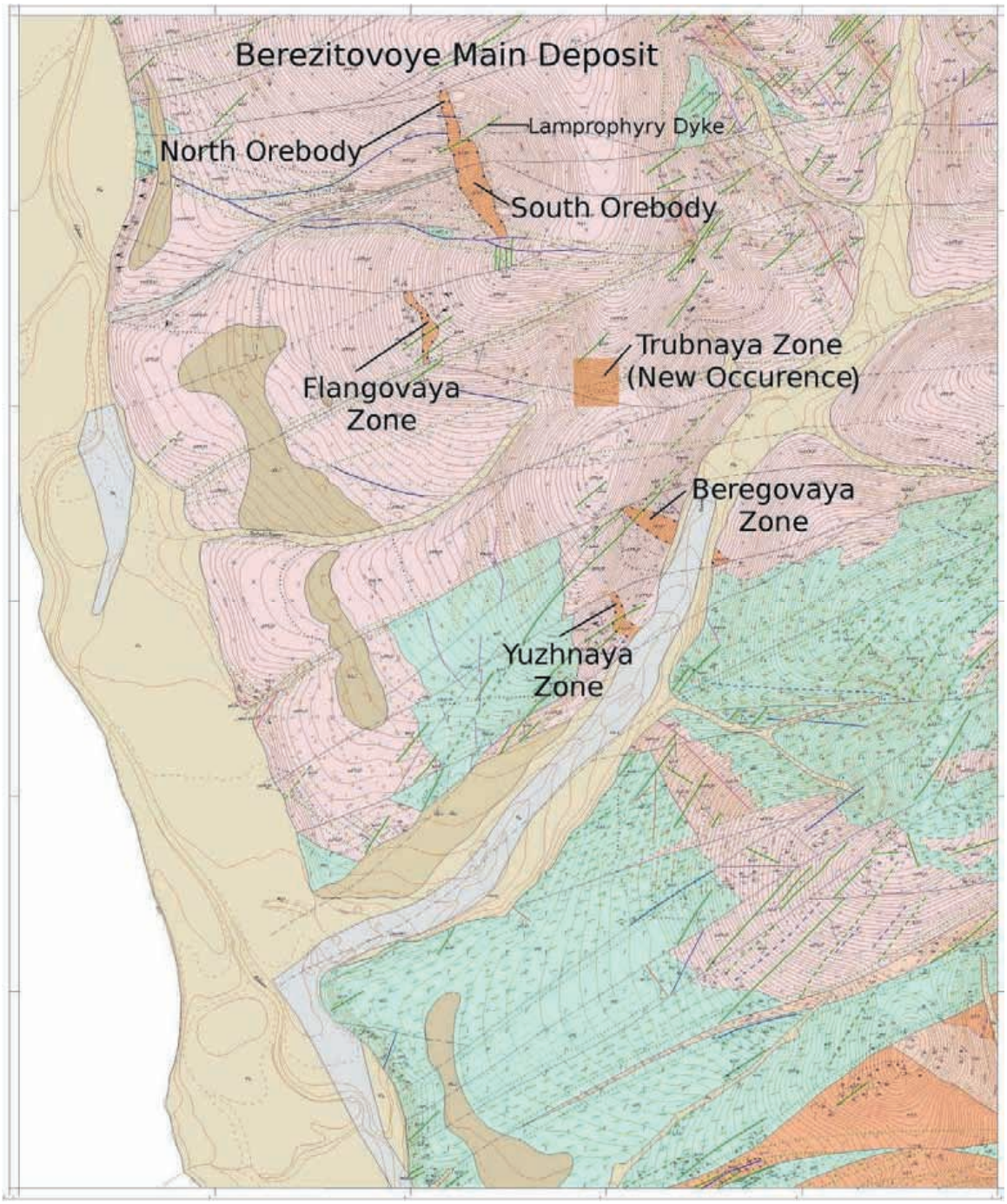


Figure 1.7: Location of the Main Gold Occurrences on the Property

Gold mineralisation in and around the Berezitovy property is related to explosive breccia within granitic gneisses. This is present within a north-northwest trending and steeply southwest dipping zone of brecciated and hydrothermally altered granodiorite.

Gold mineralisation is also associated with metasomatic alteration and quartz flooding in granitic and granodioritic rocks. The contact between the granodirite and the main metasomatic alteration zone in South Zone can clearly be identified in the open pit (Photo 1.1 below).

Photo 1.1: Contact of Main Zone of Metasomatic Alteration in South Zone with the Granodiorite - View Looking North West

The near surface oxidation zone is very shallow (5-7m deep) and mineralisation throughout is predominantly sulphides.

Higher gold values, however, commonly occur along the dyke contacts and some gold mineralisation occurs in the dykes.

A major shear (Miloniyovaya zone) has been identified running east to west through the central section of the open pit. The zone of mylonitisation is strongly sheared, contains pyrrhotite and pyrite with quartz veinlets and has a microfolded structure, together with hydroxides on clay joints. The structure is typically 6-12m wide and is known to carry elevated gold grades, up to 18.3g/t Au.

1.3.2.2 Mineralisation

At Berezitovy, gold is associated with polymetallic sulphides and quartz-sericite (berezite) metasomatic alteration. Locally, tourmaline, garnet and epidote are also common. The overall outline of the mineralised zone is due to the juxtaposition of two inverted cone shaped structures (breccia pipes), which have provided channel ways for the hydrothermal alteration and the associated gold-polymetallic mineralisation.

Gold mineralisation, commonly in the range of 0.5-15g/t Au, is present in various facies of brecciated zones with disseminated sulphides and in silicified rocks. Sulphide mineralisation consists predominantly of pyrite, sphalerite and galena. Gold is commonly present in solid solution with silver as electrum.

Metal zoning is present within the deposit although mineralogical studies (on diamond drill core) have shown that there is very little visible and mineralogical difference in mineralised material collected from the different zones.

High grade portions of mineralisation (greater than 10g/t Au) are uncommon and are present within narrow zones of limited strike extent, commonly 20-80m long and 2-10m wide. Observations based on the surface and underground geological mapping and sampling suggest the following:

- The South Zone resource mineralisation has significantly higher average gold and lead grades and similar average silver and zinc grades when compared to the North Zone;
- It is probable that some cross-faults have displaced the mineralisation vertically;
- Higher gold assays occur along some andesite dyke contacts and this may be due to localized post-mineralisation remobilisation;
- Higher grade gold mineralisation is commonly associated with sulphides; and
- Higher grade gold mineralisation is generally associated with the two "roots" in the southern and northern parts of the deposit. The higher gold, silver, zinc and lead values are centred on these roots and the metal values tend to decrease, progressively, away from these roots.

WAI Comment: *the Berezitovy ore zone is well known through extensive drilling and exploitation. The geological models for the emplacement of mineralisation appear robust and act as a good pointer for further potential mineralisation in the area.*

1.4 Exploration

1.4.1 Introduction

LLC Berezitovy Rudnik (Berezitovy) has title over two principal licence areas within the Sergatchinskaya ore field. These are the exploitation licence around and including the Berezitovy deposit, and the licences encompassing the six satellite mineralised areas, summarised below.

The total exploration budget for the Sergatchinskaya property was approved as US$0.61M for 2010, and for the larger Berezitovy Rudnik allotment, some US$2M. New budgets for 2011 have not yet been allocated.

1.4.2 Perevalniy

1.4.2.1 Introduction

The Perevalniy licence which occupies some 24.5km^2 lies some 20km to the southwest of Berezitovy.

Knowledge of gold occurrences within the Perevalniy area has been in existence for many years, primarily through extraction of placer gold from the adjacent river valleys.

1.4.2.2 Geology

The geology of the area comprises metagabbro-anorthosite and granitised metagabbros of Archean age to the north, trachyandesite and trachyrhyolites in the central part and porphyritic quartz-syenites and granite-syenites to the south.

The area is also cut by numerous dykes of various composition which are believed to play an important role in the mineralisation process.

1.4.2.3 Exploration Works

A major soil geochemical exploration programme on a 100x20m grid was undertaken in 2008 which defined two main clusters of mineralisation. The western part of the area is dominated by a Cu, Mo, W, Ag signature, whilst an Au, As, Pb, Bi anomaly lies to the south and further eastwards.

Channel sampling and trenching have been undertaken at the northeast corner of the licence area where sulphidic gold mineralisation has been uncovered.

1.4.2.4 Results

Outcrop

The small outcrop uncovered at the northeast extremity of the licence area comprises andesites cut by a porphyry dyke, where sulphide rich mineralisation (predominantly pyrite) with a grade of nearly 8g/t Au over 60cm is seen although at present it is unclear how this outcrop fits in with the mineralisation exposed in trenches on the hill to the southwest.

Trenching

The trenching programme in 2009 consisted of 7 trenches. A small outcrop of stockwork mineralisation (N-S and E-W quartz veins) in a porphyritic body was observed which carries visible molybdenite. Results have now been received which revealed one intersection of gold mineralisation of 6m @ 3.79g/t Au in Trench 5. This work also revealed a significant primary gold anomaly within the granosyenite-porphyries of some 1500 x 100m with gold concentrations of 0.1g / t.

The trenching programme in 2010 consisted of 6 trenches, two of which extended the mineralisation identified in 2009 to the northwest and southeast. In the southwest direction, at a distance of 90m, Trench 10 identified gold mineralisation showing 3m @ 1.2g/t. In the northeastern direction, Trench 11 revealed several gold intersections:

- 7m @ 1.08 g/t
- 4m @ 1.72g/t
- 3m @ 0.95g/t
- 3m @ 6.14g/t

The combined mineralised interval in Trench 11 is some 61.8m @ 0.74g/t Au.

Also, Trench 9 identified a new zone of gold mineralisation showing 15.8m @ 1.19g/t.

In total, some 1119.6m (1178 samples) were generated and all test results for gold have been received.

Drilling

In 2009 the site was drilled with 36 exploratory holes. The total volume of drilling was some 3,774m. The results revealed numerous fragmented samples with anomalous gold content, although poorly correlated. The maximum gold content of a single sample was 5.38g/t.

Two holes did trace the mineralisation identified in Trench 5 (6m @ 3.79g/t). At a depth of 50m, an intersection of 1m @ 1.01g/t was obtained whilst at 100m, 6.7m @ 1.56g/t was seen.

1.4.3 Kolbachi

1.4.3.1 Introduction

The Kolbachi licence, which occupies some 105.7km^2, lies some 10km to the south of Berezitovy.

Initial prospecting took place in 1962, followed by geochemical work in 1984 which outlined a number of anomalies defining the licence boundary as seen today.

1.4.3.2 Geology

In general terms, the north half of the licence is composed of Archean gneiss with Jurassic age grano-monzonite in the northwest and Proterozoic granites in the south.

The southwest part of the area looks the most prospective with differentiated gneiss and granite having large silicified areas with pyrite. Jurassic dykes are also present in the area.

1.4.3.3 Results

Although a recent phase of geochemical sampling has taken place, assay results are not available. However, the area remains one of Berezitovy's high priority targets for large volume low grade targets.

In 2010, minor ground geophysical surveys were undertaken in the Oldoy river valley (mouth of Kolbachi creek). A slight anomaly was detected by induced polarization which was traced by drilling to 120m depth revealing intense quartz-feldspar stockworks, but without noticeable gold mineralisation.

1.4.4 Pravoberezhniy

This licence area (17.7km^2) lies immediately to the west of Berezitovy on the other side of the valley. Many of the structures seen appear similar to those at the mine including a strong NE – SW structural trend, mylonites and relicts of Jurassic volcanic rocks.

In general terms, the area comprises Proterozoic gneisses with later Triassic granites. Within this framework, some disseminated sulphide areas have been defined and areas of quartz veining and silicification have also been identified, all overprinting the older rocks suggesting that the volcanic cover may obscure Berezitovy-style mineralisation.

Some historic geochemical and geophysical work has been undertaken with a subsequent trenching programme to test anomalies which has revealed grades up to 5.5g/t Au.

In 2010, two exploratory holes were drilled, with a total of 728m, to test the nature of geophysical anomalies identified by previous work. Boreholes intersected intense stockwork quartz-feldspar with weak pyrite mineralisation. Assay results are planned to be ready in January 2011.

1.4.5 Videnovski

This small licence area in the northwest encompasses some 7km^2. The area is remote and to-date work has comprised reconnaissance mapping, grab sampling (which has given values up to 6.2g/t Au) and stream panning. Previous work on the licence has included geochemistry and limited trenching.

The area comprises Archean gabbro to the south, volcanics to the north and east, and Proterozoic granites and Jurassic monzonite/diorite to the northwest. Felsic volcanic (rhyolitic) domes are also encountered.

The targets in this area comprise alteration zones in younger rocks and vein-type mineralisation in gabbros. To date, only short (<150m) structures have been defined.

1.4.6 Koloktikan

This small licence (3.3km^2), situated in the west of the area, comprises Archean gabbros with Triassic rhyolites. Limited trenching was completed in 1960, 1963 and 1984, and yielded results of some interest.

Mineralisation identified includes quartz veins in gabbros, up to 400m long, but only a few centimetres wide, although with average grades of 14g/t Au. The target is for wider, metasomatic zones around these veins, but as yet, no evidence for this style of mineralisation has been discovered.

1.4.7 Lazarevskii

This licence lies in the centre of the area and encompasses approximately 4.5km^2.

The terrain comprises granite-gneiss with some hornfels zones indicative of contact metamorphism. Gabbros are also present, some areas showing propylitic alteration with pyrrhotite. Alluvial gold deposits appear to coincide with these areas.

Historically, some IP and Resistivity has been done along with limited trenching and two drill profiles up to 350m depth. However, the results from this work were negative.

1.4.8 Berezitovy Exploration Areas

1.4.8.1 Introduction

Geologic evidence suggests that the Berezitovy orebody extends to the south through the Trubny structure to Beregovaya and Yuzhnaya (collectively known as Orogzhan). The Flangovaya prospect may also be part of this extension, although at the present time, waste dumps cover much of the occurrence.

1.4.8.2 Orogzhan

Although the subject of a legal dispute, the Orogzhan project, within 2km of the mine, could be of strategic importance. A sequence of outcropping gabbros and micro-gabbros host a series of dominantly E-W trending tectonic structures which vary in width from a few centimetres to several metres. Pyritised mylonitic zones also occur.

Historically, previous work had concentrated on trying to identify other “Berezitovy-type” structures with N-S trend, thus the trenching was orientated E-W and drilling orientated eastwards at approximately 60°. As a result, historic exploration results may be misleading.

Figure 1.8 shows a number of northerly dipping E-W trending tectonic structures in outcrop at Orogzhan.

1.4.8.3 Trubnaya

This N-S trending prospect lies between Berezitovy mine and Orogzhan and structurally appears to connect the two (Figure 1.9). Trenching and limited drilling have produced some interesting grade and thickness intervals including 6m @ 2.65g/t Au.

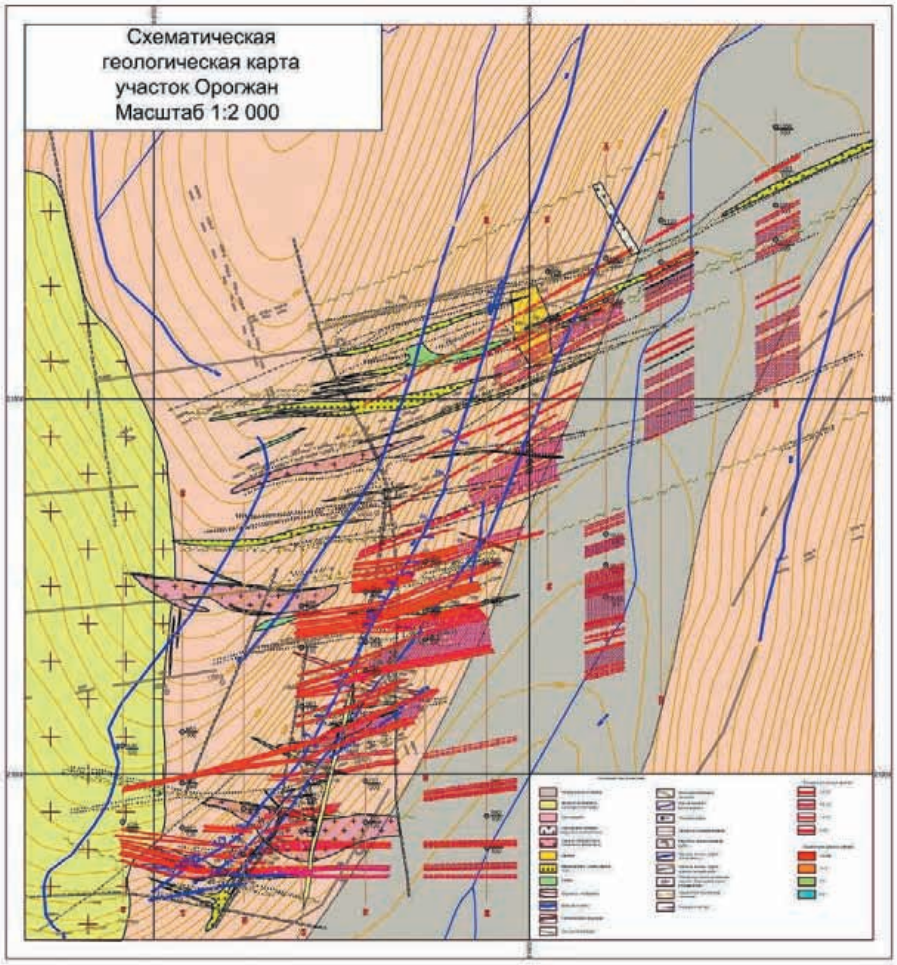


Figure 1.8: Orogzhan Mineralised Zones

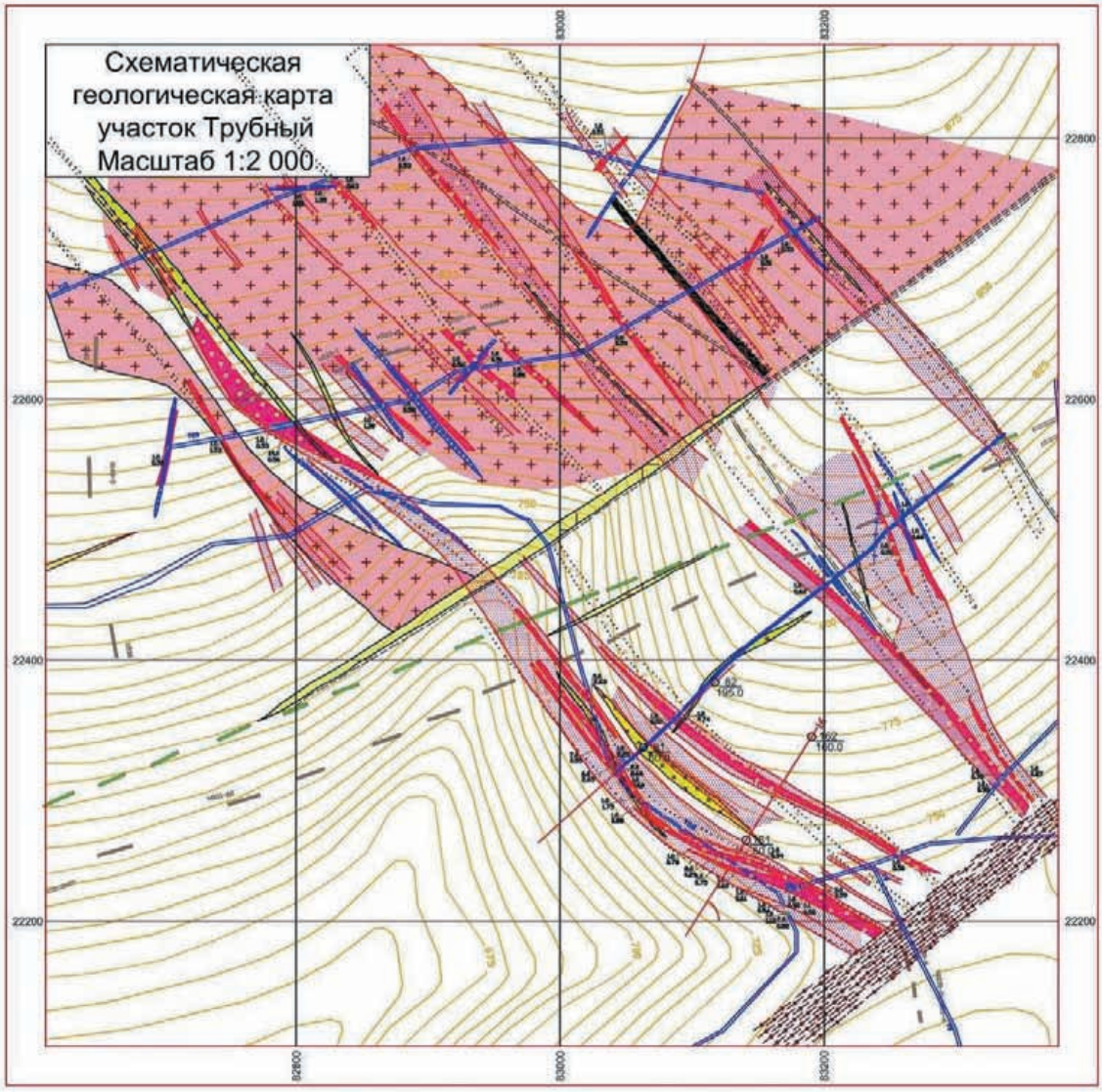


Figure 1.9: Mineralisation and Structure at Trubnaya

1.4.8.4 Berezitovy Deep Horizons

In 2010, a study of the deeper levels of the deposit was undertaken by 34 drillholes (10,880m) with a total budget of US$2M. To date, results are not complete, although some 5,422 samples are in hand giving the following results for both the potential open pit (Table 1.2) and underground mineralisation (Table 1.3).

Table 1.2: Potential Open Pit Mineralised Intersections

Hole-ID	Interval from	Interval to	length	Au g/t	Ag g/t	Cu %	Pb %	Zn %
	225.4	230.5	**5.1**	**2.92**	**28.9**	**0.03**	**0.28**	**0.28**
	245.3	262.8	17.5	0.74	8.10	0.04	0.10	0.09
	269	307.5	38.5	0.74	13.13	0.06	0.16	0.21
1412	313.5	319.7	**6.2**	**2.46**	**13.99**	**0.03**	**0.20**	**0.05**
	262.7	272.8	**10.1**	**2.28**	**12.16**	**0.07**	**0.12**	**0.16**
	272.8	290	17.2	0.76	7.65	0.04	0.09	0.19
	290	300	**10**	**2.06**	**15.48**	**0.31**	**0.09**	**0.26**
	306	331	**25**	**1.37**	**18.91**	**0.18**	**0.16**	**0.27**
1492	331	337	6	0.59	17.93	0.03	0.18	0.25
	245.1	254.4	**9.3**	**1.28**	**11.15**	**0.03**	**0.13**	**0.32**
	254.4	281.2	26.8	0.83	11.10	0.09	0.08	0.11
	281.2	293.4	**12.2**	**1.22**	**11.91**	**0.11**	**0.09**	**0.21**
	293.4	304.6	11.2	0.91	14.45	0.15	0.09	0.10
	311.8	319.5	**7.7**	**1.23**	**10.13**	**0.12**	**0.07**	**0.28**
	330.5	345.7	15.2	0.57	13.85	0.05	0.21	0.37
1452	358.8	367.1	8.3	0.54	16.30	0.02	0.16	0.06
1252	262.6	272.6	10	0.87	27.68	0.03	0.38	0.90
	247.6	266.8	**19.2**	**2.40**	**9.42**	**0.03**	**0.81**	**0.88**
	284.7	300.7	16	0.72	9.45	0.02	0.64	0.74
	308.3	330.7	**22.4**	**6.46**	**13.44**	**0.03**	**1.02**	**0.99**
1081	340	349	9	0.72	3.96	0.02	0.09	0.06
	200.5	206.5	6	0.78	19.12	0.05	0.16	0.94
1251	233.3	237.4	**4.1**	**4.10**	**21.21**	**0.03**	**0.14**	**0.90**
	274.5	294.5	20	0.63	15.08	0.07	0.16	0.19
1413	320.5	328.5	8	0.80	10.80	0.05	0.12	0.29
913	343.6	354.8	**11.2**	**5.83**	**9.93**	**0.02**	**0.39**	**0.51**
1041	259.9	269.9	**10**	**1.11**	**5.33**	**0.01**	**0.26**	**0.11**
1042	347.5	353.5	**6**	**2.32**	**1.98**	**0.02**	**0.12**	**0.55**
	293.7	310	**16.3**	**1.69**	**18.50**	**0.10**	**0.20**	**0.32**
	310	326	16	0.60	9.74	0.07	0.20	0.59
	343	350.8	7.8	0.64	7.20	0.05	0.10	0.07
	350.8	359	**8.2**	**1.18**	**14.00**	**0.06**	**0.16**	**0.16**
1453	359	365	6	0.65	10.40	0.10	0.09	0.11
911	211	216.9	**5.9**	**3.80**	**8.05**	**0.02**	**0.26**	**0.14**
	285.5	291	5.5	0.76	9.06	0.05	0.08	3.49
	297	311	14	0.80	13.45	0.07	0.18	0.22
1414	329	336	7	0.72	6.57	0.05	0.07	0.64

Note: based on minimum 5m ore interval which can include up to 5mof internal waste, 1g/t Au cut-off for ore, 0.5g/t cut-off for off-balance

Table 1.3: Potential Underground Mineralised Intersections								
	Interval							
Hole-ID	from	to	length	Au g/t	Ag g/t	Cu %	Pb %	Zn %
	225.4	228.5	3.1	3.94	41.96	0.03	0.29	0.11
1412	314.5	318.7	4.2	2.70	18.27	0.03	0.21	0.07
	262.7	265.8	3.1	3.57	15.31	0.09	0.13	0.13
	268.7	271.8	3.1	2.78	11.21	0.08	0.06	0.09
	290	293	3	3.29	24.93	0.58	0.16	0.33
1492	297	300	3	2.71	18.40	0.32	0.08	0.38
	256	264.8	8.8	2.77	10.92	0.04	0.95	0.93
	308.3	311.7	3.4	7.21	19.97	0.02	1.33	1.26
	314.9	319.1	4.2	9.32	20.06	0.03	1.42	1.40
1081	324.5	329.1	4.6	12.48	15.24	0.04	1.30	1.05
	226.5	230.4	3.9	6.76	27.09	0.04	0.23	0.60
1251	233.3	237.4	4.1	4.10	21.21	0.03	0.14	0.90
	344.7	348.8	4.1	8.83	9.28	0.05	0.52	0.86
913	351.8	354.8	3	9.12	22.90	0.02	0.69	0.69
1162	404	407.3	3.3	3.54	4.00	0.00	0.06	0.03
911	213	216.9	3.9	4.66	10.61	0.02	0.33	0.14
	276.7	280.4	3.7	4.15	10.89	0.04	0.77	1.01
1001	341	344	3	4.00	1.60	0.01	0.00	0.01
Note: based on minimum 3m mineralisation thickness and inclusive of up to 1m of internal waste, cut-off grade of 2.5g/t Au								

1.4.9 Sample Preparation

WAI understands that all sample preparation and analysis, undertaken in the on-site laboratory (process plant), is adequate for this stage of prospecting work.

1.5 Mineral Resource Estimates

1.5.1 Historical Estimate (RPA 2003)

In 1982, AGE estimated B and C_1 category resources at 14.1Mt with an average grade of 3.0g/t Au, 14.3g/t Ag, 0.93% Zn and 0.57% Pb, using a 1.5g/t equivalent Au cut-off grade (Ivanishenko and Kuzin, 1982).

In May 2003, results of the 2002 to 2003 diamond drilling programme were loaded into a Gemcom database and a mineral resource estimate started. Micon International Limited (Micon) provided a preliminary Whittle pit shell and supporting documentation during August 2003, so that the resource estimate could be constrained at depth. Roscoe Postle Associates (RPA) reviewed the Micon work and provided the mineral resource estimate to HRG, Buryatzoloto and Micon in September 2003.

In the RPA model a 0.5g/t Au cut-off grade was applied to define the mineralised envelope and to construct a 3-D wireframe model of the mineralisation. The 0.5g/t Au cut-off grade was generally effective in defining mineralisation continuity. A minimum thickness of approximately 4m was used to define mineralisation envelopes.

After reviewing the data with Buryatzoloto personnel, RPA selected a 35g/t Au top cut level for Zone 1 resource assays and a 20g/t Au top cut for Zone 2 resource assays before compositing to 3m lengths.

RPA noted that the 2003 density tests varied over a wide range and this probably reflects the variable sulphide content and different rock densities within the deposit. RPA used 2.87t/m³ as the tonnage factor for mineralisation.

The Micon Whittle pit shell was used as a guide to estimate the maximum potentially economic open pit depth and to constrain the resource at depth. Consequently, although the block model contains mineralisation below the Micon preliminary pit shell, only the mineralisation above it was included in the Mineral Resource estimate. The maximum preliminary Whittle pit shell depth is at approximately the 480m level elevation. The final pit together with the solids for the North and South zones are illustrated in Figure 1.10 below.

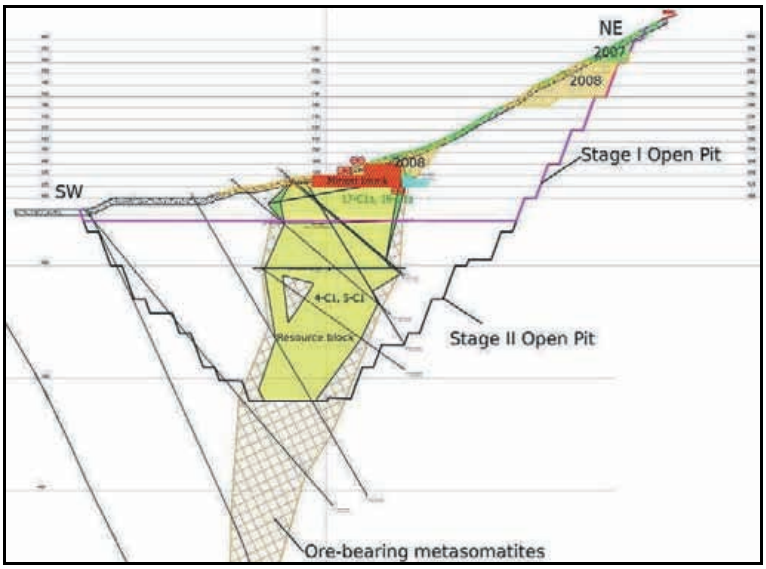


Figure 1.10: Typical Cross Section through the Two Stage Development of the Open Pit

RPA developed new Mineral Resource classifications in which the Indicated Mineral Resources were classified using a primary search radius for Zone 1 of 45x60x10m and for Zone 2 of 40x50x10m. The Inferred Mineral Resources were classified using a secondary search radius of 90x120x20m for Zone 1 and 80x100x20m for Zone 2. Almost all of the Mineral Resources were classified as Indicated. Approximately 2% of the Mineral Resources were classified as Inferred, with the majority located near surface at the southern end of the South Zone.

The Mineral Resource estimate by RPA (September 2003) is presented in Table 1.4 below.

Table 1.4: Berezitovy Mineral Resource Estimate (RPA September 2003)

Block Cut-Off Grade	Tonnes Above Cut-Off Grade	Cut Au* (g/t)	Uncut Au (g/t)	Uncut Ag (g/t)	Contained Cut Au* (oz Aug)	Contained Ag (oz Ag)
Zone 1 *Indicated* Mineral Resources						
1.4g/t Au	8,950,000	3.01	3.46	13.32	866,000	3,832,000
1.2g/t Au	9,500,000	2.91	3.33	13.00	889,000	3,971,000
1.0g/t Au	10,000,000	2.82	3.22	12.71	907,000	4,087,000
0.9g/t Au	**10,200,000**	**2.79**	**3.18**	**12.59**	**915,000**	**4,129,000**
0.8g/t Au	10,340,000	2.76	3.15	12.50	918,000	4,156,000
0.0g/t Au	10,580,000	2.71	3.09	12.34	922,000	4,197,000
Zone 2 *Indicated* Mineral Resources						
1.4g/t Au	2,340,000	2.24	2.42	13.80	168,000	1,036,000
1.2g/t Au	2,900,000	2.06	2.21	13.50	192,000	1,257,000
1.0g/t Au	3,480,000	1.90	2.02	13.18	213,000	1,476,000
0.9g/t Au	**3,840,000**	**1.81**	**1.92**	**12.99**	**223,000**	**1,602,000**
0.8g/t Au	4,040,000	1.76	1.87	12.88	228,000	1,672,000
0.0g/t Au	4,290,000	1.70	1.80	12.78	234,000	1,763,000
Total *Indicated* Resources above 0.9g/t Au	**14,040,000**	**2.52**	**2.84**	**12.70**	**1,138,000**	**5,731,000**
Zone 1 *Inferred* Mineral Resources						
1.4g/t Au	190,000	2.66	2.73	9.81	16,000	59,000
1.2g/t Au	200,000	2.57	2.64	9.65	16,000	62,000
1.0g/t Au	230,000	2.38	2.44	9.18	18,000	68,000
0.9g/t Au	**230,000**	**2.36**	**2.42**	**9.14**	**18,000**	**68,000**
0.8g/t Au	240,000	2.30	2.36	9.03	18,000	70,000
0.0g/t Au	260,000	2.20	2.25	8.73	18,000	73,000
Zone 2 *Inferred* Mineral Resources						
1.4g/t Au	30,000	1.82	2.03	13.26	2,000	14,000
1.2g/t Au	60,000	1.57	1.70	13.68	3,000	25,000
1.0g/t Au	80,000	1.45	1.55	13.89	4,000	34,000
0.9g/t Au	**80,000**	**1.42**	**1.51**	**13.89**	**4,000**	**36,000**
0.8g/t Au	80,000	1.41	1.50	13.81	4,000	37,000
0.0g/t Au	90,000	1.39	1.47	13.71	4,000	38,000
Total *Inferred* Resources above 0.9g/t Au	**310,000**	**2.12**	**2.19**	**10.54**	**21,000**	**105,000**

Notes:
(*) High Zone 1 assays cut to 35g/t Au and High Zone 2 assays cut to 20g/t Au. Silver assays were not cut;
The 0.9 g/t Au block cut-off grade is based on a US$350/oz gold price, US$5.75/t for processing, US$4.08 for G&A and royalties, and 87% gold recovery;
Mineralised wireframe models constructed based on approximately a 0.5g/t Au cut-off grade;
Blocks are 10x10x10m
Ordinary kriging grade interpolation for gold and inverse distance squared for silver;
Mineral Resources constrained at depth using a preliminary Whittle Four-X pit shell based on a US$400/oz gold price and a US$5.00 silver price and 45° pit slopes;
Tonnage factors: 2.87t/m³ for ore; 2.70t/m³ for waste rock; and 2.20t/m³ for overburden;
Gemcom Software International Inc. Resource Evaluation Edition Version 4.21 was used;
Mineral Resource estimate performed by Roscoe Postle Associates Inc; and
Qualified Person Luke Evans, M.Sc., P.Eng.

1.5.2 State Approved Resources

The latest State-approved resource estimation is dated 2004. Since this estimation was approved by GKZ, it has been used as a basis for scheduling and mine planning purposes. A cut-off-grade (COG) of 1.07g/t Au has been obtained for ore blocks (with a block size of 10mx10x10m). A COG of 0.5g/t Au has been used for off-balance ore.

Table 1.5 and Table 1.6 below quote the Berezitovy ore resources for both gold and silver as of 01 January 09, approved by GKZ.

Table 1.5: GKZ-Approved Gold Resources as of 01.01.2009

Resource Category		Balance	Off-balance
C_1	ore (kt)	9,433	2,208
C_1	Au (kg)	28,333	1,623
C_1	Au (g/t)		
C_2	ore (kt)	531	
C_2	Au (kg)	1,273	
C_2	Au (g/t)		

Table 1.6: GKZ-Approved Silver (as co-product) Resources as of 01.01.2009

Resource Category		Balance	Off-balance
C_1	ore (kt)	9,433	2,208
C_1	Ag (kg)	128.4	17.3
C_2	ore (kt)	531	
C_2	Ag (kg)	6.2	

Projected losses for all the resources within the designed open pit were estimated to be in the region of 2.3%; with a projected dilution factor of 5.3%. During the present year the mine has achieved figures for losses and dilution of 2.45% and 5.45% respectively.

> ***WAI Comment:*** *the design of the pit, which is currently used for planning and scheduling, is based on outdated economic parameters. A review of the pit design considering the current gold market situation and up-to-date operating costs and capital expenditures may potentially increase the reserve base.*

1.5.3 Current Mineral Resource Estimate (WAI 01 November 2010)

WAI was commissioned by Nord Gold to prepare a Mineral Resource estimate in accordance with the guidelines of the JORC Code (2004) for the Berezitovy Au deposit, Russian Federation. The work completed during this project is summarised below.

1.5.3.1 Topography

The pit topography supplied for Berezitovy is understood to have been generated from a mine survey carried out on 01 November 2010. The topography is shown in Figure 1.11.

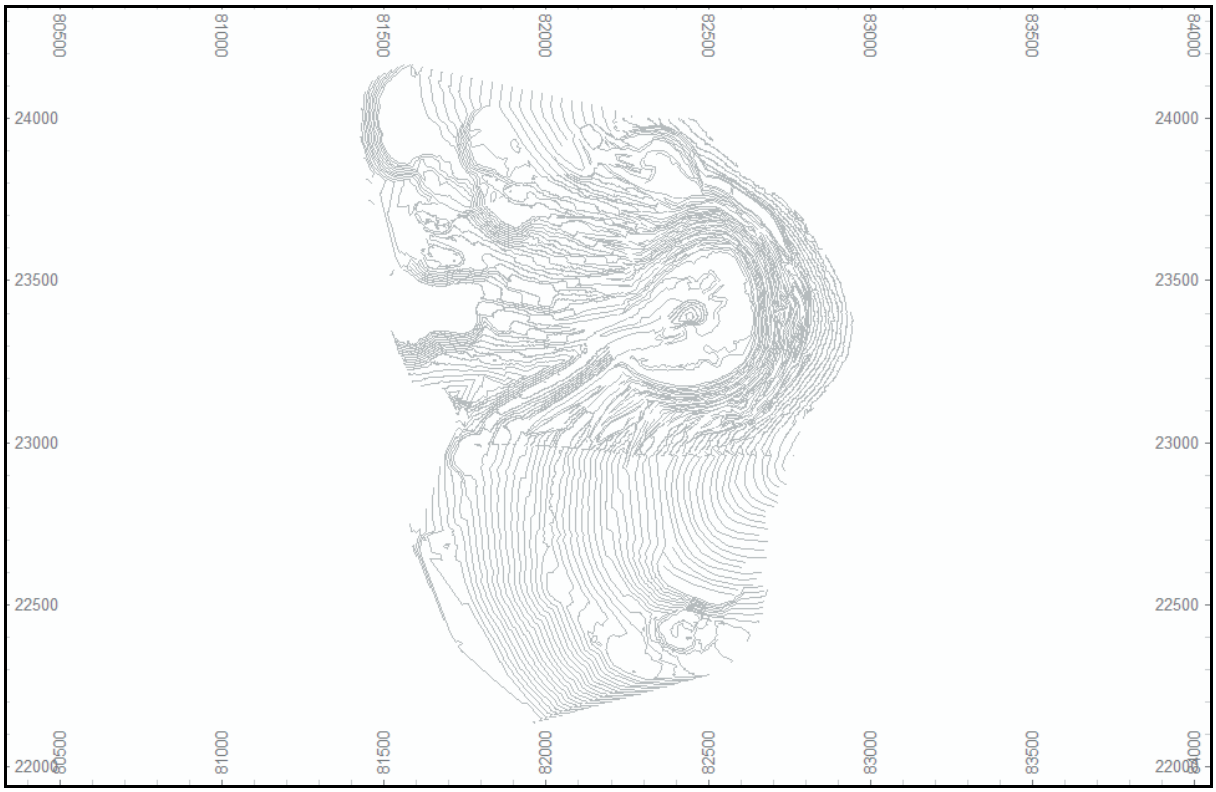


Figure 1.11: 5m Contour Topographical Survey of Berezitivoy Mine as of November 2010

1.5.3.2 Sample Database

The data collected includes diamond drillhole, trenches and underground channel samples, and are summarised below in Table 1.7.

Table 1.7: Sample Data Summary

	No. of Holes/Lines	No. of Samples	Total Length (m)
Diamond Drillholes	140	5,603	23,622
Underground Channels/Trenches	570	5,927	8,788

1.5.3.3 Mineralised Zone Modelling

Previous interpretations had been based on a gold cut-off grade of 0.5g/t Au. However, with current gold prices, economic cut-off grades lower than this, are quite possible. A contiguous sample length analysis shows that a cut-off of 0.3g/t Au would be most suitable for the project., and therefore this was used in subsequent interpretation and modelling work. Mineralised zone interpretations were defined by WAI based on the previous interpretation by Nord Gold, but the zones were expanded where necessary to reflect the lower cut-off grade. Near surface the zones were extended up into the air, to ensure proper selection of surface trench samples. A typical cross-section is shown in Figure 1.12, and as with previous interpretation work, there are two main northern and southern zones, split at approximately 23,500mN.

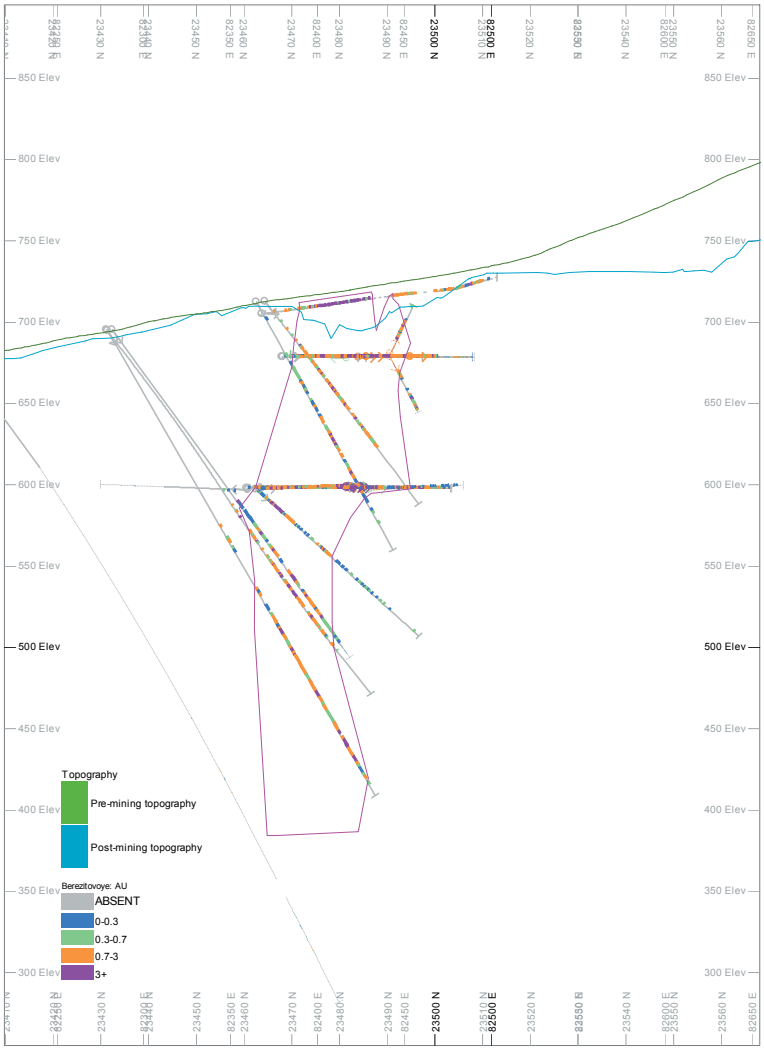


Figure 1.12: Typical Cross-Section of Mineralised Zones

1.5.3.4 Geostatistics

The sample data indicates a single log-normal population of gold grades. The similarity of the distribution of the channel samples and the drillhole samples supports the combination of both samples types for grade estimation. A geostatistical study of 5m composites indicated variogram ranges of approximately 45m along-strike, 60m down-dip and 20m cross-strike. An example modelled variogram is shown in Figure 1.13.

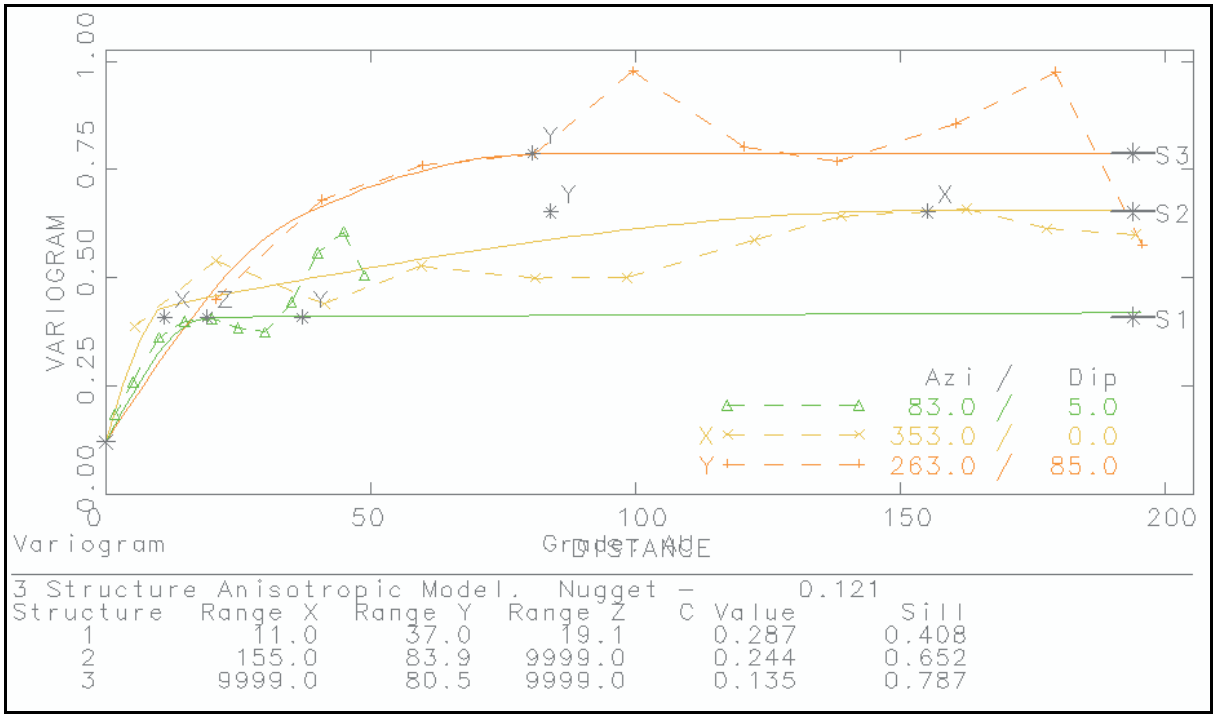


Figure 1.13: Au Variograms for Northern Zone

1.5.3.5 Block Modelling

A model prototype was set up with a 10m x 10m x 10m parent block size. This prototype was then used as the basis to set up a volumetric model, as controlled by the present (1st November 2010) topographical surfaces, the original pre-mining topography and the interpreted mineralised zones described previously. Within the mineralized zones, sub-blocks were generated, measuring 5m x 5m x 5m. The topographical data were used so that model blocks were generated up to the level of the original pre-mining topography, but with the mined-out parts separately coded. A typical section through the mined and unmined part of the volumetric block model is shown in Figure 1.14. A global density of 2.87t/m^3 was used.

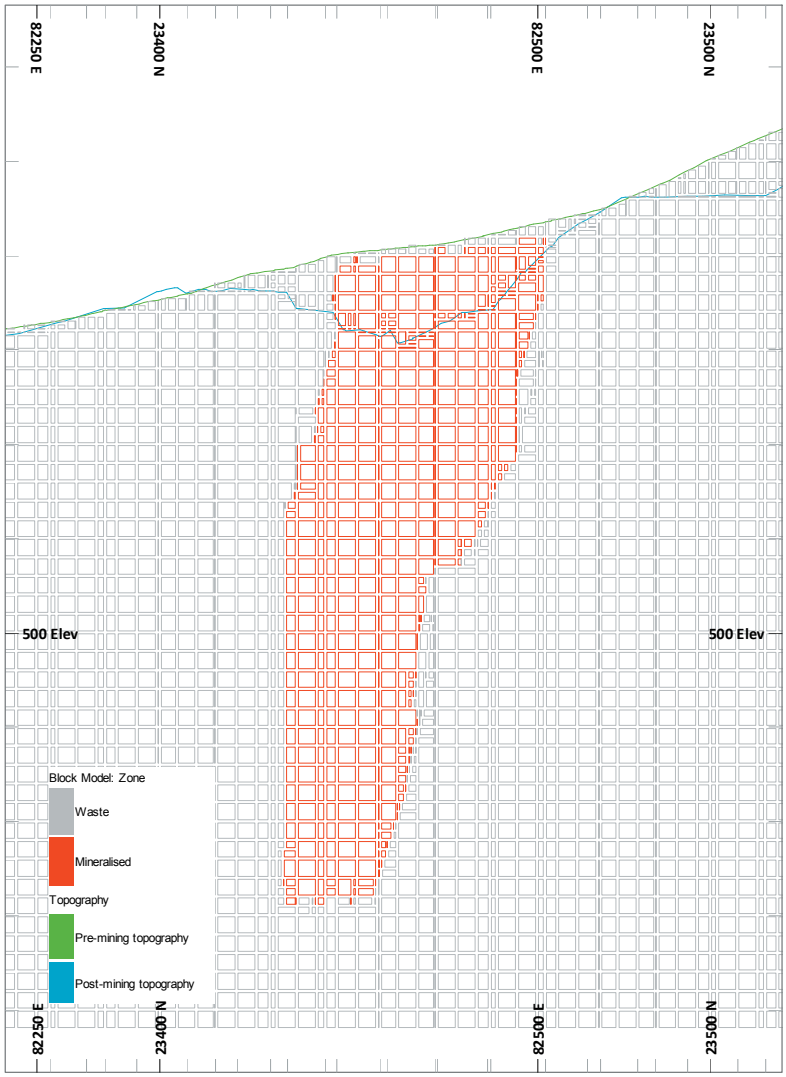


Figure 1.14: Cross-Section through Volumetric Block Model, on Section 14

1.5.3.6 Grade Estimation

Grade estimation was carried out using Ordinary Kriging (OK) as the principle interpolation method. Inverse Power of Distance Squared (IDW^2) and Nearest Neighbour (NN) were also used for comparative purposes.

1.5.3.7 Model Validation

A model validation process included the examination of block model versus composites, and the building up of a model grade profile, to compare average grades on vertical slices, as derived from the composites directly, as well as from model grades derived from different interpolation methods.

1.5.3.8 Resource Classification

Criteria for defining resource categories were also derived from the geostatistical studies. Key drillhole spacings for the allocation of *Measured* resources were 40m x 40m and 80m x 80m for *Indicated* resources. The extent of *Inferred* resources was limited to within 60m of a diamond drillholes intersection within the modelled mineralised zone. An example cross section through the block model showing resource classification is shown in Figure 1.15 below.

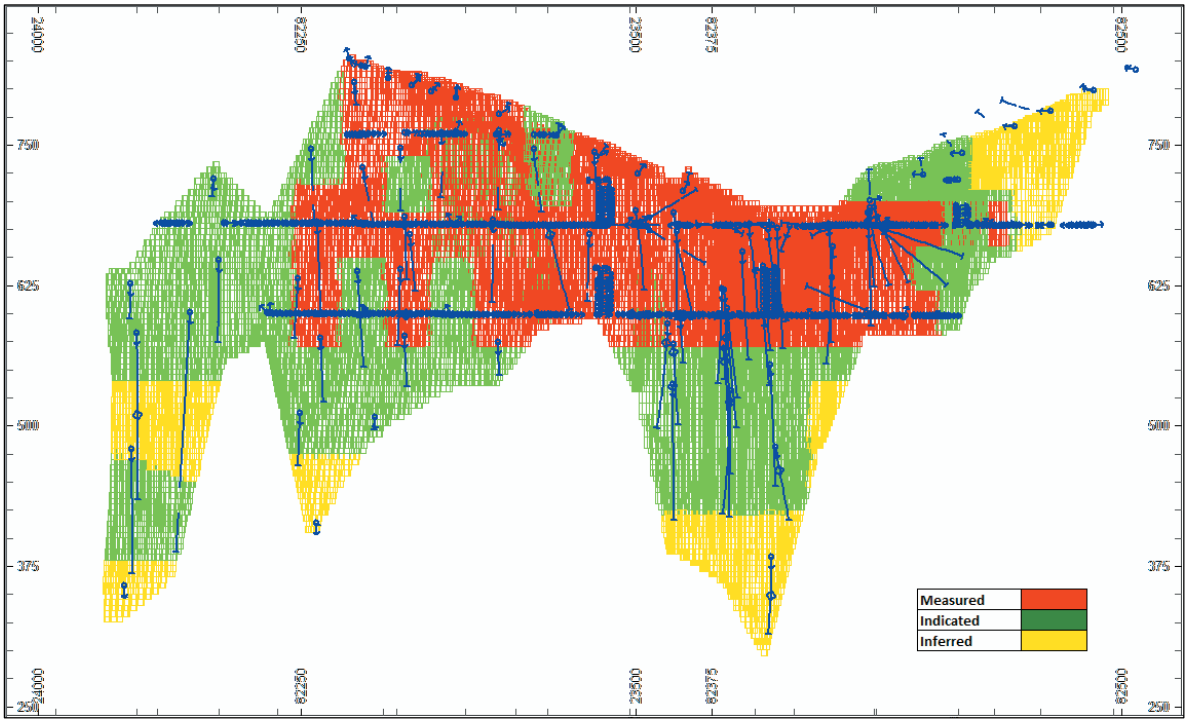


Figure 1.15: Example Long-Section showing Resource Classification

1.5.3.9 Resource Evaluation

The final block model was used as the basis for resource evaluation. Summary results of the evaluation of the un-mined, in-situ resources are shown in Table 1.8 for three different cut-off grade levels: 0.3g/t, 0.5g/t and 0.7g/t Au.

Table 1.8: Berezitovy Resource Estimate (WAI, 1st November 2010)
(in accordance with the guidelines of the JORC Code (2004)

Ore Type			Sulphide	Sulphide	Sulphide
Cut Off Grade (g/t)			0.3	0.5	0.7
Measured	Tonnage (kt)		10,764	10,577	9,905
	Au (g/t)		2.00	2.02	2.12
	Metal	kg	21,492	21,412	20,999
		oz	690,972	688,412	675,118
Indicated	Tonnage (kt)		10,357	10,184	9,479
	Au (g/t)		1.64	1.66	1.74
	Metal	kg	17,008	16,933	16,502
		oz	546,807	544,420	530,550
Measured + Indicated	Tonnage (kt)		21,121	20,761	19,384
	Au (g/t)		1.82	1.85	1.93
	Metal	kg	38,499	38,345	37,501
		oz	1,237,779	1,232,832	1,205,668
Inferred	Tonnage (kt)		2,571	2,364	1,951
	Au (g/t)		1.59	1.70	1.93
	Metal	kg	4,098	4,009	3,766
		oz	131,743	128,906	121,080

Notes:
1. Mineral Resources are not reserves until they have demonstrated economic viability based on a Feasibility study or pre-feasibility study.
2. Mineral Resources are reported inclusive of any reserves.
3. The contained Au represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

1.6 Mining

1.6.1 Pit Optimisation

1.6.1.1 Introduction

As a continuation of *Mineral Resource* modelling of Berezitovy deposit, WAI has undertaken a pit optimisation using the *Mineral Resource* Block Model prepared by WAI and updated in November 2010. The model was depleted to contain only those *Mineral Resources*, which have not been extracted as of 01 November 2010. WAI used NPV Scheduler® software for the optimisation, applying conceptual financial and technical parameters, provided by the Client.

The NPV Scheduler® programme comprises three components, the ultimate pit shell generator, a push back generator and the optimising scheduler. The ultimate pit generator is the first stage of the optimisation process and utilises a Lerchs-Grossman algorithm to generate an economic open pit shell from the *Mineral Resource* block model based on the initial input parameters.

The objective of this study was to obtain an optimised pit shell containing economically viable *Mineral Resources*, which therefore will provide an estimate of *Ore Reserves*. The estimate of *Ore Reserves* will form a basis for the mining schedule.

Prior to importing into NPV Scheduler®, the *Mineral Resource* block model was expanded by adding waste cells. A density of 2.87t/m^3 was assumed for each new waste cell. Zero grade values for gold were also assigned to each waste cell.

1.6.1.2 Pit Optimisation Parameters

The metal price used for the pit optimisation was US$900/oz Au. The major Key Performance Indicators (KPI) such as mining and processing costs, and other parameters, considered during the optimisation, has been supplied by the Client as actual results of year 2008-2010 production. These figures are given in Table 1.9 below.

Table 1.9: Berezitovy Key Performance Indicators for 2008 - 9m 2010

	Unit	2008	2009	9m 2010
Rock mined	kt	N/A	10,435	12,522
Ore mined	kt	833	1,342	1,400.1
Ore milled	kt	691	1,092	799
Average Au grade	g/t	2.5	2.8	2.3
Ore stripped	kt	N/A	9,344	11,122
Gold recovered (2008 - dore, 2009, 9m2010 - refined gold and silver)**	kg	1,365	2,715	1,718
Gold recovered (2008 - dore, 2009, 1H 2010 - refined gold and silver)**	koz	43.9	87.3	55.2
Recovery rate	%	89.4	87.3	89.6
Full cash cost	US$/oz		570	651
Normalised TCC	US$/oz		544	623
Ore mining costs	US$/t		1.61	1.50
Waste mining costs	US$/t		1.61	1.50
Ore processing costs	US$/t		11.96	15.75
General and administration costs	US$M		1.96	3.97
CAPEX	US$M		11.87	18.00
Depreciation	US$M		12.04	8.75

Note:
*** including Exploration and Evaluation**
**** including 1.15 kGEO and 0.64 kGEO of silver production in 2009 and 1H 2010; 903 and 1,017 GEO of silver production for 9 and 10 months 2010**

WAI has reviewed the results of year 2008 to 9m 2010 and technical data, obtained from the Client both during site visit and by request, together with the Client's production forecasts for 2010 and 2011, and found the following technical and economical parameters to be the best suitable for the optimisation purposes (Table 1.10).

One of the most crucial geotechnical parameters is the overall pit slope angle. The shallower the overall slope angle, the more waste cells that are included in the pit design. This has a great effect on the stripping ratio and hence economic viability. An overall pit slope angle of 55° was used for the optimisation (as supplied by the Client and observed by WAI at the site).

Table 1.10: WAI Pit Optimisation Parameters		
Parameter	**Units**	**Value**
Metal Price (Au)	US$/oz	900
Metal Price (Au)	US$/g	28.94
Selling Cost	US$/oz	56
Selling Cost	US$/g	1.80
Production Rate	ktpa	2,000
Discount Rate	%	11.2
Dilution	%	5.5
Mining Recovery Factor	%	96.6
Mining Cost (Ore and Waste)	US$/t	1.3
Processing Cost (per t of processed ore)	US$/t	12.55
Gold Recovery	%	90.1
Final Pitwall Angle	Degrees	55

1.6.1.3 Pit Optimisation Results

The results of the WAI Base case optimisation as calculated by NPV Scheduler® are presented in Table 1.11 below. The Base case optimised pit shell 3D image is shown in Figure 1.16 below.

Only the *Measured and Indicated Mineral Resources* were utilised in the optimisation as only these categories of *Mineral Resources* can become *Ore Reserves*.

Table 1.11: Berezitovy Deposit Open Pit Optimisation Results (WAI 01 November 2010, COG=0.51g/t Au)								
Case	**Gold Price (US$/oz)**	**Plant Recovery (%)**	**Annual Discount (%)**	**Ore (kt)**	**Au (g/t)**	**Au (kg)**	**Waste* (kt)**	**Stripping Ratio Estimation (t/t)**
Base Case	900	90.1	11.2	16,328	1.97	32,190	65,538	4.01

Note: 5.5% dilution and 96.6% mining recovery applied
*Waste is given inclusive of *Inferred* material

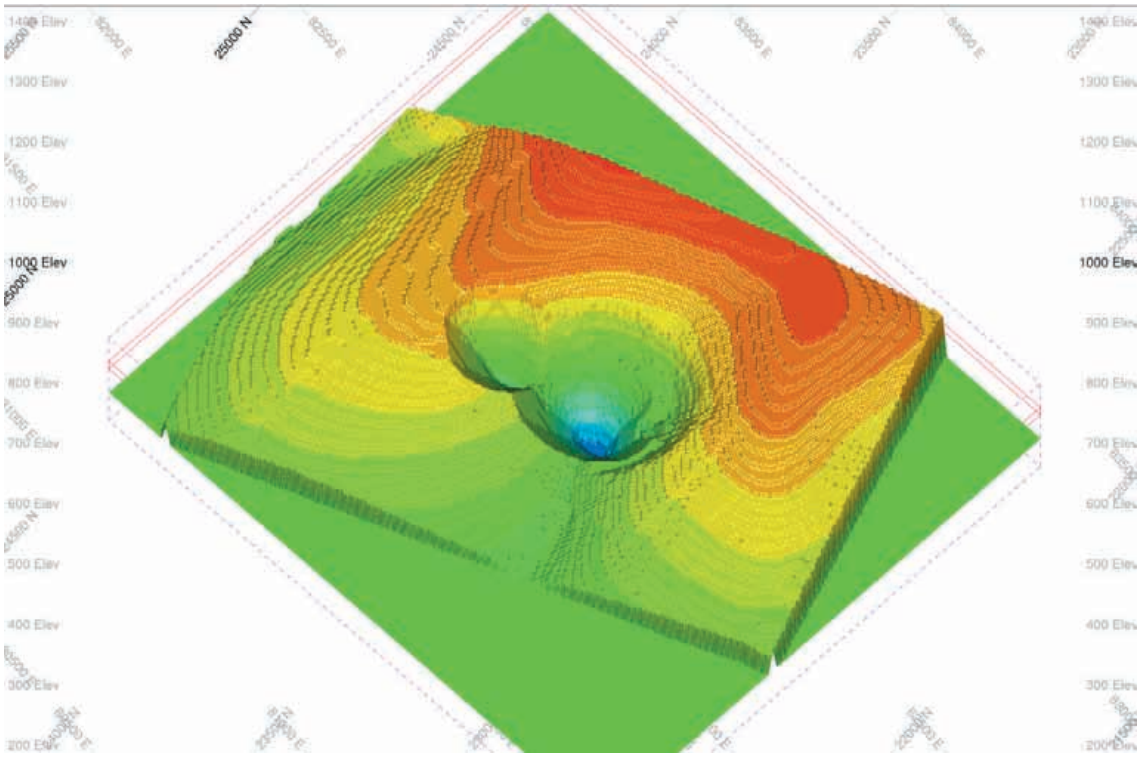


Figure 1.16: 3D View of WAI Optimised Pit

1.6.1.4 WAI Ore Reserve Estimation

Based on the open pit optimisation described above, WAI estimated Berezitovy open pit *Ore Reserves*. Detailed results of this estimation are given in Table 1.12 and Table 1.13 below.

Table 1.12: Berezitovy Open Pit Ore Reserves as of 01 November 2010 (WAI) (in accordance with the guidelines of the JORC Code (2004)

Ore Type			Sulphide
Cut Off Grade (g/t)			0.51
Proven	Tonnage (kt)		10,129
	Au (g/t)		1.98
	Metal	kg	20,105
		oz	646,450
Probable	Tonnage (kt)		6,199
	Au (g/t)		1.95
	Metal	kg	12,085
		oz	388,597
Proven + Probable	Tonnage (kt)		16,328
	Au (g/t)		1.97
	Metal	kg	32,190
		oz	1,035,047

NB-
1) Mining Factors of 5.5% Dilution and 96.6% Mining Recovery applied
2) Global density of 2.87t/m^3 applied to both ore and waste

Table 1.13: Berezitovy Reserve Pit Summary (WAI 01 November 2010)			
Ore (kt)	**Waste (kt)**	**Strip Ratio**	**Total Rock (kt)**
16,328	65,538	4.01	81,865

NB-
1) Ore Cut Off of 0.51g/t Au applied
2) Global density of 2.87t/m^3 applied to both ore and waste
3) Waste includes all of the inferred material at ore grades

1.6.2 Life of Mine Schedule (LOM)

After optimised pit shells were generated for each of the optimisation runs, WAI has designed a mining schedule for the Base case.

The schedule is targeting 2.0Mtpa with an appropriate ramp-up of production during the first two years (year 1 production equals 0.65 of full production rate – 1.3Mtpa, year 2 production equals 0.9 of full production rate – 1.8Mtpa). The Ore Reserve statement was prepared by WAI as of 01 November 2010, therefore the LOM schedule starts in 2011 with last two months of 2010 incorporated into 2011 production. As the Berezitovy mine already is in production and the majority of stripping works has been performed, the LOM schedule targets to maintain consistent total volume of ore (at a grade, equal to the average grade for the deposit) and waste extracted, providing the required planned annual ore tonnage as the result.

Table 1.14 below describes mining schedule for the Base case open pit shell designed by WAI.

An Ore Reserve open pit plan, long-section and cross-section are given in Figure 1.17, Figure 1.18 and Figure 1.19 below respectively.

Table 1.14: Berezitovy Deposit LOM Mining Schedule (WAI 01 November 2010)											
Year		**Nov-Dec 2010***	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**Total**
Rock	kt	3,117	8,037	8,845	8,928	8,841	9,862	11,983	11,718	10,030	81,361
Waste	kt	2,808	6,166	6,769	6,853	6,764	7,788	9,905	9,641	7,899	64,592
ORE	kt	310	1,871	2,076	2,075	2,077	2,074	2,078	2,077	2,131	16,770
Au	g/t	2.23	2.38	1.99	1.72	1.63	2.11	1.93	2.01	1.58	1.92
Au	kg	691	4,453	4,130	3,577	3,386	4,366	4,018	4,179	3,365	32,166
Strip	0	9.06	3.30	3.26	3.30	3.26	3.76	4.77	4.64	3.71	3.85

Note:
*** Based on the Company forecast**
5.5% dilution and 96.6% mining recovery applied
Waste is given inclusive of *Inferred* material

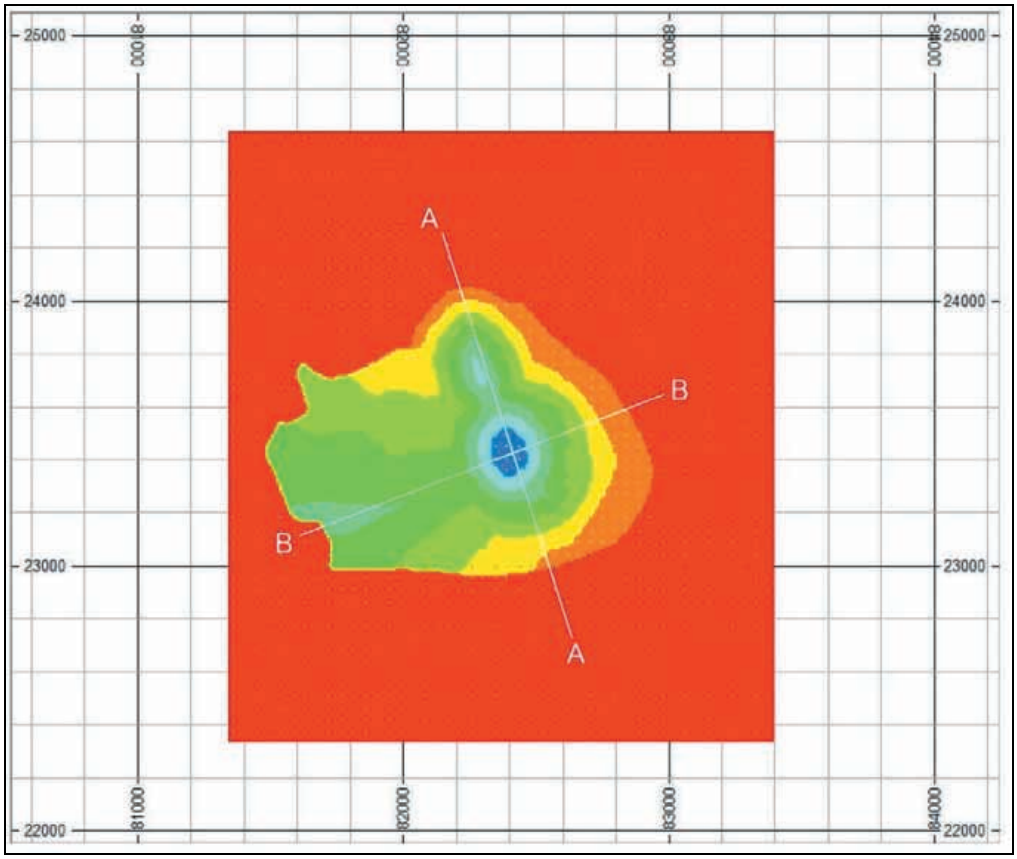


Figure 1.17: Plan View of WAI Reserve Pit, Colour Coded by Elevation

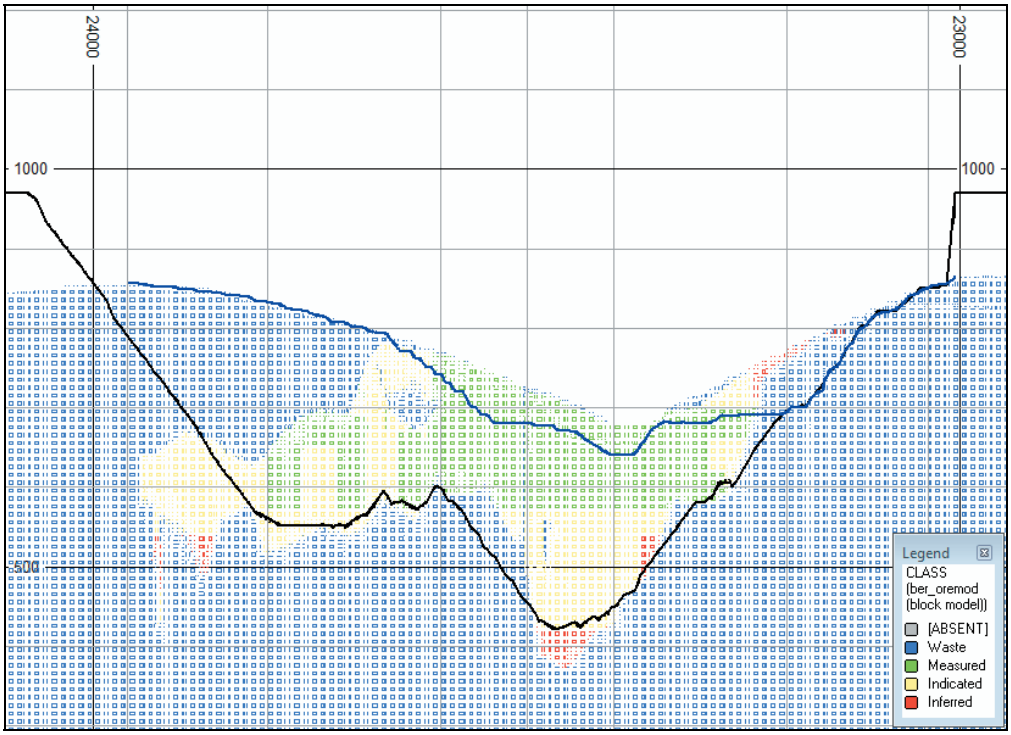


Figure 1.18: Long Section (Section A-A) of Geological Model
(Blue – Contours of Pit as of 01 November 2010; Black – Contours of WAI Reserve Pit)

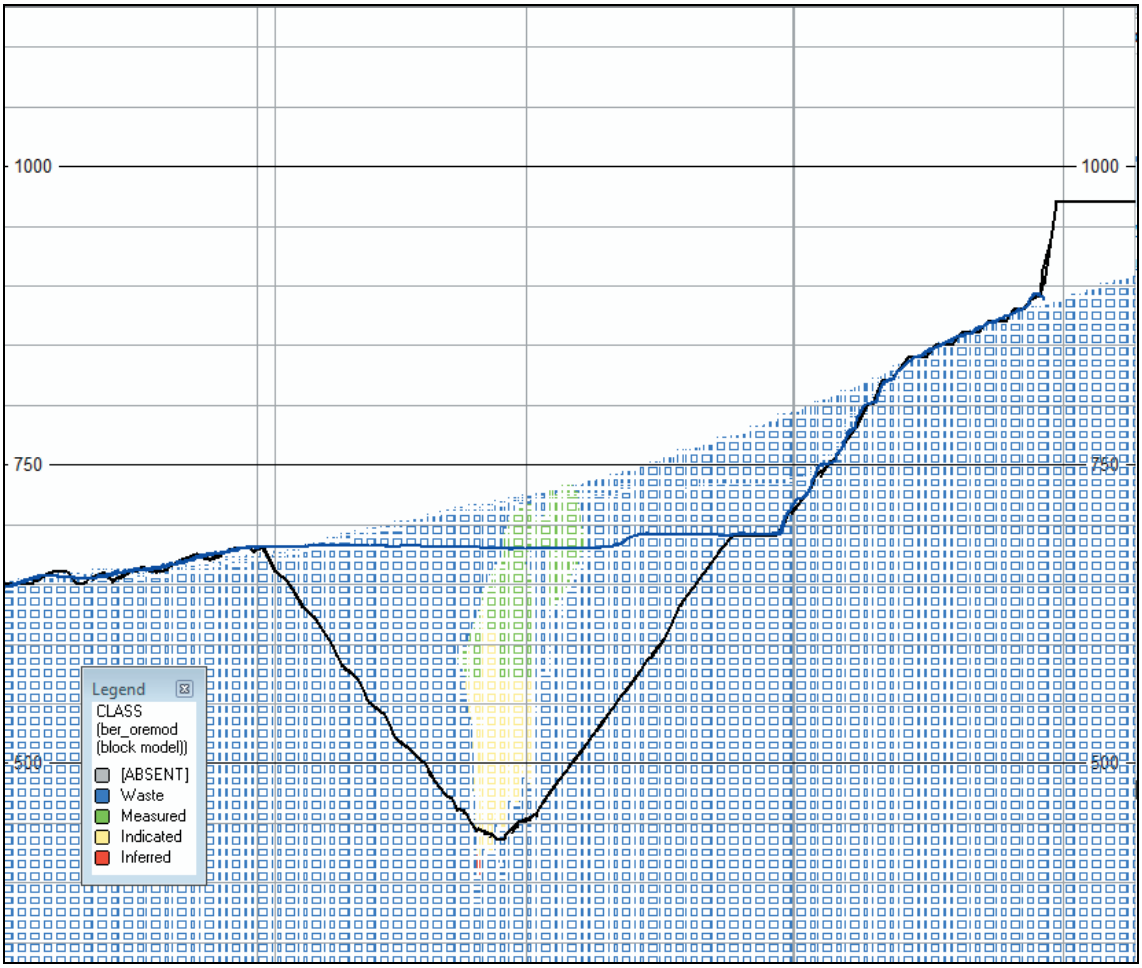


Figure 1.19: Cross-Section (Section B-B) of Geological Model
(Blue – Contours of Pit as of 01 November 2010; Black – Contours of WAI Reserve Pit)

1.6.3 Mining Activities

The pit began operating in 2006, and achieved its target commercial production rate in October 2008. The projected (design) production of the mine is 1.5Mtpa, whilst it is presently maintaining a production rate of 1.8Mtpa. Some 3,700kt of ore have been mined since the beginning of production to 01 November 2010.

Currently mining takes place on the 670m and 690m horizons, with the bottom of the pit at an elevation of 640m. Some preliminary extraction took place between the 640m-700m horizons, where extraction of the ore required less waste stripping.

Currently, the south and eastern walls have reached the final position, with the pit is expanding in a north eastern direction.

1.6.4 Current Pit Layout

The deposit is located in the Berezitovy River valley, on two of the slopes of the surrounding mountains. The current pit layout comprises the open pit itself surrounded by low-capacity waste dumps and access roads (see Figure 1.20 below).

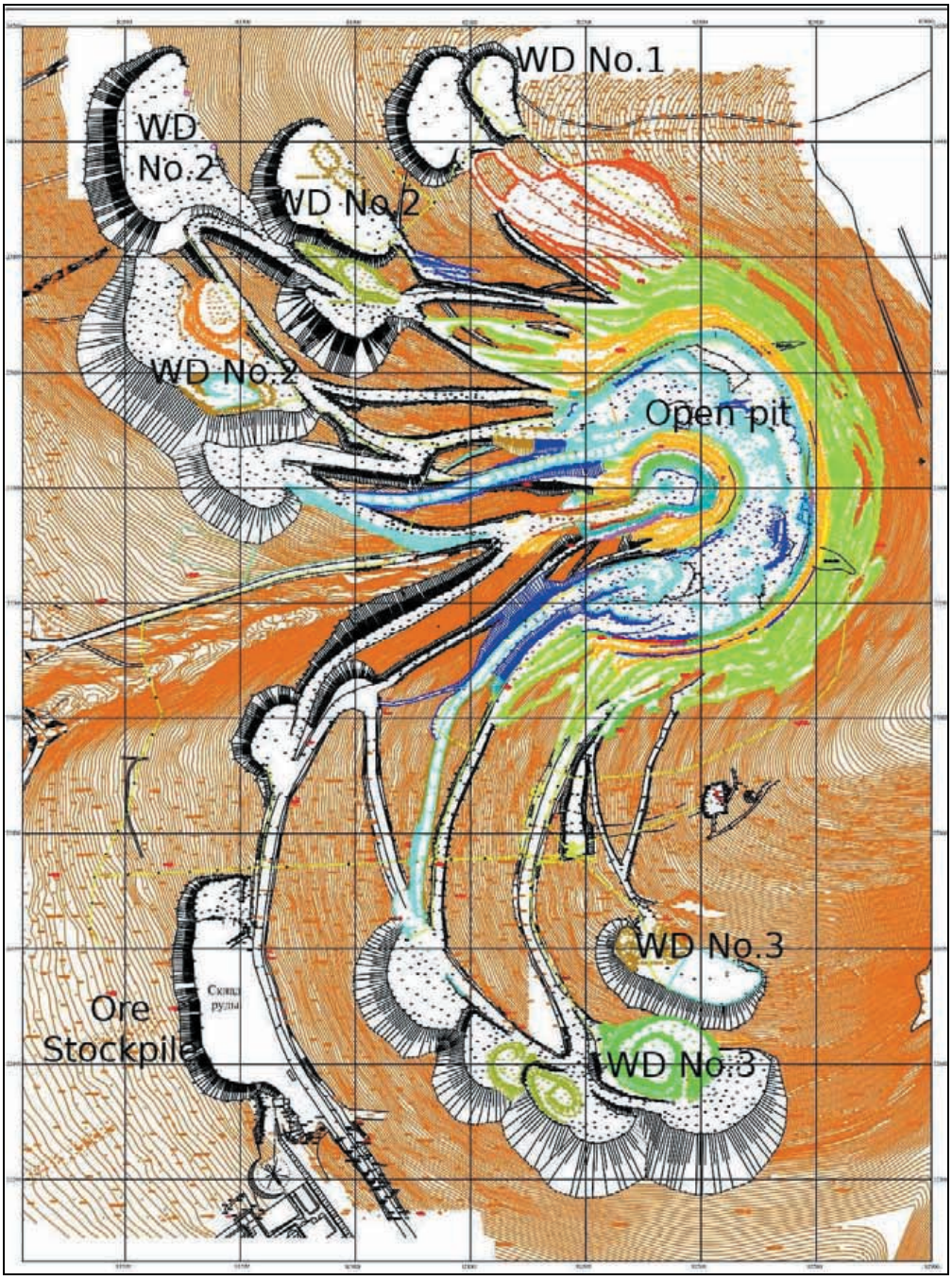


Figure 1.20: Open Pit Layout

There are main waste dumps at the site in addition to an ore stockpile, an off-balance ore stockpile, a drainage water sump and channels, an industrial area (including workshop, fuel charging area and portable cabins), as well as temporary and permanent roads.

1.6.5 Mining Method

1.6.5.1 Rock Properties and Geotechnical Conditions

The deposit is located on the slope of a mountain, partly in the Berzitovoye river valley. The orebody is covered by blocky diluvials (2m-5m), and in some areas outcrops. Permafrost areas are above the 676m horizon, with

permafrost thickness varying from 180m in the west and northwest to 60m-80m in the south, with respective variance of bedding depth from 0.5m to 50m from surface.

The deposit comprises hard rock, causing issues relating to fragmentation (in both ore and waste) during drilling and blasting operations. The ores are also acidic, which requires the use of specific types of explosives.

The bulking factor varies from 1.47 to 1.76, at an average of 1.56. The natural humidity of the rock is generally in the region of 0.10%; and rocks with increased natural humidity are generally 10%-20% softer. Rock frozen by permafrost has exhibited reductions in hardness of up to 20% compared to dry rocks.

Generally the rocks are stable, allowing the implementation of a high final wall angle (75°), corresponding to 55-61° overall pit slope angle.

1.6.5.2 Drilling and Blasting

The ore and host rocks within the deposit are hard and sulphidic, requiring specific efforts to maintain effective drilling and fragmentation of the material.

A blast hole grid of 3.5m x 3.5m is used for waste and, generally, a 3m x 3m configuration is used for ore, however for blocky rocks the grid is reduced to 2m x 2m. Specific blast patterns, hole size and charge layouts are used for final wall delineation.

1.6.5.3 Ore and Waste Loading

The extraction plan for the site is based on the results of sampling. The plan provides information regarding the location of ore, waste and low-grade ore within the blasted block. After the plan has been designed, it is applied in-situ. The boundaries between ore, low-grade ore and waste are marked by ribbons and pegs.

1.6.5.4 Ore and Waste Transportation

The principal transportation equipment at Berezitovy is 55t Caterpillar trucks and 45t Belaz trucks (see Table 1.15 below). The current average transportation distances to the ore stockpile (located adjacent to the crushing plant), is 2.0km-2.5km, and 1.5km to the waste dump.

1.6.5.5 Pit Dewatering

Pit dewatering is carried out without the use of pumping machinery, and clearly benefits from the deposit's location on a hillside. Surface water is drained via a drainage trench surrounding the pit edge, and then collected in an artificial water sump. Underground water is drained into the bottom of the pit via a system of in-pit drainages, and then discharged to the bottom of the valley from adit No.2. This adit is connected to the centre of the deposit via a raise.

The expected maximum water inflow is 36,161m^3/day (1,507m^3/h).

1.6.5.6 Ore, Low-Grade Ore and Waste Stockpiles

There are three ore types at Berezitovy: low-grade ore (or 'off-balance' ore at 0.5-1.0g/t Au), medium-grade ore (1.0-2.0g/t Au) and high-grade ore (>2.0g/t Au).

There is only one ore stockpile, used for medium and high-grade ore, which is located adjacent to the crushing plant. The ore is delivered there, stored, and then loaded into trucks to be taken to the primary crusher grizzly using wheeled loaders.

After the ore is delivered to the stockpile, it is repositioned by excavator in order to maintain stockpile design configuration.

1.6.6 Mining Equipment and Maintenance

A list of the major mining equipment in use at Berezitovy is given in Table 1.15 below.

Table 1.15: List of Major Mining Equipment

Equipment	No. of Units
Sandvik Pantera-1500 (Tamrock) Drill rig	4
Atlas Copco DM-45 Drill Rig	2
Hitachi 1200 Excavator	1
Komatsu PC-1250 Excavator	1
Hitachi-450 Excavator	1
EKG-5A Electric Shovel	3
BELAZ 45t Dump Truck	14
BELAZ 30t Dump Truck (multi-purpose)	2
Caterpillar 773D 55t Dump Truck	4
Wheel loader WA-500	3
Caterpillar D9R Dozer	3
Charging machine	1

WAI Comment: *although lacking truck washing facilities, the maintenance area is equipped sufficiently. Maintenance workshops have high standards of housekeeping. Generally, the observed maintenance facilities should ensure efficient exploitation of the equipment.*

1.6.7 Mine Personnel

The mine operates throughout the year, with two shifts per day. The length of shift is 12 hours, with 1 hour in-shift break.

The staff complement comprises 710 (153 – administration, geology, mechanics and maintenance services and 557 - open pit personnel).

1.7 Processing

1.7.1 Ore Characteristics

1.7.1.1 Introduction

Berezitovy is a gold-polymetallic orebody with moderate levels of sulphide (5 to 10%). There are two ore types, namely oxide and sulphide. Of the known mineable reserves, the primary (sulphide) ore type is predominant, with the oxidised ore accounting for less than seven percent of total reserves.

1.7.1.2 Oxidised Ore

In the oxidation zone, the sulphides minerals, with the exception of galena, are replaced with secondary minerals, with limonite predominating. The iron hydroxide levels reach 78.9% in the lower zones. In the partially oxidised ores, martite, marcasite and native copper are found.

1.7.1.3 Primary Ore

The results of laboratory and pilot-scale metallurgical testing show that the gold (including ore from the oxidation zone) is amenable to cyanide leaching) and is also amenable to gravity separation. It is, however, more difficult to recover silver, zinc and lead minerals into a saleable concentrate.

The ores contain low levels of elements which could be potentially deleterious to conventional processing technologies (Cu, Sb, Bi etc.) The exception is arsenic which varies between 0.03 and 0.29%.

1.7.1.4 Chemical and Mineralogical Analysis

In 2003, a 3.5t bulk sample was collected using core from diamond drill holes in Adit No.5. The bulk sample was considered representative for the known reserves of the deposit and contained the following proportions of ore materials: gold-polymetallic – 69%; gold-sphalerite – 16%; and gold-pyrrhotite-sphalerite – 15%.

Analysis of the sample indicated an average grade of 3.1g/t Au, 20g/t Ag, 0.5% lead and 1.0% zinc. The ore is characterised by a high silica content (70.8%) of which 52% occurs as quartz. Zinc (0.9%) and lead (0.5%) are present as sulphide minerals. Silver does not occur as separate minerals, but is associated with gold and galena.

1.7.2 Metallurgical Testwork

1.7.2.1 Introduction

In January 2003 15 representative ore samples from the Berezitovy Deposit were subjected to a programme of variability testing at Irgiredmet. The test programme concluded that the mineralogy of the economic minerals in each sample was similar and that the ore type could be described as being “gold-polymetallic”.

In August 2003, SGS Lakefield Research, Ontario, Canada (“Lakefield”) tested a smaller composite sample and determined that gold occurs as free particles and is also associated with sulphides (66% reporting to the ‘sinks’) and silicate minerals (as small inclusions).

The principal sulphide minerals were fairly well liberated and comprised pyrite, galena, sphalerite and pyrrhotite, with minor amounts of chalcopyrite, arsenopyrite and marcasite.

1.7.2.2 Pilot Testing

A 300t sample was used to investigate two process options:

- Gravity and flotation recovery – using gravity separation, flotation of gravity tailings, cyanidation of flotation concentrate, and selective flotation of lead and zinc from the cyanidation residue; and
- Gravity and cyanidation recovery for gold followed by selective flotation of lead and zinc minerals.

Standard procedures were used in analysis and testing of samples and data from the pilot plant testing indicated that the maximum recoveries that could be expected were gold 88.4%; silver 34.3%; zinc 58.2%; and lead 61.7%.

The optimum grinding size was found to be 90-95% passing 0.071mm. The optimum leaching conditions were found to be a NaCN concentration in solution of 0.05%; pulp density of up to 50%; and leach residence time of 24 hours, with cyanide consumption of 1.1kg/t ore.

1.7.3 Flowsheet Options

Based on testwork results, five potential options were studied and it was concluded that the highest gold recovery (up to 90%) would be provided by gravity separation and cyanide leaching followed by selective flotation of lead and zinc from leach tails.

This processing option also included acceptable selective recoveries of lead and zinc (61.7 percent and 58.2% respectively) which could be processed by pyrometallurgical plants.

However, cyanide leaching followed by selective flotation of zinc and lead from leach tailings was eventually preferred due to the simpler process sheet and the elimination of gravity gold concentrate refining, although it gave the lower gold recovery (86.5%). Also, testing at Lakefield Research indicated very little increase in recovery using gravity ahead of cyanidation.

1.7.4 Process Flowsheet

The plant was constructed between May 2006 and December 2007. Many of the major equipment items, including the jaw crusher, the SAG mill and the ball mill were sourced second hand and this has impacted significantly on the operational efficiency of the plant.

In October 2009 the plant was treating ore at a rate of 1.2Mtpa. The tailings filtration capacity was being increased and an additional ball mill installed which would enable the processing rate to be increased to 1.5Mtpa in 2010. The current mining and processing rate is 1.8mtpa and further expansion to 2.0Mtpa is planned in 2011.

1.7.4.1 Ore Stockpile

Ore is delivered to a coarse ore stockpile adjacent to the main plant feed hopper which, at the time of the WAI site visit, consisted of some 500,000t of ore grading 1.5 to 2.0g/t Au. Ore is transferred to the hopper using a Front End Loader.

The primary crushed ore is conveyed to a crushed ore stockpile with a nominal capacity of 15,000t. The stockpile suffers from segregation of the coarser material which tends to roll to the bottom of the stockpile.

1.7.4.2 Grinding

Ore is ground in a SAG mill–ball mill circuit to a size of 80% passing 74µm. The ball mill maximum ball loading is 20% by volume rather than the design figure of 35% resulting in the target grind of 90% passing 74 microns not being achieved and a reported recovery loss of 2%. Milling rate ranges from 185 to 220tph.

The Company has informed WAI that during the second half of 2010, a new 4.5 x 6m ball mill has been installed in parallel with the existing ball mill. The productivity of these two second stage ball mills is some 0.9Mtpa with 80-85% passing 74 microns.

In addition, an increase in grind size from 2 to 5mm in the initial SAG mill stage has allowed capacity to increase from 1.3 to 1.8Mtpa.

1.7.4.3 Leaching

The ground pulp, at 24% solids, passes to a 24m diameter thickener. The thickener overflow is returned to the grinding circuit and the underflow, at 52-53% solids, is pumped to the leach tanks.

There are eight leach tanks each with capacity of 430m^3. The tanks are air sparged but the levels of dissolved oxygen are not determined. Cyanide is added to tanks 1 and 5 and the pH is maintained at 11.5.

Carbon absorption takes place in six 330m^3 adsorption tanks. There is a total of 28t of carbon in the circuit. Carbon addition rate is 4%. The loaded carbon, typically assaying 2kg/t is screened and passes to desorption.

1.7.4.4 Desorption and Electrowinning

Gold is recovered from carbon using a conventional desorption-electowinning and smelting process.

There are three desorption circuits each capable of processing 10m^3 of carbon. Gold loadings are lower than the design figure due to lower plant head grades, the presence of wood chips in the pulp and the coarser grind size.

There are three electrowinning cells. The cathode sludge is smelted on site to produce dore.

1.7.4.5 Filtration of Leach Tailings

The Berezitivoye plant uses dry deposition for tailings disposal. The leached tailings are pumped to a filter station which consists of three disc filters (each filter has 200m^2 filtration area) with an average utilisation of 37% in 2008. Two additional disc filters (100m^2 filtration area) which were due to become operational by the end of 2009, will each give a filtration rate of 55t/h.

WAI has been informed by the Company that in the second half of 2010 that modifications and repairs within the filtration plant have allowed tails processing to increase from 1.3 to 1.8Mtpa.

The problems of filtering the leach tailings resulted in the construction of an emergency tailings dam. In 2008, it was reported that some 60% of the plant tailings was sent to this dam although this figure fell to 30% in 2009.

> ***WAI Comment:*** *the filtration of the entire plant tailings stream is a high cost operation and will be prone to technical and mechanical risk. Variable amounts of clay minerals in the plant feed will cause fluctuations in filtration rates. The ability of the filtration plant to cope with the required throughput is critical to achieving production targets, particularly when the emergency pond can no longer be used.*

1.7.5 Consumables

The 2009 budget for plant consumables is given in Table 1.16 below.

Table 1.16: Plant Consumables Budget 2009

Item	Consumption g/t ore
Mill liners	150
Grinding balls	2,030
NaCN	768
Lime	2,140
Carbon	118
NaOH	146
Electrical power	42.1kWh/t
Filter cloths	0.005m^2/t
Diesel	2.2l/t

The grinding media is sourced from Russia and the cyanide from Korea. The carbon is bought from a Malaysian supplier. The rates of consumables are typical for a moderately hard, non refractory ore.

1.7.6 Water Balance

The process water requirement for the plant is 13,800m^3 per day. Water losses to tailings are estimated at 1,100m^3 per day and internal recycling provides 12,700m^3 per day. Fresh water supply of 852m^3 per day is provided from four boreholes.

1.7.7 Production Data

The annual production summary for the plant is given in Table 1.17.

Table 1.17: Berezitovy Plant Historic Production Data

	2007	2008	2009	2010*
Ore processed (tonnes)	61,717	691,000	1,091,615	885,369
Feed grade g/t	1.87	2.49	2.78	2.3
Au Recovery %	83.4	89.4	87.3	90.0

*Year to date, 1St November 2010

Production commenced during the last month of 2007. In the first full year of production (2008) a total of 691,00t of ore was treated at an average plant head grade of 2.49g/t Au. In 2009 a total of 1,092Kt of ore had been processed. In 10 months 2010 at total of 885Kt have been processed. It is planned to increase the plant throughput to 2.0Mtpa by 2011.

The plant recovery has increased gradually over the period from 83.4% in 2007 to 87.3% in 2009.

1.7.8 Assay Laboratory

1.7.8.1 Introduction

The laboratory analyses samples from all stages of the production process including geology, mining production, plant and environmental. In 2008 the laboratory assayed over 13,000 samples. The majority of the exploration samples are only prepared in the laboratory and sent to an external laboratory for analysis.

1.7.8.2 QA/QC Procedures

The laboratory uses a range of Standard Reference standards supplied by Rocklabs. The standard ore sample assays 1.65ppm Au (tolerance +/- 0.08ppm Au) and the tailings sample assays 0.71ppm Au (tolerance +/- 0.03ppm Au).

A regular system of duplicate analysis is used and the allowable differences between the analyses are given in Table 1.18.

Table 1.18: Agreement between Duplicate Analyses

Assay ppm Au	Agreement (+/- ppm Au)
0.36	0.12
1.04	0.26
4.3	0.90
10.9	1.3
20.9	1.9

The sample batch is repeated if the agreement falls outside these limits.

1.7.8.3 Staffing

The laboratory is operated over two 10 hour shifts by a total of 20 employees and is managed by a Chief Chemist. The laboratory operates 7 days a week.

1.7.9 Labour

There are a total of 161 employed in the plant, operating on 12 hour shifts. The shift rotation is one month on and one month off. There are a total of 15 engineers and technical staff.

There are four operators per shift in the crushing plant, six in grinding, seven in leaching, desorption and electrowinning and four in dore production.

1.7.10 Capital Costs

The capital costs for the process plant are given in Table 1.19.

Table 1.19: Process Plant Capital

Year	US$
2009	3,688,590
2010	9,171,953
2011	3,770,080

In 2009 the majority remaining capital expenditure was on the installation of the two new filters, a fuel and lubricants store, design work for the new ball mill and improved dust collection in the crushing plant.

1.7.11 Conclusions

The Berezitovy plant has experienced a slow start up which has either been due to the installation of poor quality, second hand equipment or new equipment of poor construction.

At the time of the site visit, the plant gold recoveries were approaching, but still short of the target level. This was attributed to the failure to achieve the required grind size and the installation of a new ball mill should remedy this situation.

The tailings filtration section of the plant was also performing poorly due to the installation of low quality, Chinese filters. The filtration plant was being upgraded by the addition of two new filters of Russian manufacture, combined with modernised Chinese filters, which should increase productivity to 280tph.

Recent data received from the Company indicates that these issues have been remedied, although WAI has not had an opportunity to validate these claims.

In terms of the process, the installation of gravity processing may result in a minor increase in recovery although the impact on the overall plant recovery will probably not be significant.

The method of head sample analysis requires review as a significant proportion of the gold in the head sample is solubolised. The method of head sample filtration and solution analysis needs to be reviewed.

The economics of producing lead and zinc from the leach tailings should be reviewed through on-site laboratory flotation tests on samples of leach tailings. This is particularly the case in the light of recent increases in base metal process.

The sample preparation method for geological and mine samples should be changed so that all of the material is crushed to pass 1mm. The sample preparation dust extraction facility also requires an overhaul.

1.8 Environmental, Community, Health and Safety Section

1.8.1 Introduction

WAI was commissioned by the Client to undertake a review of environmental and social issues relating to development of Berezitovy Mine to establish whether or not the project is in compliance with environmental management commitments and relevant environmental legislation. WAI also compared existing practices at Nord Gold operations to international standards, codes and best management practices e.g IFC Performance Standards and World Bank Guidelines.

Russian operations have always sought to comply with Russian Federation requirements and continue to meet the applicable standards. Compliance with these standards is monitored and results are reported. Non-compliances are addressed and corrective actions are implemented and reported to regulatory authorities, it is important to note the reluctance of local Russian authorities to change their approach to standards. This does not encourage the implementation of international standards, however the client continues to strive to improve its HSEC programmes and performances.

It is understood that the project area is not designated for particular cultural interest or special protection. Nevertheless, the region supports various ecosystems and is fragile and as such requires sound regimes of environmental control.

1.8.2 Key EHSC Aspects and Issues

The key EHSC aspects and issues associated with this project include:

Corporate level:

- Health, Safety & Environmental (HSE) management;
- Social Responsibility and Community Engagement;
- Social & Environmental Assessment and HSE Permitting; and
- Labour & working conditions.

Project level:

- Processing facilities;
- Mining activities
- Project closure, rehabilitation and post-closure monitoring;
- Infrastructure and associated facilities;
- Surrounding communities;
- Protection and conservation of biodiversity.

An OVOS was prepared and approved in 2004 before operations commenced at the site and in October 2005 an Environmental Impact Assessment (EIA) was prepared which concluded that the significant impacts likely to arise from the planned operations would be mostly rated as local in character with a risk that the delivery of reagents could be rated as of narrow regional in nature. In addition an Environmental/Social Action Plan was formulated to demonstrate compliance with international guidelines.

> ***WAI Comment:*** *WAI considers that the EHSC Action Plan was well founded but has not been implemented in its entirety and requires further commitment to complete all of the proposals within the timescale specified.*

WAI agrees with the main findings of the EIA, OVOS and Action Plan; however the main environmental risk is related to the storage of non-detoxified process residues, intermediate drainage, recycling of highly polluted leachate and a zero discharge system. Although the EIA did not consider the 'Emergency Tailings Disposal Facility' as operational, WAI consider this to represent a higher degree of risk .

1.8.3 Environmental Permitting

At the time of the site visit, all licences and permits are in place (2009 to 2014) for the discharge of contaminants, and MACs, MADs and MAEs calculated and approved by the State authorities. The project is fully permitted by State and Local Authorities and is attested to be compliant with the State requirements, whilst an Ecological Passport has been issued by State authorities to control emissions throughout 2009 and no significant matters have been identified nor have any prescriptions been made based on the .minimum three yearly inspection.

WAI Comment: *WAI notes the high standards of environmental management applied at the site. Record keeping is exemplary, although this task may be improved by the introduction of a computer database as the volume of analysis results and permit requirements increase.*

1.8.4 Environmental Protection

Air, water, flora, fauna and soils are afforded protection in the project lease area. Regular, sampling and reporting is undertaken under contract as well as by Berezitovy environmental staff for comparative purposes.

WAI Comment: *The monitoring programme has been well planned and conducted, designed to high standards and very comprehensive over the short period since its inception in 2008.*

1.8.4.1 Air

Air sampling in 2008 indicated elevated levels of low level contaminants when measured against baseline conditions and emissions are considered to be within permitted MAC's for the operation until 2012. Measures taken to mitigate negative impact on the atmosphere include dust suppression, using water bowsers and cyclones with scrubbers for dust collection at the main boilers plant. All personnel were observed to be wearing appropriate personal protective equipment and dust masks.

WAI Comment: *WAI believes that dust reduction and control measures could be improved at Berezitovy Mine in the areas surrounding the tailings storage facilities. The proposed change of practice of incineration of waste oils on-site to the storage and off-site treatment by an outside contractor will lead to a reduction in the products of combustion.*

1.8.4.2 Water

The Khaikta River which is the source of industrial water had, in 2008, already undergone technogenic change and the water quality was poor, with low numbers of indicator species. Potable water is taken by boreholes from groundwater resources adjacent to the same river.

As the site is designed to operate as a 'zero discharge facility' (recycling in a closed system) no negative impact on the water environment is anticipated.

Domestic effluents are mostly drained to a Bio-disc, bacteriological plant and then to settlement ponds whilst open-pit mine water drains into Berezitovy Creek, where it is settled in a series of three, water collection dams constructed of coarse filtration gravels, before discharge to the Khaikta river.

WAI Comment: *WAI are informed that the Tailings area including the Emergency Tailings Disposal Facility was inspected by Berezitovy involving experts from Research Institute (hydraulic engineering works) KuzbasNIIOGR, Rostechnadzor (State Industrial Inspectorate), Chief Directorate of the Ministry of Emergency Situations of Russia for Amur region and a design institute in May 2010 .The Emergency Tailings Disposal Facility was "shut down". The project is been developed by the design institute IRGIREDMET to turn the ETDF into the pond of recycle water. Samples taken in groundwater beyond the emergency TMF dam wall, elevated levels of sodium cyanide have been noted. Decontamination and clean-up measures are currently being considered by Nord Gold*

The domestic water treatment system however is operating successfully and In general the range of determinands tested for in water samples exceeds the State requirements on monitoring.

1.8.4.3 Land

Sampling of waste dumps has so far indicated clean neutralisation and marginal acid generating potential (2007) whilst a new fuel storage area is to be constructed and measures will be taken to remove and treat contaminated soils at the old facility.
All 91ha of disturbed lands are required to be reclaimed at closure and a Mine Closure and Recultivation Project (2007) and has been approved by the State authorities.

WAI Comment: *The measures proposed to clean-up the fuel spillages at the present facility are acceptable.*

Standards of dump rehabilitation practiced appear poor with only token planting and very little placement of top-soil.

The mechanism for accrual of funds for mine closure and recultivation is unclear and it is recommended that the current Closure Plan should be finalised.

1.8.5 Waste Management

Berezitovy has introduced a Waste Management Plan for industrial domestic and hazardous wastes arising at their operations, based on the State concept that any damage to land, fauna and flora is reversible and insignificant in the region as a whole.

Waste rock is classified as non-toxic and is stockpiled in engineered waste rock dumps. Solid waste is classified as non-toxic and dumped at surface by 'edge-tipping'.

WAI Comment: *The heavy use of cyanide poses high risks to both the surrounding environment and human health unless properly managed and it is recommended that the Company undertakes necessary steps towards reinforcement of its cyanide management practices.*

1.8.6 EHSC Management

Occupational health and safety receives high priority and environmental reporting, management and monitoring appear to be conducted to meet State requirements.

Monitoring provides for:

- Landscape monitoring;
- Hydrology observations and reporting;
- Hydrogeological observations and analysis;
- Air sampling and analysis;
- Snow cover sampling and monitoring;

- Hydrobiosynosis reporting; and
- Waste dumps monitoring, control and management of potential acid generating material.

WAI Comment: *Monitoring plans are well formulated and conducted to a high standard by well respected independent institutes under contract to Berezitovy.*

1.8.7 Conclusions and Recommendations

The following aspects have been highlighted by this report:

- The EHSC Action Plan requires full implementation;
- The 'Emergency Tailings Disposal Facility' has been decommissioned and its long term stability will need to be ensured such that it does not pose an environmental or human health risk;
- Environmental monitoring and other data should be centralised in a database to allow the identification of any anomalous results and provide a useful benchmark by which to verify State assessments;
- Dust reduction and control measures could be improved at the TMFs in line with international best practice, and to reduce risk to personnel;
- Appropriate measures should be implemented to prevent, minimise or remove the pollution of surface waters in the open pit by explosives residues and hydrocarbons;
- The newly proposed fuel storage facility should be lined and designed so as to minimise future spillages and resultant soil contamination;
- The standards of progressive rehabilitation should conform to the conceptual mine closure and land recultivation plan, particularly with regard to the rehabilitation of Waste Dump #3; and
- To satisfy international best practice, the current Closure Plan should be extended and finalised and include an assessment and secure source of sufficient finance in an escrow account set against this and any post closure management and monitoring. Although financial security for the mine closure is not common in Russia and estimated mine closure costs are only reported as financial clarity in IFRS financial statements.

WAI recognises that to date the company has primarily sought to acheive national compliance, which has been done, and hence many of the above recommendations have hitherto not been required. However, WAI recognises the company's commitment to achieving international compliance, and highlights the issues above as areas where further actions will be required.

1.9 Life of Mine Model of the Project

1.9.1 Introduction

A life of mine production model of the Berezitovy gold deposit has been prepared in order to demonstrate the potential viability of the project and its robustness. This model was based on the Ore Reserve figures described above, capital investment requirements estimates provided by the Client and audited by WAI with a view to similar projects, and other parameters. Input parameters were implemented into a life of mine model. The details of this model are given below.

1.9.2 Financial Model Assumptions and Input Data

The assumptions made in the WAI financial model for the Berezitovy gold project were based on:

- Au price of US$900/oz;
- Ore reserves as of 01 November 2010, estimated by WAI in accordance with the guidelines of the JORC (2004) code;
- Mining Schedule, prepared by WAI. The Mining Schedule targeting a 2.0Mtpa ore mining rate with appropriate ramp-up of production;
- Long-term operating costs forecasts based on year 2008-2010 production results and considering analogous deposits in this part of Russia;
- Overall Au recovery indices obtained from actual annual results for 2008-2010;
- Annual discount factor of 11.2% (Base case);
- General and Administration costs estimated to be equal to 20% of main costs; and
- Overall tax at 20% of net income applied.

A summary of the Berezitovy life of mine model assumptions and input data is given in Table 1.20 below.

Table 1.20: Berezitovy Life Of Mine Model Assumptions and Input Data

Year	*TOTAL*	Nov-Dec2010*	2011	2012	2013	2014	2015	2016	2017	2018
Gold Price,US $/Oz	-	900	900	900	900	900	900	900	900	900
Ore production (diluted) kt	***16,770***	310	1,871	2,076	2,075	2,077	2,074	2,078	2,077	2,131
Waste mined, kt	***64,592***	2,808	6,166	6,769	6,853	6,764	7,788	9,905	9,641	7,899
Ore Mining cost, US$/t	***1.55***	1.55	1.55	1.55	1.55	1.55	1.55	1.55	1.55	1.55
Waste Mining cost, US$/t	***1.55***	1.55	1.55	1.55	1.55	1.55	1.55	1.55	1.55	1.55
Mining OPEX, k US$	***126,110***	4,832	12,457	13,710	13,839	13,703	15,286	18,573	18,163	15,547
Au Grade g/t	***1.92***	2.23	2.38	1.99	1.72	1.63	2.11	1.93	2.01	1.58
Gold mined, kg	***32,166***	691	4,453	4,130	3,577	3,386	4,366	4,018	4,179	3,365
Recovery, %	***0.90***	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
Gold recovered, kg	***28,965***	622	4,010	3,719	3,221	3,049	3,932	3,618	3,763	3,031
Gold recovered, Oz	***931,253***	19,998	128,919	119,571	103,572	98,031	126,414	116,319	120,993	97,436
Processing cost, US$/Oz	***228***	217	182	218	251	266	206	224	215	274
Processing cost, US$/t	***13***	14	13	13	13	13	13	13	13	13
Proc and Mining Costs US$/Oz	***375***	459	279	333	385	406	327	384	366	434
Depreciation (total) k US$	***97,052***	1,747	13,884	13,884	13,884	13,884	13,884	13,884	6,000	6,001
Revenue, k US$	***838,128***	17,998	116,027	107,614	93,215	88,228	113,772	104,687	108,894	87,692
Operating Costs, k US$	***337,021***	9,171	35,938	39,766	39,887	39,774	41,309	44,656	44,227	42,292
G&A, Sales	***158,263***	2,885	17,307	19,038	19,038	19,038	19,038	19,038	19,038	19,038
Royalty, 6%	***50,288***	1,080	6,962	6,457	5,593	5,294	6,826	6,281	6,534	5,262
CAPEX, k US$	***22,281***	1,781	8,500	2,000	2,000	2,000	2,000	2,000	1,000	1,000

Note:
Based on the Company forecast

1.9.3 Conclusions

The WAI life of mine model results in a positive NPV at various discount rates and at various gold prices, as well as the relatively high internal rate of return at nominal input parameters. This shows that the reserves considered in the financial model are profitable for exploitation in the current economic environment.

The deposit consists of a large gold ore resource, with potential to increase, as the area is being intensively explored.

The fact that the key financial indices remain reasonably high given the conservative cost input parameters and recovery used in the models, shows good economic potential for the project.

2 IROKINDA

2.1 Introduction

Wardell Armstrong International (WAI) was commissioned by Buryatzoloto to prepare a Competent Persons Report (CPR) on the Irokinda fully operational underground gold mine, situated in the Muiskay district, Republic of Buryatia, Russian Federation.

A previous independent audit (September 2008) by Micon International Limited (Micon) provided a conversion (2007) of the Russian classified "reserves" and "operational reserves" into Canadian Institute of Mining, Metallurgy and Petroleum (CIM) compliant mineral resources and mineral reserves according to the current definitions.

The review of Irokinda was based on published material researched by Micon, as well as data, professional opinions and unpublished material submitted by the professional staff of High River and Buryatzoloto or their consultants. Much of the data came from reports prepared and provided by High River or its Russian subsidiary, Buryatzoloto.

2.2 Background

2.2.1 Location

Irokinda gold mine is located in the far northern portion of the Republic of Buryatia in the Muiskay district.

The mine is situated 700km northeast of the Buryatian republic capital city of Ulan-Ude, which hosts an international airport with numerous regional flights to other major Russian cities, as well as international destinations, and 74km from the village of Taximo which is the nearest railhead (Figure 2.1).

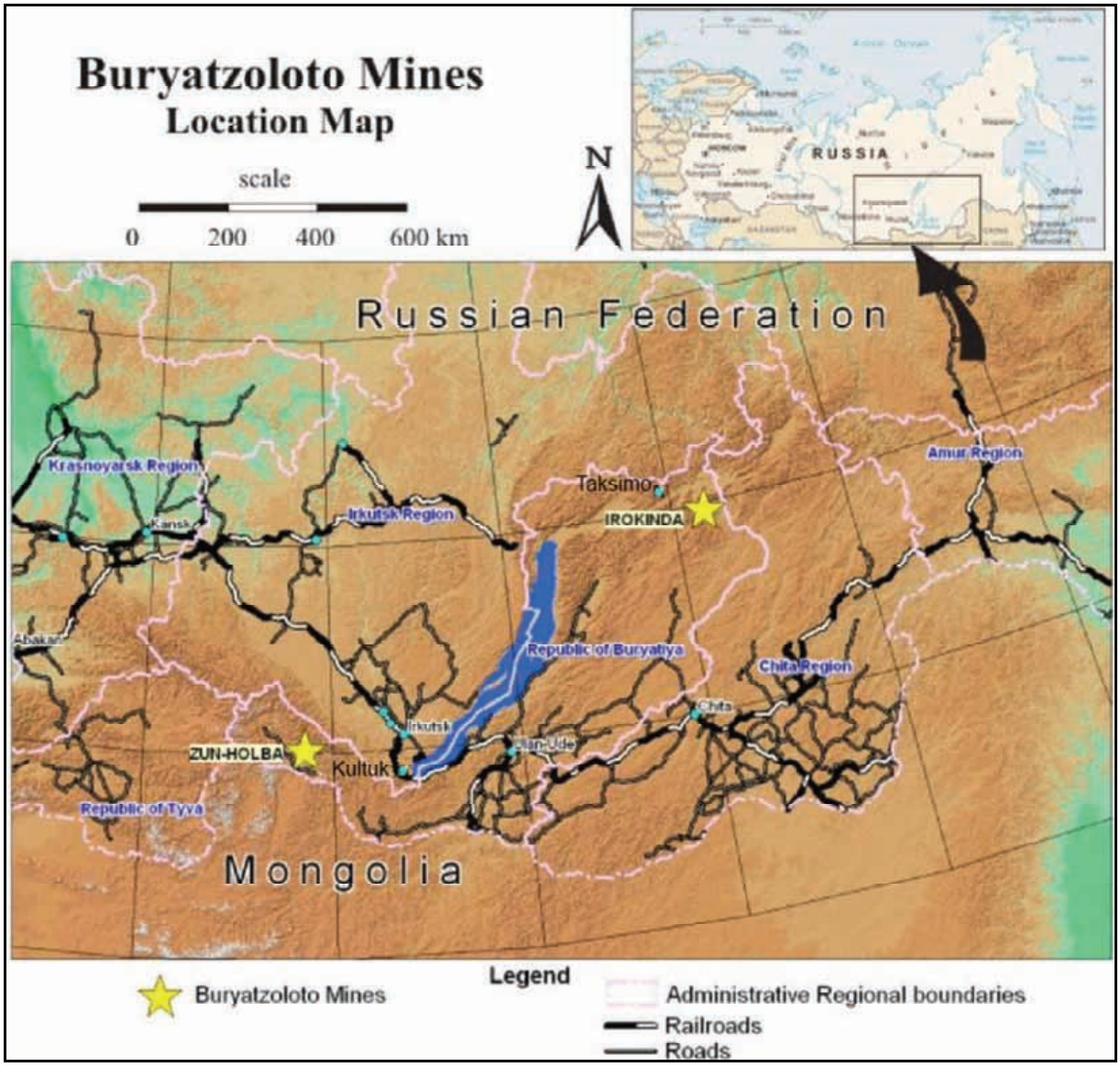


Figure 2.1: Location of Irokinda Gold Deposit within the Buryatia Region, Russian Federation

2.2.2 Access

Irokinda mine is readily accessible from the railway station at Taximo on the Baikalo-Amurskaya (Baikal-Amur) railway, via a 70km all weather gravel road, which takes approximately 2hrs by 4WD vehicle. This road crosses a high mountain pass which is prone to poor weather during the winter months, but is rarely closed, a result of regular inspections and snow clearance.

2.2.3 Infrastructure

Electric power for the Irokinda mine and settlement is supplied from a 220/110/35kV substation, located in Taximo and transmitted via a single 70km, 110kV power line.

The main water source for the Irokinda mine consists of two wells located in the western outskirts of the Irokinda settlement. Fresh water consumption at the mine is of the order of 121,400m^3/y. Industrial water for the plant is recycled, with the make-up water obtained from the wells.

Domestic waste is drained from the mine facilities into septic tanks whilst production waste from the processing plant is pumped to the tailings pond.

The population of Irokinda village comprises approximately 400 local inhabitants and 700 company employees on a rotational schedule and there is an ample supply of skilled personnel sufficient to cater for both the underground mining operations and the surface facilities.

2.2.4 Topography and Climate

Irokinda is located at an altitude of approximately 1,350m. The landscape in this region changes from alpine in the north to middle alpine taiga to the south, with elevations ranging from 1,100-1,500m. The main geographical feature in the region is the Yuzno-Muisky mountain ridge with the Irokinda deposits located on its southern flank.

Due to the rugged relief, a considerable number of road sections have been built in a serpentine pattern on the mountain slopes, with the maintenance the responsibility of the mine. In addition there are a number of local roads which provide access from the community of Irokinda to the mine and mill facilities, as well as the local exploration roads.

The climate of the region is severe continental with long cold winters and short wet summers and great fluctuations in the daily and yearly temperatures. The winter season extends over 7 months with snow cover usually lasting 234 days. The average winter temperature ranges from -22°C to -32°C in January (the coldest month). Summer temperatures in July range from 15°C to 17°C. The average annual temperature is minus 6.7°C.

The average precipitation is 462mm, most of which falls in summer (71.7%) and only 2.5% in winter, with the remainder occurring during the spring and autumn seasons.

2.2.5 Drainage

The main river drainage system in the area of Irokinda mine is the Tuldun river, and its tributaries the Tuluya and Irokinda rivers. The mine is located in the Irokinda river basin. All rivers in the area are fed by local rainfall throughout the spring and summer and by melting snow during the spring. The primary type of underground water in the region is permafrost water.

Permafrost occurs in many areas, with the depth in the area of the mine reaching 200m and the depth of seasonal thawing in the permafrost is between 0.5-3.5m.

2.2.6 Seismicity

The mine is located in a seismically active region with maximum earthquake probability ranging from 6 to 9 on the GEOFIAN scale.

2.2.7 History of Ownership

Mining at Irokinda began in 1974 and prior to 1991, operations at Irokinda mine were undertaken by the State owned Zabaikalzoloto.

In January, 1991, a state gold mining company (Buryatzoloto) was founded with its assets comprising the Tsipicanskiy placer mine and both the Irokinda and Zun-Holba mines. From 1992 to 1994, 140,000t were mined and processed producing 1,565.9kg of gold.

With the breakup of the USSR the Tsipicanskiy placer mine became a joint stock company; Buryatzoloto received exploration-mining licences for the Irokinda and Zun-Holba deposits and between 1994 and 1995

became an open joint stock company with re-registered licences. At that time 30% of the company shares were acquired by High River Gold, a Canadian gold mining company and by 2005 the company had acquired 85%.

2.2.8 Mineral Rights and Permitting

The exploitation licence is 227km^2 in area, covering the Irokinda deposit, and has the status of a "mining allotment" with an unlimited licence depth for mining gold and silver deposit as well as the right for any exploration works. The licence was granted on 08 May 1998 and expires on 31 December 2012; the corner co-ordinates of the licence are provided in the Table 2.1 below.

Table 2.1: Irokinda Licence Co-ordinates

Point	Latitude (North)	Longitude (East)
1	56° 03` 18``	115° 02`49``
2	56° 05`18``	115° 10`19``
3	56° 00`31``	115° 13`57``
4	56° 58`57``	115° 15`02``
5	55° 59`20``	115° 21`22``
6	55° 58` 53``	115° 21`25``
7	55° 55` 51``	115° 16`49``
8	55° 56` 21``	115° 15`02``
9	55° 55` 55``	115° 14`13``
10	55° 56` 53``	115° 10`58``
11	55°55` 02``	115° 10`52``
12	55° 52` 52``	115° 04`57``
13	55° 56`52``	115° 00`56``
14	55° 58` 43``	115° 00`07``
15	55° 58` 41``	115° 06`15``

The location of the licence is shown in Figure 2.2 below.

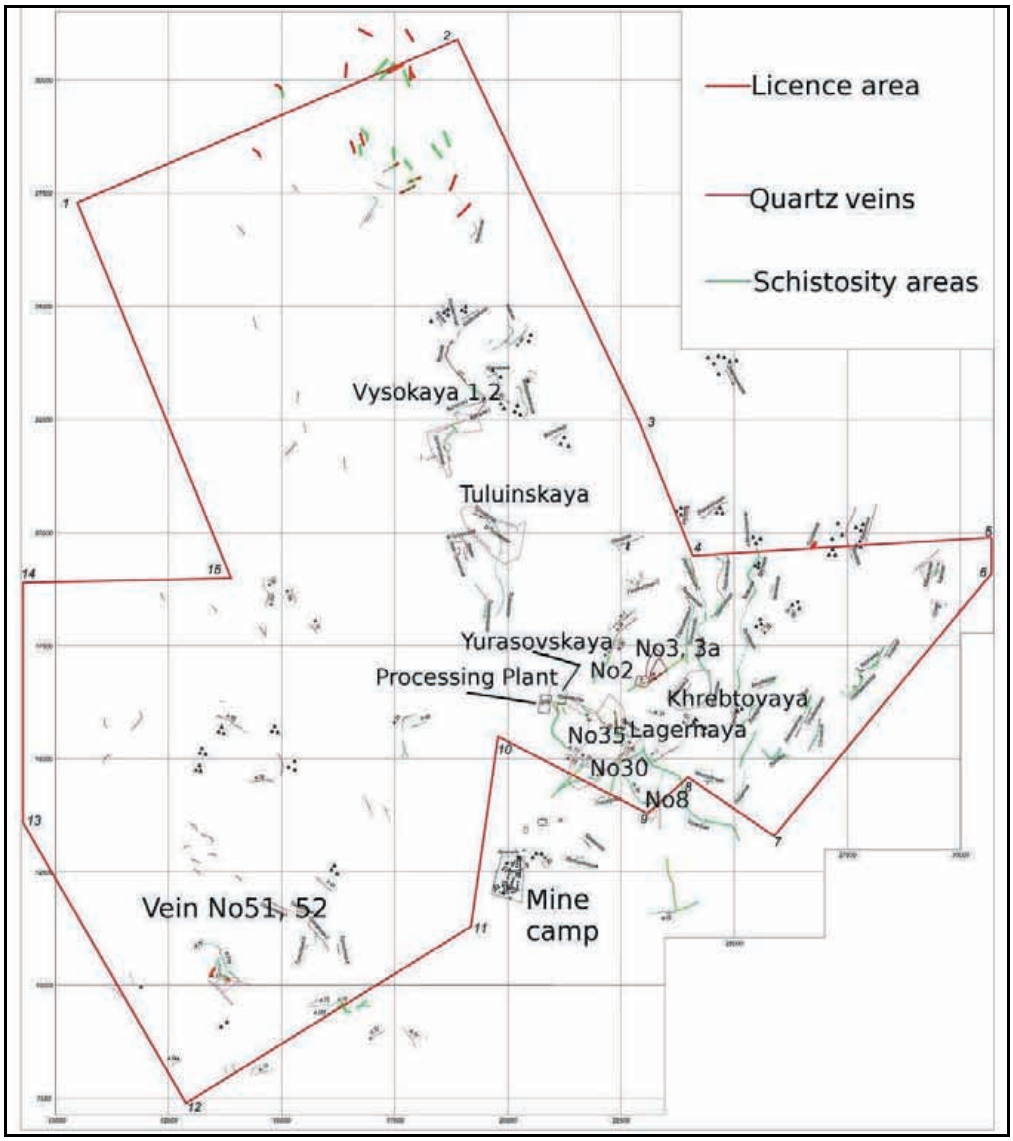


Figure 2.2: Location of the Licence Area

WAI Comment: *An inspection of the licence documentation has shown that all are in good order and suitable for the future needs of the Company.*

2.3 Geology and Mineralisation

2.3.1 Regional Geology

The main structural unit in the region is the Uzhno-Muiskaya formation, one of the blocks of the Archean basement, which is constrained on its western and eastern sides by contacts with the Kelyansakaya and Tuldunskaya mobile zones, respectively, and on the northern and southern boundaries by the Muisky and Tilishminsky faults, respectively.

A plan showing the regional geology of the Irokinda district is given in Figure 2.3.

The *Archean* formations are represented by the Uzhno-Muiskaya formation, a 2,500m thick, dislocated series of amphibole, biotite-amphibole, biotite- garnet-pyroxene gneisses and calciphyres, which have been ruptured by dykes and sills of intrusive rocks. Metamorphism of the rocks is of granulite facies.

The *early Proterozoic* rocks comprise acidic flows, green schists and carbonaceous rocks and have a thickness of approximately 7,000m.

Molasse sediments of *late Proterozoic* age comprise two sub-suites; a lower (sand schists) and an upper (sand conglomerates). The thickness of the sediments is approximately 2,500m.

Quaternary unconsolidated sediments are rare and occur in river valleys and along the slopes of the dividing ranges.

Magmatism within the region is complicated and there are a variety of magmatic products, separated either by age or petrography with both Archean and early Proterozoic intrusive units identified.

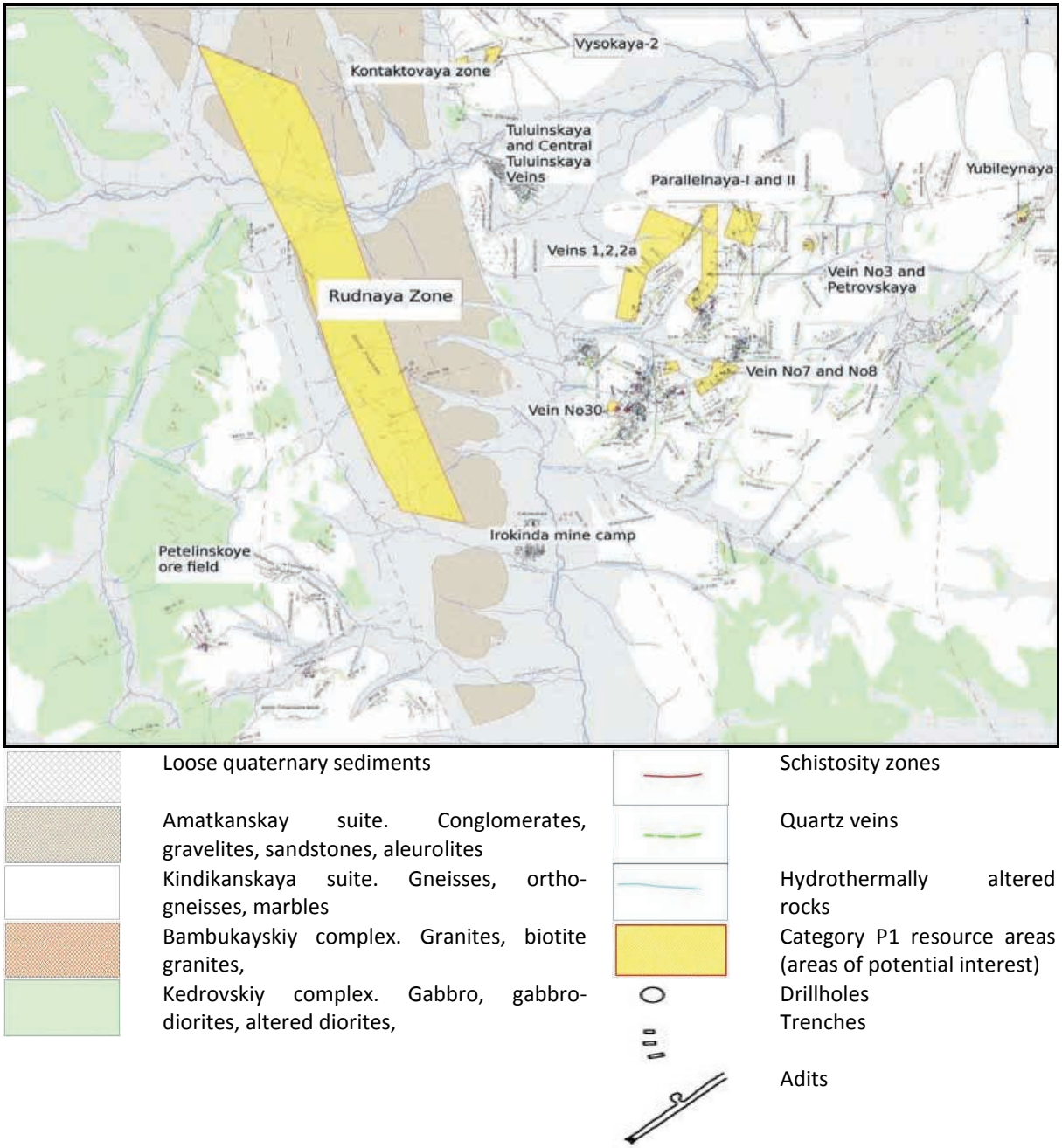


Figure 2.3: Regional Geological Map of the Irokinda District

2.3.2 Mine Geology & Mineralisation

2.3.2.1 Vein Types

In total approximately 140 quartz veins have been identified at the Irokinda deposit, of which some 30 veins contain "reserves", with average gold grades in the different veins between 10.0-36.5g/t. The gold bearing quartz veins typically show the following characteristics:

- Confined to three main fracture systems: 45-60°, 300-320° and 5-20°;
- Flat-dipping (25-45°) to the west;
- Strike lengths varying from 60 to 400m; down dip extent > 300m;
- Complicated morphology with pinching and swelling (up to 3-4m) of the veins which may alternate over a distance of 20 to 30m;
- Economic mineralisation is localised into shoots with strike lengths varying between 30 and 350m and a depth extent in some shoots up to 350m;
- Irregular gold distribution;
- Quartz veins distributed between elevations from 1,740m - 875m;
- 20m wide zones of beresitic alteration almost always present around veins;
- Contacts with the host rock contacts are clear, linear or embayed and combined generally with a 3m wide zone of schistosity; and
- Gold-quartz mineralisation was generated during several paragenetic stages and has a clear mineralogical zonation.

2.3.2.2 Mineralisation

The deposit consists primarily of quartz with less than 3%, content of other mineral, comprising a low sulphidation assemblage: the most abundant minerals are pyrite and galena, with rare chalcopyrite and fahlore, plus accessory pyrrhotite, arsenopyrite and scheelite. Epigenetic minerals include iron hydro-oxides, malachite, azurite and jarosite as well as others.

Five paragenetic mineral associations corresponding to five stages of mineral genesis have been recognised:

- Quartz;
- Tourmaline-scheelite-quartz;
- Carbon-quartz;
- Gold-sulphide-quartz; and
- Carbon.

2.4 Exploration

2.4.1 Historical Work

The Irokinda deposit was discovered in 1959 by the Oktyabrskaya partnership of the state owned company Buryatgeologia and was prospected during the 1969 to 1975 field seasons. During this period six zones were explored and proven to contain mineable "reserves".

From 1960 to 1989, the exploration of several veins was undertaken by the Bagdarinskaya expedition and after approval of the first "reserves" by the State Reserves Committee (SCR) of the USSR, continued exploration occurred in the central part of the ore field and on its flanks (1975-1991).

During the period from 1993 to 2006, Buryatzoloto has explored and taken to a mining stage 11 additional veins.

2.4.2 Current Exploration Practices

2.4.2.1 Sampling Methods

A combination of rugged landscape, the mode of occurrence and morphology of the veins, and extremely irregular distribution of gold make the exploration of this area very difficult to explore on the surface.

The exploration of vein continuity along strike and at depth is performed using surface trenches and drill holes, whilst the main exploration, unusually, is conducted using underground mine development.

2.4.2.2 Trenching

Surface exploration includes trenching at spacing of 20-40m and stripping to expose the veins on surface. The length of individual channel samples is not more 1.0m with a cross-section of 5 x 10cm, and weight varying from 13 - 15kg.

2.4.2.3 Drilling

Veins tested by trenches and stripping are further evaluated to depth by drilling both vertical and inclined holes, typically on a grid of 160 x 80-160m, with further infill drilling on a 40x40m pattern where defined mineral shoots are intersected.

The same methodology is used underground for testing the depth extension of veins being mined, with additional cross-cuts into the hangingwall where drilling chambers are excavated for this purpose. The whole core is sampled up to a maximum of 1m without splitting, together with 0.5m samples from footwall and hanging wall, and recovery through the mineralisation is typically 98 to 99%.

2.4.2.4 Exploration Underground Development

During the exploration stage the distance between adit levels along the dip of the vein does not usually exceed 80m and exposure of the vein is accomplished by driving along strike until the quartz fully pinches out, often beyond the limits of known drilling. Channel sample intervals are 2.5–3.0m and samples may also be taken from the walls and roof of the drive.

If the vein increases in width or abruptly changes strike, cross-cuts every 5m and raises at 50-60m spacing are developed to expose the hangingwall/footwall or verify the up-dip continuity respectively.

WAI has not conducted any verification sampling, but it is understood that Micon (NI 43-101 2008) believes that the sampling results can be used for the resource estimations conducted at the Irokinda mine.

2.4.2.5 Buryatzoloto Sample Preparation & Analysis

Channel samples are prepared in the crushing facility at Irokinda mine, whilst core samples are prepared at the crushing facilities of the Tuluinskaya Exploration Partnership.

Micon considered that the Quality Assurance/Quality Control (QA/QC) procedures and protocols employed at the Irokinda mine were rigorous enough to ensure that the sample data are appropriate for use in mineral resource estimations."

2.4.2.6 Surface Exploration Potential at Rudnaya

Recent surface exploration has taken place to the west (Rudnaya) on a prospective structurally complex gold occurrence hosted by terrigenous formations on the contact with intrusives of granite-gabbro. To date,

exploration works have consisted of trenching in the most accessible areas, and a small number of exploration drill holes to test at depth.

Gold mineralisation is found in hydrothermal metasomatic alteration and represented by zones of beresitisation and brown feldspathic rocks (Figure 2.4).

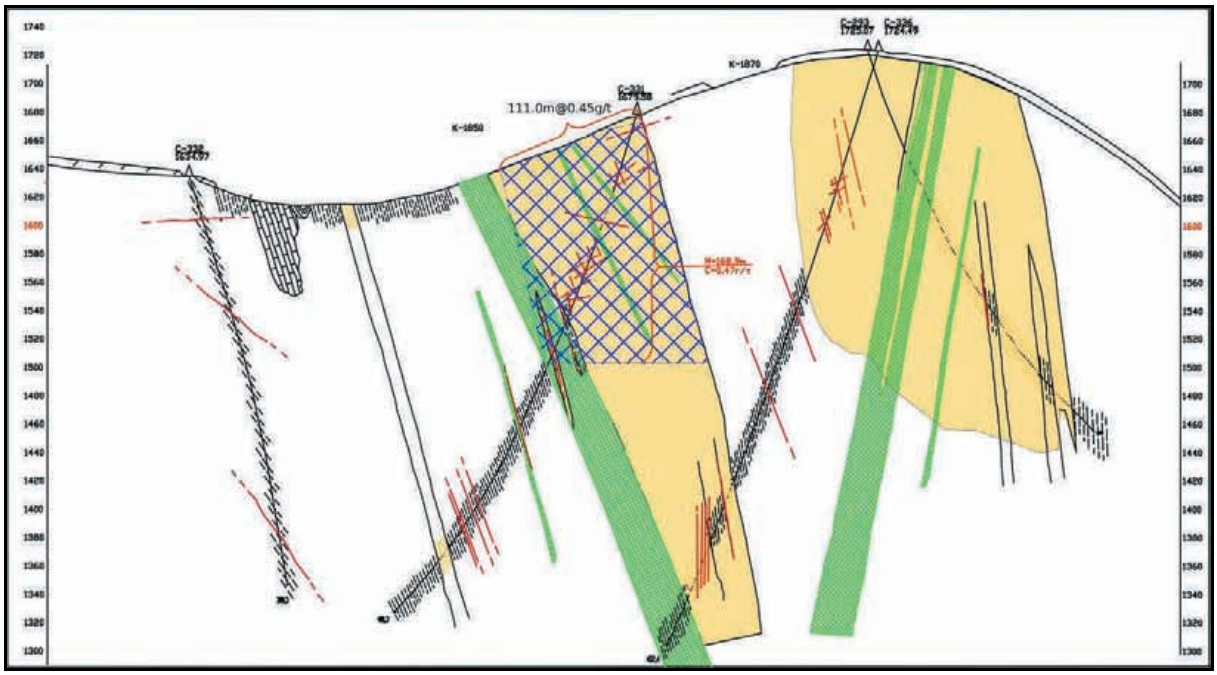


Figure 2.4: Cross Section through Rudnaya Exploration Prospect – showing Location of Metasomatic Zones

Quartz veinlets, small veins and swarms, with a thickness up to 0.8m are observed in many locations and large quartz outcrops are exposed along the overall strike of the zone.

Gold mineralisation is not uniformly distributed, with the most developed mineralisation grades varying from 0.2- 34.6g/t Au, whilst low gold grades are prevalent, with individual high grade sectors, most probably confined to quartz veins and veinlets.

The increase of grades is normally related to beresitisation and brown feldspar formation zones, where the formation of pyrite, galena and chalcopyrite have been observed.

To the north and to south from the extension (600m wide) zone, mineralisation appears to be decreasing, with limited significant assays.

2.5 Resources and Reserves

2.5.1 Methodology

All projects within the Russian Federation use the state approved classification and all reserve estimates must be approved by the state prior to any mining occurring on a mineral property. For the Irokinda deposit the "Geological Block" method (similar to the polygonal resource estimation method) has been used historically and is still being used to estimate the "reserves".

"Operational reserves" are estimated from the tonnages of the geological in-situ "reserve" by applying factors for mining recovery (loss in mining, %) and dilution.

2.5.2 Database

The Irokinda mine property database for "reserve" estimation consists of underground sampling of drifts, cross-cuts and raises, as well as numerous underground and surface diamond drill holes which are surveyed and inclinations recorded at 15m intervals.

The drill core is logged by the geological staff of the Irokinda mine and detailed information on each hole is compiled and recorded.

2.5.3 Density Used For "Reserve" Estimate

At the Irokinda mine the density varies slightly between the various mineralised zones within the deposit (2.56–2.80t/m^3). The average bulk density of 2.64tm^3 is used for "reserve" estimation.

2.5.4 Top-cutting

Capping of the high metregram value of the composite occurs by replacing the higher value with the value closest to either 110% (if the number of composites >25) or 120% (if the number of composites <25) of the average metregram (grade x thickness) value for a block.

2.5.5 Parameters Used for Current Reserve Estimation

The following parameters are used to define the "reserves" considered to be mineable:

- The minimum (cut-off) grade to outline the width of the mineralisation or vein, is 3g/t Au;
- The minimum gold grade in an estimated block is 6.8g/t Au; note:
- The minimum mining width included in a "reserve" estimate is 1.0m and, in the case of smaller widths, corresponding grade x thickness values are applied to see if the areas with smaller widths can be included;
- The maximum low grade mineralised vein (zone) strike extension included in an estimated "reserve" outline is 12.0m;
- The minimum silver grade taken into account in the "reserve" estimation is 3g/t Ag;
- The cut-off gold grade necessary to delineate off-balance "reserves" is 1.0g/t Au; and
- The minimum mineralised vein width needed to estimate off-balance "reserves" is 1.0m.

The price of gold used to estimate the parameters was US$550/oz.

> ***WAI Comment:*** *the use of such a low gold price will naturally constrain the mineralisation that will be included into the "reserve" calculation, although as the ore zones have fairly clean contacts, this may not be as limiting as thought.*

2.5.6 Buryatzoloto "Reserve" Inventory as of 01 November 2010

Buryatzoloto conducts a quarterly "Reserve" Inventory for company use and an annual "Reserve" Inventory as of December 31 each year, which is created in a special statistical form and submitted to the state as part of Buryatzoloto's reporting requirements. This "reserve" estimate is conducted according to the Russian "reserve" classification system.

Previous resource and reserve estimates conducted by the staff of Buryatzoloto's Irokinda mine were audited by Micon (2007 and 2008) which converted the current Russian classified "reserves" into Mineral resources and Ore reserves which comply with the current CIM standards and definitions for estimating resources and reserves as required by NI 43-101 regulations.

Any differences between the Russian classified "reserves" and the mineral reserves complying with CIM guidelines are the result of differences in the classification systems and not the result of the estimation methodologies used since, in general, the parameters used to classify the "reserves" according to the Russian system were maintained and used to convert the mineralisation into CIM compliant mineral resources and mineral reserves.

The Irokinda project falls into a 'vein type gold deposit of the third category' and, as a consequence the mine can report predominantly C_1 and C_2 "reserve" categories, and usually only small amounts of the B category based on underground development.

The Irokinda "Reserves" as of 1st November 2010 are shown in Table 2.2 below.

Table 2.2: Irokinda "Reserves" as of 01 November 2010

"Reserve" Category (Dilution and recovery factors are not applied)	Ore (kt)	Au (kg)	Au Grade (g/t)
C_1	149,555	2,408	16.1
C_2	191,680	3,510	18.3
C_1+C_2	341,235	5,918	17.3

WAI Comment: *taking into account the current mine production rate (approximately 300ktpa), the remaining "reserves" will provide approximately 1.5 years of mine life. However from past experience, the mine continues to discover vein extensions and new ore zones, a situation that WAI believes will continue.*

2.5.7 WAI Subjective JORC Assessment

As part of the resource review, WAI was asked to provide a subjective opinion of the Irokinda resource statement in terms of its equivalency to JORC Code (2004) guidelines.

Without re-modelling the deposit, it is clearly not possible to provide a definitive opinion on this matter, although for the purposes of this report, WAI accepts the approximate equivalence of the B category with *Measured*, C_1 category resources with *Indicated* and C_2 with *Inferred*.

Considering the above, a resource statement complying very approximately with JORC Code (2004) guidelines as of 1st November 2010 is presented in Table 2.3 below.

Table 2.3: Irokinda JORC Equivalent Resources as of 01 November 2010

Resource Category	Ore (t)	Au (kg)	Au Grade (g/t)
Indicated	149,555	2,408	16.1
Inferred	191,680	3,510	18.3

2.5.8 WAI Review of the Current Reserve Base

WAI has reviewed each of the most significant Irokinda reserve blocks on a vein by vein basis, and, in general, the results correlate well with the estimate presented above, confirming the relatively small remaining reserve tonnage at the mine.

The exploration programme was well behind schedule and as a result, the mining operations are rapidly depleting the reserve base which cannot be replenished at the same rate. However, the current aggressive drilling programme is going someway to alleviating the situation.

WAI Comment: *proving the down dip extension of a number of the key orebodies requires time and capital investment (which the Company is now implementing) and the mine may become exhausted before further reserves can be delineated. Specific targets defined by the Company are as follows:*

Vysokaya Vein - priority target for mining in 2010. Resources (extrapolated down dip 40m) are quoted as 31,294t at 19.8g/t Au containing 619.6kg of Au in the C_2 category.

Tulinskaya - access by ramp to the 890m level – but this is at the water table. An exploration plan to drill this target beneath the water table has been proposed for 2010.

Vein No.3 - this is a priority development target has a total reserve of 154,454t at a grade of 18.7g/t Au containing 2,882kg of Au, of which 117,723t at a grade of 17.1g/t Au is in the C_2 category.

Vein 35 - this is also a priority development target containing 81,691t at a grade of 14.9g/t Au containing 1,213.2kg of Au in the C_2 category. The reserves are contained within underslung blocks below the lowest level at 959m level.

Jubilanee Vein - reserves have been identified down dip of the lowest level. This structure requires further verification drilling from hangingwall cross cut drift; together with development of two new level horizons beneath the existing lowest level drive.

2.5.9 Exploration Programme for 2010-2011

Buryatzoloto has undertaken an aggressive budgeted exploration programme for 2010 and will continue to do so in 2011, in order to update the level of reserves as soon as possible. The proposed exploration programme and progress achieved to date is given in Table 2.4 below.

WAI Comment: *WAI believes that the proposed exploration programme although aggressive, is achievable, and the results of which will be key to the future of the mine.*

Table 2.4: Irokinda Exploration Work Programme For 2010 & 2011

			2010					2011			
			Volume of works		Estimated increase of resource (t Au)			Volume of works	Estimated increase of resource (t)		
Mine	**Exploration Target**	**Type of works**	**Plan**	**Actual as at 1.11.2010**	**C2**	**P1**	**P2**		**C2**	**P1**	**P2**
Irokinda	Irokindinskoye and Petelinskoye ore fields	Auxiliary mining works (m or m^3)	1,510(11,016)	1,161(8,200)	2	11.5	13	2,800(20,720)	3	10	0
		Underground drilling (m)									
		Surface drilling (m)	17,720	13,252,7				17,000			
		Trenching (m^3)	47,620	45,063.7				36,260			
		Roads and drilling site	6,000	6,000				7,000			
		preparation (m^3)	65,500	80,076				60,000			
	Kelyano-Irokindinskiy greenstone belt	Geochemical works (km squared)	50	50	0	6	13	0	0	0	0
		Geophysics (km squared)	50	50				0			
		Trenching (m^3)									
		Roads and drilling site	12,800	13,866				0			
		preparation (m^3)	15,700	4,000				0			
		Surface drilling (m)									
			0					0			

The Irokinda budget for 2010 totals RUR279.5M or US$9.572M, the fulfillment of budget as at 01 November 2010 was RUR272.3M or US$9.325M (The exchange rate used is RUR29.2/one dollar).

The Irokinda budget approved for 2011 totals RUR287.373M or US$9.579 M (The exchange rate is RUR 30 for one dollar).

2.6 Mining

2.6.1 Introduction and History

Irokinda is an operating mine, the first regular production dating back to 1974, with 140 spatially isolated gold vein occurrences, 30 of which are economically mineable, spread over a 227km^2 licence. Initial trial operations took place until 1991, when Buryatzoloto was founded and became the owner of the Irokinda and Zun-Holba mines; and a placer deposit at Tsipikanskiy. In 1995, High River Gold acquired 30% of Buryatzoloto's stock, which, along with re-organisation of the company, forced an increase in production and corresponding development of the project (Table 2.5).

Table 2.5: Production for Irokinda Mine (1985 - H1 2010)

Year	Run-of-Mine Diluted Ore (t)	Average gold grade (g/t)	Contained gold (kg)	Ore, Processed at Plant (t)	Refined gold (kg)
1985	-	-	-	3,310	125.0
1986	-	-	-	7,630	154.0
1987	-	-	-	11,800	142.0
*1988**	*7,430*	-	-	23,800	241.0
*1989**	*35,278*	-	-	32,000	344.2
*1990**	*31,600*	-	-	28,030	431.8
1991	28,840	14.6	419.8	32,460	341.7
1992	38,380	18.1	696.0	34,760	515.8
1993	50,079	15.2	762.8	48,000	685.8
1994	44,192	11.1	491.9	50,497	364.3
1995	58,057	9.9	573.2	62,888	545.2
1996	96,784	11.5	1,117.4	96,128	773.9
1997	87,618	12.9	1,133.4	83,057	891.6
1998	83,492	13.6	1,137.7	85,284	1,001.0
1999	106,556	12.5	1,331.2	96,786	1,029.1
2000	158,538	11.9	1,879.7	152,106	1,625.3
2001	235,816	11.4	2,676.7	220,691	2,274.6
2002	245,971	11.6	2,844.7	234,609	2,530.8
2003	287,929	10.4	2,980.3	255,913	2,610.9
2004	327,015	9.0	2,934.9	308,127	2,658.8
2005	329,080	8.9	2,922.0	300,317	2,590.3
2006	307,592	9.7	2,987.4	280,097	2,490.3
2007	301,954	9.2	2,783.7	275,485	2,339.7
2008	310,627	8.6	2,684.3	290,792	2,313.0
2009	332,693	8.5	2,827.0	316,004	2,340.7
1H2010	152,939			148,680	1,069.8

2.6.2 Mining Activities

Major ore production for 2009 was focused in the following areas (estimated predicted mined ore for Jan-Dec 2009 is given in brackets):

- Vein No.30 (93.5kt);
- Lagernaya Vein (72kt);
- Vein No.3 (40kt);
- Vein No.35 (19kt); and
- Vein No35a (15kt).
-

Major development is planned for vein No.3, where 200m deep shaft ("Shaft 3G") is to be sunk. This will allow to access additional resources of 140kt of ore at a grade of approximately 9g/t Au.

2.6.3 Mining Method

2.6.3.1 Rock Properties and Geotechnical Conditions

The rocks of the deposit are hard, with minimal natural moisture content (0.1-0.3%), and low porosity (1.3-3%). The vast majority of the underground workings are located in stable areas, where no ground support or stope reinforcement is required, and above the water table, which, together with permafrost, eliminates water inflow into the workings.

The majority of underground support is present in underground infrastructure workings rather than stopes. In areas of unstable ground, some bolting or screening is applied.

Although underground conditions are generally favourable, rock stability requires careful monitoring in order to minimise risk of accidents, especially in areas where recovery of pillars has taken place, and some individual failures have been registered in the past.

The water table is currently located at an elevation of approximately 890-900m level. Any mining below this level would involve installation of pumps, and potential problems with high water pressure at depths.

Reserves above the water table are rapidly being depleted and future exploration strategy will concentrate on finding additional reserves on the flanks of known veins and in new areas of the property.

2.6.3.2 Mining Methods Employed

There are three main mining methods employed in underground operations at Irokinda:

Room-and-pillar mining is used for shallow dipping (35-40°), relatively thick veins (up to 18m), when mining of the vein is performed in several layers/benches, and the ore is hosted in stable and very-stable rocks. The size of chambers (rooms) is up to 8x8m, with pillar size varying depending on rock conditions (normally in the range 1.5-2m x 1.5-2m).

Up-dip mining (selective mining) is used in stable and medium-stable ground. The implementation of the method is not restricted by vein dip angle, and is used for medium and thin veins (1.1-3.0m). Extraction of the material is performed along the strike of the vein, with mining faces of 25-30m width and a sub-face of 8-10m. Drilling is accomplished using a rig installed on a sledge. The area adjacent to the face is supported by hydraulic uprights located 5-6m from the face, depending on ground conditions. After emptying the stope, the mined area is treated by a vacuum machine in order to collect gold-bearing ore dust.

Drift mining with rock fill is employed for extraction of very thin veins (thickness <0.8m). It is a very selective and labour-intensive method, suitable for solid and moderately-stable rock. Ore and waste are blasted separately, using 1.8m blastholes. After blasting the ore is mucked by scraper into ore passes. Waste is then used to fill the void. The amount of waste to blast is chosen depending on the quantity required to fill the space and the minimum feasible height for mining the stope.

2.6.3.3 Ore, Low-Grade Ore and Waste Stockpiles

Each adit, where intensive operations take place, has similar surface infrastructure, consisting of high-grade ore, low-grade ore and waste unloading facilities and mine safety equipment storage. Waste is normally dumped in close proximity to the adit portal, and ore (both low-grade and high grade) is stored in small-sized intermediate stockpiles. In some cases the ore is delivered to the plant stockpile directly.

The high grade ore is loaded into trucks with use of scrapers or loaded into trucks directly from wagons.

2.6.3.4 Mining Equipment

A list of the major items of mining equipment is given in Table 2.6 below. It should be noted, that this is not a complete inventory of the machinery at Irokinda, but a summary list of the equipment which is directly employed in the underground operations.

Table 2.6: Major Mining Equipment

Wagon (0.8m^3)	416
Overhead loader	31
Locomotive	52
Scraper	106
Ventilation Fan	62
Hoist Cage	5

2.6.3.5 Ventilation

The vast majority of the developed veins require ventilation. The most common type of ventilation scheme is employing two main ventilation fans at the flanks of the workings and smaller local fans for assistance at stope, drift and other faces.

2.6.4 Mine Personnel

There are two departments directly involved in mining operations, each comprising a face team, stope team and shaft team, with the requisite management and specialist staff. Underground exploration and development works are performed by a separate department which also has the same organisational structure. The three teams have a combined personnel complement of 816 and the overall number of employees at Irokinda (inclusive of administration, processing plant and auxiliary services personnel) amounts to 1,430.

Mining is conducted on a daily basis, with two 10.5 hour shifts per day.

> ***WAI Comment:*** *due to the distance between the individual veins and the production sites on them, personnel requirements are higher than those generally required for similar sized operations. This has a negative effect on the operational costs of production.*

2.6.5 Conclusions

Irokinda is a mature operation, well-setup with all necessary infrastructures, labour and other facilities. Although a significant part of the main mining equipment is relatively old, the mine is still capable of meeting scheduled production. The remaining reserves are sufficient for approximately 1.5 year of production, although the mine has some potential for resource increase. The current exploration programme will go some way to alleviating the problem of the small reserve base, assuming positive results.

2.7 Processing

2.7.1 Introduction

The processing operations at Irokinda have been in operation in their present form since 1988 and comprise two plants using essentially the same flowsheet, with crushing, grinding, gravity and flotation processes. The Main Plant is capable of processing 30-31tph through two parallel lines whilst a second (''Seasonal'') plant, operates from March to September, at a rate of 12tph. This seasonal plant will operate all the year round from 2011.

The gold is recovered into one of two products:

- A gravity concentrate which is smelted on site to produce dore; and
- A flotation concentrate which is shipped to a carbon in pulp (CIP) cyanidation plant at Zun-Holba.

The ore types at Irokinda are exceptionally easy to process, with a high proportion of free gold resulting in a gravity recoverable gold (GRG) content of approximately 65%. The gravity tailings respond well to flotation and the low sulphide content of the ores results in high-grade flotation concentrates.

2.7.2 Mineralogy

The ores, of generally similar mineralogy, consist predominantly of quartz together with low levels (0.1 to 0.3%) of sulphides, the latter having a positive impact on the ores processing characteristics. The gold is relatively coarse grained and easily amalgamated (71.2-88.7%) with grades ranging from 6.6 - 27g/t and silver grades from 6.5 up to 27.0g/t.

The degree of sulphide oxidation is low as a whole with only pyrite and galena forming finely disseminated impregnations in fractures in the quartz along the grain contacts. Non-ferrous metal sulphides are sphalerite, galena, chalcopyrite, acinthite and fahlore which are intimately associated with each other. The minerals have relatively clean surfaces, although sometimes galena and chalcopyrite show signs of oxidation.

Mineralogical descriptions of different ore sources are given in Table 2.7 below.

Table 2.7: Mineralogy of Irokinda Ore Sources

Minerals	Ore Source					
	No. 35	No 3	Tuluinska	Serebrya kovskaya	No 30	Yurasovskaya
Quartz	68.1	76.0	75.0	85.1	91	85
Carbonates (calcite, siderite, dolomite)	13.8	7.2	6.0	4.8	1.3	1.8
Amphiboles, pyroxene, garnet	3.0	3.8	5.1	7.2	5.7	4.8
Feldspar, kaolinite	14.6	12.4	13.2	2.5	1	7.1
Pyrite, pyrhottite, galena, chalcopyrite	0.2	0.2	0.3	0.2	0.6	0.2
Acanthite, fahlore	trace	trace	0.2	trace	trace	Trace
Limonite,magnetite,secondary copper lead and iron minerals	0.1	0.4	0.2	0.2	0.4	1.1
Accessory: scheelite, fluorite, barite, apatite, rutile	0.2	trace	trace	trace	trace	trace

Silica is the most abundant mineral in each of the ore sources. The No. 35, No. 3 and Tuluinska veins contain significant levels of feldspar and kaolinite. The levels of other minerals are generally low with amphiboles, pyroxene and garnet present between 3.0-7.2%.

> ***WAI Comment:*** *WAI notes that no metallurgical testing of the Vysokaya deposit had taken place, despite the importance of this ore source in the near future.*

Gold in the veins occurs as free gold which is spatially associated with quartz, and with sulphides. In quartz, the gold occurs as separate impregnations and inclusions which are distributed along fractures, as well as fine interstices and cavities. Gold grain size ranges from 0.04 - 1.3mm. The deportment of the gold in different veins is summarised in Table 2.8.

Table 2.8: Gold Deportment in Ore Sources										
Form of Gold Occurrence	Sample Location									
	No 3 Adit		Tuluinskaya Vein		Serebryakov Skaya Vein		No. 30 Vein		Yurasovskaya Vein	
	g/t	%	g/t	%	g/t	%	g/t	%	g/t	%
Free grains	5.61	85	16.38	86.3	5.05	71.2	21.73	88.7	22.03	84.1
In joints but native form	0.82	12.5	2.4	12.6	1.93	27.1	2.65	10.8	3.99	15.2
Total cyanidable	6.43	**97.5**	18.78	**98.9**	6.98	**98.3**	24.38	**99.5**	26.02	**99.3**
Finely impregnated in quartz and sulphide	0.17	2.5	0.2	1.1	0.12	1.7	0.12	0.5	0.18	0.7
Total	**6.6**	**100.0**	**18.98**	**100.0**	**7.1**	**100.0**	**24.5**	**100.0**	**26.2**	**100.0**

2.7.3 Processing Flowsheet

2.7.3.1 Introduction

The metallurgical process at Irokinda involves gravity and flotation circuits with the production of both a “gold head” ready for smelting and a flotation concentrate filter cake. Flotation tailings are pumped 650m to a nearby tailings dam.

There are two processing facilities at Irokinda which both utilise similar flowsheets. The Main Plant operates throughout the year, as far as weather permits. There is also a smaller “Seasonal Plant” operating during the summer months which originally treated low grade ore, but now treats ore grades similar to the Main Plant.

2.7.4 Main Plant

2.7.4.1 Stockpile Area and Crushing

The ore is trucked to a stockpile adjacent to a feed hopper, removed from the hopper using an apron feeder and screened with the screen oversize being conveyed to a jaw crusher (400 x 900mm) and the screen undersize is conveyed to an 18mm screen. The jaw crusher product is conveyed to the 18mm screen whilst the screen oversize is conveyed to a KSM-1200 cone crusher and returned to the screen.

The feed to the plant (which operates a 2x12hour shift system) is not sampled and head grades are determined through back-calculation from production data. The maximum feed rate to the plant is 30-31t/hr.

2.7.4.2 Grinding and Gravity

The crushed ore is ground in two process lines where two stages of ball milling are used to grind the ore to 85% passing 200 mesh. As the ore contains significant levels of free gold, gravity processing is used to recover a significant proportion of the gold to a gravity concentrate which is smelted on site.

The primary mills are fitted with a grate discharge and the secondary mills are of the ‘overflow’ type.

The primary mill discharge gravitates to a jig and the jig concentrate is cleaned continuously by two stages of tabling. The final gravity concentrates are cleaned one more time on a small table on a 12 hour batch basis.

The jig tailings pass to a spiral classifier with the sands being returned to the primary mill. The classifier overflow is pumped to a cyclone and the cyclone underflow gravitates to a secondary ball mill and the overflow passes to flotation. The secondary ball mill discharge feeds a second jig and the jig concentrate is pumped to a two stages of shaking table. The jig tailings are pumped back to the cyclone.

2.7.4.3 Gold Smelting

The final gravity concentrates are dried and roasted before being smelted in two stages to a final dore product. The main fluxes are borax, sodium carbonate, silica and lead oxide.

Gravity gold recovery was 66% in 2008 and 64% in 2007.

2.7.4.4 Flotation

The gravity tailings stream is subjected to bulk flotation to recover both free gold and gold bearing sulphide minerals. There are two flotation lines consisting of rougher and scavenger flotation. The rougher concentrates are cleaned in a common three stage cleaning circuit and the cleaner tailings are split and pumped back to the two rougher flotation stages. The cells are 1.3m^2 capacity, and are all naturally aspirated.

The gold and sulphide minerals are collected using butyl xanthate and pineoil is used as a frother.

The final concentrate is thickened and the thickened pulp is filtered using a drum filter. The moisture of the filter cake is in excess of 20% and therefore a drying stage is used to reduce the moisture content to 10-12%.

The flow sheet for the Main Plant is shown as Figure 2.5 below.

2.7.5 Seasonal Plant

The “Seasonal Plant” treats ore during the summer months and was originally designed to treat low grade ore although the plant’s head grades are similar to those of the main plant.
The processing rate is 12t/hr.

The plant uses the same basic process operations as the Main Plant. The plant has a smaller cone crusher and a single stage of ball milling is used in closed circuit with a classifier and cyclones. The gravity circuit is similar to the Main Plant. The flotation section uses a rougher-cleaner cell configuration with two cleaning stages.

2.7.6 Concentrate Transport and Treatment at Zun-Holba

The dewatered flotation concentrate is loaded into large bags and transported (a journey of 10 days) to Zun-Holba where it is sampled again and then processed by fine grinding and cyanidation. The recovery of gold from the concentrates by cyanidation is high at 97-98%.

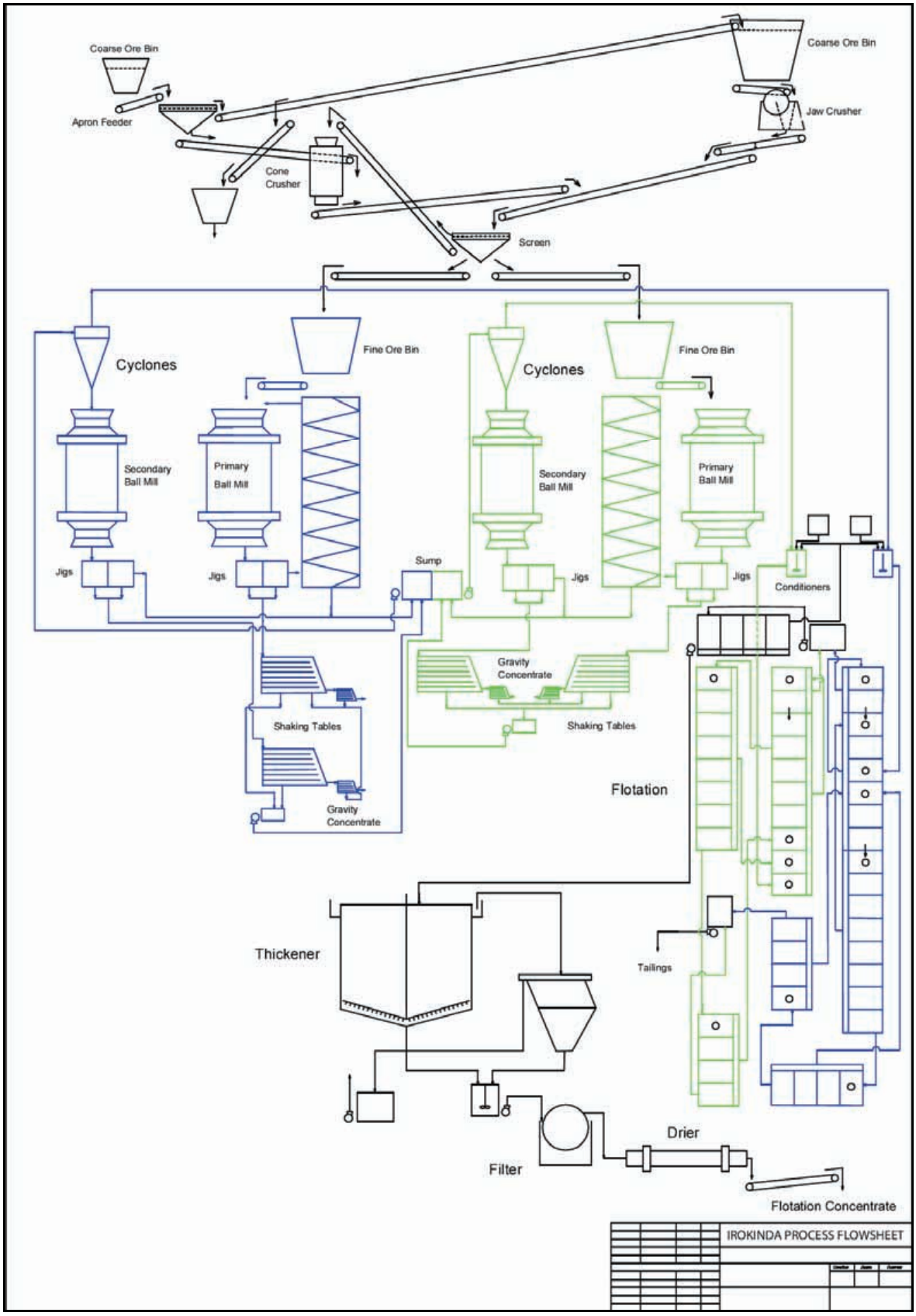


Figure 2.5: Main Plant Flowsheet

2.7.7 Consumables

The consumption of grinding balls is high at 1,700g/t reflecting the hard nature of the ores being treated. The consumption of the flotation reagents is typical for the bulk flotation of a gold sulphide ore whilst the electrical power consumption is also typical for a gravity-flotation operation.

2.7.8 Labour

The plant is operated using a 12 hour shift rota on a two months on two months off basis. There are 59 workers and 6 technical personnel.

2.7.9 Analytical Laboratory

2.7.9.1 General

The Irokinda assay laboratory is accredited by Irgiridmet (Irkutsk) until 2013. The plant analyses underground channel samples, drill core and plant samples. Environmental samples are tested in a separate laboratory.

2.7.9.2 Channel and Drill Samples

Underground channel samples are dried and crushed to pass 1cm and subsequently 2mm using a jaw crusher and a rolls crusher respectively. The crushed product is thoroughly mixed and a 1kg sub-sample riffled out and ground in a disc grinder to pass 0.074mm. Samples are assayed in duplicate. Drill samples are generally smaller (1kg) and follow the same sample preparation flow sheet.

2.7.9.3 Plant Samples

The plant samples submitted on a routine basis are tailings, flotation concentrates, dore, slags and low-grade jig ragging.

2.7.9.4 Fire Assay

The sample weights used in analysis are 50g for tailings, 25g for flotation concentrates, jig ragging and slags and 3g for dore. Standard fire assay fluxes are used.

There are two fire assay and two cupellation furnaces. The gold beads are subjected to standard nitric acid “parting” and are weighed to an accuracy of 0.001g. The limit of detection is 0.02g/t Au.

The laboratory is operated 12 hours per day using 11 personnel on two shifts per day.

2.7.9.5 QA/QC

The laboratory has a comprehensive system of QA/QC analysis using Standard Reference materials. The agreement between the assayed values and the certified values should be within set limits, depending on the reference material analysed. Duplicate analyses which fall outside these limits are repeated.

There is also a regular system of check analysis with an external laboratory (Irgiridmet) which involves 300 samples every 6 months. This exercise is undertaken by the Geological department.

2.7.10 Historic Plant Production

The production records of the Irokinda processing facilities are summarised in Table 2.9, below.

Table 2.9: Irokinda Plant Production Data (2001-2009)					
	Plant Throughput (t)			Feed	Overall *
Year	Main	Seasonal	Total	Au g/t	Recovery %
2010	239,233	73,816	313,049	7.32	95.5
2009	250,659	65,345	316,004	7.75	95.2
2008	241,441	49,351	290,792	8.33	96.0
2007	235,923	39,562	275,485	8.95	96.1
2006	240,519	39,578	280,097	9.29	96.4
2005	239,665	60,651	300,316	9.06	96.1
2004	269,054	39,073	308,127	9.11	95.8
2003	285,823		315,733	9.57	95.9
2002	234,609		239,094	11.20	96.1
2001	220,691		220,691	10.85	95.2
Note: * to dore and flotation concentrate					

It is noticeable that although production through the plant has remained fairly constant, feed grade continues to decline along with a slight decrease in recovery.

2.7.11 Tailings Retreatment

Some 2.8Mt of ore has been treated since 1988 and the resource may be amenable to re-processing using agglomeration and heap leach technology. Assuming an (original) Irokinda plant feed grade of 10.7g/t Au and a process recovery of 95%, the tailings grade should average 0.5g/t Au. At a US$900/oz gold price this is equivalent to US$15/t of tailings. Other low grade resources, following crushing, may also be amenable to heap leaching and could be used to form the base drainage layer.

> ***WAI Comment:*** *The feasibility of re-treating the Irokinda tailings resource should be reviewed by means of a desk study and limited metallurgical testwork.*

2.7.12 Conclusions

The Irokinda ores are highly amenable to processing using gravity, flotation and cyanidation technologies. The ores contain a high proportion of gravity recoverable gold and have a low sulphide content, which results in high-grade flotation concentrates and low flotation mass recoveries.

The plants achieve gold recoveries in excess of 95% using relatively old equipment. The jigs could be replaced with centrifugal concentrators such as Knelsons or Falcons and this would result in an improvement in gravity gold recovery. However, overall recovery improvements would probably be minor, as flotation will always also be required to process the gravity tailings. Installation of more efficient gravity equipment would result in simply moving gold from the flotation concentrate to the gravity concentrate.

The replacement of the rougher flotation cells with larger cells with a low pressure air supply may be should be considered.

The relationship between flotation mass pull and gold recovery should be critically reviewed to determine whether it is economically viable to produce a higher yield of flotation concentrate with higher gold recoveries but with increased concentrate transport and processing costs.

2.8 Environmental, Community, Health and Safety Section

2.8.1 Key EHSC Aspects and Issues

WAI compared existing practices at Nord Gold operations to international standards, codes and best management practices e.g IFC Performance Standards and World Bank Guidelines.

To date, Buryatzoloto has not needed to demonstrate compliance with international best practice such as IFC Performance Standards on Social and Environmental Sustainability, WB/IFC Guidelines on Health and Safety and Equator Principles, since money for the project development has been sourced internally. However, should finance be sought from international financial institutions, it will be necessary to demonstrate compliance with international guidelines.

> ***WAI Comment:*** *WAI recommends that an EHSC Action Plan is developed to demonstrate how this can be achieved, in addition to the resources and timescale required. Current staff has knowledge of, but little experience in introducing the international standards. Development and implementation work has begun, but further assistance will be needed to support this objective.*

2.8.2 Environmental Permitting

It is reported that the Irokinda project is fully permitted by State and Local Authorities and an OVOS (EIA) for the current project was presented and approved in 2001.

The most recent State audits were conducted on 26 December 2008 and 3 April 2009. No prescriptions or changes to the existing environmental standards or operations were demanded. Annual pollution emissions and discharge levels and charges were approved for 2010.

Russian operations have always sought to comply with Russian Federation requirements and continue to meet the applicable standards. Compliance with these standards is monitored and results are reported. Non-compliances are addressed and corrective actions are implemented and reported to regulatory authorities, it is important to note the reluctance of local Russian authorities to change their approach to standards. This does not encourage the implementation of international standards, however the client continues to strive to improve its HSEC programmes and performances.

2.8.3 Environmental Protection

Air, water, flora, fauna and soils are afforded protection in the project lease area and regular sampling and analysis for potential pollutants are carried out.

2.8.3.1 Air

Air quality tests are conducted annually by the Regional Environmental Research Centre for dust, suspended particulates, SOx, NOx, CO and soot amongst other determinands at selected sites. Dust in the PM_{10} range is important to assess hazard to health. Sources of air emissions are boilers, blasting fumes, dust blow, crushers and road surfaces. Air emissions total approximately 380tpa. Estimated Production Hazard Criteria (EPHC) have been calculated according to Russian Federation criteria as non-hazardous. All personnel were observed to be wearing appropriate personal protective equipment and dust masks.

> ***WAI Comment:*** *WAI believes that dust reduction and control is being addressed at Irokinda, in line with international best practice and to reduce risk to personnel.*

2.8.3.2 Water

The environmental laboratory at Irokinda samples surface water at monthly intervals in rivers and streams (May to October) and adjacent to the Tailings Management Facilities (TMF's) but WAI notes that no background samples were taken to detect any pre-existing elevated levels of pollutants or sources of pollution arising in River Irokinda, upstream.

Surface water quality studies in 2004 indicated that the impact of Irokinda mining project on the quality of the Irokinda River was minor and river-side shelter belts and water conservation zones have been defined and agreed.

Water supply for domestic consumption and production purposes is taken from groundwater and domestic effluent drains to cess pits from which it is removed for treatment in settlement ponds, believed to have less than 12 months capacity available.

2.8.3.3 Land

The likely impact on land of the Irokinda project is localised and temporary. The mine is located in an area designated for production activity and all disturbed land will be required to be reclaimed at closure. Any soil which is removed during the course of the project activities is required to be kept and used for reclamation purposes.

Non-toxic solid waste is dumped at surface by 'edge-tipping' near the mouth of production adits which can result in slopes forming at or near the limit angle for safety and WAI is not aware of any geotechnical risk assessment results for these tips. There is evidence of some natural re-vegetation of slopes around mine entrances, but insufficient to aid stabilisation.

Tailings from the plant are pumped to the Tailings Management Facility (TMF) which is close to its design capacity, with 150,000m^3 (12 months) remaining. The original design has recently been altered to create a 2m high inner wall along the centreline of the TMF and a modification of the design to allow a staged lift by increments of 2m is being considered.
Other production wastes are reused, recycled or disposed of offsite in accordance with a waste management plan. State approvals consider that any damage to land, fauna and flora is reversible and is insignificant in the region as a whole.

> ***WAI Comment:*** *the use of mercury-amalgamation which was terminated in 1989 has the potential to result in localised mercury contamination around the old mill and tailings area. It is recommended that further information on the nature, potential extent and scope of site clean-up operations are required.*

2.8.3.4 EHSC Management

Mechanisms for environmental, health and safety management have been introduced for Irokinda. An integrated Health and Safety Management system devised by Buryatzoloto was introduced in 2009 together with new regulations governing production control safety.

The environmental studies carried out at the site are well organised and appropriate to meet state requirements but would need to be extended to satisfy international best practice standards.

The costs of a mine closure plan, developed in 1999 in accordance with the regulations in effect at that date, have been re-cast to 2009 prices to between US$1M and US$7M for closure depending on whether 'book' or realistic figures are taken into account. Regulatory changes and the closure programme, techniques, and costs need to be readdressed and the lack of a closure fund presents a major impact on cash-flow and accounting. It is noted that financial security for mine closure is not common in Russia. Estimated mine closure costs are only reported as financial liability in IFRS financial statements.
A public consultation programme was carried out as part of the OVOS process and agreements are reached on an ad-hoc basis, there being no fixed budget for community development.

WAI note that in 2010 the H&S team from Nord Gold conducted moanagement system training for management and supervision personnel to familiarise them with the concepts and requirements of H&S management systems to meet corporate requirements. A detailed action plan has also been implemented to

enhance the existing safety programs. The action plan was formally put into operation in February 2010. The number of Health and Safety personnel is also being increased to meet the demands of the growing workload.

The Key elements of the action plan are:

- Change the way safety training is provided through the use of more effective communication tools, regular refresher training and more first aid courses;
- Perform detailed job safety analysis in all departments;
- Improve worker safety awareness with the help of the revised job safety analysis, communication of safety instructions during "line-up" meetings and more effective use of the internal employee newsletter to communicate safety messages;
- Implentation of a safety incentive program to motivate better safety attitudes and performances; and
- Work with technical departments to ensure continued full compliance with applicable technical requirements (Examples; manway construction, roof support, raise boring etc)

WAI Comment*: to ensure compliance with National quality standards and for a project to be fully compliant with international best practice the provisions of the latest IFC performance standards and WB Guidelines must be adopted. In particular the following should be reviewed and extended by the client as part of its on-going continual improvement process:*

- *Environmental and Social Action Plan (ESAP) –The ESAP is one of the critical components for a management plan. It is used to ensure that the company continues to improve in all aspects of the project and that IFC Performance Standards and Equator Principles are met. The existing Environmental Action Plan should be extended to cover capacity and organisation, training, operational control and management to ensure compliance with IFC Performance Standards;*
- *Community Health, Safety and Security Plan – several elements of this plan have addressed measures to protect employees at the project, notably through improved occupational health and safety measures introduced at the operations; however international best practice requires plans to ensure that the wider community is considered and an integrated plan should be prepared;*
- *Endangered and Protected Species Protection Plan – this is required to be considered, although it is unlikely to require extensive work to ensure that such species are adequately protected; and*
- *Community Development, Public Consultation and Disclosure Plans – whilst arrangements are in place for wide levels of public participation and assistance is provided on a request led basis, these plans should be formalised. This is particularly important since the community at Irokinda relies entirely on the project for its economic existence. If closure should occur, this would have a major destabilising effect on the community, for which preparation should be made.*

WAI recognises that to date the company primarily sought to acheive national compliance, which has been done, and hence many of the above recommendations have hitherto not been required. However, WAI recognises the company's commitment to achieving international compliance, and highlights the issues above as areas where further actions will be required. WAI also recognises that whilst Buryatzoloto/Irokinda has a good, well motivated, young, environmental, health and safety team, extra support and guidance may be required in order to effectively implement the range of EHSC recommendations made, particularly with regard to setting up an EHSC Management System.

2.9 LOM Plan

The Irokinda mine is an operation which has continuously produced high grade gold ores from a number of veins, with an annual production between 70koz and 83koz of Au per year from 2001 to 2009. The mine currently has a low tonnage -high grade reserve base, but with a proven track record of consistent reserve replacement in line with production since the mine commenced operations in 1974. This situation was common in the gold mining sector of Russia and has been continued to the present day.

The geological characteristics of the mine result in difficulties to prove the reserve base without significant investment in underground development. Subsequently, the mine has never kept a large "reserve" balance, preferring instead to drill and explore annually to replenish mined material. The reserve replacement track record has illustrated an annual replacement ranging from 24-34% per annum in total reserves as shown in Table 2.10 below. The exploration budget has increased under the ownership of Buryatzoloto, up to the last quarter of 2008, but as a consequence of the global "financial crisis" resulted in the company reducing the budget for 2009. This situation has now been reversed in 2010, with an allocated budget of US$11.4M for 2010 and US$17.5M for 2011 to fund an aggressive drilling campaign designed to potentially extend the mine life up to 10 years. This aggressive drilling campaign aims to delineate extensions to known mineralisation both in depth and along strike, and to explore for new orebodies which may be present both at increased depth, on the margins of existing structures and that have not been accessible to drilling until the present.

Table 2.10: Irokinda - Reserve Replenishment and Exploration Budget

Year	Exploration Budget (US$M)	Replenished Additional Reserves (Au t)	Replenished Additional Reserves (Au koz)	% of Reserve Replacement
2002	3.8	3.3	106	34.01
2003	3.1	3.4	109	33.45
2004	2.7	2.7	87	27.87
2005	4.2	1.7	55	19.04
2006	4.6	1.2	39	13.84
2007	5.0	2.7	87	33.87
2008	7.1	2.6	84	32.11
2009	2.5	1.9	61	24.02
Total	**33**	**20**	627	

The reserve development strategy was previously developed in order to maximise cash flow from operations as opposed to extending the mine life of the asset. Going forward, Buryatzoloto intends to focus on reserve development and hence extend the mine life.

At the beginning of 2010, total C_1+C_2 reserves were sufficient for the operation of Irokinda until 2012. The existing C_1 and C_2 "reserves" (comparable with *Indicated and Inferred* resources using an approximate reconciliation with JORC Code (2004) guidelines) as of 01 November 2010 are given in Table 2.11 below.

Table 2.11: Irokinda "Reserves" as of 01 November 2010

"Reserve" Category (Dilution and recovery factors are not applied)	JORC Equivalent	Ore (t)	Au Grade (g/t)	Au Metal (kg)	Au Metal (koz)
C_1	*Indicated*	149,555	16.1	2,408	74.9
C_2	*Inferred*	191,680	18.3	3,510	109.2
C_1+C_2		**341,235**	**17.4**	**5,918**	**184.1**

Based on the remaining reserve base, WAI is not able to produce a LOM plan and economic model under the guidelines of the JORC Code (2004). However, WAI has reviewed the company's Life of Mine (LOM) model which assumes a conservative continuing replenishment of reserves of the mine during the next few years. The Company's LOM model excludes any additional mineralised potential that could be identified to further supplement and increase the resource base in the future. This LOM model allows extension of the period of operation until at least 2014.

The LOM extension to the Irokinda 'reserves' proposed in the company model is given in Table 2.12 below.

Table 2.12: Irokinda - Buryatzoloto Projected LOM 'Reserve' Base

		2011	2012	2013	2014	2015
Gold 'Reserves' (beginning of the year)	**kg**	6,727	5,497	5,505	4,514	2,522
Grade (before dilution)	**g/t**	18	14	11	9	7
Ore	**000t**	382	383	502	503	375
Additional 'Reserves'	**kg**	2,000	3,000	2,000	1,000	0
Grade in Additional 'Reserves'	**g/t**	8	8	8	8	0
Ore	**000t**	244	375	256	128	0

WAI has reviewed the Company's LOM production plan and, although not currently based on Ore Reserves, estimated in accordance with the guidelines of the JORC Code (2004). WAI considers it to be reasonable.

Clearly, the mine will generate significant revenue during 2010/11 and there is every opportunity that new extensions and discoveries to mineralisation will be made, although this cannot currently be quantified.

> ***WAI Comment:*** *WAI believes that with the on-going exploration drilling programme and the historical annual replenishment of 'reserves' the potential to delineate further resources, which will ultimately transfer through to reserves and allow the mine to continue forward, is high.*

2.10 Company Financial Model

Based on the LOM plan referenced above, the Company has prepared a financial model for Irokinda and Zun Holba combined using financial assumptions based on historical 2009 and 1H2010 results. The KPI as provided by the client are given Table 2.13 below.

Table 2.13: Buryatzoloto (Irokinda and Zun-Holba Combined)
Key Performance Indicators for 2008 to 9m 2010

	Unit	2008	2009	9m 2010
Rock mined	kt	596	626	467
Ore mined	kt	596	626	467
Ore milled	kt	563	632	487
Average Au grade	g/t	8.3	7.9	6.9
Ore stripped	kt	n/a	n/a	n/a
Gold recovered (2008 - dore, 2009 and 1H 2010 - refined gold and silver)**	kg	4,535	4,808	3,118
Gold recovered (2008 - dore, 2009 and 1H 2010 - refined gold and silver)**	koz	145.8	154.6	100.3
Recovery rate	%	95.4	92.7	91.7
Full cash cost	US$/oz		466	651
Normalised TCC	US$/oz		412	512
Ore mining costs	US$/t		33.1	34.3
Waste mining costs	US$/t			
Ore processing costs	US$/t		13.4	15.7
General and administration costs	US$M		4.8	4.4
CAPEX*	US$M		13.4	19.9
Depreciation	US$M		13.4	10.6

Note:
*** including Exploration and Evaluation**
**** excluding 3.96 kGEO and 0.94 kGEO of silver production in 2009 and 1H 2010; 1,359 GEO of silver production for 9 and 10 months 2010**

Table 2.14: Key Assumptions - Financial Model

Year	Total	2011	2012	2013	2014	2015
Gold price, US$/Oz	900	900	900	900	900	900
Ore production (diluted), kt	3,061	699	699	699	648	316
Au grade, g/t	32	6.4	6.4	6.4	6.4	6.7
Gold mined, kg	19,682	4,468	4,468	4,468	4,138	2,140
Recovery, %	5	93%	93%	93%	93%	92%
Gold recovered, kg	18,194	4,163	4,163	4,163	3,851	1,854
Gold recovered, Oz	584,966	133,854	133,854	133,854	123,798	59,606
Revenue, m US$	524	120	120	120	111	53
Mining cost, US$/t	180	36	36	36	36	36
Mining costs, m US$	109	25	25	25	23	11
Processing cost, US$/t	75	15	15	15	15	15
Processing cost, m US$	45	10	10	10	10	5
Operating costs, m US$	157	36	36	36	33	16
Operating costs, US$/Oz	1,340	266	266	266	267	275
G&A, Sales, m US$	175	39	39	39	39	19
Depreciation, m US$	68	15	15	15	15	8
CAPEX, m US$	40	15	12	9	4	0

WAI has reviewed the company's LOM plan referenced above and considers it, though not currently based on Ore Reserves estimated in accordance with the guidelines of the JORC Code (2004), to be reasonable and achievable. WAI also reviewed cost assumptions in the company financial model for Buyatzoloto and considers these assumptions reasonable and indicative of future cash costs.

The Irokinda and Zun-Holba mines represent mature gold mining operations with no significant capital requirements and a lean operating cost structure, enabling a stable and positive cash flow generation over the next few years. Although the geological structures of the deposit does not allow for a traditional reserve and resource base to be identified without significant capital requirements, the past track record of incremental annual reserve replacement provides reasonable ground to expect replacements in the coming years.

Moreover a number of potential exploration targets identified at Buryatzoloto properties may lead to significant discoveries which will enable increases both in life of mine and future production.

3 ZUN HOLBA

3.1 Introduction

Wardell Armstrong International (WAI) was commissioned by Buryatzoloto (Client) to prepare a Competent Persons Report (CPR) on the Zun-Holba underground gold mine, located in the Republic of Buryatia, Russian Federation.

A team of WAI consultants visited the Zun-Holba mine during the period 28 - 31 of August 2009, as well as the Buryatzoloto office located in Ulan-Ude.

3.2 Location, Access & Infrastructure

3.2.1 Background

Gold in the Zun-Holba area was discovered in 1955 and evaluation between 1956 and 1959 was followed by full scale exploration programmes between 1959 and 1993. Gradual development of the deposit started in 1986 and full mining and processing operations commenced in 1991.

The Zun-Holba gold deposit is situated in the western portion of the Republic of Buryatia in the Okinskiy district, which borders the Irkutsk Region to the north and west, and Mongolia to the south. The location of the mine is shown in Figure 3.1.

3.2.2 Access

The mine is situated 736km west of the Buryatian republic capital city of Ulan-Ude, 150km from the village of Orlik, which is the district centre and 100km from the village of Mondy; the latter is 203km via a paved road to the village of Kultuk, at the southern end of Lake Baikal, the nearest station on the Trans-Siberian railway line with a transfer yard for Irokinda flotation concentrate and other supplies.

Figure 3.1: Location of Zun-Holba in Southern Russia

Total driving time from Ulan-Ude to the mine is approximately 10 hours, mostly along good paved roads, through the southern end of Lake Baikal and on as far as the Mongolian border, and then followed by good all-weather mine road.

Samarta is the main mine settlement connected to the mine site by a 12km gravel dirt road which reaches elevations of over 2,000m.

> ***WAI Comment:*** *although somewhat remote, the mine is served by excellent road communications which at the time of the visit, were in good condition.*

3.2.3 Topography and Climate

The Zun-Holba deposit is located in a mountainous area, highest elevation >3,000m at Ulan-Sardak mountain, whilst the deeper valleys and plains lie below 1,300m; the extreme topography led to the decision to site the plant and tailings facility away from the mine itself.

The area is well drained with the largest rivers, the Urik and Kitoy, having continuous year round water flow.

The climate is 'high altitude' continental with large daily variations in the temperature, annual precipitation of 500mm and an annual average temperature of -7.4°C with an average temperature in July of +15°C and in January of -22°C; snow cover up to 0.70m lasts from October to May.

Permafrost is pervasive in the area with depths up to 250 to 300m, and seasonal melting of the permafrost is accompanied by wide development of solifluction in the slopes and the formation of swamps in the upper parts of river valleys.

The area is situated in a seismically active region with a possible maximal intensity of earthquakes of 9 points according to the GEOFIAN scale. The Alpine-like part of the ZunHolba property is known for its avalanche hazards.

3.2.4 Infrastructure

The source of potable water supply and industrial fire prevention needs is underground derived from capped wells located on the right bank of the Samarta river.

Power is provided by an 86km long, double-line 110kV power line from Mondy to Samarta Two transformer substations distribute the power around the site and an emergency electrical supply is provided by eleven power generators with a total capacity of 8,360kWs.

3.2.5 Mineral Rights and Permitting

In Russia the subsoil rights are always held by the State, although since 1993, the right to explore and develop mineral deposits can be obtained *via* a state auction of the licences.

During the breakup of the USSR, Buryatzoloto received the exploration-mining licences for both the Irokinda and Zun-Holba deposits.

Between 1994 and 1995 Buryatzoloto became an open joint stock company and the licences were reregistered to the company. At that time 30% of the company shares were acquired by High River, a Canadian gold mining company and by 2005, High River had increased its holdings in Buryatzoloto to 85% of the stock.

The Zun-Holba mine inventory consists of the following deposits: Zun-Holba deposit (licence UDE 00213 BE), Pionerskoye deposit (licence UDE 00231 BR) and Smezhnoye and Pravoberezhnoye deposits located within licence UDE 00419 BR. A summary of the licence information is tabulated in Table 3.1.

Table 3.1: Zun-Holba Licence Details

Licence Number	Type of Licence	Date Licence Granted	Date Licence Expires	Licence Area	Annual Fees[1] (Rubles)	Annual Fee (US$/km^2)
UDE 00213	BE Exploitation	22 05 1998	11 01 2019	242.2 ha	Au – 6% Ag – 6.5% of revenue from the recovered metal.	
UDE 00419	BR Exploration/Exploitation	30 07 1999	29 06 2024	67km^2	203 Roubles per 1km^2	US$7.98
UDE 00231	BR Exploration/Exploitation	10 07 1999	9 07 2013	3.1km^2	203 Roubles per 1km^2	US$7.98

Note 1: The exchange rate used was RUR 25.45 = 1 United States Dollar. Table provided by OJSC Buryatzoloto.

WAI Comment: *an inspection of the licence documentation has shown that all are in good order and suitable for the future needs of the Company.*

3.3 Geology and Mineralisation

3.3.1 Regional Geology

3.3.1.1 Introduction

The Zun-Holba deposit is located in a Caledonide system in the north eastern peripheral part of the Garganskaya dome which comprises the core of the approximately east-west striking Gargan-Butugolskiy anticlinorium, and is included in the Holbinskoye mineralised area (Garganskiy gold district).

The Holbinskoye mineralised area includes the Zun-Holba and Baroon-Holba gold deposits along with a number of occurrences viz. Pionerskoye, Smezhnoye and Pravoberezhnoye; the geological setting of the area is summarised below.

3.3.1.2 Garganskiy Metamorphosed Complex

The Garganskaya dome basement complex occurs in the southern part of the Holbinskoye mineralised area and is represented by three gneiss-granite domes consisting of mainly Archean-Lower Proterozoic (870-460Ma Caledonides) gneisso-granites and gneisso-granodiorites.

The domes are 5 - 20km in diameter and are often ellipse-like in form with their contacts dipping towards the host rocks at an angle of 50° to 80°, rarely vertical and extremely rarely in contact with another dome. The contact surface between the gneisso-granite domes and the host rocks of the Riphean schist-carbonate complex is semi-conformable.

3.3.1.3 Vend-Silurian Schist-Carbonate Complex

The Vend-Silurian schist-carbonate complex covers the Garganskaya dome lying with both angular and stratigraphic discordance on the basement complex, comprising shallow dipping Vend-Cambrian (Irkutnay) terrigenous-siliceous-carbonate rocks and the Ordovican-Silurian (Ilchirskaya) volcanogenic-terrigenous rocks. The latter fill the Samarta-Holbinskaya intradome syncline zone, which separates the three domes, and underlie the ophiolite tectonic cover.

Most of the identified deposits and occurrences are located in the Samarta-Holbinskaya intradome zone, the Zun-Holba deposit located in the axis zone of the structure, with the Baroon-Holba and Pionerskoye deposits at the northwestern and southeastern edges respectively.

3.3.1.4 Ophiolite Association of Rocks (Tectonic Cover)

To the north and south of the Holbinskoye mineralised area ophiolites have been traced around the Garganskaya dome and the southern boundary of the pillow lava and siliceous schist (hyperbasites) form a natural northern boundary of the mineralised area.

3.3.1.5 Baroon-Holbiskiy Volcano-Plutonic Complex

Intrusive and effusive gabbros (basalt) within the mineralised area belong to the Baroon-Holbiskiy volcano-plutonic complex which includes numerous dykes of phyoritic, chloritic diabase and is partly destroyed and metamorphosed by granitoids of the Sumsunurskiy massif.

Amphibolised gabbro, gabbrodiabase and diabase porphyrite, as well as small size bodies and lenses of pyroxenite and serpentinite also occur in the complex. The relative age of upper Riphean-Vend rocks of the complex is indicated by their relationship with the Ordovician-Silurian Ilchirskaya suite rocks.

3.3.1.6 Sumsunurskiy Intrusive Complex

The Sumsunurskiy intrusive complex comprises two phases of intrusions:

- gabbro, gabbro-diorite, pyroxene diorite, pyroxenite and hornblendite occurring as small sized bodies located at the southwestern edge of the massif; and
- plagiogranite, tonalite, biotite and biotite-amphibole granodiorite, diorite, quartz diorite and leucocratic plagiogranite.

Granitoid massifs of the Sumsunurskiy complex are situated at the northwestern and northern parts of the Garganskaya dome and along the contact of different age rocks of the region such as the schist-carbonate cover of the dome and the ophiolite association.

The age of granitoids of the Sumsunurskiy complex is 400 to 420Ma and they are considered to be a source of the heat which led to the mobilisation and reprecipitation of gold.

3.3.1.7 Holbinskaya Fault Zone

The Holbinskaya fault zone represents a set of lensed and branched subparallel northwest trending faults which intersect the Samarta-Holbinskaya syncline intradome zone along the axial plane. Faults in the schists and carbonate rocks of the dome cover comprise steeply dipping shear zones of 10 to 30m thick.

The gneissogranite basement rocks of the intradome structure are represented by shear, mylonitisation and fracture zones of up to 400m in thickness, accompanied by berezitisation, silicification and sulphidisation of the substratum. Some of the faults of the Holbiskaya fault zone are mineralised, as is the case with the Zun-Holba, Baroon-Holba and Pionerskoye deposits.

Figure 3.2 shows the regional geology and locations of the main Holbinskoye gold deposits.

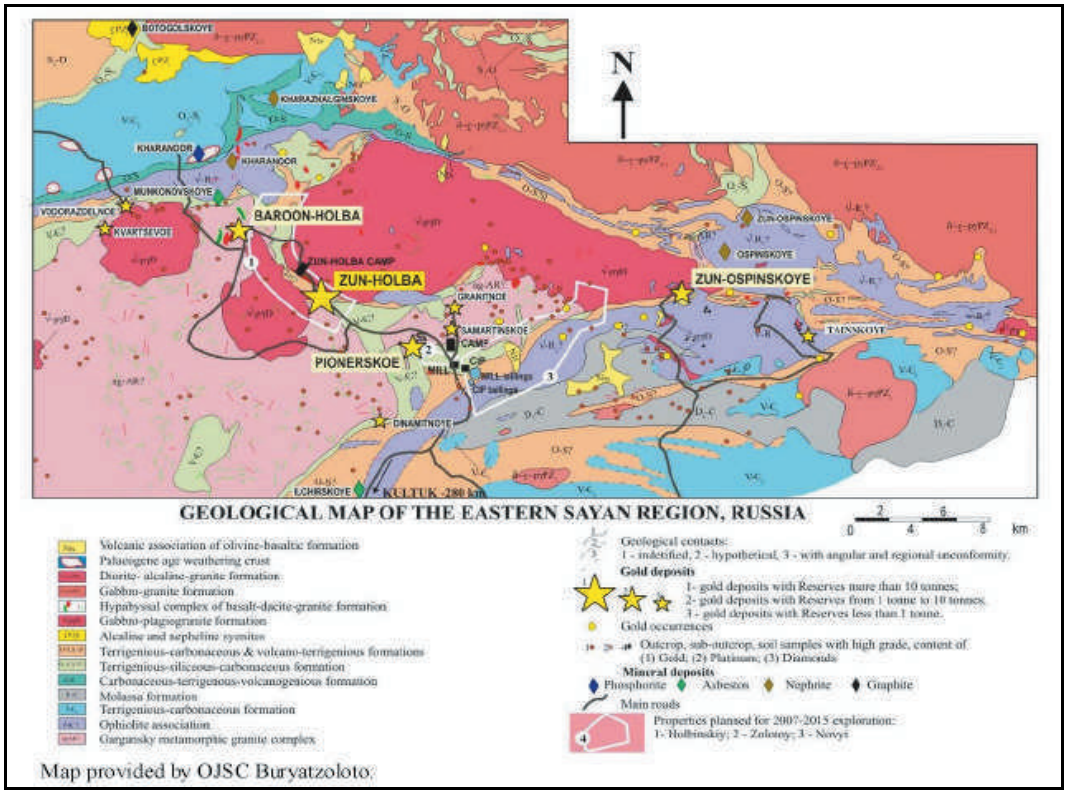


Figure 3.2: Geological Map of the Holbinskoye Area

3.3.2 Mine Geology & Mineralisation

3.3.2.1 Geology

The structure of the Zun-Holba mine property is characterised by a narrow anticline of the metamorphosed basement rocks whilst the flanks of the anticline are composed of cover rocks comprising silicified limestones and dolomite of the Irkutnaya suite, and volcanogenic-terrigenous rocks of the Ilchirskaya package.

The axial plane of the anticline, inclined to the northeast at an angle of 80 to 85°, has a northwest strike and stretches over 7km in length within the deposit limits. The northwestern closure of the anticline is obscured by granitoids whilst the southeastern periclinal closure is defined in the area of the #1 to 4 adits. The structure crest line plunges towards the southeast at an angle of 40 to 45° under a granite massif.

The mineralised zones are stepped and *en echelon* resulting from faults forming simultaneously with folding.

3.3.2.2 Mineralisation

General

The Holbiskaya fault zone in the Zun-Holba deposit limits led to formation of three northwest trending sub-parallel shear zones *viz.* Perspectivnaya, Yuzhnaya and Smezhnaya, the dominant structural features which host the gold mineralisation at the mine.

24 sub-vertical mineralised zones discovered within the Perspectivnaya shear zone and include almost all of the mineral “reserves” of the Zun-Holba deposit. The mineralised zones, located on the north eastern limb of the anticline, are hosted by the hydrothermally altered rocks of the Ilchirskaya package at the contact with the Irkutnaya suite limestone.

At Zun-Holba, there are 23 mineralised zones and the most significant in terms of contained ‘’reserves’’ are: Sulphidnoye-1 (39%), Vavilovskoye-1 (23.3%), Severnoye-1 (10%), and Vavilovskoye-3 (5%). The remaining 22.7% of the mineral “reserves” for the Zun-Holba deposit are distributed amongst the other 19 mineralised zones.

There exists a vertical zonation from surface to depth, represented by different morphological types from typical veins through vein-like bodies to mineralised zones.

Only two types of mineralised zone are considered to have economic mineralisation in the steeply dipping shear zone, and comprise mineralised bands (79% of the “reserves”) and vein-like zones (21% of the “reserves”). There is also an increase in thickness and strike length of the mineralised zones at depth from 1-2m in thickness and 20-50m in strike length to 3-5m in thickness and up to 1,000m in strike length.

Two types of mineralisation are evident:

- Quartz-sulphide rocks, and
- Gold-bearing altered host rocks such as silicate, carbonate and graphite-containing schists.

The sulphides in the mineralised rock amount to 8-9% with dominant pyrite at 90 to 95%. The secondary sulphide minerals amount to approximately 5 to 7% and are represented by galena, sphalerite, chalcopyrite as well as accessory minerals including arsenopyrite, fahlore and bournonite.

The more important mineralised bodies are described below.

Sulphidnoye-1

The thickness of the mineralised zone swells up to 4 to 7m, exhibits a massive texture and is located in limestones. It has a north western strike (290 to 300°) and a steep dip (85 to 90°) towards the southwest which becomes a little shallower at the lower levels. Within the zone there are three natural types of mineralised rocks:

1. Massive sulphide mineralisation.
2. Carbon-siliceous and carbon-siliceous-carbonate schists with sulphide mineralisation.
3. Sulphide-quartz rocks (in small amount).

The carbon-siliceous schist and other schists contain sulphide minerals, which are mainly represented by pyrite and range from 0.5% to 3.0% by volume and seldom up to 10%. The other sulphide minerals are noted as rare disseminations in pyrite or carbon-siliceous beds. The average thickness of the zone is 1.87m with an average gold grade of 12.7g/t.

The zone is being mined between the 1,640m and 1,590m levels. A cross-section showing the Sulphidnoye-1 mineral zone is presented in Figure 3.3.

Sulphidnoye-2

This zone is situated at the southeastern flank of the deposit and comprises a vein accompanied by gold-bearing beresites, hosted by a shear zone with granitisation processes and beresitised tuffaceous sandstones, sandstones and green chlorite schists.

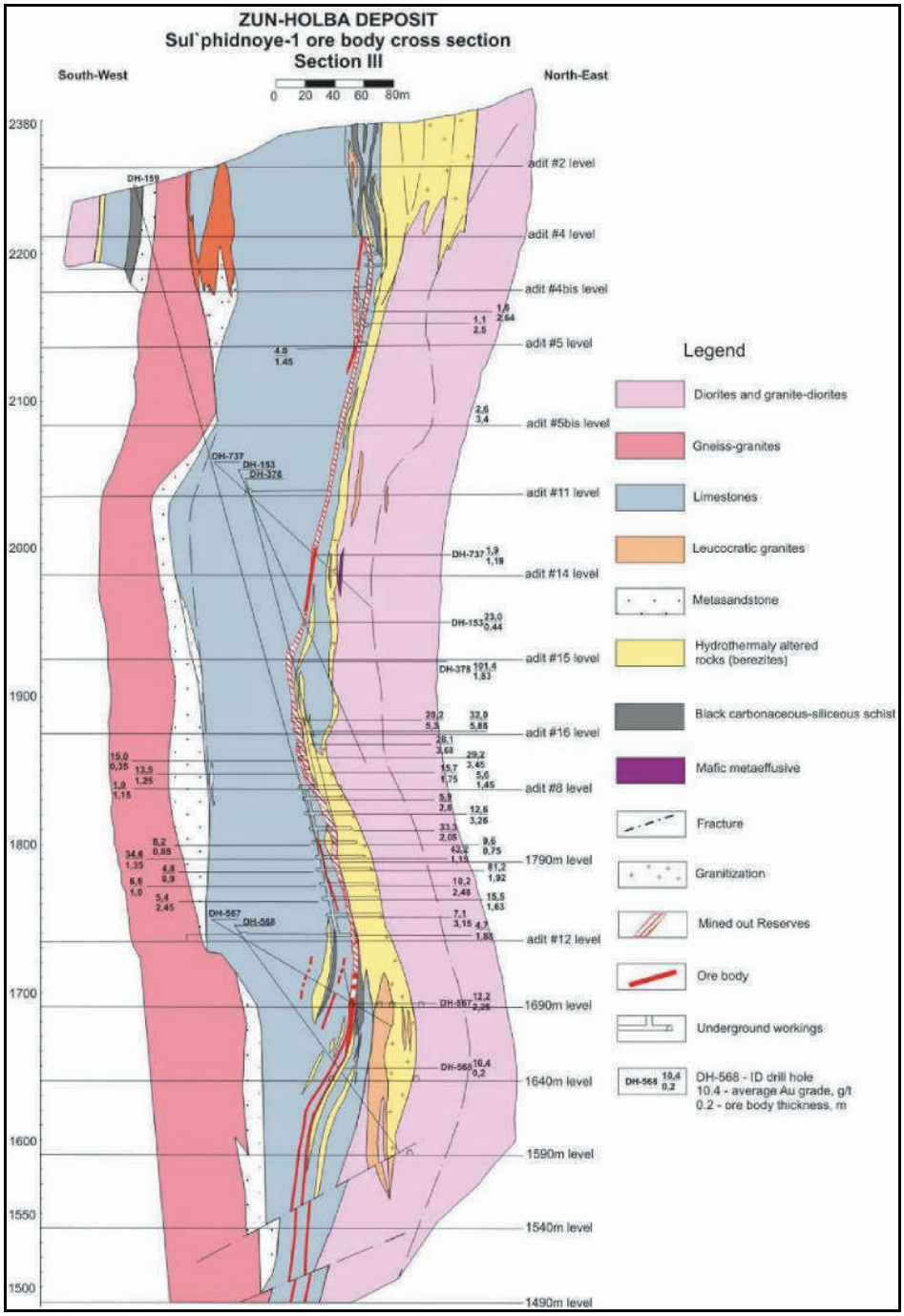


Figure 3.3: Sulphidnoya-1 Cross Section

The zone has a northwestern (310 to 330°) strike and the dip varies from vertical to 75° towards the southwest. The zone's strike length is 105m at the #8 adit level and 50m at the #12 adit level. The average thickness is 1.45m. The mineralised body consists of sulphide-quartz mineralisation (45.8%), where sulphide minerals range from 3 to 5% up to 20%, and beresitised tuffaceous sandstones, sandstones and green chlorite-quartz schists. The average gold grade is 15g/t.

The body is being mined between the 1,640m and 1,590m levels.

Vavilovskoye-1

The Vavilovskoye-1 mineralised zone is hosted by beresitised volcanogenic and terrigenous rocks of the Ilchirskaya suite and beresitised granitoids of the Holbinskiy intrusive complex. The zone is composed of bed- and dyke-like sulphide bodies in the central part, replaced along strike and down dip by sulphide-quartz veins and zones of host altered gold-bearing veins and rocks with sulphide mineralisation. The average thickness is 1.57m. The zone strikes to northwest (300° to 320°) and has a subvertical dip.

The morphological types differ in composition. The bed-like zones contain sulphide-polymetallic mineralisation (sulphide volume varies from 20% up to 40 to 60% with a maximum of 90%). The distribution of the sulphides is uniform and they are composed of pyrite, chalcopyrite, galena, quartz, calcite with molybdenite, fuchsite and magnetite.

The veins and sulphide zones have sulphide minerals ranging in volume from 0.5 to 20%. Pyrite here is the dominant mineral, with galena, sphalerite and molybdenite rarely noted. The average gold grade of the mineralised zone is 15.6g/t. The body is being mined between 1,640m and 1,490m levels. A cross-section showing the Vavilovskoye-1 mineralised zone is presented in Figure 3.4.

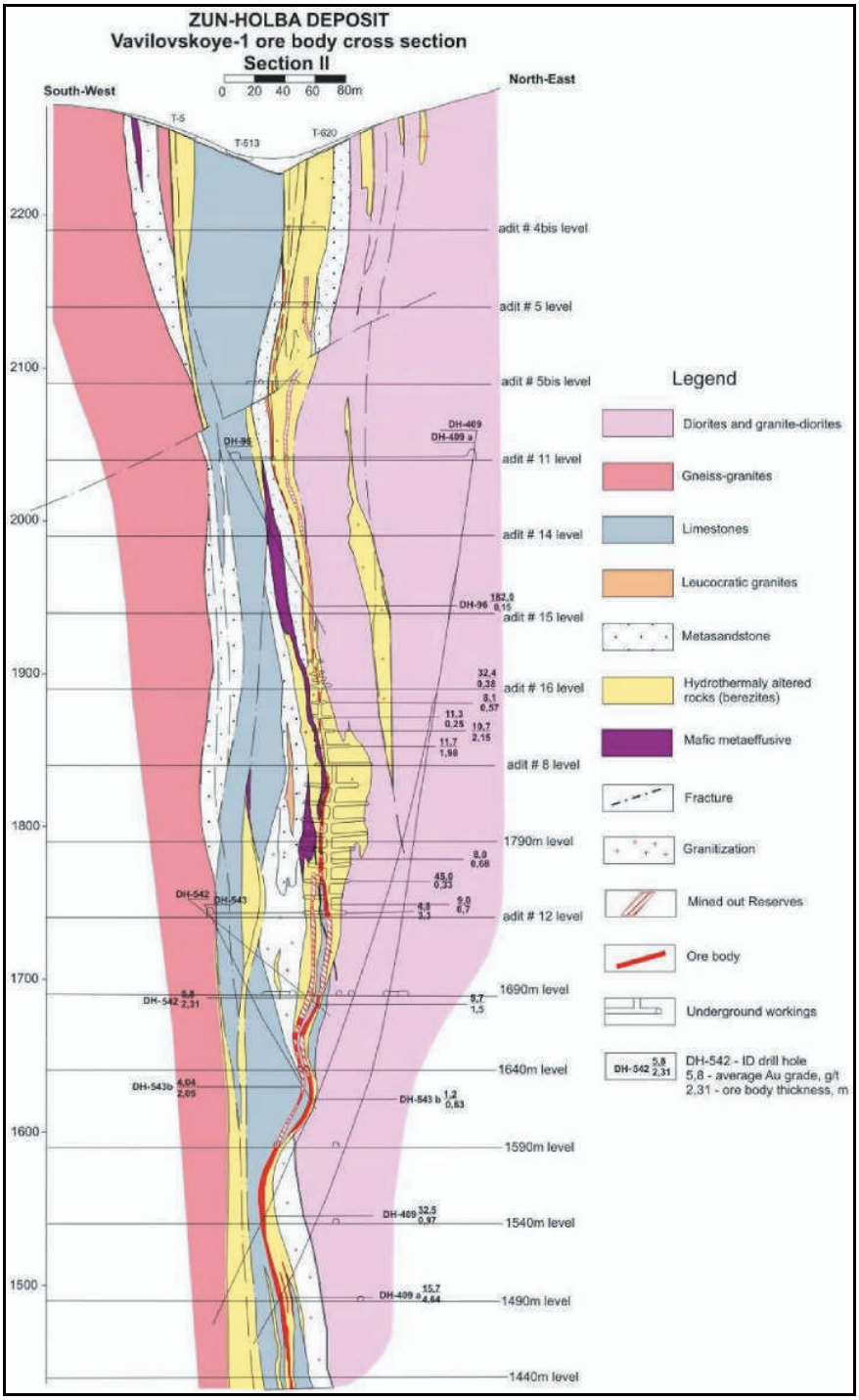


Figure 3.4: Vavilovskoye-1 Cross Section

Vavilovskoye-2

The Vavilovskoye-2 zone is situated 3 to 10m from the footwall of the Vavilovskoye-1 zone. The zone is similar to the Vavilovskoye-1 zone in mineral composition and conditional bedding but is smaller in size. The strike length is 100m at the #8 adit level, and the down-dip length varies from 12 to 140m. The average thickness is approximately 1.33m and the average gold grade is 15.6g/t.

Most of the zone has been mined and extracted.

Vavilovskoye-3

The Vavilovskoye-3 zone is localised parallel to Vavilovskoye-1 and has been determined only at the #12 adit level and 22m above the level.

The mineralised zone is represented by a steeply dipping lens with a strike length of 256m and a down-dip extent of 190m with an average thickness of 1.84m. The zone does not show any variation in thickness either up or down dip, the strike direction of the zone is between 310 and 330° and the dip angle is 80° towards the southwest. There is a quartz-sulphide vein core of 0.15 to 1.0m thickness hosted by beresites and beresitised rocks.

The amount of quartz-sulphide mineralisation within the body is 26.6% and the average gold grade is 17g/t.

The body is being mined between the 1,640m and 1,490m levels.

Severnoye-1 and 2

The zone is located on the north western flank of the property. In the upper levels the zone is represented by vein-like bodies which below the 1,740m level, change into a mineralised zone along with the appearance of black carbon-siliceous-carbonate schists with sulphide mineralisation, limestones and intrusive rocks.

The zone has a strike direction of between 310° and 330°, dips steeply to the southwest between 87° and 90°, becoming shallower (up to 68°) at depth and an average thickness of 2.15m. The average gold grade is 17.6g/t. The zone is being mined between 1,640m and 1,490m levels.

A cross-section showing the Severnoye-1 mineralised zone is presented in Figure 3.5.

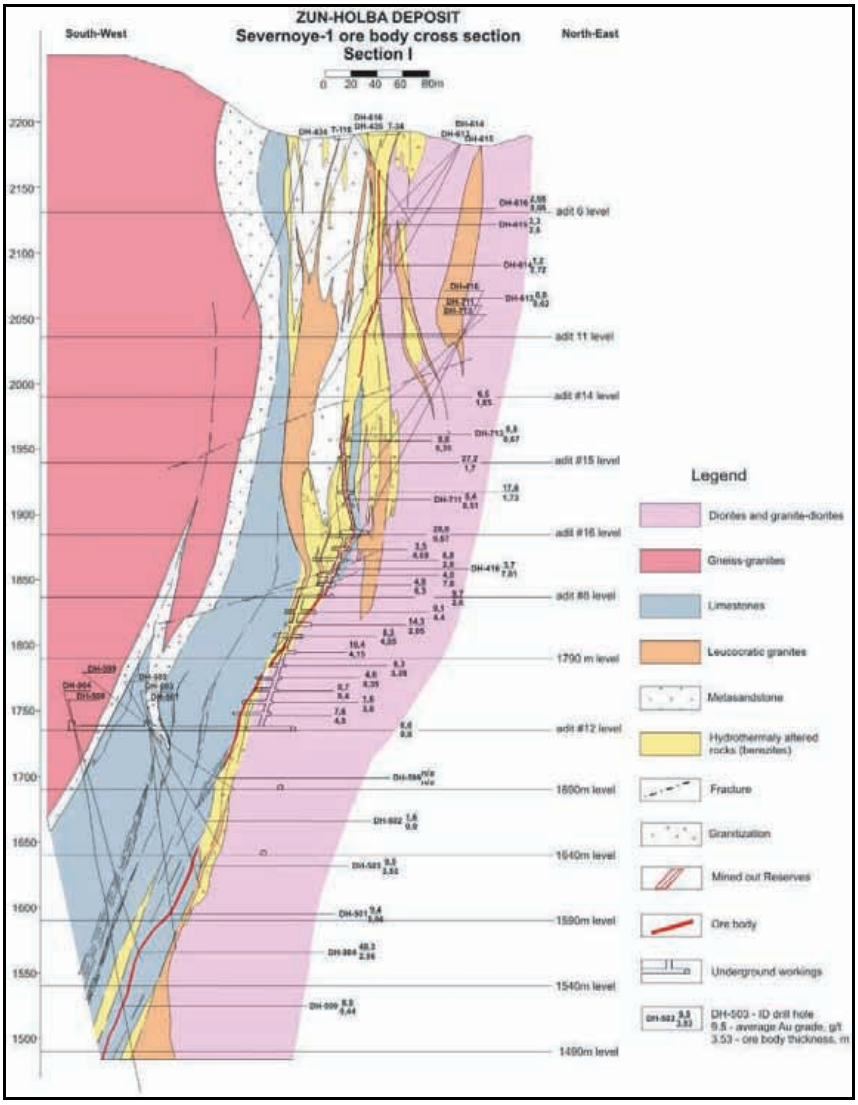


Figure 3.5: Severnoye-1 Cross Section

Severnoye-3

The Severnoye-3 mineralised zone, located 200 to 400m northwest of the Severnoye-1, comprises a steeply dipping lens with a strike length of 256m and a down-dip extent of 190m with an average thickness of 1.84m. The zone does not increase in thickness either up or down dip, the strike direction of the zone is between 310 and 330° and the dip angle is 80° towards the southwest.

The zone is situated at the contact between limestones and beresitised tuffaceous sandstones and has a complex internal structure. The amount of sulphide minerals does not exceed 20% in the quartz-sulphide mineralisation and is 3 to 5% in altered rocks. The average gold grade is 18.6g/t and the average thickness is 1.25m.

The amount of quartz-sulphide mineralisation in the body is 26.6% and the average gold grade is 17g/t. The body is being mined between the 1,640 m and 1,490 m levels.

Babkina

The Babkina mineralised zone is situated at the south-eastern end of the deposit. The average gold grade is 10.8g/t and the average thickness is 1.3m; however, all economic material has been extracted from this zone and only uneconomic mineralisation remains.

Parallelnoye

The Parallelnoye mineralised zone is situated on the north western flank of Zun-Holba with an average thickness is 1.36m and average gold grade of 11.5g/t. All economic material was extracted from this zone.

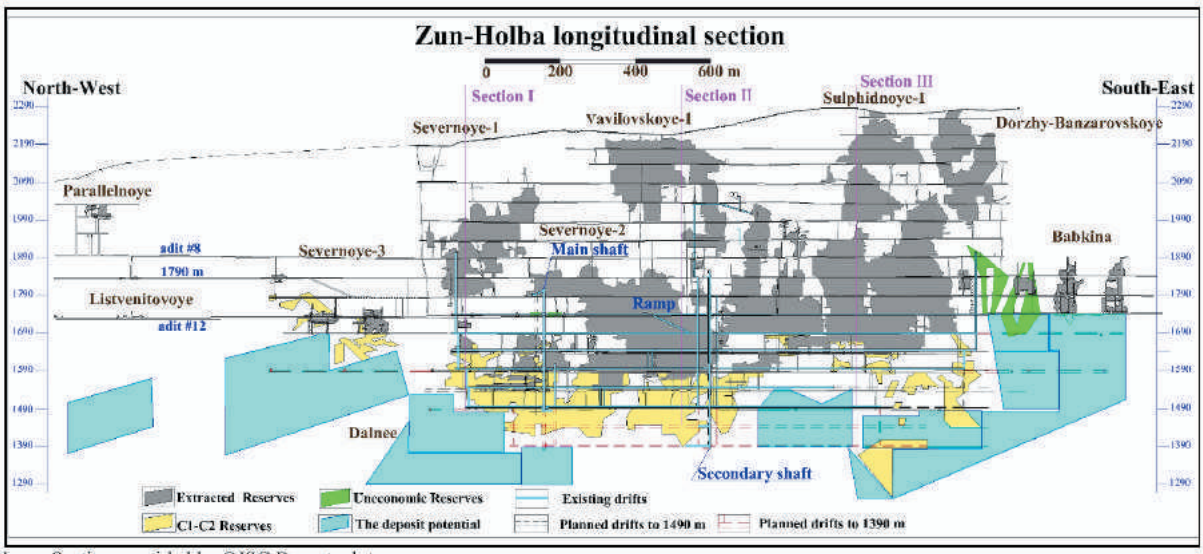
Other Zones

In addition to the main mineralised zones mentioned above, a number of other zones have been located on the Zun-Holba mine property (Table 3.2). The mineralised bodies Vavilovskoye-5, 6, Sulphidnoye-4 and Dorozhnoye-3, 4 have gold grades ranging from 3 to 5g/t and average thickness of up to 1m.

Table 3.2: Other Mineralised Zones at the Zun-Holba Mine Property

Mineral Zone Name	Average Thickness(m)	Average Gold Grade (g/t)	Comments
Dorzhy-Banzarovskoye	2.51	10.4	Economic mineralisation has been completely extracted.
Parallelnoye	1.36	11.5	Economic mineralisation has been completely extracted.
Dorozhnoye-2	0.85	16.1	Not mined out
Bulba	1.31	8.4	Economic mineralisation has been completely extracted.
Listvenitovove	1.87	23.9	Some economic mineralisation has been extracted.
Sulphidnoye	0.8	14.9	Economic mineralisation has been completely extracted.
Severnoye	1.5	6.7	Economic mineralisation has been completely extracted.
Dalnee	2.46	16.2	Exploration planned between the 1490m and 1390m levels
Table provided by OJSC Buryatzoloto.			

A long-section showing the locations of the various mineralised zones at the Zun-Holba mine is presented in Figure 3.6.



Long Section provided by OJSC Buryatzoloto.

Figure 3.6: Long Section through the Zun-Holba Mine

WAI Comment: *the Zun-Holba mine is a mature operation with a long history of underground production. The mineralised structures are persistent both in strike and depth, and overall mineralisation is continuous for over 1km vertically. Although only limited data exists below the 1,290 Level, WAI has no reason to believe the mineralisation is not continuous below this level, as well as to a lesser extent, along the flanks.*

3.4 Exploration

3.4.1 Historical Work

3.4.1.1 Introduction

From 1982 to the present there have been two organisations which have conducted exploration programmes at the Zun-Holba mine site.

3.4.1.2 State Geological Association Buryatgeologia Sampling Method

From 1982 to 1993, detailed exploration consisting of channel, core, bulk and metallurgical sampling was conducted at the deposit. The scope of the sampling is shown in Table 3.3 below.

Table 3.3: Exploration Works Undertaken Between 1982-93

Type of Work	Unit	1982 to 1993 Exploration
Excavation; mining	m	42,285
Trenches	m^3	192,848
Host rock drifts	m	6,413
Exploration drifts	m	13,247
Raises	m	3,751
Cross-cuts	m	18,573
Drilling	m	174,539
Sampling		
Channel	Number of samples	93,352
Core	Number of samples	60,940
Metallurgical	Number of samples	20
Bulk sampling	Number of samples	53
Soil sampling	Number of samples	26,499

The accuracy of the selected cross-section of channel samples was controlled by a systematic comparison of the calculated and actual weights of the samples. The quality of sampling was controlled by taking channel samples with a larger cross-section (5 x 40cm). The reliability of the channel sampling was checked by 53 bulk samples varying in weight from 1.15t to 1,009t, including 1,371 channel samples taken from 203.95m of mineralised intersections. A direct comparison of data from the bulk and channel sampling confirmed the reliability of the channel sampling.

A total of 1,006 drill holes, including 326 surface holes and 680 underground holes, was drilled, with an average core recovery of 84%.

Quality control of the sampling was conducted systematically by a comparison of calculated and actual weights of the core samples. The reliability of core sampling was evaluated by comparing the core sampling with the bulk and channel samplings. The comparison determined that there was no significant understatement of the gold and silver grades in the core sampling which was reliable enough to outline mineralised zones for the “reserve” estimates.

3.4.1.3 Buryatzoloto Sampling Method

The sampling method for the mineralisation exposed by the underground drifts was developed during the detailed exploration stage and continues to be used for exploration at the Zun-Holba deposit.

Micon (2008) concluded that the chip sampling procedures are representative of the mineralisation identified at the mine, meeting the current industry best practices guidelines, and could be used for the resource estimations conducted at the Zun-Holba mine. Entire borehole core is sampled and the core recovery for mineralisation is 98 to 99%. All drill holes are sampled.

3.4.1.4 State Sample Preparation & Analysis

The crushing and grinding processes were performed using both hand and mechanical methods in the crushing facility of the Zun-Holba exploration team located at the Zun-Holba camp.

The core samples were processed following a single stage cycle of crushing and grinding where the material was ground down to 0.074mm without splitting.

Analyses of samples for gold and silver were done using the fire assay method and a total of 3,847 internal check analyses (3.73%) of all samples from the project, and a total of 3,795 external check and 324 arbitration check analyses, were performed during the exploration programmes. Processing of the check analyses was performed in accordance with the 1982 Research Council on Analytical Methods (RSAM) guidelines.

According to the exploration reports, the internal check analyses showed that a bias existed primarily in the 0 to 1g/t gold grade range and rarely in the 1 to 4g/t and 4 to 16g/t grade ranges. The bias did not influence the "reserve" estimation because it affected only 5 to 6% of the total number of assays conducted.

The external check assays identified significant systematic biases during some periods of sampling between 1986 and 1989 and, for the sampling conducted during these periods, a number of arbitration check assays were submitted for secondary assaying. The arbitration check assays which were conducted identified that the fire assays of the main laboratory were acceptable and could be included in the "reserve" estimation.

3.4.1.5 Buryatzoloto Sample Preparation & Analysis

Preparation of the core and channel samples included crushing and grinding. The crushing and grinding processes were performed using both hand and mechanical methods in the crushing facility of the Zun-Holba mine located at Samarta.

All mineralised channel, core and chip samples are fire assayed for gold and silver. Internal check assays are performed to find random errors in the analytical work at the laboratory taken from the duplicate samples. The check assays are conducted for each of the following gold grade ranges: 0 to 1g/t (0.0 to 0.5; 0.5 to 1.0), 1 to 4g/t, 4 to 16g/t, 16 to 64g/t, 64 to 128g/t and more than 128g/t.

Both the internal and external check analyses for gold have been conducted in a systematic fashion for the Zun-Holba mine. There is a satisfactory comparison between the external and internal check assays, and these do not display any systematic biases from the external check laboratories.

Micon also considered that the Quality Assurance/Quality Control (QA/QC) procedures and protocols employed at the Zun-Holba mine are rigorous enough to ensure that the sample data are appropriate for use in mineral resource estimations.

3.4.2 Current Exploration

Exploration at Zun-Holba can be divided into three separate initiatives:

- Mine exploration to upgrade and develop the existing predominantly P_1 resources in the main ore shoots, where some 8t of Au (above 1,290m level) has been estimated which will form the basis for production exploration during 2010 in an effort to prove up 2t Au;
- Mine exploration to extend the ore shoots within the main mine area; and
- Thirdly, new target exploration on the flanks and further down dip in an effort to define new resources.

In addition to this, the Smezhnoye and Yuzhnaya deposits are also targets which the exploration teams at Zun-Holba will be investigating during 2010.

3.4.3 Exploration Programme and Budget for 2010

Buryatzoloto is implementing an aggressive budgeted exploration programme for 2010, in order to update the level of reserves as soon as possible. The proposed exploration schedule is given in Table 3.4 below.

The Zun-Holba budget for 2010 totals RUR344M or US$11.795M, and by 01 November 2010 some RUR149M or US$5.103M had been spent. The Zun-Holba budget approved for 2011 totals RUR222.197M or US$7.407M (the exchange rate is RUR30/one dollar).

At Zun Holba underground drilling during 2010 (shown as green fan holes on Long Section in Figure 3.7) is underway with an allocated budget of US$3.386M, using one LM 75 rig, three Diamec-262 rigs and one LM 55.

In addition, drilling has recently started at Smejnaya with a Russian contractor, using two LF90 rigs with a budget of US$2.611M.

The first of a planned six holes is being currently drilled on the very deep extension to Zun Holba (drilled from the southwest through the third structure known as Yuzhnaya, using an Australian rig and Russian contractor.

Also, three holes are planned in the northwest of Zun-Holba to test the northwest plunging trend, three holes in the southeast to test the southeast plunging trend, and one hole in the centre of the Zun-Holba orebody. Holes will be drilled to 1,500-1,600m, with a proposed budget of US$3.018M. A total of 1,068m from 10,880m planned were drilled as at 01 November 2010.

> ***WAI Comment:*** *WAI believes that the on-going and proposed exploration programme although aggressive, is achievable, whilst the budget costs appear to adequately reflect the proposed programme.*

Table 3.4: Zun-Holba Exploration Work Programme For 2010 & 2011

			2010					2011			
			Volume of Works		Estimated increase of Resource (t Au)			Volume of works	Estimated increase of Resource (t Au)		
Mine	**Exploration Target**	**Type of Works**	**Plan**	**Actual as at 1.11.2010**	**C2**	**P1**	**P2**		**C2**	**P1**	**P2**
Zun Holba Mine	Perspectivnaya Zone	Surface drilling (m) Underground drilling (m) Roads and drilling site preparation (m^3) Auxiliary mining works (m or m^3)	4,010 27,020 9,000 740(5,898)	4,419.3 21,917 10, 000 488(3,729)	2	8	0	9,600 23, 200 3, 600 663(5,723)	3	0	0
	Smezhnaya Zone	Surface drilling (m) Roads and drilling site preparation (m^3) Auxiliary mining works (m or m^3)	10,300 13,300 200(1,640)	9,441.6 30, 098 0(0)	0	10	0	0 0 0	0		0
	Deep Horizons of Zun-Holba	Surface drilling (m) Roads and drilling site preparation (m^3)	10,800 4,940	1, 068 14, 000	0	20	35	11, 100 9,000	0	30	0
	Pionerskoye Deposit	Surface drilling (m) Roads and drilling site preparation (m^3)	0 0	6,280.9 8,200	0	0	0	0 0			

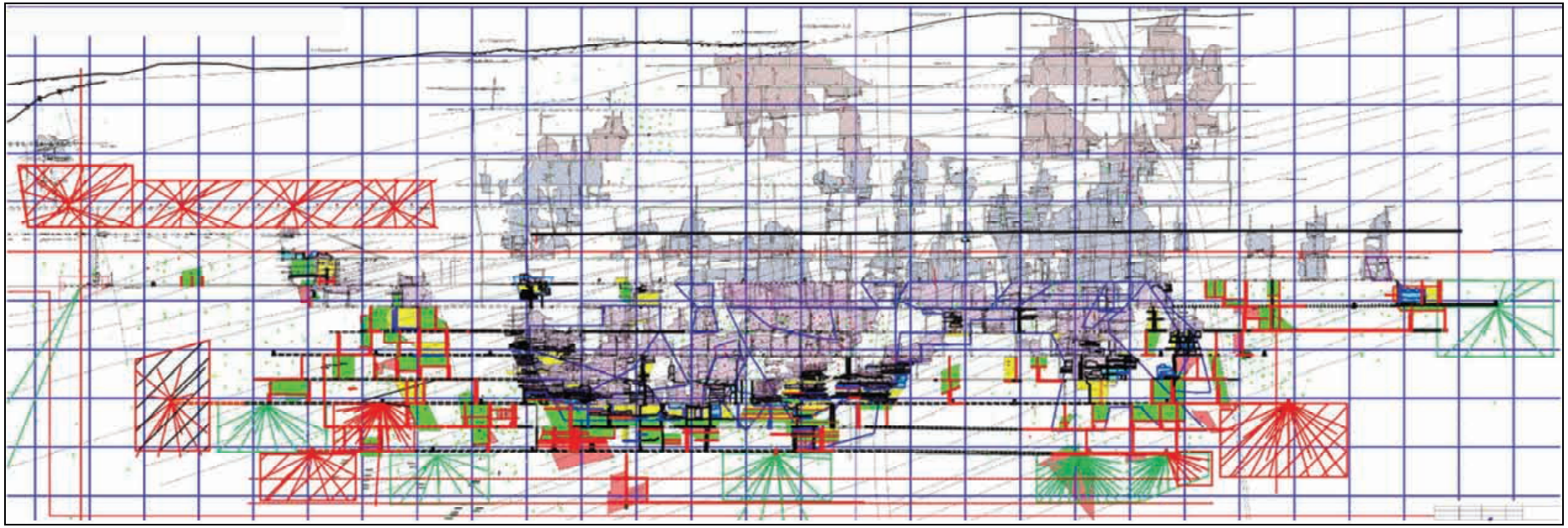


Figure 3.7: Long Section through Zun-Holba Showing 2010 Drilling (Green) and 2011 (Red)

3.4.4 New Target Exploration Drilling

In essence, new target exploration at Zun-Holba involves following extensions to the main zones to the northwest and southeast (flanks), and in depth, as well as exploring adjacent mineralised targets.

Northwest Flanks: exploration in this area will work back from the already partially explored Dalnaya Zone which hosts an estimated 12t of Au at P_1, towards the main mined areas. This will involve drilling with trenching where possible, although much of this area has previously remained untested.

The Company plans to do some 9,600m of surface drilling on an 80 x 80m grid in effort to better delineate this zone.

In addition, further to the southeast, blocks 86-C_1 (Dorojnoye-3) and 88-C_2 (Dorojnoye-2) will be properly drill tested.

Southeast Flanks: from Babkinsky zone southwards, nothing is known about the mineralisation other than the structure appears consistent and has been previously mined some 4km to the south at the Pioneerskoye deposit.

Pioneerskoye, which is now exhausted, was notable for a high grade vertical vein swarm of up to 20 individual veins, some of which had mineralisation up to 50m thick. Other smaller zones around Pioneerskoye have also been investigated with varied success.

However, the southeast flank zone between the two known areas is very deep (up to 1km beneath granites and limestone) and remains of relatively low priority.

Deep Levels: the Zun-Holba system already has a demonstrated vertical extent of more than 1km and remains open at depth. Exploration will target the zone from the 1,290m Level down to approximately 500m, initially in the 2010 programme as 3 drill lines with 7 holes each beneath Severnoye and Vavilovskoye orebodies where there is the potential for two ore shoots to merge. A 30,000m drill programme (US$5M budget) is anticipated as these zones can only really be intersected from deep surface holes.

Smejnaya: this northwest trending schistose/beresitic alteration zone, which was discovered decades ago, lies a few hundred metres to the northeast of the mining camp and has been intersected in several locations. A channel sample across the mineralisation gave 8.7m @ 1.8g/t Au, although the actual zone may well be wider than this.

For 2010, the Company plans to drill some 10,000m concentrating on the northwest end where the best exposures are seen, and at the southeast end (currently outside of the licence area) where previous trenches have intersected grades up to 54g/t Au.

Yuzhnaya: This shear zone lies approximately 1.5km to the southwest and parallel to the main mineralised zones. Limited outcrops show interesting grades up to 30g/t Au. The vein contains sulphides (predominantly pyrite) and beresite alteration. Some areas show the presence of black schists, particularly in the southeast.

> ***WAI Comment:*** *the present plans to chase mineralisation at depth and on the flanks of the veins is sound as the proven continuation of these ore zones both vertically and to a lesser extent, along strike, is well known. Similarly, the Smejnaya target also looks encouraging and warrants the planned extensive drilling programme.*

3.5 Resources and Reserves

3.5.1 Introduction

The most recent "reserve" estimate for the Zun-Holba mine was conducted 2010 and is dated as of 01 November, 2010. In addition, High River had previously requested Micon to audit their Russian "reserves" and convert the Russian classified "reserves" into a CIM compliant mineral resource and mineral reserve estimate. This exercise was done twice, in 2007 and 2008.

WAI has adopted this same approach in providing a "JORC Equivalent" resource for the mine which is reported here.

3.5.2 Zun-Holba Resource Estimate Methodology

3.5.2.1 Introduction

For the Zun-Holba deposit, the "Geological Block" method which is similar to the polygonal method has been used historically and is still being used to estimate "reserves."

"Operational reserves" are estimated from the tonnages of the geological in-situ "reserve" by applying factors for mining recovery (loss in mining, %) and dilution.

3.5.2.2 Database

The Zun-Holba mine property database for "reserve" estimation consists of underground sampling of drifts, cross-cuts and raises along the mineralised veins and zones as well as numerous underground and surface diamond drill holes. The drill hole collars were surveyed and inclinations recorded at 15m intervals for the most part.

3.5.2.3 Density Used For "Reserve" Estimate

At the Zun-Holba mine the density varies slightly between the various mineralised zones within the deposit (2.73 – 2.76t/m^3).

3.5.2.4 Top-Cutting Levels for Gold

The determination for capping a gold grade is made by reviewing if the thickness x grade (metregram) of any individual composite is greater than 10% of the metregram sum of a block. Composites which had metregrams greater than 10% of the block's metregram sum were cut.

The capping approach is based on the experience of Buryatzoloto geologists, who consider the individual influence of every high assay on the block's average grade. In the case where the value of a metregram is higher than 10% of the sum of the block's metregram due to thickness, grade capping is not performed.

3.5.2.5 Parameters Used For "Reserve" Estimation

The 2002 exploration-mining parameters currently in use are as follows:

1. Cut-off grade of 3g/t gold to outline the boundaries of the mineralised zones.
2. Minimum gold grade of 7g/t over a mineralised zone composite intercept.
3. Minimum mining thickness of 1m. When thickness is less, but the gold grade is higher, it is recommended to use the metregram.
4. Minimum gold grade in a mineable block of 15.5g/t gold.

5. Minimum gold grade of 11.9g/t gold in a block being mined with workings targeted to reach a mineable block.
6. Maximum thickness of 3m for waste rock and low grade mineralisation being included in the "reserve" estimation.
7. Uneconomic "reserves" are considered to be blocks with gold grade less than 15.5g/t or 11.9g/t but higher than 7g/t. In the Russian terminology any uneconomic "reserves" are considered to be "off balance reserves" while economic "reserves" are considered to be "balance reserves"
8. Estimate silver "reserves" within the outlines of the gold "reserves."

The price of gold used to estimate the parameters was US$550.

3.5.2.6 Buryatzoloto "Reserve" Inventory as of 01November 2010

According to the Russian classification system, the Zun-Holba mine "reserves" as of 01 November 2010 are summarised in Table 3.5. Dilution and mine recovery factors have not been applied to the "reserves" shown.

Table 3.5: Zun-Holba "Reserves" 01.11.2010 (According to 5-GR From Report)

	Ore (t)	Au g/t	Au Kg
C_1	41,496	16.8	696
C_2	589,601	13.3	7,812
C_1+C_2	631,097	13.5	8,508

The "mineable reserves" or "operational reserves" for the Zun-Holba mine are estimated by applying dilution which depends on the mineralised body thickness and mining method, and recovery factor (mining loss) which is estimated on the basis of a typical project with mineable blocks using the same mining methods.

3.5.2.7 WAI Subjective JORC Assessment

As part of the resource review, WAI was asked to provide a subjective opinion of the Zun-Holba resource statement in terms of its equivalency to JORC Code (2004) guidelines.

Without re-modelling the deposit, it is clearly not possible to provide a definitive opinion on this matter, although for the purposes of this report, WAI accepts the approximate equivalence of the B category with *Measured*, C_1 category resources with *Indicated* and C_2 with *Inferred*.

Considering the above, a resource statement complying very approximately with JORC Code (2004) guidelines as of 1st November 2010 is presented in Table 3.6 below.

Table 3.6: Zun-Holba JORC Equivalent Resources as of 01 November 2010

Resource Category	Ore (t)	Au (kg)	Au Grade (g/t)
Indicated	41,496	696	16.8
Inferred	580,653	7,812	13.5

WAI Comment: *the methodology used for the Zun-Holba "reserves" is reliable and the parameters well documented. GKZ protocols have been followed properly. However, the remaining "reserve" tonnage is small, suitable for approximately 2-3 year mine life, although the mine has good opportunities to go deeper, as well as explore the flanks further, and WAI believes that additional reserves will be defined in order to prolong the life of the mine.*

3.6 Mining

3.6.1 Reserves

WAI has been provided with a resource statement as of 01.11.2010 and 5-GR report to the State Commission of Resources (GKZ) from 01.01.2010.

Estimated losses and dilution will be 6.5% and 23.0% respectively, based on dilution (losses) estimation per each resource block, and thus, to estimate an actual reserve estimate, the resources need to have dilution and recovery applied to them.

As such, Zun-Holba does not have JORC compliant reserves as per the guidelines of that code.

3.6.2 Mine Design

The mine has three shafts, two of them are used for materials hoisting. The depth of the shafts is 300m for shaft Glavnaya and around 200m for the other two shafts. The main haulage level is adit No.12 which is slightly inclined and located in proximity to the level 1,740m.

3.6.3 Mining Method

3.6.3.1 Rock Properties and Geotechnical Conditions

The deposit is hosted by medium strength rocks with unstable sections which are normally confined to fault, slate and weak limestone areas. The presence of unstable rocks results in requirement for support of the vast majority of the underground workings.

3.6.3.2 Methods Employed

At the present time mining takes place within the central part of the deposit between horizons 1,440m–1,590m, which is below main haulage level (1,740m). At the exit portal of the adit No.12, there are stockpiles for high-grade ore (more than 10g/t), standard ore (5.5-10g/t), low-grade ore (less than 5.5g/t) and waste.

The mine employs three mining methods, a consequence of the rock properties and orebodies varying at different depths.

Cut-and-fill stoping (64% of total mined material volume) is used for relatively thick orebodies, which are unstable or moderately stable. At first the preliminary development is carried out, which includes construction of a haulage drift at the bottom of the stope level, vertical access and ventilation raises and ore pass.

Shrinkage with bolting and plastic mesh reinforcement (22%) is applied when mining thin orebodies in medium to stable rock. The method is based on temporary storing of crushed ore within the stope, providing a floor for drilling the upper material and wall support.

Shrinkage with timber settings (14%) is used for thin veins located within unstable host rocks. The method is fairly similar to shrinkage with bolting and plastic mesh reinforcement with the major difference being that the stope is filled with timber sets, allowing the crushed ore to be fed to the bottom haulage drift as well as to support stope walls.

A breakdown of ore production (for 2008 and 2009) by the extraction method is given in Table 3.7 below.

Table 3.7: Zun-Holba Mining Rates By Method			
	2008	**2009**	**1H2010**
	t*	**m^3**	**m^3**
Ore Mined Using Shrinkage With Bolting And Plastic Mesh Reinforcement	59,781	20,771	9,512
Ore Mined Using Cut And Fill Method	169,008	70,565	32,315
Ore Mined Using Timber Settings	35,767	17,414	7,975
Total Ore Mined	**264,557**	**110,759**	**49,802**
* Mining figures are given in m^3. Tonnages are obtained by applying a bulk density of 2.65t/m^3.			

3.6.3.3 Ore/Waste Transportation

The ore is transported out of the mine *via* the main haulage level (adit No.12); the average transportation distance from the mining face to the surface ore stockpiles is approximately 2.5km, which has a tendency to increase as the remote flanks of the deposit are developed.

3.6.3.4 Mine Dewatering

The deposit is formed by hard rock, and hydrological conditions are deemed to be simple. Water inflows are confined to fractured rocks in areas of intense tectonic disruption. Boreholes drilled in order to estimate hydrological conditions at several horizons which are currently being mined indicated low inflow of sub-permafrost waters, with a maximum expected in summer periods.

Total water inflow into the mine is in a region of 190m^3/h. At 1690m, the water inflow is 59m^3/h; at 1640m - 52m^3/h, 1590m – 48 m^3/h, 1540m – 16m^3/h, 1490m – 5m^3/h, 1440m - 5m^3/h, 1390m – 5m^3/h.

The water is drained from the workings to the bottom horizons, and then collected and pumped to the surface via a centralised pumping station.

3.6.4 Operational Costs (Mining only)

The nature of the deposit dictates the usage of expensive methods of mining. The methods employed are labour and material intensive which negatively affects the overall mining cost.

Nevertheless the costs of mining (Table 3.8) appear to be moderate in comparison with similar underground operations worldwide.

Table 3.8: Zun-Holba Mining Costs By Method		
	2008	2009
	Mining Cost US$/t*	Mining Cost US$/t*
Materials	5.04	4.6
Energy	1.68	1.6
Wages	6.38	4.7
Other	1.6	1.2
Total Mining Costs	**14.7**	**12.2**
*Roubles were converted into dollars at an exchange rate of 24.89 Roubles per 1 US$ for 2008 and 33.20 Roubles per 1 US$ for 2009.		

WAI Comment: *during 2009, the Company has been able to reduce overall mining costs significantly, as result of reducing the number of employees.*

3.6.5 Conclusions

Zun-Holba is a well-setup mine with all necessary infrastructure, labour and other facilities. Although a significant part of the main mining equipment is outdated, the mine is still capable of scheduled production and has reasonable potential for production increase should it be required. The remaining reserves are sufficient for at least two years of mine life, but the mine has prospective areas which can enlarge available resources considerably. Currently the exploration programme is well behind the mining operations and the programme is demanding urgent development (realisation).

3.7 Processing

3.7.1 Introduction

The processing facilities for the Zun-Holba mine are located at Samarta, some 20km to the east of the mine. Ore is transported in 20t trucks, nominally at a size of -200mm, and dumped in a small stockpile area. The plant utilises the standard processes of crushing, grinding, gravity processing, flotation and carbon in pulp leaching (CIP) of the flotation concentrate. The Zun-Holba CIP Plant also treats the flotation concentrates from the Irokinda mine.

3.7.2 Previous Testwork

An initial metallurgical test programme concluded that the different types of mineralisation all belonged to a single, “easily processed” ore type with the total recovery of gold varying from 92.9 to 97% if a gravity-flotation scheme, with cyanide leaching of the flotation concentrate was used. In 1996, 508.5t of flotation concentrate were processed at the CIP plant of the Pervomaiskiy mine located in the Chita region with a recovery of 90.9%.

The economic minerals are gold and silver with the average grades for the deposit of 15.5g/t Au and 18.5g/t Ag respectively. Gold is commonly fine-grained (less than 0.07mm), and is generally disseminated and powder-like. Carbon is present as graphite (0.2 to 1.5% by volume). There is almost no oxidation of the ore.

3.7.3 Process Description

3.7.3.1 Stockpile

Ore is dumped in an area adjacent to the main plant feeder hopper. The ore is blended at the mine and therefore no further blending is undertaken at the plant site. The trucks are weighed as they go onto the stockpile.

3.7.3.2 Crushing Plant

Ore is crushed using three stages of crushing. The ore from the main feed hopper passes to a jaw crusher and the product is conveyed to a 1200mm cone crusher. The cone crusher product is screened and the screen oversize passes to a secondary 1200mm cone crusher; the final product size is either 16 or 18mm, depending on moisture content. The crushed ore passes to one of three small (50t) ore bins each feeding a separate process line.

3.7.3.3 Grinding and Gravity Processing

The ore is ground to 95% passing 75 microns using two stages of ball milling and three lines, the basic unit operations of each being similar.

The primary mills discharge into a jig and the jig concentrates passes to a shaking table. The jig tailings pass to a spiral classifier and the sands (oversize) are returned to the mill. The spiral overflow is pumped to 250mm

cyclones, with the underflow gravitating to a secondary ball mill. The secondary ball mill also discharges into a jig and the jig tailings are pumped back to the cyclone.

The jig concentrates are cleaned using shaking tables. The concentrates from the two shaking tables pass to a cleaning table. The final table concentrate is cleaned further on a 12 hour batch basis. Here a final product is dewatered, and dried and sent to the Dore plant for smelting.

It was reported that the gravity gold recovery was typically 29% but that this figure fell to 21% when treating ore from the lower levels of the mine.

> ***WAI Comment:*** *the jigs were not working well during the WAI site visit with areas of the being blocked or over compacted. The fine grained nature of the gold at Zun-Holba suggests that alternative centrifugal concentrators such as the Knelson or Falcons concentrators would be more efficient and give significantly higher gold recoveries.*

3.7.3.4 Flotation

Bulk flotation of gold and sulphide minerals is undertaken using a basic rougher-scavenger cell arrangement. The rougher concentrates from each line are cleaned three times in a common cleaning circuit. The final flotation concentrate is pumped to the CIP Plant.

> ***WAI Comment:*** *WAI recommends that consideration be given to replacing the small rougher cells with larger volume cells supplied with a low pressure air source. The scavenger cells would remain in use.*

3.7.3.5 CIP Plant

The flotation concentrates are milled to a particle size to 99% passing 75 microns after classification with 150mm cyclones. The cyclone overflow passes to a thickener and the thickener underflow is pumped to the leach tanks where leaching takes place at 40% solids in nine 20m^3 tanks and carbon is added in a further nine 7m^3 tanks.

Gold is recovered from the loaded carbon using standard stripping and electrolysis technology and carbon is regenerated in a kiln operated at 600°C.

The separate Irokinda concentrates are loaded into a hopper from where they are fed, *via* a conveyor, to a ball mill. The mill discharge is cycloned and the underflows are returned to the mill. The cyclone overflows are thickened and pumped to the CIP Plant.

3.7.4 The New Plant

A 400tpd process line has been operational since October 2009 and the flowsheet is broadly similar to the existing plant with two stages of ball milling, each with a jig on the mill discharge.

The jig concentrates are cleaned twice using shaking tables. Jig tailings are cycloned and the cyclone overflows pass to rougher and scavenger flotation. Rougher flotation takes place in four 2m^3 cells and scavenger flotation in five 2m^3 cells. These cells are operated with a low pressure air source. The rougher concentrate is cleaned twice and pumped directly to the CIP Plant.

3.7.5 Assay Laboratory

The Assay Laboratory is certified by Irgiridmet until 2013. The laboratory uses fire assay as the main method of gold analysis giving a limit of detection of 0.2ppm Au.

Standard Reference samples are prepared in-house and with grades to match tailings, ore, Zun-Holba and Irokinda flotation concentrates and carbons with appropriate agreement limits. Between 2-5% of samples are sent to an external control laboratory (Irgiredmet) every three months.

Duplicate analyses are undertaken. The results of each assay pair should fall within set limits, depending on grade.

3.7.6 Technical Control and Plant Sampling

A metallurgical balance is prepared by the Technical Control Department. The flotation plant recovery is calculated by determining the weight of concentrate produced as determined from the CIP Plant Feed flowmeter reading and the pulp density. The CIP Plant recovery is calculated from the feed and tailings analyses.

The overall balance for the plant is obtained from the weights of the two dore products (gravity and CIP), the CIP tails weight and assay (from flow meter readings) and the flotation tailings assays and calculated weight.

> ***WAI Comment:*** *the practice of determining the weight of flotation concentrate produced should be reviewed. A more accurate measurement of this flow should be obtained from the analysis of the flotation plant feed, concentrate and tailings streams. Mass recoveries should be based on gold and sulphur analyses.*
>
> *During the WAI site visit the CIP tailings automatic sampler was deactivated due to the sampling bucket being full before the sampling for the period was complete, a result of the incorrect size of the sample split; it is recommended that the sample cutting time should be adjusted accordingly to ensure that a full sample is obtained.*

3.7.7 Agreement between Measured and Calculated Head Grades

The agreement between the measured head grade (as determined from the feed belt sampling) and the back calculated head grade is poor, the former being consistently higher (average 12.5%) and is being reviewed by Management.

> ***WAI Comment:*** *parallel sampling exercises should be undertaken in which the belt is stopped on a regular basis and all material over the width of the belt is removed manually. A more rigorous cone and quartering or riffling method should be used to split the plant feed sample, preferably after fine crushing.*
>
> *A potential error in the back calculated head grade will also be the estimate of the weight recovery to flotation concentrates determined using flow meters. This flow should be calculated using chemical analyses.*

3.7.8 Additional Recovery of Gold from Process Water

Up to 30kg of gold is also recovered from the water in the CIP tailings dam; water is decanted and pumped at a rate of 90m^3/h through seven carbon tanks, each of varying capacity. In the future it is planned to replace these tanks with the square tanks from the CIP Plant.

3.7.9 Metallurgical Performance

The metallurgical performance of the plant for 2004-2010 is summarised in Table 3.9.

Table 3.9: Zun-Holba Plant Metallurgical Performance						
Year	Ore treated Tonnes	Feed g/t Au	Recovery %			
			Gravity	Flotation	CIP	Overall
2010	347,742	6.48	14.4	74.5	97.0	86.8
2009	315,777	8.07	18.3	73.6	97.7	90.2
2008	271,948	8.34	26.4	68.3	97.5	92.9
2007	259,574	8.87	28.1	66.6	97.6	93.1
2006	245,655	9.30	30.2	64.3	97.9	93.3
2005	240,958	9.26	30.9	63.8	97.3	93.0
2004	225,208	9.51	33.6	61.0	97.4	93.1

The plant throughput has increased from 225,208 (2004) to 347,742 in 2010. Over the same period the plant head grades have fallen from 9.51 to 6.48g/t Au.

Gravity gold recovery has fallen over the period from 33.6% in 2004 to 14.4% in 2010. The fall in gravity recovery corresponds with an increase in flotation recovery over the same period from 61.0 to 74.5%. The CIP recovery has remained reasonably constant over the period, which suggests that the gold mineralisation has become more fine-grained but that it is still in a cyanide leachable form.

The overall plant recoveries have remained remarkably consistent over the period, although showed a drop in 2010 to 86.8%.

3.7.10 Consumables

The steel consumption is indicative of a moderately hard ore. The consumption of xanthate (186g/t) and cyanide (1.48kg/t) is high.

3.7.11 Labour

There are a total of 17 managerial/technical positions and 76 workers operating on a 12 hour shift system.

3.7.12 Conclusions

The Zun-Holba plant shows a high degree of metallurgical efficiency according to information provided by the Technical Control Department although there is an on-going discrepancy between the measured plant head grade and back calculated head assays from plant production data and it is strongly recommended that this issue is the subject of a detailed, independent study to determine the reasons for the inconsistencies.

There has been a reduction in the proportion of gravity recoverable gold over recent years, which suggests that the gold mineralisation is becoming finer. However, there does not appear to be a reduction in process recoveries, which indicates that the ore is not becoming more refractory.

The plant management is considering abandoning gravity processing and using flotation only to recover gold ahead of cyanidation. This would simplify operations and would enable an accurate measurement of the plant head grade to be made by sampling the grinding circuit cyclone overflow.

However, WAI believes that the removal of the gravity circuit would have a negative effect on recovery and loss of revenues would not compensate for the lower plant operating costs.

The relationship between mass recovery and gold recovery in flotation needs to be fully evaluated. It may be possible to produce a lower grade of flotation concentrate at higher stage recovery. Additional CIP Plant capacity will enable the treatment of the higher throughput from flotation. Alternatively, leach kinetics may be increased through higher cyanide additions.

The old flotation cells need to be replaced with larger machines with a low pressure air source. The use of Knelson concentrators should also be considered.

The reporting system for analysis of plant products should also be reviewed and an alternative system introduced in which all results are reported solely to the Technical Control Department.

3.8 Environmental & Social

3.8.1 Introduction

WAI compared existing practices at Nord Gold operations to international standards, codes and best management practices e.g IFC Performance Standards and World Bank Guidelines.

To date, Buryatzoloto has not needed to demonstrate compliance with international best practice such as IFC Performance Standards on Social and Environmental Sustainability, WB/IFC Guidelines on Health and Safety and Equator Principles, since money for the project development has been sourced internally.

> ***WAI Comment:*** *WAI considers that an EHSC Action Plan will need to be developed to demonstrate how this can be achieved, in addition to the resources and timescale required. Current staff have, knowledge of, but no experience in introducing the international standards. Development and implementation work has begun, but further assistance will be needed to support this objective.*

3.8.2 Environmental Permitting

It is reported that the project is fully permitted by State and Local Authorities and is compliant with the State requirements.

The latest OVOS document (equivalent to EIA) was approved in 2008 and covers the expansion of the processing facilities, namely the fourth line of beneficiation process.

A programme of environmental monitoring has been prepared, post 1999, and includes monitoring of the surface and underground water and laboratory analysis of the soils, snow and air emissions.

It has been reported that there is a licence for cyanide transportation, storage and use in place, although WAI is not able to confirm this and recommends the preparation of a plan to help minimise the risk of polluting the sensitive aquatic and biotic environments. No major breaches have been identified during state inspections and audits. Russian operations have always sought to comply with Russian Federation requirements and continue to meet the applicable standards. Compliance with these standards is monitored and results are reported. Non-compliances are addressed and corrective actions are implemented and reported to regulatory authorities, it is important to note the reluctance of local Russian authorities to change their approach to standards. This does not encourage the implementation of international standards, however the client continues to reinforce cyanide management practice and strives to improve its HSEC programmes and performances

3.8.3 Environmental Protection

3.8.3.1 Introduction

Air, water, flora, fauna and soils are afforded protection in the project lease area and regular sampling is carried out, with analysis for potential pollutants undertaken in the environmental laboratory at the mine as well as by outside state organisations.

3.8.3.2 Air

Currently, total air emissions from all air pollution sources reaches 481 tonnes per year. The measures taken at Kholbinsky Mine in order to mitigate negative impact on the atmosphere include dust suppression, using water bowsers and cyclones with scrubbers for dust collection at the main boilers plant. All personnel were observed to be wearing appropriate personal protective equipment and dust masks.

Air quality tests are conducted annually by the Regional Environmental Research Centre for dust, suspended particulates, SOx, NOx, CO and soot amongst other determinands at selected sites. Dust in the PM10 range is important to assess as a hazard to health. Estimated Production Hazard Criteria (EPHC) have been calculated according to Russian Federation criteria as non-hazardous.

> ***WAI Comment:*** *WAI believes that dust reduction and control measures are being targeted at Kholbinksy Mine, in line with international best practice and to reduce risk to personnel.*

3.8.3.3 Water

Surface water quality studies carried out by Buryatgeomonitoring Institute in 2003 have found the impact of Kholbinsky operations on the environment to be localised and temporary.

Regular monitoring of surface water within the licence area has been undertaken since 1999. Surface and ground water are sampled and analysed regularly. Fish monitoring in both the Samarta and Zun Holba rivers is undertaken once every four years.
Potable water and water for technological purposes are sourced from the underground source at a depth of 300m and domestic effluents from the mine and the settlement are mostly drained to cess pits from where it is taken for treatment in settlement ponds.

Supernatant waters from the TMFs are sent back into the beneficiation process upon settling and this is found to be efficient.

3.8.3.4 Land

The likely impact on land of the project is localised and temporary. The mine is located in an area designated for production activity, all disturbed land will be required to be reclaimed at closure and any soil which is removed is required to be kept and used for reclamation purposes. To meet this requirement, a Mine Closure Project was prepared in 1999 and has been approved with the state authorities.

The timber yard is required to be cleaned up and returned to forestry when the logging is finished.

3.8.4 Waste Management

A Waste Management Plan was introduced in 2009 and the company is in possession of a licence for the collection, use, neutralisation, transportation and disposal of wastes.

Waste rock is classified as non-toxic and is stockpiled in engineered waste rock dumps. There is a heavy use of cyanide in the processing of ore at Samarta resulting in high concentration of cyanide in the tailings (600 to 800mg/l). All waste water is recirculated to the plant for recovery of gold in precipitate within a fully closed system.

Solid waste is classified as non-toxic and dumped at surface by ‘edge-tipping’. Other production wastes are reused, recycled or disposed of offsite in accordance with a waste management plan.

State approvals consider that any damage to land, fauna and flora is not irreversible and is insignificant in the region as a whole.

WAI Comment: *overall, the waste management practices in place for Kholbinsky mine are sound and well organised. However, WAI would emphasise that such heavy use of cyanide poses high risks to both the surrounding environment and human health unless properly managed and would therefore recommend that the Company undertakes necessary steps towards reinforcement of its cyanide management practices.*

3.8.5 EHSC Management

Mechanisms for environmental, health and safety management have been introduced for Kholbinky operations. There is a Health and Safety Management System for the operations devised by Buryatzoloto which was introduced in 2009 together with new regulations governing production control safety.

A detailed action plan has also been implemented to enhance the existing safety programs. The action plan was formally put into operation in February 2010. The number of Health and Safety personnel is also being increased to meet the demands of the growing workload.

The Key elements of the action plan are:

- Change the way safety training is provided through the use of more effective communication tools, regular refresher training and more first aid courses;
- Perform detailed job safety analysis in all departments;
- Improve worker safety awareness with the help of the revised job safety analysis, communication of safety instructions during “line-up” meetings and more effective use of the internal employee newsletter to communicate safety messages;
- Implentation of a safety incentive program to motivate better safety attitudes and performances; and
- Work with technical departments to ensure continued full compliance with applicable technical requirements (Examples; manway construction, roof support, raise boring etc)

Accident statistics for the half year 2009 show marked improvements over the previous four years, showing zero fatalities for the first time and a marked reduction in severe accidents.
Environmental reporting, management and monitoring appear to be well conducted to meet state requirements. The environmental studies carried out at the site are well organised and appropriate to meet state requirements.

The current management maintains an open dialogue with community members and allocates finance on an ad-hoc basis with no fixed budget for community development in place.

Costs estimates for the 1999 mine closure plan have been re-cast to 2009 prices to around US$7M. It is not clear if or how these sums have been accounted for and the lack of a liquidation fund presents a major impact on cash-flow and accounting, in this connection. Financial security for mine closure is not common in Russia. Estimated mine closure costs are only reported as financial liability in IFRS financial statements.

Furthermore, WAI recommends that to ensure compliance with national quality standards and for a project to be fully compliant with international best practice the provisions of the latest IFC performance standards and WB Guidelines must be adopted. In particular the following system plans and programs should be addressed:

- Environmental and Social Action Plan (ESAP) –The ESAP is one of the critical components for a management plan. It is used to ensure that the company continues to improve in all aspects of the project and that IFC Performance Standards and Equator Principles are met. The existing Environmental Action Plan should be extended to cover capacity and organisation, training, operational control and management to ensure compliance with IFC Performance Standards;

- International Cyanide Management Code - Overall, WAI does not consider that operations are currently compliant with the International Cyanide Management Code. WAI would recommend that an independent specialist be contracted to inspect operations in detail, and help Buryatzoloto to identify measures that could be implemented to ensure compliance. WAI would also recommend that and action plan, to address aspects of cyanide management be compiled, to identify priority areas, and outline a timeframe within which actions will be taken;
- Community Health, Safety and Security Plan – several elements of this plan have addressed measures to protect employees at the project, notably through improved occupational health and safety measures introduced at the operations; however international best practice requires plans to ensure that the wider community is considered and an integrated plan should be prepared;
- Endangered and Protected Species Protection Plan – this is a required to be considered, although it is unlikely to require extensive work to ensure that such species are adequately protected; and
- Community Development, Public Consultation and Disclosure Plans – whilst arrangements are in place for wide levels of public participation and assistance is provided on a request led basis, these plans should be formalised. This is particularly important since the community at Zun-Holba Mine relies entirely on the project for its economic existence. If closure should occur this would have a major destabilising effect on the community, for which preparation should be made.

WAI Comment: *WAI recognises that to date the company primarily sought to achieve national compliance, which has been done, and hence many of the above recommendations have hitherto not been required. However, WAI recognises the company's commitment to achieving international compliance, and highlights the issues above as areas where further actions will be required. WAI also recognises that whilst Buryatzoloto/Zun-Holba Mine has a good, well motivated, young, environmental, health and safety team, extra support and guidance may be required in order to effectively implement the range of EHSC recommendations made, particularly with regard to setting up an EHSC Management System.*

3.9 Company LOM Plan

The Zun Holba Mine is a mature operation which has continuously produced high grade gold ores from a number of veins. As at Irokinda, historically, the mine has never kept a large "reserve" balance, preferring instead to drill and explore annually to replenish mined material.

The reserve replacement track record has illustrated an annual replacement ranging from 0-18% per annum in total reserves as shown in Table 3.10 below. The exploration budget has increased under the ownership of Buryatzoloto, up to the last quarter of 2008, but as a consequence of the global "financial crisis" resulted in the company reducing the budget for 2009. This situation has now been reversed in 2010, with an allocated budget of US$9.3M for 2010 and US$16.8M for 2011 to potentially extend the mine life up to 10 years. Again as with Irokinda, this aggressive drilling campaign aims to delineate extensions to known mineralisation both in depth and along strike, and to explore for new orebodies which may be present both at increased depth, on the margins of existing structures and that have not been accessible to drilling until the present.

Table 3.10: Zun Holba- Reserve Replenishment and Budget			
	Exploration Budget (US$M)	**Reserves' Added (Au t)**	**% of Reserve Replacement**
2002	1.7	2.0	5.68
2003	1.5	3.7	18.28
2004	0.8	0.1	0.35
2005	0.5	0.6	3.92
2006	0.3	0.0	0.01
2007	1.6	1.5	11.11
2008	2.8	1.8	14.84
2009	1.9	1.3	11.89
TOTAL	**11**	**11**	

At the beginning of 2010, total C_1+C_2 reserves were sufficient for the operation of Zun Holba until 2013. The existing C_1 and C_2 "reserves" (comparable with *Indicated and Inferred* resources using an approximate reconciliation with JORC Code (2004) guidelines) as of 01 November 2010 are given in Table 3.11 below

Table 3.11: Zun Holba"Reserves" as of 01 November 2010					
"Reserve" Category (Dilution and recovery factors are not applied)	**JORC Equivalent**	**Ore (t)**	**Au Grade (g/t)**	**Au Metal (kg)**	**Au Metal (koz)**
C_1	*Indicated*	41,496	16.8	696	21.6
C_2	*Inferred*	58,653	13.5	7,812	243.0
C_1+C_2		**622,149**	**13.7**	**8,508**	**264.6**

Based on the remaining reserve base, WAI is not able to produce a LOM plan and economic model. However, WAI has reviewed the company's Life of Mine (LOM) model which assumes a conservative continuing replenishment of reserves of the mine during the next few years. The company's LOM model excludes any additional mineralised potential that could be identified to further supplement and increase the resource base in the future. This LOM model allows extension of the period of operation until at least 2015.

The LOM extension to the Zun Holba 'reserves' proposed in the company model is given in Table 3.12 below.

Table 3.12: Zun Holba - Buryatzoloto Projected LOM 'Reserve' Base							
		2011	**2012**	**2013**	**2014**	**2015**	**2016**
Gold 'Reserves' (beginning of the year)	**kg**	9,093	7,368	6,977	5,587	4,196	2,805
Grade (before dilution)	**g/t**	13	13	12	12	10	8
Ore	**000t**	699	586	564	451	402	353
Additional 'Reserves'	**kg**	2,000	3,000	2,000	2,000	2,000	0
Grade in Additional 'Reserves'	**g/t**	11	11	11	8	8	0
Ore	**000t**	185	278	185	250	250	0

The existing C_1 and C_2 "reserves" (*Indicated and Inferred resources using an approximate reconciliation with JORC Code (2004) guidelines*) as of 1st November 2010 held by the mine are given in Table 3.13 below.

Table 3.13: Zun Holba "Reserves" as of 01 November 2010				
"Reserve" Category (Dilution and recovery factors are not applied)	**Ore (t)**	**Au Grade (g/t)**	**Au Metal (kg)**	**Au Metal (koz)**
C_1	41,496	16.8	696	21.6
C_2	58,653	13.5	7,812	243.0
C_1+C_2	**622,149**	**13.7**	**8,508**	**264.6**

Clearly, the mine will generate significant revenue during 2010/11 and there is every opportunity that new extensions and discoveries to mineralisation will be made, although this latter observation cannot be quantified at this time.

> ***WAI Comment:*** *although the current remaining "reserves" at Zun-Holba are limited, significant revenue will be generated during 2010/11. Until the exploration programme is completed, the long-term future of the mine is difficult to quantify; however WAI believes that with the on-going exploration drilling programme and the historical annual replenishment of 'reserves' the potential to delineate further resources, which will ultimately transfer through to reserves and allow the mine to continue forward, is high.*

3.10 Financial Model

The financial Model for Zun Holba has been discussed under section 2.10.

4 TABORNOE

4.1 Location, Access and Infrastructure

4.1.1 Background

The Tabornoe gold deposit is situated in a remote part in the SW of the Yakutia (Saha) region within the Russian Federation. The deposit is located approximately 125km north east of the BAM railway station of Ikabya. Access to this mining operation is possible via a new normal 4WD vehicular access road that has recently been completed. The location of Tabornoe is shown in Figure 4.1 below.

Figure 4.1: Location of Tabornoe Gold Deposit within the Yakutia Region

There is no permanent population in the area, the nearest settlements of Olekma; Khani and Ikabia being located approximately 80km south on the Baykal-Amur trunk line (BAM).

There is no external power supply to site. All power requirements are provided by on-site diesel generation.

Water supply to the mine is mainly via road tankers that transport water from Lake Usu approximately 8km due south from the mine. Springs local to the mine site are also used occasionally.

Due to the difficulties in accessing the mine site personnel are accommodated on site on a rotation basis.

The operational mine is completely self sufficient, containing staff accommodation, canteen, first aid facility, technical service offices, process plant and laboratory, mechanical workshops and various stores. All supplies are brought in to the camp in order to maintain the operation for several months continuously. The only means of communication with the outside world is by satellite telephone,radio and Internet. All staff operates on-site on a rotational basis for up to three months at a time.

The Tabornoe deposit was discovered in 1978 as a result of a government geological survey mapping programme. Continued exploration over the next 20 years culminated in the development of the trial mine at Tabornoe which was first started in 2001, with the trial heap leach being commissioned in November 2002.

4.2 Topography and Climate

The deposit is located in mountainous terrain incised by river valleys, with the elevation of the area typically ranging from 1,100 to 1,375masl.

The climate is extreme continental with a long winter and short, hot summer. The maximum temperature is +35°C (July) whereas the minimum is -55°C (December, January) with the annual average of -8.5°C.

Annual precipitation averages 370mm, with most falling in July-August. Rivers are frozen from November – April/May. Snow cover is normal between October and May. Permafrost exists to a depth of 400-450m. Soil melts during summer to depths of 1-3m; however north facing slopes barely defrost at all. Winds are predominantly from a north-westerly direction.

An all-weather road has now been completed from the mine to Ikabia**,** which has greatly improved the supply and logistics**.** Total expenditure on road construction amounted to 100M rubles.

> ***WAI Comment:*** *the remoteness of this deposit and the resultant lack of local infrastructure, has an overall negative impact on the project, reflected in the operating costs, particularly those for power, labour and logistics.*

4.3 Licences

The exploration licence (No.YaKU- 12112BP), covering an area of 58km^2, contains both the Tabornoe, Temny Tabornoe and Gross mineral occurrences, and has been held by the "LLC Neryungri-Metallic" Company, (now wholly owned by Nord Gold) since 22 December 2003; it is valid until 31 December 2008, and has now been extended until 31 December 2011.

The corner co-ordinates of the exploration licence are given in Table 4.1 below.

Table 4.1: **Tabornoe Exploration Licence Co-ordinates**

Point	Latitude (North)	Longitude (East)
1	57°39`40``	119°46`50``
2	57°40`15``	119°56`25``
3	57°40`30``	120°03`15``
4	57°39`50``	120°03`15``
5	57°38`45``	119°59`10``
6	57°37`45``	119°53`20``
7	57°37`30``	119°47`10``

The Exploitation licence for Tabornoe (No.YaKU- 01860BP), held by "LLC Neryungri-Metallic" Company, which is now wholly owned by Nord Gold, is currently valid until the end of 2020. The licence covers an area of 0.93km^2. Neryungri-Metallic received the licence in 2001.

The corner co-ordinates of the licence are provided in Table 4.2 below.

Table 4.2: Tabornoe Mining Licence Co-ordinates (Licence number is YaKU #01860 BR, expires 30.12.2020)

Point	Latitude (North)	Longitude (East)
1	57°39`10``	119°49`10``
2	57°39`20``	119°49`35``
3	57°39`15``	119°50`10``
4	57°39`20``	119°50`40``
5	57°39`25``	119°51`50``
6	57°39`20``	119°50`05``
7	57°39`20``	119°51`20``
8	57°39`10``	119°51`25``
9	57°39`05``	119°51`15``
10	57°39`05``	119°51`00``
11	57°39`00``	119°50`30``
12	57°39`00``	119°50`10``
13	57°39`00``	119°49`35``
14	57°39`05``	119°49`10``

4.4 Geological Setting

4.4.1 Regional Geology

The Tabornoe deposit is situated in the south western corner of the Uguskiy Graben. This graben is filled with Lower Proterozoic sediments which discordantly overlie early Archean gneisses and crystalline schists, intruded by later Archean granites and later cretaceous gabbro and syenite (Figure 4.2 below).

The principal tectonic features in the area consist of a set of N-S striking, steeply dipping faults a NE – SW striking and steeply dipping shear zone and a shallow thrust of E-W strike, dipping to the south at 20-30°; gold mineralisation is believed to be directly related to the latter.

The Tabornoe deposit is hosted by a potassic metasomatite zone, developed in the sandstones, which strikes E-W and dips to the south, parallel to the orientation of the thrust zone.

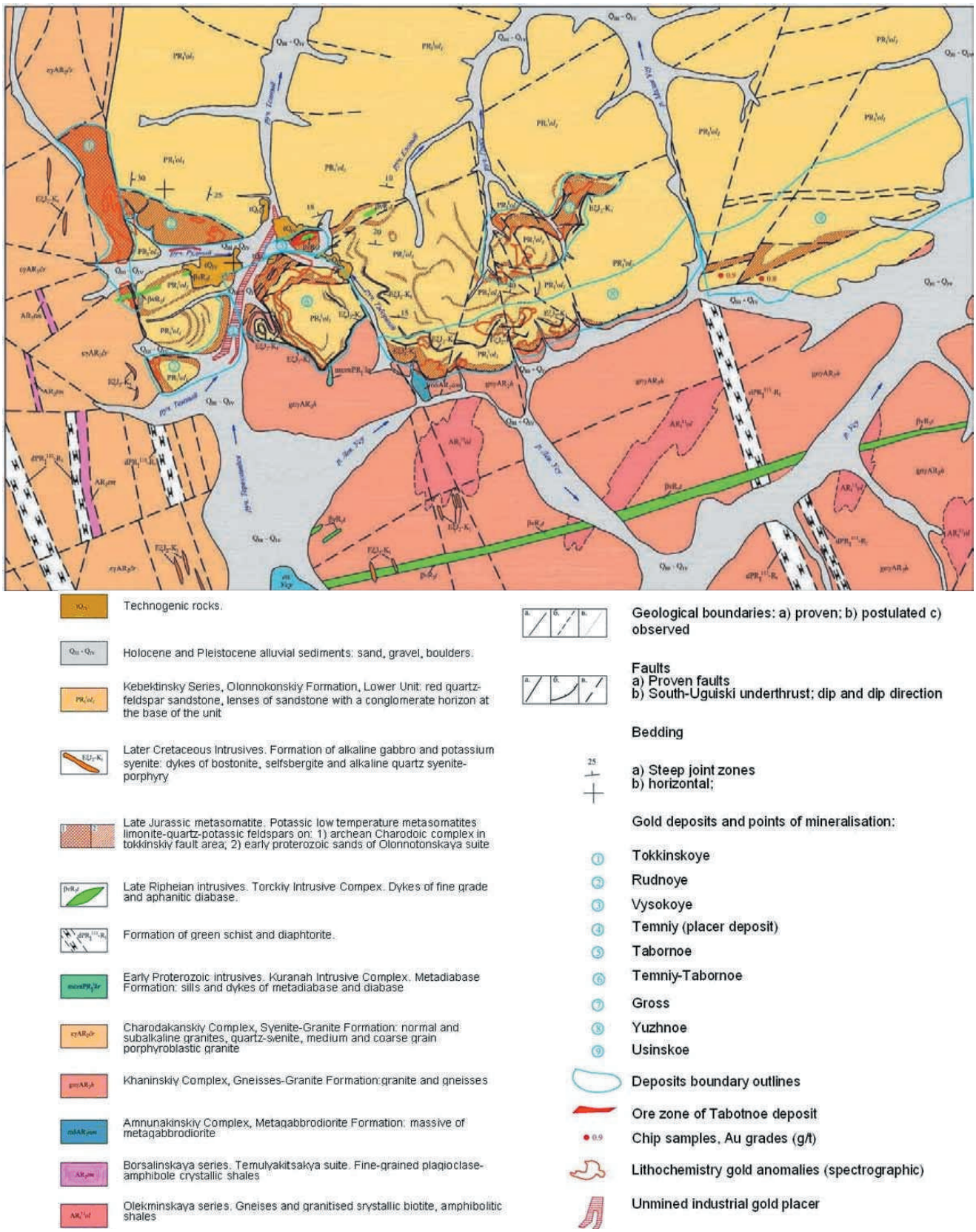


Figure 4.2: Tabornoe – Regional Geology (Not to Scale)

4.4.2 Local Geology

4.4.2.1 Stratigraphy

The Tabornoe deposit is hosted by early Proterozoic sandstones of the Olonnokonskiy Formation. The sandstones are massive, fine to medium grained, and of quartz-feldspar and feldspar-quartz composition. The sandstones are horizontal or shallow dipping to the north, northeast and east, typically at dips of 10-30°.

4.4.2.2 Intrusive Rocks

A diabase dyke of Riphean Age, striking from NEE – SWW and dipping at 70° to the south is shown in the Figure 4.3 andFigure 4.4 is heavily altered and disrupted, has a maximum thickness of 25m, a proven strike length of 700m and is mineralised, often carrying good gold grades.

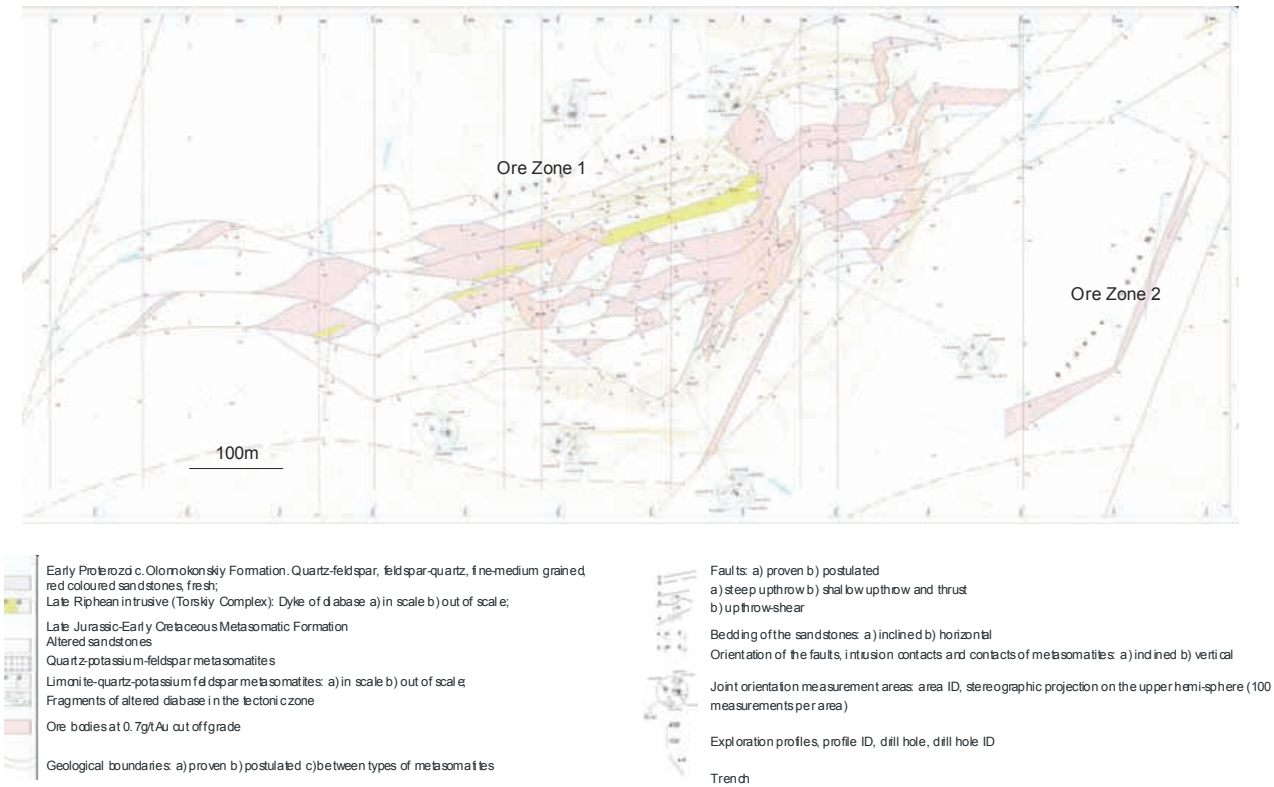


Figure 4.3: Local Geology of Tabornoe Inside the Open Pit, 2005

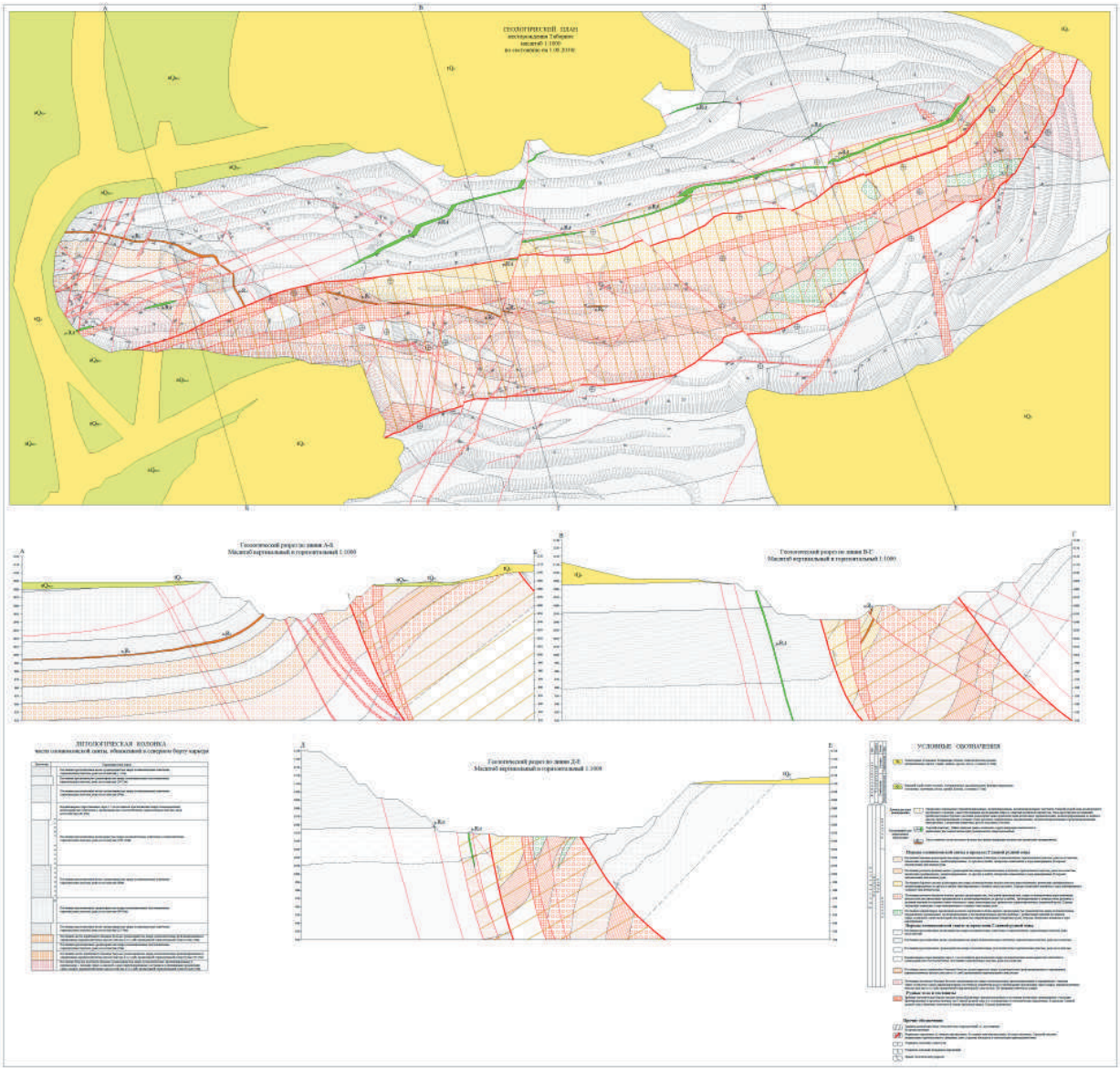


Figure 4.4: Local Geology of Tabornoe Inside the Open Pit, Yu.Zubkov&A.Babich, 2010

4.4.2.3 Structure

The Tabornoe orebody is located within a northeast striking shear zone, the fissures of which form natural west and east boundaries to Ore Zone 1. The wedge zone between the fissures is broken by a system of east-west and northeast striking upthrows, dipping to the south at 70° and thrusts, dipping at 10-35°, also to the south.

4.4.2.4 Lithology and Alteration

Three main lithological types have been identified at the deposit: sandstones, diabase dykes and metasomatites.

Four stages of alteration have been identified in the open pit, ranging from fresh sandstone to limonite-quartz-potassium feldspar metasomatites in which the original structure cannot be distinguished. Altered sandstones and metasomatites contain mono- or polymineralic veinlets of quartz, potassium feldspar and clinozoisite, 0.1 to 0.8mm in thickness, and also contain very finely disseminated molybdenite, oxidised pyrite crystals and very rare free gold.

4.4.2.5 Mineralisation

Tabornoe comprises a complex set of orebodies occurring in altered sandstone and diabase. An idealised cross section through Ore Zone 1 is shown in Figure 4.5 below. Gold bearing mineralisation occurs in all altered rocks types, notably including the diabase dyke. Typical gold grades from core samples found within each rock type are summarised in Table 4.3 below.

Table 4.3: Typical Gold Grades by Rock Type

Rock Type	No of samples	No of interse-ctions	Total length (m)	Avg. Thickness (m)	Average Au Grade (g/t)		Metal Accumulation (g/t×m)
					Min*	Max*	
Altered sandstone	1070	95	1058.7	11.1	1.64		1,763.3
					0.74	8.54	
Qtz-potassium-feldspar metasomatite	331	30	365	12.2	1.70		621.0
					0.72	3.20	
Limonite-qtz- potassium-feldspar metasomatite	528	39	512.5	13.1	1.63		835.4
					0.72	4.57	
Altered diabase	134	9	151.1	16.8	5.03		760.0
					0.91	13.83	
Total	2,063	173	2,087.6		1.89		3,979.7

Note: minimum and maximum grades are given for the intersections.

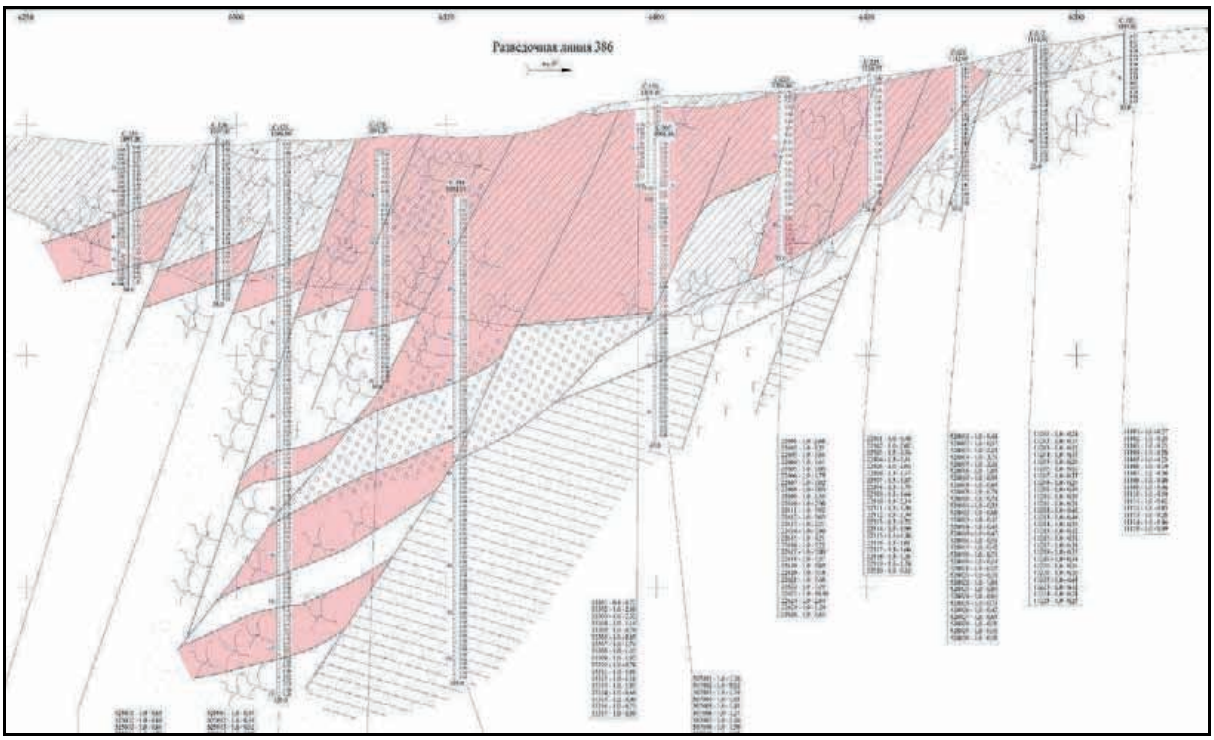


Figure 4.5: Typical Cross Section through Ore Zone 1

Tabornoe Ore Zone 1 comprises a series of lenses which normally have no geological contacts and could not be identified from field observations, the outline being identified by assay cut-off grade only. Interpretation of the individual “payable” orebodies should be considered as highly speculative and at best conceptual.

WAI Comment: *It appears that faulting plays a significant role in the control of the mineralisation and therefore further steps should be taken to improve its mapping and interpretation. The diabase dyke identified within Ore Zone 1 controls high grade mineralisation and more attention should be paid to the three dimensional interpretation of this dyke by in-pit mapping.*

In accordance with this WAI recommendation, appropriate structural investigations were performed in 2010 and an excellent detailed geological map has been compiled at a scale of 1:1,000 (see Figure 4.4).

4.5 Exploration Activities

4.5.1 Deposit History

The outcome of 1978 government exploration work was a resource estimate for placers in the Tabornoe and Temniy streams of 563kg Au in the C_2 category at average grades of 0.74g/m^3 and 1.96g/m^3 for the Tabornoe and Temniy placers respectively. Other hard rock targets identified were Black (Dark) Tabornoe and Gross with maximum grades from selected trench channel samples of 5.7-7.1g/t Au.

From the thermometry performed on one of two hydrogeological boreholes drilled during the period 1980 a thickness of permanently frozen rock of up to 400-450m was identified. Water flow in both boreholes was of the order of 0.1l per minute.

In 1990-1991 trenches were excavated on a spacing of 500m along strike and prognostic resources of the P_1 and P_2 categories were estimated at 50t of gold at grades of 2.14-2.37g/t Au.

In 1995-1999 the Uzhyakutgeology Government Exploration Company continued exploration at Tabornoe. Metallurgical testwork for the Tabornoe ore was undertaken on one sample, which demonstrated a recovery of 75-85% in laboratory heap leach tests. Resources of the C_2 category were estimated to a depth of 120-150m (using an 0.7g/t Au cut off grade (COG)) at approximately 17.4Mt of ore containing 27t of gold at average grade 1.55g/t Au. Low grade resources as well as prognostic resources were also estimated.

Trial mining at Tabornoe first started in 2001, with the trial heap leach being commissioned in November 2002.

A summary of all exploration works undertaken from 1976-2004 is given in the Table 4.4 below.

Table 4.4: Summary of Exploration Work (1976-2004)

Year	Geological mapping (km^2)		Trenching (m^3)	Drilling (m)	
	1:50,000	1:10,000		Core	Churn
1976-1979	150	-	-	-	-
1977-1981	-	20	15,000	-	1,000
1990-1991	-	30	20,000	-	-
1995-1998	-	48	24,500	4,570	709
2001-2004	-	-	80,978	11,766	-

Further exploration works conducted from 2004 -2010 are summarised in Table 4.5 below.

Table 4.5: Summary of Exploration Work (2004 – 2010)			
Type of Works	**Unit**	**Totals**	
		Planned	**Actual**
Exploration mapping	km	460	588
Sampling of trench material	sample	1,160	2,600
Soil sampling with grid 50-100x10-20m	sample	5,150*	12,000
Soil sampling with spacing 20m	sample	-	528
Magnet surveying with grid 100x5m.	km^2	50	58
Magnet surveying in profiles (spacing 5m)	km	20	23
Gamma-surveying grid 100x5m	km^2	-	20
Trenching by dozer	m^3	211,190	80,000
Manual trenching	m^3	696	650
Core drilling	m	163,377	204,706
Downhole surveying	m	163,377	202,500
Individual sampling	sample	5,000	6500
Channel sampling (1.0m intervals)	sample	4,640	1,277
Core sampling	sample	2,176	204,706
Slime sampling	sample	-	948
Schlich sampling from core	sample	442	500
Schlich sampling from channels $0.04m^3$	sample	812	127
Boring boarding 0.7m	km	144,2	201,6
Boring boarding 1.0m	km	20	29,9
Atomic emission spectrometry with ICP with multi acid digestion	analysis	-	12,500
Semiquantative spectral analysis	analysis	17,660	-
Atomic absorption analysis for gold	analysis	12,355	29,180
Fire assay analysis for gold (external geological control AAA)	analysis	1,200	5000
Preparation and description of thin and polished sections	pcs	550	125

During this same period further diamond drilling was undertaken; this is broken down on an annual basis in Table 4.6 below.

Table 4.6: Core Drilling Summary (2004-2007)		
Year	**No of Drillholes**	**Total Meterage (m)**
2004	3	302
2005	60	1,402.2
2006	234	6,256.8
2007	90	6,774.9
Total	387	14,735.9

A summary of core drilling conducted from 2008-2010 is given in Table 4.7 below.

Table 4.7: Core Drilling Summary (2008-2010)			
Year	**Area**	**No of Drillholes**	**Total Meterage (m)**
2008-2009	Gross	310	68,793
	Gross Technology Holes	30	4,907
	Temny-Taborny	222	42,686
	Temny-Taborny – Technology Holes	12	810
	Vysokiy	51	8,440
	Rydny	10	1,848
	Tokkinskiy	19	3,956
	Yuzhny	26	3,445
	Condemnation DH	8	798
	Taborny Open Pit	132	12,501
Total 2008-2009		**820**	**148,184**
2010	Gross - spacing of 20x20m	141	35,271
	Gross - spacing of 80x80m	12	4,449
	Temny-Taborny	35	11,129
	VysokyI	4	691
	Yuzhny	15	3,704
	Geotechnical Holes	15	1,281
Total 2010		**222**	**56,522**
Total 2008-2010		**1,042**	**204,706**

In 2006 Nerungri-Metallic prepared resource and reserves estimates for Tabornoe which after amendment were subsequently approved by GKZ. C_1 and C_2 category reserves have been defined by borehole spacing of 40-60 x 40-60m and 100 x 100m respectively. Up to the end of 2005, some 19,785m had been drilled on the deposit and these data have allowed Tabornoe resources and reserves to be estimated and approved by GKZ in 2006.

4.5.2 Exploration Targets

Exploration to date has been focused both within the known Tabornoe reserves (in-pit ore definition drilling on a 20 x 20m basis) as well as at the following mineral occurrences and prospecting areas (see Figure 4.6 below).

4.5.2.1 Western Extension of Reserves to the Current Open Pit

A possible western extension of Ore Zone 1, contained within the current open pit, has been recognised and a diamond drill programme has been conducted to define these potential resources. The programme consisted of approximately 5,000m of drilling on 6 profiles on a drill spacing of 20x20m.

4.5.3 Budgets of the Exploration of Tabornoe & Gross (2010-2011)

The total budget for 2010 exploration activities was US$13.3M; and
The proposed budget for 2011 exploration activities is US412.7M

> ***WAI Comment:*** *WAI considers the budget for 2011 is adequate to cover the proposed exploration program.*

4.5.4 Main Exploration Targets For 2011

The main exploration works for 2011 are to include:

- Infill drilling (to grid 80x80 and 40X40 m) and step out drilling to the south and at depth at Gross;
- Preparation of a Pre-feasibility Study and preparation of report for GKZ to accelerate the commissioning of Gross in 2013; and
- Continuation of metallurgical, geotechnical, environmental investigation etc at Gross.

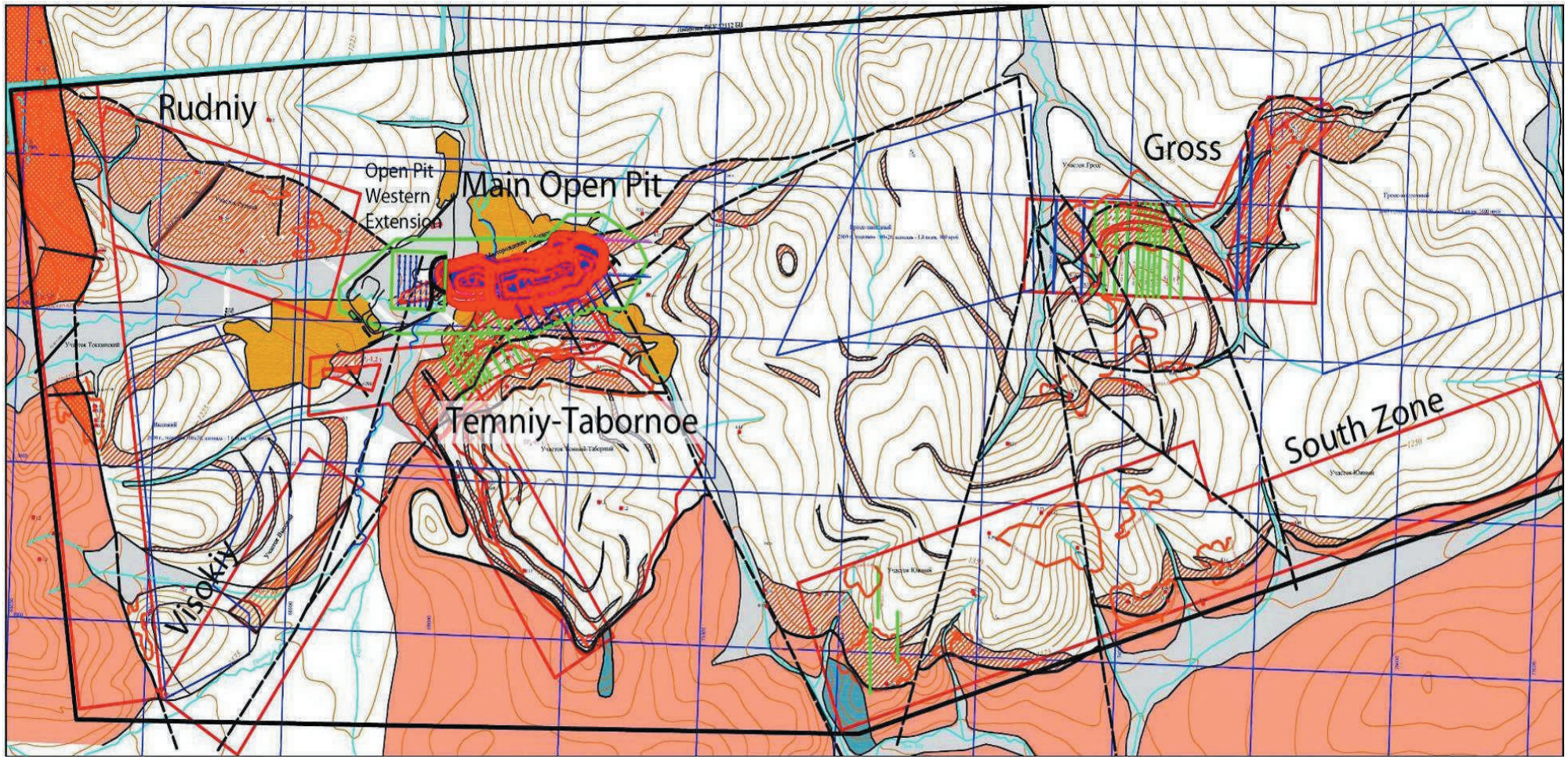


Figure 4.6: Map showing Location of the Principle Exploration Targets

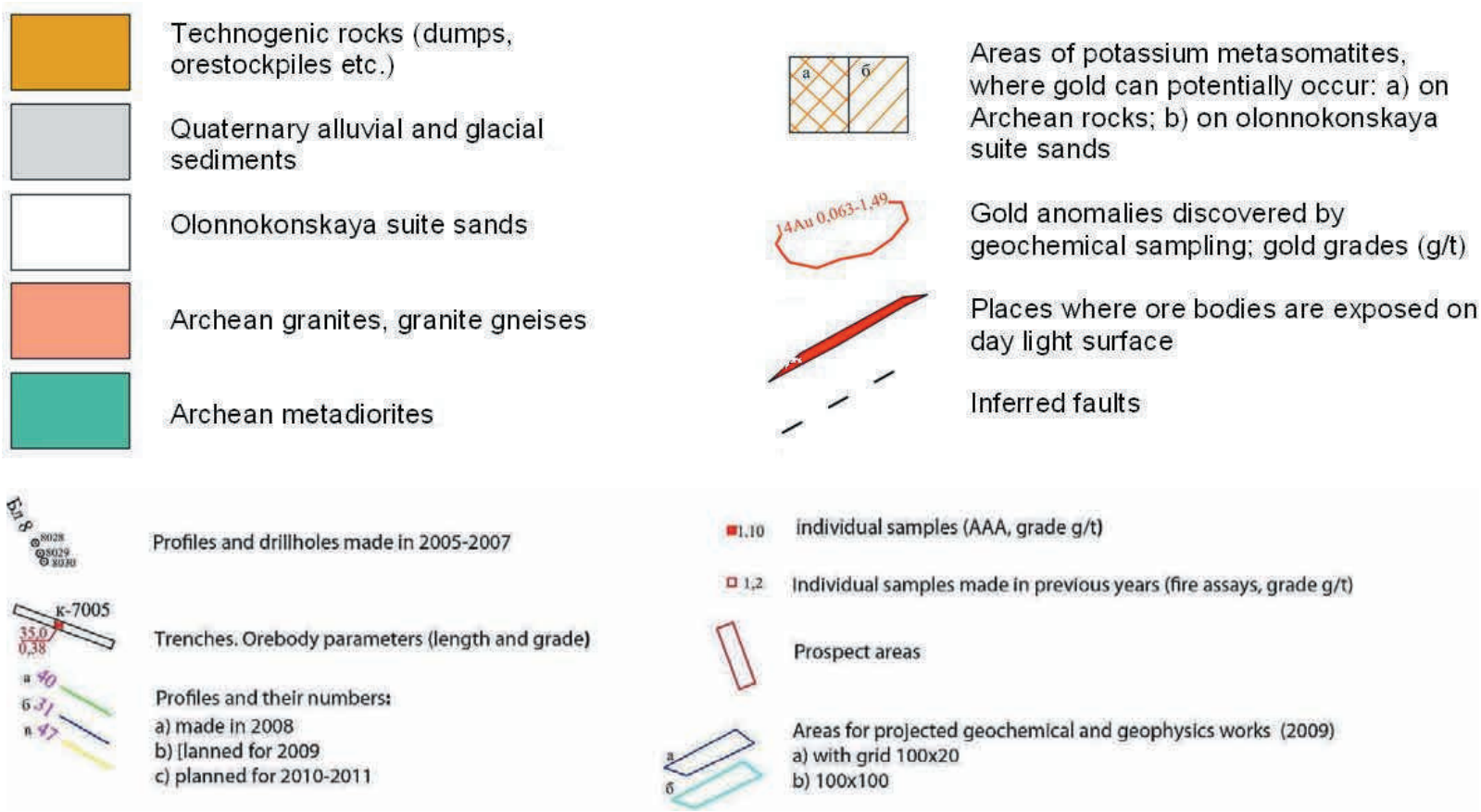


Figure 4.7: Legend for Figure 4.6 above

4.5.4.1 Temny Tabornoe

The Temny Tabornoe occurrence, located approximately 500m to the south of Tabornoe Ore Zone 1 (Figure 4.8) below appears to comprise two zones of gold bearing metasomatites lying parallel to the Tabornoe Ore Zone 1. Historical drilling on the lower zone intersected a few intervals of potentially economic mineralisation and this was followed up by an intensive drill programme by Neryungri-Metallik in 2008 the results of which have been disappointing; a limited programme of approximately 4,200m of drilling was undertaken in 2009 to trace the ore zones to the southwest of the currently defined limit.

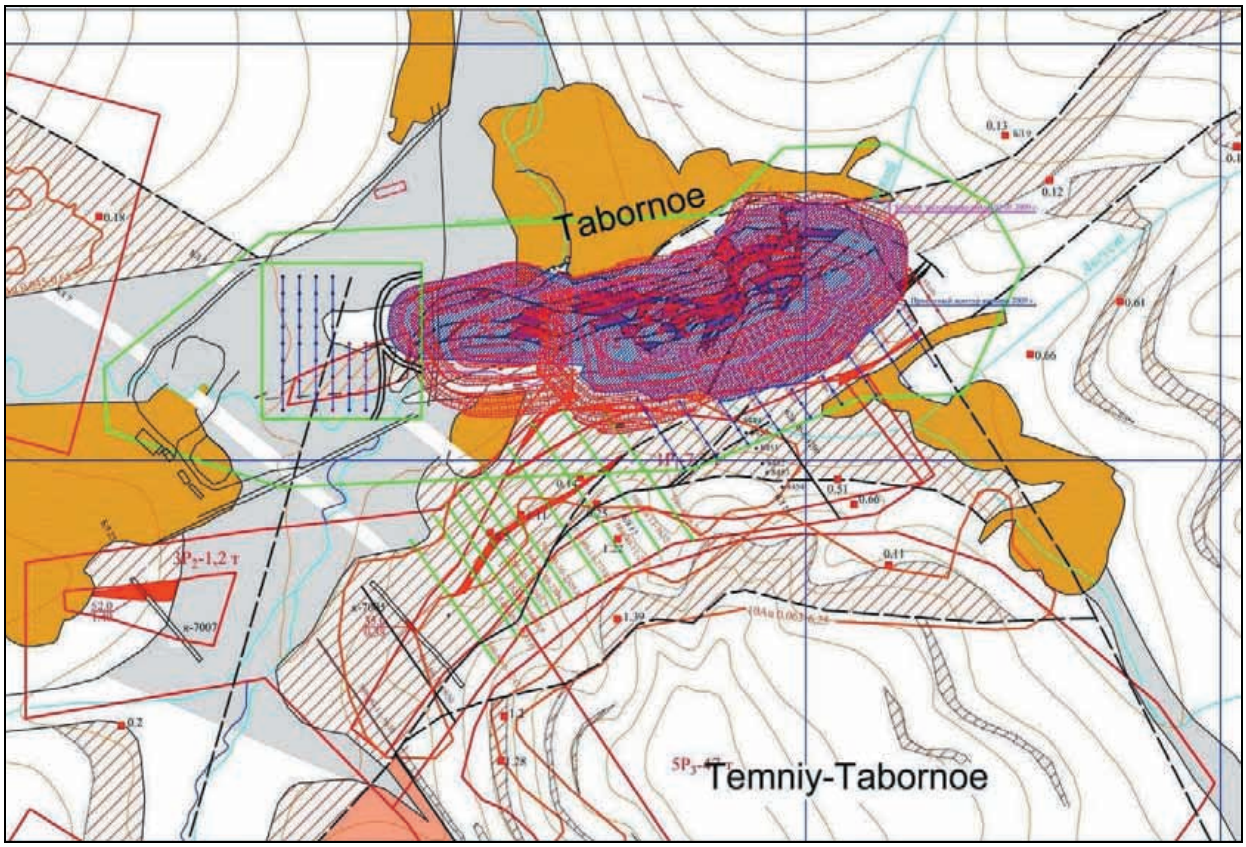


Figure 4.8: Temniy Tabornoe – Exploration Programme Drill Hole Layout (2008 Drill Profiles in Green; 2009 Drill Profiles in blue - Not to Scale Legend as for the Western Extension)

4.5.4.2 Visokiy

Visokiy is a small NE-SW trending structure that lies to the south west of Temniy Tabornoe. An exploration drilling programme was conducted during 2009-2010 to evaluate it as a satellite target to the main open pit at Tabornoe.

4.5.4.3 Gross

The Gross Mineral occurrence is situated approximately 4km to the east of Tabornoe in the upper reaches of the Gross stream. It comprises a zone of potassium feldspar metasomatite hosted by early Proterozoic sandstones of the Olonnokonskiy Formation. The zone was originally traced by geochemical and radiometric geophysical anomalies. A schematic plan of the geology is given in Figure 4.9 and a cross section through one of the 2008 drill profiles in the central section of the occurrence in Figure 4.10.

Four profiles were drilled at Gross in 2004-2005 by the previous owner, Arlan, with a total of 87 holes. Low grade Tabornoe-type gold mineralisation was intersected on Drill profile No.1, whereas adjusted profiles drilled at approximately 260m to the west and 800m to the east from profile No.1 showed limited economic mineralisation.

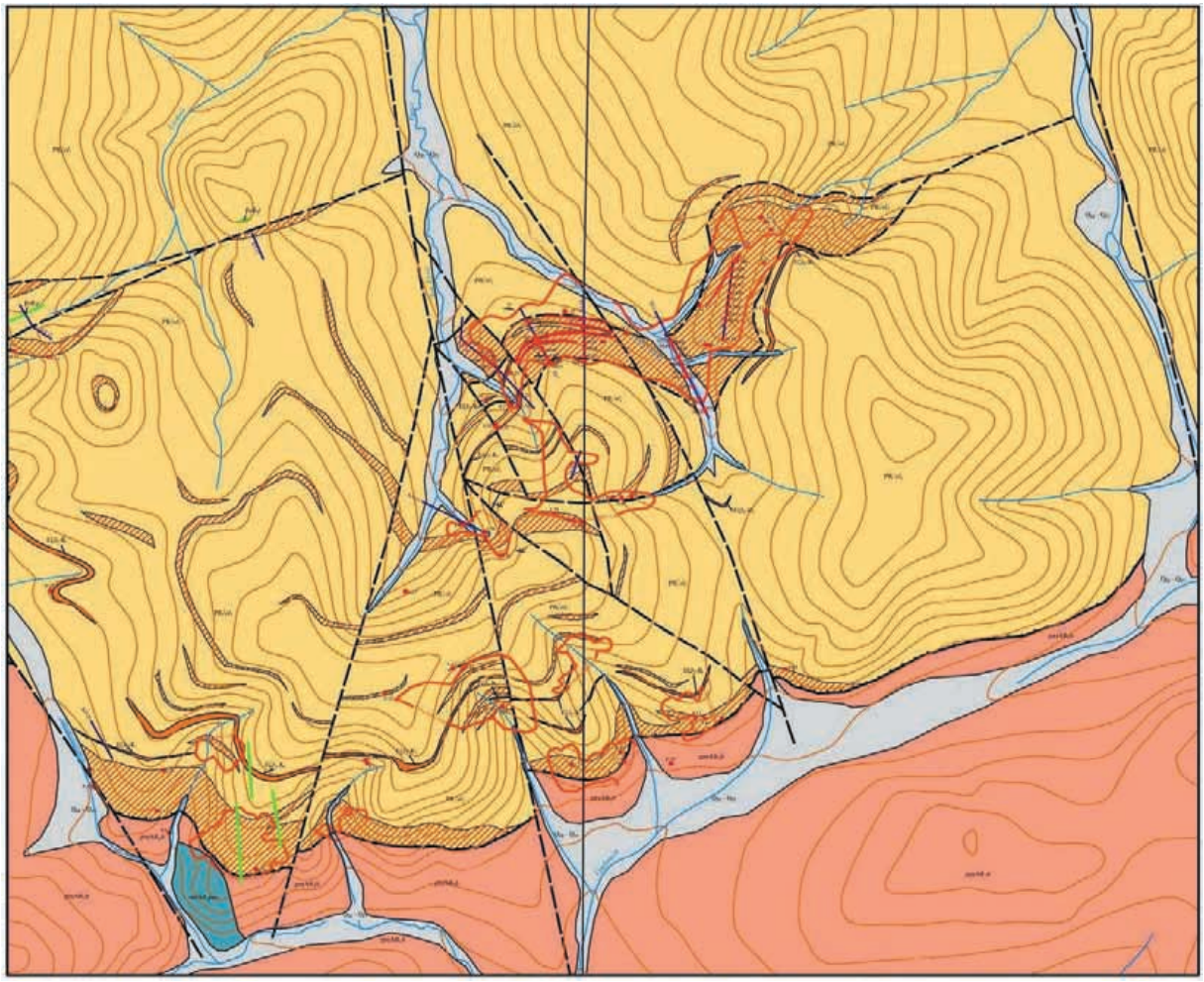


Figure 4.9: Geological Plan of the Gross Exploration Target
(Legend as for the Western Extension)

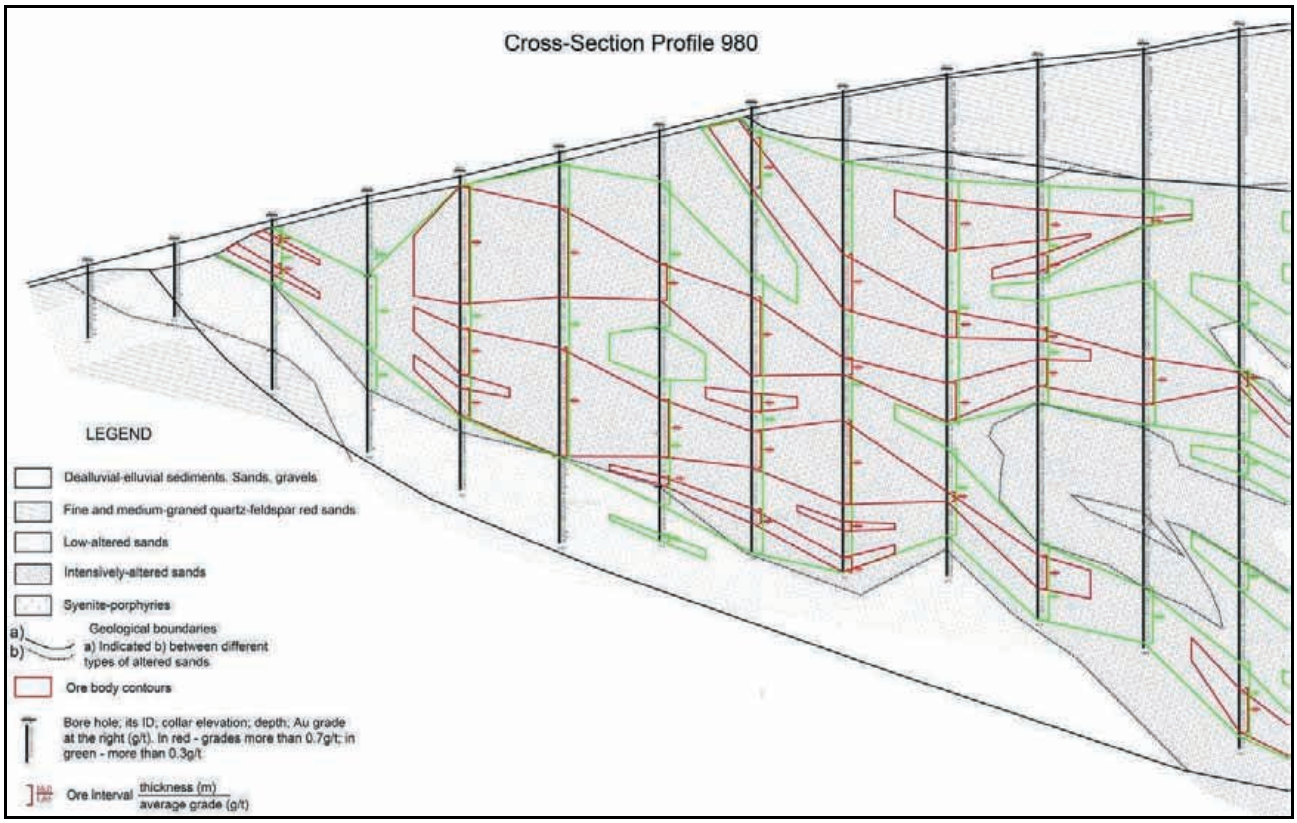


Figure 4.10: Typical Cross Section through the Centre of the Gross Deposit - Showing Potential Ore Envelopes

Further intensive drilling by Neryungri in 2008 identified four shallow southerly dipping (30°) consistent zones of gold mineralisation, containing comparable grades to that of the Tabornoe Ore Zone No.1, which could be traced both down dip (up to 440m) and along strike for a distance of at least 640m (see Figure 4.11).

Further drilling has taken place at Gross in 2009 and 2010, to explore both the eastern flank extension of the deposit and by extension of all profiles to the south of the deposit to identify whether or not the deposit forms a bowl shaped synform i.e. the dip of the deposit to the south changes to a northern dip, thereby potentially allowing exploitation of the whole of the hill and not just the northern flank. However, the shallow north flank of the hill and the gentle southerly dip of the ore zones into it, potentially will give a very low strip ratio, consequently making this target extremely attractive to open pit mining.

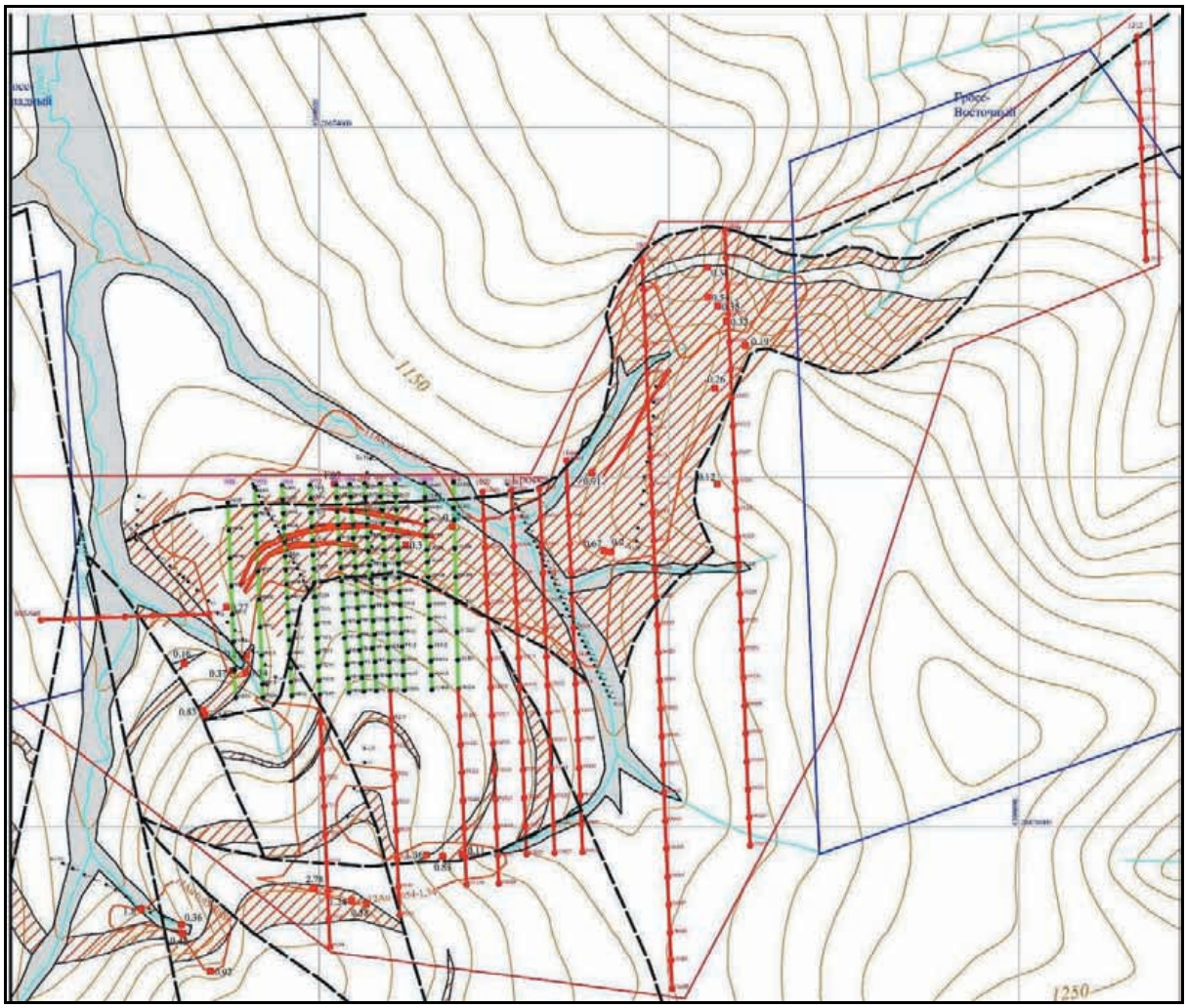


Figure 4.11: Gross – Exploration Programme Drill Hole Layout
(Green Profile Lines – Completed Drill Holes during 2008;
Red Profile Lines –Drill Holes Completed during 2009-2010)

4.5.5 Current Exploration Practices

4.5.5.1 Drilling

During 2009-2010, Neryungri undertook an extensive drilling programme on the west and SE side of the Tabornoe open pit in order to improve confidence in the open pit reserves for short and medium term planning.

WAI observed both rigs drilling HQ wireline core on site and overall was impressed by both the performance of the rigs and the drill practices of the two crews. Core recoveries averaged between 90-95%.

Reported borehole average core recovery for 2002-2005 drilling varied from 67 to 100% with an overall average of 86%. At some selected intervals, however, there was no recovery.

132 boreholes were drilled during 2009-2010 on a 20×20m and 40x40 grid spacing for a total of 17,500m. Of these x44 HQ diameter drill holes range in depth from 55 to 275m and average 94.7m.

4.5.5.2 Core Logging

It is understood that, historically, much of the exploration drilling for the project was undertaken during the short summer season. All core during this period was logged and sampled at the site of the drill rig, thereby negating the need for a core logging room. The core from the previous owner, Arlan is now held in a storage facility.

In 2008, Neryungri built an all purpose exploration building to house both a comprehensive core logging room, geological office and sample preparation facility.

The core logging room is substantial both in size and the facilities that it contains and benefits from having good lighting and heating, as well as well designed and appropriate core logging benches

> ***WAI Comment:*** *WAI has inspected the logging facilities and reviewed the core logging procedures and protocols established for describing lithological types, structure, mineralisation and alteration and have found them all to be of a very high standard that would equal any established western best practices.*

4.5.5.3 Core Cutting

All exploration core to be assayed is split into two, utilising diamond saws. The remaining half of the core is returned to the core box and thereafter transferred to the adjacent core storage shed.

4.5.6 Sample Preparation

Samples received at the laboratory are first dried in ovens at 95-105°C, and then crushed using a Rocklabs Boyd – RSD (Rotating Sample Divider) combination jaw crusher and splitter to a nominal top size of 2mm. The crushed material is automatically split to produce a sample of approximately 1kg for further size reduction. The "coarse rejects" are retained and stored in plastic bags in the core storage shed.

The ~1kg sample is pulverised and then divided, with 250g, duly labelled and stored in transportation boxes to be sent to SGS Vostok Limited based in Chita. The remaining pulp is stored in plastic bags in the core storage shed.

> ***WAI Comment:*** *the condition of the sample preparation facility was excellent, in terms of the state of the equipment and the cleanliness of the facility, as well as housekeeping and safety (PPE – personal protection equipment) procedures.*

4.5.7 Storage of Assay Sample Pulps and Rejects

After completion of core sample preparation, the coarse "reject" portion of a given sample is retained, together with the pulp reject from the assay laboratory (material of <75microns). This reference material is stored in the core storage facility.

4.5.8 Core Storage Facility

A purpose built wooden core storage facility has been constructed to house the cut core and sample reference material. The facility is well constructed and laid-out.

4.5.9 Exploration Assays

All prepared sample pulps (250g) from the Neryungri exploration drill programmes at Tabornoe are sent to SGS Vostok Limited located in the regional centre in Chita. Samples are fire assayed for gold.

This laboratory is internationally accredited and has a worldwide reputation for high quality analysis. This facility was no different.

> ***WAI Comment:*** *WAI has visited these laboratories and found them to be run to the highest of standards.*

4.5.10 Exploration Management Structure

The exploration management team located on-site at Tabornoe appears to be very competent in relation to international equivalents.

> ***WAI Comment:*** *A clear cut separation has been made between Exploration and Mine production - Grade Control Departments. WAI considers that this partition of work duties to be commendable. Both departments appear to be adequately staffed (by employees who are suitably trained in each of these two quite separate disciplines) on-site to cope with the workload.*
>
> *The exploration department at Tabornoe has complete control of assay sample preparation. Under the current regime, direct responsibility for the preparation of samples lies in the hands of the Chief Geologist, who can control the priority of sample preparation and be in a position to insert additional samples into the system in order to validate the assay samples, without the prior knowledge of the external laboratory (SGS), thereby improving the QA/QC protocols. This is an important requirement under the guidelines of the JORC Code (2004).*

4.6 Mine Production – Grade Control

4.6.1 Grade Control

The mine uses a combination of blast hole sampling in the hard ores, which appears to work efficiently, and rip line sampling in the softer ores.

4.6.1.1 Blast Hole Sampling

One composite sample is taken from each blasthole (drilled on either a 3.3 x 3.3m or 3.7 x 3.7m spacing to a depth of 5.0m) and from the assay results an ROM 0.7g/t Au contour plotted on the bench plans. A contour is also plotted for a grade range of 0.4-0.7g/t Au which is mined separately and delivered to a low grade out-of-pit stockpile. Both ROM (normal grade) and low grade envelopes are taped out on the bench by the surveyors in readiness for excavation.

> ***WAI Comment:*** *the current procedures appear adequate to control ROM grade.*

4.6.1.2 Rip Line Sampling

Samples are collected along each rip line by hand using a shovel on approximately 6 x 6m centres from material along a trench for approximately 1.0m, centred around a point. These sample points are then surveyed at the same time the samples are collected.

The large variations in both rock hardness, as well as the high variability of gold grades means that these “spot” samples are unlikely to be as representative as those from blast sampling (i.e. two point samples taken close together are likely to give quite different values). Blast hole drilling and sampling is not possible in these softer rock types and it is accepted that point sampling, although far from ideal, is the only realistic alternative to blast hole drilling.

4.6.1.3 Reconciliation between Reserves Ore Outlines and Rip line Point Samples

Ore reserves have been delineated for Tabornoe based on diamond drilling on a spacing of 20x20m. Conventional polygonal techniques have been employed using plans and cross sections to identify the reserve envelopes.

A comparison between the results of definition drilling and blasthole/rip line sampling is illustrated for the 1,060m elevation bench plan in Figure 4.12 below.

The original geological interpretation for reserve purposes is delineated by red outlines whilst normal ore is shown hatched in squares, and low grade material by inclined lines. Considerable variation in the ore/waste boundaries is evident.

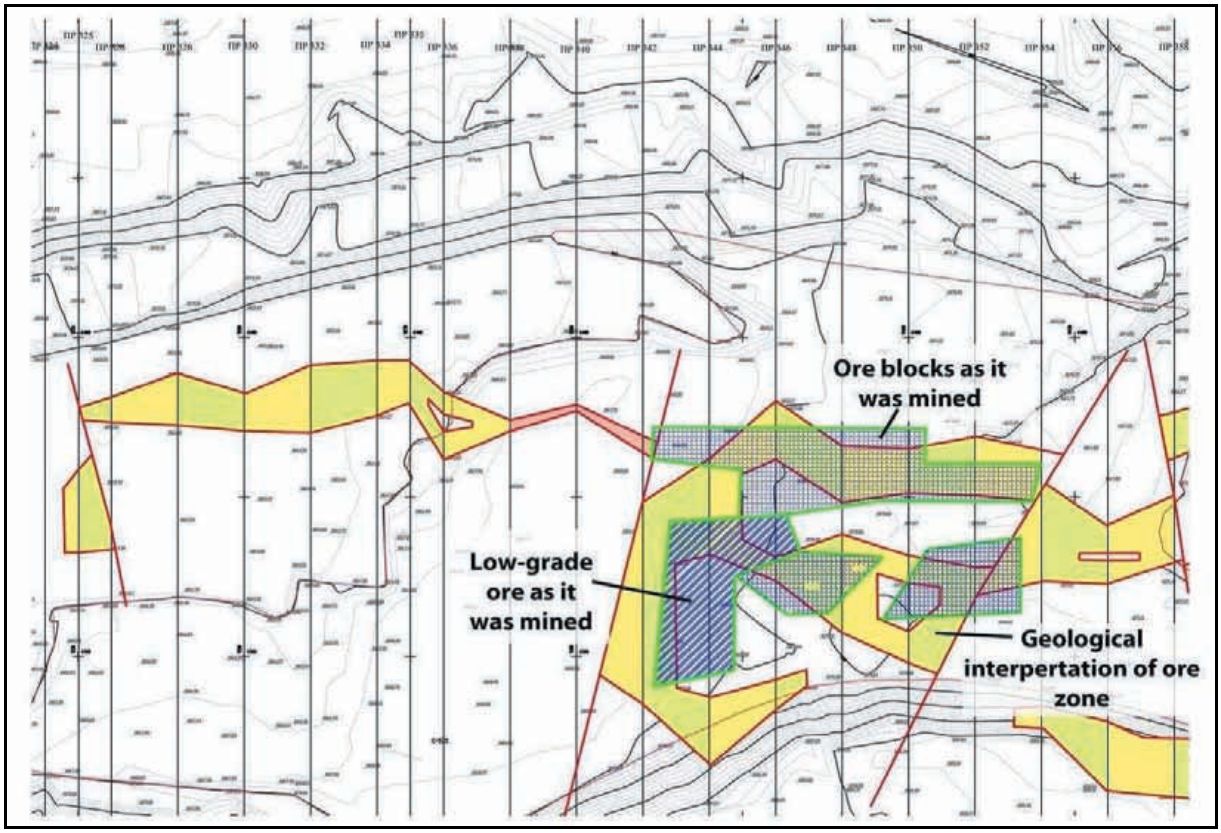


Figure 4.12: Comparison of Geological Interpretation of Ore Reserve Envelopes (based on 20x20m drillholes) vs Actual Ore Outlines Applied during Mining – based on rip sampling (Elevation 1,060m)

4.6.2 Sample Preparation and Assay of Rip Line and Blast Hole Samples

All production samples, including rip and blast hole drilling are prepared and assayed in separate facilities to those for the exploration undertaken at Tabornoe.

WAI has inspected the facilities and found them to be adequate, but considers that they would greatly benefit from some modernisation. Modernisation is likely to improve assay quality, reliability and consequently turnaround time.

In early 2010, a server was installed on the mine site with a SQL server client; with a replication to the central office in Moscow. WAI's block-model of the Tabornoe site has shown a good quality of estimation and was approved by the mining site geologists on the basis of 2010 exploration and mining. Currently, both mine site

engineers and geologists constantly update the block-model, hence facilitating the update of both annual and monthly mining plans. WAI is confident that shortly, mine planning will be done on a weekly basis.

4.7 Mineral Resource Estimate

4.7.1 Historical Estimates

4.7.1.1 July 2000 Estimate

The first estimates for Tabornoe were reviewed and approved in July 2000. A cut-off grade (COG) of 0.7g/t Au was used to delineate economic resources whilst 0.4g/t Au COG was employed to identify low grade resources.

A bulk density of 2.41t/m^3 was used in this resource estimation (Table 4.8).

Table 4.8: Tabornoe Resources as at July 2000

Type and category	On-Balance Resources (0.7g/t Au COG)			Low grade (Off Balance) Resources (0.4g/t Au COG)		
	Ore (kt)	Gold (kg)	Average Grade (g/t Au)	Ore (kt)	Gold (kg)	Average Grade (g/t Au)
C_2 Category	17,378	26,935	1.55	15,581	7,946	0.51
P_1 Category	-	-	-	3,039	1,984	0.65

Note that the "Soviet-style" reserves are reported not taking into account mining losses and mining dilution.

4.7.1.2 September 2002 Estimate

In order to evaluate ore reserves for the test mining operation, in-fill and confirmation drilling were undertaken by Neryungri Metallic in 2002, inside the proposed test pit shell.

Later in 2002, additional bulk density tests were performed and a more accurate average density estimate of 2.16t/m^3 was obtained. The test pit reserve estimate was updated using this figure:

- 513.6kt containing 1016.9kg of gold at average grade of 1.98g/t Au (economic grade- on-balance); and
- 39.8kt containing 20kg of gold at average grade of 0.50g/t Au (low grade-off balance).

Despite the new lower density figure, the overall resources for the deposit remained relatively unchanged.

4.7.1.3 January 2006 Estimate

Mineral Resources inside a final pit shell were estimated in 2006 and subsequently approved by GKZ (Russian State Commission of Resources) as a part of resource estimation for the deposit. The resources reported within this pit boundary were then converted into reserves after mining recovery and dilution were applied. The ultimate pit shell was designed using a cut-off grade of 0.7g/t Au.

Table 4.9 below summarises the Mineral Resource estimate inside and outside the designed ultimate pit boundary.

Table 4.9: Tabornoe Mineral Resource Statement (GKZ Approved 2006)			
Resource category	**Tonnage (kt)**	**Au Grade (g/t)**	**Au (kg)**
C_1 (within pit boundary)	5,279.30	1.65	8,688.5
C_2 (within pit boundary)	4,015.40	1.52	6,086.7
Total C_1+C_2 (within pit boundary)	**9,294.70**	**1.59**	**14,775.2**
C_1 (out of pit boundary)	823.4	1.43	1,179.3
C_2 (out of pit boundary)	3,057.00	1.45	4,428.1
Total C_1+C_2 (out of pit boundary)	**3,880.40**	**1.45**	**5,607.4**

Table 4.10 below provides the revised resource statement for 01 January 2008 based on depletion of the resource estimation from 2006.

Table 4.10: Tabornoe Mineral Resources as to 01 January 2008 (Depleted from GKZ Approved Resource Statement 2006)			
Resource Category	**Tonnage (kt)**	**Au grade (g/t)**	**Au (kg)**
C_1 (within pit boundary)	3,822	1.61	6138.7
C_2 (within pit boundary)	3,654	1.51	5507.2
Total C_1+C_2 (within pit boundary)	**7,476**	**1.56**	**11646.0**
C_1 (out of pit boundary)	783	1.42	1111.5
C_2 (out of pit boundary)	3,057	1.45	4428.1
Total C_1+C_2 (out of pit boundary)	**3,840**	**1.44**	**5539.6**

The 'polygonal method' was employed for resource and reserve estimation. A typical cross-section through profile 392 (central part of the deposit) is shown in Figure 4.13. The polygonal method, although approved by the Russian state authorities, does not reflect the orebody contours in detail. This has subsequently been proven by comparison of the original GKZ approved block resources with the actual ore production figures.

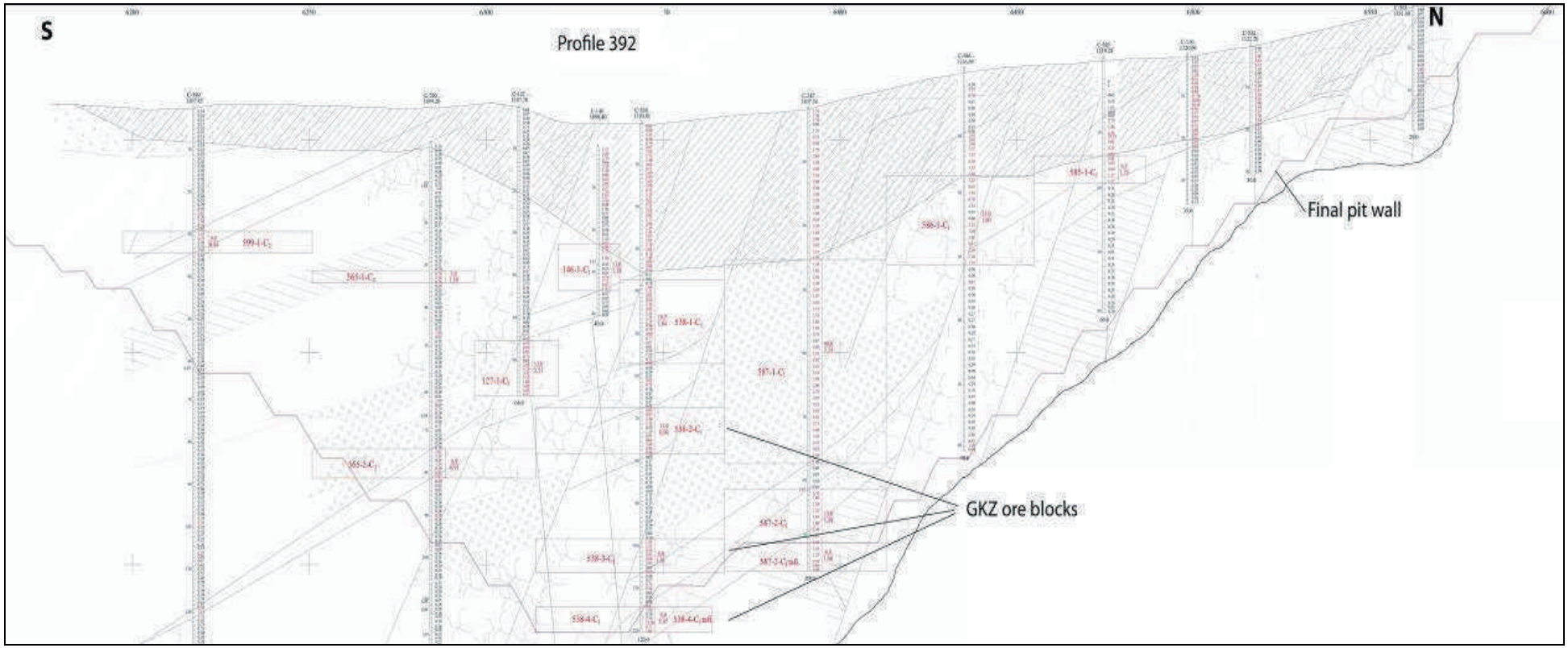


Figure 4.13: Cross-Section through Profile 392 – GKZ Resource Estimation 2006

4.7.2 Current Resource Estimate for Tabornoe and Temny Tabornoe (WAI 01 November 2010)

WAI was commissioned by Nord Gold to prepare a Mineral Resource estimate in accordance with the guidelines of the JORC Code (2004) for the Tabornoe and Temny Tabornoe. The work completed during this project is summarised below.

4.7.2.1 Topography

The pit topography used in this study is understood to have been based on a mine survey carried out on the 01 November 2010 and is shown in Figure 4.14.

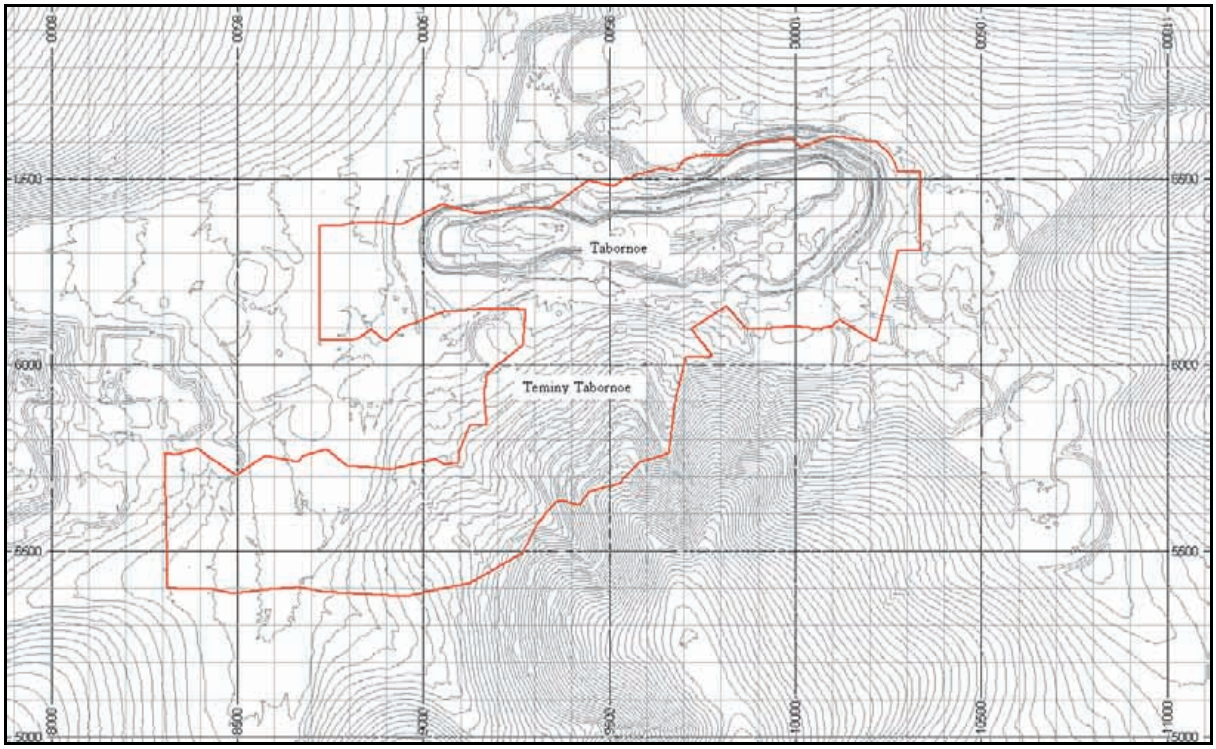


Figure 4.14: 5m Contour Plan of Tabornoe and Temny Tabornoe Area

4.7.2.2 Sample Database

The data collected includes diamond-drillhole data, rip-line grab sample data and blasthole data. A summary of the sample database is shown in Table 4.11.

Table 4.11: Sample Data Summary

	No. of Holes	No of Samples	Total Length (m)
Diamond Drillholes	1,246	109,780	111,896
Blastholes	2,370	2,370	10,699
Rip-line Grab Samples	-	21,734	-

4.7.2.3 Mineralised Zone Interpretation

Although it is thought that the mineralised parts of the deposit are strongly influenced by faulting, there is insufficient evidence from surface exposures and diamond drillholes to allow identification and modelling of individual fault surfaces. An overall analysis of gold grades does not exhibit any clear 'natural' gold grade cut-offs. However, an analysis of contiguous mineralised lengths, as well as considering potential grades that may be amenable to heap leaching, have led to the use of 0.3g/t grade cut-off for mineralised zone modelling

purposes. A wireframe defining the current drillbase/mineralisation limit was constructed and is shown in Figure 4.15. The mineralised zones were extrapolated using a dynamic-anisotropy approach, with orientation data being derived from the previous interpretations carried out by on-site geologists.

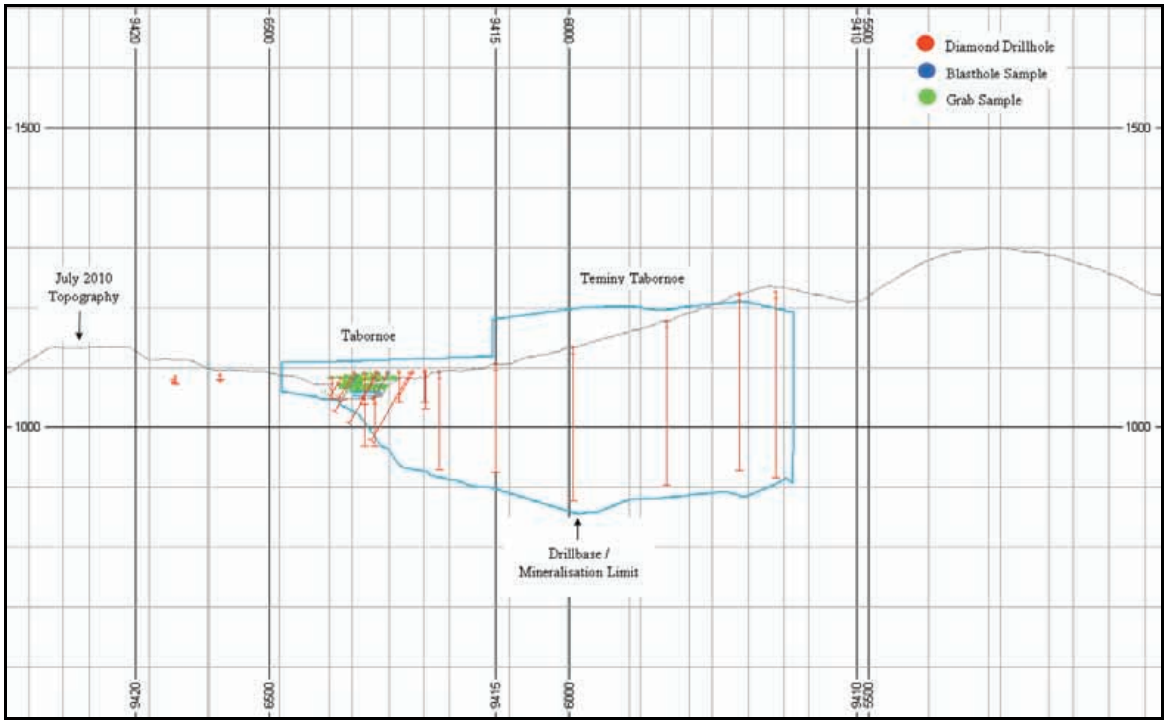


Figure 4.15: N-S Section at 9,400Em Showing Drillbase/Mineralisation Limit

4.7.2.4 Geostatistics

The sample data indicates a single log-normal population of gold grades. A geostatistical study of 5m diamond-drillhole composites indicated variogram ranges of approximately 45m along-strike, 60m down-dip and 20m cross-strike. These ranges were also supported by analyses of the available grab sample and blasthole data. A modelled variogram is shown in Figure 4.16.

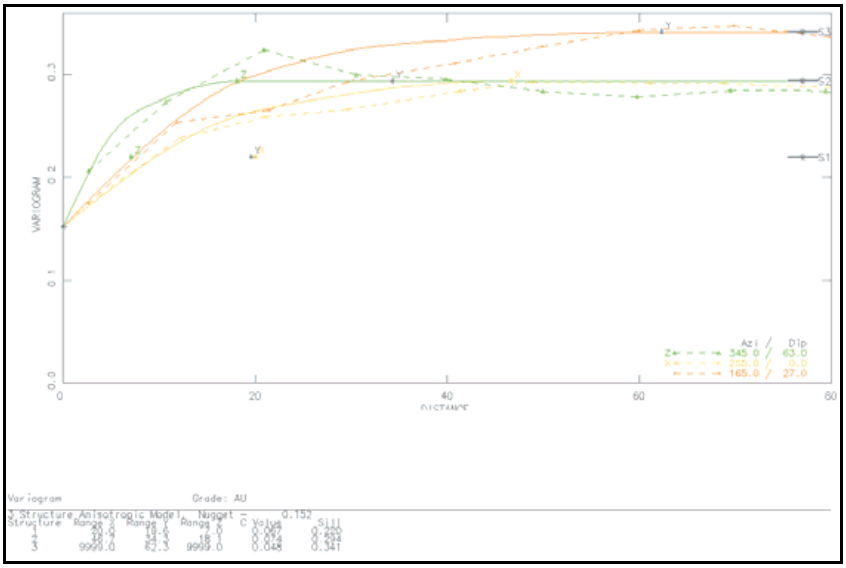


Figure 4.16: Modelled Semi Variogram
(Based on Diamond Drillhole Data)

4.7.2.5 Block Model

A volumetric block model was set-up with a 20m x 20m x 10m block size. Within the mineralized zones, sub-blocks were generated measuring 5m x 5m x 5m as shown in Figure 4.17. Topographical data were used so that model blocks were generated up to the level of the original pre-mining topography, but with the mined-out parts separately coded. A global density value of 2.4t/m^3 was used.

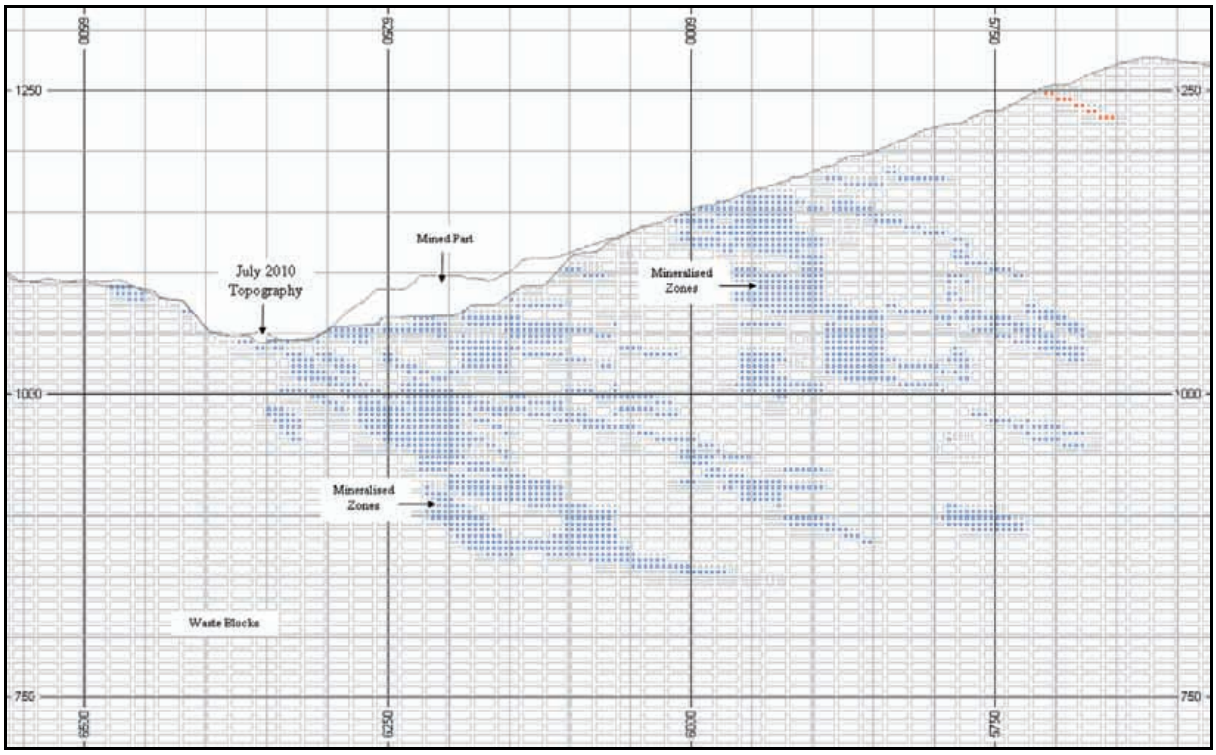


Figure 4.17: S-N Section through Volumetric Block Model at 9,500mE

4.7.2.6 Grade Estimation

Gold grades were estimated within the mineralized zones using ordinary kriging. Alternative grades were also interpolated using inverse distance weighting and nearest neighbour estimations for validation purposes Grades stemming from diamond drillholes, grab samples and blasthole data were utilised.

4.7.2.7 Model Validation

A model validation process included the examination of block model versus composites, and the building up of a model grade profile, to compare average grades on vertical slices, as derived from the composites directly, as well as from model grades derived from different interpolation methods.

4.7.3 Resource Classification

Criteria for defining resource categories were also derived from the geostatistical studies. Key drillhole spacings for the allocation of *Measured* resources were 20m x 25m (along-strike x down-dip) and 45m x 60m for *Indicated* resources. The extent of *Inferred* resources was limited to 60m down-dip, from any individual diamond drillhole intersections. A plan projection showing the resource classification is shown in Figure 4.18.

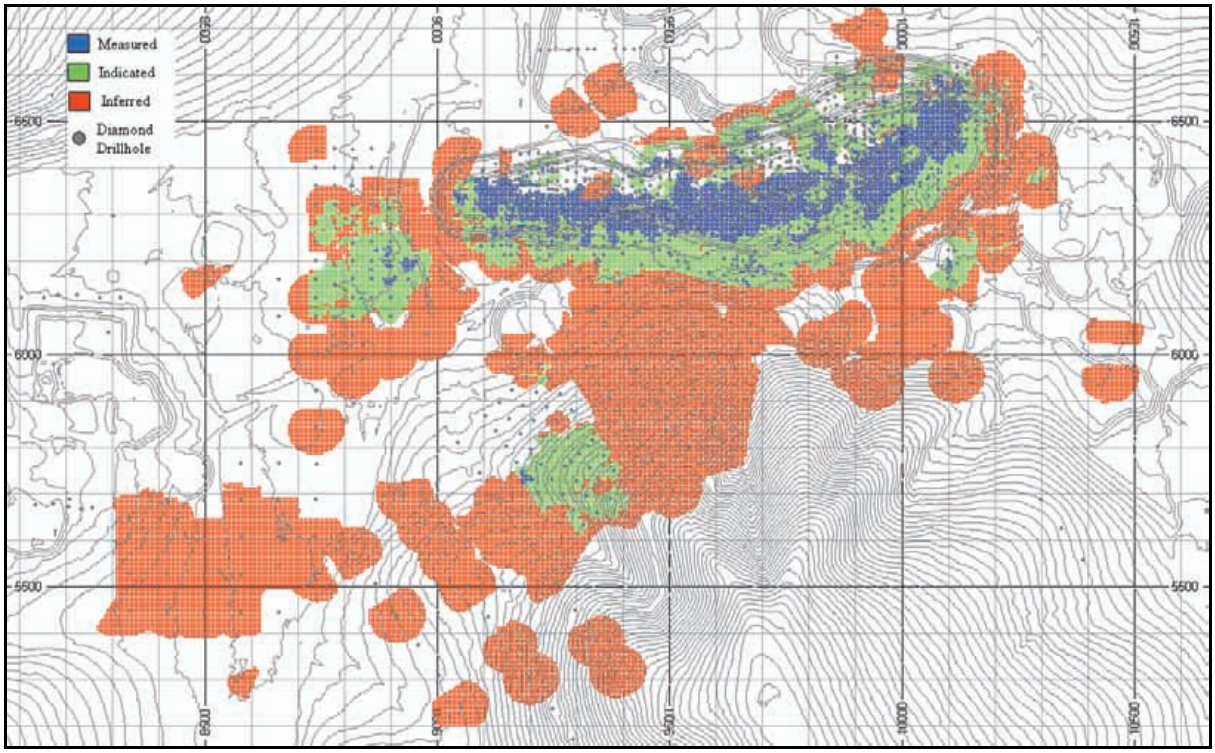


Figure 4.18: Plan View Showing Resource Classification

4.7.3.1 Resource Evaluation

The final block model was used as the basis for resource evaluation. A summary of results of the evaluation of the in-situ resources are shown in Table 4.12 below, for three different cut-off grade levels: 0.3g/t, 0.5g/t and 0.7g/t Au.

Table 4.12: Tabornoe and Temny Tabornoe Resource Estimate (WAI, 1st November 2010) (in accordance with the guidelines of the JORC Code (2004)

Ore Type			Oxide	Oxide	Oxide
Cut Off Grade (g/t)			0.3	0.5	0.7
Measured	Tonnage (kt)		5,079	4,475	3,197
	Au (g/t)		0.98	1.05	1.23
	Metal	kg	4,966	4,701	3,937
		oz	159,673	151,153	126,562
Indicated	Tonnage (kt)		25,097	19,576	12,048
	Au (g/t)		0.79	0.88	1.07
	Metal	kg	19,757	17,322	12,852
		oz	635,212	556,906	413,207
Measured + Indicated	Tonnage (kt)		30,177	24,051	15,245
	Au (g/t)		0.82	0.92	1.10
	Metal	kg	24,724	22,023	16,789
		oz	794,885	708,059	539,769
Inferred	Tonnage (kt)		73,315	39,590	20,375
	Au (g/t)		0.63	0.82	1.05
	Metal	kg	46,453	32,439	21,317
		oz	1,493,496	1,042,947	685,362

Notes:
1. Mineral Resources are not reserves until they have demonstrated economic viability based on a Feasibility study or pre-feasibility study.
2. Mineral Resources are reported inclusive of any reserves.
3. The contained Au represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

4.8 Current Resource Estimate for Visokiy (WAI 01 July 2010)

WAI was commissioned by the Client to prepare a Mineral Resource estimate in accordance with the guidelines of the JORC Code (2004) for the Visokiy prospect. The work completed during this project is summarised below.

4.8.1 Topography

Topography has been provided by the Client as wireframes.

The topography of the Visokiy deposit is based on a 1m contour interval with the survey understood to have been carried out in October 2009 and is shown in Figure 4.19. It is understood that no mining operations have been carried out at Visokiy since this date. A digital terrain model (DTM) wireframe was constructed using the survey data points.

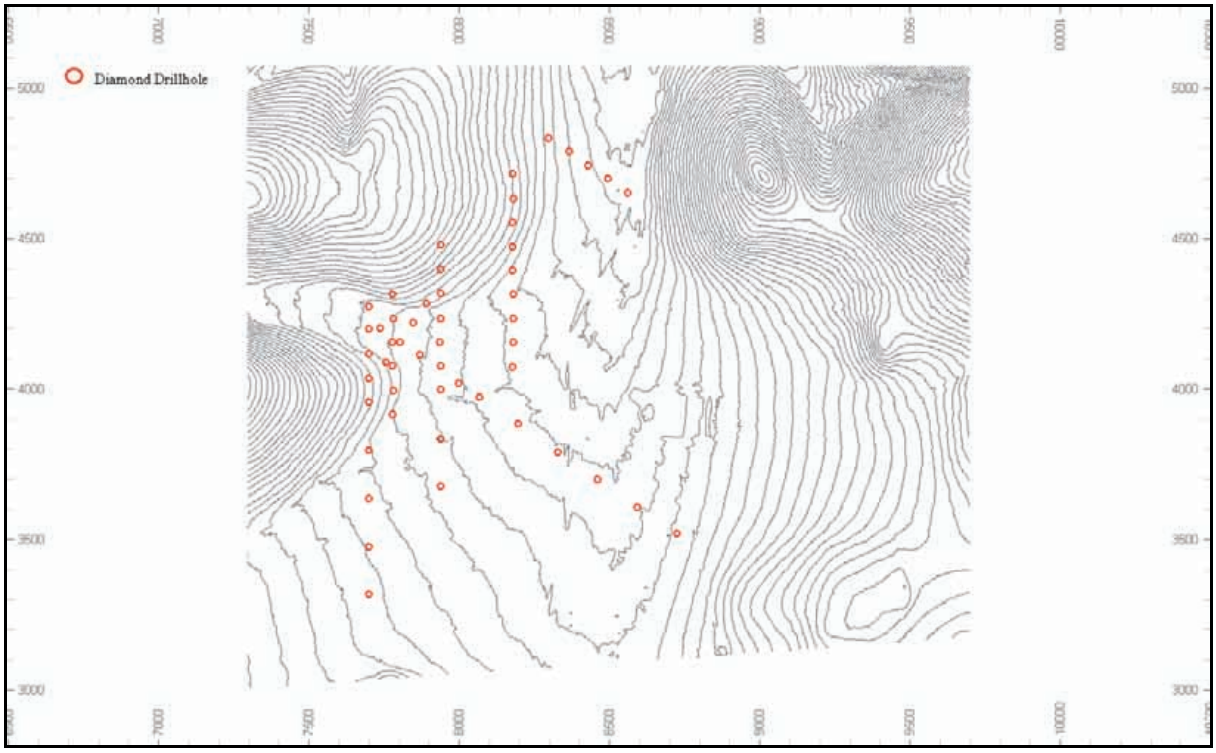


Figure 4.19: 5m Contour Plan of Visokiy Area

4.8.2 Database Compilation

4.8.2.1 Drilling

Diamond drilling (wireline) has been carried out at Visokiy during 2009 and 2010. The drillhole data has been provided in Excel format by the Client.

An isometric view of the drillholes and topography is shown in Figure 4.20 below.

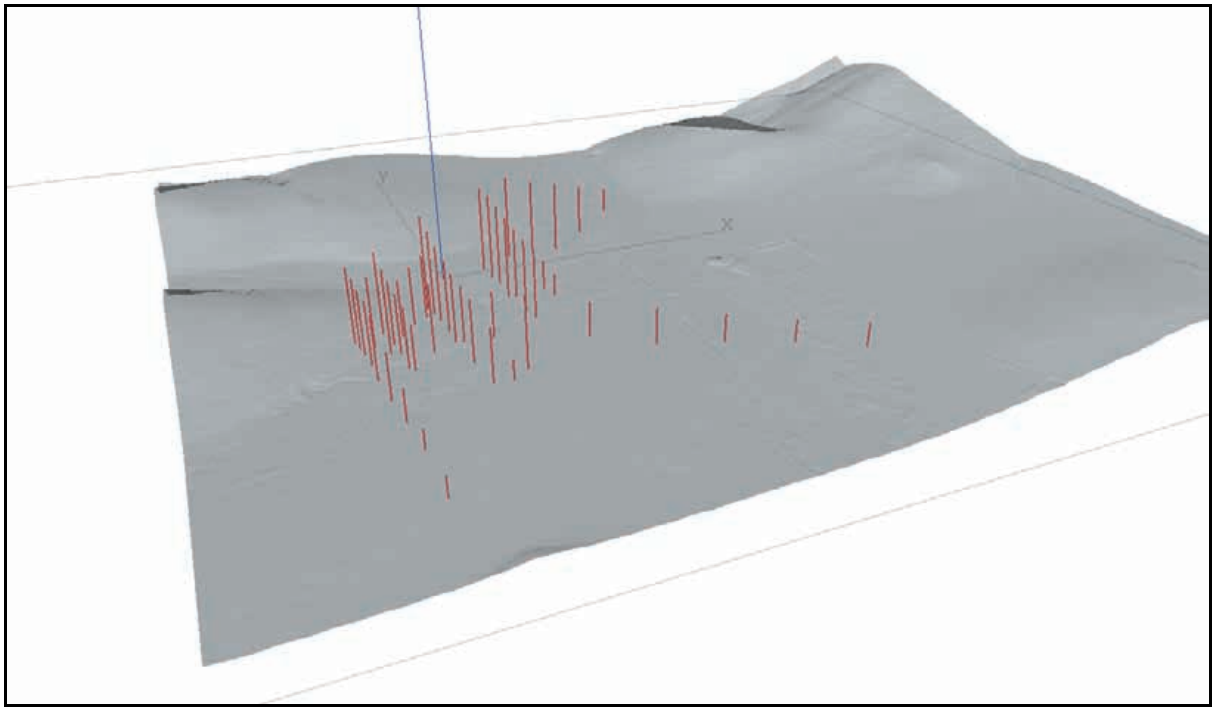


Figure 4.20: Ismoetric View of Diamond Drillholes at Visokiy

All of the diamond drillhole data collated for the current project are summarised below in Table 4.13.

Table 4.13: Sample Data Summary

	No. of Diamond Drillholes	No of Samples	Total Length
Diamond Drillholes	51	8,431	8,439

The drillhole database contains information on Au grade, sample number, section, year, collar, survey and lithology information. Ag grades are also present in the drillhole database however, the Ag grades are considered to be particularly low, therefore no further modelling with respect to Ag has been carried out.

Drilling has been based on an irregular grid pattern aligned at approximately north-south across the deposit. The drill spacing varies between 80m and 160m and all diamond drillholes were designed as vertical holes, and have been assumed to be as such.

4.8.3 Mineralised Zone Interpretation

An analysis of the Visokiy dataset was carried out to attempt to define a natural cut off grade upon which to define the mineralisation.

4.8.4 Wireframe Modelling

Given that natural breaks are evident in the log probability plots at 0.3g/t Au and that this is also considered to be an appropriate grade level in terms of what might also qualify as ore for heap leach purposes, a cut off grade of 0.3g/t Au has been chosen for subsequent interpretation and modelling work. Two separate ore zones were modelled within the Visokiy deposit and strike east-west for up to 270m and dip between 15° and 40° to the south for up to 250m. A plunge of approximately 25° to the east has also been modelled. It is possible that faulting is also present in this area however; the location of any faults could not be identified from the limited amount of drilling. In general the mineralised zones were not extrapolated more than 50m beyond any final drillhole intersection. The mineralised zone wireframes have been tested for open edges,

shared surfaces and crossovers and verified correctly. A plan and section of the interpreted mineralised zones is shown in Figure 4.21.

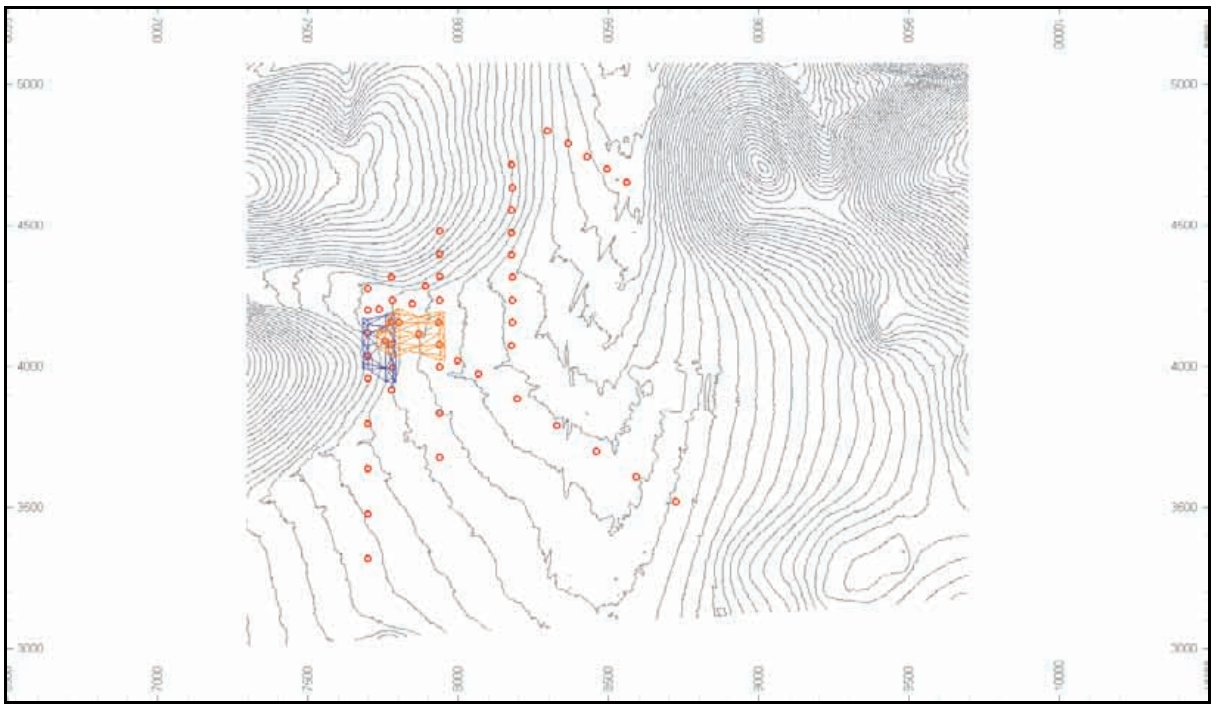


Figure 4.21: Plan of Interpreted Mineralised Zones

4.8.5 Data Processing Summary

The statistical analysis of the Visokiy sample database is summarised below:

- The population distribution for Au for each ore zone exhibit approximately log-normal distributions with very similar distributions shown for each ore zone;
- Top cutting has been carried out by ore zone. Because of the very few number of samples in each ore zone the top-cuts applied have resulted in a reduction in the mean grade from 1.24g/t Au to 1.07g/t Au for ore zone 1 and 0.87g/t Au to 0.68g/t Au for ore zone 2; and
- A 5m composite interval has been applied to standardise sample length.

4.8.6 Variography

Variography was not undertaken at the Visokiy deposit due to the small number of available samples. Additional drilling will need to be carried out before any meaningful variography can be generated.

4.8.7 Block Modelling

The block model parameters are shown in Table 4.14. The block model used a parent cell size of 5m x 5m x 5m (along strike x across strike x vertical) with sub cell splitting down to a minimum cell size of 2.5m x 2.5m x 2.5m. The model is not rotated. A summary of the block model codes is shown in Table 4.15. A sectional view of the block model coded by ore zone is shown in Figure 4.22.

Table 4.14: Block Model Prototype		
Property	**Direction**	**Metres (m)**
Model Origin	X	7,400
	Y	3,700
	Z	900
Parent Cell Size	X	5
	Y	5
	Z	5
No. of Cells	X	160
	Y	120
	Z	100

Table 4.15: Summary of Block Model Codes		
Code	**Values**	**Description**
ORE	1 to 2	Ore zone Number (absent is non mineralised)
AU_ID	-	Au value from Inverse Distance estimate
AU_NN	-	Au value from Nearset Neighbour estimate
NUM	0 to 15	Number of composites used in block estimate
SVOL	1 to 3	Search volume number
DIST	-	Distance to nearest composite (as ratio of 1st ellipse)
CLASS	1-3	Resource Class (1) Measured (2) Indicated (3) Inferred (0) Not Classified

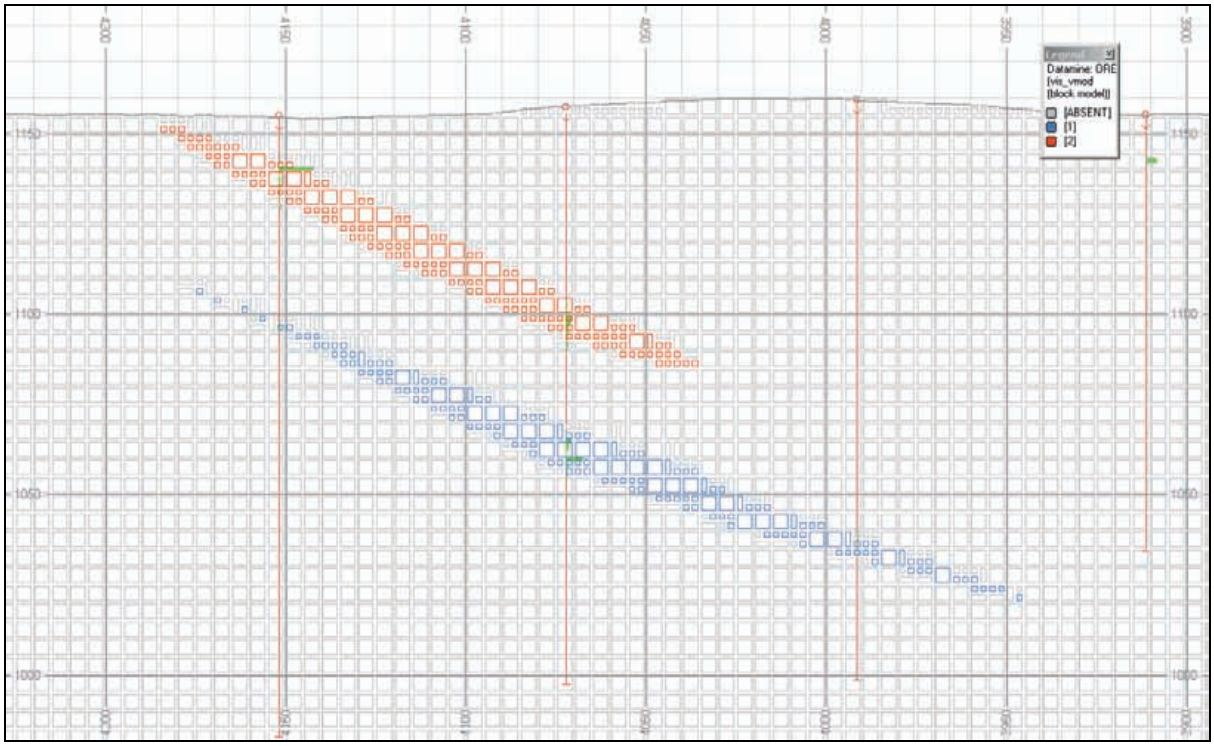


Figure 4.22: Example Section Showing Block Model Ore Zone Codes

4.8.8 Density

A constant global density of 2.4t/m^3 has been applied to the model. This figure is the same as the density determinations carried out at Tabornoe. It is recommended that separate density determinations be carried out at Visokiy, however, until these results are available WAI consider using a global density of 2.4t/m^3 to be appropriate.

4.8.9 Grade Estimation

Grade estimation was carried out using Inverse Power of Distance Squared (IDW^2). Nearest Neighbour (NN) was also used for comparative purposes. The estimation was performed only on mineralised material defined by each ore zone. Grade estimation using kriging was not carried out as no suitable variogram parameters were derived from the geostatistical studies at Visokiy.

4.8.10 Model 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:

- A visual assessment of grade;
- Global statistical grade validation; and
- SWATH plot (model grade profile) analysis.

Globally no indications of significant over or under estimation are apparent in the model nor were any obvious interpolation issues identified. From the perspective of conformance of the average model grade to the input data, WAI considers the model to be a satisfactory representation of the drillhole data used and an indication that the grade interpolation has performed as expected. In terms of conformance to the drillhole composite data WAI consider the ID interpolation method to most closely represent the drillhole data in both the cross validation and swath analysis plots. The resource estimate is therefore based upon the ID grade estimation.

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

4.8.11 Resource Classification

The criteria used for defining the resource categories at Visokiy are based on the geostatistical studies carried out at Tabornoe. The key drillhole spacings for the allocation of resources at Tabornoe can be summarised as follows:

- ***Measured*** resources - at least 20m x 25m (along-strike x down-dip) drilling grid;
- ***Indicated*** resources - at least 20m x 25m (along-strike x down-dip) drilling grid; and
- ***Inferred*** resources – Maximum of 60m from any individual drillhole intersection.

Based on these criteria and given the limited amount of drilling carried out at Visokiy the deposit has been classified as an *Inferred* resource. An example plan projection demonstrating the resultant resource class categories which have been set into the block model is shown in Figure 4.23. An example cross-section of the resource classification is shown in Figure 4.24.

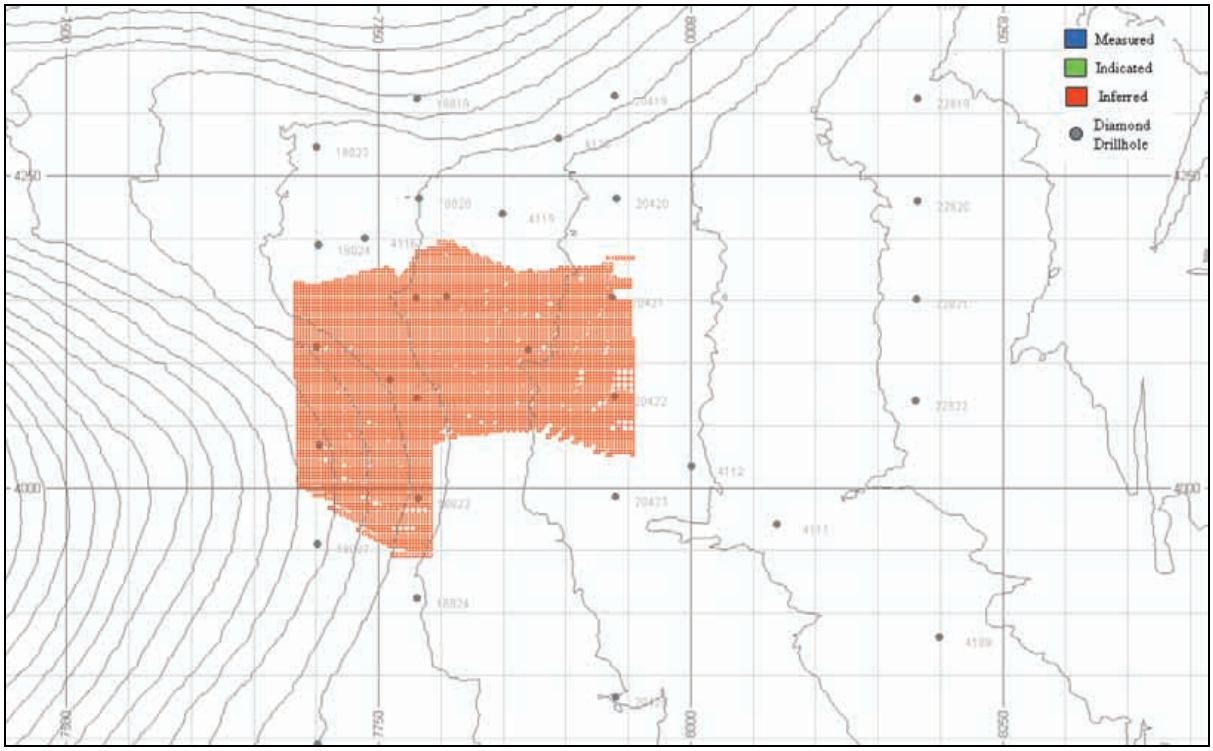


Figure 4.23: Example Plan Projection of Block Model Resource Classification

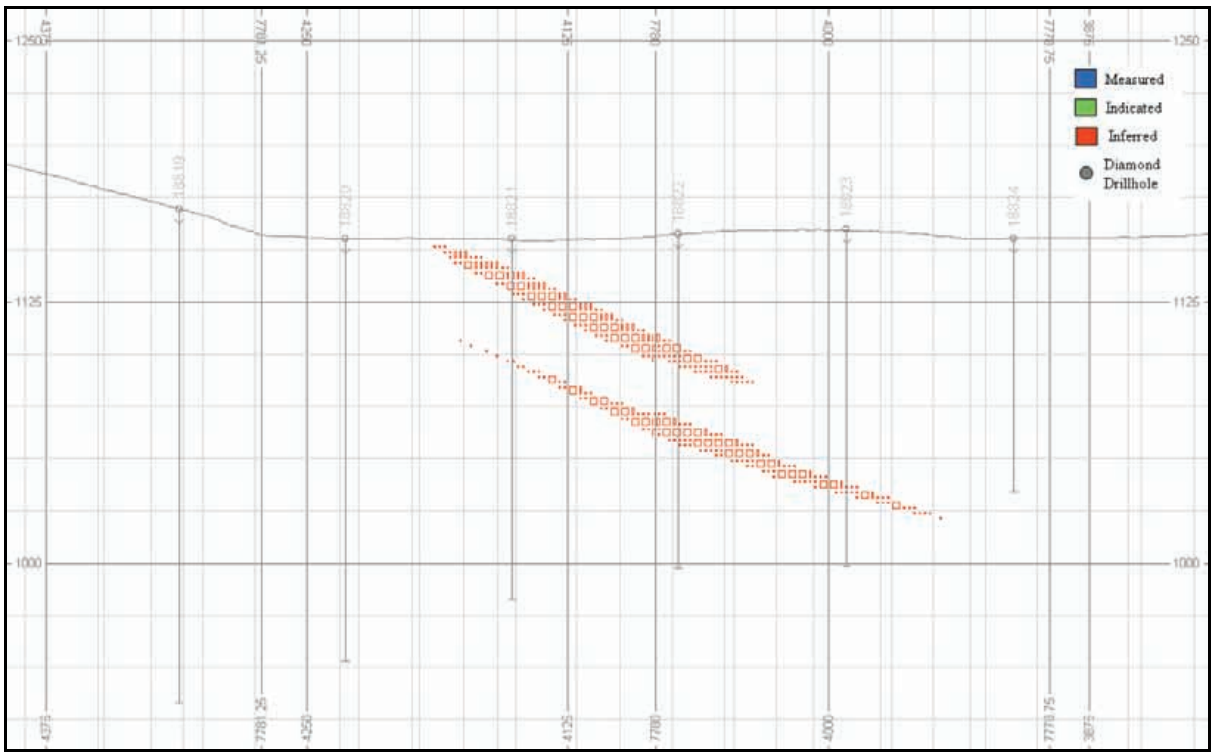


Figure 4.24: Section 7780mE Showing Resource Classification

4.8.12 Resource Estimate

The resource classification for the Visokiy deposit is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2004)].

The final block model was used as the basis for resource evaluation. Summary results of the evaluation of the in-situ resources are shown in Table 4.16 for three different cut-off grade levels: 0.3g/t, 0.5g/t and 0.7g/t Au.

Table 4.16: Visokiy Resource Estimate (WAI, 1st July 2010)
(in accordance with the guidelines of the JORC Code (2004)

Ore Type			Sulphide	Sulphide	Sulphide
Cut Off Grade (g/t)			0.3	0.5	0.7
Measured	Tonnage (kt)		-	-	-
	Au (g/t)		-	-	-
	Metal	kg	-	-	-
		oz	-	-	-
Indicated	Tonnage (kt)		-	-	-
	Au (g/t)		-	-	-
	Metal	kg	-	-	-
		oz	-	-	-
Measured + Indicated	Tonnage (kt)		-	-	-
	Au (g/t)		-	-	-
	Metal	kg	-	-	-
		oz	-	-	-
Inferred	Tonnage (kt)		1,261	1,048	795
	Au (g/t)		0.87	0.96	1.07
	Metal	kg	1,094	1,005	850
		oz	35,162	32,305	27,340

4.9 Mining

4.9.1 WAI Pit Optimisation

4.9.1.1 Introduction

As a continuation of *Mineral Resource* modelling of Tabornoe deposit WAI has undertaken a pit optimisation using the *Mineral Resource* Block Model prepared by WAI and updated in November 2010. The model was depleted to contain only those *Mineral Resources*, which have not been extracted as of 01 November 2010. WAI used NPV Scheduler® software for the optimisation, applying conceptual financial and technical parameters, provided by the Client.

The NPV Scheduler® programme comprises three components, the ultimate pit shell generator, a push back generator and the optimising scheduler. The ultimate pit generator is the first stage of the optimisation process and utilises a Lerchs-Grossman algorithm to generate an economic open pit shell from the *Mineral Resource* block model based on the initial input parameters.

The objective of this study was to obtain an optimised pit shell containing economically viable *Mineral Resources*, which therefore will provide an estimate of *Ore Reserves*. The estimate of *Ore Reserves* will form a basis for the mining schedule.

Prior to importing into NPV Scheduler®, the *Mineral Resource* block model was expanded by adding waste cells. Zero grade values for gold were also assigned to each waste cell.

4.9.1.2 Pit Optimisation Parameters

The metal price used for the pit optimisation was US$900/oz Au. The major Key Performance Indicators (KPI) such as mining and processing costs, and other parameters, considered during the optimisation, have been supplied by the Client as actual results of year 2008 to 9m 2010 production. These figures are given in Table 4.17 below.

Table 4.17: Tabornoe Key Performance Indicators for 2008 to 9m 2010

	Unit	2008	2009	9m 2010
Rock mined	Kt	5,735	10,479	8,906
Ore mined	Kt	1,526	2,230	1,696
Ore processed	Kt	1,677	2,251	1,711
Average Au grade (ore processed)	g/t	1.13	1.12	1.22
Ore stripped	Kt	4,208	8,238	7,210
Gold recovered**	kg	1,166	1,562	1,076
Gold recovered**	koz	37.5	50.2	34.6
Recovery rate	%	75	75.0	N/A
Full cash cost	US$/oz	567	659	819
Normalised TCC	US$/oz	530	539	679
Ore mining costs	US$/t	1.5	1.3	1.3
Waste mining costs	US$/t	1.5	1.2	1.2
Ore processing costs	US$/t	5	5	5
General and administration costs	US$M	2.7	3.4	3.1
CAPEX*	US$M	34.1	28.1	19.5
Depreciation	US$M	6.6	6.7	5.5

Note:
*** including Exploration and Evaluation**
**** including 73 GEO of silver production in March 2010**

WAI has reviewed the results of year 2008-2010 and technical data, obtained from the Client both during site visit and by request, together with the Client's production forecasts for 2010 and 2011, and found the following technical and economical parameters to be the best suitable for the optimisation purposes (Table 4.18).

Table 4.18: WAI Pit Optimisation Parameters

Parameter	Units	Value
Metal Price (Au)	US$/oz	900
Metal Price (Au)	US$/g	28.94
Production Rate	ktpa	2,200
Discount Rate	%	11.2
Dilution	%	9
Mining Recovery Factor	%	97
Mining Cost (Ore and Waste)	US$/t	1.3
Processing Cost (per t of processed ore)	US$/t	5.0
Gold Recovery	%	75
Final Pitwall Angle	Degrees	34

One of the most crucial geotechnical parameters is the overall pit slope angle. The shallower the overall slope angle, the more waste cells that are included in the pit design. This has a great effect on the stripping ratio and hence economic viability. An overall pit slope angle of 34° was used for the optimisation (as supplied by the Client and observed by WAI at the site).

4.9.1.3 Pit Optimisation Results

The results of the WAI Base case optimisation as calculated by NPV Scheduler® are presented in Table 4.19 below. The Base case optimised pit shell 3D image is shown in Figure 4.25.

Only the *Measured and Indicated Mineral Resources* were utilised in the optimisation as only these categories of *Mineral Resources* can become *Ore Reserves*.

Table 4.19: Tabornoe Deposit Open Pit Optimisation Results (WAI 2010, COG=0.33g/t)

Case	Gold Price (US$/oz)	Plant Recovery (%)	Annual Discount (%)	Ore (kt)	Au (g/t)	Au (kg)	Waste* (kt)	Stripping Ratio Estimation (t/t)
Base Case	900	75	11.2	23,129	0.80	18,536	64,618	2.79

Note:
9% dilution and 97% mining recovery applied
Waste is given inclusive of *Inferred* material, which is also treated as waste

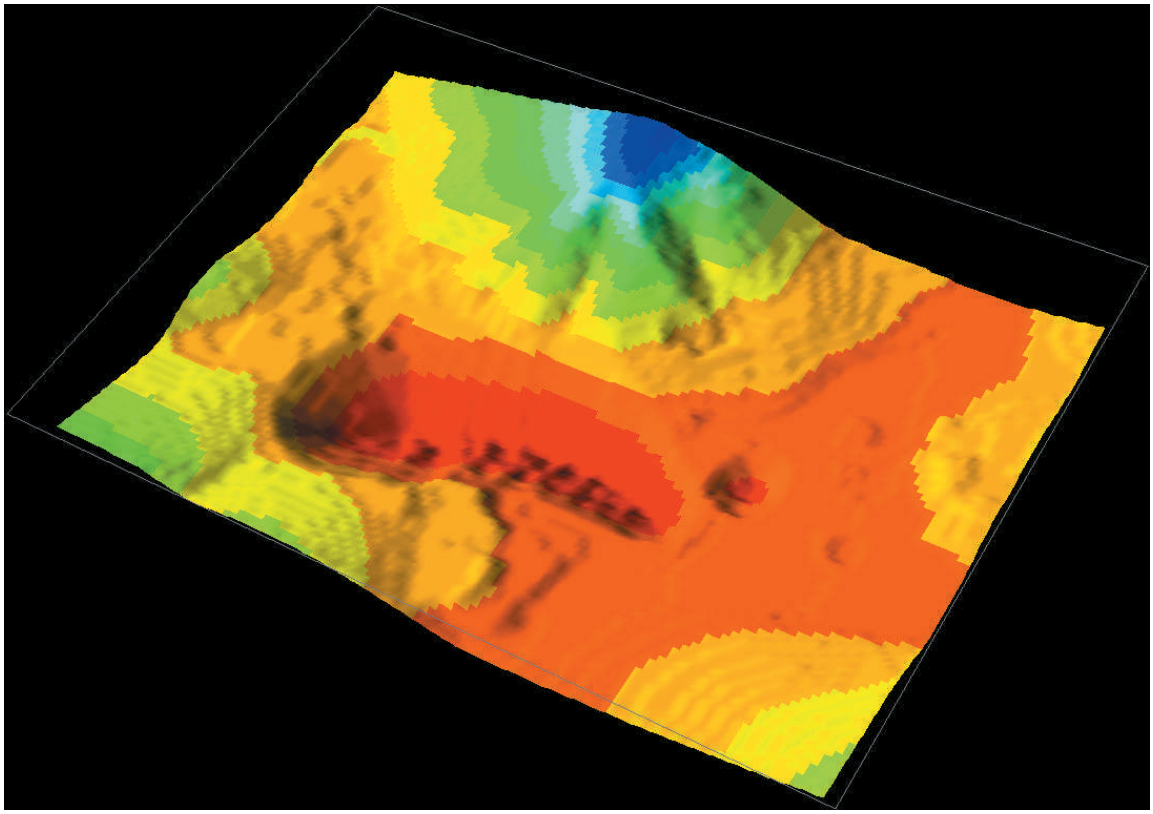


Figure 4.25: 3D View of WAI Optimised Pit

4.9.2 Ore Reserve Estimation (WAI 01 November 2010)

Based on the described above open pit optimisation, WAI estimated Tabornoe open pit *Ore Reserves*. Detailed results of this estimation are given in Table 4.20 and Table 4.21 below.

Table 4.20: Tabornoe Open Pit Ore Reserves as of 01.11.2010 (WAI) (in accordance with the guidelines of the JORC Code (2004)			
Cut Off Grade (g/t)			**0.27**
Proven	**Tonnage (kt)**		5,194
	Au (g/t)		0.91
	Metal	kg	4,703
		oz	151,210
Probable	**Tonnage (kt)**		17,867
	Au (g/t)		0.77
	Metal	kg	13,773
		oz	442,848
Proven + Probable	**Tonnage (kt)**		23,062
	Au (g/t)		0.80
	Metal	kg	18,475
		oz	594,058
Note **Mining Factors of 9% Dilution and 97% Mining Recovery applied** **Global density of 2.4t/m^3 applied to both ore and waste**			

Table 4.21: Tabornoe Reserve Pit Summary (WAI, 2010)			
Ore (kt)	**Waste (kt)**	**Strip Ratio**	**Total Rock (kt)**
23,062	64,534	2.80	87,595
Note **Ore Cut Off of 0.27g/t Au applied** **Global density of 2.87t/m^3 applied to both ore and waste** **Waste is given inclusive of *Inferred* material, which is also treated as waste**			

4.9.3 Life of Mine Schedule (LOM)

After optimised pitshells were generated for each of the optimisation runs, WAI has designed a mining schedule for the Base case.

The Ore Reserve statement was prepared by WAI as of 01 November 2010, therefore the LOM schedule considers only the second half of year 2010. As the Tabornoe mine already is in production and the majority of stripping works has been performed, the LOM schedule targets to maintain consistent total volume of ore (at a grade, equal to the average grade for the deposit) and waste extracted, providing the required planned annual ore tonnage of 2.2Mt as the result.

Table 4.22 below describes mining schedule for the Base case open pit shell designed by WAI.

An Ore Reserve open pit plan, long-section and cross-section are given in Figure 4.26, Figure 4.27 and Figure 4.28 below respectively.

Table 4.22: Tabornoe Deposit LOM Mining Schedule (WAI 01 November 2010)

Year		Nov-Dec 2010*	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	Total
Rock	kt	1,766	6,502	6,296	6,224	7,294	6,240	9,608	10,272	10,172	10,489	10,320	2,564	87,748
Waste	kt	1,727	4,293	4,105	4,024	5,093	4,037	7,410	8,069	7,977	8,288	8,113	1,482	64,618
ORE	kt	39	2,209	2,191	2,200	2,201	2,203	2,198	2,203	2,195	2,201	2,207	1,081	23,129
Au	g/t	1.20	0.84	0.89	0.85	0.90	0.84	0.80	0.69	0.72	0.79	0.80	0.55	0.80
Au	kg	46	1,855	1,953	1,879	1,978	1,859	1,769	1,526	1,577	1,744	1,756	592	18,536
Strip		45	1.94	1.87	1.83	2.31	1.83	3.37	3.66	3.63	3.77	3.68	1.37	2.79

Note:
*** Based on the Company forecast**
9% dilution and 97% mining recovery applied
Waste is given inclusive of Inferred material, which is also treated as waste

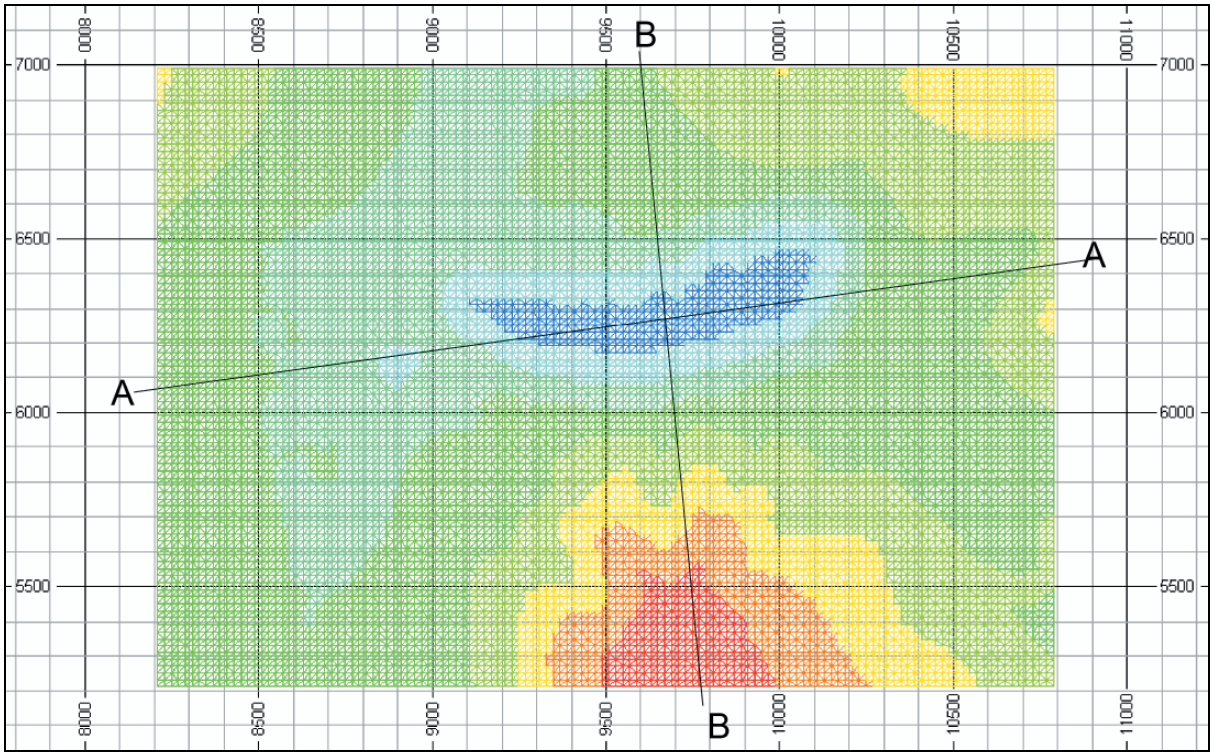


Figure 4.26: Plan View of WAI Reserve Pit, Colour Coded by Elevation

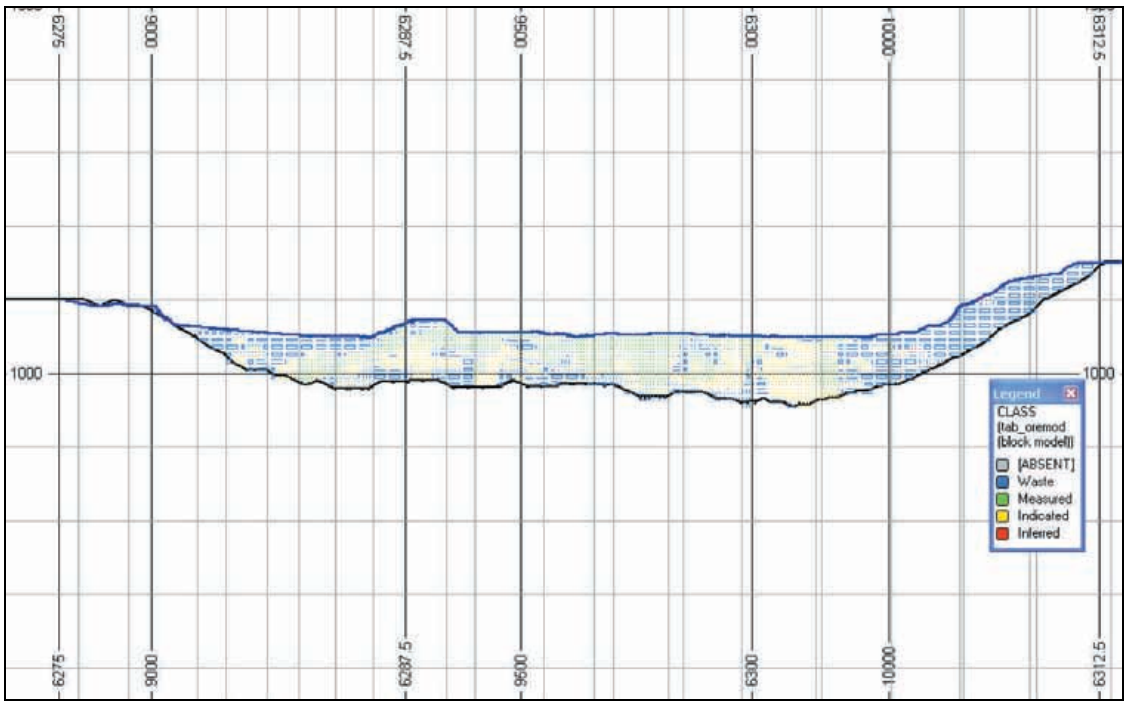


Figure 4.27: Long Section (Section A-A) of Geological Model
(Blue – Contours of Pit as of 01 November 2010; Black – Contours of WAI Reserve Pit)

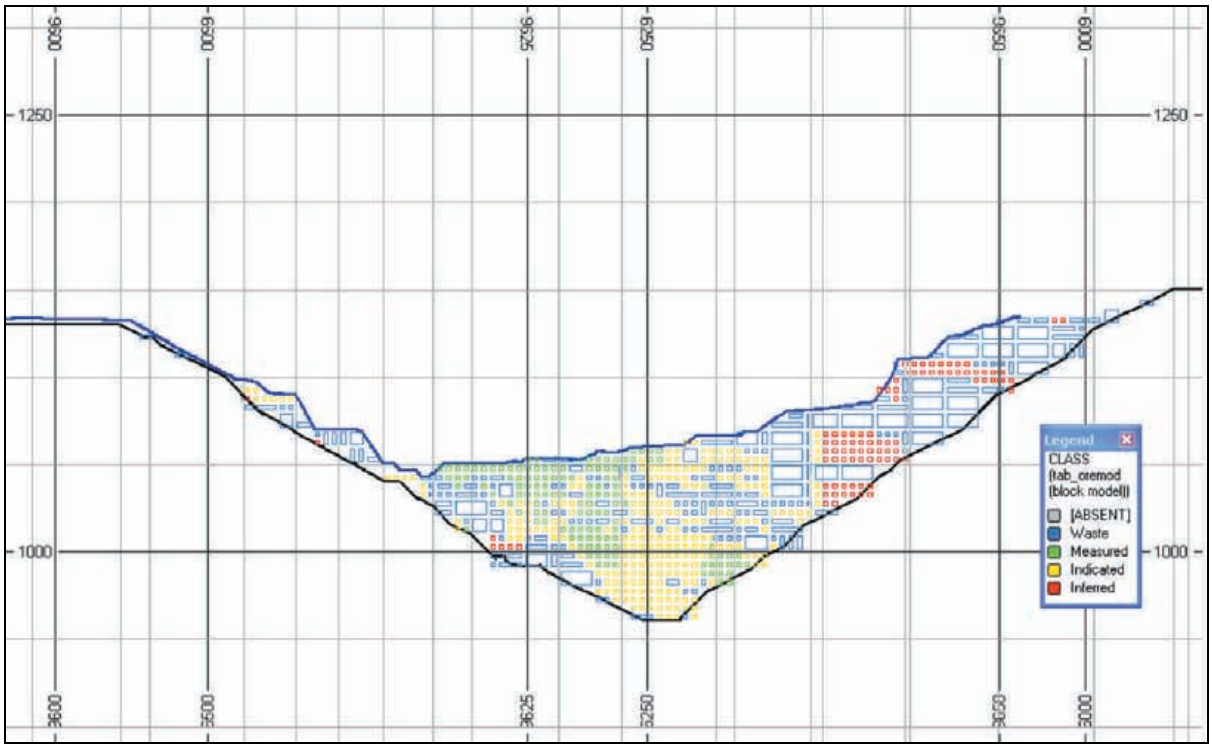


Figure 4.28: Cross-Section (Section B-B) of Geological Model
(Blue – Contours of Pit as of 01.11.2010; Black – Contours of WAI Reserve Pit)

4.9.4 Mining Method

The mine employs standard drill-and-blast techniques for hard rock fragmentation, whilst a proportion of the material within the open pit is amenable to ripping by bulldozer (without blasting). The ratio of hard to soft rocks is approximately 60:40. The ore and waste are excavated by hydraulic excavators and transported out of pit by 30t and 50t trucks.

Drilling and blasting on site are done in house. The depth of hole is adjusted according to its collar elevation in order to maintain a level profile.

Some of the rock within the pit boundary is loose enough to be stripped by a dozer. The bench is initially ripped by dozer on a spacing of approximately 6m between rip lines. The rip lines are then sampled, and rock is marked out as either ore or waste depending on the results of sampling.

Loose rock is extracted by hydraulic excavators. The ore is loaded on to 30t or 50t BELAZ trucks, and then delivered to either the ROM stockpile, low-grade ore stockpile or directly to the crusher. The roads at the site are well-maintained and in a good condition.

Waste is end-tipped directly over the dump edge, straight from the truck, contrary to recognised international safety regulations (best practice).

4.9.5 Fleet Maintenance and Workshops

During the WAI visit it was evident that number of machines which need to be serviced exceeds the workshop capacity. The maintenance area is poorly equipped; which results in an increase in man-shifts spent on standard service operations, and the absence of large tyre repairs facilities for the 50t trucks negatively affects their down-time and availability.

WAI Comment: *WAI recommends that more attention is paid to maintenance workshop improvement. The presence of a well-equipped and highly organized repair service is critical when rigid production targets are to be met.*

4.9.6 ROM and Low-grade Ore Stockpiles

The majority of the ore from the pit is stored in an out-of-pit ROM stockpile, located close to the primary heap leach crusher. This ROM stockpile is maintained by a wheel loader and a single 30t truck, which transports ore to the crusher.

Although this is an intermediate ROM stockpile, ore is transported from this ROM stockpile to the crusher on a first-in-first-out basis. The residence time of some of the ore could be considerable and as a consequence comparison of grade from mine to mill over short periods of time is irreconcilable.

4.10 Metallurgical Review

4.10.1 Process Description

Waste is mined preferentially during the winter; as the crushers do not operate during the coldest months of the year. Should it be necessary to mine payable grade material it is stockpiled adjacent to the crushers for processing when the crushers have restarted in the spring.

Gold is extracted in a cyanide heap leach and then recovered onto activated carbon.
Run of mine ore is crushed and stacked on heaps that are irrigated with dilute sodium cyanide solution. Pregnant solution is recovered and the gold adsorbed onto activated carbon. The gold is then eluted from the carbon and electrowon from the eluate prior to smelting to doré.

The operation treated approximately 1.6Mtpa in 2008 and 2.2Mtpa in 2009 and in xMtpa 2010.

4.10.2 Crushing

The current product size is visually at least 45mm. Reducing the top size of the crusher product to 10mm could result in an increase in extraction to over 90% and a one third reduction in cyanide consumption. To date no attempt has been made to decrease the product size. However, a new pad was constructed in 2010, in line with best standards with a total height of no more that 20.0m and extraction has been considerably improved as a consequence.

A cursory comparison of the gold production under a range of production scenarios shows that there is a benefit of approximately 10% greater gold production in operating the crushers at higher throughput and coarser product. This benefit can be increased to 50%, with additional cost savings arising from the lower cyanide consumption, through crushing to 10mm. For 2011, Nord Gold, believes that another new pad constructed to height of no more than 20.0m will provide largest potential for improvement.

4.10.2.1 Accounting

The ore trucks are supposed to be weighed en route from the mine to the crushers, but this is not always the case. There are no other mass measurement devices anywhere else in the crushing circuit and the crushers are therefore operated “blind”, other than for a tally of trucks tipped during a shift. Since a significant proportion of the ore mined is placed on a stockpile prior to being fed to the crushers there is no comparison on a shift by shift basis between the amount from mining and to crushing.

WAI Comment: *strongly recommended that belt scales be installed on the crushed product belts from each stream and on the conveyor removing the leach pad foundation material. This will enable reconciliation of tonnage mined, crushed and stacked on each of the leach heaps. Installation is now being planned for when the new No 1 heap leach pad comes on line although exactly where they will be located and how complete a balance will result could not be ascertained.*

A belt weightometer has been purchased and will be installed in 2011. Currently, run of mine gold grade is estimated by manual sampling.

4.10.2.2 Safety

Safety practices in the Tabornoe crushing plant, although in compliance with Russian requirements, were not up to international standards. Selected examples of areas that required improvement include:

- Stairs and access ways are irregular, steep and of substandard materials;
- Some guards on moving machinery are inadequate; and
- There are no facilities to implement lock-out of equipment.

In 2010, a HSE manager has been appointed to the Tabornoe site and a supervisory manager has been appointed to the Moscow office. For 2011, Nord Gold plans to implement a programme of best HSE practices at Tabornoe.

4.10.3 Heap Leaching

4.10.3.1 Leaching Circuit

Leaching is conducted by irrigating heaps of crushed material with dilute sodium cyanide solution.

The solution with which the heaps are irrigated has a target sodium cyanide concentration of 0.5g/l. A semi-permeable membrane is placed at the base of each new lift to reduce the amount of liquor that has to permeate to the base of the heap before entering the drainage system.

Currently there are three pads in production; one new pad - 12m high, second – 30m high and the oldest ole in approximately 40m high. In 2011 Nord Gold plan to construct another new pad (12m high) and complete a technical project regarding optimal pads design until 2015.

4.10.3.2 Heap Leaching Commentary

Production data from 2010 indicates that 2.9Mt were stacked on the leach heaps and the gold extraction achieved was 72%. Given the points noted above regarding the possible improvements that could be made, this is considered a high level of extraction and seems to confirm the relative insensitivity to crusher product top size Despite this it is likely that improved extraction would be achieved if the crushed product was finer and it is recommended that confirmatory tests be conducted.

4.10.4 Gold Recovery

4.10.4.1 Gold Recovery Circuit

Pregnant solution is collected in tanks and pumped through five parallel pairs of upflow carbon columns. Each pair of columns has the capacity to treat 100m^3/h solution.

Loaded carbon is transferred to two elution columns, each of 1t capacity. This section of the plant was originally designed to recover the gold from the solution arising from leaching 1.2Mtpa through three pairs of carbon columns. Two additional pairs of columns have now been installed.

Liquid losses are replenished by adding fresh water, the quantity of which varies from 55m^3/d during the winter and summer to almost nothing during the spring due to melting of snow and ice.

4.10.4.2 Plant Performance

Production data for 2010 indicates that the recovery of gold from solution was 90%. The barren solution grade of approximately 0.05mg/l is higher than would normally be expected from a pregnant solution of 0.5mg/l. Nord Gold have acquired a new regeneration kilm to increase carbon activity. This should reduce the barren solution grade in 2011.

There are no concerns over the adequacy of the power generation system.

4.10.5 Conclusions and Recommendations

The metallurgical operation at Tabornoe is basic but functions as required in terms of throughput and recovery. There are, however, areas where relatively minor changes could enhance performance and control significantly, the most notable of which are:

- Reduction of the top size of material fed to the leach heaps. This is currently not planned, but the existing equipment has been adjusted to crush below 45mm;
- Change the configuration of the secondary crushers to closed circuit. This is currently not planned for 2011, but is under consideration by Nord Gold;
- Use a cone crusher for secondary crushing. This is planned for 2011 but is under consideration by Nord Gold.
- Improve the metal accounting system around the crushers, which is now being implemented; and
- Implement more stringent safety practices.

In order to achieve these aspects, the following recommendations are made:

- Conduct confirmatory column leach tests on current arisings from the mine crushed to a range of sizes;
- Determine the optimal crusher product size in terms of maximising the amount of gold extracted;
- Install belt scales on the final product conveyor from each of the crushing sections;
- Install automatic samplers (cross stream or hammer type) on the final product conveyor from each of the crushing sections;
- Conduct a validation exercise on the design of the new No.1 heap leach foundation;
- Review the structure of the bonus system; and
- Conduct an independent safety audit.

4.11 Environmental & Social issues

4.11.1 Permitting

It has been reported that there is an OVOS document (equivalent to EIA) in place for the operations and MAC's/MAD's have been approved by the State authorities. All storage facilities are reported to have been designed, constructed and located in a manner consistent with the local regulations.

4.11.2 Water Management

During the site visit, the water management arrangements for the mine were reviewed and generally conform to modern best practice. Water supply both for technical and potable purposes is from the Pryamoi stream in summer and from Lake Usu in winter.

Within the plant, water is recycled continuously and is added to the process when necessary to compensate the losses occurred as a result of evaporation, etc.

The runoff waters during high rainfall are collected in 3 surge capacity plastic-lined reservoirs located between the heap leach pads and the processing plant.

4.11.3 Air Quality

Emissions to air from the mining, processing and related activities are relatively small and any effects are localised.

4.11.4 Waste Management

The main type of waste generated by the mining operations at Tabornoe is the waste rock which is classified as the lowest danger class (class V) and is disposed of in two separate dumps north of the current open pit.

Overall, waste management at Tabornoe operations seems well organised.

4.11.5 Environmental Monitoring

At the time of the site visit, WAI inspected the monitoring boreholes around the heap leach area, 50 perforated boreholes in total approximately 4m deep on the average.

For the surface (local streams) and underground waters, it was reported that the determinands monitored have not been exceeding the thresholds established for the operations, including that for CN.

> ***WAI Comment:*** *WAI believes that the monitoring conducted by Tabornoe staff is likely to pick up any chronic or long term pollution, but will not identify short term or acute incidents due to the absence of water in the monitoring boreholes around the heap leach area.*

4.11.6 Environmental and Social Management

At the time of the site visit, WAI has not been provided with an Environmental and Social Management System (ESMS), nor has been advised if the latter is in place for the operations.

> ***WAI Comment:*** *in this connection, WAI advises that the introduction, implementation and maintenance of ESMS will be required to ensure continued compliance with the Equator Principles and IFC Performance Standards.*

4.11.7 Mine Closure Planning and Land Rehabilitation

To WAI's knowledge there is no financial provision set aside for the final closure cost. Financial security for mine closure is not common in Russia. Estimated mine closure costs are only reported as financial liability in IFRS financial statements.

> ***WAI Comment:*** *it is recommended that the Company should prepare a Closure Plan.*

4.11.8 Occupational Health and Safety

The Labour Protection Department has 1 staff based in Neryungri who visits the mine regularly. At the mine, a nurse is present full time.

> ***WAI Comment:*** *it is understood that the mine is generally compliant with the regulatory requirements in Russia with no registered deviations from the norm and, based on a site walkover observations, it appears that H&S procedures are generally sound.*

4.11.9 Geotechnical Issues

The waste tips have not been designed from a geotechnical perspective and as such constitute a risk of failure. However, it has been reported that steps are taken to improve the construction methods, the practice of tipping, implement continuous monitoring over the current waste tips which are of concern and develop an Action Plan for the waste dump and the open pit to assist in monitoring stability.

> ***WAI Comment:*** *WAI considers that the Company is aware of the instability problems and is proactive in seeking means to minimise risks related to those.*

4.12 Life of Mine Plan of Tabornoe

4.12.1 Introduction

A life of mine production model for the Tabornoe gold deposit has been prepared in order to demonstrate the potential viability of the project and its robustness. This model was based on the Ore Reserve figures described above, capital investment requirements estimates provided by the Client and audited by WAI with a view to similar projects, and other parameters. Input parameters were implemented into the life of mine model. The details of this model are given below.

4.12.2 Life of Mine Model Assumptions and Input Data

The assumptions made in the WAI life of mine model for the Tabornoe were based on:

- Au price of US$900/oz;
- Ore reserves as of 01.11.2010, estimated by WAI in accordance with the guidelines of the JORC (2004) code;
- Mining Schedule, prepared by WAI. The Mining Schedule targeting a 2.2Mtpa ore mining rate with appropriate ramp-up of production;
- Long-term operating costs forecasts based on year 2008-2010 production results and considering analogous deposits in this part of Russia;
- Overall Au recovery indices obtained from actual annual results for 2008-2010;
- Annual discount factor of 11.2% (Base case);
- Royalty of 6% of revenue is applied;
- General and Administration costs estimated on the basis of the Client's forecast, and vary in a region of US$4.5-5.0/t; and

- Overall tax at 20% of net income applied.

A summary of the Tabornoe life of mine model assumptions and input data is given in Table 4.23 below.

Table 4.23: Tabornoe Life Of Mine Model Assumptions and Input Data

Year	TOTAL	Nov-Dec 2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Gold Price, US$/Oz	-	900	900	900	900	900	900	900	900	900	900	900	900
Ore production (diluted) kt	***23,129***	39	2,209	2,191	2,200	2,201	2,203	2,198	2,203	2,195	2,201	2,207	1,081
Waste mined, kt	***64,618***	1,727	4,293	4,105	4,024	5,093	4,037	7,410	8,069	7,977	8,288	8,113	1,482
Ore Mining cost,US$/t	***1.34***	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34
Waste Mining cost, US$/t	***1.23***	1.23	1.23	1.23	1.23	1.23	1.23	1.23	1.23	1.23	1.23	1.23	1.23
Mining OPEX, k US$	***110,474***	2,176	8,240	7,986	7,898	9,213	7,917	12,060	12,877	12,754	13,144	12,936	3,272
Au Grade g/t	***0.80***	1.20	0.84	0.89	0.85	0.90	0.84	0.80	0.69	0.72	0.79	0.80	0.55
Gold mined, kg	***18,536***	46	1,855	1,953	1,879	1,978	1,859	1,769	1,526	1,577	1,744	1,756	592
Recovery, %	***71%***	71%	71%	71%	71%	71%	71%	71%	71%	71%	71%	71%	71%
Gold recovered, kg	***13,645***	517	1,317	1,387	1,334	1,405	1,320	1,256	1,084	1,120	1,238	1,247	420
Gold recovered, Oz	***438,682***	16,622	42,349	44,582	42,887	45,160	42,440	40,388	34,838	36,003	39,811	40,083	13,519
Processing cost, US$/Oz	***260***	12	261	246	257	244	259	272	316	305	276	275	400
Processing cost, US$/t	***5.0***	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0
Proc and Mining Costs US$/Oz	***510***	143	455	425	441	448	446	571	686	659	607	598	642
Depreciation (total) k US$	***77,093***	5,020	7,439	7,439	7,439	7,439	7,439	7,439	7,439	5,000	5,000	5,000	5,000
Revenue, k US$	***394,813***	14,960	38,114	40,124	38,598	40,644	38,196	36,350	31,354	32,403	35,830	36,075	12,167
Operating Costs, k US$	***226,120***	2,369	19,283	18,943	18,900	20,218	18,930	23,051	23,893	23,730	24,150	23,973	8,679
G&A, Sales	***86,866***	4,355	7,501	7,501	7,501	7,501	7,501	7,501	7,501	7,501	7,501	7,501	7,501
Royalty, 6%	***23,689***	898	2,287	2,407	2,316	2,439	2,292	2,181	1,881	1,944	2,150	2,164	730
CAPEX, k US$	***29,400***	4,400	8,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	1,000	1,000	1,000

4.12.3 Conclusions

The positive NPV at various discount rates and at various gold prices, as well as the relatively high internal rate of return at nominal input parameters show, that the reserves considered in the financial model are profitable for exploitation in the current economic environment.

The deposit consists of a reasonable gold ore resource with potential to increase, as the area is being intensively explored. Current available reserve provides a mine life of around 10 years.

The fact that the key financial indices remain reasonably high given the conservative cost input parameters and recovery used in the models, shows good economic potential for the project.