SECTION XIV: INDEPENDENT TECHNICAL REPORTS

SUB-SECTION F: KAZZINC REPORT

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KAZZINC LIMITED

Mineral Resource and Ore Reserve estimations in accordance with the guidelines of the JORC Code (2004), and provision of a Competent Person's Report for the Assets held by Kazzinc Limited in both Kazakhstan and Russia

4 May 2011

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March 2011

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4 May 2011

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Dear Sirs

Competent Person's Report for the Assets held by Kazzinc Limited in Kazakhstan and Russia

Background

Kazzinc Limited (the "Client" or "Kazzinc") commissioned Wardell Armstrong International Ltd ("WAI") to prepare a Competent Person's Report (the "CPR") on its gold mining, development and exploration assets held in Kazakhstan and Russia. WAI understands that the CPR will be included as part of a prospectus (the "Prospectus") to be published in connection with a proposed offering of ordinary shares by Glencore International plc ("Glencore") to be admitted to the premium listing segment of the Official List of the United Kingdom Financial Services Authority and to trading on the main market for listed securities of the London Stock Exchange Main Listing and the main board of the Hong Kong Exchange (together, "Admission").

WAI hereby consents to the inclusion of this letter and the CPR in the Prospectus, with the inclusion of its name, in the form and context in which it appears in the Prospectus, to be published in connection with Admission.

WAI understands that Glencore International plc will be the ultimate parent company of the group.

For the purposes of Item 1.1 and 23.1 of Annex I and Item 1.1 and 10.3 of Annex III to Commission Regulation (EC) No. 809/2004 of 29 April 2004 (the "Prospectus Directive Regulation") WAI is responsible for this letter and the CPR as part of the Prospectus and declares that it has taken all reasonable care to ensure that the information contained in this letter and the CPR is, to the best of its knowledge, in accordance with the facts and contains no omission likely to affect its import. This declaration is included in the Prospectus in accordance with Item 1.2 of Annex 1 and 1.2 of Annex III to the Prospectus Directive Regulation**.**

The principal Mineral Assets in which the Client is interested comprise of production sites at Vasilkovskoye Open Pit, Maleevskoye Mine, Ridder-Sokolniy Mine, Tishinskiy Mine, Shubinskiy Mine, Shaimerden Open Pit, and reclamation of tailings from the Staroye (Old) tailings dam facility; together with advanced exploration properties at Dolinnoe & Obruchevskoe and Chekmar, all of which are located near to Ridder-Sokolniy in Kazakhstan. A further underground production mine, Novoshirokinskoye is located in the Chita region of the Russian Federation. Additional unexploited resources are to be found in tailings dam facilities (TMF's) of Chashinskoye located close to Ridder-Sokolniy Mine; and Tishinskiy TMF, located at Tishinskiy Mine, both in Kazakhstan.

In addition Kazzinc has three exploration licences located in Kazakhstan, namely, Solovievskiy Block which is located to the south of Ridder-Sokolniy Mine, Butachikhinsko-Kedrovskiy Block located west of Ridder-Sokolniy Mine; and Western Torgai located in the Kostanai region of north western Kazakhstan.

These projects are at various stages of exploration, development and mining, all of which are discussed in detail in the CPR. WAI considers that the relevant areas have sufficient technical merit to justify proposed programmes and associated expenditures.

The Prospectus contains an appropriate summary of each of the assets, and WAI is satisfied with the integrity of the information contained in the Prospectus based on the limited validation work performed by WAI, but more importantly, reliance on the legal due diligence performed by Kazzinc and its subsidiaries in the projects geographical locations.

WAI has been requested to provide an Independent valuation of the individual assets, although it has not been asked to comment on the fairness and reasonableness of any vendor or promoter considerations.

Requirement and Structure of the CPR

This report has been prepared by WAI in accordance with the requirements of the "Prospectus Rules" published by the UK Financial Services Authority from time to time and governed by the UK Listing Authority, the "Prospectus Directive" (2003/71/EC/) and the Prospectus Regulations (809/2004), "CESR's (now the European Securities and Markets Authority) recommendations for the consistent implementation of the European Commissions Regulation on Prospectuses No. 809/2004" (as updated by the European Securities and Markets Authority on 23 March 2011 following the publication of a consultation paper in 2010, in relation to the content of the prospectuses regarding mineral companies) and Chapter 18 of the Hong Kong Listing Rules.

WAI has prepared independent resource and reserve estimates for Kazzinc's material assets in accordance with the 2004 edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (the "JORC Code (2004)"), all of which are shown in the CPR.

The CPR has been structured on a technical discipline basis into sections on Geology, Mineral Resources and Ore Reserves, Mining Engineering, Mineral Processing, Infrastructure, Occupational Health and Safety, Environmental Management, and a Financial Assessment for each of the Mineral Assets.

Site visits were made by WAI to all the assets mentioned above.

Verification, Validation and Reliance

The CPR is dependent upon technical, financial and legal input. The technical information as provided by Kazzinc to, and taken in good faith by, WAI has not been independently verified by means of re-calculation, but all reserve and resource estimates as presented have been re-modelled, by evidence from WAI's site visits and observations, are supported by details of exploration results, analyses and other evidence and take account of all relevant information supplied by Kazzinc. WAI has conducted a review and assessment of all material technical issues likely to influence the future performance of the Mineral Assets which included the following:

- Inspection visits to the mining operations, processing facilities, surface structures and associated infrastructure, undertaken in the last quarter of 2010, with:
    - discussion and enquiry following access to key on-site and corporate personnel;
    - an examination of historical information and results made available by Kazzinc in respect of the mining;
    - re-modelling Kazzinc's resource and reserve estimates where appropriate; and
    - a review of the Kazzinc's production forecasts and costs and undertaken all necessary investigations to ensure compliance with the Prospectus Directive Regulation in conjunction with the CESR (now the European Securities and Markets Authority) Recommendations (as updated by the European Securities and Markets Authority on 23 March 2011) and the JORC Code (2004), (where appropriate) in terms of the level of disclosure.

The resource and reserve estimates presented in the CPR for the principal mineral assets have been prepared in accordance with the guidelines of the JORC Code (2004).

WAI has placed reliance on Kazzinc that the following information provided by Kazzinc to WAI is both valid and accurate for the purpose of compiling the CPR:

- all technical information; and
- that the legal ownership of all mineral and surface rights has been verified and save as disclosed in the CPR that no significant legal issues exists which would affect the likely viability of a project and/or the mineral resources and ore reserves as reported herein.

Limitations, Declarations, Consent and Copyright

Limitations

Kazzinc has confirmed to WAI that to its knowledge the information provided by Kazzinc was true, accurate and complete and not incorrect, misleading or irrelevant in any aspect. WAI has no reason to believe that any facts have been withheld.

The achievability of production forecasts and costs are neither warranted nor guaranteed by WAI. The forecasts as presented and discussed herein have been proposed by Kazzinc management and adjusted where appropriate by WAI, based on up-to-date reserve estimates, and cannot be assured. They are necessarily based on economic assumptions, many of which are beyond the control of Kazzinc.

Declarations

WAI will receive a fee for the preparation of the CPR in accordance with normal professional consulting practice. This fee is not contingent on the outcome of the Admission or value of Glencore and WAI will receive no other benefit for the preparation of the CPR.

None of WAI or its directors, staff or subcontractors who contributed to the CPR have, at the date of this letter, and has not had within the previous two years, any shareholding in or other relationship with Glencore, Glencore International AG, Kazzinc or the principal current assets in which Kazzinc is interested which include the Vasilkovskoye Open Pit operation, Maleevskoye Mine, Ridder-Sokolniy Mine; (and its satellites which include Tishinskiy Mine, Shubinskiy Mine, Dolinnoe & Obruchevskoe development projects and Chekmar exploration project) and Shaimerden Open Pit operation, all of which are located in Kazakhstan.

WAI considers itself to be independent of Glencore, Glencore International AG and Kazzinc. None of WAI or its directors, staff or subcontractors who contributed to the CPR has any interest in the proposed offering of ordinary shares in the capital of Glencore.

In the CPR, WAI provides assurances to the Directors of Glencore and the Directors of Kazzinc that certain Technical and Economic data including production profiles, operating expenditures and capital expenditures, of the Mineral Assets as provided to WAI by Kazzinc and reviewed and where appropriately modified by WAI are reasonable.

The CPR includes technical information, which requires subsequent calculations to derive subtotals, totals and weighted averages. Such calculations may involve a degree of rounding and consequently introduce an error. Where such errors occur; WAI does not consider these to be material.

Furthermore, WAI is responsible for this letter and the CPR as part of the Prospectus and declares that it has taken all reasonable care to ensure that the information contained in this letter and the CPR is, to the best of its knowledge, in accordance with the facts and contains no omission likely to affect its import.

Consent and Copyright

WAI consents to the inclusion of its name and all references to WAI in the Prospectus, the issuing of this letter and the CPR in the form and content in which it is to be included in the Prospectus.

Neither the whole nor any part of this letter and the CPR nor any reference thereto may be included in any other document without the prior written consent of WAI regarding the form and context in which it appears.

Copyright of all text and other matter in this document, including the manner of presentation, is the exclusive property of WAI. It is an offence to publish this document or any part of the document under a different cover, or to reproduce and or use, without written consent, any technical procedure and or technique contained in this letter and the CPR. The intellectual property reflected in the contents resides with WAI and shall not be used for any activity that does not involve WAI, without the written consent of WAI.

Responsibility for the Competent Person's Report and No Material Change

WAI accepts responsibility for the CPR for the purposes of Item 1.2 of Annex 1 and 1.2 of Annex III to the Prospectus Directive Regulation. The CPR is complete up to and including 4 May 2011. Having taken all reasonable care to ensure that such is the case, WAI confirms that, to the best of its knowledge, the information contained in the CPR is in accordance with the facts, contains no omission likely to affect its import, and no material change has occurred from 4 May 2011 to the date hereof that would require any amendment to the CPR.

Qualification of Consultants

WAI comprises over 50 staff, offering expertise in a wide range of resource and engineering disciplines. WAI's independence is ensured by the fact that it holds no equity in any project. This permits WAI to provide its clients with conflict-free and objective recommendations on crucial judgment issues. WAI has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, MER's and CPR's, and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide.

The CPR has been prepared based on a technical and economic review by a team of consultants sourced from the WAI offices in Europe over a 4 month period. These consultants are specialists in the fields of geology, resource and reserve estimation and classification, open pit mining, rock engineering, mineral processing, hydrogeology and hydrology, tailings management, infrastructure, environmental management and mineral economics.

The individuals listed below have provided input to the CPR and have extensive experience in the mining industry and are members in good standing of appropriate professional institutions:

- Phil Newall, MCSM, BSc, PhD , CEng, FIMMM, is Director of Minerals and Geologist with WAI and has practised his profession as a mine and exploration geologist for over 25 years for both base and precious metals;
- Mark Owen, BSc, MSc, (MCSM), CGeol, EurGeol, FGS, is a Technical Director and Geologist with WAI and has over 28 years international experience as a mine and exploration geologist in both surface and underground mining operations;
- Che Osmond, BSc, MSc (MCSM), ProfGradMIMMM, CGeol, Euro Geol, FGS; is a Principal Geologist with WAI and has over 15 years experience in mining and exploration geology;
- Adam Wheeler, BSc MSc CEng EurIng, MIMMM; is Principal Resource Analyst and Mining Engineer, with WAI specialising in the application, customisation and management of mining and geological software systems;
- Richard Ellis, MCSM BSc MSc FGS; is a Senior Resource Geologist with WAI who has some 8 years operational experience within the industrial minerals sector in the UK;
- Colin Taylor, MA, Pr.Sci.Nat, is an Associate Geologist with WAI who has over 40 years of international experience in exploration and resource geology, initially in South Africa and Namibia before being based in the UK.
- Owen Mihalop, BSc (Hons), MSc, MCSM, CEng, MIMMM, is a Technical Director of Mining with WAI with 15 years broad based experience in the mining and quarrying industries. He has gained experience in grass-roots exploration through to large scale open-pit and underground mining projects across Ireland, Bulgaria, Spain and Canada;
- Bruce Pilcher, BE(Mining)Syd, MAusIMM(CP), MIMMM, CEng, Eur Ing is an Associate Director and Principal Mining Engineer with WAI with 25 years experience in underground and surface mining operations in South Africa, Australia and UK. Bruce has served as Mine Superintendent, Regional and Technical Services Manager. He has considerable experience in production management, mine design and planning, and contract management in the coal and metalliferous mining industries
- Lewis Meyer, BEng, ACSM, MSc, MCSM, PhD, CEng, FIMMM is a Principal Mining Engineer with WAI who has over 18 years experience in mining, underground civil construction and rock mechanics of surface and underground mining operations;
- Daniil Lunev, DipEng (SPMI), PhD (SPMI) is a Mining Engineer with WAI whose specialist area is with NPV© scheduler mining software and mining machinery.
- Stuart Richardson, BEng, ACSM, GradMIMMM is a Mining Engineer with WAI who joined Wardell Armstrong in early 2007 as a Graduate Mining Engineer who now assists in the production of of pre-feasibility and feasibility studies using Mine 2-4D software;
- Colin Hunter, BSc (Hons), PhD, is an Associate Consultant with WAI who has 32 years minerals processing experience ranging from laboratory test work and pilot plant operations through to plant commissioning, operations and trouble-shooting;
- Phil King, ARSM, BSc, FIMMM is a Technical Director and Metallurgist with WAI and has over 25 years experience within the minerals industry in both process testwork and design for metallic and industrial minerals worldwide;

- Chris Broadbent, BSc, PhD, CEng, FIMMM, is a Partner with Wardell Armstrong and is a well-recognised authority on pyrometalurgical processing and the treatment and disposal of wastes from mining and metallurgical plants;
- Andrei Kudrin, BS (Hons), MEng (Hons), as a mature student has recently graduated from the University of Exeter with an engineering degree in Renewable Energy. Before joining WAI he has had placements with various companies such as Scottish and Southern Energy and Cornwall Sustainable Energy Partnership;
- Nick Coppin, BSc, MSc, is Managing Director of WAI and an Environmental Scientist with over 30 years of broadly based experience in the mining and minerals industry. He has worked extensively with government and regional authorities and institutions for mineral planning, mining administration, environmental protection, industrial development and finance. He is fully conversant with national and international guidelines, procedures, regulations and standards for the mining and quarrying sector
- Kim-Marie Clothier, BSc (Hons) MRes AIEEM, MIEMA, Grad IMMM, ACMI; is a Senior Environmental Scientist with WAI working on Environmental and Social Impact Assessments on mining projects overseas in Uzbekistan, Macedonia, Kyrgyzstan;
- Kathy Hicks, BSc (Hons) MSc MCSM FGS is a Geo-Environmental Scientist with WAI, Kathy specialising in the field of mineral extraction and the remediation of land affected by past mining activities;
- Christine Blackmore, BSc, MSc, CEnv, FIMMM is a Principal Environmental Geologist with WA who has over 10 years of experience in environmental work both in the UK and overseas. Her expertise is focused in environmental management (EMS, EIA ect) and environmental auditing within the mining industry; and
- Julia Boiko, BSc, Regional Manager – Kazakhstan for WAI, has worked for Western mining companies in Kazakhstan on nickel and gold projects as logistics manager and technical translator.

The Competent Person who has supervised the production of the CPR is Dr Phil Newall who is Director of Minerals with WAI and a Geologist with over 25 years experience in the mining industry.

Yours faithfully
for and on behalf of
Wardell Armstrong International Ltd

P. N

P Newall
Director of Minerals

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EXECUTIVE SUMMARY

Kazzinc Limited (the "Client" or "Kazzinc") commissioned Wardell Armstrong International Ltd ("WAI") to prepare a Competent Person's Report (the "CPR") on its gold mining, development and exploration assets held in Kazakhstan and Russia. WAI understands that the CPR will be included as part of a prospectus (the "Prospectus") to be published in connection with a proposed offering of ordinary shares by Glencore International Plc ("Glencore") to be admitted to the premium listing segment of the Official List of the United Kingdom Financial Services Authority and to trading on the main market for listed securities of the London Stock Exchange Main Listing and the main board of the Hong Kong Exchange (together, "Admission").

WAI understands that Glencore International plc will be the ultimate parent company of the group.

Kazzinc has acquired a significant portfolio of mineral assets in Kazakhstan and Russia. These vary from mature mining operations through to grass-roots exploration terrain and are summarised below.

Glencore owns 50.69% of Kazzinc. The remainder of the business is owned by Verny Capital JSC (48.73%), a Kazakh investment fund unrelated to Glencore, with certain small shareholders accounting for the remaining 0.58%.

Production Mines

Kazakhstan

- Vasilkovskoye Mine;
- Maleevskoye Mine and satellite including Grechovsky Mine;
- Ridder-Sokolniy Mine; and satellites including Tishinskiy Mine and Shubinskiy Mine;
- Staroye (Old) tailings management facility (TMF);
- Ridder Smelter; and
- Shaimerden Open Pit.

Russia

- Novoshirokinskoye Mine.

Advanced Exploration Properties

Kazakhstan

- Dolinnoe & Obruchevskoe;
- Chekmar;
- Chashinskoye TMF; and
- Tishinskiy Slimes Ponds.

Exploration Licences

Kazakhstan

- Solovievskiy Block located south of Maleevskoye;
- Butachikhinsko-Kedrovskiy Block located west of Tishinskiy ; and
- Western Torgai located in the Kostanai region of northern western Kazkahstan.

In addition to these mineral assets Kazzinc also operates smelters and a hydroelectric power station (which are included in this report as part of the operational structure).

These assets are described in more detail below. All Resource and Reserve estimates presented in the CPR have been prepared by WAI and in accordance with the JORC Code (2004).

Production Mines

Vasilkovskoye

The world-class Vasilkovskoye gold deposit is located in northern Kazakhstan, 17km north of the city of Kokshetau. It represents a stockwork located at the junction of gabbro/gabbrodiorite and granite/granodiorite rocks, recognising that the ore control was the intersection of faulting parallel to the NW trending Dongulagashsky fault with the NE trending Vasilkovskoye fault. The mineralisation forms a flattened zone, partially pinching out at depth, although an aggressive in-pit exploration programme is underway to properly test the depth extension to the mineralisation.

Au mineralisation is spatially associated with quartz and quartz-arsenopyrite veins and veinlets of hydrothermal origin.

Exploration within the licence area has led to the discovery of other, nearby exciting targets which are, and will be, the focus of future exploration activity.

The mineral resource estimate for the Vasilkovskoye Gold Project is based on the Ordinary Kriging model prepared by Kazakhstan Mineral Company (KMC) in October 2009, which has been depleted upto a pit survey dated 01 January 2011.

Vasilkovskoye Mineral Resource Estimate (WAI 01.01.2011)
(In accordance with the Guidelines of the JORC Code (2004))

Cut Off Grade Au (g/t)			0.4	0.9	1.5	2.0
Measured	**Tonnage (Mt)**		45.23	30.77	19.52	13.71
	Au (g/t)		1.75	2.28	2.92	3.41
	Metal	**kg**	79,331	70,089	56,924	46,780
		oz	2,550,552	2,253,406	1,830,142	1,504,002
Indicated	**Tonnage (Mt)**		141.56	97.72	58.96	39.95
	Au (g/t)		1.72	2.21	2.89	3.44
	Metal	**kg**	243,862	215,792	170,309	137,371
		oz	7,840,344	6,937,853	5,475,553	4,416,563
Measured + Indicated	**Tonnage (Mt)**		186.80	128.49	78.48	53.66
	Au (g/t)		1.73	2.22	2.90	3.43
	Metal	**kg**	323,193	285,880	227,233	184,150
		oz	10,390,893	9,191,256	7,305,694	5,920,566
Inferred	**Tonnage (Mt)**		99.08	68.63	39.74	26.21
	Au (g/t)		1.77	2.27	3.07	3.77
	Metal	**kg**	175,747	156,032	122,157	98,883
		oz	5,650,405	5,016,526	3,927,426	3,179,143

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) Mined areas removed from evaluation

The Vasilkovskoye open pit mine was originally developed during the early 1980's to exploit an oxide gold resource. Mining of the oxide material continued from 1980 until 2005, when the resource was finally depleted. In total 14Mt of oxide ore were mined and processed by heap leaching. A significant primary sulphide resource is present below the oxide layer and the decision to re-develop the mine and build a sulphide processing facility was made during 2006.

The new plant was designed to process up to 8Mtpa of sulphide ore and a mining study was completed by AMC Consultants in order to design a mining operation to feed the plant at this rate. AMC re-estimated the resources in accordance with the JORC Code (2004) guidelines and undertook geotechnical, hydro-geological, pit optimisation and preliminary mine design studies. The initial AMC pit design was to a depth of 360m but following re-optimisation at higher gold prices, the final pit design by Tomsk Institute was for a 440m deep pit.

Waste stripping, ore stockpiling and construction of the in-pit crushing facilities commenced during 2007. Production itself officially began with the completion of the processing facilities in November 2009. Due to problems with the processing plant, however, the mine is currently running at less than half of the design capacity, producing in the region of 270kt of ore per month rather than 600kt per month.

The Vasilkovskoye mine is a conventional open pit mine. The ore and waste rock is drilled and blasted prior to excavation by diesel-hydraulic excavators. The ore is loaded into diesel powered off-highway rigid dump trucks and hauled to either one of three in-pit stockpiles, depending on grade, or dumped directly into an in-pit crusher. The waste rock is hauled to one of two waste dumps depending on which side of the pit is being excavated. The mining fleet is owned and operated by VasGold.

Reserve estimations, related to pit design and scheduling, may be summarised as:

- **Pit Optimisation**: Pit optimisation runs were completed on the available KMC and WAI resource block models, based on parameters supplied by Vasgold;
- **Pit Design**: Based on the optimisation results, a new pit design was completed. This goes beyond the existing 'final design (Cutback #4) and incorporates a significant pit extension to the west. A summary of the reserves from this revised pit design estimated in accordance with the guidelines of the JORC Code (2004) is shown in the table below.

Vasilkovskoye Ore Reserve Estimate (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnes Mt	Au g/t	Au Contained t	Rock Mt	Waste Mt	Strip Ratio
Proven	33.3	1.95	64.95	-	-	-
Probable	90.7	1.94	175.76	-	-	-
Total	***123.97***	***1.94***	***240.71***	***870.65***	***746.68***	***6.02***

Notes:
Reserves determined with 0.9g/t Au cut-off, using KMC resource block model
Mining factors applies of 17.5% dilution and 95% mining recovery
Waste includes 38Mt of Inferred material at ore grades

- **Scheduling**: A life-of-mine schedule was developed, based on an 8Mtpa mill production rate. Material from the final cutback is not mined until year 5, with the bulk of production from this final cutback coming after year 6.

For processing, ore is subjected to primary crushing in the open pit and the -550mm fraction is conveyed to a coarse ore stockpile by overland conveyors. Further cone crushing reduces this material to 80% passing 5.2mm.

From here, the ore is ground in two parallel circuits, each using a single stage 6.7 x 11.3m Outukumpu ball mills. Further treatment involves flotation and gravity concentration using Knelson units.

The gravity and flotation concentrates are combined and ground in a 3.6 x 5.5m ball mill to achieve a product size of 95% passing 45μm.

The combined concentrates are then subjected to ultrafine grinding, using Deswick mills, to achieve a product grading 80-90% passing 4µm.

The gold in the gravity and flotation concentrates is partially refractory and the sulphide minerals need to be oxidised to ensure maximum gold recovery. This is achieved via the "Leachox Process", supplied by Maelgwyn Minerals Services (MMS). This is followed by Detox.

Gold is recovered from carbon using the standard Zardra method at 140-150°C and 2.5 atmospheres. The electrolysis product is smelted to produce doré assaying 85-92% Au in 12kg bars.

The technological recoveries for the process plant for 2010 show that there was a gradual increase in recovery throughout the year. The maximum recovery was 67.2% in August and it appears that recoveries generally range between 60-67%. The plant head grade has generally increased throughout the period, ranging from 1.65ppm Au (January) to 2.28ppm Au (December).

The tonnage throughput peaked in May and April, but decreased again in August and September before reaching 556kt in October. However, throughput decreased during November and December due to refurbishment of the Outotec mills.

In summary, the Plant has experienced a long ramp up period due to problems with equipment and lower than expected process recoveries. However, recent initiatives by the company have addressed many of these issues to allow plant production and recovery to approach design parameters, although optimal conditions have yet to be reached.

Vasilkovskoye has a very high standard of environmental, social, health and safety management. It is compliant with Kazakh licences, permits and norms. Both the environmental and labour safety functions of the company are of high quality and well organised, and supported by a strong corporate policy, commitment and team.

There are no known social, community or cultural issues or impacts that need to be addressed, and no displacement or compensation requirements. The mine appears to have a good relationship with the local communities, and does not put undue pressure on the social infrastructure of the District or Oblast.

Maleevskoye

The Maleevskoye underground polymetallic mine is situated some 17km to the north of the town of Zyryanovsk (where the concentrator is located) which in turn is situated 186km east of Ust-Kamenogorsk, northeastern Kazakhstan.

The deposit comprises seven gently dipping stratabound zones of lead-zinc-copper-gold-silver mineralisation: Rodnikovaya, Maleevskaya, Octyabrskaya, Holodnaya, Lugovaya, Bobrovskaya and Platovskaya, confined to broad mushroom-shaped brecciated domes passing downwards into steeply dipping stockwork feeder channels, which are generally aligned along NW-trending fractures. The largest and richest mineralised zones (Maleevskaya, Rodnikovaya, Holodnaya and Lugovaya) occur within a 150-200m thick Maleevsky Member of the Maslovskaya Formation. Strike lengths exceed 1km in Orebody 3 at Maleevskaya zone and in the Rodnikovaya zone.

Mineralisation is divided into polymetallic (lead-zinc with >0.6% Pb) and copper-zinc (<0.6% Pb and high Cu). In general, polymetallic mineralisation, accompanied by barite, occurs on the flanks and on the hangingwall side of the lenses, whilst copper-zinc mineralisation tends to prevail in the central parts of the lenses and on the footwall side.

The main metalliferous minerals are pyrite, sphalerite, chalcopyrite and galena, whilst the main gangue minerals are quartz, chlorite, calcite, barite, actinolite, tremolite and less common sericite, albite, epidote and biotite.

The main recoverable by-products are gold and silver. Gold occurs in its native form as very fine grains (5-8 microns) dispersed mainly in pyritic copper mineralisation and barite-bearing lead-zinc mineralisation. A small percentage of gold occurs as larger visible grains of up to 0.8mm across. The fineness of gold is 794.

For resources, WAI has prepared a Mineral Resource estimate dated 01.01.2011 for Maleevskaya in accordance with the guidelines of the JORC Code (2004), with grade estimation carried out using Ordinary Kriging (OK) as the principal interpolation method with Inverse Power of Distance (IPD) used for comparative purposes. For orebodies 3b in the Maleevskaya Zone, 18 and 19 in the Octyabrskaya Zone and orebody 5 in the Rodnikovaya Zone, IPD was used as the principal interpolation method.

The results of this study are shown in the table below:

Maleevskoye Mineral Resource Estimate - For All Orebodies
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Type	Tonnage (Mt)	Density	Pb		Zn		Cu		Au		Ag		Ba		Sp		EQV
				%	t	%	t	%	t	g/t	oz	g/t	oz	%	t	%	t	%
Measured	CuZn	5.16	3.57	0.31	16,118	3.17	163,709	2.37	122,268	0.46	76,838	45.78	7,592,815	2.34	12,074	20.83	107,488	6.62
	PbZn	7.77	3.81	1.68	130,483	9.41	730,724	2.39	185,584	0.73	181,078	98.97	24,707,132	5.19	40,308	18.06	140,209	13.44
	Total	12.92	3.71	1.13	146,601	6.92	894,433	2.39	307,852	0.62	257,917	77.74	32,299,948	4.05	52,382	19.17	247,696	10.71
Indicated	CuZn	3.68	3.22	0.35	12,826	3.28	120,504	1.65	60,638	0.39	45,517	41.84	4,943,748	2.30	8,438	11.46	42,095	5.71
	PbZn	7.36	3.62	1.55	114,356	8.59	631,546	2.10	154,476	0.65	154,837	76.29	18,043,611	3.78	27,792	11.49	84,529	12.15
	Total	11.03	3.49	1.15	127,182	6.82	752,050	1.95	215,114	0.56	200,354	64.82	22,987,359	3.28	36,231	11.48	126,625	10.00
Measured + Indicated	CuZn	8.83	3.43	0.33	28,944	3.22	284,213	2.07	182,906	0.43	122,355	44.14	12,536,563	2.32	20,512	16.93	149,583	6.24
	PbZn	15.12	3.72	1.62	244,838	9.01	1,362,270	2.25	340,060	0.69	335,915	87.94	42,750,744	4.50	68,101	14.86	224,738	12.81
	Total	23.96	3.61	1.14	273,783	6.87	1,646,483	2.18	522,966	0.60	458,271	71.79	55,287,307	3.70	88,613	15.63	374,321	10.39
Inferred	CuZn	1.77	3.06	0.29	5,047	2.80	49,408	1.22	21,533	0.18	10,145	24.28	1,377,707	1.15	2,038	5.80	10,234	4.55
	PbZn	3.11	3.21	2.31	71,746	6.23	193,725	0.84	26,099	0.29	28,611	60.67	6,061,129	2.75	8,538	5.83	18,101	8.55
	Total	4.87	3.16	1.58	76,794	4.99	243,133	0.97	47,632	0.25	38,756	47.49	7,438,836	2.17	10,576	5.82	28,334	7.10

Maleevskoye is the largest underground mine in the Kazzinc group in terms of ore production. Operations at Maleevskoye started in 2000 at a rate of 1.5Mtpa, with full scale production of 2.25Mtpa being reached in 2002. Two ore types are mined at Maleevskoye, copper-zinc ore and polymetallic ore. Both products are transported by road to the Zyryanovskiy Mining and Concentrating Complex ("ZGOK") located some 25km to the south of the mine site. The mine utilises modern high capacity trackless mining methods combined with a traditional tracked haulage system. The main mining methods in operation are sub-level caving, which accounts for approximately 5% of production, and sub-level open stoping with backfill, which accounts for the remaining 95% of production.

Both mining methods utilise mechanised development with electric-hydraulic face-jumbos and diesel LHDs for drilling and ore extraction. Ore and waste is transported either directly to ore/waste passes using LHDs or loaded into trucks and transported to a central ore/waste pass. Tailings are used for backfill.

Despite the fact that the mine is located 25km from the processing facility, the modern high productivity mining methods allow a good overall mining cost (approximately US$18/t of ore mined). The reserves appear sufficient to maintain operations at the present level until 2015, but after that production rate drops as a result of fewer faces being available.

WAI has carried out a mine design, and produced Ore Reserves for the Maleevskoye deposit, based upon the most recent Mineral Resource Block Model (WAI 2010). WAI has used GijimaAST Mine2-4D® software to prepare the mine design, and EPS® software to produce Ore Reserves.

During the mine design process, losses (4-8%) and dilution (10-12%) were applied to the mined tonnages at Maleevskoye. The stope blocks for Maleevskoye have been designed to a minimal average block grade of 4% Zn, or 4% Zinc Equivalent (ZnEQV).

Ore Reserves for the Maleevskoye deposit have been calculated in accordance with the guidelines of the JORC Code (2004). A summary of the Ore Reserves is presented below with the production schedule. It should be noted that the production schedule includes 171,010t of Inferred material (at 5.14% Zn, 1.31% Cu, 0.93% Pb, 0.30g/t Au and 41.8g/t Ag), which have not been reported as Ore Reserves. This material has been included in the production schedule, as it is not realistic to leave this material in-situ.

Maleevskoye Ore Reserve Estimate (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (kt)	Grade					Contained Metal (t)				
		Zn (%)	Cu (%)	Pb (%)	Au (g/t)	Ag (g/t)	Zn (t)	Cu (t)	Pb (t)	Au (oz)	Ag (oz)
Proven	5,042	6.46	1.92	1.00	0.56	68.13	325,627	96,770	50,358	90,779	11,044,289
Probable	7,061	6.29	1.69	1.04	0.51	56.23	444,011	119,630	73,522	115,777	12,764,963
Total	***12,103***	***6.36***	***1.79***	***1.02***	***0.53***	***61.19***	***769,638***	***216,400***	***123,880***	***260,233***	***23,810,222***

Maleevskoye Production Schedule (WAI 01.01.2011)

Year	Units	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	Total
Total Mined Tonnage	kt	2,313	2,137	2,196	1,710	822	899	420	393	386	328	352	200	118	**12,274**
Zinc Grade	% Zn	5.22	6.14	6.57	7.41	6.69	7.10	6.90	6.41	6.44	7.15	5.27	4.74	5.08	**6.34**
Copper Grade	% Cu	1.75	1.66	1.60	1.59	1.62	1.76	1.97	2.05	2.42	2.55	2.82	1.94	2.82	**1.78**
Lead Grade	% Pb	0.89	1.00	1.00	1.16	0.94	1.04	1.22	1.10	1.18	1.53	0.88	0.85	0.85	**1.02**
Gold Grade	g/t Au	0.42	0.48	0.42	0.64	0.50	0.56	0.67	0.55	0.71	0.77	0.82	0.48	0.63	**0.52**
Silver Grade	g/t Ag	68.07	58.03	49.82	58.55	55.97	56.58	77.36	78.25	79.81	69.70	68.14	59.65	59.42	**60.92**

The concentrator at Zyryanovskiy was constructed in 1953 and has undergone several stages of refurbishment. The plant utilises the standard processes of crushing, heavy media concentration (HMS) and froth flotation to produce copper, lead, zinc concentrates as well as a gold rich gravity concentrate. The plant went through a substantial upgrade in 2000 when all sections of the plant were refurbished and the grinding and flotation sections were renewed. The plant was originally designed to treat 1.5Mtpa but by 2001 the throughput had increased to 2.25Mtpa. The maximum plant throughput is reported to be 3.5Mtpa.

The plant treats ore from three sources:

- Maleevskoye mine copper-zinc and polymetallic ores;
- Grekovskoe mine – polymetallic ore; and
- "Foreign" ores.

All ore types contain copper, lead, zinc, silver and gold. The gold is present predominantly in the native form and can therefore be recovered using gravity processing methods.

The zinc concentrates are transported by rail to either the zinc smelters at Ridder or Ust-Kamenogorsk. The lead and gravity gold concentrates are sent to smelters in Ust-Kamenogorsk. The copper concentrates are shipped to customers in China. The company is in the process of constructing an IsaSmelt furnace in Ust-Kamenogorsk and this is scheduled for completion in 2011. This facility will be able to process the copper concentrates.

The metallurgical balance shows a high degree of metallurgical efficiency with low losses of metals to the HMS tailings and satisfactory grades of base metal concentrates at good recovery levels. The recovery of gold to the gravity concentrates was 37.1% with a further 34% reporting to the copper concentrate.

Copper recovery to the copper concentrate was 86.1% and the concentrate assayed 24.6% Cu, 3.63% Pb, 2.92% Zn, 2.63ppm Au and 572ppm Ag. The lead concentrate was rather low at 46% Pb, 2.3% Cu and 8.5% Zn at a lead recovery of 58.2%. The zinc concentrate assayed 55.4% Zn, 0.92% Cu and 0.46% Pb at a zinc recovery of 88.9%.

The average grade of ore treated was 1.98% Cu, 0.91% Pb, 6.28% Zn, 0.53ppm Au and 59.3ppm Ag.

For the most part, the Zyryanovskiy Complex and Maleevskoye Mine have a very high standard of environmental, social, health and safety management. It is compliant with Kazakh licences, permits and norms. Both the environmental and labour safety functions of the company are of high quality and well organised, and supported by a strong corporate policy, commitment and team.

There are no known social, community or cultural issues or impacts that need to be addressed, and no displacement or compensation requirements. The mine appears to have a good relationship with the local communities, and does not put undue pressure on the social infrastructure of the District or Oblast.

Ridder-Sokolniy

Kazzinc own and operate three polymetallic deposits within the Ridder Area including Ridder-Sokolniy, Tishinskiy, Shubinskiy, and are planning to develop a further three at Chekmar, Dolinnoe and Obruchevskoe.

The Ridder-Sokolniy deposit was discovered in 1786 and has been worked since 1789, initially exploiting silver and gold-bearing ore, before producing gold-polymetallic ore from surface and underground workings since 1920.

The deposit is situated in the northern part of the Ridder mining district in the Rudnyi Altay. Geologically, this district coincides with an unusual graben structure oriented perpendicular to the regional north west structural trend and preserving Silurian to Middle Devonian volcano-sedimentary sequences almost in the same position as they were laid down. The graben is filled with four volcano-sedimentary formations with the

Kriukovskaya Formation hosting most of the polymetallic mineralised ore. The deposit is a cluster of approximately 20 ore zones of veinlet-type and disseminated sulphide mineralised ore found at the same stratigraphic position within the Middle Devonian volcano-sedimentary Kriukovskaya Formation. Four other ore zones are known from deeper stratigraphic levels. The mineralised ore covers an area of approximately $20km^2$ and extends down to a depth of at least 700m.

The most distinguishing feature of the Ridder-Sokolniy mineralised ore is the dominance of zinc over copper and lead. The average ratio of Cu:Pb:Zn is 0.2:1:2.4, except in copper and zinc-copper veins where copper predominates. The other characteristic feature is the presence of substantial quantities of gold and, to a lesser degree, silver.

Significant exploration has been conducted at Ridder-Sokolniy and Kazzinc continues to investigate extensions to mineralisation both on the flanks and at depth to known deposits. In 2010 over 62,000m of drilling was completed from surface and a further 77,200m completed underground.

Ridder-Sokolniy has a long mining history with underground operations commencing in 1791. Originally there were three separate mines: '40 Years of VLKSM', 'Ridder' and 'Leninogorskiy', which eventually merged into one operation to enable a combined access to the ore. Mining operations have been conducted in the 17 major lodes and future plans are to develop the eastern flank of the Perspectivnaya Lode at the Bakhrushinsky deposit. The mine has an extensively developed underground infrastructure on 11 haulage levels and in 11 different deposits. The deposits are accessed by 12 shafts (10 in operation), with a maximum depth of 460m. The main hoisting shafts are Skipovaya (for Pb-Zn ore) and Novaya (for Cu ore and development waste).

The mine has a staffing of approximately 1,500 and most of the production is performed using hand-held equipment and scrapers and the main haulage is tracked. Trackless methods have been approached recently, and are now focused in the Pobeda area (narrow high-grade gold vein) in the southern part of the mine. It is proposed to implement 30% trackless mining machinery (especially in development and haulage) in 2011 and to increase utilisation to 50% of overall production by 2013. The majority of the ore is extracted using Sub-Level Open Stoping method or Open Stoping method, which are relatively similar. Where conditions require, Sublevel Open Stoping with Backfill is used to prevent caving of empty stopes or where the risk of potential damage to buildings on surface exists. Shrinkage stoping (<5% of production) is applied in thin steeply dipping ore bodies.

The Ridder mining and metallurgical complex employs some 600 staff and treats ore from several sources including Ridder-Sokolniy mine, Tishinskiy mine, Shubinskiy mine, Staroye tailings, Aged and Fresh slimes from crushing, and Shaimerden zinc oxide and other oxide ores. The latter is transported some 1,200km by rail. All but the zinc oxide ores are initially sent for crushing at one of three crushing facilities with one ore type, a high gold grading silica rich ore from the Ridder-Sokolniy mine, dispatched to Ust-Kamenogorsk (smelter) directly after crushing. The remaining ores are further treated by crushing, milling, gravity and froth flotation in one of two concentrator buildings.

The copper concentrate is sold to a Chinese smelter and the lead concentrate is dispatched to the Kazzinc lead smelter in Ust-Kamenogorsk. The gold concentrate is also dispatched to Ust-Kamenogorsk where it is fed into the lead smelter. The zinc concentrate is transported by truck a few kilometres to the Kazzinc zinc smelter (located in Ridder) and several other ores and concentrates, including Shaimerden and other oxide ores, are also treated at the zinc smelter. Tailings from the complex are pumped some 5km to the Talovskoye TMF that has a remaining capacity of 50Mt.

There has been mining activity at Ridder since 1786, via surface and underground workings, with concentration facilities since the 1920s. Prior to 1991, mining and processing at Ridder-Sokolniy was State owned and operated; Kazzinc started operating in 1997. The mineral rights for Ridder-Sokolniy mine remain State owned, and operations take place under the Subsoil Use Contract with the government. Kazzinc is responsible only for liabilities generated via their current operations and does not have any liabilities associated with contaminations prior to 1997.

Mineral Resources for individual Ridder-Sokolniy ore zones have been calculated in accordance with the guidelines of the JORC Code (2004) and are shown in the tables together with a summary below.

Centralny Resource Estimate (WAI 01.01.11) Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore and 0.6% CuEQV for Cu Ore (In Accordance with the Guidelines of the JORC Code (2004))											
Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	20.09	1.00	648,768	8.43	5,446,956	0.9	18,020	0.31	61480	0.8	160,800
Indicated	36.21	1.23	1,437,071	8.46	9,847,050	0.44	15,950	0.36	130750	0.85	308,310
Measured + Indicated	56.30	1.15	2,085,839	8.45	15,294,006	0.60	33,970	0.34	192,230	0.83	469,110
Inferred	1.53	0.83	40,763	3.74	183,365	0.45	690	0.15	2320	0.71	10,880

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Bystrushinskoe Resource Estimate (WAI 01.01.11) Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore and 0.6% CuEQV for Cu Ore (In Accordance with the Guidelines of the JORC Code (2004))											
Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
Measured + Indicated	16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
Inferred	0.04	1.53	1,768	3.44	3,922	0.18	63	0.15	53	0.33	118

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

North Bystrushinskoe Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore and 0.6% CuEQV for Cu Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
Measured + Indicated	1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
Inferred	0.09	2.46	7,106	6.90	19,997	0.86	776	0.29	264	1.03	925

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Belkina Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	3.76	1.26	152,331	18.98	2,294,630	0.09	3,384	0.47	17,670	0.98	36,850
Indicated	4.77	1.11	170,126	16.85	2,582,538	0.08	3,814	0.41	19,550	0.85	40,520
Measured + Indicated	8.53	1.18	322,456	17.79	4,877,168	0.08	7,200	0.44	37,220	0.91	77,370
Inferred	0.52	0.88	14,669	23.42	390,393	0.28	1,450	0.32	1,660	0.90	4,670

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Perspectivnaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	4.59	1.55	228,476	16.01	2,359,940	0.13	5,960	0.57	26,130	1.07	49,060
Indicated	3.94	1.36	172,076	16.79	2,124,381	0.13	5,120	0.53	20,860	1.03	40,530
Measured + Indicated	8.52	1.46	400,552	16.37	4,484,321	0.13	11,080	0.55	46,990	1.05	89,590
Inferred	2.03	1.19	77,607	12.08	787,806	0.14	2,840	0.49	9,940	0.95	19,270

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Sokolok Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
Measured + Indicated	0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
Inferred	0.37	1.40	16,715	15.27	182,315	0.14	520	1.48	5,500	1.98	7,350

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

New Glubokaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
Measured + Indicated	0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
Inferred	0.07	0.14	314	2.68	6,004	0.31	220	0.74	520	2.10	1,460

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Glubokaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	-	-	-	-	-	-	-	-	-	-	-
Measured + Indicated	-	-	-	-	-	-	-	-	-	-	-
Inferred	0.51	0.10	1,640	1.48	24,277	0.08	410	0.83	4,230	1.91	9,740

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Dalnaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
Measured + Indicated	3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
Inferred	1.16	0.18	6,855	3.18	118,984	0.51	5,958	0.40	4,620	2.07	24,083

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Zavodskaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	1.78	0.82	46,922	22.60	1,293,223	0.12	2,111	0.79	14,105	1.53	27,187
Indicated	0.51	0.97	15,816	21.62	352,524	0.10	509	0.67	3,397	1.29	6,523
Measured + Indicated	2.29	0.85	62,739	22.38	1,645,747	0.12	2,620	0.76	17,502	1.48	33,710
Inferred	0.33	1.00	10,567	21.10	222,961	0.14	459	0.60	1,958	1.21	3,991

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Kazzinc Mineral Resources Ridder-Sokolniy Mine (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004)

Deposit	Resources	Cut Off Grade	Date	Tonnes (Mt)	Gold (Au) Grade (g/t)	Gold (Au) Metal Content (oz)	Silver (Ag) Grade (g/t)	Silver (Ag) Metal Content (oz)	Copper (Cu) Grade (%)	Copper (Cu) Metal Content (t)	Lead (Pb) Grade (%)	Lead (Pb) Metal Content (t)	Zinc (Zn) Grade (%)	Zinc (Zn) Metal Content (t)
Centralniy	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01-Jan-11	20.09	1.00	648,768	8.43	5,446,956	0.9	18,020	0.31	6,1480	0.8	160,800
	Indicated			36.21	1.23	1,437,071	8.46	9,847,050	0.44	15,950	0.36	130,750	0.85	308,310
	Measured + Indicated			56.30	1.15	2,085,839	8.45	15,294,006	0.60	33,970	0.34	192,230	0.83	469,110
	Inferred			1.53	0.83	40,763	3.74	183,365	0.45	690	0.15	2,320	0.71	10,880
	Total			57.83	1.14	2,126,602	8.33	15,477,371	0.60	34,660	0.34	194,550	0.83	479,990
Bystrushinskoe	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
	Measured + Indicated			16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
	Inferred			0.04	1.53	1,768	3.44	3,922	0.18	63	0.15	53	0.33	118
	Total			17.02	2.72	1,488,032	12.28	6,715,069	0.28	48,127	0.42	71,563	0.90	152,930
Bystrushinskoe North Flank	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
	Measured + Indicated			1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
	Inferred			0.09	2.46	7,106	6.90	19,997	0.86	776	0.29	264	1.03	925
	Total			1.84	1.31	77,902	5.69	337,325	1.01	18,666	0.29	5,404	1.07	19,793
Belkina_RSM_WA	Measured	1.7% ZnEq	01-Jan-11	3.76	1.26	152,331	18.98	2,294,630	0.09	3,384	0.47	17,670	0.98	36,850
	Indicated			4.77	1.11	170,126	16.85	2,582,538	0.08	3,814	0.41	19,550	0.85	40,520
	Measured + Indicated			8.53	1.18	322,456	17.79	4,877,168	0.08	7,200	0.44	37,220	0.91	77,370
	Inferred			0.52	0.88	14,669	23.42	390,393	0.28	1,450	0.32	1,660	0.90	4,670
	Total			9.05	1.16	337,125	18.11	5,267,561	0.10	8,650	0.43	38,880	0.91	82,040
Perspectivnaya	Measured	1.7% ZnEq	01-Jan-11	4.59	1.55	228,476	16.01	2,359,940	0.13	5,960	0.57	26,130	1.07	49,060
	Indicated			3.94	1.36	172,076	16.79	2,124,381	0.13	5,120	0.53	20,860	1.03	40,530
	Measured + Indicated			8.52	1.46	400,552	16.37	4,484,321	0.13	11,080	0.55	46,990	1.05	89,590
	Inferred			2.03	1.19	77,607	12.08	787,806	0.14	2,840	0.49	9,940	0.95	19,270
	Total			10.55	1.41	478,159	15.55	5,272,127	0.13	13,920	0.54	56,930	1.03	108,860
Sokolok	Measured	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
	Measured + Indicated			0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
	Inferred			0.37	1.40	16,715	15.27	182,315	0.14	520	1.48	5,500	1.98	7,350
	Total			0.67	1.38	29,554	61.94	1,323,123	0.14	900	1.27	8,440	1.94	12,900
New Glubokaya	Measured	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
	Measured + Indicated			0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
	Inferred			0.07	0.14	314	2.68	6,004	0.31	220	0.74	520	2.10	1,460
	Total			0.39	0.17	2,181	2.51	31,727	0.10	410	0.68	2,680	1.85	7,270

Glubokaya	**Measured**	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			-	-	-	-	-	-	-	-	-	-	-
	Measured + Indicated			-	-	-	-	-	-	-	-	-	-	-
	Inferred			0.51	0.10	1,640	1.48	24,277	0.08	410	0.83	4,230	1.91	9,740
	Total			0.51	0.10	1,640	1.48	24,277	0.08	410	0.83	4,230	1.91	9,740
Dalnaya	**Measured**	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
	Measured + Indicated			3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
	Inferred			1.16	0.18	6,855	3.18	118,984	0.51	5,958	0.40	4,620	2.07	24,083
	Total			4.30	0.27	37,959	3.41	471,535	0.63	27,239	0.27	11,595	2.24	96,451
Zavodskaya	**Measured**	1.7% ZnEq	01-Jan-11	1.78	0.82	46,922	22.60	1,293,223	0.12	2,111	0.79	14,105	1.53	27,187
	Indicated			0.51	0.97	15,816	21.62	352,524	0.10	509	0.67	3,397	1.29	6,523
	Measured + Indicated			2.29	0.85	62,739	22.38	1,645,747	0.12	2,620	0.76	17,502	1.48	33,710
	Inferred			0.33	1.00	10,567	21.10	222,961	0.14	459	0.60	1,958	1.21	3,991
	Total			2.62	0.87	73,305	22.22	1,868,709	0.12	3,079	0.74	19,460	1.44	37,701
Ridder-Sokolny Mine	**Measured**			30.21	1.11	1,076,497	11.73	11,394,749	0.64	29,475	0.40	119,385	0.91	273,897
	Indicated			67.91	1.55	3,397,959	10.74	23,454,050	0.38	113,198	0.40	269,542	0.95	645,031
	Measured + Indicated			98.12	1.42	4,474,456	11.05	34,848,799	0.46	142,675	0.40	388,927	0.94	918,928
	Inferred			6.64	0.83	178,004	9.09	1,940,024	0.29	13,386	0.59	39,365	1.12	74,187
	Total			104.76	1.38	4,652,459	10.92	36,788,824	0.45	156,061	0.41	428,292	0.95	993,115

Ore Reserves for individual ore zones at Ridder-Sokolniy have been calculated in accordance with the guidelines of the JORC Code (2004) and are shown in the table below.

Ridder-Sokolniy Ore Reserve Estimate
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Deposit	Reserves	Ore (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
			Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Centralny	Proven	5.02	0.71	114,830	5.19	837,614	0.69	34,566	0.18	9,076	0.52	26,208
	Probable	9.17	1.03	303,446	5.95	1,756,434	0.26	23,699	0.34	31,365	0.64	58,683
	Total	**14.19**	**0.92**	**418,276**	**5.69**	**2,594,048**	**0.41**	**58,264**	**0.28**	**40,440**	**0.60**	**84,891**
Belkina	Proven	2.24	0.98	70,492	25.38	1,826,106	0.07	1,541	0.45	9,960	0.93	20,814
	Probable	1.22	0.82	31,972	13.16	514,297	0.06	672	0.29	3,499	0.61	7,369
	Total	**3.46**	**0.92**	**102,463**	**21.08**	**2,340,403**	**0.06**	**2,213**	**0.39**	**13,459**	**0.82**	**28,183**
Perspectivnaya	Proven	1.45	1.54	71,946	18.55	864,235	0.11	1,632	0.52	7,501	0.97	14,126
	Probable	0.83	1.33	35,682	13.82	369,402	0.09	785	0.37	3,086	0.69	5,711
	Total	**2.28**	**1.47**	**107,628**	**16.82**	**1,233,637**	**0.11**	**2,416**	**0.46**	**10,587**	**0.87**	**19,837**
Zavodskaya	Proven	0.24	0.72	5,511	21.73	165,757	0.13	299	0.86	2,050	1.71	4,047
	Probable	0.03	1.56	1,278	19.65	16,120	0.10	26	0.70	179	1.28	326
	Total	**0.27**	**0.80**	**6,789**	**21.53**	**181,876**	**0.12**	**324**	**0.85**	**2,228**	**1.66**	**4,373**
North Bestrushinskoye	Proven	-	-	-	-	-	-	-	-	-	-	-
	Probable	0.80	1.05	26,971	23.76	612,977	0.12	971	0.85	6,791	1.54	12,387
	Total	**0.80**	**1.05**	**26,971**	**23.76**	**612,977**	**0.12**	**971**	**0.85**	**6,791**	**1.54**	**12,387**
Total	***Proven***	*8.95*	*0.91*	*262,779*	*12.85*	*3,693,712*	*0.43*	*38,038*	*0.32*	*28,587*	*0.73*	*65,195*
	Probable	*12.05*	*1.03*	*399,349*	*8.44*	*3,269,230*	*0.22*	*26,153*	*0.37*	*44,920*	*0.70*	*84,476*
	Total	***21.00***	***0.98***	***662,127***	***10.32***	***6,962,941***	***0.31***	***64,188***	***0.35***	***73,505***	***0.71***	***149,671***

Tishinskiy

The Tishinskiy deposit is situated 18km south west of Ridder. The Tishinskiy mine is linked by rail and asphalt road, which lies adjacent to the mine with Ridder and Ust-Kamenogorsk.

The Tishinskiy deposit was discovered in 1958. Preliminary and detailed exploration of the new discovery was completed in mid-1963. Mine development and detailed underground exploration began in July 1963. Whilst underground work was in progress, drilling from surface continued until the end of 1983 focusing primarily on deep levels of the deposit. The central part of the Main Lode and its eastern flank were delineated to depth of 1,200m and the western flank to a depth of 900m (Tishinsky Shaft collar being at 644m).

The Tishinskiy deposit is situated in the central portion of the Butachihinsko-Kedrova shear zone, which adjoins the south-western flank of the Ridder graben. The Palaeozoic host rocks have undergone strong polyphase folding and intense shearing. The deformed formations comprise high grade metamorphic Ordovician basement rocks and weakly metamorphosed volcanogenic-sedimentary Devonian-Upper Carboniferous rocks.

Tishinskiy is a stratabound deposit developed along the E-W trending subvertical contact between the Ilinskaya and Sokolnaya Formations, both of Middle Devonian (Eifelian) age.

The Main Orebody, also known as Orebody No 1 extends over a strike length of 1,250m from surface to a depth of 1,270m corresponding to Level 22 at the absolute elevation of minus 590m. The central part of this body, about 500m in strike length and up to 60m in width, contains three subvertical lenses of massive sulphides (Western, Central and Eastern) with a combined strike length of about 200m, enveloped in disseminated mineralised ore. The Western and Central lenses peter out downwards on Levels 14 to 18 at the absolute elevations of -110m and -350m respectively and the Eastern lens pinches out at about the zero datum (10m below Level 12). Average widths range from 6.5m to 17m. Lower grade disseminated mineralised ore with much reduced widths (generally less than 10m) occurs in the Western Shaft section of the mine on the western flank of the deposit above Level 8 and on the eastern flank, where it peters out rapidly towards a major transverse fault.

Orebody No 67 has a strike length of 1,000m, dip extent of 700m and an average width of 3.7m. Orebody No.1011 has a strike length of 550m, dip extent of 400m and an average width of 1.7m. On its flanks, the Main Lode divides into a number of parallel tapering branches to the east and west and gradually fades out. Lenses and small vein-like bodies on the flanks of the deposit range from 25m to 50m in strike length and from 40m to 150m in down dip extent.

Four primary and two secondary mineral associations have been recognised at the Tishinskiy deposit. The prevalent associations are: (1) A polymetallic (chalcopyrite-galena-pyrite-sphalerite) association, and (2) A copper-zinc (chalcopyrite-sphalerite-pyrite) association. Disseminated veinlet-type polymetallic mineralised ore predominates, but massive sulphide lenses of various sizes occur in the central parts of mineralised bodies.

The resources for the Tishinskiy deposit are classified in accordance with the guidelines of Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves JORC Code (2004). Criteria for defining resource categories were also derived from the geostatistical studies.

The grades in the final resource model were derived from the Ordinary Kriging estimation method for all elements, using a cut–off grade of 2.2% Zn equivalent:

Tishinskiy Mineral Resource Estimate - At COG of 2.2% ZnEqv (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))													
Classification	**Tonnes (Mt)**	**Density**	**Zn**		**Pb**		**Cu**		**Au**		**Ag**		**ZNEQV**
			%	**Kt**	**%**	**kt**	**%**	**kt**	**g/t**	**oz**	**g/t**	**oz**	**%**
Measured	21.23	3	4.7	1,000	1.0	212	0.6	125	0.60	412,000	9.02	6,154,000	7.4
Indicated	7.01	3	4.3	305	0.9	66	0.4	31	0.46	104,000	9.75	2,072,000	6.3
Inferred	5.19	3	4.5	231	1.4	70	0.6	29	0.33	55,000	11.94	2,380,000	9.4

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Mining operations at the Tishinskiy mine started in 1975 using both surface and underground mining methods. To date, approximately 49Mt of ore have been extracted (including open pit ore).

Historically, the main mining method employed in the underground mine has been underhand open stoping with hydraulic cemented backfill. Trackless mining equipment is employed for the development and stoping activities, but a track haulage system is used to transport ore and waste from drawpoints, and ore passes to the main hoisting shaft. The current ore production rate is between 1,300-1,400ktpa.

The mining method utilises jumbo development, electric over hydraulic production drilling and diesel load haul dump (LHD) loaders. The mine has its own backfill preparation plant, where backfill is mixed and pumped to a pipeline connected to the underground infrastructure. From this pipeline the backfill is distributed to stopes via backfill boreholes (normally one for the backfill, and an additional borehole for control) and pipeline extensions.

With depletion of the ore reserves from the central part of the deposit, the mine will be developed towards its flanks, and towards the bottom. Current exploration programmes are aimed at bringing the levels between 18 Level to 22 Level into production. An additional resource estimation study is currently being conducted for the eastern flank of the deposit, with production from this area due to commence in 2012.

WAI has performed an evaluation of the Tishinskiy mine Ore Reserves in accordance with the requirements of the JORC Code (2004). The evaluation has been performed using a geological block model produced by WAI dated 01.01.2011 and considers modifying factors, such as dilution and losses, with respect to the current mining operations and methods.

The stope boundaries were generated to correspond with the standardised dimensions of the existing stopes and development in the mine, and also to be above the minimum cut-off grade for exploitation.

A total of 2,725 stopes was produced to justify the Tishinskiy ore reserves. The greatest part of the reserve is contained in the central part of the deposit, but there is a significant portion of reserves located in the western part of the deposit. A summary of ore reserves is given in the table below.

Tishinskiy Ore Reserve Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))											
	Ore	Zinc		Copper		Lead		Gold		Silver	
	kt	%	kt	%	kt	%	kt	g/t	kg	g/t	kg
Proven	18,886	4.22	797	0.52	98	0.91	172	0.54	10,198	8.12	153,353
Probable	4,928	4.13	204	0.4	20	0.88	43	0.47	2,316	9.36	46,124
Total Proven and Probable	***23,814***	***4.20***	***1,001***	***0.50***	***118***	***0.90***	***215***	***0.53***	***12,514***	***8.38***	***199,477***
Inferred material within stopes	*156*	*11.17*	*17*	*0.45*	*1*	*1.2*	*2*	*0.45*	*70*	*11.05*	*1,728*

Tishinskiy mine is a mature operation. Despite relatively unfavourable rock conditions, resulting in the requirement for additional ground support, mining is performed to a high standard. The utilisation of modern western underground mining equipment and qualified personnel provide confidence that production targets are being met. The remaining Ore Reserve together with available Mineral Resources estimated by both internal and independent sources, show long-term potential for this operation.

Kazzinc's environmental management and monitoring are performed in line with national requirements, and Kazzinc has achieved accreditation under international environmental, quality and health and safety management systems. Further reviews of requirements to achieve international compliance should be undertaken. These should include a review of monitoring and closure request. A key area for attention is water recycling and management.

In 2010 an independent environmental audit was conducted at Kazzinc including Tishinskiy Mine. During the audit all environmental aspects, risks, targets and environmental management programmes applied by these operations and the efficiency of planned impact mitigation actions were assessed. The first step in moving towards best practice, would be to review all procedures against international standards.

Shubinskiy

The Shubinskiy deposit is situated in the Glubokovskoe region of East Kazakhstan province 14km north-east of Ridder from which it is readily accessible by an all weather graded road.

The mine development consists of five underground levels connecting two vertical shafts located on the north-western and south-eastern flanks of the deposit.

The deposit covers an area of 600*110m extending from north west to south east along the contact of the Uspenskaya Formation, comprising volcanogenic-sedimentary rocks and the Beloubinskaya Formation terrigenous sediments.

The deposit consists of two clusters of sub-parallel, tabular and lenticular mineralised bodies, the North-Western Lode and the South-Eastern Lode respectively.

Five ore bodies and 24 lenses of pyritic polymetallic mineralisation have been identified at Shubinsky. The majority of the reserves (78.7%) are concentrated in the No.1 and No.2 ore bodies, and 63.9% in these bodies in the NW Lode where the grades are higher.

The ore bodies and lenses themselves, dip 65-80° NE and strike 320-340°. They vary from 100-130m to 200-220m in length, extend down dip from 190m to 500-725m with widths between 0.26-0.40m to 21-28m and average values from 1.6-9.0m.

The primary sulphide and mixed zones are defined by the degree of oxidation.

In the oxide zone, base metal grades are low and for this reason the oxide zone is not included in the reserves.

Gold grade is usually low (average 0.2-0.6g/t), although higher values have been recorded occasionally, in the upper part of the deposit.

Underground production began at Shubinskiy with trial mining in 1990/1991 and continued intermittently at low output rates for over six years. In 1999 the production exceeded 100,000t and has since risen to reach its full capacity of 200,000tpa in 2009. During the period 1992-2009, the mine produced 1.6 Mt of ore grading 0.66 g/t Au, 17.9 g/t Ag, 1.74% Cu, 0.28% Pb and 2.25% Zn. The mine is operated by TOO Shubinskoye, who had worked as Kazzinc's mining contractors prior to April 2009.

Ore is extracted from the stopes on the 1-5 Levels, it is loaded in rail wagons and hauled by electric locomotives to the hoisting shaft. The ore is hauled by road trucks to the Ridder processing plant, where it is stockpiled for separate treatment.

The mining method is chamber mining with backfill and is used down to 5 Level. It employs conventional handheld pneumatic machine drift development and stope production drilling, electric scraper mucking and electric locomotive haulage. Drifting is carried out with handheld portable rock drills using compressed air. Blast holes for the stopes are drilled with handheld drill rigs using compressed air. In the stoping operations, the ore from the drifts and stopes is mucked using 30LC-2CM and 55LC-2CM scraper winches.

The chamber mining with backfill method is also known as sublevel open stoping with classified cemented tailings backfill. The extraction sequence is top-down (under hand), working from the centre of each lode to the strike flanks.

It is intended that mining below 5 Level will be via a decline and will use rubber tyred diesel mining equipment.

The Shubinsky mine uses a zinc equivalent (ZnEq) cut-off for the polymetallic lodes. The ZnEq grade is calculated as Zn% + 0.4Pb% + 1.7Cu%. The cut-off grades are based on historic metal prices used for the 1989 TEO for the Shubinsky project. The ZnEq cut-offs are unlikely to be optimal, from an economic perspective, for 2010 costs, metal prices and process terms.

The mining reserves currently used at the Shubinskiy mine site have been estimated using conventional Soviet methods. At present no Mineral Resources or Ore Reserves have been prepared for Shubinskiy in accordance with the guidelines of the JORC Code (2004).

Staroye (Old) Tailings Management Facility (TMF)

The Staroye tailings dam, adjacent to the Ridder-Sokolniy Mine, was in operation from 1926 to 1953 inclusive accepting material from the Leninogorsk Polymetallic Concentrator (LPC). The feed during this period was derived exclusively from processing polymetallic ore of the Ridder-Sokolny deposit. Kazzinc commenced a reprocessing operation in 1991 that by September 2010 had reprocessed almost 6Mt returning an average head grade (from concentrator reports) of 1.41g/t Au, 12.70g/t Ag, 0.08% Cu, 0.31% Pb and 0.79% Zn. The process involves a two stage grind followed by a gravity flotation scheme with the recovery of gravity concentrate, auriferous concentrate and zinc concentrate.

WAI has performed a resource and reserve evaluation for Staroye TMF in accordance with the guidelines of the JORC Code (2004). The details of this evaluation are given in tables below.

Staroye Tailings Mineral Resource Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))							
Category	Tonnes	Au	Ag	Cu	Pb	Zn	AuEq
	t	g/t	g/t	%	%	%	g/t
Indicated	824,258	2.01	18.78	0.05	0.48	1.11	3.19
Inferred	5,899,119	0.91	11.16	0.04	0.30	0.63	1.72

Notes:

- AuEq calculation based on prices of

  Au 1287 US$/oz

  Ag 23 US$/oz

  Cu 7341 US$/t

  Pb 2422 US$/t

  Zn 2420 US$/t

WAI has calculated Ore Reserves based upon the above Mineral Resource Estimate. WAI has applied losses of 4.4% based upon historical data, and dilution of 0.5% to take into account minimal dilution from the bunds and floor.

Staroye Tailings Ore Reserve Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))											
Reserves	Ore (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (koz)	Grade (g/t)	Metal Content (koz)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)
Proven	-	-	-	-	-	-	-	-	-	-	-
Probable	0.79	2.00	50.97	18.69	475.89	0.05	0.38	0.48	3.80	1.10	8.72
Total	***0.79***	***2.00***	***50.97***	***18.69***	***475.89***	***0.05***	***0.38***	***0.48***	***3.80***	***1.10***	***8.72***

Ridder Smelter

Kazzinc operates integrated metallurgical complexes in Ust-Kamenogorsk and Ridder in Eastern Kazakhstan. At Ust-Kamenogorsk there is a currently operational lead plant (traditional sinter plant-blast furnace-refinery); a conventional electrolytic zinc plant; sulphuric acid plants and a precious metals refinery. At the time of the site visits (October 2010) a copper Isa-smelt furnace and tank house; a lead Isa-smelt furnace and a dual contact sulphuric acid plant were under construction with commissioning and production scheduled for 2011. At Ridder there is an operational, conventional electrolytic zinc plant with associated conventional sulphuric acid plant.

In 2009 at Ust-Kamenogorsk lead metal production was almost 75kt, although the plant can and has produced over 100kt per annum and zinc production (metal and zinc alloy) was over 190kt. The precious metals refinery can produce over 170,000kg silver and 18,500kg gold. The Ridder zinc refinery utilises essentially the same process flowsheet as the plant at Ust-Kamenogorsk and produced over 110kt saleable zinc, in 2009.

The lead Isa-smelt furnace under construction currently will eventually replace the existing Dwight-Lloyd sinter plant. The copper Isa-smelt furnace and tank house will offer potential for Kazzinc to become a low cost copper producer and provide opportunities to toll smelt third party copper ores/concentrates.

Shaimerden

The Shaimerden structure was discovered in 1956 during prospecting for bauxite and in 1989 a programme to investigate scandium levels in the bauxite deposits revealed anomalous lead and zinc concentrations in the Shaimerden area.

In 1996 the deposit was taken over by the Zinc Corporation of Kazakhstan (ZCK) and in 2001 ZincOx acquired a 95% interest which, due to the deteriorating market conditions, was sold to Kazzinc in 2003.

ZincOx had envisaged mining over 17 years averaging 250,000t/year whilst Kazzinc planned to complete the development of the same volume within 5.5 years (averaging up to 1Mt/year). Latest projections indicate that stockpiled ore will provide raw material to the Ridder Zinc Plant until 2019.

Shaimerden is located some 200km SSW of the city of Kostenay in northern Kazakhstan with access via metalled roads, initially of a reasonable standard comprising a four lane highway to Rudnay but deteriorating thereafter, whilst rail links are excellent, connecting the Kostenay district with Russia and the rest of Kazakhstan at Tobol and Arka stations, and there also exists a short spur to the mine loading site.

Shaimerden is located on the eastern side of the north-south trending transcontinental Urals Orogenic belt within the 700km long Valerianovsky belt, characterised by Carboniferous sediments and volcanics, and the southern end of the Kostenay megasyncline and on the eastern limb of the Krasnoktyabr syncline.

The deposit is hosted in a massive, clean Carboniferous limestone and has resulted from the in-situ oxidation during the Triassic-Cretaceous period of a body of massive sphalerite mineralisation. The deposit occurs within a weathered depression measuring 450m E-W and 150m N-S with mineralisation occurring to a depth of 240m below surface.

Emplacement of the orebody is thought to have resulted from the circulation of ore-bearing geothermal solutions in the period after deposition of the rocks and prior to a period of Triassic weathering.

The main Shaimerden orebody comprises supergene oxidised zinc ores with remnant sulphides towards the centre, indicating oxidation from the outside margins to the centre of the deposit. There is an outward zoning comprising five categories:

- Massive sulphides – (2.8% of the reserves) preserved in the centre of the orebody and consisting mainly of remnant massive sulphides-90% sphalerite with minor galena and pyrite and an average grade of 46% Zn, 1.2% Pb, 6.1g/t Ag;
- Massive hemimorphite-smithsonite – (11.7% of the reserve) surrounding and intimately related to the massive sulphides. This ore type assays 30 to 45% Zn, with an average resource grade of 35.2% Zn;
- Stony ore – (8.2% of the reserve) dark green to dark red weathered, with a relict breccia texture. It is an intermediate weathered material between the massive hemimorphite-smithsonite and the mineralised clays;
- Mineralised clays – characterised by their grey/green and brown colour and high percentage of gritty and fragmental material, comprising CL3 - (38.1% of the reserve) with grades averaging 24.9% Zn, CL2 - (28.8% of the reserve) with grades averaging 13.1% Zn and CL1 - (4.5% of the reserve) with grades averaging 1.8% Zn; and
- Mineralised limestone – (5.9% of the reserve) occurs along the margin of the weathered trough and consists of intervals of mineralised clays within the limestone.

115 boreholes were drilled to investigate the deposit itself; the grid was 25-50x12.5-25m for the C_1 category reserves and 50 x 100m for the C_2 category:

The Shaimerden mine has been in production since 2005, but is expected to cease mining in May 2011. The near surface zinc orebody has meant it is suitable for conventional truck and shovel open pit mining. Zinc ore is mined from a single pit, hauled to stockpiles, crushed and transported by rail to Ridder in east Kazakhstan.

All material with a Zn grade less than 2% is hauled to the waste dump.

The remaining 'in pit' reserves at the time of the WAI visit (October 2010) (estimated using a mine based Datamine® programme) Comprises 883,842t @21.05% Zn. At the time of the issue of this report, WAI believes that the in-pit reserves will be exhausted or near to exhaustion.

The remaining Shaimerden Resources and Reserve Estimate in accordance with the guidelines of the JORC Code (2004) will thereafter be those of the dry stockpiles which equate to a total of 1,631kt with a grade of 22.33% Zn containing 364.1kt of metal (WAI 01.01.2011).

Shaimerden Total Stockpile Mineral Resource Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004)			
Category	**Tonnage (t)**	**Grade (% Zn)**	**Metal (t)**
Indicated	2,483.86	21.71	539,143

Shaimerden Total Stockpile Ore Reserve Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004)			
Category	**Tonnage (t)**	**Grade (% Zn)**	**Metal (t)**
Probable	2,483.86	21.71	539,143

Novoshirokinskoye

The Novoshirokinskoye underground gold-polymetallic mine is located in East Zabaykal in the Gazimuro region of Chita province, Russian Federation. Transport to the mine is via a 10km long un-metalled road which joins the Sretensk - Nerchinsk provincial highway. The total distance to the railway siding at Priiskovaya where concentrates are loaded for transport to Ust-Kamenogorsk is some 230km.

The Novoshirokinskoye area is characterised by a sequence of Jurassic sediments folded into a synclinal structure with variable dip. Sediments are cut by many faults with hydrothermal alteration and brecciation associated with their development.

Economic mineralisation at Novoshirokinskoye is constrained within broader zones of hydrothermal alteration, which in turn are related to several fracturing systems. Mineralisation within the alteration zones is seen as lensoid and can be continuous over several hundred metres along strike and has been traced to over 900m in depth where the structure remains open in some parts of the system.

Orebodies are generally oxidised in their upper parts (where close to surface), followed by a thin transition zone grading into sulphide where the predominant economic mineralisation comprises galena and sphalerite with associated gold and silver.

The Main ore zone is the most persistent, and in the central part, 16 orebodies have been identified. In the footwall of the Main orebody are found orebodies 5, 11, 7, 4, 3, 2, 1, 13, 14 and 16, whilst in the hangingwall 8, 6 and 12 are located.

The Novoshirokinskoye tectonic zone is characterised by strong metasomatism of quartz-micaceous-dolomite composition. The width of the metasomatic zone varies from 20-300m, and has been traced down-dip for more than 750m. Ore mineralisation is concentrated in the centre of the zone.

Disseminated mineralisation which is developed along the systems of small fractures predominates, whilst brecciated mineralisation is also common as is massive ore which is found within the breccias.

Overall, the Novoshirokinskoye zone can be considered a linear stockwork with a strike varying from 255-335°, and dipping generally to the southwest at angles from 30-90°. The zone extends for 50-1,450m along strike, and from 40-760m (Main orebody) down-dip, and economic mineralisation has been proven down to the 450m level.

The principal orebodies within the Novoshirokinskoye mineralised area are Main, 5 and 7. The Main orebody typically pinches and swells, but in the centre is persistent over considerable distance, with an overall length of 1,330m, maximum down-dip depth of 760m, strike of 290-300°, and dip of between 33-82°. Thickness varies from 0.20 to 22.1m. At a 3g/t Au equivalent cut-off grade, the average thickness is 4.52m.

Orebody 7 is the second largest and occurs in the central and southeastern part of the deposit on the footwall of the Main and Orebody 5. The orebody thins to the southeast and northwest both along strike and down-dip. At the surface, the length of the orebody is some 480m, whereas at the 700m level, strike length approaches 1,510m, with a maximum down-dip extension of 460m. The general dip is to the southwest, varying from 54-80°, and averaging 65°. Thickness varies from 0.26-15m, with an average thickness at a 3g/t Au equivalent cut-off grade of 2.73m.

Orebody 5, which is the third most important orebody at Novoshirokinskoye, lies in the central part of the area, in the footwall of the Main orebody, some 5-50m away from it, although sometimes merging, although down dip, the orebodies become more separate.

The maximum strike length (270-335° azimuth) of Orebody 5 is some 1,060m, and has been traced down-dip for some 775m. Thickness is highly variable from 0.19 – 18.15m, with an average of 3.04m using the 3g/t Au equivalent cut-off grade.

More than 60 ore and vein minerals have been identified at Novoshirokinskoye. Principal mineralogy comprises galena, pyrite and sphalerite, with hydroxides of iron and manganese, lead ochres, quartz, dolomite and sericite. The typical average grades are 3.72% lead, 1.79% zinc, 3.51g/t Au and 96.91g/t Ag.

In November 2010, WAI produced a new resource and reserve estimate for five of the ore bodies (Main, 5, 7, 8 and 16), prepared in accordance with JORC (2004) guidelines and updated as of 01.01.2011.

The resource estimate for Novoshirokinskoye based on Inverse Power Distance Cubed (IPD^3) method is given below.

Novoshirokinskoye Mineral Resource Estimate at COG of 3g/t Au_{eq}
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

JORC Classification	Volume (m^3)	Ore Tonnes (t)	Density (t/m^3)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
Measured	835,326	2,425,050	2.90	4.43	87.74	3.43	1.47
Indicated	1,603,523	4,636,718	2.89	4.30	94.82	3.07	1.15
Measured + Indicated	2,438,849	7,061,767	2.89	4.34	92.39	3.19	1.26
Inferred	525,673	1,510,303	2.87	2.08	57.02	2.44	1.81

Mining operations at Novoshirokinskoye mine started relatively recently with industrial scale production first achieved early in 2010. Currently the mine is producing around 35-38kt of ore per month, targeting 450kt annual production.

The original mine design dates to the late 1960's when the deposit was intensively studied and pilot underground mine design took place. In 2004 an updated design was performed, considering the requirement for rehabilitation works (as the underground workings remained abandoned for some time) and modernised infrastructure facilities. According to this design, the mine is accessed via two shafts. One of the shafts "Skiovaya" is used to hoist the ore (In skips), whilst the other "Kletievaya" is used for man, materials and waste hoisting (cage), providing an annual production rate of 450kt.

Currently, the majority of production comes from sub-level stoping (79% of production) and ore shrinkage stoping (conventional short hole shrinkage stoping). Expected dilution and mining losses are 12.5% and 8.8% for sublevel stoping respectively; and 21.7% and 6.1% for shrinkage stoping respectively.

The shafts extend to the 750m level (with skip loading and weighing facilities below that level). First production from the mine took place in January 2010.

Mining works, based on a tracked system, are progressing from the centre of the deposit to the flanks, extracting reserves on the top levels first. Currently, one of the most developed levels is the 850m level. It is intended that the current production rate of 450ktpa be maintained in 2011 and increased it to 550ktpa by 2013.

Novoshirokinskoye ore reserves are given in table below.

Novoshirokinskoye Ore Reserve Estimate (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

	Ore (kt)	Au_{Eq} (g/t)	Au_{Eq} (kg)	Ag (g/t)	Ag (kg)	Au (g/t)	Au (kg)	Pb (%)	Pb (t)	Zn (%)	Zn (t)
Proven	2,438	7.89	19,233	77.0	187,676	3.89	9,473	2.98	72,601	1.28	31,089
Probable	4,427	7.78	34,463	84.3	373,276	3.89	17,200	2.69	118,896	0.99	43,912
Total Proven and Probable	***6,865***	***7.82***	***53,695***	***81.7***	***560,953***	***3.89***	***26,673***	***2.79***	***191,498***	***1.09***	***75,000***
Inferred material within stopes	48	5.16	249	53.3	2,577	1.5	74	1.94	937	1.14	549

The plant at Novoshirokinskoye began treating ore in December 2009. Construction of the plant buildings commenced in the early 1990's but work was stopped due to the break up of the Soviet Union. The plant uses the standard minerals processing techniques of SAG-ball milling, gravity and flotation to produce a gold-rich lead concentrate and a zinc concentrate. Current plant performance is broadly in line with planned values.

The environmental, social and H&S matters are generally being addressed in a competent and locally compliant manner.

Advanced Exploration Properties

Dolinnoe and Obruchevskoe

The Dolinnoe and Obruchevskoe deposits in east Kazakhstan are located 7.5 and 11km respectively from Ridder, the nearest railhead. The deposits are situated in the centre and deepest south-eastern portions respectively of the Ridder mining district in the Rudnyi Altay geotectonic block

The Dolinnoe deposit, which lies at a depth of between 450-650m, was discovered in 1987 and was followed up by integrated deep drilling, geophysical prospecting and geochemical investigations.

The Obruchevskoe deposit is situated at a depth of 800-1,000m and was discovered by DDH 1798 which was drilled in March 1987 to follow up a geochemical anomaly.

Geologically, the Ridder mining district coincides with an unusual graben structure preserving Silurian to Middle Devonian volcano-sedimentary sequences.

The two lodes comprising the Dolinnoe deposit are centred on anticlinal structures with gentle to moderate dips in all directions (domes) separated by a shallow trough. Each lode contains discrete mineralised bodies of two contrasting forms:

- Conformable, stratabound, mineralised mound-like bodies in altered calcareous siltstones; and
- Moderately to steeply dipping cross-cutting lenticular vein-like bodies in the gravelites which underlie the calcareous siltstone horizon.

The stratabound North-Eastern Lode is represented by a single mineralised body, Orebody 3 and a few small and isolated lenses. The underlying mineralisation is represented by a swarm of twelve moderately to steeply dipping north-northwest trending veins.

The South Western Lode occurs at a depth of 495-580m below surface and steeply dipping feeder veins extend to deeper levels. Grades tend to be lower than those recorded in the North-Eastern Lode.

The Obruchevskoe stratabound polymetallic sulphide mineralisation was outlined by drilling on two cupola-shaped lodes, Northern and Southern, and traced over the intervening ground, the Central Prospect, within the same lithological unit of the Kryukovskaya Formation.

The mineralisation occurs as tabular and lenticular veinlet-disseminated and massive sulphide bodies on three horizons (Nos.1, 2 and 3) in both Southern and Northern Lodes.

Mineralogically, the ores comprise sphalerite, galena, chalcopyrite and pyrite which, depending on their content in different ores, can constitute major, abundant or accessory minerals, although in solid sulphide polymetallic ores they may all be major components. Accessory minerals include tetrahedrite and tennantite with a high zinc content (fahlores which can be present in varying amounts); rare minerals consist of free gold, gold-silver alloy (electrum), pyrrhotite and, specifically at Obruchevskoe, bornite and some chalcopyrite and arsenical pyrite.

Gold is the main mineral of economic interest at Dolinnoe; veinlet-disseminated polymetallic mineralisation predominates at Obruchevskoe.

The 2008 Technical Economic Assessment (TEO) or Feasibility Study considered three scenarios for accessing the Dolinnoe and Obruchevskoe deposits:

- Option 1: Three shafts from surface independent of Ridder-Sokolniy mine;
- Option 2, 2A and 2B: Opening jointly with Ridder-Sokolniy mine, and
- Option 3: Using two shafts from surface but independent of Ridder-Sokolniy mine.

Kazzinc has chosen Option 3 because this option provides the greatest NPV and Internal Rate of Return and the lowest Discounted Payback Period.

Mineral resource estimates which have been performed for the Dolinnoe and Obruchevskoe ore zones are based on the models prepared by Kazakhstan Mineral Company (KMC) in January and February 2011 and have been calculated in accordance with the guidelines of the JORC Code (2004). The mineral resource models considers drilling available to this date and the effective date of this resource estimate is 02/02/2011.

Dolinnoe & Obruchevskoe Mineral Resource Estimate - COG 1.7% EqvZn (WAI 02.02. 2011) (In Accordance with the Guidelines of the JORC Code (2004))							
Classification	**Tonnage (kt)**	**EqvZn %**	**Au**	**Ag**	**Cu**	**Pb**	**Zn**
			g/t	**g/t**	**%**	**%**	**%**
Dolinnoe							
Measured	5,043	10.65	3.85	50.47	0.20	0.74	1.39
Indicated	2,707	6.60	2.32	28.05	0.14	0.48	1.00
Measured + Indicated	7,750	9.24	3.32	42.64	0.18	0.65	1.25
Inferred	6,907	4.84	1.59	15.88	0.12	0.48	0.86
Obruchevskoe							
Measured	1,154	17.75	1.62	40.68	0.88	4.02	8.87
Indicated	7,783	9.61	0.67	25.36	0.73	1.78	4.64
Measured + Indicated	8,937	10.66	0.79	27.34	0.75	2.07	5.18
Inferred	5,500	4.56	0.5	24.97	0.41	0.64	1.75

Ore Reserves for the Dolinnoe and Obruchevskoe deposits have been calculated in accordance with the guidelines of the JORC Code (2004). A summary of the Ore Reserves is presented in the table below.

Dolinnoe* & Obruchevskoe Ore Reserve Estimate
(WAI 01.01. 2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Deposit	Reserves	Ore (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
			Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Dolinnoe**	Proven	3.66	3.93	462,258	53.76	6,325,710	0.20	7,385	0.75	27,351	1.41	51,730
	Probable	0.96	2.38	73,822	29.82	923,296	0.14	1,338	0.50	4,849	1.02	9,792
	Total	***4.62***	***3.61***	***536,080***	***48.77***	***7,249,006***	***0.19***	***8,723***	***0.70***	***32,380***	***1.33***	***61,523***
Obruchevskoe	Proven	0.89	1.73	49,363	42.80	1,219,753	0.81	7,161	4.27	37,829	8.98	79,581
	Probable	3.25	0.90	94,019	33.21	3,466,977	0.83	26,845	2.66	86,520	6.50	211,092
	Total	***4.14***	***1.08***	***143,382***	***35.26***	***4,686,731***	***0.82***	***34,006***	***3.01***	***124,349***	***7.03***	***290,673***

*Dolinnoe Reserve estimate includes only Measured and Indicated Resources, and excludes 43,000t of Inferred material at 3.52g/t Au, 18.42g/t Ag, 0.13% Cu, 0.68%Pb & 1.24%Zn

**Dilution of 20% and losses of 5% applied

Chekmar

The deposit is situated 46km north of Ridder and linked with Ridder by a graded road.

The Chekmar deposit comprises three polymetallic ore zones, Chekmar, South-Eastern and Gusliakov. They are located in close proximity to each other in the centre of an 8km long mineralised trend.

The Chekmar ore zone contains about 40 mineralised bodies, predominantly of stockwork and stringer type ore.

The South Eastern ore is a blind zone of which only the top part has been explored. The structure is a NW-trending tight to isoclinal anticline with a moderate to steep (40-60°) NW plunge. Eleven lenses of stockwork and stringer mineralisation have been delineated with strike lengths in the range of 175-290m.

The Gusliakov Ore Zone occurs in a NW-trending double plunging tight to isoclinal anticline, which is correlated with the anticline of the South-Eastern ore zone. Due to faulting, this area is lifted up at least 200m in relation to the South-Eastern ore zone. Mineralisation at Gusliakov is polymetallic with barite-rich lenses at high stratigraphic levels. Stockwork, stringer and disseminated mineralisation predominates, but massive sulphide lenses are more abundant than at the South-Eastern deposit and account for over 5% of the mineralised volume. Twenty four mineralised lenses have been delineated with strike lengths of 60-150m. Down dip dimensions vary from 100-375m and widths from 7-10.5m.

The ore zones were explored by core drilling from surface augmented by underground exploration from two adits at Chekmar (Adit No.1 at +800m level and Adit No.2 at +670m level) and from another adit at Gusliakov (at +600m level).

At present no Mineral Resources or Ore Reserves have been calculated in accordance with the guidelines of the JORC Code (2004) for the Chekmar deposit.

Chashinskoye TMF

The Chashinskoye tailings dam, located some 3.5km to the east of the Ridder-Sokolniy Mine, was in operation from 1953 to 1978 inclusive, taking over from the Staroye tailings dam in containing material from the LPC and processed ore predominantly from the Ridder-Sokolny deposit. Although much larger, covering an area of some 250ha, the grade of the tailings is much lower as a result of reduced head grade to the LPC and improved recovery over time.

A summary results of the evaluation of the resources in accordance with the guidelines of the JORC Code (2004) are shown in the table below.

Chashinskoye Tailings Mineral Resource Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))							
Classification	**Tonnes (Mt)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**AuEq* (g/t)**
Indicated	57.8	0.67	5.16	0.05	0.15	0.38	1.15
Inferred	30.0	0.50	4.57	0.06	0.19	0.45	1.06

WAI has calculated Ore Reserves based upon the above Mineral Resource Estimate and in accordance with the guidelines of the JORC Code (2004). WAI has applied losses of 4.4% based upon historical data, and dilution of 0.5% to take into account minimal dilution from the bunds and floor.

Chashinskoye Tailings Ore Reserve Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))													
Deposit	Reserves	Ore (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
			Grade (g/t)	Metal Content (koz)	Grade (g/t)	Metal Content (koz)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)
Chashinskoye Tailing Dam	Proven	-	-	-	-	-	-	-	-	-	-	-
	Probable	55.53	0.70	1,245	5.37	9,589	0.05	28.9	0.16	86.7	0.40	219.6
	Total	***55.53***	***0.70***	***1,245***	***5.37***	***9,589***	***0.05***	***28.9***	***0.16***	***86.7***	***0.40***	***219.6***

Tishinskiy Slime Ponds

The Tishinskiy slimes ponds, located at the Tishinskiy Mine, are the accumulation of 'fines' deposited via a pipeline from the Ridder concentrator where, prior to August 2002, sulphide ore from the Tishinskiy deposit was processed. In August 2002 a filter press was commissioned at the Ridder Mining and Concentrating Complex (RMCC) which allowed dewatering of the slimes to occur and to be re-processed. Pond №1 is currently used as an emergency discharge facility for the Ridder complex and is flooded, whereas Pond №2 has been evaluated and is currently being reprocessed.

WAI has performed an evaluation of mineral resources and ore reserves for Tishinskiy Slime Ponds in accordance with the guidelines of the JORC Code (2004). The details of the evaluation are given in the tables below.

Tishinskiy Slimes Mineral Resource Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))								
CLASS	**Volume**	**Tonnes ***	**Au**	**Ag**	**Cu**	**Pb**	**Zn**	**AuEq**
	m^3	**t**	**g/t**	**g/t**	**%**	**%**	**%**	**g/t**
Indicated	378,744	333,295	0.33	9.93	0.22	0.76	2.46	2.77
Inferred	50,909	44,800	0.58	8.73	0.23	0.56	2.64	3.01

* Based on volume and an average density value (mean volumetric weight) of 0.88t/m3.

WAI has calculated Ore Reserves based upon the above Mineral Resource Estimate. WAI has applied losses of 4.4% based upon historical data, and dilution of 0.5% to take into account minimal dilution from the bunds and floor.

Tishinskiy Slimes Ore Reserve Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004)											
Reserves	**Ore (Mt)**	**Gold (Au)**		**Silver (Ag)**		**Copper (Cu)**		**Lead (pb)**		**Zinc (Zn)**	
		Grade (g/t)	**Metal Content (koz)**	**Grade (g/t)**	**Metal Content (koz)**	**Grade (%)**	**Metal Content (kt)**	**Grade (%)**	**Metal Content (kt)**	**Grade (%)**	**Metal Content (kt)**
Proven	-	-	-	-	-	-	-	-	-	-	
Probable	0.32	0.33	3.41	9.89	101.79	0.22	0.71	0.76	2.42	2.44	7.82
Total	***0.32***	***0.33***	***3.41***	***9.89***	***101.79***	***0.22***	***0.71***	***0.76***	***2.42***	***2.44***	***7.82***

Exploration Licences

Solovievskiy Block

The Solovievskiy Block, covering 4,300km^2 and registered under Contract No.2114 dated 25.07.2006, is located in the eastern Kazakhstan region, Katon-Karagaiskiy District, some 100km south-southeast of Ridder and 140km east southeast of Ust-Kamenogorsk. The nearest large town is Zyryanovsk where Kazzinc operate the Maleevskoye mine and processing plant. The Solovievskiy Block is the most advanced exploration licence currently in operation by Kazzinc and the only one to possess a valid exploration licence (at the time of the site visits in 2010).

The Solovievskiy Block lies within the Rudny Altai Palaeozoic province, or belt, and predominantly comprises Carboniferous and Devonian geological units. Kazzinc are undertaking a comprehensive and thorough exploration programme within the Solovievskiy block that commenced in 2005. This structured work has resulted in the identification of three targets; Novokhairuzovskoie (Au), Khairuzovskoie (Cu-Zn) and Greizenovoie, which are all located in the south-southwest of the licence block. The most advanced of these, Novokhairuzovskoie, is currently being drilled.. To date (01.07.2010) some 30,510m of surface trenching and 16,544.8m of diamond drilling (plus 41,180m of Reverse Circulation drilling has been completed).

Butachikhinsko-Kedrovskiy

The Butachikhinsko-Kedrovskiy Block is located in the East Kazakhstan Oblast, some 40km northeast of Ust-Kamenogorsk and to the southwest of Ridder. The licence block covers an area of 700km^2 and a contract was sent to the Ministry of Industry and New Technologies for approval, the receipt of which is awaiting by Kazzinc.

It should be noted that the Tishinskiy deposit/mine is enclosed within this licence block. The Butachikhinsko-Kedrovskiy Block lies within the Rudny Altay (VMS) province and predominantly comprises Lower to Upper Devonian geological units, with a small area of Carbonaceous in the extreme west of the block.

Regional prospecting has been conducted within the block since the early 1900's, initially involving geological mapping and later through geochemical and geophysical surveys. Since 2005 Kazzinc has undertaken a limited amount of exploration within the block, prior to the issuing of the licence but with State agreement. Initial work comprised the collation of available historic data, including drillholes and trench data, along with a confirmatory programme with sampling that identified elevated levels of mineralisation, particularly copper. Although preliminary in nature the work identified several promising targets.

Western Torgai

The Western Torgai exploration asset, comprising the Karabaitalsky and Sakharovsko-Adaevskiy sites, lies immediately to the southwest of the original Shaimerden exploration licence and contracts are in the process of registration at the Ministry of Industry and New Technologies. At the Karabaitalsky site a 3 year programme of drilling (19,000m) and geophysics is planned to investigate previously identified anomalies representing porphyry gold and polymetallic deposits; at the Sakharovsko-Adaevskiy site a similar 3 year programme has been drawn up to investigate the Klochkovskoye quartz diorite porphyry Cu/Mo deposit (intersected in old soviet boreholes) with 13,000m drilling and geophysics.

Bukhtarma Hydroelectric Power Plant

The plant was constructed in stages between 1953 and 1968. The Standard design life for hydro stations is generally in the region of 30 years. Currently the Bukhtarma Hydro Station is in the last stage of major refurbishment works, which are expected to be completed in 2011. The reconstruction programme has led to the increase in capacity from 675MW to 720MW. The dam is a reinforced concrete gravity type dam. The Bukhtarma reservoir is located in East Kazakhstan and Semipalatinsk oblasts of Kazakhstan and has an area of 5,500km^2.

Valuations

WAI has undertaken a valuation of the Kazzinc operations using discounted cash flow analysis to determine the Net Present Value ("NPV") of both the operations as a whole and each individual mine. A post-tax un-geared cash flow model was constructed for each operation based on the Ore Reserves compliant with the guidelines of the JORC Code (2004) estimated by WAI.

Ore production rates, operating costs and metallurgical recoveries for each mine were estimated on the performance over the last 3 years along with Kazzinc's forecast for 2011 production. Capital expenditure for 2011 to 2016 is based on Kazzinc's 5 year plan; thereafter, WAI has estimated likely capital expenditure based on the 2008 to 2016 expenditure.

In order to derive an individual valuation for each mine, a Net Smelter Return ("NSR") has been assumed for each concentrate product based on typical refining and treatment charges for that concentrate type. For the combined operations scenario, Kazzinc smelts its own concentrate and therefore benefits from the sale of by-products and other penalty elements, which are not included in the individual mine valuations. Third party concentrates smelted by Kazzinc are also included in the combined scenario but not in the individual mine valuations.

Kazzinc Life of Mine Production and Valuation

Operation	Ore Mined/Concentrate Processed (Mt)	Metal Production					Revenue (US$M)	Total Operating Costs (US$M)	EBIT (US$M)	NPV (US$M)
		Zn (kt)	Pb (kt)	Cu (kt)	Au (koz)	Ag (koz)				
Combined Kazzinc Operations including Smelters and Third Party Concentrates*	**263.2/9.9**	**2,628.0**	**610.9**	**314.1**	**8,650.0**	**48,993.8**	**21,330.8**	**9,941.5**	**10,186.1**	**4,513.8**
Vasilkovskoye	124.0	-	-	-	5,617.8	-	5,675.7	2,072.1	3,028.8	1,466.6
Maleevskoye	12.1	744.6	149.8	131.5	158.0	19,574.0	2,393.3	633.0	1,825.4	1,075.3
Shaimerden	2.5	539.2	-	-	-	-	804.7	216.6	535.5	287.3
Chashinskoye Tailings	55.5	-	-	-	1,076.5	3,659.2	1,040.1	515.1	461.3	188.4
Tishinskiy	25.1	1,015.4	125.6	75.5	338.0	5,234.9	2,601.8	1,877.3	473.9	220.6
Novoshirokinskoye	7.3	84.3	176.0	17.2	650.3	13,029.9	1,136.8	569.3	392.7	187.3
Ridder-Sokolniy	21.1	100.2	57.9	63.5	837.2	5,592.8	1,469.5	1,028.5	304.6	134.4
Obruchevskoe	4.1	281.8	115.3	33.5	107.5	94.0	849.9	235.0	426.0	65.8
Staroye Tailings	6.9	23.5	9.4	2.0	132.3	1,454.8	207.7	99.8	64.7	36.0
Dolinnoe	4.6	53.4	28.0	7.7	169.3	221.4	285.7	255.5	-78.8	10.2
Total	**263.2**	**2,842.4**	**662.0**	**330.9**	**9,086.9**	**48,861.0**	**16,465.2**	**7,502.2**	**7,434.1**	**3,671.9**

*Includes Net Revenue from third party concentrates but not volumes processed. Smelter recovery factors applied.

A summary of the combined Kazzinc operations cash flow forecast, including third party concentrates, between 2011 and 2027 is provided in the table below. Total operating costs for the operations are estimated at US$9,941.5M with a cumulative operating free cash flow of US$8,395.2M and EBIT of US$10,186.1M. The operations generate a positive NPV of **US$4,513.8M.**

Kazzinc Combined Operations - Summary Economic Statistics		
Summary	**Units**	**Value**
Ore Mined	**Mt**	263.2
Zn Concentrate Produced	**kt**	6,490.1
Pb Concentrate Produced	**kt**	1,127.1
Cu Concentrate Produced	**kt**	1,209.6
Au Concentrate Produced	**kt**	1,085.4
Zinc Metal Recovered	**kt**	2,628.0
Lead Metal Recovered	**kt**	610.9
Zinc Metal Recovered	**kt**	314.1
Gold Metal Recovered	**t**	278.1
Silver Metal Recovered	**t**	1,523.9
Gross Revenue Generated	**US$M**	21,330.8
Operating Costs	**US$M**	9,941.5
Capital Expenditure	**US$M**	1,323.2
Depreciation	**US$M**	1,203.2
Cash Taxes	**US$M**	1,671.0
Total Operating Free Cash Flow	**US$M**	**8,395.2**
EBIT	**US$M**	**10,186.1**
NPV	**US$M**	**4,513.8**

CONSULTANTS AND INTERESTS

Wardell Armstrong International ("WAI") is an internationally recognised, independent minerals industry consultancy. All consultants used in the preparation of this report are employed directly by WAI or are Associates of WAI and have relevant professional experience, including prior field experience of the geology and mineralisation of gold and polmetallic deposits in Kazakhstan and Russia.

Details of the principal consultants involved in the preparation of this document are as follows:

Phil Newall, BSc (ARSM), PhD (ACSM), CEng, FIMMM, **Director of Minerals**, ***Geology and Project Management***
Phil is a Mining Geologist with over 25 years experience of providing consultancy services to minerals companies throughout the world, with particular specialisation in CIS, Europe, Central and West Africa, and China. He has developed an extensive portfolio of exploration and mining-related contracts, from project management through to technical audits of a large variety of metalliferous and industrial mineral deposits. In particular, Phil has recently managed the Kempirsai Pre-feasibility Study for Bekem Metals on a Ni laterite in Kazakhstan, ARICOM's London CPR on their Fe assets in Russia, CPR for Kazakhaltyn for their London listing, PHM's London MER report, a large technical Due Diligence for Kazakhmys and the Feasibility Study for Oriel Resources on their Shevchenko Ni project.

Mark Owen, BSc, MSc (MCSM), CGeol, FGS, EurGeol, **Technical Director*, Geology and Mineral Resources***
Mark has over 27 years as a Mining Geologist, in both the metalliferous and industrial mineral mining sectors. He has considerable expertise of front line production mining, both in the underground and exploration environments, working on mines in the UK, Saudi Arabia and Venezuela. Throughout his experience he has had responsibility for technical teams addressing resource estimation, exploration planning and management, mining and process issues and environmental impact assessment. At WAI over the last 11 years, he has gained considerable experience of completing gold projects in Russia and the CIS, from scoping level through to full Bankable Feasibility Study and at the same time is fully conversant with the requirements of both the London and North American stock exchanges both acting as a Competent Person and author of numerous NI 43-101 reports.

Che Osmond, BSc, MSc (MCSM), ProfGradMIMMM, CGeol, Euro Geol, FGS; **Principal Geologist**, ***Exploration Geology***
Che is a Geologist with over 15 years experience of implementation and management of geological, geotechnical, environmental and civil engineering contracts. Upon completion of his MSc in Mining Geology he undertook a position with CSMA to plan and supervise nickel and copper exploration projects in Southern Africa before going on to other projects including exploration programmes for Cyprus-type VMS deposits within the ophiolite belt in the Sultanate of Oman. Che is also experienced with Datamine® modelling software. Since joining WAI in 2005, Che has been involved with exploration programme proposals, mineral resource estimation and audits, technical audits and CPR's as well as preparation of NI 43-101 Technical Reports of industrial and metalliferous minerals in Eastern Europe, CIS, North and West Africa and the Middle East.

Colin Taylor, BA (Hons), MA, Pr. Sci Nat, **Associate Consultant**, ***Exploration Geology***
Colin has some 40 years of international experience in exploration and resource geology, initially in South Africa and Namibia before being based in the UK. Colin has held such positions as Exploration, Mine and Chief Geologist and is widely versed in a variety of mineral commodities, including base and precious metals, industrial minerals and coal. Colin continues to contribute to technical studies, due diligence reviews and competent persons reports in the CIS, Europe and Africa and is fully conversant with the JORC Code (2004) and in the preparation of NI 43-101 reports.

Adam Wheeler, BSc MSc CEng Eur Ing, **Manager of Resources**, ***Resource Modelling and Estimation***
Adam is a Mining Engineer specialising in the application, customisation and management of mining and geological software systems. He has particular expertise relating to general mining/geological software systems used in the geostatistical resource and reserve assessment for both open pit and underground

optimisation. His skills include undertaking geostatistical studies, ore reserve estimation, geological modeling and mine planning, and training of personnel. Adam has broad international experience, but has been more involved in projects in CIS, Europe, Africa and South America.

Richard Ellis, MCSM, BSc, MSc, FGS; **Senior Resource Geologist**, ***Database & Resource Modelling***
Richard is a Geologist with nearly 10 years operational experience within the industrial minerals sector in the UK. He has conducted reserve and resource reports for annual corporate financial reporting and has been responsible for pit optimisation, production scheduling, grade control and reconciliation. As a resource modelling geologist with WAI, Richard has worked on numerous resource modelling projects including the Koksay Cu-Mo porphyry deposits, Kazakhstan, the Bapy iron ore deposit, Kazakhstan and recently conducted resource auditing of the Mount Nimba, iron ore deposit, Guinea.

Owen Mihalop, BSc (Hons), MSc, MCSM, CEng, MIMMM, **Technical Director**, ***Mining Engineering and Financial Modelling***
Owen is a chartered mining engineer with 15 years broad based experience in the mining and quarrying industries. He has gained experience in grass-roots exploration through to large scale open-pit and underground mining projects across Ireland, Bulgaria, Spain and Canada. He has worked as an operations manager in industrial mineral mining and quarrying operations in the UK and has gained considerable project management and financial evaluation experience through these roles.

Bruce Pilcher, BE(Mining)Syd, MAusIMM(CP), MIMMM, CEng, Eur Ing A**ssociate Director - Principal Mining Engineer**, ***WAI, Mining and Infrastructure***
Bruce is a Chartered Mining Engineer with 25 years experience in underground and surface mining operations in South Africa, Australia and UK. Bruce has served as Mine Superintendent, Regional and Technical Services Manager. He has considerable experience in production management, mine design and planning, and contract management in the coal and metalliferous mining industries. Bruce also has worked within the explosives industry focussing on drilling & blasting applications and contracting which included managing optimisation projects in Australasia, UK, France, Spain, Germany, Poland and Kyrgyzstan. Since joining WAI, Bruce has worked on several metal projects including the Macusani uranium deposit in Peru, the Namib lead and zinc mine in Namibia, the Bjorkdal gold mine in Sweden and the Kemparsai nickel mines in Kazakhstan

Lewis Meyer, BEng, ACSM, MSc, MCSM, PhD, CEng, FIMMM, **Principal Mining Engineer**, ***Geotechnics and Mine Design***
Lewis is a Chartered Mining Engineer with 18 years experience in mining, underground civil construction and rock mechanics of surface and underground mining operations. The early part of his career was spent in production mining in the platinum mines of South Africa. On returning to the United Kingdom, he completed a MSc, before joining a civil engineering consultancy gaining experience in project and contract management of various tunneling projects in London. Following completion of a PhD in geomechanics, Lewis spent 6 years working as a consultant rock mechanics engineer specializing in underground excavation and support design, on mining projects located in South Africa, India, Bangladesh, Kazakhstan, Indonesia and the UK. Since joining WAI in 2007, Lewis has specialised in open pit and underground mine design using Mine 2-4D software.

Stuart Richardson, BEng, ACSM, GradMIMMM, **Mining Engineer**, ***Mine Design and Scheduling***
Stuart is a mining engineer, graduating from Camborne School of Mines in 2006. He was employed as a student mining engineer by Deno Gold, Republic of Armenia, in 2005 gaining experience in metalliferous mining methodologies. He joined Wardell Armstrong in early 2007 as a Graduate Mining Engineer and has been involved in several projects, including the analysis of coal resources in Kentucky, USA. He has also gained experience in managing site works, acting as the Resident Engineer for a project at the Great Laxey Mine on the Isle of Man and as CQA Engineer for drilling works at Sutton Courtenay Landfill Site, Oxfordshire. In 2008, Stuart transferred to Wardell Armstrong International, working as a mining engineer and assisting in the production of a number of pre-feasibility and feasibility studies using Mine 2-4D software.

Daniil Lunev, DipEng (SPMI), PhD (SPMI) **Mining Engineer**, ***Mine Design and Scheduling***
Daniil is a specialist in mining machinery. His skills include the following: optimisation of schemes of underground and land mining equipment; calculations of mining transport systems and estimation of efficiency and reliability of mining machinery. He is particularly experienced in belt conveyor system development, modernisation of construction, and resolving conveyor application problems. He has 5 patents related to belt conveyor innovation. His has recently been involved in the Bapy Fe Ore Project in Kazakhstan, which involved the review of a feasibility study, advising on mining operation and scheduling, and the Feasibility Study of the Akbakai gold project in the same country.

Philip King, BSc (Eng) Mineral Technology (Hons), **Technical Director**, ***Processing***
Phil has 20 years minerals processing experience ranging from laboratory test work and pilot plant operations through to plant commissioning, operations and trouble-shooting. He has considerable experience in the technical and financial evaluation of many mining projects through the completion of both pre-feasibility and feasibility studies, and has been involved in process design and engineering studies, equipment selection, and capital and operating cost estimates. He has also participated in a number of multi-disciplinary projects throughout Central Asia, Africa and Europe involving base and precious metals, industrial minerals and coal. In particular, Phil has considerable experience of the metallurgy and extraction from gold ores.

Colin Hunter, BSc (Hons), PhD, **Associate Consultant**, ***Processing***
Colin has 32 years minerals processing experience ranging from laboratory test work and pilot plant operations through to plant commissioning, operations and trouble-shooting. He has considerable experience in the technical and financial evaluation of many mining projects through the completion of both pre-feasibility and feasibility studies, and has been involved in process design and engineering studies, equipment selection, and capital and operating cost estimates. He has also participated in a number of multi-disciplinary projects throughout Russia, Western Australia, West Africa, Central Asia, and Europe involving base and precious metals. In particular, Colin has considerable experience of commissioning and operation of Biox plants and bacterial leaching technology.

Chris Broadbent, BSc, PhD, CEng, FIMMM, **Partner, Wardell Armstrong**, ***Pyrometallurgy***
Chris is a well-recognised authority on pyrometalurgical processing and the treatment and disposal of wastes from mining and metallurgical plants. He has worked on R & D projects at universities as well as with ISP Ltd, RTZ Technical Services Ltd and Billiton (Shell) Research B.V. Whilst at Shell he chaired the Nickel Research Group and managed the FeNi R&D team. He has also published widely on the pysio-chemistry of nickel smelting slags. Since 1994, as head of the pollution group in Wardell Armstrong, he has become experienced in assessing the environment consequences of general industrial pollution and the use of environmental chemistry in contaminated land investigations. In the UK he works closely with the metallurgical industries, especially the ferrous foundry sector and was a member of the Environment Agency Working Parties revising Process Guidance to the Ferrous Foundry, non-ferrous metal and glass making sectors.

Andrei Kudrin, BS (Hons), MEng (Hons), **Engineer**, ***Renewable Energy***
Andrei, as a mature student has recently graduated from the University of Exeter with an engineering degree in Renewable Energy. Before joining Wardell Armstrong International he has had placements with various companies such as Scottish and Southern Energy and Cornwall Sustainable Energy Partnership. He is a member of the Institute of Engineering and Technology and the Energy Institute. Andrei has worked on various wind and solar farm development projects, CHP and hydro schemes. He continues to contribute to technical, economical and environmental assessments within Wardell Armstrong.

Nick Coppin, Bsc, MSc, **Managing Director WAI**, ***Environmental Scientist, Environment, Social and Health & Safety***
Nick is an environmental scientist with over 30 years of broadly based experience in the mining and minerals industry. He has worked extensively with government and regional authorities and institutions for mineral planning, mining administration, environmental protection, industrial development and finance. He is fully conversant with national and international guidelines, procedures, regulations and standards for the mining and quarrying sector.

Kim-Marie Clothier, BSc (Hons) MRes AIEEM, MIEMA, Grad IMMM, ACMI; **Senior Environmental Scientist**, ***Environment, Social and Health & Safety***
Kim-Marie is a Senior Environmental Scientist, mainly dealing with Environmental and Social Impact Assessments on mining projects overseas. She has worked with the company for over 5 years on projects located in Uzbekistan, Macedonia, Kyrgyzstan, Portugal, DRC, Sierra Leone, Greenland, Russia and Kazakhstan, providing environmental and social impact assessments of mining projects as part of feasibility studies and CPR or due diligence processes. She has contributed to projects for improving environmental performance and implementing Environmental Management Systems and plans as well as ESIAs. She also regularly undertakes environmental audits for UK mineral sites and has extensive experience in the assessment and remediation of contaminated land and has a strong interest in sustainable treatment options for Brownfield sites.

Kathy Hicks BSc (Hons) MSc MCSM FGS **Geo-Environmental Scientist**, ***Environment***
Kathy is a Geo-environmental Scientist specialising in the field of mineral extraction and the remediation of land affected by past mining activities. After graduating from Camborne School of Mines, Kathy has worked as a Geotechnical Engineer with Fugro Engineering Services Ltd on sites with a range of issues throughout the UK. As a Mining Consultant for a further 4½ years, Kathy has acquired further experience in the assessment and remediation of mining impacted land. Since working for WAI, Kathy has worked as part of a multidisciplinary environmental team on a wide range of mining and development projects and has acquired experience in the remediation of contaminated land, risk assessment, gas, soils, water and environmental condition monitoring. Kathy has also worked on specific mining related projects within the UK, EU and CIS. She is conversant with OVOS/ESIA procedures and writing environmental reviews. Kathy has been responsible for formulating on-site health and safety guidance and is conversant with health and safety, environmental management and permitting requirements.

Christine Blackmore, BSc, MSc, CEnv, FIMMM, **Principal Environmental Geologist**, ***WA, Enviroment***
Christine has over 10 years of experience in environmental work both in the UK and overseas. Her expertise is focused in environmental management (EMS, EIA ect) and environmental auditing within the mining industry, with some of the work including aspects of health and safety procedures and protocols. Her recent experience has included work in Mauritania (Copper and Gold), Burkina Faso (Gold), Mexico (Gold), Kazahkstan (Zinc/Copper) and China (Silca). Clients include mining companies, investment bankers, government departments and design institutes. She has experience in design and construction using both natural and geosynthetic materials for tailings dams, lagoons, landfills, together with the design of environmental monitoring programmes. Lately she has been involved in transportation of dangerous goods audits for cyanide and waste oil, which have involved preparing accompanying management plans.

Elena Bagayeva, BA, **Regional Manager and Translator**, ***WAI Kazakhstan - Logistics***
Elena joined the WAI Kazakhstan team at the end of 2009 as Executive Secretary/ Technical Translator. She has gained her diversified professional experience in mineral mining sector through working with a number of international mining companies, including 4 years full-time employment in UK-based Central Asia Metals Ltd which holds exploration/mining properties across Kazakhstan and Mongolia. Elena's wealth of knowledge in both the mining industry and the technical terminology thereof is a valuable asset for WAI.

Yelena Borovikova, **Senior Translator**, ***WAI Kazakhstan***
Yelena has recently joined the WAI Kazakhstan team as Senior Translator. Her previous experience was gained through working with a number Western consultancies operating in FSU.

Timur Mussin, MA, **Technical Manager*, WAI Kazakhstan***
Timur currently works at WAI's Kazakhstan office to assist the Regional Manager in technical aspects of liaison with clients and technical translation for many of our FSU projects, as well as support in reconciliation of FSU standards and practices with international ones. He has 12 years experience in various aspects of the minerals and mining industry of the FSU. Previously, he has worked in R&D Dept of ALROSA (Russian diamond producer) and several Western mining companies in Kazakhstan on manganese, gold and copper projects as a project manager and an operations manager and has gained considerable project management and evaluation

experience through these roles, as well as good understanding of both FSU and Western standards and practices.

Neither WAI, its directors, employees nor company associates hold any securities in Glencore, Glencore International AG or Kazzinc, its subsidiaries or affiliates, nor have they:

- any rights to subscribe for any Glencore, Glencore International AG or Kazzinc securities either now or in the future;
- any vested interest or any rights to subscribe to any interest in any properties or concessions, or in any adjacent properties and concessions held by Kazzinc; or
- been promised or led to believe that any such rights would be granted to WAI.

The only commercial interest WAI has in relation to Glencore, Glencore International AG or Kazzinc is the right to charge professional fees to Glencore, Glencore International AG or Kazzinc (as applicable) at normal commercial rates, plus normal overhead costs, for work carried out in connection with the investigations reported herein.

Disclaimer/Reliance on Experts

The observations, comments and results of technical analyses presented in these reports represent the opinion of WAI as of 4 May 2011 and are based on the work as stated in the report. While WAI is confident that the opinions presented are reasonable, a substantial amount of data has been accepted in good faith. Although WAI has visited all of the assets described in this report, WAI did not conduct any verification or quality control sampling. WAI cannot therefore accept any liability, either direct or consequential, for the validity of such information accepted in good faith.

1 SITE VISITS

1.1 Background

In order to prepare the Competent Person's Report, WAI personnel undertook site visits to the various assets as follows:

Kazakhstan

Vasilkovskoye Open Pit and Maleevskoye Mine

A team of WAI consultants visited the Vasilkovskoye open pit operation during the period 12 and 15 October 2010 and Maleevskoye Mine during the period 16 and 18 October 2010. During the site visit the WAI team inspected current exploration, production and process activities, discussed many aspects of the project with Kazzinc technical staff and collected sufficient data to complete the studies.

The team from WAI visiting the project comprised Phil Newall, Director of Minerals and Mining Geologist; Richard Ellis, Resource Geologist, Owen Mihalop, Technical Director and Mining Engineer; Phil King, Technical Director and Processing Engineer; and Nick Coppin, Managing Director of WAI and Environment Scientist. All the technical team are full time employees of WAI.

Ridder-Sokolniy Mine and Satellite Mines and Advanced Exploration Properties

A large team of WAI consultants visited the Ridder-Sokolniy, Shubinskiy and Tishinskiy mines, Dolinnoe & Obruchevskoe development projects and Chekmar exploration licence during the period 12 to 22 October 2010. During the various site visits the WAI team inspected current exploration, production and process activities, discussed many aspects of the project with Kazzinc technical staff and collected sufficient data to complete the studies.

The team from WAI visiting the project comprised Mark Owen, Technical Director and Mining Geologist; Adam Wheeler, Resource Geologist; Owen Mihalop, Technical Director and Mining Engineer; Bruce Pilcher, Lewis Meyer, Stuart Richardson and Daniil Lunev, Mining Engineers; Coin Hunter, Processing Engineer; and Chris Broadbent, Director of WA and Pyrometallurgist. All the technical team are full time employees of WAI or WAI.

A separate visit was made by Kim-Marie Clothier, Kathy Hicks and Christine Blackmore, all Environmental Scientists with WAI on 10-21 November 2010.

Kazzinc Smelters – Ust-Kamenogorsk

The Kazzinc smelters in Ust-Komenogorsk were visted by Chris Broadbent, Director of WA and Pyrometallurgist during the period 12 to 22 October 2010. During the visit he inspected the processing activities, discussed many aspects of the project with Kazzinc technical staff and collected sufficient data to complete the study.

Shaimerden Open Pit

WAI personnel Bruce Pilcher (Mining Engineer), Colin Taylor (Geologist) and Colin Hunter (Process Engineer) visited the Shaimerden mine and offices during the period 12 to 14 October 2010.

A separate visit was made by Anne Mihalop, Environmental Scientist with WAI on 23 to 25 November 2010.

Solovievskiy Exploration Block

On 06 October 2010, Che Osmond, Senior Geologist with WAI visited the TOPAZ Mining Co. LLP analytical laboratory in Ust-Kamenogorsk where the fire assay analysis is conducted for the current Solovievskiy Block exploration works.

Che Osmond visited the Solovievskiy Block licence area on 07 and 08 November 2010, specifically the Novokhairuzovskoie deposit, and the Kazzinc Geological Camp where core is logged, prepared and stored from the current exploration work. The Kazzinc sample preparation laboratory in Zyryanovsk was also inspected as well as the Kazzinc analytical laboratory located in the Zyryanovskiy Mining and Processing Complex.

The Plazma Analit LLP analytical laboratory (Ust-Kamenogorsk) was also visited on the 08 October 2010, where polymetallic ore is analysed by ICP methods.

Tishinskiy Slimes Ponds & Staroye and Chanshinskoye TMF's

From 09 to 11 October 2010, Che Osmond visited the Tishinskiy slimes ponds, Staroye and Chanshinskoye TMF's as well as the Kazzinc office in Ridder to review data with senior technical staff. The Kazzinc 'Technical Control Dept.' sample preparation and analysis laboratory in Ridder was also visited where samples from the mining and process plant are analysed.

Bukhtarma Hydroelectric Power Plant

Andrei Kudrin visited the Bukhtarma Hydroelectric Power Plant between 17 and 18 November 2010.

RUSSIA

Novoshirokinskoye Mine

A team of WAI consultants visited the Novoshirokinskoye Mine during the period 10 and 11 November 2010. During the site visit the WAI team inspected current exploration, production and process activities, discussed many aspects of the project with Kazzinc technical staff and collected sufficient data to complete the studies.

The team from WAI visiting the project comprised Phil Newall, Director of Minerals and Mining Geologist;, Owen Mihalop, Technical Director and Mining Engineer; Phil King, Technical Director and Processing Engineer; and Chris Broadbent, Director of WA and Environment Scientist. All the technical team are full time employees of WAI or WA.

1.2 Study Strategy

The basic strategy for this CPR has been to examine and report on the existing information available on the properties held globally by the Client, which includes geological, resources/reserves, mining and metallurgical data and basic economic parameters. All resource and reserve estimates presented in the CPR have been prepared by WAI in accordance with the JORC Code (2004).

During the visits, further information was gathered on infrastructure, equipment, costs, potential mining methods, permitting and environmental issues.
Locally-based and publicly available documentation was viewed by WAI and in addition, WAI held meetings with key staff at the project sites.

For the environmental aspects of the projects WAI has adopted the following strategy throughout the study:

- Carry out a review of environmental and social issues associated with each mine and processing facilities development in line with national and international requirements;

- Provide a preliminary evaluation of the project in this respect and to identify the main issues likely to affect valuation, viability, and access to funds from international financial institutions; and
- Identify and comment on areas of particular risk or exemplary practice.

The methodology used by WAI for the environmental study has been to:

- Review project information and seek further clarification of project description as necessary;
- Review the environmental studies previously undertaken;
- Visit the sites and make a visual inspection of the project area and its surroundings;
- Review and comment on key environmental and social issues;
- Assess projects in line with international requirements (generally including the Equator Principles, IFC Performance Standards and World Bank EHS guidelines);
- Advise on recommendations to satisfy 'best practice environmental management'; and
- Assess the adequacy of planned rehabilitation and closure costs.

2 OVERVIEW OF THE ASSETS

2.1 Introduction

This CPR covers all the main project technical areas including geology, resources, mining methods, processing, infrastructure, markets, Capex, Opex and environmental. The core assets form the basis of the CPR and comprise the following assets as shown in Figure 2.1 and Figure 2.2 below.

Figure 2.1: Location of Kazzinc Assets in Kazakhstan
(Note: Ridder-Sokolniy & Satellites includes: Ridder-Sokolniy Mine, Tishinskiy Mine, Shubinskiy Mine; Dolinnoe, Obruchevskoe and Chekmar advanced exploration projects; and Staroye and Chashinskoye TMF's and Tishinskiy Slime Ponds)

Figure 2.2: Location of Novoshirokinskoye Mine in the Russian Federation

2.2 Summary of Assets

A summary of the assets is provided in Table 2.1 below.

Table 2.1: Summary of Assets - Kazakhstan						
Asset	Holder	Interest	Status	Licence Expiry Date	Licence Area (km²)	Comments
Vasilkovskoye Deposit Contract No.1185 dd. 07.07.2003	Kazzinc	100%	Mining	07 July 2025	28.3	Valid for 20 years Permit for mining to depth of 660m
Maleevskoye Deposit Contract No.95 dd 21.05.1997	Kazzinc	100%	Mining	21 May 2022	7.703	Valid for 25 years Permit for mining to depth of 1,100m
Ridder-Sokolniy Deposit Contract No.91 dd. 21.05.1997	Kazzinc	100%	Mining	21 May 2022	8.558	Valid for 25 years Permit for mining to depth of Level 18
Ridder-Sokolniy Deposit Contract No.91 dd. 21.05.1997	Kazzinc	100%	Mining	21 May 2022	11.5	Valid for 25 years Permit for mining to depth of Level 24
Ridder-Sokolniy Deposit Contract No.91 dd. 21.05.1997	Kazzinc	100%	Exploration	21 May 2022	12.1	Valid for 19 years
Tishinskiy Deposit Contract No.92 dd. 21.05.1997	Kazzinc	100%	Mining	21 May 2022	3.8k	Valid for 25 years Permit for mining to depth of 590m
Shubinsky Deposit Contract No. 1296 dd. 30.12.2003	Kazzinc (indirectly through TOO Shubinskiy)	100%	Mining	7 April 2015	0.97	Valid for 25 years Permit for mining to depth of 510m
Staroye TMF Contract No.559 dd. 07.11.2000	Kazzinc	100%	Mining	07 November 2030	0.615	Valid for 30 years Permit for mining to a depth of 14m
Chashinskoye TMF Contract No.559 dd. 07.11.2000	Kazzinc	100%	Mining	07 November 2030	2.65	Valid for 30 years Permit for mining to a depth of 60m
Slimesdumps No.1 and No.2 Contract No.2693 dd. 19.06.2008	Kazzinc	100%	Mining	19 May 2013	0.3	Valid for 30 (5) years Permit for mining to a depth of 12m
Shaimerden Deposit Contract No.298 dd. 04.03.1999	Kazzinc (indirectly through Shaimerden JSC)	100%	Mining	04 March 2024	3.23	Valid for 25 years Permit for mining to depth of 245m
Dolinnoe Contract No.2450 dd. 20.08.2007	Kazzinc	100%	Exploration	20 August 2026	3.3	Valid for 19 years Permit for mining to depth of 520m
Obruchevskoe Contract 450 dd. 20.08.2007	Kazzinc	100%	Exploration	20 August 2026	1.61	Valid for 19 years Permit for mining to depth of 1,015m
Novoshirokinskoye Deposit Licence ЧИТ 12697 ТЭ dd. 30.09.2004	Kazzinc (indirectly through OAO Novoshirokinskoye Rudnik)	48.3%	Mining	01 October 2024	1.4	Valid for 20 years Permit for mining to depth of 660m

Note:
The Chekmar subsoil use contract is expected to be finalised and registered with the relevant Kazakhstan authority by the end of 2011. The Chekmar mining plan, which was submitted by Kazzinc to the relevant Kazakhstan authority, is currently going through the approval process.

As of 1 January 2011, those Mineral resources estimated in accordance with the guidelines of the JORC Code (2004) held by Kazzinc in Kazakhstan and Russia are provided in Table 2.2, whilst Ore reserves estimated in accordance with the the guidelines of the JORC Code (2004) and derived from these are given in Table 2.3 and Table 2.4 below.

Table 2.2: Summary of all Kazzinc Mineral Resource Estimates (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))

Deposit	Resources	Cut Off Grade	Date	Tonnes (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
					Grade (g/t)	Metal Content (oz)***	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Vasilkovskoye	Measured	0.4g/t Au	01 January 2011	45.23	1.75	2,550,552								
	Indicated			141.56	1.72	7,840,344								
	Measured + Indicated			186.80	1.73	10,390,893								
	Inferred			99.08	1.77	5,650,405								
	Total			285.87	1.74	16,041,301								
Maleevskoye	Measured	n/a	01 January 2011	12.92	0.62	257,917	77.74	32,299,948	2.39	307,852	1.13	146,601	6.92	894433
	Indicated			11.03	0.56	200,354	64.82	22,987,359	1.95	215,144	1.15	127,182	6.82	752050
	Measured + Indicated			23.96	0.59	458,271	71.79	55,287,307	2.19	522,966	1.14	273,783	6.87	1,646,483
	Inferred			4.87	0.25	38,756	47.79	7,438,836	0.97	47,632	1.58	76794	4.99	243133
	Total			28.83	0.53	497,027	67.73	62,726,143	1.98	570,598	1.21	350,577	6.56	1,889,616
Novoshirokinskoye	Measured	3g/t AuEq*	01 January 2011	2.43	4.43	345,395	87.74	6,840,843			3.43	83,179	1.47	35,648
	Indicated			4.64	4.30	641,022	94.82	14,135,204			3.07	142,347	1.15	53,322
	Measured + Indicated			7.06	4.34	985,356	92.39	20,976,337			3.19	225,270	1.26	88,978
	Inferred			1.51	2.08	100,985	57.02	2,768,726			2.44	36,851	1.81	27,336
	Total			8.57	3.95	1,086,341	86.18	23,745,063			3.06	262,121	1.36	116,314
Shaimerden (Stockpiles)	Measured	n/a	01 January 2011											
	Indicated			2.48									21.71	539,143
	Measured + Indicated			2.48									21.71	539,143
	Inferred													
	Total			2.48									21.71	539,143
Tishinskiy	Measured	2.2% ZnEq**	01 January 2011	21.23	0.60	412,348	9.02	6,153,840	0.587	124,635	1.00	212,031	4.71	1,000,316
	Indicated			7.01	0.46	103,607	9.75	2,072,238	0.446	31,289	0.95	66,298	4.35	304,634
	Measured + Indicated			28.24	0.57	515,956	9.20	8,226,078	0.552	155,924	0.99	278,329	4.62	1,304,950
	Inferred			5.19	0.33	55,131	11.94	2,379,670	0.554	28,748	1.36	70,285	4.46	231,443
	Total			33.43	0.53	571,087	9.87	10,605,748	0.553	184,672	1.04	348,614	4.60	1,536,393
Dolinnoe	Measured	1.7% ZnEq	01 January 2011	5.04	3.85	624,252	50.47	8,183,378	0.20	10,090	0.74	37,320	1.39	70,100
	Indicated			2.70	2.32	201,879	28.05	2,440,821	0.14	3,790	0.48	12,990	1.00	27,070
	Measured + Indicated			7.74	3.32	826,131	42.64	10,624,199	0.18	13,880	0.65	50,310	1.25	97,170
	Inferred			6.90	1.59	353,085	15.88	3,526,405	0.12	8,290	0.48	33,150	0.86	59,400
	Total			14.64	2.50	1,179,216	30.03	14,150,604	0.15	22,170	0.57	83,460	1.07	156,570
Obruchevskoe	Measured	1.7% ZnEq	01 January 2011	1,154	1.62	60,105	40.68	1,509,308	0.88	10,155	4.02	46,391	8.87	102,360
	Indicated			7,783	0.67	167,654	25.36	6,345,814	0.73	56,816	1.78	138,537	4.64	361,131
	Measured + Indicated			8,937	0.79	226,992	27.34	7,855,636	0.75	67,028	2.07	184,996	5.18	462,937
	Inferred			5,500	0.50	88,415	24.97	4,415,423	0.41	22,550	0.64	35,200	1.75	96,250
	Total			14,437	0.68	315,407	26.44	12,271,059	0.62	89,578	1.53	220,196	3.87	559,187
Chashinskoye Tailing Dam	Measured	n/a	01 January 2011	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Indicated			57.80	0.67	1,245,000	5.16	9,589,000	0.05	28,900	0.15	86,700	0.38	219,640

	Measured + Indicated			57.80	0.67	1,245,000	5.16	9,589,000	0.05	28,900	0.15	86,700	0.38	219,640
	Inferred			30.00	0.50	482,261	4.57	4,407,861	0.06	18,000	0.19	57,000	0.45	135,000
	Total			87.80	0.61	1,727,261	4.96	13,996,861	0.05	46,900	0.16	143,700	0.40	354,640
Tishinskiy Tailing Dam	Measured	n/a	01 January 2011											
	Indicated			0.33	0.33	3,536	9.96	106,407	0.22	733	0.76	2,533	2.46	8,199
	Measured + Indicated			0.33	0.33	3,536	9.96	106,407	0.22	733	0.76	2,533	2.46	8,199
	Inferred			0.04	0.58	835	8.73	12,574	0.23	103	0.56	251	2.64	1,183
	Total			0.38	0.36	4,372	9.79	118,981	0.22	836	0.74	2,784	2.48	9,382
Staroye Tailing Dam	Measured	n/a	01 January 2011											
	Indicated			0.82	2.01	53,266	18.78	497,680	0.05	412	0.48	3,956	1.11	9,149
	Measured + Indicated			0.82	2.01	53,266	18.78	497,680	0.05	412	0.48	3,956	1.11	9,149
	Inferred			5.90	0.91	172,592	11.16	2,116,618	0.04	2,360	0.3	17,697	0.63	37,164
	Total			6.72	1.04	225,858	12.09	2,614,297	0.04	2,772	0.32	21,654	0.69	46,314
Ridder-Sokolniy	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu Ore	01 January 2011	30.21	1.11	1,076,497	11.73	11,394,749	0.64	29,475	0.40	119,385	0.91	273,897
	Indicated			67.91	1.55	3,397,959	10.74	23,454,050	0.38	113,198	0.40	269,542	0.95	645,031
	Measured + Indicated			98.12	1.42	4,474,456	11.05	34,848,799	0.46	142,675	0.40	388,927	0.94	918,928
	Inferred			6.64	0.83	178,004	9.09	1,940,024	0.29	13,386	0.59	39,365	1.12	74,187
	Total			104.76	1.38	4,652,459	10.92	36,788,824	0.45	156,061	0.41	428,292	0.95	993,115

*Gold plus other metals expressed in 'equivalent ounces' of gold use a conversion ratio dependent on prevailing gold and metal prices, or as stated.
**Zinc equivalent is the zinc grade plus other metals in equivalent units, using a conversion ratio dependent on prevailing zinc and other metal prices, or those stated.
***Metal content is based on tonnage and grade with no account of mining losses and dilution, and metallurgical recoveries.

Table 2.3:Kazzinc Mineral Resources Ridder-Sokolniy Mine
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004)

Deposit	Resources	Cut Off Grade	Date	Tonnes (Mt)	Gold (Au) Grade (g/t)	Gold (Au) Metal Content (oz)	Silver (Ag) Grade (g/t)	Silver (Ag) Metal Content (oz)	Copper (Cu) Grade (%)	Copper (Cu) Metal Content (t)	Lead (Pb) Grade (%)	Lead (Pb) Metal Content (t)	Zinc (Zn) Grade (%)	Zinc (Zn) Metal Content (t)
Centralny	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01-Jan-11	20.09	1.00	648,768	8.43	5,446,956	0.9	18,020	0.31	6,1480	0.8	160,800
	Indicated			36.21	1.23	1,437,071	8.46	9,847,050	0.44	15,950	0.36	130,750	0.85	308,310
	Measured + Indicated			56.30	1.15	2,085,839	8.45	15,294,006	0.60	33,970	0.34	192,230	0.83	469,110
	Inferred			1.53	0.83	40,763	3.74	183,365	0.45	690	0.15	2,320	0.71	10,880
	Total			57.83	1.14	2,126,602	8.33	15,477,371	0.60	34,660	0.34	194,550	0.83	479,990
Bystrushinskoe	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
	Measured + Indicated			16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
	Inferred			0.04	1.53	1,768	3.44	3,922	0.18	63	0.15	53	0.33	118
	Total			17.02	2.72	1,488,032	12.28	6,715,069	0.28	48,127	0.42	71,563	0.90	152,930
Bystrushinskoe North Flank	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
	Measured + Indicated			1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
	Inferred			0.09	2.46	7,106	6.90	19,997	0.86	776	0.29	264	1.03	925
	Total			1.84	1.31	77,902	5.69	337,325	1.01	18,666	0.29	5,404	1.07	19,793
Belkina_RSM_WA	Measured	1.7% ZnEq	01-Jan-11	3.76	1.26	152,331	18.98	2,294,630	0.09	3,384	0.47	17,670	0.98	36,850
	Indicated			4.77	1.11	170,126	16.85	2,582,538	0.08	3,814	0.41	19,550	0.85	40,520
	Measured + Indicated			8.53	1.18	322,456	17.79	4,877,168	0.08	7,200	0.44	37,220	0.91	77,370
	Inferred			0.52	0.88	14,669	23.42	390,393	0.28	1,450	0.32	1,660	0.90	4,670
	Total			9.05	1.16	337,125	18.11	5,267,561	0.10	8,650	0.43	38,880	0.91	82,040
Perspectivnaya	Measured	1.7% ZnEq	01-Jan-11	4.59	1.55	228,476	16.01	2,359,940	0.13	5,960	0.57	26,130	1.07	49,060
	Indicated			3.94	1.36	172,076	16.79	2,124,381	0.13	5,120	0.53	20,860	1.03	40,530
	Measured + Indicated			8.52	1.46	400,552	16.37	4,484,321	0.13	11,080	0.55	46,990	1.05	89,590
	Inferred			2.03	1.19	77,607	12.08	787,806	0.14	2,840	0.49	9,940	0.95	19,270
	Total			10.55	1.41	478,159	15.55	5,272,127	0.13	13,920	0.54	56,930	1.03	108,860
Sokolok	Measured	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.89	5,550	1.00	2,940

	Measured + Indicated			0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.89	5,550	1.00	2,940
	Inferred			0.37	1.40	16,715	15.27	182,315	0.14	520	1.98	7,350	1.48	5,500
	Total			0.66	1.38	29,554	61.94	1,323,123	0.14	900	1.94	12,900	1.27	8,440
New Glubokaya	Measured	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			0.32	0.18	1,867	2.48	25,723	0.06	190	1.80	5,810	0.67	2,160
	Measured + Indicated			0.32	0.18	1,867	2.48	25,723	0.06	190	1.80	5,810	0.67	2,160
	Inferred			0.07	0.14	314	2.68	6,004	0.31	220	2.10	1,460	0.74	520
	Total			0.39	0.17	2,181	2.51	31,727	0.10	410	1.85	7,270	0.68	2,680
Glubokaya	Measured	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			-	-	-	-	-	-	-	-	-	-	-
	Measured + Indicated			-	-	-	-	-	-	-	-	-	-	-
	Inferred			0.51	0.10	1,640	1.48	24,277	0.08	410	1.91	9,740	0.83	4,230
	Total			0.51	0.10	1,640	1.48	24,277	0.08	410	1.91	9,740	0.83	4,230
Dalnaya	Measured	1.7% ZnEq	01-Jan-11	-	-	-	-	-	-	-	-	-	-	-
	Indicated			3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
	Measured + Indicated			3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
	Inferred			1.16	0.18	6,855	3.18	118,984	0.51	5,958	0.40	4,620	2.07	24,083
	Total			4.30	0.27	37,959	3.41	471,535	0.63	27,239	0.27	11,595	2.24	96,451
Zavodskaya	Measured	1.7% ZnEq	01-Jan-11	1.78	0.82	46,922	22.60	1,293,223	0.12	2,111	0.79	14,105	1.53	27,187
	Indicated			0.51	0.97	15,816	21.62	352,524	0.10	509	0.67	3,397	1.29	6,523
	Measured + Indicated			2.29	0.85	62,739	22.38	1,645,747	0.12	2,620	0.76	17,502	1.48	33,710
	Inferred			0.33	1.00	10,567	21.10	222,961	0.14	459	0.60	1,958	1.21	3,991
	Total			2.62	0.87	73,305	22.22	1,868,709	0.12	3,079	0.74	19,460	1.44	37,701
Ridder-Sokolny Mine	Measured			30.21	1.11	1,076,497	11.73	11,394,749	0.64	29,475	0.40	119,385	0.91	273,897
	Indicated			67.91	1.55	3,397,959	10.74	23,454,050	0.38	113,198	0.40	269,542	0.95	645,031
	Measured + Indicated			98.12	1.42	4,474,456	11.05	34,848,799	0.46	142,675	0.40	388,927	0.94	918,928
	Inferred			6.64	0.83	178,004	9.09	1,940,024	0.29	13,386	0.59	39,365	1.12	74,187
	Total			104.76	1.38	4,652,459	10.92	36,788,824	0.45	156,061	0.41	428,292	0.95	993,115

Table 2.4: Kazzinc Ore Reserve Estimates
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Deposit	Reserves	Ore (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
			Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Vasilkovskoye	Proven	33.30	1.95	2,087,709					-	-	-	-
	Probable	90.70	1.94	5,657,181	-	-	-	-	-	-	-	-
	Total	124.00	1.94	7,744,890	-	-	-	-	-	-	-	-
Maleevskoye	Proven	5.04	0.56	90,779	68.13	11,044,289	1.92	96,770	1.00	50,358	6.46	325,627
	Probable	7.06	0.51	115,777	56.23	12,764,963	1.69	119,630	1.04	73,522	6.29	444,011
	Total	**12.10**	**0.53**	**206,233**	**61.19**	**23,810,222**	**1.79**	**216,400**	**1.02**	**123,880**	**6.36**	**769,638**
Novoshirokinskoye	Proven	2.44	3.89	304,564	77.00	6,033,915	-	-	2.98	72,601	1.28	31,089
	Probable	4.43	3.89	552,992	84.30	12,001,085	-	-	2.69	118,896	0.99	43,912
	Total	**6.87**	**3.89**	**857,556**	**81.70**	**18,035,032**	**-**	**-**	**2.79**	**191,498**	**1.09**	**75,000**
Ridder-Sokolniy	Proven	8.95	0.91	262,779	12.85	3,693,712	0.43	38,038	0.32	28,587	0.73	65,195
	Probable	12.05	1.03	339,349	8.44	3,269,230	0.22	26,153	0.37	44,920	0.70	84,476
	Total	**21.00**	**0.98**	**662,127**	**10.32**	**6,962,941**	**0.31**	**64,188**	**0.35**	**73,505**	**0.71**	**149,671**
Shaimerden (Stockpiles)	Proven	-	-	-	-	-	-	-	-	-	-	-
	Probable	2.48	-	-	-	-	-	-	-	-	21.71	539,143
	Total	**2.48**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**21.71**	**539,143**
Tishinskiy	Proven	18.89	0.54	330,299	8.12	4,930,099	0.52	98,415	0.91	172,093	4.22	796,457
	Probable	4.93	0.47	74,056	9.36	1,482,209	0.40	19,621	0.88	43,139	4.13	203,681
	Total	**23.81**	**0.53**	**405,782**	**8.38**	**6,415,955**	**0.50**	**119,068**	**0.90**	**214,323**	**4.20**	**1,000,174**
Dolinnoe	Proven	3.66	3.93	462,258	53.76	6,325,710	0.20	7,385	0.75	27,351	1.41	51,730
	Probable	0.96	2.38	73,822	29.82	923,296	0.14	1,338	0.50	4,849	1.02	9,792
	Total	**4.62**	**3.61**	**536,080**	**48.77**	**7,249,006**	**0.19**	**8,723**	**0.70**	**32,380**	**1.33**	**61,523**
Obruchevskoe	Proven	0.89	1.73	49,363	42.80	1,219,753	0.81	7,161	4.27	37,829	8.98	79,581
	Probable	3.25	0.90	94,019	33.21	3,466,977	0.83	26,845	2.66	86,520	6.50	211,092
	Total	**4.14**	**1.08**	**143,382**	**35.26**	**4,686,731**	**0.82**	**34,006**	**3.01**	**124,349**	**7.03**	**290,673**
Chashinskoye Tailing Dam	Proven	-	-	-	-	-	-	-	-	-	-	-
	Probable	55.53	0.70	1,245,070	5.37	9,588,896	0.05	28,900	0.16	86,700	0.40	219,640
	Total	**55.53**	**0.70**	**1,245,070**	**5.37**	**9,588,896**	**0.05**	**28,900**	**0.16**	**86,700**	**0.40**	**219,640**
Tishinskiy Tailing Dam	Proven	0.00	-	-	-	-	-	-	-	-	-	-
	Probable	0.32	0.33	3,410	9.89	101,790	0.22	710	0.76	3,420	2.44	7,820
	Total	**0.32**	**0.33**	**3,410**	**9.89**	**101,790**	**0.22**	**710**	**0.76**	**3,420**	**2.44**	**7,820**
Staroye Tailing Dam	Proven	0.00	-	-	-	-	-	-	-	-	-	-
	Probable	0.79	2.00	51,000	18.69	475,900	0.05	380	0.48	3,800	1.10	8.720
	Total	**0.79**	**2.00**	**51,000**	**18.69**	**475,900**	**0.05**	**380**	**0.48**	**3,800**	**1.10**	**8.720**

2.3 Extraction Table

Extraction tables for each of the deposits are shown in Table 2.5: Kazzinc Group Extraction to Table 2.17 below.

Table 2.5: Kazzinc Group Extraction

		2008A	2009A	2010A	2011E	2012E	2013E	2014E	2015E
Finished metal production capacity	t								
Zinc	t	300,900	300,900	300,900	300,900	300,900	300,900	300,900	300,900
Lead	t	130,000	130,000	130,000	130,000	172,500	172,500	172,500	172,500
Copper	t	-	-	-	70,000	87,500	87,500	87,500	87,500
Gold	oz	1,511,084	1,511,084	1,511,084	1,511,084	1,511,084	1,511,084	1,511,084	1,511,084
Silver	oz	46,297,041	46,297,041	46,297,041	46,297,041	46,297,041	46,297,041	46,297,041	46,297,041
Finished metal actual / forecast production									
Total									
Zinc	t	299,443	301,104	300,750	300,800	300,800	300,800	300,800	300,800
Lead	t	90,240	79,041	100,789	99,249	160,135	169,054	171,848	171,523
Copper	t	55,956****	59,420****	49,782****	47,700	70,000	87,500	87,500	87,500
Gold	oz	183,428	238,226	347,778	625,068	762,417	815,583	803,270	791,934
Silver	oz	7,617,831	6,285,871	6,730,970	6,470,494	19,349,636	22,857,937	24,375,092	25,212,401
From own mines' concentrate									
Zinc	t	261,904	226,853	239,050	259,543	205,756	253,246	242,516	182,752
Lead	t	43,276	46,472	33,142	42,086	51,986	51,706	44,532	33,748
Copper	t	55,956	57,324	47,955	45,308	30,945	31,456	29,805	19,574
Gold	oz	179,293	232,674	326,315	619,663	542,060	614,432	636,135	598,832
Silver	oz	5,312,570	5,335,199	5,182,404	5,899,184	5,221,692	5,063,672	4,829,110	3,348,772
From third party concentrate*									
Zinc	t	37,538	74,251	61,700	41,257	95,044	47,554	58,284	118,048
Lead	t	46,964	32,570	67,647	57,163	108,149	117,348	127,316	137,775
Copper	t	0	2,096	1,827	2,392	39,055	56,044	57,695	67,926
Gold	oz	4,135	5,552	21,463	5,405	220,357	201,151	167,135	193,102
Silver	oz	2,305,262	950,672	1,548,566	571,310	14,127,944	17,794,265	19,545,982	21,863,629
Cash cost (excl. royalties, before by-product revenues) ***	US$M	997	725	1,040	1,500	2,077	2,460	2,415	2,444
By-products revenues	US$M	411	368	542	491	523	552	477	495
Royalties (as a % market price)									
Zinc		3.5%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead		3.5%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper		3.5%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%

Gold		4.0%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver		4.0%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Royalties	US$M	19.05	67.71	101.61	123.77	106.98	119.39	117.74	97.37
Depreciation & amortisation	US$M	116	147	236					
Tax rate %		30.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%
Capex	US$M	**568****	**367****	**350****	**345**	**299**	**190**	**174**	**312**
Sustaining		199	86	95	243	248	164	139	208
Expansionary		369	280	256	102	51	26	35	104

Note:

* Kazzinc will purchase third party concentrate to load all available capacity in smelters.

**Kazzinc's CapEx for 2008-2010 excludes development loans to Novoshirokinskoye and VasGold in the amount of US$642 million (US$277 million in 2008, US$324 million in 2009, and US$41 million in 2010)

***Cash cost calculations include the cost of purchased concentrates.

****Copper contained in copper concentrates and blister copper

Table 2.6: Maleevskoye Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t	2,250,000	2,250,000	2,250,000	2,250,000	2,250,000	2,250,000	2,250,000	2,250,000
Milled actual / forecast	t	2,216,329	2,357,243	2,230,246	2,250,000	2,137,000	2,196,000	1,710,000	822,000
Grades	g/t								
Zinc	%	6.90	6.71	6.37	6.31	6.14	6.57	7.41	6.69
Lead	%	1.14	1.00	0.99	0.97	1.66	1.6	1.5	1.4
Copper	%	2.06	1.90	1.96	2.01	1	1	1.16	0.94
Gold	g/t	0.61	0.57	0.56	0.55	0.48	0.42	0.64	0.5
Silver	g/t	65.69	59.22	59.73	62.58	58.03	49.82	58.55	55.97
Concentrate produced									
Zn Conc	dmt	248,148	257,050	229,644	225,739	212,647	233,822	205,353	89,122
Metals in concentrate									
Zinc	t	137,603	141,601	126,616	127,087	117,435	129,128	113,406	49,218
Lead	t	1,256	1,302	1,264	1,038	1,703	1,687	1,231	552
Copper	t	2,144	2,052	2,230	1,672	791	813	734	286
Gold	oz	3,379	3,495	3,151	2,396	1,979	1,779	2,111	793
Silver	oz	274,968	297,461	277,138	247,569	219,310	193,481	177,062	81,363
Pb Conc	dmt	33,318	31,452	28,462	29,564	49,214	48,745	35,585	15,965
Metals in concentrate									
Zinc	t	2,632	2,669	2,485	2,513	2,321	2,552	2,241	973
Lead	t	15,642	13,680	12,765	13,599	22,194	21,983	16,048	7,200
Copper	t	773	1,177	1,015	579	274	282	254	99
Gold	oz	1,353	2,113	1,704	2,289	1,888	1,698	2,014	756
Silver	oz	867,105	969,074	878,877	1,066,158	970,770	856,437	783,759	360,153
Cu Conc	dmt	155,254	166,036	157,137	158,225	77,402	79,539	71,846	27,986
Metals in concentrate									
Zinc	t	4,687	5,240	4,853	3,501	3,233	3,555	3,122	1,355
Lead	t	5,510	5,849	5,477	4,400	7,182	7,113	5,193	2,330
Copper	t	39,126	37,798	36,730	38,917	18,407	18,915	17,085	6,655
Gold	oz	14,920	15,134	12,687	12,760	10,526	9,464	11,230	4,217
Silver	oz	3,015,129	2,696,398	2,640,619	2,630,407	2,320,573	2,047,266	1,873,532	860,925
Au Conc	dmt	4,812	4,778	5,341	4,731	4,697	4,826	3,758	1,807

Metals in concentrate									
Zinc	t	151	126	114	165	152	167	147	64
Lead	t	347	295	377	438	715	709	517	232
Copper	t	42	35	30	49	23	24	22	8
Gold	oz	15,846	13,786	14,374	14,256	11,759	10,574	12,546	4,712
Silver	oz	36,193	33,835	44,117	30,399	27,640	24,385	22,316	10,255
Cash cost (excl. royalties, before by-product revenues)	US$M	92.76	89.69	100.13	120.86	115.29	117.21	101.54	72.90
By-products revenues	US$M	323.44	271.33	388.54	342.36				
Royalties (as a % of market price)									
Zinc		2.17%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead		1.95%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper		2.62%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%
Gold		2.07%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver		1.56%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Total Royalty	US$M	16.80	37.46	52.25	47.96	42.79	43.89	37.31	15.45

Table 2.7: Vasilkovskoye Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t			3,520,650	8,000,000	8,000,000	8,000,000	8,000,000	8,000,000
Milled actual / forecast	t			3,520,650	7,470,000	8,000,000	8,000,000	8,000,000	8,000,000
Grade	g/t								
Gold				1.93	2.19	2.03	2.05	2.11	1.97
Concentrate produced				977	0	0	0	0	0
Metals in concentrate	oz								
Gold				5,977	0	0	0	0	0
Alloy Doré	t			4.954	14.587				
Metals									
Gold	oz			117,982	402,296	365,490	369,091	379,893	354,687
Cash cost (excl. royalties, before by-product revenues)	US$M			106.93	123.89	132.68	132.68	132.68	132.68
By-products revenues	US$M			0	0	0	0	0	0
Royalties (as a % of market price)									
Gold				5%	5%	5%	5%	5%	5%
Total Royalty	US$M			15.17	31.63	24.55	24.66	25.26	23.46

Table 2.8: Ridder-Sokolniy Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t	2,354,000	2,150,000	2,300,000	2,125,000	2,125,000	2,125,000	2,125,000	2,125,000
Milled actual / forecast	t	2,126,780	2,086,024	2,036,102	2,125,000	2,125,000	2,125,000	2,125,000	2,125,000
Grades									
Zinc	%	0.51	0.46	0.52	0.47	0.71	0.71	0.71	0.71
Lead	%	0.25	0.24	0.27	0.20	0.35	0.35	0.35	0.35
Copper	%	0.23	0.34	0.30	0.35	0.31	0.31	0.31	0.31
Gold	g/t	1.98	2.22	2.23	1.99	0.98	0.98	0.98	0.98
Silver	g/t	10.35	7.78	13.10	13.98	10.35	10.35	10.35	10.35
Concentrate produced									
Zn Conc	dmt	10,515	8,452	8,947	8,802	13,366	13,366	13,366	13,366
Metals in concentrate									
Zinc	t	5,904	4,590	4,858	4,753	7,217	7,217	7,217	7,217
Lead	t	120	140	247	110	196	196	196	196
Copper	t	144	278	294	256	226	226	226	226
Gold	oz	2,018	2,159	2,137	2,395	1,180	1,180	1,180	1,180
Silver	oz	27,603	20,674	34,083	37,134	27,495	27,495	27,495	27,495
Pb Conc	dmt	0	0	0	20,588	17,700	17,700	17,700	17,700
Metals in concentrate									
Zinc	t	0	0	0	1,222	1,380	1,380	1,380	1,380
Lead	t	0	0	0	2,633	4,850	4,850	4,850	4,850
Copper	t	0	0	0	519	690	690	690	690
Gold	oz	0	0	0	19,798	10,084	10,084	10,084	10,084
Silver	oz	0	0	0	256,434	177,788	177,788	177,788	177,788
Cu Conc	dmt	5,484	24,393	24,341	27,777	24,505	24,505	24,505	24,505
Metals in concentrate									
Zinc	t	166	552	751	853	1,295	1,295	1,295	1,295
Lead	t	65	234	974	304	543	543	543	543
Copper	t	1,351	2,891	2,710	5,768	5,089	5,089	5,089	5,089
Gold	oz	1,149	10,744	17,036	34,871	17,183	17,183	17,183	17,183
Silver	oz	18,792	48,960	124,070	409,761	303,397	303,397	303,397	303,397
Au Conc	dmt	30,805	36,988	32,055	8,170	7,208	7,208	7,208	7,208
Metals in concentrate									
Zinc	t	2,305	1,988	2,420	455	691	691	691	691

Lead	t	4,154	3,262	3,458	327	584	584	584	584
Copper	t	2,765	2,835	2,215	185	163	163	163	163
Gold	oz	83,801	72,718	70,317	31,463	15,504	15,504	15,504	15,504
Silver	oz	486,102	304,866	542,346	79,830	59,108	59,108	59,108	59,108
Ore Flux	dmt	44,470	60,316	59,820	104,394	65,433	65,433	65,433	65,433
Metals in concentrate (oz)									
Zinc	t	365	351	401	417	411	411	411	411
Lead	t	132	147	223	268	196	196	196	196
Copper	t	110	274	369	227	242	242	242	242
Gold	oz	28,332	52,012	37,906	34,444	38,174	38,174	38,174	38,174
Silver	oz	13,908	26,818	37,700	28,431	26,714	26,714	26,714	26,714
Cash cost (excl. royalties, before by-product revenues)	US$M	86.27	65.16	90.25	112.03	112.03	112.03	112.03	112.03
By-products revenues	US$M	26.89	32.36	50.70	47.18				
Royalties (as a % of market price)									
Zinc		0.27%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead		0.21%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper		0.27%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%
Gold		0.16%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver		0.16%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Total Royalty	US$M	0.39	10.89	13.10	13.34	12.83	12.60	12.34	12.06

Table 2.9: Tishinskiy Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t	1,400,000	1,187,500	1,330,000	1,330,000	1,330,000	1,330,000	1,330,000	1,330,000
Milled actual / forecast	t	1,421,038	1,262,510	1,476,486	1,330,000	1,187,500	1,187,500	1,187,500	1,187,500
Grades									
Zinc	%	5.01	5.23	4.61	4.81	5.08	5.08	5.08	5.08
Lead	%	0.72	0.79	0.63	0.55	0.57	0.57	0.57	0.57
Copper	%	0.40	0.43	0.41	0.38	0.35	0.35	0.35	0.35
Gold	g/t	0.81	0.87	0.82	0.78	0.70	0.65	0.65	0.60
Silver	g/t	10.06	10.83	9.69	9.00	9.00	9.00	9.00	9.00
Concentrate produced									
Zn Conc	dmt	115,908	110,061	113,758	106,918	99,834	99,834	99,834	99,834
Metals in concentrate									
Zinc	t	64,703	61,234	60,956	58,245	54,924	54,924	54,924	54,924
Lead	t	1,352	1,497	1,654	1,136	1,060	1,060	1,060	1,060
Copper	t	605	498	784	573	477	477	477	477
Gold	oz	9,369	8,974	10,742	8,578	6,873	6,382	6,382	5,891
Silver	oz	166,658	172,497	181,645	144,389	133,315	133,315	133,315	133,315
Pb Conc	dmt	9,839	9,649	8,747	6,827	6,583	6,583	6,583	6,583
Metals in concentrate									
Zinc	t	762	758	820	660	622	622	622	622
Lead	t	6,993	6,821	5,700	4,693	4,541	4,541	4,541	4,541
Copper	t	80	84	121	72	52	52	52	52
Gold	oz	3,784	3,274	3,659	2,733	2,190	2,033	2,033	1,877
Silver	oz	85,065	81,292	69,038	56,585	52,245	52,245	52,245	52,245
Cu Conc	dmt	15,453	15,211	15,458	13,268	11,322	11,322	11,322	11,322
Metals in concentrate									
Zinc	t	881	837	950	896	845	845	845	845
Lead	t	350	409	434	288	269	269	269	269
Copper	t	4,357	4,020	4,363	3,754	3,127	3,127	3,127	3,127
Gold	oz	8,938	8,644	9,172	8,552	6,853	6,363	6,363	5,874
Silver	oz	89,662	91,003	88,149	70,249	62,844	62,844	62,844	62,844
Au Conc	dmt	445	1,247	421	1,851	753	753	753	753
Metals in concentrate									

Zinc	t	25	117	35	72	72	72	72	72
Lead	t	81	158	68	85	84	84	84	84
Copper	t	6	79	13	24	22	22	22	22
Gold	oz	839	2,051	762	4,463	3,682	3,419	3,419	3,156
Silver	oz	1,807	4,798	1,604	3,537	3,381	3,381	3,381	3,381
Cash cost (excl. royalties, before by-product revenues)	US$M	81.852	50.674	72.105	86.782	77.93	77.93	77.93	77.93
By-products revenues	US$M	46.538	42.266	64.228	54.842				
Royalties (as a % of market price)									
Zinc		0.74%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead		0.57%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper		0.74%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%
Gold		0.44%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver		0.44%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Total Royalty	US$M	1.40	12.94	16.34	14.19	14.54	14.16	13.86	13.39

Table 2.10: Shubinskiy Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t	190,000	200,000	190,000	190,000	190,000	190,000	190,000	190,000
Milled actual / forecast	t	179,457	211,650	194,414	190,000	190,000	190,000	190,000	190,000
Grades									
Zinc	%	1.66	1.74	1.65	1.66	1.66	1.66	1.66	1.66
Lead	%	0.19	0.23	0.22	0.23	0.23	0.23	0.23	0.23
Copper	%	1.15	1.37	0.91	1.00	1.00	1.00	1.00	1.00
Gold	g/t	0.54	0.72	0.55	0.31	0.31	0.31	0.31	0.31
Silver	g/t	12.49	13.94	11.51	8.48	8.48	8.48	8.48	8.48
Concentrate produced									
Zn Conc	dmt	4,947	6,331	5,709	5,196	5,316	5,316	5,316	5,316
Metals in concentrate									
Zinc	t	2,333	2,873	2,432	2,400	2,395	2,395	2,395	2,395
Lead	t	37	57	43	53	53	53	53	53
Copper	t	116	179	127	171	171	171	171	171
Gold	oz	211	330	218	132	134	134	134	134
Silver	oz	9,226	12,782	9,091	6,494	6,491	6,491	6,491	6,491
Cu Conc	dmt	8,084	11,290	6,593	7,025	7,000	7,000	7,000	7,000
Metals in concentrate									
Zinc	t	303	367	330	302	301	301	301	301
Lead	t	249	368	312	310	307	307	307	307
Copper	t	1,821	2,526	1,475	1,568	1,570	1,570	1,570	1,570
Gold	oz	885	1,388	739	402	408	408	408	408
Silver	oz	33,364	42,870	39,989	24,049	24,037	24,037	24,037	24,037
Au Conc	dmt	13	44	0	0	0	0	0	0
Metals in concentrate									
Zinc	t	0	1	0	0	0	0	0	0
Lead	t	1	1	0	0	0	0	0	0
Copper	t	0	1	0	0	0	0	0	0
Gold	oz	8	100	0	0	0	0	0	0
Silver	oz	28	100	0	0	0	0	0	0
Cash cost (excl. royalties, before by-product	US$M	10.35	7.73	10.00	10.30	10.30	10.30	10.30	10.30

revenues)									
By-products revenues	US$M	12.31	11.44	12.85	10.20				
Royalties (as a % of market price)		0.50%							
Zinc			7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead			8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper			5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%
Gold			5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver			5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Total Royalty	US$M	0.04	1.52	1.46	1.38	1.53	1.48	1.42	1.37

Table 2.11: Staroye Tailing Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t	500,000	1,100,000	1,100,000	1,100,000	1,100,000	1,100,000	1,100,000	1,100,000
Milled actual / forecast	t	421,885	936,966	414,636	660,850	660,850	660,850	660,850	660,850
Grades									
Zinc	%	0.92	0.97	0.72	0.67	0.67	0.67	0.67	0.67
Lead	%	0.37	0.35	0.26	0.31	0.31	0.31	0.31	0.31
Copper	%	0.07	0.15	0.06	0.04	0.04	0.04	0.04	0.04
Gold	g/t	1.64	1.76	1.12	1.05	1.05	1.05	1.05	1.05
Silver	g/t	16.56	14.99	11.16	12.25	12.25	12.25	12.25	12.25
Concentrate produced									
Zn Conc	dmt	2,190	6,842	1,635	3,824	3,824	3,824	3,824	3,824
Metals in concentrate									
Zinc	t	771	2,178	354	1,109	1,109	1,109	1,109	1,109
Lead	t	84	303	69	122	122	122	122	122
Copper	t	40	291	54	60	60	60	60	60
Gold	oz	1,324	5,393	941	1,720	1,720	1,720	1,720	1,720
Silver	oz	18,166	49,135	11,950	30,448	30,448	30,448	30,448	30,448
Cu Conc	dmt	0	2,127	3,419	0	0	0	0	0
Metals in concentrate									
Zinc	t	0	234	68	0	0	0	0	0
Lead	t	0	63	50	0	0	0	0	0
Copper	t	0	29	26	0	0	0	0	0
Gold	oz	0	1,364	805	0	0	0	0	0
Silver	oz	0	6,181	6,236	0	0	0	0	0
Au Conc	dmt	11,123	20,838	6,673	8,713	8,713	8,713	8,713	8,713
Metals in concentrate									
Zinc	t	970	2,510	769	1,145	1,145	1,145	1,145	1,145
Lead	t	563	1,411	320	780	780	780	780	780
Copper	t	176	842	123	133	133	133	133	133
Gold	oz	14,129	28,655	6,707	10,950	10,950	10,950	10,950	10,950
Silver	oz	117,585	195,988	57,338	108,900	108,900	108,900	108,900	108,900
Cash cost (excl. royalties, before by-product revenues) ()	US$M	6.30	11.55	7.14	12.16	12.16	12.16	12.16	12.16
By-products revenues	US$M	2.08	10.50	2.24	3.41				

Royalties (as a % of market price)									
Zinc		0.58%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead		0.58%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper		0.58%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%
Gold		0.88%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver		0.68%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Total Royalty	US$M	0.27	4.91	1.71	2.19	1.72	1.70	1.68	1.65

Table 2.12: Shaimerden Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t	305,000	74,711	212,898	276,386	276,386	276,386	276,386	276,386
Milled actual / forecast	t	305,519	74,711	212,898	276,386	100,000	280,000	296,270	300,000
Grades	g/t								
Zinc	%	20.64	21.23	20.66	20.71	21.71	21.71	21.71	21.71
Concentrate produced		305,519	74,711	212,898	287,856	104,154	291,630	308,578	312,461
Metals in concentrate									
Zinc	t	63,074	15,861	43,980	60,001	21,710	60,788	64,321	65,130
Cash cost (excl. royalties, before by-product revenues)	US$M	13.35	0.53	7.04	18.16	7.70	18.37	19.34	19.56
By-products revenues	US$M	0.000	0.02	0.80	1.23				
Royalties (as a % of market price)									
Zinc		1.50%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Total Royalty	US$M	0.17	0.00	0.00	9.06	3.76	10.34	10.75	10.60

Table 2.13: Novoshirokinskoye Deposit

		2008A	2009A	2010A	2011E	2012E	2013 E	2014E	2015E
Milled Capacity	t	-	-	450,000	450,000	450,000	450,000	450,000	450,000
Milled actual / forecast	t	-	-	392,720	450,000	450,000	450,000	450,000	450,000
Grades		-	-						
Zinc	%	-	-	0.81	1.27	1.23	1.23	1.23	1.23
Lead	%	-	-	2.04	2.80	2.70	2.70	2.70	2.70
Copper	%	-	-	0.14	0.00	0.17	0.17	0.17	0.17
Gold	g/t	-	-	3.30	3.48	3.36	3.36	3.36	3.36
Silver	g/t	-	-	60.37	64.90	62.48	62.48	62.48	62.48
Concentrate produced									
Zn Conc	dmt	-	-	4,003	6,723	6,723	6,723	6,723	6,723
Metals in concentrate									
Zinc	t	-	-	1,752	3,051	3,051	3,051	3,051	3,051
Lead	t	-	-	68	0	0	0	0	0
Copper	t	-	-	21	0	0	0	0	0
Gold	oz	-	-	673	1,660	1,660	1,660	1,660	1,660
Silver	oz	-	-	16,459	15,953	15,953	15,953	15,953	15,953
Pb Conc	dmt	-	-	19,136	26,634	26,634	26,634	26,634	26,634
Metals in concentrate									
Zinc	t	-	-	850	2,131	2,131	2,131	2,131	2,131
Lead	t	-	-	6,992	10,823	10,823	10,823	10,823	10,823
Copper	t	-	-	450	1,057	1,057	1,057	1,057	1,057
Gold	oz	-	-	32,087	38,356	38,356	38,356	38,356	38,356
Silver	oz	-	-	634,379	785,571	785,571	785,571	785,571	785,571
Cash cost (excl. royalties, before by-product revenues)	US$M	-	-	33.23	39.84	39.84	39.84	39.84	39.84
By-products revenues	US$M	-	-	22.73	31.85				
Royalties (as a % of market price of concentrate)		-	-	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%
Total Royalty	US$M	-	-	1.57	4.02	5.27	5.21	5.14	5.07

Table 2.14: Dolinnoe Deposit

		2014E	2015E	2016E	2017E	2018E	2019E	2020E	2021E
Milled Capacity	t	100,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000
Milled actual / forecast	t	100,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000
Grades	g/t								
Zinc	%	1.33	1.33	1.33	1.33	1.33	1.33	1.33	1.33
Lead	%	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70
Copper	%	0.19	0.19	0.19	0.19	0.19	0.19	0.19	0.19
Gold	g/t	3.61	3.61	3.61	3.61	3.61	3.61	3.61	3.61
Silver	g/t	48.77	48.77	48.77	48.77	48.77	48.77	48.77	48.77
Zn Conc	dmt	1,793	4,483	4,483	4,483	4,483	4,483	4,483	4,483
Metals in concentrate									
Zinc	t	1,004	2,510	2,510	2,510	2,510	2,510	2,510	2,510
Lead	t	21	52	52	52	52	52	52	52
Copper	t	12	30	30	30	30	30	30	30
Gold	oz	20	49	49	49	49	49	49	49
Silver	oz	392	980	980	980	980	980	980	980
Pb Conc	dmt	2,842	5,683	2,323	2,323	2,323	2,323	2,323	2,323
Metals in concentrate									
Zinc	t	53	133	133	133	133	133	133	133
Lead	t	511	1,278	1,278	1,278	1,278	1,278	1,278	1,278
Copper	t	4	10	10	10	10	10	10	10
Gold	oz	25	62	62	62	62	62	62	62
Silver	oz	858	2,146	2,146	2,146	2,146	2,146	2,146	2,146
Cu Conc	dmt	1,873	3,745	1,273	1,273	1,273	1,273	1,273	1,273
Metals in concentrate									
Zinc	t	11	27	27	27	27	27	27	27
Lead	t	9	23	23	23	23	23	23	23
Copper	t	138	344	344	344	344	344	344	344
Gold	oz	63	157	157	157	157	157	157	157
Silver	oz	1,444	3,609	3,609	3,609	3,609	3,609	3,609	3,609
Au Conc	dmt	938	2,344	2,344	2,344	2,344	2,344	2,344	2,344
Metals in concentrate									
Zinc	t	285	571	184	184	184	184	184	184
Lead	t	206	411	139	139	139	139	139	139

Copper	t	46	93	26	26	26	26	26	26
Gold	oz	3,554	8,885	8,885	8,885	8,885	8,885	8,885	8,885
Silver	oz	2,096	5,239	5,239	5,239	5,239	5,239	5,239	5,239
Cash cost (excl. royalties, before by-product revenues)	US$M	5.36	13.39	13.39	13.39	13.39	13.39	13.39	13.39
By-products revenues	US$M								
Royalties (as a % of market price)									
Zinc		7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%
Lead		8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%
Copper		5.7%	5.7%	5.7%	5.7%	5.7%	5.7%	5.7%	5.7%
Gold		5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
Silver		5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
Total Royalty	US$M	4.48	8.85	1.48	1.41	1.24	1.21	1.20	1.20

Table 2.15: Chashinskoye Tailings

		2013E	2014E	2015E	2016E	2017E	2018E	2019E	2020E
Milled Capacity	t	3,869,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000
Milled actual / forecast	t	3,869,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000
Grades	g/t								
Zinc	%	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40
Lead	%	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16
Copper	%	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Gold	g/t	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70
Silver	g/t	5.37	5.37	5.37	5.37	5.37	5.37	5.37	5.37
Au Conc	dmt	10,240	10,585	10,585	10,585	10,585	10,585	10,585	10,585
Metals in concentrate									
Zinc	t	-	-	-	-	-	-	-	-
Lead	t	-	-	-	-	-	-	-	-
Copper	t	-	-	-	-	-	-	-	-
Gold	oz	74,571	77,089	77,089	77,089	77,089	77,089	77,089	77,089
Silver	oz	254,606	261,899	261,899	261,899	261,899	261,899	261,899	261,899
Cash cost (excl. royalties, before by-product revenues)	US$M	36.94	35.59	35.59	35.59	35.59	35.59	35.59	35.59
By-products revenues	US$M								
Royalties (as a % of market price)									
Zinc		7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead		8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper		5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%
Gold		5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver		5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Total Royalty	US$M	**5.35**	**5.50**	**5.47**	**5.31**	**4.85**	**3.56**	**3.51**	**3.46**

Table 2.16: Obruchevskoe Deposit

		2016E	2017E	2018E	2019E	2020E	2021E	2022E	2023E
Milled Capacity	t	70,000	350,000	350,000	350,000	350,000	350,000	350,000	350,000
Milled actual / forecast	t	70,000	350,000	350,000	350,000	350,000	350,000	350,000	350,000
Grades	g/t								
Zinc	%	7.03	7.03	7.03	7.03	7.03	7.03	7.03	7.03
Lead	%	3.01	3.01	3.01	3.01	3.01	3.01	3.01	3.01
Copper	%	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82
Gold	g/t	1.08	1.08	1.08	1.08	1.08	1.08	1.08	1.08
Silver	g/t	35.26	35.26	35.26	35.26	35.26	35.26	35.26	35.26
Concentrate produced		13,513	67,563	67,563	67,563	67,563	67,563	67,563	67,563
Zn Conc	**dmt**	7,988	39,939	39,939	39,939	39,939	39,939	39,939	39,939
Metals in concentrate									
Zinc	t	3,071	22,366	22,366	22,366	22,366	22,366	22,366	22,366
Lead	t	63	314	314	314	314	314	314	314
Copper	t	36	180	180	180	180	180	180	180
Gold	oz	4.1	21	21	21	21	21	21	21
Silver	oz	198.4	992	992	992	992	992	992	992
Pb Conc	**dmt**	3,082	15,410	15,410	15,410	15,410	15,410	15,410	15,410
Metals in concentrate									
Zinc	t	197	984	984	984	984	984	984	984
Lead	t	1,818	9,092	9,092	9,092	9,092	9,092	9,092	9,092
Copper	t	13	63	63	63	63	63	63	63
Gold	oz	5	26	26	26	26	26	26	26
Silver	oz	434	2,172	2,172	2,172	2,172	2,172	2,172	2,172
Cu Conc	**dmt**	1,786	8,932	8,932	8,932	8,932	8,932	8,932	8,932
Metals in concentrate									
Zinc	t	41	203	203	203	203	203	203	203
Lead	t	27	136	136	136	136	136	136	136
Copper	t	509	2,546	2,546	2,546	2,546	2,546	2,546	2,546
Gold	oz	13	66	66	66	66	66	66	66
Silver	oz	731	3,653	3,653	3,653	3,653	3,653	3,653	3,653
Au Conc	**dmt**	656	3,282	3,282	3,282	3,282	3,282	3,282	3,282
Metals in concentrate									
Zinc	t	285	285	285	285	285	285	285	285

Lead	t	206	206	206	206	206	206	206	206
Copper	t	46	46	46	46	46	46	46	46
Gold	oz	1,799	8,994	8,994	8,994	8,994	8,994	8,994	8,994
Silver	oz	229	1,145	1,145	1,145	1,145	1,145	1,145	1,145
Cash cost (excl. royalties, before by-product revenues)	US$M	3.08	15.38	15.38	15.38	15.38	15.38	15.38	15.38
By-products revenues	US$M								
Royalties (as a % of market price)									
Zinc		7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%	7.00%
Lead		8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%
Copper		5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%	5.70%
Gold		5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Silver		5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Total Royalty	US$M	1.52	6.98	6.68	6.52	6.43	6.43	6.43	6.43

Table 2.17: Kazzinc Smelter

		2008	2009	2010	2011	2012	2013	2014	2015
Concentrate Processed actual / forecast	t	-	-	-	-	752,903	973,591	947,940	778,971
Finished Metal									
Zinc	t	-	-	-	-	205,756	253,246	242,516	182,752
Lead	t	-	-	-	-	51,986	51,706	44,532	33,748
Copper	t	-	-	-	-	30,945	31,456	29,805	19,574
Gold	oz	-	-	-	-	542,060	614,432	636,135	598,832
Silver	oz	-	-	-	-	5,221,692	5,063,672	4,829,110	3,348,772
Smelter Cash cost (excl. royalties, before by-product revenues)	US$M	-	-	-	-	144.2	163.8	154.3	114.8

3 VASILKOVSKOYE DEPOSIT

3.1 Introduction

3.1.1 Location & Access

The Vasilkovskoye gold deposit is located in northern Kazakhstan, 17km north of the city of Kokshetau. Its geographical coordinates are 53° 26′ 13.8′′ North and 69° 14′ 59.33′′ East (Figure 3.1).

Figure 3.1: Location of Vasilkovskoye, Northern Kazakhstan

Kokshetau city is the administrative centre of Akmola Oblast and has a population of approximately 147,000 (2010), and is some three hours by car northwest of Astana (approximately 300km), the State capital, which is served by regular international and internal flights.

Road connections are excellent, as are power, water and communications. Overall, the district is economically developed and has well established infrastructure.

The deposit was discovered in 1963, but it was not until 1980-1986 that a pilot open pit mining project was undertaken, where some of the ore was processed through a pilot plant, although most was stacked.

Due to the complicated economic situation within the country, mining was suspended, but then continued again in 1995 until the end of 2007, mining the oxide ores which were treated using heap leach technology. Production up to this time is believed to be approximately 14Mt.

3.1.2 Topography & Climate

The Vasilkovskoye mine lies in an area of typical Kazakh steppe, with little or no change in elevation, dominated by grasslands with occasional isolated scrub and small trees. The area is drained by the Ishim and Nura rivers.

Kokshetau exhibits a typical harsh continental climate with hot summers and cold winters. Average January temperature is -16 to -18°C, and +19 to +21°C in July. Annual precipitation is some 250-400mm.

3.1.3 Infrastructure

Kokshetau city was established on the edge of Kopa Lake, a major feature of the town. There are urban light industries such as food production, whilst the region produces low grades of wheat and other crops such as potatoes, vegetables and sunflower.

The world's only power line with a voltage greater than 1000kV is installed near Kokshetau, connecting the town of Ekibastuz with the central part of Russia.

In terms of mineral resources, some 19 ore deposits and 155 non-metallic deposits have been described in the region, including 6 operating gold mines, 4 uranium, 1 titanium, 1 antimony, 2 ornamental stone, 6 coal, 6 limestone, 14 mortar sand and 16 building stone deposits.

3.1.4 Mineral Rights & Permitting

In August 2002, JSC Vasilkovskoye Zoloto obtained a Mining Allotment for the extraction of gold ores from an area of 2.65km^2 to a depth of 660m (level 425m).

In July 2003 a contract with the Kazakhstan Government was awarded granting the right to extract gold bearing ore within the abovementioned Mining Allotment till 2022. Moreover, a mining allotment was obtained for development of the whole mine infrastructure.

In 2010, the existing Mining Contract was amended by a Supplementary Agreement which mainly resulted in the following:

- Extended Mining Allotment of 28.3km^2 (see co-ordinates in Table 3.1);
- Ore processing expanded up to 8Mtpa;
- Extended validity of the Contract till 2025;
- Assignment of the contractual right from JSC "JV "Vasilkovskoye Zoloto" in favour of Kazzinc Ltd.

Table 3.1: Mining Licence Coordinates		
Co-ordinate	**Easting**	**Northing**
1	69° 12' 54.70"	53° 27' 34.35"
2	69° 15' 34.32"	53° 27' 32.33"
3	69° 18' 10.02"	53° 28' 01.22"
4	69° 18' 30.13"	53° 27' 37.79"
5	69° 17' 26.32"	53° 27' 05.03"
6	69° 17' 48.06"	53° 26' 18.84"
7	69° 16' 02.33"	53° 24' 30.80"
8	69° 12' 50.14"	53° 24' 39.50"

WAI Comment: *The licence documentation has been inspected and is in order. Moreover, the conditions of the licence are sufficient for the life of mine, particularly now that the Mining Allotment has been extended around the property and which now also includes adjacent exploration properties.*

3.2 Geology & Mineralisation

3.2.1 Regional Tectonics

The Vasilkovskoye deposit is located within the Altai-Sayan Orogenic Belt, more precisely within the endo-exocontact of the Zerendinsky Granitoid Complex Massif, which intruded Pre-Cambrian metamorphic rocks. Both are overlaid by Paleozoic and Cenozoic sediments (the Mesozoic period is represented by a weathering phase).

The Altai-Sayan Orogenic Belt extends from northern Kazakhstan, through China, Mongolia and into Russia. In Kazakhstan the western extension of this belt is referred to as the Charsk Gold Belt, which hosts numerous gold deposits. The belt forms part of the Altaids orogen, which is at the same time included in the Central Asia orogenic region.

Central Asia is an orogenic supercollage consisting of three generations of orogenic collages that are grouped into the Timanides-Baikalides, Altaids and Mongolides. The supercollage is separated by the giant Trans-Eurasian fault system into the Altai-Mongolian and Kazakhstan-Khingan domains.

The Altaids have significant metallogenic importance (including world-class Au, Ag, Cu-Mo, Pb-Zn, and Ni, of late Proterozoic to Mesozoic age). It is widely accepted that subduction-related orogenesis of the Altaids started in the late Precambrian and gradually migrated southward (present co-ordinates).

3.2.2 Regional Stratigraphy

3.2.2.1 General

The regional geology of Vasilkovskoye deposit comprises a Precambrian metamorphic basal complex, intruded by Ordovician granitoids, and overlain by later Paleozoic sediments. The area was subjected to Mesozoic weathering and overlain by Cenozoic sand and clay sediments. Figure 3.2 shows the geology of Vasilkovskoye and the surrounding area.

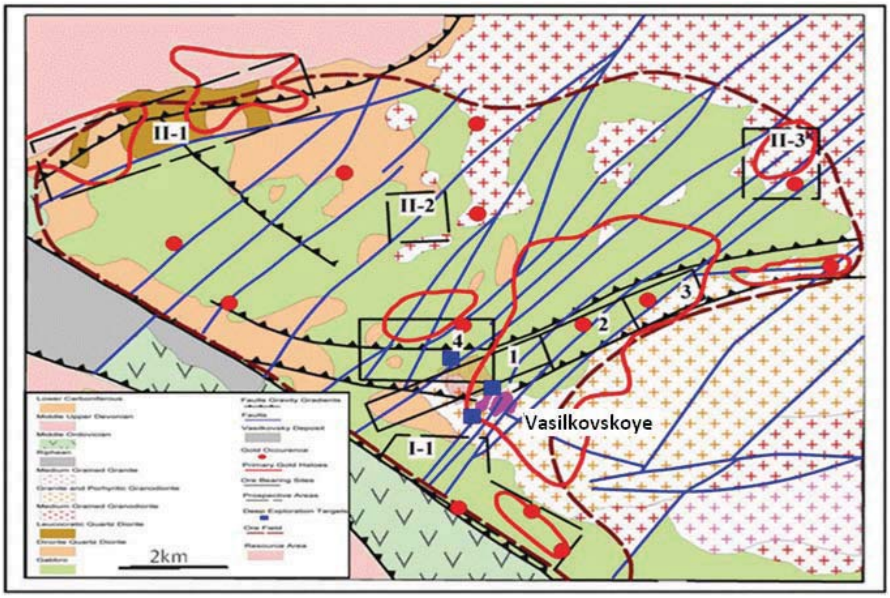


Figure 3.2: Geology of Vasilkovskoye and Surrounding Area

3.2.3 Mine Geology

The Vasilkovskoye deposit belongs to the group of intruded-related Au deposits. It is located in the western part of the Shatskaya metaliogenic zone of the North Kazakhstan province, with a northwest trend.

The deposit is located in the endo-exo contact of the Altybaysky granite intrusion (part of the Zerendinsky Complex) with the Precambrian metamorphic rocks. Within this contact occurs a large gold gas chemical halo, which represents a thermal aureole caused by the granitic intrusion, and where the gold enrichment took place.

The Altybaysky intrusion is crossed by a considerable number of faults, with main directions being NW and NE. These two main faults are the Vasilkovskoye fault (NE strike) and a major northwest trending structure parallel to the Dongulagashsky fault (NW strike). The deposit is situated in the northeast quadrant of this intersection.

The Altybaysky granite intrusion is represented by two major units at the Vasilkovskoye deposit; a diorite/gabbrodiorite unit and a granite/granodiorite unit. Planar and sharp contacts between the two units are common, although sometimes contacts are different.

The diorite/gabbrodiorite unit is located in the northern part of the deposit, and shows generally weak mineralisation. This unit represents the transitional zone between the metamorphic host rocks and the Altybaysky intrusion granite rocks. The Granite/Granodiorite Unit occupies the majority of the deposit and hosts the mineralised body.

In the southern part of the deposit, at depths of 400-500m, quartz diorites, diorites and gabbro-diorites have been intersected which may be related to the Stepnyaksky Complex (the main gold deposits of North Kazakhstan province are paragenetically related to this complex).

3.2.4 Mine Structure

The deposit's area is characterised by the presence of intense faulting and sheared zones. First, second and third order faults can be distinguished, with the main first order faults represented by the NW major structure parallel to the Dongulagashsky fault and the NE trending Vasilkovskoye fault. The rest of the faults trend in various directions such as N, NW or NE.

After detailed structural mapping, seven dislocation orientations have been identified:

- North Western (NW 300-320°) of variable dip, widespread and occur as isolated fractures and zones of jointing. They are 2-10cm wide, rectilinear, contain ferruginous fault gouge and fine schistosity along vein margins. They are related to the Dongulagashsky regional fault. They are pre-ore but were reactivated during the ore forming event;
- North Western (NW 330-350°) dipping steeply NE and SW (relatively uncommon);
- North (N 350-10°) vertical and rarely dip steeply to the NE and SW. These are numerous and occur as thick (5-7m) zones of schistosity, lenses of brecciated rocks and narrow zones of tectonic fault gouge. They are curvilinear and were reactivated into shear zones;
- North Eastern (NE 15-30°) dipping steeply to vertical, predominantly SE (65 to 90°). These are predominant and may be part of a very thick zone of a large regional fault. Steeply dipping (75 to 90°), narrow fractures (0.5 to 25-30cm) control the distribution of ore veins of the deposit. The longest of these veins is 143m, but they generally pinch out between 1-2m and tens of metres. Quartz-arsenopyrite and arsenopyrite veinlets (0.5 to 3cm) develop at the contacts of the quartz veins. The frequency of the veins and veinlets varies from one per 5-7m to hundreds per metre. The frequency is determined by intersecting fractures. The highest frequency occurs in blocks bounded by submeridional dislocations. At the intersection of the NE veins and veinlets with NW dislocations, clusters of veinlets merge into veins along NW fractures;
- North Eastern (NE 40-60°), dipping 30 to 55°. They are predominant and may be part of a very thick zone of a large regional fault. The thickness of isolated schistosity zones is up to 5-10m. Rectilinear surfaces, considerable extension and do not enclose productive assemblages;
- East (E 80-100°) various dips predominantly to the north (45°). Uncommon; and
- North Western (NW 290-310°), gently dipping to the northeast at 10 to 15°. They are up to 30cm wide and are related to the ore phase.

3.2.5 Alteration

Alteration zones exist in the deposit area and are seen as steeply dipping lenticular zones.

The different types of alteration are: chlorite; chlorite-albite; albite; carbonate (known as listvenite, a term used for CO_2 metasomatism of an ultramafic rock); quartz-sericite (known as beresite); quartz-k-feldspar; k-feldspar; argillite alteration.

The bodies decrease in frequency with increasing distance from the centre of the mineralisation. The most intensive alteration is confined to areas of increased fracturing.

Syn-mineral alteration accompanying the sulphide-quartz auriferous stockwork and veins are on beresitised rocks. It comprises grey quartz, the filling of vesicles and minor recrystallisation of rock minerals (sericite, carbonate etc).

The post mineralisation stage comprises the development of argillite alteration comprising 2-5cm veins, predominantly of carbonate composition.

3.2.6 Style of Mineralisation

Vasilkovskoye represents a stockwork located at the junction of the gabbro/gabbrodiorite and the granite/granodiorite rocks, recognising that the ore control was the intersection of faulting parallel to the NW trending Dongulagashsky fault with the NE trending Vasilkovskoye fault. The mineralisation forms a flattened zone, pinching out at depth.

3.2.7 Mineralogy

Au mineralisation is spatially associated with quartz and quartz-arsenopyrite veins and veinlets of hydrothermal origin. The dominant sulphides are arsenopyrite (which occurs as sulphide veinlets, with quartz veins and as disseminations), with minor pyrite plus chalcopyrite (which occur only in trace amounts), bismuth sulphides and bismuth tellurides. Uranium mineralisation is spatially associated with the Au mineralisation, but is younger than the main Au mineralising event. Minor Au occurs within carbonate and silicates,

Uranium mineralisation is present within the deposit, and in the area of the pit, values over 0.1% uranium can be found, with rare, narrow higher grades. Outside of the pit envelope, uranium grades >1% occur.

All the sulphides occur in both major rock units, but pyrite and chalcopyrite are more common in the mafic rocks. Due to the extensive weathering in the upper levels of the pit (now mined out) the sulphides were typically altered to various oxide minerals.

Microcrystalline grey quartz veins and veinlets carry the higher grade Au mineralisation. The veins are commonly high angle, either en echelon or slightly oblique to the direction of the main faults, and relatively thin (< 3cm). In the ore zones the concentration of veinlets ranges from 1-5/m to >10/m. Au ranges in size from 1 to 63 microns, and the average size in the gabbro is 2.5 microns and in granodiorite 4 microns. Detailed mineralogy has indicated that most of the gold occurs as less than 10 micron grains within arsenopyrite.

Mineralisation has been divided into early and late stages. All stages include some sulphides, especially arsenopyrite, complicating interpretation of element associations within the ore deposit, especially the As and Au relationships.

Pre Ore Phase: The earliest sulphides include chalcopyrite-pyrrhotite-pyrite, arsenopyrite-pyrite and Au assemblages. The sulphides are well developed within the gabbro suite, especially adjacent to the contact with the granitic suite and in strongly jointed areas. The chalcopyrite-pyrrhotite-pyrite assemblage is usually associated with the development of mafic minerals.

Early Ore Phase: The early ore phase is dominated by the introduction of quartz, carbonates, sericite and pyrite (listvenite and beresite alteration).

Main Ore Phase: The main ore phase is essentially a quartz, carbonate and chalcophile mineral assemblage, with the introduction of complex copper, bismuth, antimony, arsenic and tellurium minerals.

Post Ore Phase: The post ore phase is a waning of the system with the introduction of silica, carbonate and minor complex sulphides. Uranium was deposited during the post ore phase.

3.2.8 Exploration Potential

In February 2010, the Company expanded its Mining Allotment to 28.3km^2 to include some surrounding exciting exploration projects.

Geophysical exploration has indicated that the Vasilkovskoye deposit appears localised on an IP Gradient area, particularly noticeable from chargeability, but also from resistivity.

A limited volume of geophysical works (IP-R) was undertaken at Promozutichnoye, located 2km to the northeast of the main deposit in 2008. This followed on from a small hydrocore drilling and trenching (1,100m) programme undertaken in 2002 where trenching revealed 36 samples with "industrial grades", with a maximum grade of 5.6g/t, and drilling revealed 39 samples with a maximum grade of 7.7g/t.

In 2010, some 13,943m of hydrocore drilling was undertaken, producing 6,400 samples from 462 holes. Chemical analysis is being undertaken at Vasilkovskoye, although only some 2,200 sample assays have been returned with values up to 0.3g/t Au.

Although not spectacular, the geophysical signatures still appear promising for the delineation of deeper-seated mineralisation (historic Soviet deep drilling (>200m) had picked up some high grades in this area).

A budget for this 2010 programme was set at US$1.3M which included geophysical and topographical baseline works (180km and 190km respectively), trenching, hydrocore drilling and analytical work. Results are awaited from this work.

In addition, the Company has designed an exploration programme for some 8 years within an expanded 1,600km^2 area which includes Turan in the west which is currently being explored by Kostanai Expedition (a sub-contractor to Kazakhstan Mineral Corporation).

> ***WAI Comment:*** *The geology and mineralisation of the Vasilkovskoye deposit are well known through years of exploration and development. Importantly, controls to mineralisation are also well defined, and are clearly related to prominent fracture systems. Thus, WAI has a high degree of confidence in the magnitude, tenor and morphology of mineralisation.*
>
> *Furthermore, this understanding has led to the discovery of other, nearby exciting targets which are, and will be, the focus of future exploration activity. WAI believes that the potential of the Vasilkovskoye area for further significant discoveries is high.*

3.3 Exploration History

3.3.1 General

During the 1960s, the deposit was explored on a 60m x 60m rectangular grid, with a more detailed 30m x 30m grid in the central part of the deposit.

The deposit has been investigated by means of core drilling from surface along 18 exploration profiles (azimuth 315°) each 60m apart, as well as from underground. All surface holes are inclined, and the underground holes are drilled directionally.

Between 360-400m, the drill grid was 60 x 60m, from 360-600m, the grid was 60 x 60m and 120 x 120m, with a maximum exploration depth of between 800-1,000m.

Three shafts were sunk to depths of 60m and 180m, with horizontal drives on three levels at 60m intervals (60m, 120m and 180m elevations) from which raises and drilling chambers were developed. Cross-cuts were developed from the longitudinal drives every 60m along the exploration profiles of core drilling. The length of the cross-cuts was selected so as to intersect the whole width of the mineralised zone.

Within the central part of the deposit short longitudinal drives were developed to detail the deposit structure and grade. All the exploration works (Table 3.2 and Table 3.3) have been surveyed in a local grid coordinate system, with no UTM equivalent coordinates provided.

Table 3.2: Historical Exploration Works Up To 1990							
Work types	Unit	Years					Total
		1966 -1970	1971-1975	1976-1980	1980-1987	1988-1990	1971-1990
Core drilling from surface	m	36,500	27,900	48,000	73,308	4,471	190,179
Underground core drilling	m	5,495	7,305	3,729	1,303	-	17,832
Exploration shafts	m	60	180	180	-	-	420
Raises	m	-	436	260	121	-	817
Test pits 4m^2 size	m	-	403	93		-	496
Drives from the shafts	m	2,439	5,394	9,615	1,015	-	18,463
Lateral drives from the test pits	m	-	1,018	-	-	-	1,018
* From KMC's Resource Estimation report 2009							

Table 3.3: Historical Sampling Methods					
Sampling Method	Year	Number of holes	Metres drilled	Metres sampled	Number of samples
Drillholes	1966-2007	580	212,969	176,482	99,707
Channel	1966-1987	1,129	32,125	31,755	31,230
Blastholes	1995-2007	65,628	196,884	196,884	65,628
* 2009 drillholes not included					

3.3.2 Core Drilling

3.3.2.1 Historical Drilling

The deposit has been drilled using core drilling on profiles oriented at an azimuth of approximately 315°. This direction is approximately 90° to the orientation of the contact between the two main rock units and the main NE trending fault (Vasilkovskoye fault).

The drillholes were drilled at a diameter of either 59mm or 76mm for the majority of the deposit. All holes were drilled conventionally, and most of them sub-vertical (approximately 80°) for the first 200m or so, and then gradually deviated to a dip of 45°. The holes were drilled on 60m spaced sections with some widening of this spacing at the edges of the drill pattern. Spacing of collars along drill sections is generally between 30m and 60m, reducing to 5m or less on some profiles.

Up until 2009, a total of 580 holes has been drilled over the mine area, of which 192 were drilled from underground. A current drilling campaign is underway to extend and infill the resource, details of which are provided below. For the historic data, the average core recovery is 84.3% for Soviet holes (1975-1990). Placer Dome's core recovery was reported in AMC Consultant's report as 100%. Micromine's report (2005) indicates that drillholes completed prior to 1975 had core recoveries of 70% applied to all samples.

From 1963 to 1995 a total of 570 drillholes was drilled (including four geotechnical drillholes) and 96,358 samples collected. Most of the core samples were assayed for Au, As and Bi.

In 2007, 10 more drillholes were drilled, under the advice of the AMC Consultants (Australia), and 3,349 samples collected, as a part of a validation campaign.

3.3.2.2 Current Drill Programme

During 2009, a further programme of core drilling was undertaken to update the Mineral Resources (within the open pit to 450m depth) currently classified in the Inferred Category to Indicated status, under the guidelines of the JORC Code (2004).

Following this, starting in April 2010, an extensive drilling programme has been taking place for detailed exploration of the deeper levels of the deposit. Currently, 8 rigs are operating in the pit, provided by the contractor Iscander, managed by KMC, with samples going to both Ridder laboratory and Stewart Group laboratories, Karabalta, Kyrgyzstan.

The rigs comprise one LF 90 (LongYear), five XY44 (Chinese) and two Brasilian (BBS56). Another rig is planned to join the programme shortly. All rigs drill NQ core, with holes planned to variable depths, with a maximum of 1,050m.

Holes are drilled on a 315° azimuth, but with variable inclination, usually 60° although some future holes are planned to be drilled at 115°. Hole spacing is 60m.

A total of 31,457m of drilling was undertaken inside the open pit and 17,946m was drilled outside the confines of the pit in 2010 and a further 25,000m is scheduled in 2011. However, full results from this deep drilling programme are not expected until November 2011. The budget for this work is some US$12M.

3.3.2.3 Sampling Methodology

WAI inspected several of the drill rigs operating in the pit. The quality of both the drilling and core handling appeared good. All drilling utilised wireline techniques with LongYear tools.

Core is collected on site in well built and labelled six-row wooden trays. These are then transported to the core storage facility where the core is logged, photographed and marked up for sampling. The Company uses "Ballmark" core orientation system for downhole structural control.

Samples are usually 1m in ore and 3m in waste. Once samples are marked up, they go to the core cutting room for splitting. Currently, three core saws are in operation. From here, samples are placed into large sacks for transportation to either the Ridder laboratory, or Alex Stewart in Kyrgyzstan.

3.3.2.4 Sample Analysis

The Company has currently sent some 22,187 samples to the Stewart Group laboratories and has received some 4,000 results back.

To accelerate results, the Kyrgyzstan laboratory has sent 2,500 samples to the Ridder laboratory as well as 5% check samples comprising standards (0.5, 1.52 and 6.0g/t Au) and blanks. The Company has also used repeat samples as standards in an effort to test the competence of the Ridder facility.

3.3.3 Channel Sampling

The surface and underground workings were sampled by channel sampling, with a length of 0.1-2.5m and an average sample length of 1.0m. The channel size was 10x3cm and a 1 metre-long sample typically weighted 8kg. In total, some 31,230 samples were collected, 94% of them 1.0m long.

The drive faces were sampled every 2-3 blasts. The long cross-drives were continuously sampled on both walls, while the shorter drives on one wall only. In raises, samples were collected from two opposite walls every metre.

3.3.4 Core Sampling

During the Soviet period, sectional core sampling was performed with a section length of 3.0-3.2m, average length of sample was 1.6-1.7m. Typically one drilling run core comprised one sample. All the mineralised core intervals were fully sampled with sampling of inter-mineral intervals of enclosing rocks. Unaltered rocks beyond mineralised ore zones were not sampled – the total volume of non-sampled core was 25.4%.

For drilling diameter 76mm and more, a half core was taken as a sample, for 59mm, the core was fully sampled. The average weight of core sample ranged from 2.2kg to 11.0kg depending on drilling diameter and sample length. Slimes were not sampled.

Core holes drilled in 1994-95 by Placer Dome and in 2007 by AMC (drilling diameter 76mm), were cut into two parts. One half was sent to PDI (Canada) and Alex Stewart (Kyrgyzstan) laboratories. The second half was used for metallurgical testwork, whilst the remaining part was stored.

3.3.5 Blastholes

65,628 blastholes to 7.5m depth have been drilled on different bench levels, from the level 226mRL to the level 175mRL.

The vertical blastholes were spaced between 5x5 to 7x7m. Up to 2009, blastholes were sampled by one sample to the whole depth of 7.5m, but since 2009, three samples are taken from the hole, 2.5m in length, as ore is mined on 2.5 flitches. Generally the material size is -10mm, with a sample weight of 8-10kg.

In 2009-10, tests on operational sampling using RC inclined holes were performed, in an attempt to better delineate the predominantly vertical/sub-vertical mineralisation.

In order to develop the RC drilling procedure and compare the RC drilling with blastholes, a trial block (167-2) has been drilled on the south-western flank of the Main ore zone. Blastholes were drilled using a ROC-L8 rig with a simple air hammer. All the holes were vertical. RC drillholes were drilled using a ROC-L8 with a reverse flushing and automated sampling setup. All the drillholes were 53° inclined to ensure maximum ore intersections. The inclined drillholes azimuth complies with the exploration grid of 315°.

Each borehole was sampled by sections (three samples for each section). Sample length was 2.5m in the vertical drillholes, and 3m in the inclined drillholes. The results of this work are shown in Table 3.4 below.

Table 3.4: Inclined RC vs Vertical Blasthole Comparison

	Drilling with ROC L-8 RC			Drilling with ROC L-8 DBH			RC/ DBH	
	ore, tons	grade, g/t	Au, g	ore, tons	grade, g/t	Au, g	grade, g/t	Au, g/t
1 level	24 411	3.06	74 746	24 411	1.90	46 305	1.61	1.61
2 level	24 411	2.80	68 418	24 411	1.74	42 484	1.61	1.61
3 level	24 411	2.41	58 843	24 411	1.89	46 093	1.28	1.28
Total	73 233	2.76	202 006	73 233	1.84	134 882	1.50	1.50

The comparison was carried out within the RC drilling outlines. The results showed that the vertical blastholes underestimate the grade by approximately 30%.

As a result of this work, from the end of January 2011, a 10x15m spacing RC drilling programme, at 53°, will be implemented when the Company will principally use the inclined data for grade control.

3.3.6 Density

WAI Comment: *The Vasilkovskoye project has been well studied with historic exploration works done to a high standard. WAI has, as part of the resource estimation process, conducted a detailed audit of the data and in general, been satisfied of the procedures used and results received.*

The latest on-going drilling works are also being undertaken to a good standard, but the current problems of getting analysis done is of concern, as has been experienced by many other explorers and producers in Kazakhstan who have historically used the Alex Stewart facility in Kyrgyzstan.

Although no assays are available for the deep drilling project, preliminary geological interpretation does indicate that overall resources are likely to be increased.

3.4 Mineral Resource

3.4.1 Introduction

In 2006, AMC prepared a test resource model ("the 2006 Model") using an interpretation provided by Vasgold (operator of Vasilkovskoye Mine), and recommended that Vasgold should drill ten more diamond drillholes for resource confirmation. On completion of these holes AMC compared the results with the previous drilling and updated the resource model where required.

The confirmation drill holes were completed in late 2007 and the assay results sent to AMC in June 2008. AMC compared the results with the previous drilling and found that although there were differences in the data sets, globally the new data supported the older data and as such they were satisfied that the old data was suitable for use in resource estimation at Vasilkovskoye.

AMC revised the interpretation of the mineralisation outline, as defined by a 0.4g/t Au cut-off, with more emphasis placed on the new data where there was discordance between the old and new data. The directional variography was updated to reflect the trend of the higher grades in the primary grade domain (called "the Main Zone") and was oriented with the major axes sub-parallel to the Vasilkovskoye fault.

In situ bulk density for the Main Zone was updated, with the additional information provided by the 2007 drilling and the density for the Main Zone was set at 2.75t/m^3. The density of 2.68t/m^3 used in the 2009 Model was used for all other domains.

Ordinary kriging was used to estimate Au and As grades into the block model (2008 Model) and the reported Mineral Resource above a 0.4g/t Au cut-off was 150Mt @ 1.90g/t Au (*Indicated*) and 120Mt @ 1.50g/t Au (*Inferred*).

The tonnes and grade above a 0.0g/t Au cut-off within the interpretation wireframe for the 2009 Model were compared to the 2006 Model:

2006 Model 310Mt @ 1.5g/t Au; and
2009 Model 296Mt @ 1.6g/t Au.

3.4.2 WAI 2010 Resource Estimate

3.4.2.1 Introduction

The mineral resource estimate presented here for the Vasilkovskoye Gold Project is based on the model prepared by Kazakhstan Mineral Company (KMC) in October 2009, which has been depleted up to a pit survey dated 01 January 2011.

Database Compilation

Three distinct sampling techniques have been used at the Vasilkovskoye deposit; diamond drilling, underground channel sampling and blasthole sampling. A summary of these data can be found in Table 3.5.

Table 3.5: Summary of Assay Data by Type

Sampling Method	Year	Number of Collars	Total Metres	Total Assays	Number of Au Assays
Drillholes	1966-2009	633	236,117	122,838	120,128
Blastholes	1995 - 2007	64,990	194,970	65,628	65,628
Channel Samples	1966 - 1987	1,129	33,321	31,260	31,197

The drillholes have mainly been drilled on profiles orientated at an azimuth of 315°. This orientation is at approximately 90° to the NE trending Vasilkovskoye fault. Most of the holes are vertical to sub-vertical within the first 200m and thereafter deviate gradually towards 45°. The profiles are 60m apart and although in general the collars are spaced 30-60m along the section there are some collars less than 5m apart. Ten of the holes were drilled in 2008 as twinned holes to verify the older data. Forty seven of the holes were drilled in 2009.

Three shafts were sunk with horizontal drives on three levels at 60m intervals (60, 120 and 180m levels).

Cross-cuts were developed every 60m from horizontal workings along the survey profiles of core drilling. Cross-cut lengths were sufficient to cross the whole length of the mineralised area.

Short longitudinal workings were developed in the central part of the deposit in order to get more detailed information about structure and content.
These underground workings were sampled by channel sampling with the drive faces sampled every 2-3 blasts, the longer drives sampled on both walls and the shorter drives sampled on one wall only. Samples were also taken from raises connecting the three levels.

In addition, the upper levels of the pit were sampled by channel sampling (sample length was 2.0m and section was 10x3cm).

Blasthole details are provided above.

> ***WAI Comment****: WAI has checked and verified the individual assay files for duplicate samples, overlapping samples and missing collar and survey data. The database verifies correctly and is organised in a sensible and professional manner.*

3.4.2.2 Domaining

The gold mineralisation at the Vasilkovskoye deposit is associated with steeply dipping vein sets and associated alteration. The alteration halo is associated with a 0.4g/t Au cut-off and this grade was used to construct wireframes defining the mineralisation outlines.

As the mineralisation at Vasilkovskoye depends on the concentration of quartz veining, veinlets and sulphide mineralisation, the interpretation is based on the outer edge of the alteration halo rather than the interpretation of individual veins.

An indicator approach was used to define the edges of mineralisation at 0.4g/t Au. The gold assay values were converted to one or zero to indicate that the original grade was above or below the chosen cut-off and an indicator kriged block model was created using these data that estimated the probability of a block having an Au grade greater than 0.4g/t. Blocks that had a greater than 50% probability were subsequently used to guide the interpretation to define the outer limits of the mineralised zones. The outline strings were then modified to fit drillhole data more accurately and the final strings were used in the creation of domain wireframes. KMC created 27 domains based on the cut-off grade of 0.4g/t Au. These domains were separated in to 4 ore zones. Figure 3.3 below is an isometric view of the wireframes delineating these four ore zones.

In addition to the grade shell, wireframes of the two main lithological units, diorite and granite, were created by KMC. These were created from a combination of drillhole logging and underground mapping. These wireframes were used to control density assignation.

> ***WAI Comment:*** *The methodology behind the creation of the mineralised zone wireframes is appropriate to the situation at Vasilkovskoye. Attempting an interpretation of individual veins considering distances between boreholes is more than likely to give a wrong interpretation. During snapping of strings to boreholes, where there is ambiguity over interpretation, preference was given to the more recent assay information. Wireframes were verified within Datamine Studio 3 to check for crossovers and open edges and were found to verify correctly.*

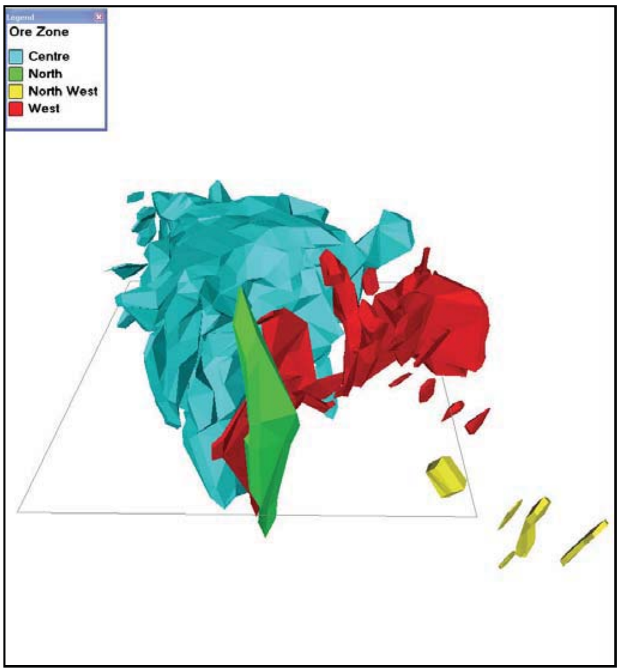


Figure 3.3: Isometric View Looking Towards South of Wireframes Delineating the Four Ore Zones

3.4.2.3 Geostatistical Analysis

Log probability plots of Au in the individual files were constructed before amalgamation of the files to form a complete database. The population of assay values in the blast holes was significantly different from the population of the diamond drillhole samples and channel samples. The blast hole assays were not included in the final assay database due to this indication of a separate statistical population. The final assay database was made up of all the diamond drilling assays and the channel samples.

The data shows a strong positive skew with a mean value of 1.10g/t Au and a few very high grade samples with a maximum of 395g/t Au. The statistical analysis of the global data by sample type as described above is listed in Table 3.6. Each of the sample campaigns shows a similar distribution with a strong positive skew indicating that the database as a whole covers a single population.

Log probability plots of Au assays were constructed for each of the main lithological types, diorite and granite, within the overall sample set. Each of the individual lithological types shows a similar type of distribution with a strong positive skew. There is slight deviation between lithological types at higher grades. The statistical analysis of the global data by lithological type is listed in Table 3.7.

> ***WAI Comment:*** *The final sample database is verified and robust and suitable for further processing and grade estimation, with no significant bias present due to sample type or lithology.*

Table 3.6: Standard Statistical Analysis By Borehole Type

Borehole Type	Pre-2008 Drillholes	2008 Drillholes	UG	2009 Drillholes
Field	**Au**	**Au**	**Au**	**Au**
No. of Samples	96,370	3,563	33,518	18,228
Minimum	0	0.045	0	0.045
Maximum	395	57.33	104.8	86.58
Range	395	57.29	104.8	86.54
Mean	1.01	0.69	1.43	1.08
Variance	9.94	5.92	9.99	8.31
StandDev	3.15	2.43	3.16	2.88
Skewness	29.03	10.22	6.86	7.86
Kurtosis	2,687.02	157.46	92.95	106.23
Coefficient of Variation	3.13	3.54	2.20	2.67

Table 3.7: Standard Statistical Analysis By Lithology

Lithology	Diorite	Granite
Field	**Au**	**Au**
No. of Samples	43,558	107,002
Minimum	0	0
Maximum	118.53	395.00
Range	118.53	395.00
Mean	0.95	1.15
Variance	8.29	10.18
StandDev	2.88	3.19
Skewness	9.77	25.82
Kurtosis	181.64	2,341.68
Coefficient of Variation	3.03	2.76

KMC applied no top-cuts to the raw data and believe that their search ellipses, estimation parameters and domains adequately account for their influence during Kriging. The mean sample length is 1.3m and the most common sample length is 1m. KMC chose to use a composite length of 1m for the resource estimation.

> ***WAI Comment:*** *Typically top-cuts are applied to outlier samples which have grades outside of the main population in order to avoid any undue influence during grade estimation. The log-probability plot for Au assays selected within the mineralised zones shows that in the main there is a single major population within the mineralised domains suggesting good stationarity. There is, however, a very minor high grade population which, left uncut, could lead to localised over estimation within the resource model. Any over estimation though will be minimal on a deposit wide scale. Although the most common sample length is 1m, approximately 34% of samples within the mineralised zones have a larger length than this. A 1m composite length would lead to a large amount of decompositing*

where samples are split in to two or more samples of equal grade. This has the effect of artificially reducing the local variability of data, in particular suggesting that the nugget value of the data is lower than reality. This may lead to over smoothing of grade within the resource model.

3.4.2.4 Variography

KMC calculated a single set of directional experimental variograms for the Vasilkovskoye deposit in Micromine using the composited samples. Using a lag distance of 10m, horizontal and then vertical experimental variograms were created to define the main axis of anisotropy. This direction corresponded roughly to the trend of the main mineralisation controlling fault. Perpendicular directions to this main axis were assigned for across strike and down dip directions. Single structure spherical models were assigned to the selected experimental directions. Table 3.8 lists the model parameters.

Table 3.8: Summary of Variogram Models by Domain				
Nugget	**Along Strike Range**	**Across Strike Range**	**Down Dip Range**	**Sill**
5.3	40.1	52.3	17.0	10.4

WAI Comment: *The directions chosen for experimental variogram creation are appropriate considering the interpreted trends of mineralisation. The variogram models assigned are appropriate to the experimental variograms.*

3.4.2.5 Block Modelling

Block models were created within the mineralised zone wireframes. KMC used a block model with parent cells of 20m x 20m x 15m in the X, Y and Z directions respectively. Ten subcells were allowed in each direction so that the smallest possible call size was 2m x 2m x 1.5m.

WAI Comment: *The block model parent cell sizes are appropriate to the sample spacing and the subcell splitting achieves a good fit against the mineralised zone wireframes.*

3.4.2.6 Density

Density calculations were based on 1199 wax-treated samples and 16 pillars testing results received in Soviet times, with 147 and 829 additional samples analysed respectively in 2007 and 2009.

Two sets of density data are available for the Vasilkovskoye deposit. Density results were organised by rock type and then amalgamated to form groups representing the two major lithological units; diorite and granite.

Based on these results KMC assigned an average bulk density of 2.83 to the diorite unit and an average density of 2.68 to the granodiorite into the block model. Blocks within the model were selected using the lithological wireframes and assigned the average grade for that unit.

3.4.2.7 Grade Estimation

Gold grades were estimated in to the block model using a zonal control based on domain. Gold grades were estimated using Ordinary Kriging (OK) as the main estimator and Inverse Power of Distance Squared for comparative purposes. Grades were estimated into parent cell volumes with any subcells all receiving the same grade as the parent. Gold grade in the subcells was the same as in the parent cells. Cell discretisation was set at 2 x 2 x 2.

A multiple pass approach was used with up to an initial possible three expansions of the search ellipse if minimum conditions were not met in the first or second ellipse. The first search ellipse is equal in size to 2/3 of the variogram ranges as outlined above. The second ellipse was equal in size to the range of the variograms and the third equal to twice the variogram range. With any blocks left unestimated after the third pass, further expansions of the search ellipse were carried out until all blocks within the mineralised zone were estimated.

The key estimation parameters used by KMC are listed in Table 3.9. The pass used in estimating blocks formed the basis used in resource classification.

3.4.2.8 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 interpolated model grades. An example Swath plot from this process is shown below in Figure 3.4.

Table 3.9: Estimation Parameters

Ellipse	**Parameter**	
1st	Search Radii 1 (m) – across strike	34
	Search Radii 2 (m) – along strike	26
	Search Radii 3 (m) – down dip	11
	Minimum Composites	12
	Maximum Composites	30
	Minimum Drill Holes/Trenches/Cross Cut	3
2nd	Search Radii 1 (m) – across strike	52
	Search Radii 2 (m) – along strike	40
	Search Radii 3 (m) – down dip	17
	Minimum Composites	6
	Maximum Composites	30
	Minimum Drill Holes/Trenches/Cross Cut	2
3rd	Search Radii 1 (m) – across strike	104
	Search Radii 2 (m) – along strike	80
	Search Radii 3 (m) – down dip	34
	Minimum Composites	3
	Maximum Composites	30
	Minimum Drill Holes/Trenches/Cross Cut	1

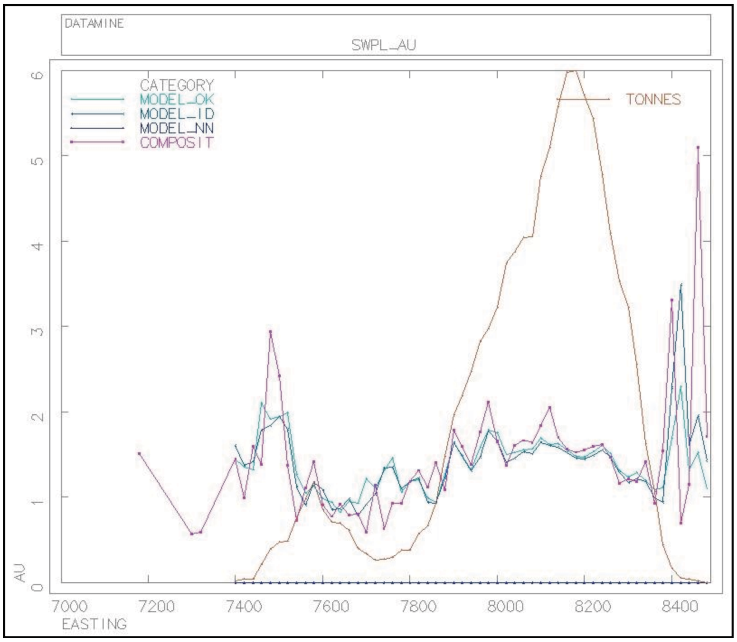


Figure 3.4: Grade Profile of KMC Model and Composites by Easting

3.4.2.9 Depletion

WAI depleted the KMC model up to the pit survey supplied by Kazzinc dated 01 January 2011 by removing blocks entirely above the pit surface DTM and splitting blocks intersected by the DTM to give an exact fit.

3.4.2.10 Resource Classification

The basis of the definition of the resource categories is the pass used in estimating blocks. Essentially this gives approximate drillhole spacings for the allocation of resources which can be summarised as 25m x 10m for *Measured* resources and 40m x 20m for *Indicated* resources. All other blocks within the mineralised domains were classified as *Inferred*.

3.4.2.11 Resource Evaluation

Grade-tonnage information for the Vasilkovskoye Au deposit (for all domains) is summarised in Table 3.10 for total in-situ resources estimated in accordance with the guidelines of the JORC Code (2004) at a range of cut-off grades. The grades and classifications are reported as estimated by the OK method in the KMC model, which has been depleted up to a pit survey dated 01 January 2011 by WAI.

Table 3.10: Vasilkovskoye Mineral Resource Estimate (WAI 01.01.2011) (In Accordance with the guidelines of the JORC Code (2004)

Cut Off Grade (g/t)			0.4	0.9	1.5	2.0
Measured	Tonnage (Mt)		45.23	30.77	19.52	13.71
	Au (g/t)		1.75	2.28	2.92	3.41
	Metal	kg	79,331	70,089	56,924	46,780
		oz	2,550,552	2,253,406	1,830,142	1,504,002
Indicated	Tonnage (Mt)		141.56	97.72	58.96	39.95
	Au (g/t)		1.72	2.21	2.89	3.44
	Metal	kg	243,862	215,792	170,309	137,371
		oz	7,840,344	6,937,853	5,475,553	4,416,563
Measured + Indicated	Tonnage (Mt)		186.80	128.49	78.48	53.66
	Au (g/t)		1.73	2.22	2.90	3.43
	Metal	kg	323,193	285,880	227,233	184,150
		oz	10,390,893	9,191,256	7,305,694	5,920,566
Inferred	Tonnage (Mt)		99.08	68.63	39.74	26.21
	Au (g/t)		1.77	2.27	3.07	3.77
	Metal	kg	175,747	156,032	122,157	98,883
		oz	5,650,405	5,016,526	3,927,426	3,179,143

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) Mined areas removed from evaluation

WAI Comment: *WAI has also prepared a geostatistical resource model for the Vasilkovskoye deposit which presents a more global picture. Although the data are not reported here, overall, tonnages are higher and grades lower.*

However, since production began, it appears that the KMC model better represents the grades of mineralisation being recovered from the pit. As such, a decision was taken in 2010 to utilise the KMC model for short term planning, whilst the WAI model may be more suitable for long term planning purposes.

In any event, the large drilling campaign currently taking place at the mine will provide the opportunity for an updated resource model later in 2011.

3.5 Mining

3.5.1 Introduction

The Vasilkovskoye open pit mine was originally developed during the early 1980's to exploit an oxide gold resource. Mining of the oxide material continued from 1980 until 2005, when the resource was finally depleted. In total 14Mt of oxide ore was mined and processed by heap leaching. A significant primary sulphide resource is present below the oxide layer and the decision to re-develop the mine and build a sulphides processing facility was made during 2006.

The new plant was designed to process up to 8Mtpa of sulphide ore and a mining study was completed by AMC Consultants in order to design a mining operation to feed the plant at this rate. AMC re-estimated the resources in accordance with the JORC Code (2004) guidelines and undertook geotechnical, hydro-geological, pit optimisation and preliminary mine design studies. The initial AMC pit design was to a depth of 360m but following re-optimisation at higher gold prices, the final pit design by Tomsk Institute was for a 440m deep pit.

Waste stripping, ore stockpiling and construction of the in-pit crushing facilities commenced during 2007. Production itself officially began with the completion of the processing facilities in November 2009. Due to problems with the processing plant, however, the mine is currently running at less than half of the design capacity, producing at present in the region of 270kt of ore per month rather than 600kt per month.

3.5.2 Mining Operations

The Vasilkovskoye mine is a conventional open pit mine. The ore and waste rock is drilled and blasted prior to excavation by diesel-hydraulic excavators. The ore is loaded into diesel powered off-highway rigid dump trucks and hauled to either one of three in-pit stockpiles, depending on grade, or dumped directly into an in-pit crusher. The waste rock is hauled to one of two waste dumps depending on which side of the pit is being excavated. The mining fleet is owned and operated by VasGold. **Error! Reference source not found.** shows a general view of the open pit mining operations at Vasilkovskoye.

Photo 3.1: General View of Mining Operations at Vasilkovskoye

3.5.3 Drilling and Blasting

Blast hole drilling operations at Vasilkovskoye are conducted using 7 blast hole drilling rigs of which 4 units are generally operating at any one time. Three types of rig are in operation, 2 x Atlas Copco Roc L8 DTH drills, 3 x Atlas Copco Pit Viper PV 275 rotary drills and 2 x Atlas Copco DM45 drill rigs capable of both rotary and DTH drilling.

The Pit Viper rigs are fitted with 251mm drill bits and are used to drill both ore and waste but generally in softer areas of the pit. The DM45's are fitted with 171mm drill bits and are used to drill both ore and waste in areas of harder ground. The Roc L8's are fitted with 165mm drill bits and are used for drilling ore and trim blasting of the final faces. The drill patterns vary depending on whether it is ore or waste, the hardness of the rock and the type of drill rig used. The patterns range from 8.5m x 8.5m for a Pit Viper drilling waste rock down to a 5.0m x 5.0m pattern when using Roc L8's in ore. The drilling depth in both ore and waste is 8.5m for a 7.5m bench height.

All blast holes are loaded with Rioflex bulk emulsion explosives which are supplied by the local division of Maxam, an international explosives supplier and blasting contractor. Maxam has an explosive mixing and storage facility located adjacent to the Vasilkovskoye pit and supplies all explosives and blasting accessories for each blast. An average blast will contain between 120 and 150 holes and liberate between 30k and 50k bank cubic metres (BCMs) of material.

The Rioflex emulsion is loaded into the blast hole directly using a bulk explosives delivery truck with a capacity of 11,000kg of bulk emulsion explosives. Each hole contains between 25kg and 55kg of Rioflex depending on the diameter of the blast hole being loaded (170mm to 250mm). The Rioflex column charges are detonated using 450g Pentolite boosters which are placed in the bottom of each blast hole and initiated using non-electric (Nonel) detonators. All surface delays and connectors are also Nonel. The blast design work is undertaken by VasGold's Mine Planning department.

3.5.4 Loading and Hauling

The main loading and haulage fleet at Vasilkovskoye consists of 6 hydraulic excavators, 5 x Komatsu PC1800 backhoes and 1 x Terex RH120 face shovel, 2 x Komatsu WA800 wheel loaders and 20 x Caterpillar 777F rigid dump trucks. Additional support equipment includes 3 x Caterpillar D10 bulldozers, 3 x Caterpillar graders, 2 x water trucks for dust suppression and 1 x fuel tanker for re-fuelling the excavators and drill rigs.

Excavators are used for loading waste rock whereas ore is usually loaded using the wheel loaders.

The mining fleet is currently operating well below its design capacity due to the problems with the processing plant. During the WAI visit only 3 excavators, 1 wheel loader and 13 trucks were operating.

Currently, up to 4 blocks are in operation at any one time. Blocks may be waste only blocks or may be mixed ore and waste blocks. In the mixed ore and waste blocks loading of the material is selective with marker tapes and a banksman used to assist the loader operator. On average 8,000BCMs of ore and 23,000BCMs of waste are excavated each day. It is anticipated that the number of blocks in operation and the amount of equipment operating will increase when the plant throughput issues are resolved. The operations are continuous with 2 x 12 hour shifts operating per day. Effective operating hours are 20 hours per day due to meal breaks, shift changes and re-fuelling/servicing of equipment.

Due to the fact that the majority of the mining equipment is diesel-hydraulic, if the temperature drops below -38°C then operations cease to avoid damage to the hydraulic systems due to freezing of oil. Last year, 4 days were lost due to low temperatures and snow storms.

3.5.5 Ore Crushing and Stockpiling

The ore is hauled from the working faces to an in-pit crushing facility located on the south side of the pit. Haul trucks dump the ore into 1 of 2 x 150t rock hoppers which feed 2 x Sandvik CJ615 single toggle jaw crushers at a rate of up to 750tph each. The crushers reduce the ore to -350mm and discharge onto a 1,200m long inclined conveyor belt which transports the ore to an 18,000t live coarse ore stockpile that in-turn feeds the secondary crushing plant.

Located adjacent to the in-pit crushing facility are three run-of-mine (ROM) stockpiles for low grade ore (0.4g/t to 0.9g/t), medium grade ore (0.9g/t to 2.0g/t) and high grade ore (>2.0g/t). Ore is tipped on the stockpiles when the live stockpile is full, when the crushing plant is not operating due to maintenance or the ore is required to be blended.

The secondary crushing plant consists of 2 x cone crushers operating in closed circuit, which reduce the ore to -30mm in size.

3.5.5.1 Waste Rock

Waste rock from the mining operations is transported to one of two rock dumps located on the east and west sides of the pit. Waste is transported to the closest dump to the face being excavated. Sufficient waste dump capacity is available for the whole mine life but waste haulage distances increase as the mine deepens and the waste dumps extend further from the pit rim. The current average haul distance is 1.5km from the face to the waste dump. VasGold anticipates that the waste haulage fleet will need to increase in future to accommodate longer haulage distances. Five additional trucks will be made available in 2011 when the Shaimerden open pit ceases operations.

3.5.5.2 Dispatch and Control

VasGold is in the process of upgrading all the vehicles in the mining fleet with GPS transmitters with the aim of implementing a computerised dispatch and control system. Currently, the shift supervisor and dispatcher are located in an office overlooking the pit and control and record all truck movements manually by radio. This system works well but does not always provide optimal truck movements/waiting times, requires all data to be manually recorded and suffers in poor weather due to reduced visibility. The GPS dispatch system will automatically record all movements of materials and direct each truck to the nearest available excavator, ensuring reduced haul distances and lower waiting times. The GPS dispatch system is due to be operational during 2011.

3.5.5.3 Maintenance Facilities

Vasilkovskoye is equipped with very good maintenance facilities for both the mobile and static plant equipment. The main maintenance facility for the mining equipment consists of a "Pit Stop" workshop for minor maintenance, servicing, re-fuelling, tyre changing and lubrication of the truck and mobile equipment fleet. Each truck visits the Pit Stop once per shift for re-fuelling and lubrication. A mobile tanker truck which is based at this facility also visits the tracked vehicles (drill rigs, excavators and bulldozers) to re-fuel and lubricate them without the need for them to be tracked too far.

In addition to the Pit Stop, there is a large workshop where both planned and unplanned maintenance is carried out. This workshop is equipped with full servicing, fabrication and machining facilities and is capable of undertaking most servicing and repair tasks. The main items within the mining fleet are maintained under a service contract with their respective Original Equipment Manufacturers (OEMs). Each OEM is allocated a space within this workshop and conducts all necessary maintenance to its fleet of vehicles. Currently, Komatsu, Caterpillar and Atlas Copco are contracted to maintain the fleet of excavators, trucks and drill rigs.

3.5.5.4 Power and Pumping

The Vasilkovskoye pit is reported to be relatively dry and requires very little pumping. In winter the ground water is frozen and has no effect on the operations. A borehole fitted with a submersible pump is located in the middle of the pit and does operate during the summer months. WAI did not note any significant ground water during its site visit in October when air temperatures were still above freezing.

The is no major requirement for electrical power in the pit due to the fact that all the mining equipment is diesel powered; however, a system of low voltage overhead lines is present to supply power to the in-pit submersible water pump and flood lights for night time operations. Portable diesel lighting sets are also used to supplement the mains powered units.

3.5.6 Mine Planning

Long term mine planning at Vasilkovskoye is conducted by the Mine Planning Department and short term planning is conducted by the Mine Operations Department.

The Mine Planning Department is primarily responsible for the production of the annual plan, a 3 month plan and 1 month plan. The primary tool used by the Mine Planning Department is Surpac for the production of survey plans and the Gemcom Gems suite of software for blast design. The Mine Operations Department has an input to the monthly plan and then produces their own weekly and daily plans to ensure that the operations are scheduled and managed effectively. There is close communication between the Mine Operations and the Mine Planning Departments. Other key inputs to the management of the mining operations are provided by the Geology Department (Resources and Grade Control), the Survey Department and the Maintenance Department.

During the WAI site visit the mining operations appeared to be well managed and the operations were conducted to a very high standard. Inter-department communications and data sharing tools were well implemented with the effective sharing of basic geological data, survey data and production data to all interested parties.

3.5.7 Ore Reserves

3.5.7.1 Optimisation

A summary of the pit optimisation parameters used are shown in Table 3.11.

Table 3.11: Optimisation Parameters

Description	Unit	Value
Gold Price	US$/oz	1,000
	US$/g	32.15
Selling Cost	US$/g	0.30
Ore Production Rate	Mtpa	8.0
Discount Factor		10.0%
Mining Cost	US$/t	1.45
Dilution Factor		17.0%
Mining Recovery Factor		95.0%
Processing Cost	US$/t ore	10.74
Processing Recovery		82%
Overall Slope Angle	Degrees	41
Derived Cut-Off Grade	g/t	0.48

Optimisation runs were completed on both the KMC and WAI resource block models, and for different cut-off strategies. Free cut-off runs were made with the cut-off effectively calculated directly from the supplied parameters (which gives 0.48g/t). Fixed cut-off runs were also made at a fixed cut-off of 0.9g/t Au. All of the optimisation results, for maximum cashflow pits, are summarised in Table 3.12. The NPV's and other financial indices shown here are used to select an option pit shell and do not reflect the project valuation. For reference, the same results for final design pit are also shown in the same table. Graphs depicting the results for the KMC free optimisation runs are shown in Figure 3.5 and Figure 3.6, with the selected pit (shell 87), which includes the pit's western extension.

Table 3.12: Pit Optimisation Summary

	Run + Model	Cut-off g/t	Price US$/oz	Profit US$M	Revenue US$M	Processing Cost US$M	Mining Cost US$M	Total Rock Mt	In-Situ Mt	Plant Feed Mt	Total Waste Mt	Strip Ratio	Mes Mt	Ind Mt	Au g/t	Au Recovered t
Optimisations – Max Cashflow	KMC Free cut-off	0.48	900	3,898	6,430	1,612	920	635	135	151	500	3.70	41	94	1.92	202
	KMC Free cut-off	0.48	1,000	3,909	6,618	1,670	1,038	716	140	156	576	4.12	41	99	1.91	208
	KMC Free cut-off*	0.48	1,060	3,891	6,980	1,782	1,307	901	149	167	752	5.04	43	106	1.89	219
	KMC Fixed cut-off	0.9	900	3,733	5,872	1,219	920	634	102	114	532	5.21	30	72	2.32	184
	KMC Fixed cut-off	0.9	1,000	3,740	5,991	1,247	1,004	692	104	117	588	5.63	30	74	2.31	188
	WAI Free*	0.48	900	4,323	8,169	2,594	1,253	864	217	243	647	2.98	16	201	1.52	256
	WAI Free	0.48	1,000	4,352	8,845	2,885	1,608	1,109	242	270	868	3.59	18	224	1.48	278
	WAI Fixed	0.9	900	3,927	6,485	1,562	996	687	131	146	556	4.25	9	122	2.00	204
	WAI Fixed	0.9	1,000	3,963	7,060	1,710	1,387	956	143	160	813	5.68	10	133	1.99	222
	Design + KMC Model	0.9	1,000	3,897	6,575	1,373	1,306	901	115	128	773	6.05	36	92	1.97	206

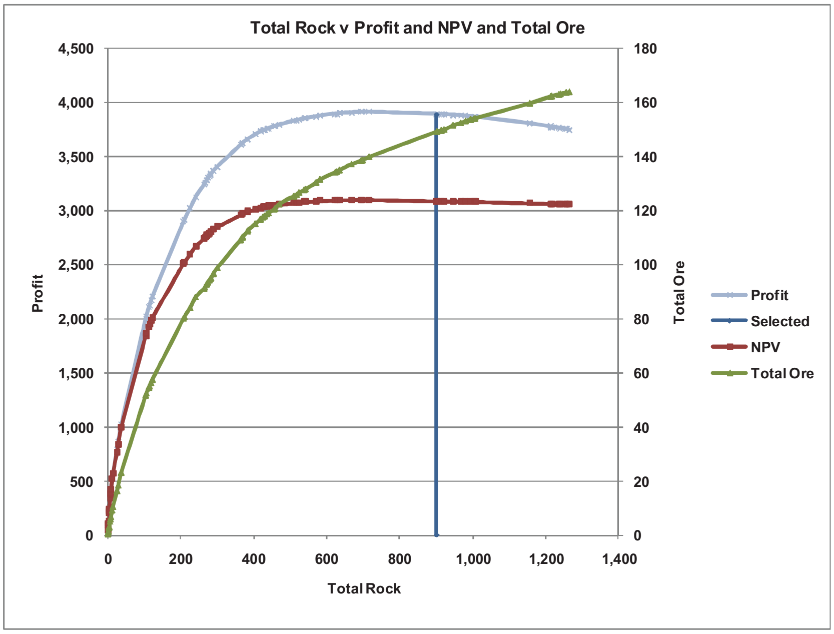


Figure 3.5: Optimisation Results – KMC Model - Rock v Profit/Ore

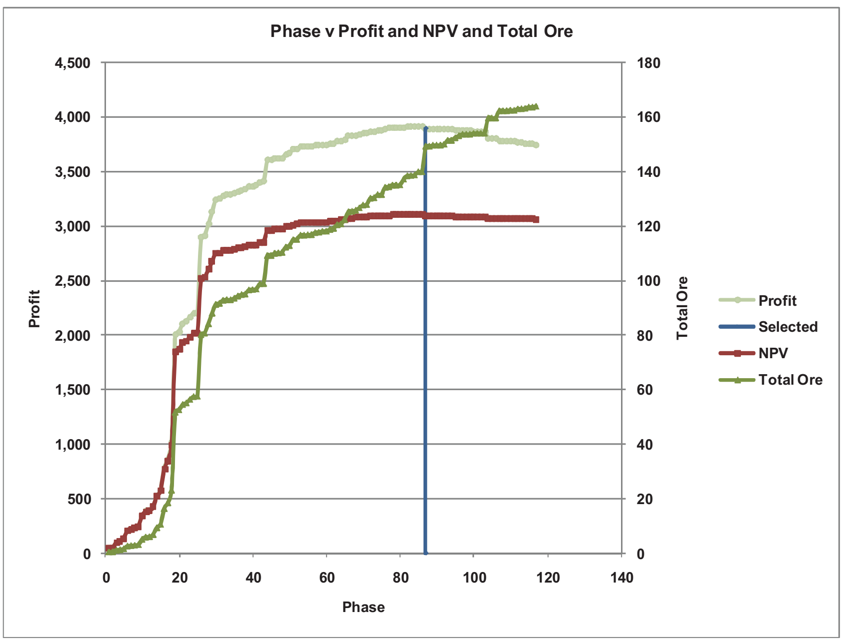


Figure 3.6: Optimisation Results – KMC Model - Phase v Profit/Ore

3.5.7.2 Pit Design

Based on the pit optimisation results, specifically those resulting from the optimisation of the KMC model (Phase 87, for a price of US$1060/oz), as well as the optimisation of the WAI model (US$900/oz), a final pit design was developed. This used the design parameters summarised in Table 3.13.

Table 3.13: Pit Design Parameters		
Description	**Unit**	**Value**
Face Angles		
Above 145mRl	Degrees	60
Below 145mRl	Degrees	65
Berm Width	m	10
Bench Height between Berms	m	30
Ramp Width	m	32
Ramp Gradient		10%
Minimum Working Width between Cutback #4 and Final Design	m	50

The final pit design is shown in a plan in Figure 3.7, with reference sections in Figure 3.8 and Figure 3.9. The ramp was designed so as to exit in the south, and at the south end a horizontal pad was also incorporated, for the crusher and ore stockpile area, similar to that in cutback #4.

A summary of this overall pit evaluation and the estimated reserves at Vasilkovskoye calculated in accordance with the guidelines of the JORC Code (2004), against the KMC model, are shown in Table 3.14.

Table 3.14: Vasilkovskoye Pit Evaluation Summary (WAI 01 01.2011) (In Accordance with the guidelines of the JORC Code (2004)						
Class	**Reserves**			**Other Pit Contents**		
	Tonnes Mt	**Au g/t**	**Au Contained Moz**	**Rock Mt**	**Waste Mt**	**Strip Ratio**
Proven	33.3	1.95	2.088	-	-	-
Probable	90.7	1.94	5.6576	-	-	-
Total	***123.97***	***1.94***	***7.732***	***870.65***	***746.68***	***6.02***

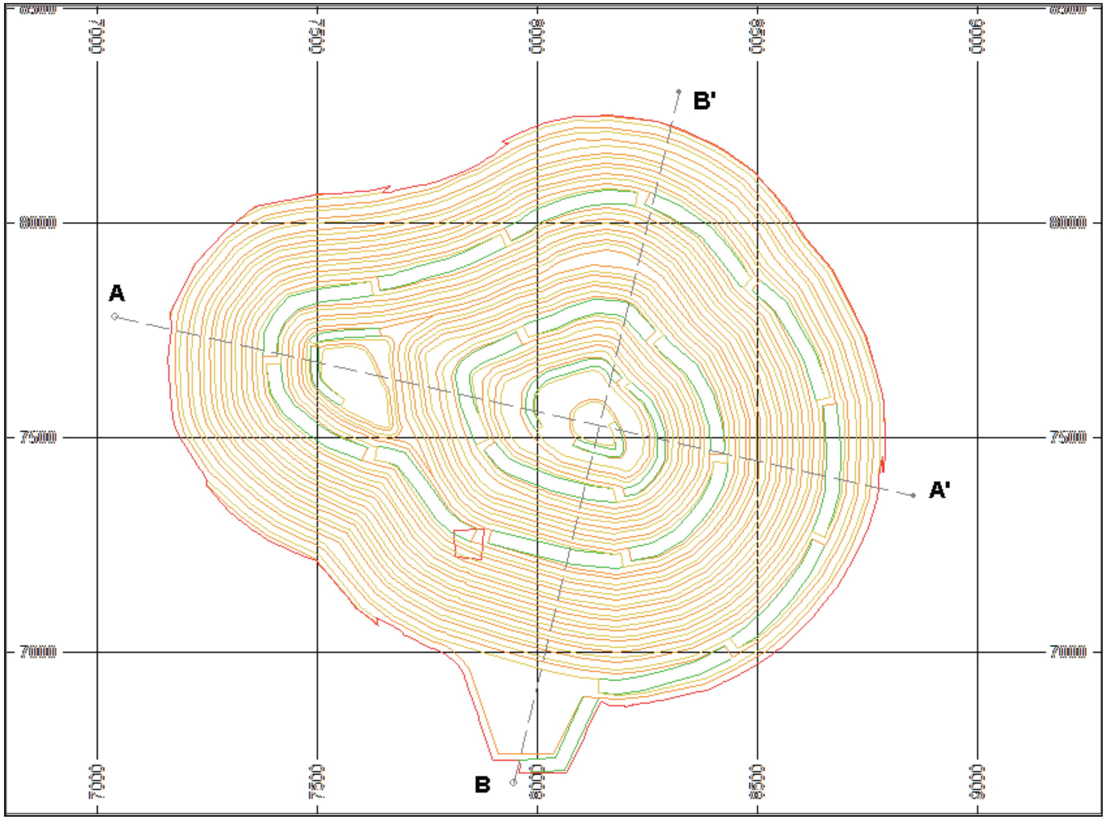


Figure 3.7: Plan of Final Pit Design

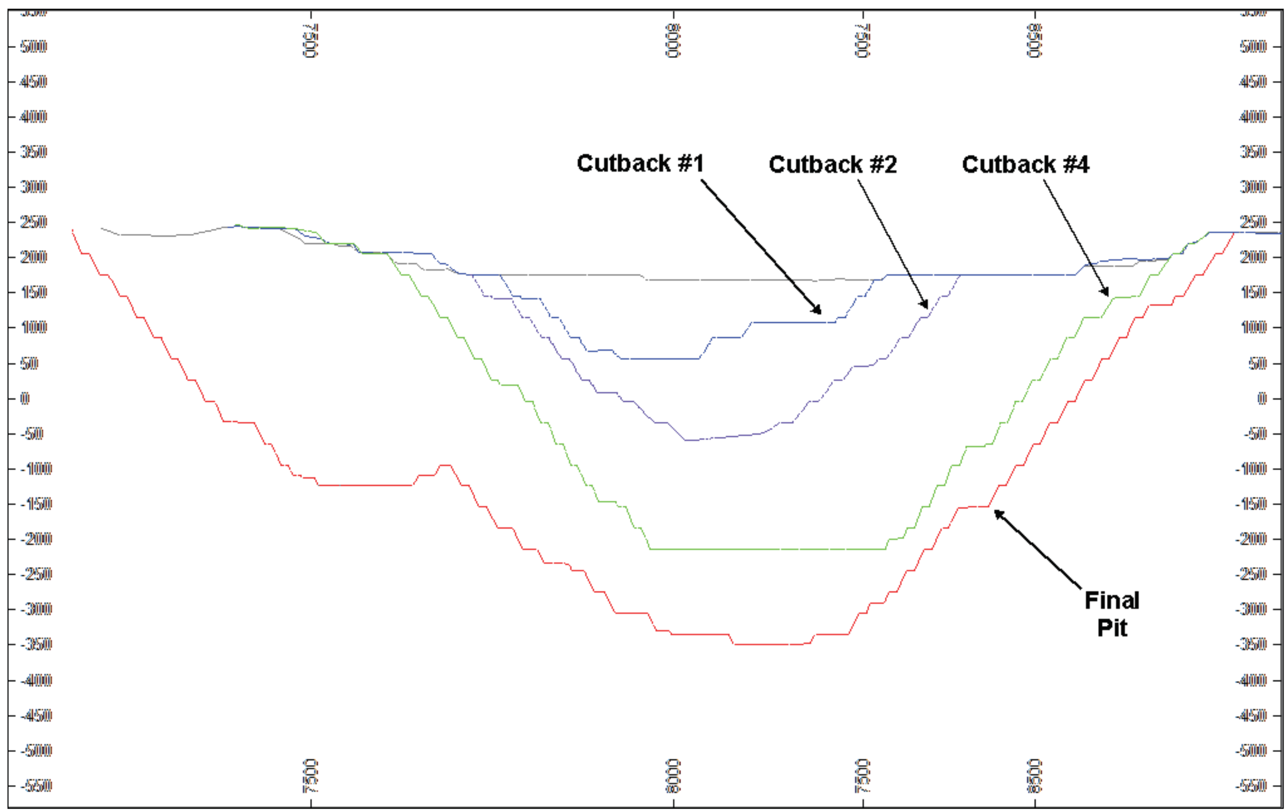


Figure 3.8: Section A-A' Of Final Pit Design

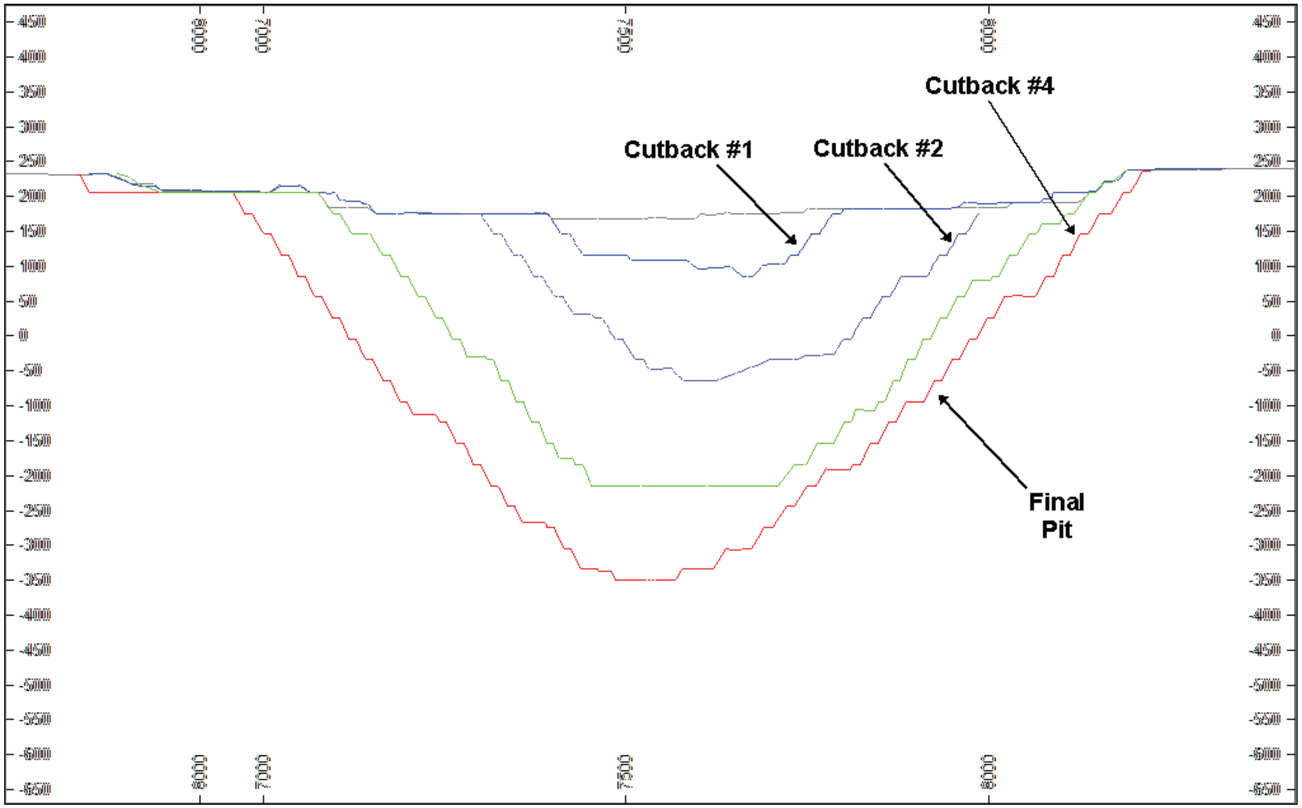


Figure 3.9: Section B-B' Of Final Pit Design

3.5.7.3 Scheduling

A mining schedule was developed for the final pit, incorporating the supplied earlier cutbacks (#1, #2 and #4) and based on an 8Mtpa mill production rate. Material from the final cutback is not mined until year 5, with the bulk of production from this final cutback coming after year 6.

A summary of this schedule is shown in Table 3.15, with a diagrammatic cross-section in Figure 3.10. This schedule has been developed with a relatively low stripping ratio for the first 5 years.

3.5.8 Grade Control

Within the open pit, grade control is done using blastholes, drilled on a 6x6m grid to a depth of 7.5m for the 7m bench. Several blasthole rigs are used including Atlas Copco ROC, drilling 151 and 171mm holes, and the Viper which drills 250mm holes.

Table 3.15: Schedule Summary

	Year	Unit	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	Total
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	
	Profit	US$M	236	297	285	292	307	230	185	238	220	260	231	246	176	178	181	236	**3,798**
	Revenue	US$M	372	435	424	428	440	411	368	423	404	461	428	431	373	369	348	360	**6,475**
	Processing Cost	US$M	86	86	86	86	86	86	86	86	86	86	86	86	86	86	86	82	**1,371**
	Mining Cost	US$M	51	52	53	50	47	95	97	9	99	115	111	100	110	105	91	42	**1,307**
	NPV	US$M	213	243	211	196	186	126	92	107	89	96	77	74	48	44	40	48	**1,891**
	Total Rock	Mt	35	36	36	35	32	66	67	68	68	79	77	69	79	72	56	29	**901**
	Total Waste	Mt	27	28	28	27	24	58	59	60	0	71	69	61	68	64	48	21	**773**
	Strip Ratio		3.4	3.5	3.5	3.4	3.0	7.2	7.4	7.5	7.5	8.9	8.6	7.6	8.5	8.1	6.0	2.8	**6.1**
	Ore	Mt	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	7.7	**127.7**
	Au	g/t	1.78	2.08	2.03	2.05	2.11	1.97	1.76	2.03	1.94	2.21	2.05	2.06	1.78	1.76	1.67	1.79	**1.94**
	Au R Product	t	11.46	13.65	13.30	13.44	13.81	12.91	11.56	13.29	12.69	14.49	13.43	13.54	11.70	11.58	10.93	11.30	**203**
Cutback #1	Rock	Mt	16	9	5	2	-	-	-	-	-	-	-	-	-	-	-	-	**31.79**
	Ore	Mt	6.02	5.38	3.29	1.65	-	-	-	-	-	-	-	-	-	-	-	-	**16.34**
	Waste	Mt	10	4	2	0	-	-	-	-	-	-	-	-	-	-	-	-	**15.45**
Cutback #2	Rock	Mt	19	13	14	15	13	8	3	4	-	1	-	-	-	-	-	-	**90.21**
	Ore	Mt	1.98	2.62	4.47	6.15	7.75	5.19	1.90	2.89	-	0.87	-	-	-	-	-	-	**33.83**
	Waste	Mt	17	10	10	9	5	3	1	1	-	0	-	-	-	-	-	-	**56.39**
Cutback #4	Rock	Mt		14	17	18	17	7	52	25	26	17	17	14	13	6	-	-	**293.81**
	Ore	Mt	-	-	0.24	0.19	0.25	2.81	6.10	5.11	7.99	7.09	7.53	7.11	5.98	3.76	-	-	**54.15**
	Waste	Mt	-	14	17	17	16	55	46	20	18	10	10	7	7	2	-	-	**239.66**
To Final Wall	Rock	Mt	-	-	-	-	3	-	12	39	42	61	60	54	64	66	56	29	**485.25**
	Ore	Mt	-	-	-	-	-	-	-	-	0.01	0.04	0.47	0.89	2.02	4.24	8.00	7.68	**23.36**
	Waste	Mt	-	-	-	-	3	-	12	39	42	61	59	54	62	62	48	21	**461.90**

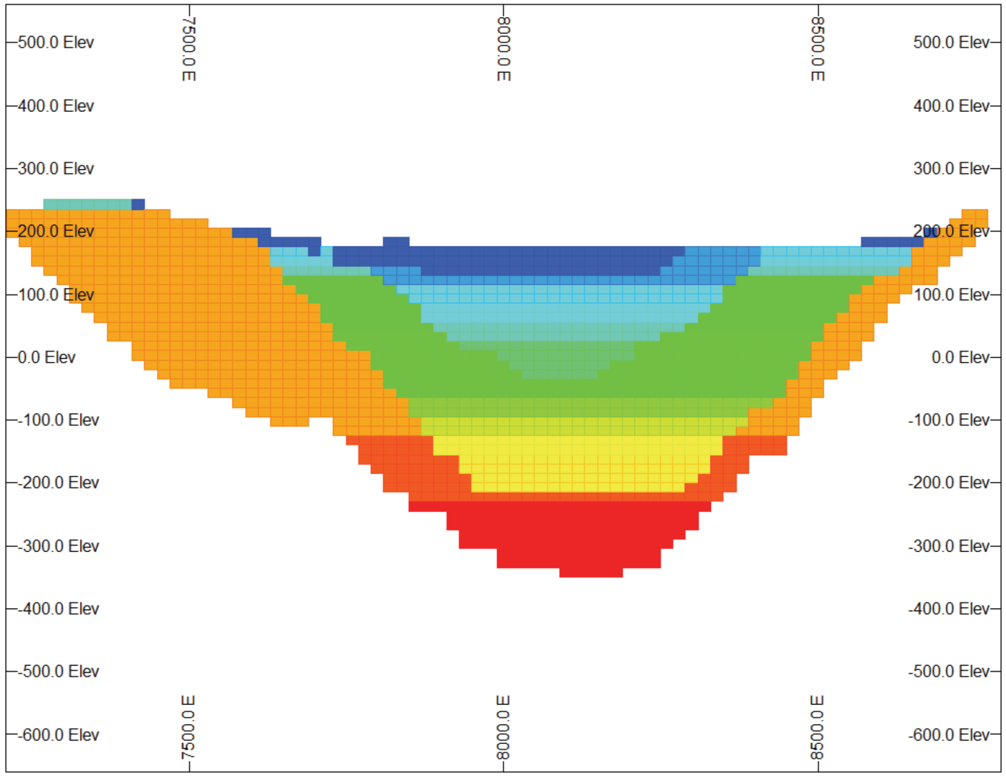


Figure 3.10: W-E Section, Depicting Mining Schedule

The Viper takes one sample per blasthole, the ROC's three samples (2.5x3m). Holes are laid out by GPS and tagged using nails hammered into the ground.

During the early development of the mine, extensive tests were undertaken to determine the optimum grade control methodology. This involved changing the drillhole spacing and importantly, using angled RC holes.

Bearing in mind that the mineralisation is broadly a sub-vertical stockwork, it is logical that angled holes will be better at delineating the steeply dipping mineralised features as opposed to vertical holes. This was in fact the case, with the angled holes providing a better reconciliation between predicted and actual grades.

However, the mine currently does not operate an RC rig, although angled exploration holes are planned for early 2011 to improve grade estimation.

> ***WAI Comment:*** *The introduction of angled RC holes for grade control should greatly improve the ability of the mine to predict bench grades thus allowing more efficient management of stockpile/direct feed/waste issues.*

3.5.9 Reconciliation

In an effort to establish how accurate the existing KMC resource model is in relation to mined production, the Company has undertaken some preliminary reconciliation studies. The results of this work for 2010 are shown in Table 3.16 below.

Table 3.16: Reconciliation Data of Block Model to Mining For 2010

	Ore Tonnes	Grade g/t	Au kg
Balanced Ores			
KMC Block Model	3,278,406	2.43	7,976.4
Production	3,271,189	2.29	7,496.2
Difference	-7,217	-0.14	-480.1
Off-balanced Ores			
KMC Block Model	1,177,973	0.62	735.1
Production	2,332,834	0.59	1,371.8
Difference	1,154,861	-0.04	636.8

Reconciling model grade to the production grade is always difficult due to the uncertainty in sampling the mined material. At Vasilkovskoye, the production grade was estimated from sampling the belt at the plant and assuming that this material has increased by a 12% dilution, thus allowing grade to be back-calculated.

Broadly, the results from the Balanced Ore (run-of-mine material) were good with only a small decrease in grade and produced 5% less gold in comparison to the model which appears to be well within the tolerance for normal losses within the pit. For the Off-balance Ore (low-grade material), there was a significant production tonnage increase, although at the approximate predicted grade.

The reasons for this are that the mine has consistently mined low-grade material that is not in the model, either on the periphery of the ore zone or as additional lenses. With high gold prices, off-balance material may well be economic to mine – a view shared by the Company.

3.5.10 Stockpile

At present, there are three stockpiles used by the mine:

- High clay content;
- High grade, and

- Run-of-mine.

The high clay content ore is not currently treated and therefore this stockpile does not play a part in everyday mining activities. The other two remaining stockpiles are formed by trucks dumping ore. Each truck is grab sampled (several small pieces) to give approximately 6kg of material. The stockpile is then levelled and a simple average grade applied to the horizon.

When the company plans to utilise a part of the stockpile, local averages are calculated both in plan and section, based on the original grab sample data to provide a grade for that particular slice of the stockpile.

During 2010, some (864kt @ 2.57g/t Au) 23% of total ore to the concentrator came from the stockpiles. This was in contrast (2,779kt @ 1.78g/t Au) to the balanced ore that went directly from the pit to the concentrator as well as (113kt @3.83g/t Au) of high grade ore.

> ***WAI Comment:*** *The use of 6kg grab samples to provide grade information for each truck is flawed as it is extremely unlikely that any degree of "representability" can be achieved. It may be possible with the introduction of the angled RC holes for grade control that the amount of ore passing through the stockpiles might be reduced, with an associated decrease in numbers of assays required.*

3.6 Process

3.6.1 Introduction

Processing of the Vasilkovskoye ores was initially undertaken in the 1990's using heap leach technology to treat the oxide ores. In 2006 the oxide ores were exhausted and a decision was made to begin the treatment of the sulphide ores by using flotation and cyanide leaching.

The design and plant construction was undertaken by Toms (Irkutsk) in conjunction with Kazzinc staff. The plant design was approved by the Kazak regulatory authorities and the construction of an 8Mtpa processing facility was begun in late 2007. Plant commissioning began in November 2009. In 2010, the equipment pre-commissioning was undertaken and in October 556kt of ore was processed which represents 83% of design plant capacity.

3.6.2 Process Testwork of Sulphide Ores

The results of rational analysis undertaken on two samples of Vasilkovskoye primary ores are given in Table 3.17.

Table 3.17: Rational Analysis Results

Phase	Sample 1	Sample 2
Free gold	11.65	7.16
Cyanide leachable gold	63.82	74.95
Total cyanide recoverable	**75.47**	**82.11**
Gold in Sulphides	20.28	15.81
Gold in Silicates	4.25	2.08
Total	100.0	100.0

The tests were undertaken at a grind size of 95% passing 71µm. The samples contained relatively low levels of free gold (11.6% for Sample 1 and 7.16% for Sample 2). The cyanide recoverable gold contents were 75.47% and 82.11%. A significant proportion of the gold was associated with sulphide minerals.

3.6.3 Testwork Undertaken at AMMTEC for Bateman Minerals

A sample assaying 3.12g/t Au, 0.3g/t Ag, 2.3% As and 3.2% Fe was subjected to a programme of metallurgical testwork at AMMTEC (Australia).

The ore minerals were found to be sulphides with significant quantities of arsenopyrite and lesser amounts of pyrite. Mineralogical analysis showed the presence of fine-grained gold particles (less than 10 microns) predominantly occurring in arsenopyrite.

Direct cyanidation of the ore showed that gold recovery increased with decrease in particle size, with a maximum recovery of 75.4% obtained at a grind size of 80% passing 45µm. Gravity concentration tests gave low upgrading with 40.2% of the gold being recovered at a 2.51% mass recovery and a concentrate grade of 57.2g/t.

Flotation tests on the gravity tailings gave high gold recoveries of 95-97%. The recovery of gold from the head sample by flotation was lower, at 84.6%. The overall recovery by gravity and flotation was 90.8%.

Direct cyanidation of the flotation concentrate after ultrafine grinding to 100% passing 25µm, gave a gold recovery of 77.3%. Preliminary alkalization of the concentrate with NaOH for 8 hours increased the gold recovery up to 83.8% from the cyanidation operation.

A summary of the test results undertaken by AMMTEC is summarised in Table 3.18.

Table 3.18: AMMTEC Test Results

Technology	Overall gold Recovery %	Comments
Direct ore leaching	75.4	Passing size 80% -0,045 mm
Gravity concentration – cyanidation of gravity concentrate and tailings	80.3	Passing size 95%-0,074 mm,
Gravity concentration with flotation and cyanidation of concentrates	79.2	Grind size 95% -0,074 mm, Regrind - 25µm (concentrates).
Gravity concentration + cyanidation of concentrates – cyanidation of flotation tailings	85.9	Passing size 95%-0,074 mm Less than 25 µm (concentrates)

Further testing at Kazzinc's ZMCC laboratory broadly confirmed these results.

The main conclusions of the testwork programmes were as follows:

1. Gravity concentration and flotation will give a recovery of gold into concentrates of 85-90.8%;
2. Recovery of gold from the concentrates is complicated due to their refractory nature. This was attributed to the fine disseminations of gold (less than 10 microns) and the presence of cyanicides which decrease the efficiency of the cyanidation process. The leach recovery of gold from the concentrates without regrinding was 66-69%;
3. The application of ultrafine grinding down to 25 microns resulted in the additional exposure of fine-grained gold with leach recoveries increasing to 87.0-88.4%) but also lead to increased consumptions of cyanide (of up to 39kg/t of concentrate); and
4. An alkalization pre-treatment with fine grinding increased gold recovery and resulted in lower cyanide consumptions.

3.6.4 Process Flowsheet

3.6.4.1 Crushing

Ore is subjected to primary crushing in the open pit and the -550mm fraction is conveyed to a coarse ore stockpile by overland conveyors. The primary crushing and ore transport are the responsibility of the mine department.

The coarse ore stockpile has a live capacity of 18,000t. Ore is recovered using 13 vibrating feeders and is conveyed via a surge hopper to an inclined double deck vibrating screen fitted with 50 and 30mm decks. The screen oversize passes to a Sandvik H8800C cone crusher (open size setting of 40mm) and the crushed product is further screened at 30mm with the oversize passing to a tertiary Sandvik H8800EF cone crusher. The tertiary crushed product and the screen undersize products are combined and are conveyed to one of two fine ore bins, each of 4,200m^3 capacity.

The products from the fine ore bins are conveyed via surge bins to two parallel lines utilising KHD high pressure grinding rolls (HPGR) which are designed to crush the material to 80% passing 5.2mm. The roller presses are fitted with two 1745kW motors each. The design feed rate is 1,280tph (1,380tph maximum). The edge products are recirculated to the surge bins.

3.6.4.2 Milling

The ore is ground in two parallel circuits, each using a single stage 6.7 x 11.3m Outukumpu ball mill, with each mill being powered using two 4.9MW motors. The ball mill discharge is cycloned using Krebs D75 cyclones and the underflow is returned to the mill. The cyclone overflow passes to "Intercycle flotation" which consists of three 130m^3 Outotec flotation tank cells to recover coarsely liberated sulphide minerals. The flotation tailings are cycloned and the overflows pass to rougher flotation. The secondary cyclone underflows gravitate via a 1.5mm screen to an XD70 Knelson concentrator. The Knelson concentrates are pumped to a regrind ball mill.

In October 2010, ore processing reached 556kt and in December all the required repairs and maintenance works were completed, including replacement and new design of gearboxes and reinforcement of foundations.

3.6.4.3 Flotation

Rougher flotation takes place in three 130m^3 Outotec flotation tank cells which give a rougher flotation time of 16 minutes. Rougher tailings pass to scavenger (Control) flotation where a further four 130m^3 cells give an additional 20 minutes flotation time. The scavenger tailings are pumped via a trash screen to three Knelson KC-CVD60 concentrators in parallel (continual discharge). The concentrates are cleaned using a single KC-CVD20. The concentrate from this machine joins the concentrate from the grinding circuit for regrinding.

The rougher concentrate is cleaned in two stages, with the first stage consisting of four 10m^3 cells and the second stage of three 5m^3 cells. The residence times are 12 and 10 minutes respectively. The final flotation concentrates are pumped to the regrind mill. The mass pull to the flotation concentrate is 4.8%. The plant has experienced problems in 2010 with insufficient pump capacities within the flotation circuit. This has now been overcome with the replacement of many 6" x 4" pumps with 8" x 6" models.

During the WAI site visit, high pulp levels and minimum froth depths were observed in the flotation plant. However, recent plant recovery statistics (Table 3.20) indicate that the plant is nearing design efficiency.

3.6.4.4 Concentrate Regrind

The gravity and flotation concentrates are combined and ground in a 3.6 x 5.5m ball mill fitted with a 1250kW motor to achieve a product size of 95% passing 45µm.

3.6.4.5 Ultrafine Grinding

The combined concentrates are then subjected to ultrafine grinding, using Deswick mills, to achieve a product grading 80-90% passing 4µm. There are three $2m^3$ mills operating in parallel. The grinding medium is obtained from a French supplier (SiLi beads, 1-6 –1.8mm). The Deswick mills have failed to achieve the target is grind size and are currently only achieving 60-65% passing 4µm.

Since December, the mill's vendors have been controlling the works to assure the production target grind size achieves not less than 85% passing 4µm, which involves the use of a hydrocyclone prior to the Deswick mills, scheduled for implementation in March 2011.

3.6.4.6 Gold Recovery from the Gravity and Sulphide Concentrates

The gold in the gravity and flotation concentrates is partially refractory and the sulphide minerals need to be oxidised to ensure maximum gold recovery. This is achieved via the "Leachox Process", supplied by Maelgwyn Minerals Services (MMS).

After ultrafine grinding, the concentrates, typically assaying 40g/t Au and 25-30% As, are pumped at 30-40% solids to four oxidation tanks in series, each fitted with two "Aachen Reactors", with one operating and the other on standby. These are externally mounted and supply oxygen ($900m^3/h$), which is obtained from an Oxygen Plant, as ultrafine bubbles, thus giving the opportunity for oxidation of the sulphide minerals. Dissolved oxygen levels are maintained at 20ppm. However, the Company plans to replace the Aachen reactors with special "Telescopic Dispersion Units" within the period January to March 2011 in order to increase the level of dissolved oxygen to 30-35ppm.

Lime is added to control pH. After oxidation the pulp is pumped to a preliminary leach column where lime and cyanide is added to a pH of 10-11 and a cyanide concentration of 0.2g/l. There are a further two leach tanks ($250m^3$) and a further eight tanks for carbon in leach (CIL)

3.6.4.7 Detox

There are three tanks for detox. The first two are dosed with lime and sodium metabisulphite for cyanide destruction. Iron sulphate (1,140g/t) and lime are added to the third tank for arsenic precipitation. The Company has installed a Mintec analyser to control cyanide levels for direct measurement of Total, WAD and Free cyanide.

3.6.4.8 Gold Recovery from Carbon

Gold is recovered from carbon using the standard Zardra method at 140-150°C and 2.5 atmospheres. Final gold loadings are approximately 3.5-5.8kg/t. 6t of carbon is eluted giving a gold production per cycle of approximately 25-30kg. The elution cycle is 12 hours and gold stripping efficiency is reported to be 95%. Gold is recovered from the eluate by electrolysis in two cells with 40 cathodes. The gold bearing sludge is removed by high pressure water sprays and filtered and dried. The electrolysis product is smelted to produce Doré assaying 85-92% Au in 12kg bars. The carbon is regenerated at a rate of 500kg/h in a furnace at 600°C.

3.6.5 Consumables

The consumables for the process plant are given in Table 3.19.

Table 3.19: Vasilkovskoye Process Consumables	
Crushing and grinding	**kg/tonne**
Steel balls	1.4336
80mm	0.6846
70mm	0.2251
60mm	0.4259
40mm	0.1091
Soda ash	0.4000
Flotation and Gravity	
Metal balls 25 mm	0.0489
Potassium butyl xanthogenate (BKK)	0.1318
Sodium-butyl Aeroflot	0.0242
Pine oil	0.0331
Frother X-133	0.0003
Collector AERO 7279A	0.0003
Depressor AERO 7261A	0.0015
Flocculant Magnafloc 10	0.0040
Lime	0.0024
Coagulator Mafnafloc 1697	-
Ceramic balls	0.0116
Flotation agent (OXAL) T-92	0.0009
Flotation agent Flotanol C7	0.0009
Thickening, industrial water supply and tailings	
Flocculant Magnafloc 10	0.0380
Coagulant Magnafloc 1697	0.0139
Hydrometallurgy	
Sodium cyanide, 88% activity	0.3455
Lime 70% activity	1.3087
Diesel fuel	-
Caustic ash NaOH, 96% activity	0.0352
Hydrochloric acid (HCl), 35 % activity	0.0117
Sulphuric acid	0.0014
Sodium metabisulphite	1.2130
Ferrous sulfate grade II,	1.3931

3.6.6 Metallurgical Performance

The technological recoveries for the process plant for 2010 are given in Table 3.20 below.

Table 3.20: Vasilkovskoye Process Plant Recoveries					
Month	**Tonnes treated**	**Head Grade Au ppm**	**Recovery %**	**% Flotation Recovery**	**% Hydrometallurgical Recovery**
January	154,383	1.65	47.9	64.2	74.5
February	130,661	1.78	50.2	67.4	74.3
March	230,131	1.65	42.5	63.5	68.5
April	247,041	1.78	61.2	71.9	85.1
May	450,915	1.7	59.0	66.9	81.3
June	441,540	1.77	59.3	55.0	82.5
July	324,685	1.82	62.6	77.4	80.9
August	373,276	2.07	67.2	81.4	82.5
September	257,633	2.02	62.5	79.2	77.8
October	556,868	2.13	61.8	79.7	77.1
November	435,297	2.04	59.0	82.1	71.8
December	203,261	2.28	57.6	77.1	74.7

There was a gradual increase in recovery throughout the year. The maximum recovery was 67.2% and it appears that recoveries are ranging between 60-67%. The plant head grade has generally increased throughout the period, ranging from 1.65ppm Au (January) to 2.28ppm Au (December).

The tonnage throughput peaked in May and April, but decreased again in August and September before reaching 556kt in October. However, throughput decreased during November and December due to refurbishment of the Outotec mills.

3.6.7 Labour

A total of 597 persons is employed in the Plant. There are three senior managerial positions, and a total of 18 supervisors. There are four senior mechanical positions and an electrical foreman with two electrical engineers.

A further 48 are classified as "technical and engineering" staff and there are 516 workers.

3.6.8 Assay Laboratory

The Vasilkovskoye mine contains a well-equipped assay laboratory. The laboratory is not accredited but there are plans to apply for accreditation in the near future and also install a Quality Management system. The laboratory typically assays 8,000 mine and geological samples and 1,500 plant samples per month.

Sample preparation is achieved in three stages using a combination of jaw crushers, rolls crushers and ring mills to produce material that is 100% passing 200 mesh. The equipment is both Russian and Australian (Rocklabs). The sample preparation laboratory was well equipped and maintained.

Gold analysis is undertaken using fire assay (in duplicate) on 50g trials for geological or plant feed and tails samples and 25g for plant concentrates. The laboratory has four furnaces, each capable of holding 24 crucibles. There are also three cupellation furnaces.

The beads are parted using nitric acid and the gold weighed using a Sartorius micro balance to a limit of detection of 0.2ppm Au. The laboratory is also equipped with atomic adsorption machines (Perkin Elmer and Kvant) for solution analysis. The laboratory also has an "Express Laboratory" for Lasersizer particle size analysis and rapid analysis of process solutions (Analysis, pH and cyanide).

The laboratories QA/QC procedures include duplicate gold analysis, where the agreements between analyses must be within certain limits which are dictated by state protocol. There is a regular programme of external check analyses. The laboratory also uses international reference materials such as those supplied by "Geostats".

3.6.9 Conclusions

The plant has experienced a long ramp up period due to problems with equipment and lower than expected process recoveries. However, recent initiatives by the company have addressed many of these issues to allow plant production and recovery to approach design parameters, although optimal conditions have yet to be reached.

It is reported by Kazzinc that the key to the Vasilkovskoye process is the ultrafine grinding of the bulk concentrate in order to liberate the fine gold. This is achieved through the use of Deswik ultrafine grinding mills which take the particle size to 90% <4µm. The ultrafinely ground pulp is then subjected to an intensive aeration process in four columns. Oxygen is injected through telescopic gas dispergators (produced through Kazzinc's JV with Bakor Ceramics of Russia), where oxygen is forced into a ceramic membrane and finely diffused into the pulp, with approximately 20-25ppm dissolved oxygen content. The process results in an Iron hydroxide fim coating the outer shell of the mineral particles, thereby passivating the refractory mineralisation prior to the cyanide leach process. The pulp passes through cavitators, which are required in order to clean the surface area of the mineral particles through rapidly changing pressure. The pulp is then processed through a conventional CIL system.

The process is designed to keep the arsenopyrite molecular structure intact, oxidising only the outer surface in order to reduce subsequent cyanide consumption and increase gold dissolution. A small proportion of the arsenic will be dissolved, which is subsequently handled through the addition of iron sulphate into the pulp in order to neutralise the arsenic into scorodite prior to discharge into the sorbtion tailings dam.

Empirical tests to date, performed by Kazzinc, show that sorbtion tailings contain only unliberated gold which has been interpreted by Kazzinc as indicating the success of the aeration process but highlights the sub-optimal nature of the current grind size achieved by the Deswick mills.

Going forward, WAI believes that the problem of low process recoveries may be difficult to fully solve as although the Leachox process is not designed to achieve a high degree of oxidation of the sulphide minerals it is very dependent on grind size to liberate refractory gold particles. The low pulp temperatures being achieved in the Leachox Plant certainly indicate that very little oxidation of sulphide minerals is occurring, therefore, without the grind performance, it is difficult for WAI to see clearly how the refractory or locked gold will be liberated.

3.7 Environmental, Social and Health & Safety Issues

3.7.1 Introduction

3.7.1.1 Scope of Study

This review of the environmental and social performance of the Vasilkovskoye Mine and processing project is based on a brief site visit and reconnaissance, together with discussions with staff of the geology department and Department of Health Safety & Environment. In the short time available it is only possible to have an overview of the project and the way that the company manages its health, safety, environmental and social obligations.

Whilst WAI believes it has gained sufficient insight into the key issues and performance, there may be additional information that was not seen, or variations in interpretation of the available data that could not be explored further.

3.7.1.2 Method of Study and Information Sources

A site visit was undertaken over 2 days between 12 and 14 October 2010. Documents and plans were inspected but not translated from Russian, and key data were provided by the company staff. The main documents inspected are listed as appropriate in this Chapter.

In addition to site specific information, the overall Kazzinc Company Environmental Policy and Programme was discussed with the Chief Environmental Manager.

3.7.2 Environmental & Social Setting and Context

3.7.2.1 Landscape, Topography

The Vasilkovskoye mine is located about 10km from Kokshetau town, in a flat, open steppe landscape. The mine buildings, structures and dumps dominate the landscape.

3.7.2.2 Climate

The climate is characterised by a semi-arid continental climate, with hot dry summers and cold dry winters. Average annual rainfall is around 260mm.

3.7.2.3 Land Use and Land Cover

The land cover is dominated by short dry grassland (mainly Stipa species) with no trees and only very occasional shrubs. Lower lying areas consist of wetland and ponds, probably seasonal in nature.

3.7.2.4 Water Resources

Surface Waters

Whilst the mine area has few surface watercourses, the area drains generally to the south and southwest into the River Chaglinka, which flows northeast to Lake Chagly (Shaghalaltengiz).

To the south of the mine, in the vicinity of the old heap leach dumps and current TMF, the land is lower lying and there are a number of small seasonal ponds, probably arising from the high water table (Photo 3.2). This zone generally drains east into the river (visible in Photo 3.3), though there is no defined watercourse or drain.

Groundwater

The groundwater is sparse and there are no major aquifers. In the area of the TMF it is within 10m of the surface and is reported to be saline. In the vicinity of the open pit the GKZ groundwater reserve is stated to be 3,800m^3/day. Groundwater flows are very slow, due to the low permeability of the rocks.

Water Supply

The mine is licensed to obtain water from the River Chaglinka (potable), groundwater from the open pit, boreholes (near the river) and from the Murzakolsol Lake (industrial water). The water sources for Murzakolsol Lake are surface waters, drainage water from Kokshetau city after treatment and precipitation. Water from Murzakolsol Lake is neutralised and treated using ultraviolet lamps.

Nearby villages obtain their water supply from boreholes.

3.7.3 Communities and Livelihoods

There are 3 local villages:

a) Konysbay, 2km to the east of mine, on the River Chaglinka and close to the mine entrance (see Photo 3.2);
b) Altybay, adjacent to Konysbay and the other side of the river; and
c) Dongulagash, 5km to the west of the mine.

Kokshetau is the nearest major settlement, 20km to the south-east of the mine. Kokshetau is the capital of Akmola Oblast and has a population of around 150,000. Akmola Oblast has a population of around 810,000. The mine is located in the District of Zerendinsky, which has a population of 41,000. The district centre is Zerendi, about 60km south of the mine.

The area has a low population density, concentrated in settlements. The majority of the people in the local villages and a significant number in Kokshetau work for the mining company, directly or indirectly via contractors and suppliers.

Apart from the mine, the main livelihoods are agricultural, particularly cattle and sheep grazing, and some grain production. In the vicinity of the mine the only land use is for subsistence grazing, and the mine provides the only source of livelihood.

3.7.3.1 Infrastructure & Communications

The mine has road connections and a rail connection. Most of the plant and equipment for the mine was delivered by rail.

3.7.4 Project Status, Activities, Effects, Releases & Controls

3.7.4.1 Project Description & Activities

Past activities

Originally the open pit extracted oxide ores and produced gold by heap leaching. It is understood that some ore remains in the heaps (Photo 3.2).

Photo 3.2: Old Heap Leach Dumps and Associated Infrastructure
Note low lying areas of wetland

Current Operations

The arrangement of the current and past mine components in relation to the highway, River Chaglinka and Komysbay village is shown in Photo 3.3.

Photo 3.3: Aerial View Showing Location of Main Mine Components. The Outline of the new TMF is shown approximately

The open pit has a large surface area, but is still relatively shallow, the company having utilised the current excavation capacity to strip overburden whilst normal production through the plant has been limited due to construction and ramp up. Associated with the pit are two waste rock dumps – one to the north-west and one to the south-east (Photo 3.3).

Along the west perimeter of the pit is a water storage pond, into which the pit water was pumped. However, this is now decommissioned and water is pumped to the TMF.
The plant area has a number of facilities:

- Mobile plant maintenance area;
- Primary crusher and ROM stockpile;
- Milling (ball mills);

- Flotation concentrator and sorption (Leachox Process) plant;
- Rail head for equipment delivery, and warehousing;
- Heating plant – coal fired;
- Tailings treatment and detox plant; and
- Offices and administration.

In addition, to the south and south-west of the main plant area are located:

- Tailings pipelines and return water lines;
- The main flotation tailings disposal area (TMF);
- Sorption tailings disposal area (STMF);
- Water storage and conditioning lagoon (previously a tailings disposal area from oxide ore treatment); and
- Old heap leach dumps and associated derelict infrastructure.

The mining methods and processing flowsheet are described in a separate section of this report.

The sulphide ore has a high content of both arsenic and pyrite, which give rise to environmental complications as well as processing difficulties.

3.7.4.2 Energy Consumption & Source

Energy is used as follows:

- Electricity - Connection to Kazakh grid. Majority of power requirement; main use in process plant, conveyors, etc;
- Coal – Heating Plant; and
- Diesel fuel – Mobile plant in the mine.

Whilst the mine is connected to the Kazakhstan power grid, which has high carbon intensity (being coal, oil and gas dominated), Kazzinc operates a hydroelectric power station, which produces much of the required power of the company. Thus the total carbon emissions of the company from electricity are much reduced.

3.7.4.3 Mine Wastes – Rock

Overburden and waste rock is disposed of in two dumps, adjacent to the open pit (Photo 3.3). The waste rock production so far is high in relation to ore production. This means that the mine has used its mobile plant capacity to pre-strip overburden in preparation for ore production.

Geochemical information on the sulphide and sulphate content of the waste is very limited; however it is considered that the environmental stability of this waste material is good. The results of runoff monitoring indicate an alkaline pH, which is regularly controlled. Off balance ore is dumped separately and is intended for future processing. The cone of depression caused by the open pit results in all drainage waters collecting in the pit base. Water is pumped from the open pit to the flotation tailings dam as recycled water.

> ***WAI Comment:*** *Whilst the waste rock may be largely inert, there may be areas or zones from mineralised parts of the orebody, which have the potential to be acid generating or contain metals (particularly As). Waste rock should be tested for ARD and metals (total content and leachability) on a routine basis on the dumps. Zones of potential contamination can then be placed within the dump to isolate them in safer conditions. Currently, the geochemical composition of pit drainage waters, including waste dumps is regularly monitored.*

3.7.4.4 Mine Wastes – Tailings

Tailings Properties & Treatment

The process plant produces two tailings streams:

1. Flotation tailings, from the first stage of concentration; and
2. Sorption tailings, from treatment of the Au (and As) rich concentrate by oxidation to release the Au, and subsequent cyanidation.

Recent analysis of tailings solids provided by Vasgold is given in Table 3.21 below.

Table 3.21: Tailings Analysis

Element	Unit	Flotation tailings	Sorption tailings
Fe	%	0.74	15.92
Pb	%	0.002	0.009
Zn	%	0.002	0.008
Cu	%	0.003	0.15
As	%	0.17	19.80
S	%	0.14	11.17
CN total	mg/l		11 - 160
CN free	mg/l		7.4 - 3
Average of monthly samples, April – September 2010			

It can be seen that the Sorption tailings contains a substantial quantity of iron and sulphur (pyrite), which may have acid production potential, and contain arsenic. Even the flotation tailings contains significant quantities of Fe, As and S, and may be geochemically active. As a result, the tailings may represent an environmental hazard, and the sorption tailings may also be a potential hazard. The alkaline character of the tailings decant water will reduce the dissolution of sulphide minerals, and the thick layer of natural clay below the waste dump is considered to reduce the risk of environmental contamination.

Prior to disposal, the sorption tailings are treated in a detox unit in a two-stage process:

1. CN destruction, in two tanks – addition of lime and sodium metabisulphite. Residual CN levels are normally within the range 7mg/l to 10mg/l; and
2. Arsenic precipitation – addition of Iron sulphate (1,140g/t) and lime to precipitate arsenic as the more stable calcium iron arsenate (also referred to as 'scorodite', though this is slightly different from naturally occurring scorodite).

Levels of As, CN and pH are regularly monitored at the processing plant discharge to assess the effectiveness of the detox and further analyses for these compounds are performed in the TMF. Cyanide is destroyed by sunlight, and analyses indicate a consequent reduction in levels. In 2010 pH has not dropped below 9, with an average between 9-12.

Arsenic levels in the liquid phase in the tailings flow and decant water are monitored weekly and there is no indication of solubilisation of As in the short term. CN levels are monitored daily.

Tailings Pipelines

The flotation and sorption tailings are delivered to the TMF in separate pipelines, two lines for each (one operational, one backup). Pipelines are conventional high density plastic, and are laid directly on the ground surface, with no secondary pipeline containment; however there is an emergency spill discharge pond in the event of an incident, into which tailings will flow if there is a pipeline breach. Pipelines are inspected twice per shift by trained personnel and once per week by the Chief Engineer.

Tailings Disposal

The tailings management facility (TMF) was designed by Mechanobr Enginerring, St Petersburg, in 2009 and was subject to a full OVOS concluded in 2009 (along with the mine and plant facilities). A particular issue of concern was the safe disposal of the tailings and whether the TMF or STMF needed to be lined.

The selected site for tailings disposal is underlain by 6m to 10m depth of low permeability clay (<1x10-9m/sec), with a regulatory requirement of 1x10-7m/sec. This depth decreases towards the south-east of the TMF, the lower lying areas. The sorption tailings are contained within a separate paddock, in the north-west portion of the TMF area (see Photo 3.2), i.e. where the underlying clay is thickest.

The paddocks for both the sorption and flotation tailings have been developed and in situ tests were performed to assess permeability and the natural clay layer was not compacted. The tailings dam embankments are compacted rockfill and will be constructed in two stages – the first stage comprising 3 lifts, the first of which has been completed and the second has commenced.

The main TMF parameters are summarised in Table 3.22.

Table 3.22: Tailings Management Facility Parameters

Parameter	Units	1st Stage	Total on 2nd Stage (20 years)	
		Flotation tailings	Flotation tailings	Sorption tailings
Elevation, datum	metres	240	254	240
Area	Mm^2	5.96	6.20	2.16
Volume	Mm^3	54.54	144.02	6.5
	Mt (@1.7t/m^3)	93	245	11
Perimeter of dam	metres	8805	11,108	5801
Max height of dam to crest	metres	20	34	8

The intention is that the sorption tailings will retain a cover of water over them, in order to prevent pyrite oxidation. It is reported that since the TMF has recently been developed, to date, insufficient water has collected to cover the TMF surface. The company considers this water will be accumulated in 1-2 years time. It is also intended to rework the tailings beach in the future. The sorption tailings material has been assumed to be of the highest hazard class, which has informed the design of the TMF. To date, the company has designed all storage structures in line with national standards, which has been approved by the required regulatory authorities. The company considers that the pH of tailings decant waters which is maintained at pH 9 is sufficiently high to neutralise any potential acid generation.

> ***WAI Comment:*** *WAI has not independently examined the TMF design and OVOS, and has thus not verified the potential seepage rates and long term risks to water resources from the sorption tailings. WAI would recommend that the character of the sorption tailings material should be further clarified via specific ARD testwork, and that a review against compliance with international standards (IFC Performance Standards and EU Mine Waste Directive) should be undertaken as soon as possible.*
>
> *In Kazakhstan, tailings and other mine wastes are considered as a mineral resource or asset (an industrial raw material under the Law of Republic of Kazakhstan, on Mineral Resource & Subsoil use), on the basis that either technology development or economics (i.e. increased commodity prices) will enable them to be economically re-processed in the future. Whilst this may be true in some cases, the consequence of this policy is that the storage of potentially toxic materials can be considered as a temporary requirement. Thus long term or permanent risks to land and water resources are less critical, and storage requirements might be to a lower standard than would be for permanent final disposal.*

The TMF has a naturally very low permeability clay base layer, as indicated by the company, so the risk of seepage, particularly from the sorption tailings, is very low. WAI does not consider that there is a risk of acid being generated; as this is mitigated by the high pH of the tailings decant water in the long term.

3.7.4.5 Water Management & Effluents

The Vasilkovskoye Mine operates as a mainly closed system with maximum water recycling. This is necessitated by the scarcity of water supply, as well as by environmental considerations and minimisation of discharges.

- Groundwater from the open pit is pumped to the TMF for purification and recycling;
- Water decanted from the flotation tailings is stored in the water clarification dam, where natural biological activity provides some treatment;
- Water from the water clarification dam is recycled to the plant as industrial water;
- Water is not recycled from the sorption tailings, in order to maintain a water cover over the tailings;
- Industrial makeup water is abstracted from the Murzakolsol Lake and potable water is sourced from the Chaglinskoye water dam; and
- 'Domestic' waste water is treated in a biological treatment plant and the effluent recycled via the TMF.

Losses of water are likely to include:

- Process losses;
- Evaporation from the TMFs and water clarification dam; and
- Seepage to groundwater from the TMFs and water clarification dam.

In addition to the pumped pit water discharge, it is possible that some discharge occurs via shallow groundwater seepage reaching the natural drainage from the south-eastern corner of the TMF/leach pads area, although company monitoring data indicates that there has not yet been any contamination of water via this route. Potential drainage is intercepted by one pumping station (and another under construction) which collects and pumps any discharge to the old TMF, although none has been encountered to date.

Vasgold have commissioned hydrogeological modelling of the potential groundwater drawdown from pit dewatering. At its full extent this will extend to a radius of 7km from the pit, except towards the river which is 4.8km distant. It is possible therefore that pit dewatering will have a significant effect on some surface waters, and may affect the river, although the river bed is comprised of low permeability clays, and the cone of depression caused by the open pit is considered to intercept all drainage waters. Pit water is pumped at a rate of 750,000m^3 per year, whilst the average annual volume of Chaglinka River is 22,450,000m^3 per year.

3.7.5 Emissions to Air

Emissions to air are summarised in Table 3.23. Emissions are generally of low significance, although WAI considers that following emission is significant dust blow from the open ore stockpile, which is particularly prevalent during dry, windy conditions. Currently, this is stated to be controlled by water spray on the conveyors as an interim measure. Three options for treatment in 2011 are being considered: 1. enclosure by a concrete structure, 2. snow covering of the dump in winter and water spray in summer, 3. Dust removal by telescopic extractor duct.

Table 3.23: Potential Emissions to Air		
Fugitive dust from drilling, blasting	Open pit	Inert and mineralised rock dust. Controlled by dust filters on rigs.
Fugitive dust from loading and haulage of ore	Open pit and haul roads.	Mineralised rock dust. Controlled by watering of haul roads.
Deposition and storage of coarse ore prior to milling	Ore stockpiles	Mineralised rock dust. Conveyors are covered; dust from stockpile is uncontrolled (winter only).
Process emissions	Process plant stacks and ventilation	Dust and acid vapours. Controlled by filters and enclosure.
Combustion gasses from coal-fired heating plant	Chimney stack at heating plant.	Combustion gasses, smoke particulate. Particulate filters in place, no scrubbing of gasses (low emissions).
Fugitive dust from tailings surface	TMFs	Hazardous dust, containing As and metals, which could contaminate surrounding land. (not confirmed by company monitoring) Controlled by maintaining water cover and wet tailings beaches; effectiveness is dependent on water availability and it is estimated to improve within 1-2 years as more water accumulates.

3.7.5.1 Waste Management – General

General waste and recyclable materials are collected and stored separately as appropriate, for disposal by specialist contractors. Non-recyclable waste goes to the local municipal landfill. As part of the environmental license the company maintains records and makes returns on the quantities of waste produced, under different waste categories.

3.7.5.2 Hazardous Materials Storage & Handling

Oils and fuel (hydrocarbons) – are stored in proper bunded facilities in accordance with licence requirements and international standards.

Cyanide – separate and secure storage and handling facilities. The company has recently undertaken a review of requirements under the International Cyanide Code and has prepared a detailed and comprehensive specification of requirements and actions for compliance (copy received by WAI in draft) and the company is currently considering the implementation of such requirements and actions.

3.7.5.3 General Housekeeping

In general housekeeping at the mine is excellent and all areas are maintained well. No evidence was seen of uncontrolled tipping or abandonment of disused equipment, scrap, containers or wastes.

3.7.5.4 Fire Safety

Good quality systems and arrangements are in place.

3.7.5.5 Security

The site is well secured within a perimeter fence and public safety is maintained. The plant area is accessed via manned security gates and scanners.

3.7.6 Permitting

3.7.6.1 ESIA/OVOS

The OVOS (EIA in accordance with Kazakhstan Environmental Code) process commenced in 2008 and was completed in 2009. This covered all elements of the current project (open pit mining, processing, ancillary activities, waste rock and tailings disposal) with the exception of the chemical storage, which was added as an addendum. The work was carried out by Ecotera PLC of Almaty, which is a specialist environmental design institute licensed by the Ministry of Environment.

A copy of the conclusion of the 'Expertise' (expert review and issuing of environmental permit) was provided to WAI. No major issues of concern were identified by the Ministry of Environment.

3.7.6.2 Environmental Permits and Licences

Under the Environmental Code the following licences are required, and copies were provided to WAI:

- Emission Permit, setting out annual limits of releases to air, water and wastes;
    - Air emissions – dust, heating plant, and process plant stacks/vents;
    - Solid wastes – tailings, waste rock, heating plant slimes, containers, oil & filters, office/household waste; and
- Water use permit – open pit, boreholes, Chaglinka river/reservoir and municipal waste water use.

There are no water discharge permits as there is no planned discharge of water to surface or groundwater.

> ***WAI Comment:*** *The company has the necessary environmental permits and licences in place, in accordance with Kazakh Environmental Code.*

3.7.7 Environmental Management

3.7.7.1 Environmental Policy and Company Approach

Kazzinc has a strong central corporate environmental management function, based in Ust-Kamenogorsk. There is a Chief Environmental Manager, reporting to the Vice President of Operations, and two environmental specialists at the headquarters, supporting the site-based staff. The company therefore has a consistent approach to environmental and social management systems, procedures and standards.

The company is certified across all its sites for:

ISO9001 – Quality Assurance	Certified in 2004 and audited in 2010
ISO14001 – Environmental Management System and ***OHSAS18001*** - Occupational Health and Safety Management System	Certified in 2006, recertification audit in 2009 and supervisory audits of company facilities alternately each year, starting 2007.

Glencore is a majority shareholder in Kazzinc and thus also exercises an overarching commitment to Health, Safety, Environment and Community, through the Glencore Corporate Practice commitment applied to all subsidiary companies and operations. WAI understands that all Kazzinc assets were audited by a Glencore team in early 2010. The findings of this are not available to WAI, though it is understood that the main recommendation was in connection with implementing the International Cyanide Code and the company is currently considering the implementation of such requirements and actions.

Kazzinc also commissioned specialist environmental institute ECOTERA LLC of Almaty to undertake a full audit of all the company's assets. The report: Assessment of Compliance of Activities of Kazzinc LLC with the Nature Protection Laws of the Republic of Kazakhstan and with the Requirements of Controlling Agencies, Auditors Report to Management, was presented in September 2010.

> ***WAI Comment:*** *It is clear that, at a corporate level, Kazzinc is progressive and responsible in its approach to environmental, social, health & safety management across all their operations. They have progressively implemented measures to improve air quality and water management and tailings management and maintain consistent standards in accordance with Kazakh norms, policies and requirements.*

3.7.7.2 Environmental Management Staff & Resources

Health & Safety and Environmental are separate departments, under the Department of Production and Safety, reporting to the Director of Production.

The Head of the Environmental Protection department has 4 staff: two responsible for Control (processes, checks, operations, etc.) and two responsible for monitoring (including emissions control, environmental taxes, etc.). In addition, sampling and analytical requirements are outsourced to contractors.

3.7.7.3 Systems and Work Procedures

Vasgold has prepared a series of work procedures, dated 2009, copies were provided to WAI:

- PC-EP-016 Process card for environmental management; and
- PLC-EP-011 Environmental policy and procedures.

These set out the main processes and procedures to be followed. Parameters and norms that these apply to are set out in detail in the Environmental Permits. The company adheres to these requirements; there are no additional self-imposed requirements.

Vasgold are due to be audited on 2011 in accordance with ISO14001 and OHSAS18001 certification requirements.

As indicated above, Vasgold is considering the implementation of measures in accordance with the International Cyanide Code, covering transportation, storage, handling and disposal of CN used in connection with gold recovery.

3.7.7.4 Environmental Monitoring, Compliance & Reporting

Monitoring of a range of parameters as required by the Environmental Permit, and to an agreed protocol with the Government, is undertaken quarterly by an independent laboratory. In addition, weekly samples from boreholes are monitored for a limited range of basic parameters by the mine laboratory.

Groundwater boreholes are located downstream (south-east) of the TMF, but none upstream and none around the mine or process plant. CN and As levels are monitored in these boreholes and in surface waters. Data over the last 2 years seen by WAI did not show any significant levels of either contaminant, although SO_4- Cu+ and Fe+ levels could also be measured.

Soils are monitored once per year around the mine. It is reported that As levels are naturally high background; no significant elevated metal levels are evident so far.

***WAI Comment:** As the mine has only been operational at a reduced production and for a short period, significant contamination would not be expected and it is not yet possible to comment on compliance. WAI considers that the mine has a good monitoring regime in place.*

3.7.7.5 Emergency Preparedness & Response

This is included within the Health & Safety policy and covers environmental aspects. There are plans covering the pit, plant, tailings, fuel storage and chemical storage, and other aspects such as tailings pipeline burst, as part of a full Emergency Preparedness and Response Plan.

3.7.7.6 Training

The Environmental Manager and staff carry out inductions and specific training on environmental awareness and procedures to supervisory and management staff.

3.7.8 Social and Community Management

3.7.8.1 Stakeholder Dialogue and Grievance Mechanisms

WAI understand that 80% of the local population in the vicinity of the mine are employed in the mine. The mine therefore has a very close relationship with its neighbours.

There is regular communication with communities through public meetings and the local press, and regular meetings with the heads of local villages. No issues or complaints have been raised in the last 4 years and the community see the mine as a positive benefit to the area, bringing jobs and prosperity.

3.7.8.2 Social Initiatives and Community Development

Vasgold has a range of community development and support activities:

- Use of equipment for small works;
- Tree planting and support for local forestation initiatives;
- Participation and financial support for local and community events and projects; and
- Provision of sports ground for the disabled.

Kokshetau is a significant town and has a good social infrastructure, including culture, health and recreational facilities, housing and communications.

3.7.9 Health & Safety

3.7.9.1 Health & Safety Management Arrangements

The Head of Labour Safety reports to the Health and Safety Director, similar to the Head of Environmental Protection. He has a staff of 7 occupational safety specialists and 9 first aid and medical specialists (total 17).

The company has in place an extensive management system and documentation, including the following (copies provided to WAI):

1. Safe Labour Code, setting out the mission, principles and rules, dated March 2010;
2. Process card for implementation of the health care and safety of the workforce – setting out the requirements for all aspects of the mine, inputs and outputs, legal responsibilities and documents;
3. Workflow tables – detailed responsibilities, resources, parameters, reporting procedures, training, maintenance of records, etc; and
4. Specific activity 'passports' (working procedures) – 54 covering various stages and activities.

There are regular health checks for employees involved in hazardous activities and who control machinery, including blood pressure, sight/hearing, alcohol and drug use.

Signage and PPE observed around the mine by WAI were good and employees clearly respected and followed safety requirements.

3.7.9.2 Performance and Accident Records

The mine considers that it has a good no-blame culture and incidents are reported reliably, for both employees and contractors. The reporting procedures for accidents and incidents cover 5 main categories and are reported monthly. Data seen by WAI showed that for 2010 to September, there had been 73 no-lost-time incidents, 11 lost-time incidents and 0 fatalities amongst employees, and only 1 no-lost-time incident for contractors.

Workplace monitoring for light, noise exposure, respirable air quality and exposure to hazardous chemicals (such as CN) is carried out on a regular basis by specialist contractors.

3.7.10 Mine Closure & Rehabilitation

3.7.10.1 Mine Closure Plans

Vasgold has prepared a closure and rehabilitation plan for the open pit, waste rock dumps and objects directly serving the mining process, in accordance with the requirements of the Kazakh Subsoil Use Code. This is the responsibility of the mine manager and does not involve the environmental department. There are no plans covering the old heap leach areas, the process plant (licensed under a separate Industrial Code) and the TMF (considered as industrial raw material rather than mine waste).

The mine closure plan was seen but not reviewed in detail by WAI. In general it included the following:

- Construction of a safety berm around the open pit;
- Dump slopes graded and planted with vegetation; and
- Buildings and mine related structures (offices, conveyor, vehicle maintenance, refuelling area) dismantled and removed.

There are no proposals for the pit itself, as it is considered unusable due to the steep slopes, arsenic and acid contents. It is likely to flood to natural groundwater levels.

WAI understand that the company is considering the rehabilitation of the heap leach areas, dumps and unused infrastructure, but no details or plans were reviewed. There is a plan to reprocess the heaps again, and there is a contract with a Russian company to assess different processing circuits for this material, and a specialist has been employed to investigate processing options.

3.7.10.2 Financial Provision for Closure

The Subsoil Use Code imposes contractual obligations on the mine licence holder, to fund the closure requirements. A sum of 0.1% of all mine operating costs (i.e. not including the plant and other activities) has to be placed in an escrow account, separate from the company assets. In addition, a liquidation programme has to be agreed with the Government and the fund made up to cover this. This fund cannot be drawn down until mining ceases and can only be used for closure requirements.

WAI did not receive information on the detail of the closure requirements, since an action plan is not required until 2-3 years prior to closure under Kazakh standards. The sum of accrued estimated costs for closure as at 31.12.10 was US$2,120,900 as per the contract requirements. With a 'good will' sum of US$37,420,000 set aside by the Company to cover all closure requests.

WAI Comment: *WAI considers that the present arrangements for mine closure and restoration of the Vasilkovskoye Mine are fully compliant with Kazzakh requirements,although they are insufficient when compared with international requirements.*

It is advisable that a full and comprehensive mine closure and rehabilitation plan, with estimated costs, is prepared, covering not only the mine and waste dumps, but also the process plant, TMFs, the old heap leach areas, administration/offices, power plant, warehousing and related infrastructure. The safe long term (maybe permanent) storage of the sorption tailings is a particular concern.

Once the closure and long term risks and liabilities are quantified it is advisable that a financial provision and guarantee is put in place to cover these. This could probably be arranged through Kazzinc and/or Glencore corporate, as parent company guarantors, subject to whether the assets are publicly or privately owned.

3.7.11 Conclusions

3.7.11.1 Environmental and Social Liabilities & Risks

For the most part, the Vasilkovskoye Mine has a very high standard of environmental, social, health and safety management. It is compliant with Kazakh licences, permits and norms. Both the environmental and labour safety functions of the company are of high quality and well organised, and supported by a strong corporate policy, commitment and team.

There are no known social, community or cultural issues or impacts that need to be addressed, and no displacement or compensation requirements. The mine appears to have a good relationship with the local communities, and does not put undue pressure on the social infrastructure of the District or Oblast.

WAI emphasise the following areas requiring further Company attention:

1. The characterisation and safe long term disposal of arsenic and acid-producing tailings, notably the sorption tailings. Monitoring and protection of groundwater and related surface water resources is important regionally, via the River Chaglinka;
2. Dust control from the exposed tailings surface, particularly a backup irrigation system during periods of water shortage;
3. Dust control from the dry ore stockpile, from the open pit and haul roads, to prevent contamination of surrounding land; and
4. Long term closure liabilities for all the project areas need to be addressed, with a full closure plan, rehabilitation and costs, and a long term environmental liability risk assessment.

The Company considers that the above areas already receive regular attention and are being fully managed with regard to national standards, but is prepared to assess any further requirements to achieve international practice compliance.

3.7.11.2 Compliance with Local and International Standards and Expectations

The project is generally compliant with international standards and expectations, with the exception of the potential liabilities and risks identified above. It is likely that the management of hazardous tailings and the mine closure provisions fall short of international practice, although WAI believes that the company's attention is focussed on further improvement in connection with these issues and there are no reasons why they cannot be readily resolved in the near future.

3.7.11.3 Recommendations for ESAP

Table 3.24 shows WAI recommendations for inclusions in an environmental and social action plan for the mine to achieve international compliance. All these issues are being addressed by the Company.

Table 3.24: Recommendations for Environmental & Social Action Plan

	Action	Priority & timescale
1.	Geochemical and environmental characterisation of the waste rock and tailings (flotation and sorption) – acid-base accounting and TCLP/SPLP leaching tests.	Immediate, high priority.
2.	Further review of tailings disposal arrangements, particularly the sorption tailings, given information from (1) above and hydrogeological modelling of seepage movements.	Q2 2011, high priority.
3.	Depending on the outcome of (1) and (2) above, implement remedial measures and improved containment for sorption tailings.	Q4 2011, 2012, priority depends on previous.
4.	Design and implement dust control for ore stockpile; possibly by containment or wind barriers.	2012; medium priority.
5.	Prepare a full and comprehensive mine closure and rehabilitation plan for all current and past project areas, with costs and timescales. Include a post closure Environmental Liability Risk Assessment.	Q4 2011; medium priority.
6.	Based on (7) above, it is advisable to provide a mine closure guarantee and fund for implementation.	2012; medium priority.

4 MALEEVSKOYE DEPOSIT

4.1 Introduction

4.1.1 Location & Access

The Maleevskoye polymetallic mine is situated some 17km to the north of the town of Zyryanovsk (where the concentrator is located) which in turn is situated 186km east of Ust-Kamenogorsk, northeastern Kazakhstan.

The geographic coordinates for the mine are latitude 49°52'57"N, longitude 84°16'49"E (Figure 4.1).

Figure 4.1: Location of Maleevskoye, Northeast Kazakhstan

The mine is linked by a graded road to Zyryanovsk (a town established on the site of historic lead-zinc production), and a paved road to Ust-Kamenogorsk, with a driving time between the two of approximately 3 hours. The Russian border lies about 45km to the northeast and the Chinese border about 200km to the southeast.

4.1.2 Topography & Climate

The area surrounding the mine shows moderate mountain relief with absolute elevations from 450m to 920m. Much of the area is covered by superficial diluvial-proluvial material which ranges from 3m to 15m in thickness effectively masking bedrock.

Vegetation consists of high mixed grass and thick bushy undergrowth with small groves of birch, aspen, poplar and bird-cherry.

The Bukhtarma River flows south of the deposit and is the main artery of the region along with its tributaries the Khamir and Bobrovka, the latter flowing through the western part of the deposit.

The region has a typical extreme continental climate characterised by a long winter (November-February) and short summer (June-August), both with sharp variations of daily temperatures. The average annual temperature is 1-2°C, with annual absolute temperature variations from -52°C in winter to +40°C in summer. The average annual precipitation at Kutiha, west of the deposit, is 700mm, distributed evenly throughout the year. Snow cover normally prevails from November to May.

4.1.3 Infrastructure

The region is sparsely populated with the majority being concentrated in the town of Zyryanovsk, which has some 50,000 inhabitants. The town has grown up around historic polymetallic mining and processing, although the deposits close to the town are now exhausted. As a result, it is only mining and agriculture that are the main sources of income in the region.

Electrical power is supplied to the mine from the Bukharma and Ust-Kamenogorsk power station, whilst water is taken from the Khamir River.

4.1.4 Mineral Rights & Permitting

Kazzinc holds the right to mine polymetallic ores under the terms of the Contract for Subsurface Use dated 21 May 1997. The contract supersedes Licence Series MG No 59 D granted to OAO Kazzinc on 28 March 1997. As amended, the Contract is valid for 25 years from the date of the licence issue and can be extended by mutual agreement between Kazzinc and the issuing authority.

The current mining lease (7.703km^2) was issued by the Ministry of Energy and Mineral Resources of the Republic of Kazakhstan in August 2001, superseding an earlier released mining lease. The boundaries of the lease are defined by 14 corner points as detailed in Table 4.1. The legal depth limit for mining defined in the mining lease is to the elevation of 1,500m below Baltic Sea datum. The deepest mine level is currently at approximately -70m level, the deepest shaft reaches -303m level and the deepest mineralised zone extends to an elevation of -1,220m.

Table 4.1: Mining Licence Coordinates

Coordinate	Easting	Northing
1	84°14'40"	49°51'58"
2	84°14'10"	49°52'51"
3	84°15'54"	49°53'43"
4	84°16'36"	49°53'40"
5	84°17'05"	49°53'37"
6	84°17'29"	49°53'25"
7	84°18'45"	49°53'41"
8	84°18'47"	49°53'37"
9	84°17'54"	49°53'24"
10	84°17'50"	49°53'12"
11	84°17'35"	49°53'02"
12	84°16'51"	49°52'39"
13	84°15'49"	49°52'37"
14	84°15'49"	49°52'20"

WAI Comment: *The Maleevskoye mine benefits extensively from the proximity of Zyryanovsk which provides the majority of the workforce as well as being well connected by rail, road and power to the rest of Kazakhstan. In this respect, the mine is very well located.*

In addition, although the climate can be harsh, the underground nature of operations insulates against weather extremes. Also, the licence documentation has been inspected and is in order. Moreover, the conditions of the licence are sufficient for the life of mine.

4.2 Geology & Mineralisation

4.2.1 Regional Geology

The Maleevskoye deposit is situated on the western limb of the Maleeevsko-Putintsevskaya anticline between the Bukhtarmy River and the northern periclinal closure of the anticline. The Revniushinskaya Formation occurs in the core of the anticline, whilst terrigenous sediments of the Maslyanskaya, Khamirskaya and Turgusunskaya Series form the limbs.

In a broader regional context, the Maleevsko-Putintsevskaya anticline forms the northern closure of the Revniushinskaya anticlinorium, which covers an area of 250km2 and contains six other polymetallic deposits, including the now exhausted Zyryanovskiy and Grekhovskoe.

With one exception, all these deposits are hosted by Middle Devonian volcano-sedimentary rocks and closely associated with sills of quartz-plagioclase porphyry and felsite either of submarine volcanic or subvolcanic origin.

Polymetallic and copper mineralisation is found in several stratigraphic levels within the Revniushinskaya and Maslyanskaya Formations and can be considered a fairly typical VMS Exhalative type. On the western side, the Revniushinskaya anticlinorium abuts against the Schebniushinsky intrusive massif composed mostly of granite, tonalite and diorite.

Figure 4.2 shows the regional geology of the Maleevskoye area and the main stratigraphic units.

4.2.2 Local Geology

4.2.2.1 Structure

The Maleevskoye deposit comprises seven stratabound zones of lead-zinc-copper-gold-silver mineralisation: Rodnikovaya, Maleevskaya, Octyabrskaya, Holodnaya, Lugovaya, Bobrovskaya and Platovskaya.

They are found at three stratigraphic levels within a 600-700m interval of the Revniushinskaya and Maslovskaya Formations, and are confined to broad mushroom-shaped brecciated domes passing downwards into steeply dipping stockwork feeder channels, which are generally aligned along a NW-trending fractures.

The stratabound bodies generally strike 300-310°, parallel to axial traces of the domes, dip at low to moderate angles SW and plunge NW.

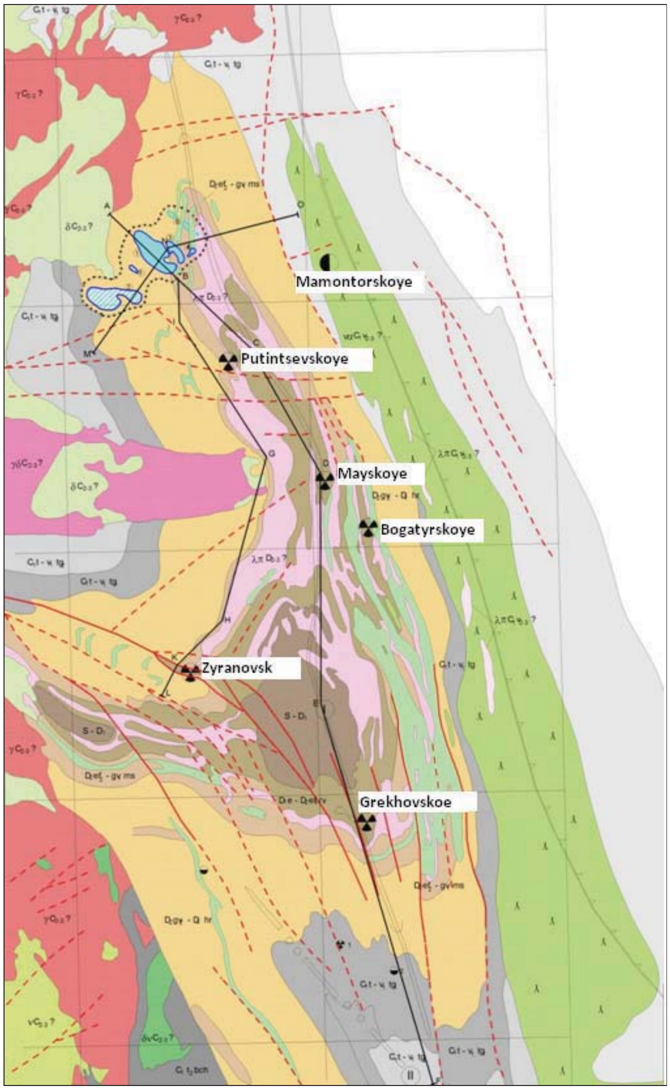
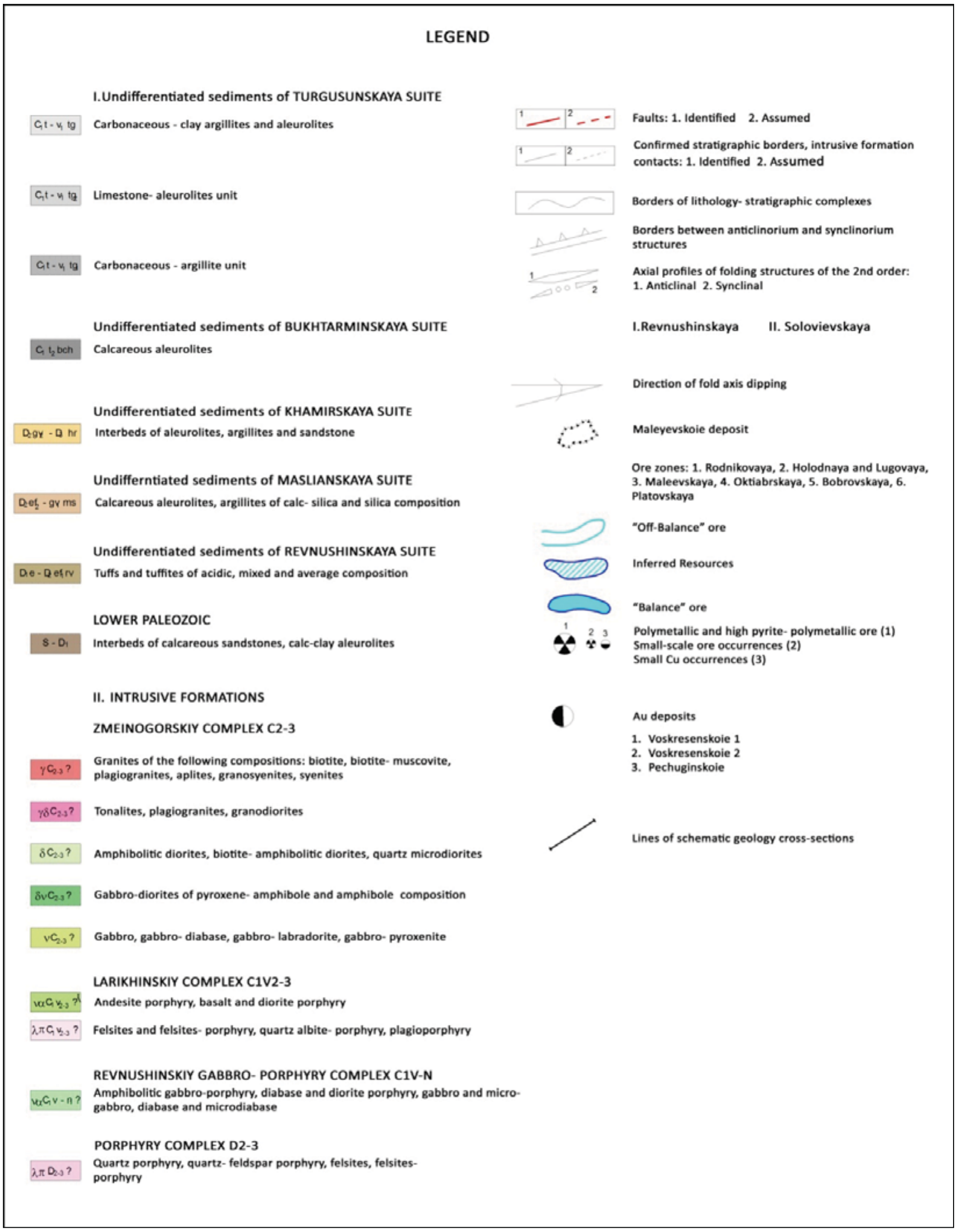


Figure 4.2: Regional Geology of the Maleevskoye Area



Legend for Figures

4.2.2.2 Mineralisation

Mineralisation is divided into polymetallic (lead-zinc with >0.6% Pb) and copper-zinc (<0.6% Pb and high Cu), the division defined by the lead content.

In general, polymetallic mineralisation, accompanied by barite, occurs on the flanks and on the hangingwall side of the lenses, whilst copper-zinc mineralisation tends to prevail in the central parts of the lenses and on the footwall side.

Soviet geologists initially thought that the mineralisation formed within thick zones of intrabed brecciation and cleavage ("traps") in pelitic-siliceous siltstones after the emplacement of the overlying rhyolite, which acted as a screen. The deposit is now regarded as syngenetic exhalative VMS mineralisation developed on the sea floor.

The largest and richest mineralised zones (Maleevskaya, Rodnikovaya, Holodnaya and Lugovaya) occur within a 150-200m thick Maleevskoye Member of the Maslovskaya Formation. They consist of massive sulphide stratabound bodies of various forms (saddles, lenses, tabular bodies) which pass downwards into veinlet-disseminated mineralisation.

Figure 4.3 shows a plan outline of the Maleevskaya Zone on 12 Level, whilst Figure 4.4 shows a section through this zone.

The upper boundary to mineralisation is generally sharp, although veinlet-disseminated mineralisation occurs locally above the massive sulphides.

The footwall boundary is generally inconspicuous as massive sulphide bodies pass into fracture filling veinlets and replacements of cement in hydrothermal breccias. This boundary has to be defined by sampling. The hangingwall is controlled by brecciated rhyolite (69%), porphyritic andesite sills and dykes (26%) and hydrothermal quartzites (5%).

Veinlet-disseminated mineralisation of the Bobrovskaya and Platovskaya zones occur at the level called Platovsky, 100-150m below the Maleevskoye level in the hinge zone of the Maleevsko-Putintsevska anticline. This type of mineralisation is characterised by poor continuity and erratic metal grades. Resource delineation requires closely spaced sampling, especially in areas where veinlet-type and disseminated mineralisation is dispersed throughout the Maslovskaya Formation between the upper and lower rhyolite bodies (e.g. on the northwestern flank of the Rodnikovaya zone and the Bobrovskaya zone).

The veinlet-disseminated mineralisation of the Octyabrskiy level occurs in mottled slates at and below the interface of the Revniushinskaya and Maslovskaya Formations. Another level of veinlet-disseminated mineralisation has been encountered in the middle of the Revniushinskaya Formation, 50-150m below the Octyabrskiy level. The mineralisation at this level occurs in quartz rhyolites and andesite-dacite porphyries and is also characterised by erratic grades and poor continuity.

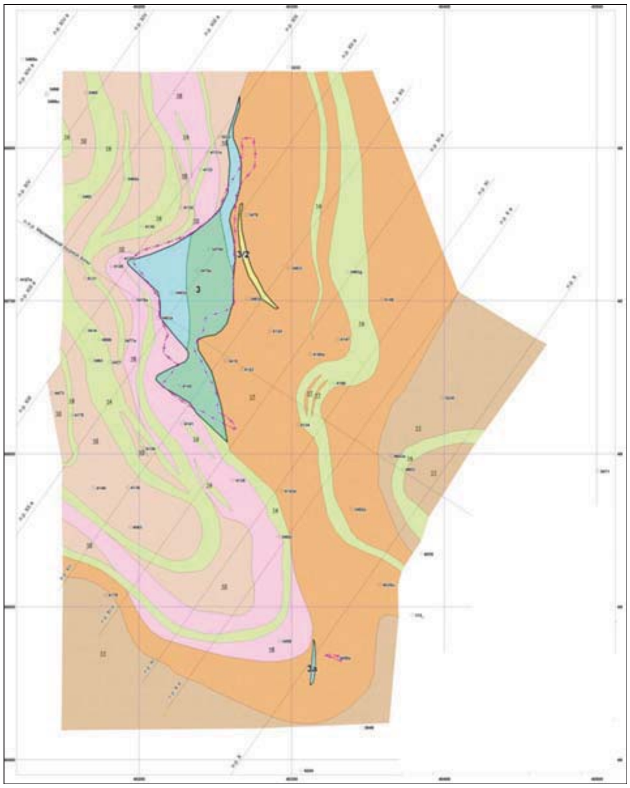


Figure 4.3: Outline of the Maleevskaya Zone (Orebody 3), 12 Level (Grid:100m)

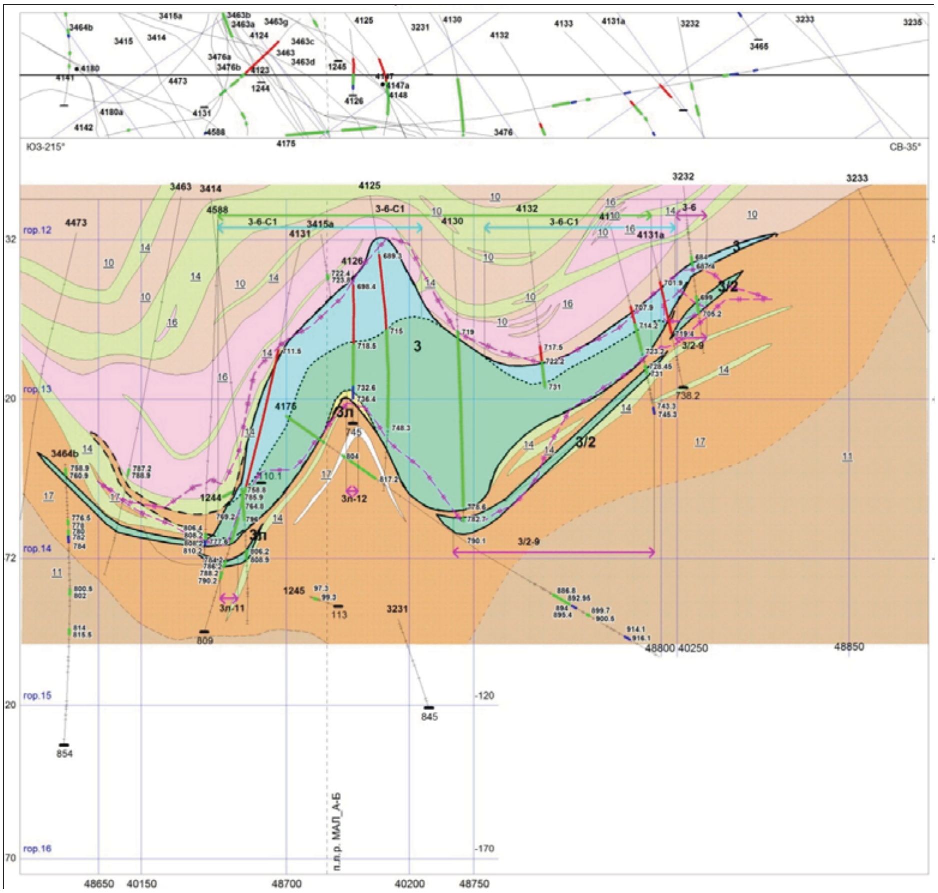


Figure 4.4: Section XIII through Maleevskaya Mineralised Zone (Orebody 3)

4.2.3 Orebodies

4.2.3.1 Maleevskaya Zone

The outcrop of this zone was mined in a small open pit and from shallow adits. The blind extension of this zone, called Novo-Maleevskaya, was discovered in 1979. A series of folded stratabound lenses of complicated geometry has subsequently been traced to a depth of 700-750m. The overall strike of the zone is 315˚ with a 32-35˚ northwestern plunge. The combined strike length is 2.3km.

The largest mineralised body in this zone, identified as No 3 Orebody, has a strike length of 1,175m and extends from Level 4 (430m) down to Level 16 (-170m). Its larger north¬western segment measures 575m in strike length with a down-dip extent of up to 250m. Dips are highly variable but generally less than 40˚SW flattening and turning NE at the north¬western end. The southeastern part is about 410m in strike length and extends down-dip for up to 195m. The average dip is 20-25˚SW.

In addition, this zone contains small satellite bodies. Mineralised body 3a is located at Levels 10 to 12 (130-32m) at the same stratigraphic level as No 3 Orebody. Mineralised lenses 3b and 3v are located at a slightly higher stratigraphic level, 90-125m from the south-eastern segment of No.3 Orebody. Lenses 3/2 (Figure 4.5 and Figure 4.6) and 3l are found in the footwall of No.3 Orebody at levels 11 to 13 and 13 to 15 respectively.

4.2.3.2 Rodnikovaya Zone

This is the main mineralized zone at the Maleevskoye deposit. It extends from Level 9 at 180m down to Level -550m along a dome structure plunging 25-30˚ northwest. The total strike length is 1,750m.

The zone contains 18 stratabound mineralised lenses with complicated forms due to folding and the presence of cross-cutting dykes. Orebodies 6 and 7, which hold the bulk of the Maleevskoye resource, represent a single predominantly massive sulphide body cut by a dyke of porphyritic dolerite and its apophyses.

Orebody 6 is situated above the steeply dipping porphyrite dyke. On average, it dips 35°WSW and plunges 27°NNW. The vertical span is 434m between absolute elevations of 318m and -116m.

Orebody 7 is situated below the dioritic porphyrite dyke. It dips 36° WSW and plunges 23° NNW. The vertical span is 383m from 213m to -170m. The combined strike length of both orebodies exceeds 1,500m, the combined true width is in the range of 120-380m and the horizontal width reaches 900m in the central part of the zone.

There is strong vertical mineral zoning. From top to bottom the main zones are:

- Thin discontinuous pyritic zone along the hangingwall;
- Barite-lead-zinc mineralisation;
- Zinc mineralisation;
- Pyrite-polymetallic mineralisation with barite;
- Pyrite-copper-zinc mineralisation;
- Copper-pyrite mineralisation, and
- Pyritic zone in the footwall (0.5-7m) passing down into disseminated pyrite.

Massive sulphides, represented by pyritic-polymetallic, copper-zinc and barite-bearing varieties are best developed in Orebody 6 (over 70% of Total volume). Disseminated ore types occur mostly in the footwall and on the flanks.

Overall, the dyke and its apophyses make up about 11% of the volume of the Rodnikovaya Zone. As dolerite apophyses are more abundant in disseminated mineralisation, they contribute about 8% of volume within copper-zinc mineralisation and only 3% of volume within predominantly massive sulphide polymetallic mineralisation. Other barren and sub-economic partings are represented mainly by weakly mineralised quartzites. On average, barren and sub-economic partings form 8.3% of the Rodnikovaya Zone.

The footwall contact is transitional. The pyritic zone passes into microquartzites with relics of bedding and disseminated syngenetic pyrite (0.001-0.05mm) forming bands and small lenses. These rocks are cut by steep feeder channels marked by brecciation with quartz-sericite-chlorite alteration and cubic pyrite (0.04-0.1mm).

In contrast, the hangingwall contact with rhyolites, siltstones and porphyritic andesites is sharp with no, or very weak alteration.

4.2.3.3 Bobrovskaya Zone

This zone is situated southwest of the Rodnikovaya Zone at a depth of 1,000-1,500m below the surface. It contains at least three lenticular bodies over a strike length of 1,750m.

4.2.3.4 Octyabrskaya Zone

This zone is situated a short distance to the northeast of the Maleevskaya Zone. In contrast to other zones, it is bowl shaped. The strike length is 600m. Mineralised bodies forming this zone dip west and northwest.

4.2.3.5 Holodnaya Zone

This is the deepest zone so far discovered at the Maleevskoye deposit. It is situated at a depth of 1,400-1,700m, 2km from the central part of the Rodnikovaya Zone. It has been followed by drilling over a strike length of 1,500m. From the north and northwest, the zone is bounded by intrusive rocks of the Schebniushinskiy massif. Its prognosticated southwest extension, called Lugovaya Zone, was confirmed in drillholes 4452a, 4503 and 4510. In 2001-2008 Kazzinc drilled 12 diamond drillholes totalling 21,579m, including 4,068m of deflections. Drillhole 7005 intercepted rich massive pyritic-polymetallic mineralisation. Four other holes intercepted lower grade veinlet-type mineralisation.

4.2.3.6 Platovskaya Zone

This zone occurs within a NNW trending subsidiary situated in the northeastern part of the deposit, 700m from the Maleevskaya Zone. It extends over a vertical distance of about 120m with widths in the range of 5.5m to 23m (6m on average). The lode plunges 20° NW.

4.2.4 Mineralogy

The main metalliferous minerals are pyrite, sphalerite, chalcopyrite and galena. Black ore (tetrahedrite-tennantite), pyrrhotite and magnetite occur in subordinate amounts. Accessories include native gold, silver, antimony, bismuth, electrum, bismuthite, molybdenite, cobaltine, pyrargyrite. The main gangue minerals are quartz, chlorite, calcite, barite, actinolite, tremolite and less common sericite, albite, epidote and biotite.

With respect to chemical composition, the Maleevskoye mineralisation is pyritic polymetallic with the average Pb:Zn:Cu ratio of 1:6:1.7.

As mentioned above, from a technological standpoint, the ore is divided into pyritic copper-zinc (<0.6% Pb) and polymetallic (≥ 0.6% Pb), with further subdivisions depending on the relative abundance of sphalerite, galena, chalcopyrite and pyrite. The lead content is the key parameter as ore with less than 0.6% does not produce a lead concentrate. The relative abundance of the two technological types and the mineralogical subtypes are given in Table 4.2.

Pyritic copper-zinc mineralisation generally prevails in the central portions of mineralised bodies.

Although the various mineralogical types alternate, there is a weak mineral zoning within the main mineralised zones expressed, from footwall to hangingwall, by the following sequence: pyritic - pyritic copper - copper-zinc – zinc - lead-zinc -polymetallic.

Table 4.2: Technological and Mineralogical Classification of Maleevskoye Mineralisation

Type	Subtype	Relative Abundance %
Polymetallic (>0.6% Pb)	Polymetallic	10
	Lead-zinc	7
	Zinc	34
Pyritic copper-zinc (<0.6% Pb)	Copper-zinc	37
	Pyritic copper	10
	Pyritic	2

Pyritic mineralisation in the footwall is highly variable in thickness (up to 7m). The pyrite content is in the range of 30-60% in disseminated varieties and up to 95% in massive mineralisation. Chalcopyrite is the main associated ore mineral (1-3%). The boundary between the pyritic subtype zone and the overlying mineralisation (generally pyritic copper) is arbitrarily set at 1% ZnEq[1].

Massive sulphides have a complex internal composition with massive, banded, lenticular and rare brecciated and mottled textures. Disseminated mineralisation, which makes up about 40% of the total volume of the mineralisation, occurs mainly at the base and on the flanks of massive sulphide bodies. It is represented by veinlet, veinlet-disseminated and disseminated textures. The matrix is formed by hybrid rocks (95.9%) represented by quartzites and silicified siltstones and porphyries (4.1%). Polymetallic and lead-zinc mineralisation often contains barite either as bands or lenses.

The main recoverable by-products are gold and silver. Gold occurs in its native form as very fine grains (5-8 microns) dispersed mainly in pyritic copper mineralisation and barite-bearing lead-zinc mineralisation. A small percentage of gold occurs as larger visible grains of up to 0.8mm across. The fineness of gold is 794.

Silver is also very fine and occurs as native silver and electrum in massive and banded barite-bearing polymetallic mineralisation and, in lesser quantities, in lead-zinc mineralisation. It is associated with pyrrhotite, barite and black ore.
Coarser electrum (20-70 microns) occurs as inclusions in pyrrhotite in pyritic copper-zinc mineralisation.

Barite-bearing polymetallic mineralisation also contains silver in freibergite and pyrargyrite.

*ZnEq[1] (zinc equivalent) is defined as 0.4*Pb + Zn + 1.4*Cu, with grades less than 0.4% Pb, less than 0.1% Zn and less than 0.1% Pb not being subject to conversion*

4.2.5 Exploration Potential

A number of other small occurrences/deposits have been identified which either have been depleted (Grekhovskoe) or are too small to be of interest.

However, the main potential appears to be down-dip from the known mineralisation towards the Holodnaya zone, although as yet, no other commercial scale mineralisation has been defined.

> ***WAI Comment:*** *The geology and mineralisation of the Maleevskoye deposit are well known through decades of exploration and nearly ten years of exploitation. It is characterised as a Volcanogenic Massive Sulphide (Exhalative Type) deposit. The orebody geometry and mineralogy are relatively straightforward and well known and efficient mining methods maximise ore extraction.*
>
> *Maleevskoye deposit is world class in terms of the base and precious metal grades it contains. Coupled with the generally large dimensions of the various ore zones which lend themselves to bulk mining techniques, Maleevskoye represents a tremendous asset to Kazzinc.*

4.3 Exploration Works & Procedures

4.3.1 Background

The Maleevskoye mine has had a long history of exploration before production finally commenced in 2000.

Surface discoveries were made in the 19th Century, but it was not until the 1950's that systematic exploration took place which has continued following a series of specified protocols through to the present day.

4.3.2 Drilling

Table 4.3 below presents data on exploration undertaken at Maleevskoye from 1950 – 2004.

Table 4.3: Historical Exploration Works

Activity	Dates	Drilling (km)	Drilling (holes)
Preliminary Reconnaissance Works	1950-1955	14.0	56
Follow-up Exploration	1967-1974	6.4	10
Continued Exploration	1979-1983	93.6	159
Reconnaissance	1981-1986	159.6	281
Detailed Reconnaissance	1983-1986	111.3	237
Additional flanks exploration	1987-1988	31.2	47
Exploration on flanks and deep horizons	1986-1992	176.1	153
Exploration on the flanks	1988-1993	124.8	121
Holodnaya Exploration	2001-2004	10.3	6
Total:		**724.3**	**1070**

The main surface drilling campaign was carried out in 1979-1986 using conventional drill rigs, including hydro-percussive rigs. Depending on depth of drilling, the drilling diameter was 76mm or 59mm.

Unusually, some of the core remains from this period, representing mineralised intersections through ore that has not yet been mined - protocols allowed for core to be discarded once that particular part of the orebody had been exploited.

Wireline drilling with the NQ size (75mm) has been used in most recent drilling programmes, including the most recent deep drilling to delineate the Holodnaya zone.

Drilling profiles at Holodnaya are aligned on an azimuth of 055˚, which appears to be perpendicular to plunge, whereas early drillholes, prior to the discovery of the Rodnikovaya zone, were sited on profiles trending 074-075˚.

The majority of surface drillholes are sited along profile lines on an azimuth of 035˚, perpendicular to the prevailing plunge direction, and drilled at various angles with or without wedging to reach expected intercepts at predetermined points. As most mineralised bodies have lenticular forms elongated along the plunge direction and highly variable strikes and dips, grid orientations vary to suit local conditions.

Most drillholes were angled as required to secure intersections at angles approximating 90˚. Many holes had up to three or wedged deflections (total of 302 wedged deflections for 103,000m). In most instances, drillholes were naturally deflecting due to complex folds typically 60-300m in wavelength and 200-300m in strike length.

Recent exploration in the developed sections of the mine from Level 3 to Level 14 has been conducted entirely by underground exploration methods, including underground diamond core drilling. A grid of 50x50m is regarded as optimal for the delineation of a C_1 resource and a grid of 30m along strike or plunge by 25m along dip for an upgrade to a B category. Closer spaced drilling is applied is areas of complex geology.

Sludge drilling combined with electrical logging to confirm the hangingwall contact had been used in addition to channel sampling prior to 1999.

Exploration below the current development and elsewhere in undeveloped areas relies on surface diamond drilling which is subcontracted to Zyryanovsk Geological-Exploration Expedition (Zyryanovsk GRE). Drilling grids vary depending on the structural complexity and grade variability. A grid of 100m by 100m is currently considered optimal for the delineation of a C_2 category resource. Grids used in the past were generally 200m along strike or plunge by 25-50m down dip for the delineation of a C_2 category resource and 100m by 25m respectively for its upgrade to the C_1 category.

4.3.3 Surveying

All underground exploratory workings, collar positions of surface and underground exploration drillholes and geophysical survey points were surveyed using instrumental methods. This is considered sufficiently accurate for resource modelling and mine planning purposes.

Downhole surveys have been conducted in all drillholes at 20m intervals and at 5-10m intervals in wedged intervals. Various instruments were used at various times. In the 1987 report, it is stated that absolute errors of zenithal angles and azimuths were 21′ and azimuth 1˚53′ respectively. Higher errors are mentioned in the report on the Holodnaya drilling (series 7000 holes). At deviations to 10˚, errors in measurements of zenithal angle and azimuth did not exceed 0.5˚ and 5˚ respectively. At deviations exceeding 10˚, errors in measurements of zenithal angle and azimuth did not exceed 1˚ and 5° respectively.

Geophysical drillhole logging, including electrical logging and roentgen-radiometric grade logging, has been routinely conducted in most surface drillholes to monitor depths of mineralised intercepts recorded in drill logs and to monitor grades.

Oriented drill core was collected from 111 holes in 1980 and compared with telephoto logging records.

4.3.4 Core Recovery

4.3.4.1 Surface Drilling

Geoincentr (2005) reported that core recoveries in excess of 80% were achieved for 61.7% of the total length of mineralised intercepts included in the resource estimate approved by GKZ RK. Core recoveries improved in surface drillholes completed after 1986.

Geoincentr (2005) also reported results of a comparison of linear core recoveries with weight core recoveries for the post-1987 drilling. No systematic bias in recording linear core recovery was observed.

Differential loss is suspected to have occurred only in pre-1986 drilling. Correction coefficients have been proposed to account for it. The correction coefficients are: 1.2 for Pb, 1.2 for Zn, 1.1 for Cu, 1.3 for gold and 1.1 for Ag.

4.3.4.2 Underground Drilling

Geoincentr (2005) reported some quality control data for the underground diamond core drilling and sludge drilling.

4.3.4.3 Underground Sampling

Horizontal channel samples were taken with a chisel and hammer from one or two walls of each underground opening with visible sulphide mineralisation. Channels are 10cm wide and 2.5cm deep.

There has been a considerable amount of work done to check the quality of channel sampling. It included duplicate sampling, larger channels, control drilling and bulk sampling. Geoincentr (2005) reported rather large relative differences, but no systematic bias.

4.3.4.4 Core Sampling

All mineralised intervals have been sampled, by splitting drill core in a mechanical splitter up to 1968, and then by sawing the core in two halves. From 1974, all drill core recovered from 59mm diameter holes was taken for sample preparation. Core from subsequent drilling programmes was sawn in half. Lengths of core samples range from 0.1m to 2m.

Geoincentr (2005) mentioned that that 5,784 core samples were included in the 1986 resource and those samples comprised 39 samples from 93mm diameter holes, 610 samples from 76mm diameter holes and 5,135 samples from 59mm diameter holes. Core sampling variance depends on the core diameter, the smaller the diameter the larger the variance.

Pulverised sample rejects were composited over intervals with similar characteristics for analyses for potential by-product elements and deleterious elements. The analyses also included the main elements. The samples were typically in the range of 300-500g each. Geoincentr (2005) reported that a statistical analysis of composite sample results confirmed that no mineralisation of potentially economic grades was overlooked.

Geochemical samples were collected from each lithological unit outside mineralised zones.

4.3.5 Sample Collection

The sample preparation scheme is based on the Richard-Czeczott formula Q=kda, where Q is the minimum sample quantity at a given stage of volume reduction, d is the diameter of the largest fragments defined as the screen size that retains the largest 5% of the mass, k is a coefficient dependent on the distribution irregularity of the mineral of interest and a is a coefficient related to the roundness of mineral grains (generally approximately 2).

The coefficient k is the key parameter. In general terms, the lower coefficient k is, the better it accounts for the erratic distribution of minerals.

Different schemes were used at different times. The k coefficient for processing drill core from surface drilling was initially 0.2, then 0.6 from 1978 and 0.2 again from February 1984.

Experimental tests confirmed that the 0.2 coefficient was appropriate. The k coefficient for processing underground drill core was initially 0.2 and then it was changed to 0.1 in 1993.

A series of tests conducted on four bulk samples representing polymetallic and copper-zinc mineralisation confirmed that the scheme based on the coefficient 0.2 was appropriate.

4.3.6 Sample Analysis

Routine analyses were performed mainly by the Zyryanovsk GRE laboratory. All samples with spectral Pb+Zn+Cu results greater than 0.5% or Zn results greater than 0.3% were analysed by the AAS method, which was introduced towards the end of 1980. Reported lower and upper detection limits are: 0.005-20% for Cu and Zn and 0.02-20% for Pb. Samples containing more than 20% Zn or more than 20% Pb were analysed by the titrometric method. A small number of drill core samples from surface exploration was analysed by the polarographic method on electron polarograph CLA-2122.

External control analyses were performed by Vostkazgeologia in Ust-Kamenogorsk. Analyses for by-product elements and deleterious elements were conducted by Vostkazgeologia and external control analyses by Yuzhkazgeologia in Almaty.

Out of a Total of 14,524 analysis made in 1980-1986, 2,016 duplicate samples (13.9%) were subjected to internal control analyses and 1,765 duplicate samples (12.2%) to external control analyses. Average relative differences of internal control samples were within acceptable limits.

However, based on external control analyses, some systematic errors were detected in specific periods, although samples from these periods represented different parts of the deposit, most of which have since been delineated in detail during underground development. On the whole, therefore these errors are not considered to have biased the average grades.

4.3.7 Current Activities

The mine currently operates three Diamec 262 rigs completing around 50,000m of development exploration drilling per year.

All the core from this drilling is crushed and sampled, as the 42mm diameter core does not allow sub-samples to be statistically valid. The results from these samples are input to the resource model to improve future mine planning and resource estimation.

For grade control, a small 5-9kg grab sample is taken after each blast. In addition, all wagons pass over a weightometer where sample analysis is done using a portable XRF, which allows classification of the ore into either polymetallic or copper-zinc (or some mixture of the two).

Continued surface exploration drilling uses a half core for analysis, the remainder are currently kept at the mine in a new core storage unit, which also has a small laboratory attached.

> ***WAI Comment:*** *The Maleevskoye deposit has been thoroughly explored and developed over a period of approximately sixty years. As such, the huge volume of exploration data available on the mine, coupled with the knowledge that these data will have been collected under strict Soviet protocols, provides strong comfort to the quality of data. Moreover, nearly ten years of production has allowed comparisons of exploration data with production information, and in general, exploration (both past and present) has proved satisfactory in delineating the sometimes complex mineralised zones.*
>
> *In particular, the use of small, versatile underground core rigs for development exploration has proved vital in efficient orebody definition.*

4.4 Mineral Resources

4.4.1 Introduction

Maleevskoye is unusual in Kazakhstan in that the most recent GKZ approved resource was estimated using contemporary techniques and software. The initial estimate was created in the Datamine software by Computing Resource Services Ltd (CRS) in late 1999. CRS then went on to train Kazzinc personnel before handing over to them in 2002/03.

After the handover, Kazzinc retained Almaty-based TOO Geoincentr and ZF TOO Geos to prepare a resource report in the format required by GKZ RK. The report (Geoincentr, 2005) contains two resource estimates:

- Geostatistical estimate in Datamine (Kriging) with about 80% of the estimated resource checked using a locally-developed GST programme; and
- Conventional estimate using the method of parallel vertical cross sections in the developed part of the deposit and the method of geological blocks in the undeveloped part.

The estimates were accepted and approved by GKZ RK in February 2005 (Protocol No 379-05-U dated 11 February 2005).

4.4.2 Resource Conditions

The currently valid resource conditions were approved by GKZ RK on 22 July 2004 (Memorandum No 327-04-K).

The approved conditions require a resource to be estimated and reported for two types of mineralisation, namely polymetallic mineralisation (with the lead grade equal to or more than 0.6% Pb) and copper-zinc mineralisation (with the mean lead grade less than 0.6%).

For the developed part of the Maleevskoye deposit, the resource conditions are as follows:

- Sample cut-off grade to define mineralised intercept intervals - 1.0% ZnEq, where ZnEq = 0.4*Pb + Zn + 1.4*Cu, with grades less than 0.4 % Pb, less than 0.1% Zn and less than 0.1% Pb not being subject to conversion.
- Minimum grade in estimation block - 3.3% ZnEq.
- No top-cut applied.
- Minimum thickness – 2m.
- Maximum thickness of barren and below cut-off inclusions – 4m.
- Cut-off grade for delineation of off-balance resource – 0.8% ZnEq.

Grades of lead, zinc, copper, gold, silver, cadmium, bismuth, indium, selenium, tellurium, arsenic, antimony, total sulphur, pyritic sulphur and barite should be estimated within mineralisation shells. Blocks with mean grades of less than 3.3% ZnEq should be classified as an off-balance resource. Estimates of pyritic sulphur and total sulphur should be reported in the off-balance category.

Correction coefficients should be applied to mean block grades estimated from drilling results reported prior to 1987. The correction coefficients are: 1.2 for Pb, 1.2 for Zn, 1.1 for Cu, 1.3 for gold and 1.1 for Ag.

4.4.3 2005 GKZ Approved Estimate

Details of the 2005 Balanced Ore Conventional Reserve calculation are given in Table 4.4 below.

Table 4.4: 2005 Conventional Reserve Estimate

	Reserves						**Average content**				
	Ore '000t	Pb '000t	Zn '000t	Cu '000t	Au kg	Ag t	Pb %	Zn %	Cu %	Au g/t	Ag g/t
Total	39400.5	548.3	3415.3	1012.4	28627.0	3233.3	1.39	8.67	2.57	0.73	82.06
Made up of:											
Polymetallic	19294.6	491.7	2643.2	379.2	16851.6	2427.4	2.55	13.70	1.97	0.87	125.81
Copper-zinc	20105.8	56.6	772.1	633.2	11775.4	805.9	0.28	3.84	3.15	0.59	40.08

In comparison, detailed below is the 2005 Geostatistical Reserve Estimate which has been approved by GKZ (Table 4.5).

Table 4.5: Geostatistical GKZ-Approved Resources

	Reserves						Average Content				
	Ore '000t	Pb '000t	Zn '000t	Cu '000t	Au kg	Ag t	Pb %	Zn %	Cu %	Au g/t	Ag g/t
Total	42703.5	641.9	3499.6	1017.4	33139.9	3478.4	1.50	8.20	2.38	0.78	81.45
Made up of:											
Polymetallic	28084.6	592.00	2894.8	624.8	24088.9	2631.9	2.11	10.31	2.22	0.86	93.71
Copper-zinc	14618.9	49.9	604.9	392.5	9051.0	846.5	0.34	4.14	2.69	0.62	57.91

The approved resource estimate has also been classified according to the various parts of the deposit, the results of which are shown in Table 4.6.

Table 4.6: Classified Geostatistical Approved Resource				
	Ore Reserves	Metal content		
		Lead	Zinc	Copper
	t	%	%	%
Category C_1				
Reserves of Developed part of Mine				
Total	23,399,055	1.35	8.56	2.76
Reserves of Undeveloped part of Mine				
Reserves of Rodnikovaya ore zone				
Total	8,852,260	1.41	7.43	2.00
Reserves of Maleevskaya ore zone				
Total	6,276,000	1.36	8.02	2.39
Total Category C_1				
Total	38,527,215	1.37	8.21	2.53
Category C_2				
Reserves of Undeveloped part of Mine				
Reserves of Rodnikovaya ore zone				
Total	2,101,500	3.36	8.17	1.38
Reserves of Maleevskaya ore zone				
Total	113,800	3.00	10.21	0.79
Reserves of Oktyabrskaya ore zone				
Total	1,909,900	2.13	8.02	0.73
Total C_2 category				
Total	4,125,200	2.78	8.16	1.06
Total C_1+C_2 categories				
Total	42,652,415	1.50	8.20	2.39

WAI Comment: *It is very encouraging that GKZ has allowed the use of a geostatistical resource estimate to become part of the State balance, albeit still classified under the Russian system. Moreover, the closeness of the geostatistical and conventional estimates provides extra comfort as to the robustness of the resource. Overall, the Maleevskoye resource base is sound and can be relied upon for future mine scheduling and development.*

4.5 WAI 2010 Resource Estimate

4.5.1 Introduction – Previous Resource Estimates

The mineral resource model presented and evaluated here for the Maleevskoye polymetallic deposit has been supplied by Kazzinc and created by the methods laid out below in accordance with the guidelines of the JORC Code (2004). The effective date of the resource estimate is 01 January 2011.

4.5.2 Database Compilation

The sample database was supplied by Kazzinc in Datamine Format separated by mineralised zone and orebody. Checks were made for overlapping or duplicate samples and the database was found to be in good order. Orebody 3 in the Maleevskaya Zone and orebodies 6 and 7 in the Rodnikovaya Zone are the most extensively sampled. Table 4.7 below lists the number of samples by orebody. As different sampling intervals were used for the different elements, the lengths and number of samples reported in Table 4.7 are for Pb/Zn/Cu analysis.

4.5.3 Domaining

Mineralised zone wireframes have been created within Datamine to a sample cut-off grade of 1.0% Zinc Equivalent (ZnEq) where ZnEq= 0.4*Pb + Zn + 1.4*Cu where grades of less than 0.4% Pb, 0.1% Zn and 0.1% Cu were not converted. The minimum thickness of mineralised intercept is 2m and the maximum included thickness of barren or below cut-off grade samples is 4m. As examples, Figure 4.5 and Figure 4.6 below are isometric views of the Maleevskaya Zone orebodies and ore body 6 and orebody 7 from the Rodnikovaya Zone.

Table 4.7: Sample Database					
Orebody	Number of Samples	Minimum Length	Maximum Length	Mean Length	Total Length
Maleevskaya Zone					
3	4126	0.10	4.30	1.57	6497.77
3b	29	0.40	2.10	1.45	42.00
Octyabrskaya Zone					
18	89	0.50	2.80	1.72	153.10
19	131	0.20	2.00	1.41	184.60
Rodnikovaya Zone					
5	464	0.10	2.50	1.61	747.15
6	12728	0.10	16.00	1.64	20864.33
7	15365	0.10	26.00	1.61	24804.30
7_4	463	0.20	2.20	1.64	758.80
9	51	0.20	2.00	1.02	52.22
10	272	0.10	5.70	1.19	324.64
10a	37	0.20	2.10	1.22	45.20
10b	6	0.65	1.50	0.90	5.40
11	17	0.10	2.00	0.83	14.05
13	7	0.40	2.00	1.07	7.50
14	49	0.20	2.05	1.08	53.00
14a	6	0.35	2.05	0.68	4.10
72	15	0.20	1.50	0.85	12.70
7a	130	0.20	2.00	0.99	128.85

Figure 4.5: Isometric View of Maleevskaya Zone Orebodies 3 and 3b looking approximately SE

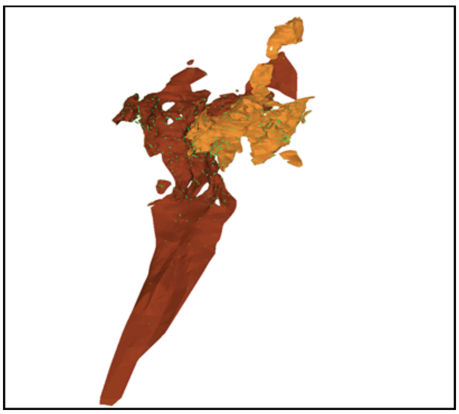


Figure 4.6: Isometric View Rodnikovaya Zone Orebodies 6 and 7 looking approximately SE

4.5.4 Global Geostatistical Analysis

The database is very robust and contains a significant number of samples for the larger orebodies. Statistical analysis indicates roughly log-normal populations of grades for each individual orebody with no significant bias present. As examples, Figure 4.7 shows the log-probability plots for orebody 3 Pb, Zn, Cu grades. Figure 4.8 shows the log-probability plot of Au within Orebody 7 and Figure 4.9 is the log-probability plot of Ag also within orebody 7.

Different sample intervals have been used for Pb/Zn/Cu, Au/Ag and Ba/S analysis. For this reason different composite lengths have been selected. A composite length of 3m has been selected for Pb/Zn/Cu assays, a composite length of 5m has been selected for Au/Ag analysis and a composite length of 4.5m has been selected for S/Ba assays.

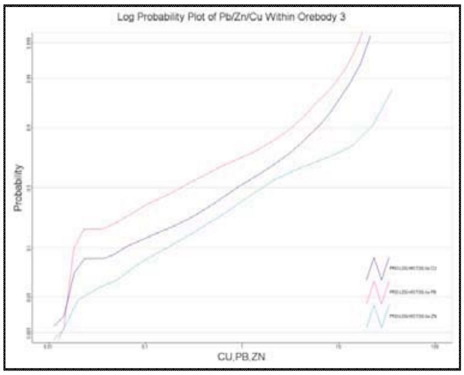


Figure 4.7: Log Probability Plot of Pb/Zn/Cu within Orebody 3 Maleevskaya Zone

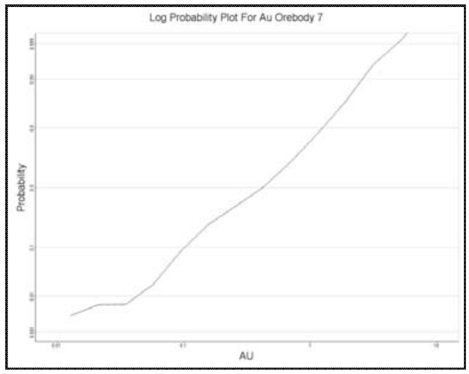


Figure 4.8: Log Probability Plot for Au in Orebody 7 Rodnikovaya Zone

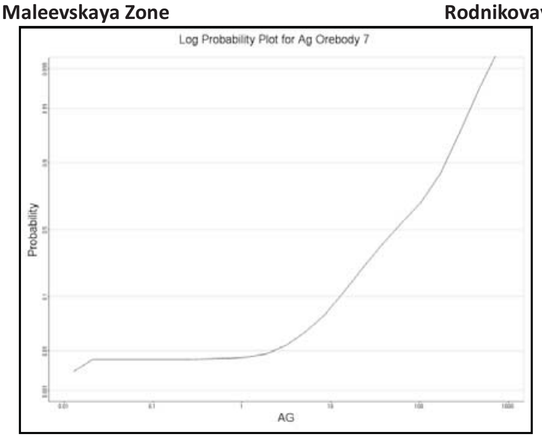


Figure 4.9: Log probability Plot of Ag within Orebody 7 Rodnikovaya Zone

WAI Comment: *WAI considers the composite lengths chosen to be appropriate with regard to the mean and the spread of the sample lengths.*

4.5.5 Variography

The initial estimate for the Maleevskoye deposit was created using Datamine software by Computing Resources Limited (CRS) in 1999. CRS continued to update the model and train Kazzinc personnel until the model was handed over to Kazzinc in 2002/2003. Part of this process was a variographic study which has been updated as required. Directional semi-variograms for the along strike, down dip and across strike directions were generated for Pb, Zn, Cu, Ag, Au, Ba and pyritic Sulphur. Examples of some of these experimental variograms and their associated models are shown below in
Figure 4.10 and Figure 4.11.

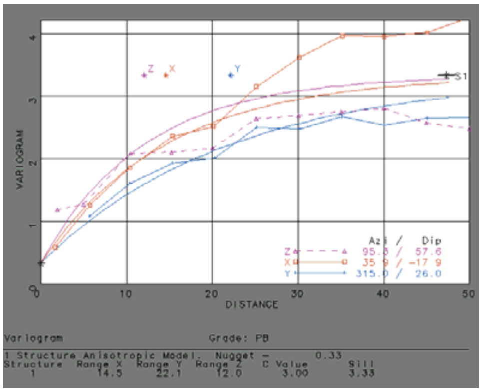


Figure 4.10: Variogram Model for Pb in Orebody 6

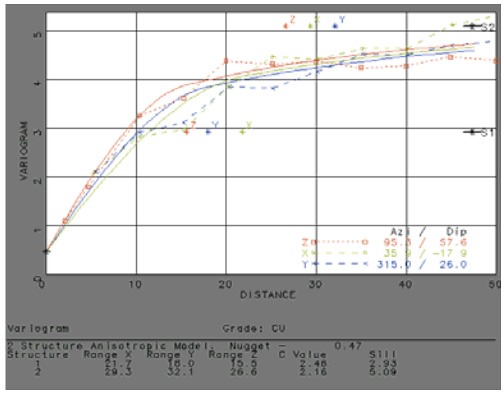


Figure 4.11: Variogram Model for Cu in Orebody 7

4.5.6 Block Modelling

Block models were created within each of the mineralised zone wireframes. A parent cell size of 10x10x10m was selected. Sub cell splitting to a minimum block size of 2.5x2.5m in the X strike and Y directions has been used with precise fitting to wireframe boundaries in the Z direction where additional cell resolution is required.

4.5.7 Density and Zinc Equivalent

Bulk density has been determined on approximately 15,000 samples (mostly drill core). This included 4,135 samples from the Maleevskaya Zone and 5,939 samples from the Rodnikovaya Zone. Regression formulae have been developed for each of the zones and these were used post grade estimation to calculate density in the block models. These regression formulae are listed below in Table 4.8. A zinc equivalent grade (ZnEq) is calculated in to the block models from the estimated grades using the formula ZnEq= (0.4 x Pb) + Zn + (1.4 x Cu).

Table 4.8: Density Regression Formulas

Zone	Density Regression Formulae
Maleevskaya	2.7351+0.101*(PB+0.8*ZN+CU)-0.0019*(PB+0.8*ZN+CU)
Octyabrskaya	*2.773 + (0.0403*Cu) + (0.0134*Pb) + (0.0365 Zn)*
Rodnikovaya	*For samples with analysis for Cu, Pb, Zn, pyritic S and Ba* =100/(36.9-(0.303*Pb+0.186*Zn+0.395*Cu+0.31*S+0.262*Ba) For samples with <8.39% ZnEq (see below) =2.8306+0.1789*(0.9Pb+1.0Zn+1.0*Cu) For samples with >=8.39% ZnEq =4.3397-0.0009*(0.9*Pb+1.0*Zn+1.0*Cu)

4.5.8 Grade Estimation

Grade estimation was carried out using Ordinary Kriging (OK) as the principal interpolation method with Inverse Power of Distance (IPD) used for comparative purposes. For orebodies 3b in the Maleevskaya Zone, 18 and 19 in the Octyabrskaya Zone and orebody 5 in the Rodnikovaya Zone, IPD was used as the principal interpolation method.

The OK method used estimation parameters defined by the variography. The estimation was performed only on mineralised material defined by each ore zone and only drillholes contained within each ore zone were used in the grade estimation (i.e. no drillholes from adjacent ore zones have been used).

Where used, the OK estimation was run in a three pass Kriging plan, the second and third passes using progressively larger search radii to enable the estimation of blocks unestimated on the previous pass. The search parameters were derived from the variographic analysis, with the first search distances corresponding to the variogram range where a spherical mode was used, the second search distance twice the variogram range and the third search distance three times the variogram range. Where exponential models were used the search ellipses were limited to one third of the variogram model range.

Sample weighting during grade estimation was determined by variogram model parameters for the OK method. Block discretisation was set to 3x3x3 to estimate block grades. Grades were estimated in to each individual subcell.

4.5.9 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 interpolated model grades. An example Swath plot from this process is shown below in Figure 4.12 for Pb grades within orebody 6.

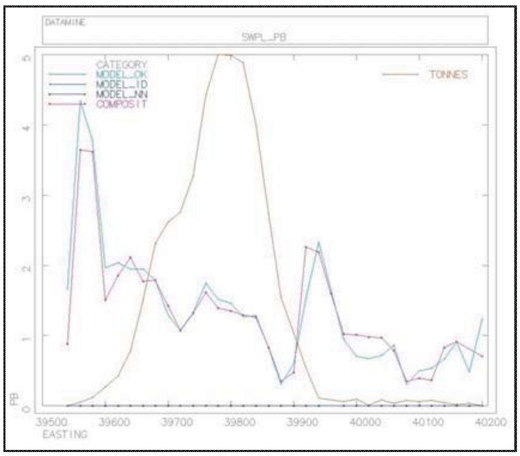


Figure 4.12: Swath Plot for Orebody 6 Pb Grades

4.5.10 Depletion

The block models were coded based on existing stopes and other underground development. The effective date of the resource assessment is 01 July 2010.

4.5.11 Resource Classification

WAI has classified the Maleevskoye deposit in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves JORC Code (2004).

Key drillhole spacings for the allocation of resources by area can be summarised as follows:

- ***Measured*** resources - 20m x 25m (down dip and along strike) with underground development within this area;
- ***Indicated*** resources – 40m x 50m (down dip and along strike); and
- ***Inferred*** resources – 80m x 100m (down dip and along strike);

An example vertical section is shown in Figure 4.13 for orebody 6, demonstrating the resultant resource class categories which have been set into the block model. An example isometric view of the resource classification for orebody 6 is shown in Figure 4.14.

4.5.12 Resource Evaluation

The resource classification for the Maleevskoye deposit is classified in accordance with the guidelines of the JORC Code (2004).

The final block models were used as the basis for resource evaluation. The resource is reported to different types of mineralisation; polymetallic mineralisation (PbZn) with a Pb grade more than or equal to 0.6% Pb and copper-zinc mineralisation (CuZn) with a mean Pb grade of less than 0.6% Pb.

Summary results of the evaluation of the unmined, in-situ resources for the whole of the Maleevskoye deposit are shown below in Table 4.9.

> ***WAI Comment:*** *The resource modelling process at Maleevskoye is, in WAI's opinion, robust and accurately reflects the remaining resources both in the developed part of the mine, and in those other undeveloped areas.*
>
> *As the parameters set for the Zinc Equivalent cut-off were set in 2004, it is likely that a late-2010 economic cut-off might be a little lower which would have an effect of increasing the resource tonnage, but probably decreasing the grade. However, WAI does not believe that this change would be significant and moreover would have minimal effect on the overall life-of-mine schedule.*

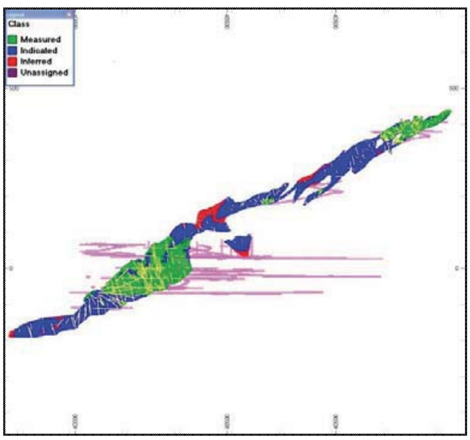


Figure 4.13: Orebody 3 NW-SE Vertical Section Showing JORC Classification

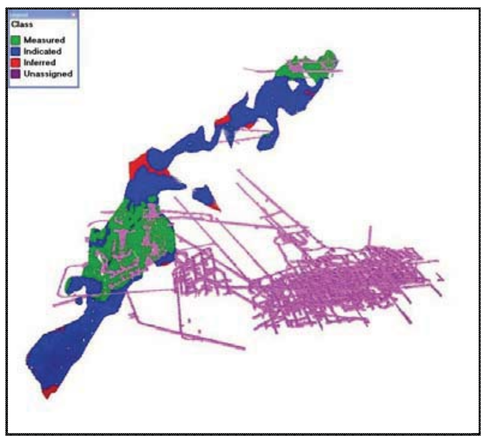


Figure 4.14: Orebody 3 Isometric View from North Showing JORC Classification

Table 4.9: Maleevskoye Resource Estimate - For All Orebodies
(WAI 01.01.2011)
(In Accordance with the guidelines of the JORC Code (2004))

Classification	Ore Type	Tonnage (Mt)	Density	Pb		Zn		Cu		Au		Ag		Ba		Sp		EQV
				%	t	%	t	%	t	g/t	oz	g/t	oz	%	t	%	t	%
Measured	CuZn	5.16	3.57	0.31	16,118	3.17	163,709	2.37	122,268	0.46	76,838	45.78	7,592,815	2.34	12,074	20.83	107,488	6.62
	PbZn	7.77	3.81	1.68	130,483	9.41	730,724	2.39	185,584	0.73	181,078	98.97	24,707,132	5.19	40,308	18.06	140,209	13.44
	Total	12.92	3.71	1.13	146,601	6.92	894,433	2.39	307,852	0.62	257,917	77.74	32,299,948	4.05	52,382	19.17	247,696	10.71
Indicated	CuZn	3.68	3.22	0.35	12,826	3.28	120,504	1.65	60,638	0.39	45,517	41.84	4,943,748	2.30	8,438	11.46	42,095	5.71
	PbZn	7.36	3.62	1.55	114,356	8.59	631,546	2.10	154,476	0.65	154,837	76.29	18,043,611	3.78	27,792	11.49	84,529	12.15
	Total	11.03	3.49	1.15	127,182	6.82	752,050	1.95	215,114	0.56	200,354	64.82	22,987,359	3.28	36,231	11.48	126,625	10.00
Measured + Indicated	CuZn	8.83	3.43	0.33	28,944	3.22	284,213	2.07	182,906	0.43	122,355	44.14	12,536,563	2.32	20,512	16.93	149,583	6.24
	PbZn	15.12	3.72	1.62	244,838	9.01	1,362,270	2.25	340,060	0.69	335,915	87.94	42,750,744	4.50	68,101	14.86	224,738	12.81
	Total	23.96	3.61	1.14	273,783	6.87	1,646,483	2.18	522,966	0.60	458,271	71.79	55,287,307	3.70	88,613	15.63	374,321	10.39
Inferred	CuZn	1.77	3.06	0.29	5,047	2.80	49,408	1.22	21,533	0.18	10,145	24.28	1,377,707	1.15	2,038	5.80	10,234	4.55
	PbZn	3.11	3.21	2.31	71,746	6.23	193,725	0.84	26,099	0.29	28,611	60.67	6,061,129	2.75	8,538	5.83	18,101	8.55
	Total	4.87	3.16	1.58	76,794	4.99	243,133	0.97	47,632	0.25	38,756	47.49	7,438,836	2.17	10,576	5.82	28,334	7.10

4.6 Mining

4.6.1 Introduction

Maleevskoye is the largest underground mine in the Kazzinc group in terms of ore production. Operations at Maleevskoye started in 2000 at a rate of 1.5Mtpa, with full scale production of 2.25Mtpa being reached in 2002. Two ore types are mined at Maleevskoye, copper-zinc ore and polymetallic ore. Both products are transported by road to the Zyryanovsky Mining and Concentrating Complex ("ZGOK") located some 25km to the south of the mine site. The mine utilises modern high capacity trackless mining methods combined with a traditional tracked haulage system. The main mining methods in operation are sub-level caving, which accounts for approximately 5% of production, and sub-level open stoping with backfill, which accounts for the remaining 95% of production.

4.6.2 Mining Methods

Two main mining methods are employed at Maleevskoye, being:

- Block Caving (5%); and
- Sub-level Open Stoping ("SLOS") with Backfill (95%).

Both mining methods utilise mechanised development with electric-hydraulic face-jumbos and diesel LHDs for drilling and ore extraction. Ore and waste is transported either directly to ore/waste passes using LHDs or loaded into trucks and transported to a central ore/waste pass. Both sub-level caving and SLOS mining methods use electric-hydraulic production jumbos for blast hole drilling

The sub-level caving method is essentially open stope mining but the hangingwall is allowed to cave into the stope void after the ore has been extracted. Blast holes drilled from hangingwall development drives are used to initiate and control the caving. Stopes are orientated longitudinally along the strike of the lode and are extracted in a top-down sequence. The caving method is only used between Level 2 and Level 7 (inclusive) of the Maleevskiy lode. The minimum mining width is approximately 5m; maximum width is approximately 20m and sub-level spacing is typically 25-50m.

SLOS mining with backfill is a variation of Open Stope Mining with the final void being backfilled with cemented classified tailings, typically with a final strength of 1.5-2.5MPa. Extraction takes place from multiple sub-levels located vertically between main mine levels.

The SLOS stopes are transversely orientated across the orebody strike. The method employs up-hole drilling and stope extraction usually takes place as a series of four stopes mined along strike (20m width each) with each stope containing several sub-level stopes with a vertical spacing of 12.5m between sub-levels. The first stope in each block is always mined top-down over several sub-levels and opened up as one large open stope and subsequently backfilled with cemented fill. The second and third stopes in the sequence are mined bottom-up and always adjacent to a stope side wall fill exposure. The last (4th) stope in the block is the closest to the access point and is essentially a pillar recovery stope. It is mined bottom-up as per stages 2 and 3.

The SLOS mine method is employed from Level 11 down to Level 16 (inclusive) in both the Maleevskaya and Rodnikovaya Zones and represents approximately 95% of current and future production.

For mining blocks that span more than four sub-levels, the lower (down-dip) series of stopes are mined under the backfill. When this occurs (which is not often), the floor of the overlying stope incorporates tank-buster style steel frames embedded in higher strength cemented backfill to ensure safe working conditions in the underlying stope.

In areas where the ore does not extend to an extraction level, it is mined with an intermediate sub-level to reduce waste rock dilution. The majority of stopes employ up-hole drilling but some down-hole drilling is employed for resources that don't extend the full extent between sub-levels. All stopes have a uniform 20m strike width.

The stope extraction sequence is 1 to 3 (right to left) followed by the 4th, on the right hand side of 1 (see Figure 4.15). The extraction sequence within the primary stope is top-down; for example 13L+37m, 13L+25m, 13L+12m, 13L and then all sub-levels are backfilled. The extraction sequence within the 2nd 3rd and 4th stopes is bottom-up; for example 13L+12m, 13L followed by backfilling of these two sub-levels, then 13L+37m, 13L+25m followed by backfilling of these two sub-levels. Adjacent stopes on the same sub-level have a strike step stagger, for example Stope 2 – 13L+37m sub-level and Stope 2 13L+25m sub-level are mined, backfilled and allowed to cure for 90 days before production starts in the adjacent Stope 3 – 13L+12m sub-level.

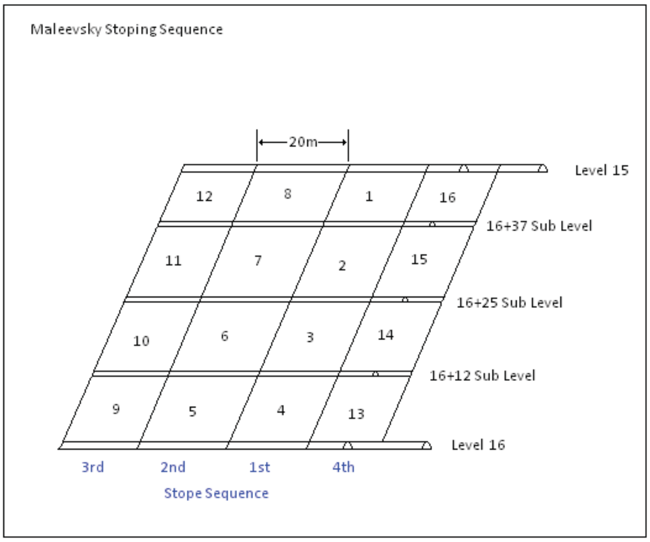


Figure 4.15: SLOS Stope Extraction Sequence at Maleevskoye Mine

Where the orebody thickness (from hangingwall to footwall) is greater than 50-60m, the stope may be split into two transverse stopes as well as 4 sub-levels. Again, a stepped extraction sequence is employed between transverse stopes.

Typically each stope can take 9-12 months to complete from development through to backfilling. Approximately 10 to 11 stopes are active at any time to achieve the 2.25Mtpa production rate.

Ore transportation from the stopes to the ore and waste passes is via loaders (LHDs) and trucks

Ground conditions are very good underground with little or no support required. Kazzinc reports very few in-situ or mining induced stress problems and none were viewed during WAI's site visit.

4.6.3 Dilution, Recoveries and Losses

Dilution factors are applied to the Mineral Resource depending on the mining method to be employed in extracting the resource. Total dilutions (primary and secondary) typically vary between 10 – 12% (mine dilution) with the overall average being 11%. Dilutions are typically higher for the 4th stage pillar recovery stopes.

The dilution factors applied assume zero grade for material classified as waste even if it may contain some mineralisation. Off balance material that is included within stope boundaries is assigned its respective grade for mining and planning purposes.

Exploitation losses (under-break, mucking losses, spillages, misallocation etc) are applied to the Mineral Resource depending on the mining method to be employed when extracting the resource. Total losses (primary and secondary) typically vary between 4-8% with the overall average for stope extraction method being 4%.

4.6.4 Mining Equipment

The following main mining equipment fleet is employed at Maleevskoye mine:

- 3 x Atlas Copco Boomer – Twin boom electric-hydraulic face drilling jumbos for mine development;
- 7 x Caterpillar R1700G LHDs for ore extraction and haulage;
- 4 x Sandvik Solo 7 electric-hydraulic production drilling jumbos;
- 1 x Kiruna K635E – 35t electric haul truck; and
- 2 x Caterpillar 740 – 40t diesel haul trucks.

The mine is equipped with an underground workshop for maintaining all underground mining equipment. All equipment is owner operated and maintained.

4.6.5 Backfill

Backfill for the SLOS mining operations at Maleevskoye is produced in an on-site backfill plant. The principal ingredients in the fill are:

- Tailings from ZGOK – 1,140kg/m^3 (60%);
- Granulated smelter slag from Karaganda – 240kg/m^3 (12.6%);
- Smelter slag from Ust-Kamenogorsk – 260kg/m^3 (13.7%);
- Cement – 60kg/m^3 (3.2%); and
- Water – 200kg/m^3 (10.5%).

The combined bulk density of the backfill is 1.9t/m^3 with 3.2% cement content.

The tailings are excavated from the ZGOK tailings dam located 14km south of the Maleevskoye mine and transported as a return load by the trucks used to transport ore from the mine to the metallurgical plant.

The smelter slags from Ust-Kamenogorsk and Karaganda are transported to Zyryanovsk by rail and then to the mine site by truck.

The backfill plant consists of two ball mills in series which mill the tailings and smelter slags down to the correct particle size before the cement is added. All ingredients are measured and added by their dry weight and then water is added to make up to the prescribed pulp density.

The fill is transported underground by gravity through a borehole within the backfill plant. Underground there is a distribution system comprising of steel pipes and valves to direct the backfill to the required stope for filling.

The backfill plant is fully automated and control by a SCADA system located in a central control room.

4.6.6 Ore and Waste Extraction

The majority of the ore production at Maleevskoye is produced between Levels 11 and 14 (1,200ktpa) and Levels 4 to 7 (550ktpa). The remaining 500ktpa is produced between Levels 14 and 16.

The main haulage level is located on Level 14. All ore produced above level 14 is dropped down a series of ore passes to Level 14 where it is collected and loaded into rail wagons. The ore produced on Level 14 and below is transported up a dedicated haulage ramp, using a Kiruna electric haul truck, to a central ore pass located just above Level 14 on the 14+25m sub-level. From here it drops down to Level 14 and loaded into wagons.

The ore and waste is transported by rail along Level 14 to an automated sampling point where the ore type is determined. The wagons are analysed and then proceed to a wagon tipping point which discharges into a series of storage bins depending on whether it is polymetallic ore or copper-zinc ore. The ore is then drawn from the various storage bins, crushed and hoisted to surface via Skipovaya Shaft, which is equipped with an automated twin skip hoisting system.

Waste rock is sent to Maleevskaya Shaft where it is hoisted to surface in a single skip hoist without being crushed.

All men and material enter the mine through Vozduzhovdyushtaya (Exhaust) Shaft, which is equipped with a man riding cage. Maleevskaya Shaft also serves as a second means of egress.

4.6.7 Dewatering System

The main pumping station at the mine is located in close proximity to Maleevskaya Shaft. Mine water is collected in a series of sumps close to the bottom of the shaft from where it is pumped 660m to surface using 5 pumps (one on stand-by) each with a capacity of $300m^3$/hr.

4.6.8 Ventilation

The mine utilises forced ventilation. Fresh air enters the mine via Ventilatsioniya Shaft and Maleevskaya Shaft using VOD 30 ($180m^3$/m) and VOD 40 ($300m^3$/m) main ventilation fans, respectively. The total air intake is $480m^3$/m and is heated when required.

The main exhaust is Vozduzhovdyushtaya (Exhaust) Shaft, which is equipped with a VCD 47.5 ventilation fan extracting $500m^3$/hr.

Two new ventilation shafts are currently being constructed by raise boring and are due for completion in early 2012. This will increase the fresh air intake capacity by $150m^3$/m. The airflow in the existing Exhaust Shaft can be increased to match.

4.6.9 Conclusions

The Maleevskoye Mine is a modern underground mine with high production capacity, which utilises modern mining methods and equipment. The mine employs many automated systems and has been developed to a very high standard.

The mine is also operated to a very high standard and the workforce appears to be qualified and skilled providing stable production rates.

Despite the fact that the mine is located 25km from the processing facility, the modern high productivity mining methods allow a good overall mining cost (approximately US$18/t of ore mined). The reserves appear sufficient to maintain operations at the present level until 2015, but after that production rate drops as a result of fewer faces being available.

4.7 Ore Reserves

4.7.1 Introduction

WAI has carried out a mine design, and produced Ore Reserves for the Maleevskoye deposit in accordance with the guidelines of the JORC Code (2004), based upon the most recent Mineral Resource Block Model (WAI 2010). WAI has used GijimaAST Mine2-4D® software to prepare the mine design, and EPS® software to produce Ore Reserves. The objective of the mine design is the best utilisation of the economic resources with a safe and productive mining method, making maximum use of the existing infrastructure.

4.7.2 Mine 2-4D and EPS Software

The mine design for the Maleevskoye deposit was carried out using Mine2-4D software, which is an automated mining software package developed in Australia and currently marketed by GijimaAST. It allows the user to accurately design mine excavations such as development and stoping, and then allows the operator to apply time-dependent mining activities such as backfilling and cable bolting in a fully three-dimensional graphical environment. Following the design of excavations and associated activities (i.e. bolting or backfilling), mining activities can be sequenced, with time delays built into the sequence where appropriate. Once the design is complete and the associated activities applied, the mine design is exported into the Enhanced Production Scheduler (EPS) which is a direct representation of the 3D mine design in a Gantt chart format. The sequence and delays built into the model in Mine2-4D are also exported to EPS.

EPS allows the user to input specific production rates for individual activities and manipulate, in detail, the various mining activities by applying mining resources such as drilling crews, mining equipment and personnel to tasks. A mining schedule and production report including mined tonnages, grade, and development metres over the life of the mine or for specific time periods can then be produced.

4.7.3 Mining Parameters

The stope blocks for Maleevskoye have been designed to a minimal average block grade of 4% Zn, or 4% Zinc Equivalent (ZnEQV).

4.7.4 Mine Layout

The Maleevskoye mine has been laid out following the proposed mining methods laid out in Section 4.6.

4.7.5 Mine Development

The principal mining levels at Maleevskoye currently incorporate ramps, main mining levels, crosscuts and repair/storage areas, at 50m vertical intervals, with 12-13m sub-levels, extending to the -175m level.

WAI has carried out a mine design by utilising existing development to access proposed new stoping areas on the +400m to -175m levels, and designed extensions to the existing development to provide access to new mining horizons to the -275m levels.

4.7.6 Stoping

All proposed new stopes for the Maleevskoye deposit have been designed as sub-level chamber stopes with backfill as outlined in the Mining chapter of this document, above.

> ***WAI Comment:*** *WAI considers that the stoping methods outlined are well suited to the deposit and in addition will not require significant investment in new equipment or training of personnel.*

Figure 4.16 shows the proposed stope layout for Maleevskoye.

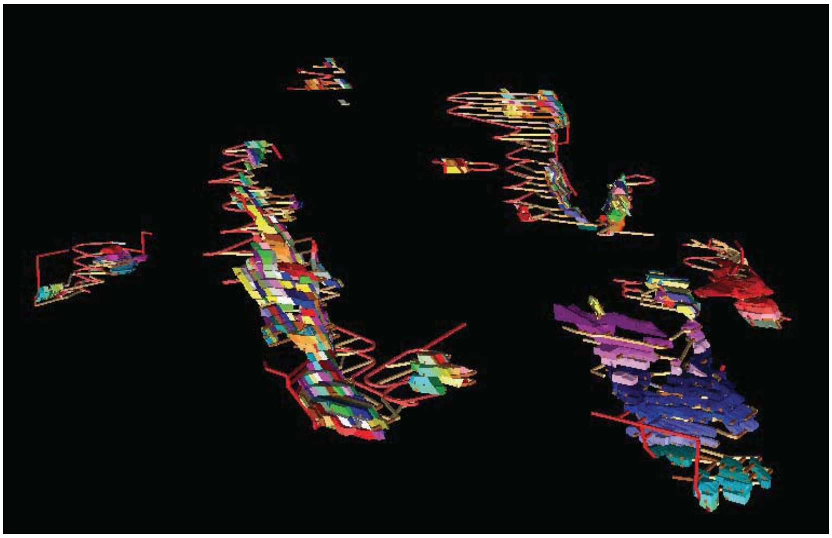


Figure 4.16: Proposed Stopes with Development

4.7.7 Losses and Dilution

During the mine design process, losses (4-8%) and dilution (10-12%) were applied to the mined tonnages at Maleevskoye.

4.7.8 Mining Schedule

The production schedule for Maleevskoye is shown in Table 4.10 below.

> ***WAI Comment**: It should be noted that the production schedule includes 171,010t of Inferred material (at 5.14% Zn, 1.31% Cu, 0.93% Pb, 0.30g/t Au and 41.8g/t Ag), which have not been reported as Ore Reserves. This material has been included in the production schedule, as it is not realistic to leave this material in-situ.*

4.7.9 Ore Reserves

Ore Reserves for the Maleevskoye deposit have been calculated in accordance with the guidelines of the JORC Code (2004). A summary of the Ore Reserves is presented in Table 4.11.

Table 4.10: Maleevskoye Production Schedule (WAI)

Year	Units	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	Total
Total Mined Tonnage	kt	2,313	2,137	2,196	1,710	822	899	420	393	386	328	352	200	118	**12,274**
Zinc Grade	% Zn	5.22	6.14	6.57	7.41	6.69	7.10	6.90	6.41	6.44	7.15	5.27	4.74	5.08	**6.34**
Copper Grade	% Cu	1.75	1.66	1.60	1.59	1.62	1.76	1.97	2.05	2.42	2.55	2.82	1.94	2.82	**1.78**
Lead Grade	% Pb	0.89	1.00	1.00	1.16	0.94	1.04	1.22	1.10	1.18	1.53	0.88	0.85	0.85	**1.02**
Gold Grade	g/t Au	0.42	0.48	0.42	0.64	0.50	0.56	0.67	0.55	0.71	0.77	0.82	0.48	0.63	**0.52**
Silver Grade	g/t Ag	68.07	58.03	49.82	58.55	55.97	56.58	77.36	78.25	79.81	69.70	68.14	59.65	59.42	**60.92**

Table 4.11: Maleevskoye Ore Reserves
(WAI 01.01.2011)
(In Accordance with the guidelines of the JORC Code (2004))

Ore Reserves	Tonnage (Mt)	Grade					Contained Metal				
		Zn (%)	Cu (%)	Pb (%)	Au (g/t)	Ag (g/t)	Zn (t)	Cu (t)	Pb (t)	Au (oz)	Ag (oz)
Proven	5.04	6.46	1.92	1.00	0.56	68.13	325,627	96,770	50,358	90,779	11,044,289
Probable	7.06	6.29	1.69	1.04	0.51	56.23	444,011	119,630	73,522	115,777	12,764,963
Total	***12.10***	***6.36***	***1.79***	***1.02***	***0.53***	***61.19***	***769,638***	***216,400***	***123,880***	***260,233***	***23,810,222***

4.8 Process

4.8.1 Introduction

The concentrator at Zyryanovski was constructed in 1953 and has undergone several stages of refurbishment. The plant utilises the standard processes of crushing, heavy media concentration (HMS) and froth flotation to produce copper, lead and zinc concentrates as well as a gold rich gravity concentrate. The plant went through a substantial upgrade in 2000 when all sections of the plant were refurbished and the grinding and flotation sections were renewed. The plant was originally designed to treat 1.5Mtpa but by 2001 the throughput had increased to 2.25Mtpa. The maximum plant throughput is reported to be 3.5Mtpa.

The plant treats ore from three sources:

- Maleevskoye mine copper-zinc and polymetallic ores;
- Grekhovskoe mine – polymetallic ore; and
- "Foreign" ores.

All ore types contain copper, lead, zinc, silver and gold. The gold is present predominantly in the native form and can therefore be recovered using gravity processing methods.

At Maleevskoye there are two ore types: a copper-zinc ore and a polymetallic copper-lead-zinc ore. The ore types are classified based on the lead content – those containing <0.6% Pb are deemed to be copper zinc ores and those with > 0.6% are classified as polymetallic ores. The copper-zinc ores also have high copper and zinc contents than the polymetallic ores.

The plant also processes a small amount of ore (70ktpa) from the Grekhovskoe mine, which is located some 12km from the plant. This ore source is only expected to last for another three years.

4.8.2 Ore Mineralogy

The ore minerals are chalcopyrite, tennantite, tetrahedrite, galena, sphalerite and pyrite.

4.8.3 Flowsheet Description

4.8.3.1 Crushing and HMS feed Preparation

The crushing plant is designed to operate 21 hours per day. Ore is trucked to site, after primary crushing at the mine site to -100mm, and dumped directly into the main plant feed hopper (170m^3). Ore is further crushed in a Telman (2,120 x 1,500mm) jaw crusher, which crushes any oversize material that the mine has produced.

The crushed produce passes to three stages of feed preparation screening with an intermediate cone crushing stage to produce a –70+5 mm feed product to the HMS. This product is conveyed to one of three bunkers with 8,000, 5,400 and 5,400m^3 capacities.

The crushed material is transferred via conveyors fitted with belt weightometers to two 5mm Feed Preparation screens in series. The screen undersize gravitates to spiral classifiers and the classifier sands are conveyed to flotation. The classifier overflows are pumped to hydrocyclones and the underflows are returned to the spiral classifiers. The cyclone overflows are thickened and pass to flotation.

4.8.4 Heavy Media Separation

Heavy media separation (HMS) takes place on the –70+5mm material in two 6m diameter cones each with a capacity of 170t/h. A combination of ferrosilicon and magnetite is used at separating densities of between 2.8-

3.0g/cc. Weight rejection to the "floats" product ranges between 13-30%. The "floats" and "sinks" are crushed to pass 10mm and the floats used as backfill. The "sinks" and fines products passes to flotation.

4.8.5 Grinding

Milling of the HMS and fines product is achieved using a total of nine 3.2 x 3.9m diameter ball mills, with five being used as primary mills and four as secondary mills. The primary mill discharge gravitates to a jig for coarse gold recovery and the jig tailings spiral classifier and the sands products are returned to the mill.

The spiral classifier overflows are cycloned and the underflows are also returned to the mill. The cyclone overflow is pumped to secondary cyclones and the underflows pass to jigs and intercycle bulk Cu-Pb flotation. Copper and lead minerals are floated using potassium amyl xanthate at pH 7.6. Zinc sulphate and sodium cyanide are used as pyrite and sphalerite depressants. The concentrate is pumped to the second stage Cu-Pb cleaning cells.

The intercycle flotation tailings pass to the secondary mills. The secondary cyclone overflow, at 85-90% passing 74µm passes to Cu-Pb bulk rougher flotation.

The jig concentrates are pumped to a magnetic drum separator to remove grinding steel and then are cleaned using shaking tables to produce a gold rich lead concentrate. The final gravity concentrate, which assays typically 100g/t Au, is free draining and is loaded into Big Bags for transport.

4.8.6 Flotation

Separate copper lead and zinc concentrates are produced by floating a bulk copper lead concentrate and then depressing the lead using sodium dichromate. Zinc is recovered from the copper-lead flotation tailings. There are two parallel flotation circuit using Russian RHYF flotation machines with automatic levels control and low pressure air supply.

Rougher copper lead flotation takes place in seven 25m^3 cells with scavenger flotation taking place in a further six cells. Zinc sulphate and cyanide are used as zinc and pyrite depressants. The scavenger concentrates are pumped back to rougher flotation. The rougher concentrate is cleaned in three stages with a ball mill regrind stage on the cleaner tailings product. The regrind ball mill is currently being replaced with Metso SMD mills which will result in a finer particle size and improved metallurgical performance.

The bulk concentrate is then conditioned with sodium dichromate to depress galena and the copper minerals are re-floated in a separation circuit. The copper and lead products are pumped to thickeners.

The copper lead scavenger tailings are conditioned with copper sulphate to activate sphalerite and the pH is increased to 11-12 using lime. Rougher flotation takes place in four 25m^3 cells and scavenger flotation takes place in a further 11 cells. The rougher concentrate is cleaned in three stages using seven, five and three cells of 8.5m^3 capacity.

4.8.7 Process Control

The plant is controlled with a SCADA system using Delta V software. Feed rate to the plant is controlled automatically and motor temperature and oil pressures are monitored. The system also monitors cyclone pressures, bunker levels, flotation cell pulp levels, air flows, pH and reagent controls. The efficiency of the plant is monitored with a Courier 11 stream on stream analyser.

4.8.8 Concentrate Dewatering

The concentrates are pumped to 15m conventional thickeners. The copper concentrate is filtered using a Svedala VPA Filter Press. The gold rich lead concentrates are filtered using an Outotec Ceramic filter and the zinc concentrates are filtered using a Ceramic filter (Kazakh), or a vacuum drum filter or a Svedala 1540 Filter Press.

4.8.9 Concentrates Transport and Sales

The zinc concentrates are transported by rail to either the zinc smelters at Ridder or Ust-Kamenogorsk. The lead and gravity gold concentrates are sent to smelters in Ust-Kamenogosk. The copper concentrates are shipped to customers in China. The company is in the process of constructing an IsaSmelt furnace in Ust-Kamensogorsk and this is scheduled for completion in 2011. This facility will be able to process the copper concentrates.

4.8.10 Concentrate Management and Labour

There are a total of 415 persons employed within the concentrator. The plant is managed by the "Head of Department" to whom the Chief Engineer reports.

A breakdown of the other personnel employed is given in Table 4.12.

Table 4.12: Process Plant Manning Levels

Chief Technologist	1
Health and Safety	4
Information centre	4
Testwork research	32
Crushing and Beneficiation	121
Grinding and flotation	90
Thickening and filtration	90
TMF	16
Reagents	48
Financial	1
Plant managers assistant	1
Project Manager	1
Copper sulphate production	13

The manning levels would be considered high by western standards but are typical for an operation in the CIS.

4.8.11 Metallurgical Performance

The metallurgical balance for the treatment of all ore types in 2009 is given in Table 4.13.

Table 4.13: Maleevskoye Plant Metallurgical Balance (2009)

Product	Tonnes (t)	Assay %, ppm					Distribution				
		Zn	Pb	Cu	Au	Ag	Zn	Pb	Cu	Au	Ag
Feed	2,405,000	6.28	0.91	1.98	0.53	59.28	100.00	100.00	100.00	100.00	100.00
Zn Concentrate	242,405	55.40	0.46	0.92	0.31	31.24	88.91	5.14	4.68	5.88	5.31
Pb Concentrate	27,603	8.50	46.00	2.30	1.28	757.04	1.55	58.25	1.33	2.76	14.66
Cu Concentrate	166,354	2.92	3.63	24.64	2.63	572.14	3.21	27.73	86.13	33.99	66.76
Gravity Concentrate	4,770	3.52	7.94	1.05	100.00	192.63	0.11	1.74	0.10	37.12	0.64
Tailings	1,642,086	0.51	0.07	0.19	0.09	8.61	5.58	5.32	6.66	11.98	9.92
HMS Tailings	307390	0.17	0.06	0.16	0.06	8.33	0.34	0.89	1.06	1.33	1.80

The balance shows a high degree of metallurgical efficiency with low losses of metals to the HMS tailings and satisfactory grades of base metal concentrates at good recovery levels. The recovery of gold to the gravity concentrates was 37.1% with a further 34% reporting to the copper concentrate.

Copper recovery to the copper concentrate was 86.1% and the concentrate assayed 24.6% Cu, 3.63% Pb, 2.92% Zn, 2.63ppm Au and 572ppm Ag.

The lead concentrate was rather low at 46% Pb, 2.3% Cu and 8.5% Zn at a lead recovery of 58.2%.

The zinc concentrate assayed 55.4% Zn, 0.92% Cu and 0.46% Pb at a zinc recovery of 88.9%.

The average grade of ore treated was 1.98% Cu, 0.91% Pb, 6.28% Zn, 0.53ppm Au and 59.3ppm Ag.

4.8.12 Process Consumables

The major processing consumables are given in Table 4.14 below.

Table 4.14: Maleevskoye Plant Consumables

Item	Consumption kg/t
Ferrosilicon	0.12
Balls	1.09
$ZnSO_4$	0.76
$CuSO_4$	0.36
Xanthate	0.22
NaCN	0.077
Na_2S	0.032
Lime	4.1
Dichromate	0.10
Frother	0.017
Thionocarbamate	0.043
$FeSO_4$	0.30
Electricity	40.4kWh/t
Industrial Water	2.86m^3/t

The rates of consumable usage are typical for the treatment of a moderately hard polymetallic base metal ore.

4.8.13 Process Capital Budget 2011

The total capital expenditure budget for 2011 is US$24.8M. The major items are

- Modernisation of the main flotation plant (US$10.9M);
- New cells for the Cu-Pb separation circuit (US$4.2M);
- A new treatment facility to process lead dust from the Isasmelt process (US$3.0M); and
- One new SMD mill (US$1.69M).

4.8.14 Conclusions

The Maleekskoye Plant operates with a high degree of efficiency on a range of plant feeds from different ore bodies and ore types. The concentrates so produced are typical of those obtained from polymetallic ores. The HMS section is rather old, perhaps overly complex, and in need of modernisation yet it produces a satisfactory upgrade of plant feed with minimal metal losses to the floats products.

The flotation plant operates with a high level of process control and the operators are clearly highly experienced in the treatment of such ores.

The intstallation of Metso SMD mills for copper-lead regrind should result in improved metallurgical performance.

4.9 Environmental, Social, Health & Safety

4.9.1 Introduction

This review of the environmental and social performance of the Zyryanovsk Mining & Concentrator Complex (ZGOK), including the Maleevskoye mine, is based on a brief site visit and reconnaissance, together with discussions with staff of the geology department and Department of Health Safety & Environment. In the short time available it was only possible to have an overview of the project and the way that the company manages its health, safety, environmental and social obligations.

Whilst WAI believes it has gained sufficient insight into the key issues and performance, there may be additional information that was not seen, or variations in interpretation of the available data that could not be explored further.

4.9.2 Environmental & Social Setting and Context

4.9.2.1 Landscape, Topography

The main ZGOK processing plant is located within the industrial zone at the north-east side of the town of Zyryanovsk. This zone is characterised by the largely disused remains of former mining related industry and works, and has an air of dereliction and abandonment. Immediately adjacent to the plant are a large water-filled open pit and a range of mine waste dumps, which are owned by the Government.

The Maleevskoye mine and the TMF are located in rural areas to the north of Zyryanovsk.

The Zyryan region is mountainous, located in the foothills of the southern Altai Mountains, with a landscape dominated by steep hills and open valleys, many of which are wide and fertile.

4.9.2.2 Climate

The climate is sharply continental with hot summers and cold dry winters.

4.9.2.3 Land Use and Land Cover

Zyrynaovsk Concentrator Plant

The surrounding land is mainly post-industrial urban fringe, with a high proportion of derelict or unused land. There is quite a lot of natural colonisation with scrub and trees, as well as municipal and company planting of trees. Old open pits are now open water, usually with steep hazardous slopes into the water.

Previous attempts at 'reclamation' by state authorities have been largely ineffective.

Within the urban fringe landscape there are many residences and informal smallholdings, mainly of lower standard.

Zyryanovsk Tailings Storage Facility

This is located in a fertile river valley, dominated by cultivation and cattle grazing. There are a number of rural communities such as the village of Maleevsk.

Maleevskoye Mine

The mine is located in a mountainous area, with a high proportion of the land forested. Land use consists of localised cultivation in sheltered valleys and pastoral grazing.

4.9.2.4 Water Resources

The area is in the catchment of the Bukhtarma/Hamir River, which flows west into the Irtysh River, one of the main river catchments of eastern Kazakhstan and rising in the southern Altai Mountains. Water resources are mainly surface waters – fast flowing streams and rivers, with shallow groundwater resources in the river valley alluvial deposits The Zyryanovsk tailings facility is within 2km of the river, in the river plain, and overlying the valley alluvial aquifer.

4.9.2.5 Communities and Livelihoods

The town of Zyryanovsk is a significant industrial town, though clearly prosperity has declined in recent years. The community has grown up and developed around the mining and concentrator plant, and a significant proportion of the community will depend on Kazzinc for employment as employees, contractors or providing services.

Outside of the town smaller communities have developed, mainly relying on agriculture, though tourism is also significant (as a base for the mountains). There are a number of village communities in the Bukhtarma River valley, close to the tailings facility.

Zyryanovsk is the administrative centre for Zyryan District, within which the mine and concentrator complex sits. Zyryan is within the East Kazakhstan Oblast, the provincial capital being Ust-Kamenogorsk.

4.9.2.6 Infrastructure & Communications

Zyryranovsk has the usual road, rail and communication infrastructure associated with an industrial town and complex. Power is available from the Kazakhstan grid.

4.9.3 Project Status, Activities, Effects, Releases & Controls

4.9.3.1 Project Description & Activities

Project Components

The Zyryanovsk Mining and Concentrator Complex (ZGOK) consist of 4 main components:

1. The Maleevskoye Mine, 25km to the north of Zyryanovsk. This is the main ore production facility;
2. The Grekhovskoe Mine, 12km to the south of Zyryanovsk. This is exhausted and will shortly be closed, though some areas will continue to be used for waste disposal. This site is not considered further in this report;
3. The Zyryanovskok Concentrator facility itself; and
4. The Tailings Management Facility (TMF), midway between Zyryanovsk and Maleevskoye.

Ore is hauled by truck from the mine to the concentrator, partly on a dedicated haul road and partly on the public highway. On the return journey, trucks are loaded with tailings excavated from the TMF and hauled to the mine for backfill underground.

Maleevskoye Mine

The mine is a self-contained complex with its own facilities and infrastructure, remote from Zyryanovsk. The layout is shown in Photo 4.1.

- Shafts and adit to access underground, with a main ore haulage shaft. Ventilation and emergency access shafts are separate from the main complex;
- ROM ore stockpile and loadout onto trucks;

- Mine offices, administration, maintenance, facilities and lamproom.
- Backfill plant utilising tailings;
- Heating plant and other infrastructure, much of which is disused;
- Electricity substation; and
- Pumped mine water treatment plant and lagoons.

Around and below the mine are derelict and disused areas. These have poor drainage and there is evidence of ochre and acid rock drainage.

The ore mined has very high sulphur (pyrite) content, up to 20% S. Under certain conditions, such as exposure to air, this can give rise to pyrite oxidation and resultant problems with acid production (referred to as Acid Rock Drainage or Acid Mine Drainage, depending on the context).

Ore is loaded at the ROM stockpile by front loader direct into trucks for transport to the concentrator. It was noted during the site visit that trucks are frequently overloaded and spill ore onto the road surface, which will therefore be contaminated with heavy metals. In addition the trucks are uncovered.

Photo 4.1: Aerial View of Maleevskoye Mine

Zyryanovsk Concentrator

The concentrator was originally commissioned in 1955. However the current plant dates from a modernisation completed in 2006, to a capacity to treat a variety of Pb, Zn and Cu ores, up to 3,500kt/yr. At present the concentrator treats ore from Maleevskoye and also 800kt/yr of polymetallic ore from Kazakhmys.

The process flowsheet involves the following stages: crushing and HMS (waste rock) - grinding – gravity – flotation (flotation tailings) – thickening – filtration of concentrates. CN and dichromate are hazardous materials used in the plant. There is a detox unit in the plant for converting the toxic chromium(IV) to the less toxic (and insoluble) chromium(III) form.

In 2008 water recirculation was introduced, to reduce the need to abstract industrial water from local supplies, and eliminate the discharge of waste process water.

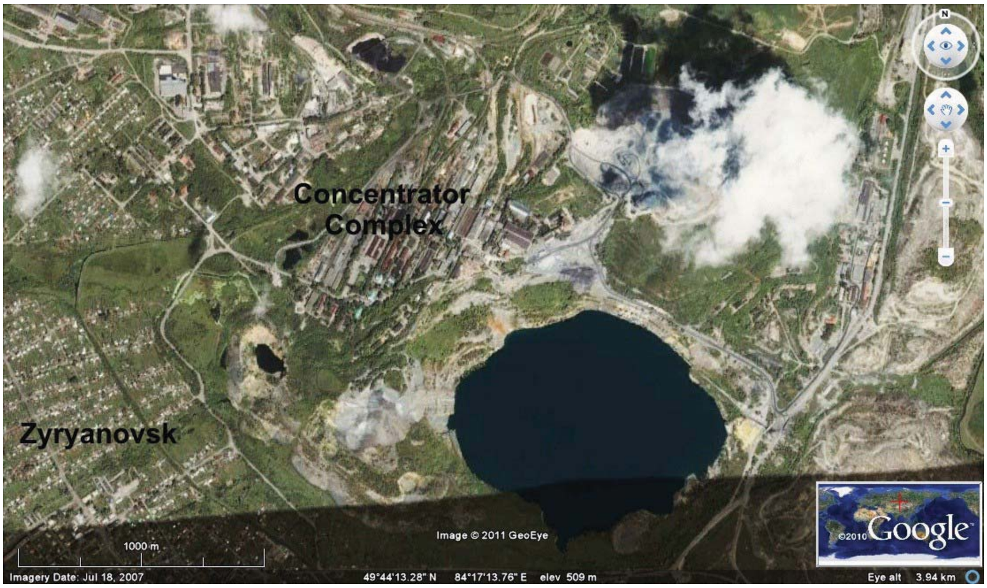


Photo 4.2: Concentrator Complex in Zyryanovsk
(Note the old open pit mine and waste dumps are owned by the Government.)

Zyryanovsk TMF

Flotation tailings produced by the concentrator are piped 4.3km to the TMF, via two steel 820mm diameter slurry lines, one operational and one backup. The pipelines were initially installed in 1986. They are visually inspected daily (walking) and x-rayed annually. A pumping station at the base of the TMF raises the tailings onto the lagoon. Slurry discharge rates are typically 2,000m^3/hr, but vary between 1,800m^3/hr and 2,500m^3/hr; the slurry pulp density is 900kg/m^3.

There are also 2 return water lines, 600mm and 530mm diam, to the plant. Return water rates are usually 1,100m^3/hr to 1,400m^3/hr, though at the time of the visit, were 600m^3/hr.

The area of the TMF is 368.5ha in total and the dam structure is 4.35km in length. The predicted flow to the spillway (residence time) is 15 – 25 days. The dam and tailings basin are unlined and the western half (downslope) is on former alluvial deposits, connected to the river. Infiltration/seepage through the base is estimated by the company at around 800m^3/hr.

The TMF was last raised in 2005 to give a total capacity of 85Mm^3; the current volume is 84Mm^3, equivalent to 120Mt @1.8t/m^3 density. Thus the dam is close to capacity. A new raise is planned which will bring the capacity to 95Mm^3 up to 2016.

Photo 4.3: Zyryanovskoye TMF, Showing Local Communities and Connection to Bukhtarma River

The technical study "Operation of concentrator tailing storage" for 2008 – 2010 has been approved by the State Committee of Emergency Control and Industrial Safety (Report 47 dated 11.01.2008).

An Emergency response plan for the TMF was developed and approved by the Company Operations Manager, and approved by the State Emergency Control Agency, dated 15.12.2009.

4.9.3.2 Mine Wastes – Rock

The existing external waste dumps at Maleevskoye and Grekhovskoe mines pre-date 1993 and are owned by the State; Kazzinc has no liability for them. All mine waste produced at Maleevskoye is used for backfilling underground and for raising the TMF dam, so there are no current waste rock dumps associated with the mine.

4.9.3.3 Mine Wastes – Tailings

Tailings waste from the concentrator has a small particle size – 97% <0.15mm. Recent chemical analysis of the tailings solids is as follows:

Table 4.15: Chemical Analysis

Element	% of solids
Total S	32.82
Fe	5.2
Pb	0.083
Zn	0.2
Cu	0.033
Co	0.005
Ni	0.006
Cd	0.002
Mn	0.08

The tailings have a very high pyrite S content and significant metal content, so are considered hazardous waste. The tailings will be acid generating and there is visible evidence of this on the tailings beaches, however no acid-base accounting or leaching testwork has been undertaken.

The tailings liquid phase (and return water) has a very high pH (9 to 12), together with a high Ca, bicarbonate and sulphate content (as expected), and generally a low heavy metal content. This strong alkalinity will neutralise any acid produced by pyrite oxidation, as long as there is process water on the TMF. However, dispersed tailings, dry beaches and zones where there is no high pH water (such as after the TMF is closed) will be net acid generating and no provision has been made for this.

4.9.3.4 Other Solid Mineral Wastes

Sludge residue, containing precipitated heavy metals, is produced by the mine water treatment plant (see below). Recent analysis of this is as follows:

Element	% of solids
Pb	0.8
Zn	3.5
Cu	1.91

This sludge is transported to an old open-pit at Grekhovskoe; the long term safety and stability of this facility is not known.

The Company is proposing to use the TMF for the disposal of imported 'Scorodite' waste from Vasilkovskoye Mine and the smelters at Ust-Kamenorgorsk. This material is a synthetic Scorodite, being the residue from treatment of arsenic-containing wastes, by addition of lime and iron sulphate, to precipitate insoluble calcium iron arsenate.

The design of this facility is still in preparation. WAI understands that it is proposed to construct a geomembrane lined cell within the tailings and to deposit the Scorodite within this containment. The nature of this containment would seem to be adequate, however the long term risk of depositing arsenic waste in this location, close to groundwater and an important river, will need to be carefully examined.

4.9.3.5 Water Management & Effluents

The recent policy of the company is to conserve water, and water recycling has been introduced at Zyryanovskoye. Water stored on the TMF is returned to the Concentrator as technical water. Potable water required for the plant (drinking and reagent mixing) is supplied from the town supply, and technical water for makeup etc. comes from boreholes.

The expected water balance provisions for the TMF are given in Table 4.16 below.

Table 4.16: Water Balance for the TMF	
Balance items (planned on 2010)	**Total 2010**
Tailings Storage Basin	
A. Water supply thousand cubic metres	
Drainage basin dregs	3248,0
Concentrator production waste water	16095,24
Total of A	**19343,24**
B. Waterloss, thousand cubic metres	
Evaporation	2805,43
- pool surface	1133,59
- land surface (beach)	1671,84
Filtration through dams	635,04
Filtration through bead	113,88
Bead steeping	85,20
Filtration through base	486,96
Total of B	**4126,51**
C. Total volume of water (A-B), thousand cubic metres	15216,73
D. LOOSE WATER km^3	0,0
E. Tailings storage water(C-Г km^3	15216,73
- porewater	690,0
- store water (to the pool)	-373,26
- spill water (discharge)	14900,0
INWASH SPECIFICATION	
Tailings inflow (kT)	2700,0
The same as previous, but packed, thousand cubic metres	1500,0
Solid particles in pool bulk, %	65,0
Tailings storage space filling, %	-
Displaced water volume, $000m^3$	970,83
Raising the water level because of displacement, m	0,61
Water level rise by reason of accumulation, m	-0,24
General water level rise in the pool, m	0,37
Inwash height, m	0,63
Summarized tailings volume at the end of the year, thnd. cub. m	84576,0
Beach length, m	200,0
Inwash line length, m	4000,0
Pool water volume at the end of the year, thnd. cub. m	1955,74
Pool area at the end of the year km^3	1622,49
Pool water average depth, m	1,18
Water mark in pool year at the end of the year, м	469,67
Minimal crest level of warp tailings at the end of the year, m	472,0
Feather excess above the water line at the end of the year, m	2,33
Water Treatment Oxidation Pond	
A. Water in flow km^3	
Drainage basin dregs	522,0
Tailings storage discharge	14900,0
Filtration from the tailings storage	253,80
Total of A	15675,80
B. Waterloss, thousand cubic metres	
Vaporation	546,0
- pool surface	420,0
- land surface (beach)	126,0
Filtration underflow	9920,64
Total of B	10466,64
C. LOOSE WATER km^3	5225,0

Prior to the installation of water recycling in 2008, the decant overflow from the TMF was discharged to the river via a biological oxidation treatment lagoon; see Photo 4.3 and Photo 4.4. At times of high rainfall and snowmelt, excess water is still discharged via the lagoon system – the current licence permits a total of 800,000m^3/year. Monitoring data indicates that this discharge is within acceptable limits.

Seepage from beneath the TMF is significant and will contain contaminated process water leached through the tailings. Groundwater flows from east to west, towards the river, and there is the potential for significant polluted water to reach the river. There is a network of 27 boreholes along the western perimeter and northwest corner of the TMF, used for monitoring and for pumping intercepted groundwater, to prevent migration of contaminated groundwater. This interception is apparently effective and pumped water is returned to the TMF; however it is a long term liability and pumping may be required for many years post-closure of the TMF.

Photo 4.4: Biological (Oxidation) Water Clarification Lagoon at the TMF

Since the water recirculation scheme was introduced, this facility is only used when high rainfall or snowmelt require water to be discharged from the dam.

The Maleevskoye Mine is too far away from the Concentrator for use of the mine water in the Concentrator to be feasible. Minewater is pumped from the mine and is treated in a treatment plant before discharge to the Buktarma river. Treatment involves two stages:

i. Lime dosing with hydrated lime to raise the pH and precipitate metals; and
ii. Settlement in lagoons to remove the sediment. The 3 lagoons are used in rotation and excavated periodically to remove the sludge for disposal.

The resulting effluent is monitored and falls within discharge limits, with the exception of ammonia (NH_3-), which originates from underground blasting residues. The company has planned to reduce ammonia discharge levels by 2012, by utilising the treated mine water in the backfill plant (thus reducing the total quantity discharged).

In addition to the mine water, there are other fugitive uncontrolled discharges from the mine site:

- Drainage from the road during heavy rainfall will transport contaminated sediment (ore spilt from overloaded trucks) on to surrounding land and into streams; and

- Storm rainfall runoff from the mine site area contains acid and metals leached from ore residues deposited or spilled on the ground over a period of years. This discharges via local streams to the river.

The Company has identified the need to reconstruct the stormwater drainage system for the mine area and has an action plan to improve this.

4.9.3.6 Emissions to Air

The significant emissions to air are summarised in Table 4.17.

Table 4.17: Emissions to Air

Mine ventilation	Maleevskoye ventilation shafts	Inert and mineralised rock dust; combustion gases from underground diesel plant.
Deposition and storage of coarse ore at RoM stockpile	ROM stockpiles at Maleevskoye	Mineralised rock dust. Conveyors are covered.
Fugitive dust from loading and haulage of ore	ROM stockpile and haul roads.	Mineralised rock dust. Controlled by watering of haul roads. Spillage of ore from overloaded trucks onto haul roads will increase this source considerably.
Process emissions	Concentrator plant stacks and ventilation	Dust and acid vapours. Controlled by filters and enclosure.
Fugitive dust from tailings surface	Zyryanovskoye TMF	Hazardous dust, containing metals, which could contaminate surrounding land. Controlled by maintaining water cover and wet tailings beaches; effectiveness is dependent on water availability.

One of the most significant sources is fugitive ore dust from the ROM stockpile, due to conveyor deposition and truck loading activities. Dust will contain heavy metals and will disperse on to the surrounding land during dry windy periods.

Fugitive ore dust is also emitted from the haul road between the mine and concentrator. The passage of trucks at speed emits significant dust from the haul road surface and by windblow from the load itself.

4.9.3.7 Waste Management – General

General waste and recyclable materials are collected and stored separately as appropriate, for disposal by specialist contractors. Non-recyclable waste goes to the local municipal landfill.

As part of the environmental licence the company maintains records and makes returns on the quantities of waste produced, under different waste categories.

4.9.3.8 Hazardous Materials Storage & Handling

Oils and fuel (hydrocarbons) – are stored in proper bunded facilities in accordance with licence requirements and international standards. The Company maintains full monthly records and provides annual returns to the State for all consumption and outflows of hazardous materials and wastes.

Particular hazardous materials in use at the concentrator are cyanide and Dichromate. Facilities for storage and handling are in accordance with requirements. There is a detox plant in the Concentrator for treating the streams containing toxic Cr(VI) and converting it to the non-toxic Cr(III) form.

4.9.3.9 General Housekeeping

In general housekeeping at the mine is excellent and all areas are maintained well. No evidence was seen of uncontrolled tipping or abandonment of disused equipment, scrap, containers or wastes.

4.9.3.10 Fire Safety

Good quality systems and arrangements are in place.

4.9.3.11 Security

The site is well secured within a perimeter fence and public safety is maintained. The plant area is accessed via manned security gates and scanners.

4.9.4 Permitting

4.9.4.1 ESIA/OVOS

The Zyryanovskoye complex and Maleevskoye Mine have been operational for many years and have therefore developed progressively. Most of the activities pre-date the requirement for an OVOS so have not been through this process.

WAI understands that an OVOS is in progress as part of the proposed project to stockpile scorodite waste in the TMF dam. This will be undertaken in accordance with Kazakh norms and regulatory requirements.

4.9.4.2 Environmental Permits and Licenses

The Kazzinc audit of its facilities confirms that all necessary permits and licences are in place for the mining and processing operations. The latest TMF dam raise, in 2005, was linked to the increase in mine and plant production. Required permits from the fishing authorities and water basin authorities (both part of the Ministry of Agriculture) are in place. The project was subject to State Expertise in 2006 and received full permits, including the requirements of the Emergency Department.

The Sanitary Zone for the Concentrator plant and TMF has been set at 100m from the perimeter of each facility.

4.9.5 Environmental Management

4.9.5.1 Environmental Policy and Company Approach

Kazzinc has a strong central corporate environmental management function, based in Ust-Kamenogorsk. There is a Chief Environmental Manager, reporting to the First Vice President, and two environmental specialists at headquarters, supporting the site-based staff. The company therefore has a consistent approach to environmental and social management systems, procedures and standards.

The company is certified across all its sites for:

ISO9001 – Quality Assurance	Certified in 2004 and audited in 2010
ISO14001 – Environmental Management System and ***OHSAS18001*** - Occupational Health and Safety Management System	Certified in 2006, recertification audit in 2009 and supervisory audits of company facilities alternately each year, starting 2010.

Glencore is a majority shareholder in Kazzinc and thus also exercises an overarching commitment to Health, Safety, Environment and Community, through the Glencore Corporate Practice commitment applied to all subsidiary companies and operations. WAI understands that all Kazzinc assets were audited by a Glencore team in early 2010. The findings of this are not available to WAI, though it is understood that the main recommendation was in connection with implementing the International Cyanide Code.

Kazzinc also commissioned specialist environmental institute ECOTERA LLC of Almaty to undertake a full audit of all the company's assets. The report: Assessment of Compliance of Activities of Kazzinc LLC with the Nature Protection Laws of the Republic of Kazakhstan and with the Requirements of Controlling Agencies, Auditors Report to Management, was presented in September 2010.

> ***WAI Comment:*** *It is clear that, at a corporate level, Kazzinc is progressive and responsible in its approach to environmental, social, health & safety management across all their operations. They have progressively implemented measures to improve air quality and water management, and maintain consistent standards in accordance with Kazakh norms, policies and requirements.*

4.9.5.2 Environmental Management Staff & Resources

There are 2 environmental staff covering all of the facilities at the operation – a Head Environmental Specialist and an Environmental Engineer. These two both report to the Kazzinc central Environmental Service Department and locally to the Operations Support Manager (and thus to the Chief Engineer).

The Head Environmental Specialist attends weekly management conferences with all function heads, and any environmental issues are dealt with here. There is also regular dialogue with the District Environmental Inspector (Ministry of Environment and Nature Protection - MENP).

4.9.5.3 Systems and Work Procedures

The company has an Environmental Activity Standard and all operations have their individual work procedures and requirements documented. There are regular quarterly and annual reports to MENP detailing all emissions, discharges and wastes, hazardous raw materials and activities, against annual permit levels and norms.

The company prepares a 5 year plan in consultation with MENP, setting out the significant environmental impacts, targets, aims and specific actions and budget to implement the plan. The current action plan has 5 specific actions identified:

1. Reduction of dust emission from Tailings surface, target reduction of 1.5t/yr by 2015;
2. Reduction of discharge of NH_3 in mine water by 0.8t/yr by 2012;
3. Safe disposal of Scorodite (As containing waste) in TMF by 2011;
4. Reconstruction of surface stormwater drainage for Maleevskoye Mine by 2011; and
5. Improvements within the Sanitary Protection Zone of the Concentrator, 49.7ha, by 2012.

4.9.5.4 Environmental Monitoring, Compliance & Reporting

There is an extensive groundwater and surface water monitoring regime around the project areas, and particularly the TMF, which is contracted out to a specialist supplier. Monitoring is undertaken 4 times per year and a detailed annual report is prepared. This monitoring is supplemented by the company with 10-day monitoring of critical parameters (pH, conductivity, metals, etc) at key locations such as discharges.

No details of air quality or soil monitoring were reviewed.

In addition to physio-chemical monitoring, regular monthly inspections are carried out by environmental staff. Whilst there are no pro-forma reports or checklists, any problems or issues are noted and reported.

Quarterly and annual reports are prepared for company management, and the Environmental Manager participates in the weekly management meetings.

The independent Kazzinc audit confirms that the compliance record with standards and norms is good, with only a few problem areas as identified in this report

> ***WAI Comment:*** *The monitoring of water quality, which is the main risk area, is thorough and sufficiently comprehensive, and compliance with the required standards is good. However, as is usual it is based on measuring physio-chemical parameters of effluents and pathways, and does not give a good picture of the impacts or health of the receiving environment. For example, periodic sampling of soils, vegetation and stream/river sediments, gardens, drinking water, and biological monitoring of benthic invertebrates, would give an indication of the long term impacts of the mining operation on the surrounding environment and communities.*
>
> *Monitoring air quality, particularly dust deposition, should also be considered.*

4.9.6 Social and Community Management

4.9.6.1 Stakeholder Dialogue and Grievance Mechanisms

Zyryanovsk is a mining community and has a very close relationship with the company; it is reported that there is regular dialogue with the local administration and the Mayor of Zyryanovsk. However, whilst the OVOS law prescribes structured public hearings, none have apparently been carried out yet under these regulations. Nevertheless the Company does hold public meetings and involves representatives of the community in the planning stages.

Company news and initiatives are regularly reported and discussed in local and Oblast newspapers and the local radio broadcast channel. The Company employs a journalist specifically to report on company activities. There is ample opportunity for the public to voice opinions, concerns or support for the company activities through these media.

The local administration authority holds open public meetings twice per year, at which any member of the public can raise issues or request information from any industrial enterprise in the District. GOK Directors are usually present at these meetings and present reports on the company activities.

Specific complaints or grievances are received by the GOK Director's office and recorded in a register. This register also gives a response to the comment and any actions taken. The results of this are published in the local newspapers.

It was reported to WAI that there are no NGOs active in the region.

4.9.6.2 Social Initiatives and Community Development

The Company has an active social sponsorship and community development programme, which is set out in a Memorandum between Kazzinc and the Mayor of Zyryanovsk, the current one being dated May 2008. This sets out a commitment to 8 specific initiatives for community support, mainly covering donations and provision of facilities for orphans, disabled and elderly within the District.

These initiatives are mainly of a charitable nature and are clearly an important contribution to the District. No information was provided to WAI on other types of community support such as economic diversification, local enterprise and training that the Company could consider as part of a wider and longer term support.

4.9.7 Health & Safety

4.9.7.1 Health & Safety Management Arrangements

The mine and the concentrator plant both have labour safety departments, responsible to the mine and concentrator managers respectively, but overseen by the Kazzinc team (responsibility for Labour is 1st VP of Kazzinc). Staffing is organised as follows:

- Safety Manager for ZGOK
- 3 Safety Inspectors & 2 Clerks reporting to the Safety Manager
- 2 Safety Specialists at the Mine
- 1 Safety Specialist at the Concentrator
- 2 Safety Specialists at auxiliary departments

There are overall standardised policies and procedures for Kazzinc together with local procedures in each facility appropriate to the activities carried out there. These follow the statutory prescribed procedures in the Law on Industrial Safety, which sets out the functions, responsibilities and procedures to be followed. These are implemented with the prescribed 3 layers of safety control and responsibility: Foreman; Senior Specialist at each facility and Senior Specialist of Company.

The medical service at the mine and plant is outsourced to a specialist company, who provide a medical centre and trained staff.

All staff receive 5 days of safety and other induction training at the dedicated Kazzinc training centre at commencement of employment.

4.9.7.2 Occupational Issues at ZGOK

The Company is aware of potential problems at the mine due to the high level of silica in the rock. Mine ventilation is being improved with the proposed installation of a larger fan, to reduce the risk further and ensure that air quality is within required norms.

4.9.7.3 Performance and Accident Records

The company performs regular occupational health and exposure surveys, with annual monitoring and reports prepared by an independent specialist company. Air quality (dust and gases from blasting) is monitored daily by the mine rescue department, in accordance with sanitary norms.

Occupational health and safety data was seen by WAI and no particular issues were noted. There has been one case of silicosis in 5 years, and the incidence of bronchitis is slightly above normal expectations.

4.9.8 Mine Closure & Rehabilitation

4.9.8.1 Mine Closure Plans

According to the Subsoil Use Code, all stockpiles and tailings formed before 1992 revert to State supervision. Thus all old waste dumps around Zyryanovskoye are not connected with the Company and are not a liability for the present operation. The TMF was originally constructed in 1968 so the deeper layers technically fall under State control and are not the responsibility of the Company. However, in practice it is not possible to separate the pre and post-1992 parts of the TMF and all is treated as one entity.

Under the Subsoil Use Code licence requirements, the mining operation has prepared a closure plan, dealing with the physical closure only; there is no provision for social closure, other than the statutory redundancy payments. The costs of the closure plan have been estimated at US$2.5million.

The Concentrator and TMF are licensed as an Industrial complex and thus have no requirement for a formal closure plan.

The TMF is considered as a man-made mineral formation, with a metal content that may, one day, be of economic or beneficial value, and is therefore considered as an asset. It is thus treated as a potential resource that will either be re-worked by the company or returned to the State. Kazzinc have stated that, in the long term, they will continue to be responsible for the TMF and will prepare a design for decomissioning and recultivation, but that this is too far in the future for consideration now. Thus, no provision has been made for its long term and post closure condition, or funding.

> ***WAI Comment:*** *Whilst the Maleevskoye Mine has made provision for closure, there is no provision for the concentrator or TMF, and only short-term plans are made. There is no consideration of long term and post-closure physical, geochemical or land use stability, and long term risks do not appear to be well considered. Thus the TMF in particular must be considered a potential long term liability, particularly as the tailings material itself is potentially hazardous and deposition of imported arsenic-containing scorodite waste increases the potential risk.*

4.9.8.2 Financial Provision for Closure

Under the Subsoil Use Code licence the closure fund for the Maleevskoye Mine (US$2.5M) has been created and is held in a special Company account, as part of the Mine Closure Programme that has been developed.

There is no financial provision for closure of the Concentrator complex and TMF, and in the absence of a closure plan there is no basis for estimating the magnitude of the closure and post-closure liability.

4.9.9 Conclusions

4.9.9.1 Environmental and Social Liabilities & Risks

For the most part, Maleevskoye Mine has a very high standard of environmental, social, health and safety management. It is compliant with Kazakh licences, permits and norms. Both the environmental and labour safety functions of the company are of high quality and well organised, and supported by a strong corporate policy, commitment and team.

There are no known social, community or cultural issues or impacts that need to be addressed, and no displacement or compensation requirements. The mine appears to have a good relationship with the local communities, and does not put undue pressure on the social infrastructure of the District or Oblast.

The town of Zyryanovsk is essentially a mining town, and grew up around the developing mining operations over 200 years. It is heavily dependent on the continuation of mining for its economy.

Whilst there are a number of historic mining liabilities around Zyryanovsk, these are not part of the company's current liability and responsibility for them lies with the State.

WAI has identified the following areas of potential liability and risk, which do not meet international expectations and standards:

1. The characterisation and risk assessment for long term disposal (storage) of acid-producing tailings. Monitoring and protection of groundwater and related surface water resources are important regionally;

2. Dust control from the exposed tailings surface, particularly a backup irrigation system during periods of water shortage;
3. Dust control from the ROM ore stockpile, from the haul roads between the mine and concentrator, to prevent contamination of surrounding land and sediment in runoff into watercourses. Overloading of haul trucks, and the absence of load sheeting, should be addressed by the company urgently; and
4. Long term closure liabilities for the concentrator and the TMF Facilities within the company's responsibility need to be addressed, with a full closure plan, rehabilitation and costs, and a long term environmental liability risk assessment.

4.9.9.2 Compliance with Local and International Standards and Expectations

The project is generally compliant with international standards and expectations, with the exception of the potential liabilities and risks identified above. It is likely that the management of hazardous tailings and the mine closure provisions fall short of international practice, although WAI believes that the company's attention is focussed on further improvement in connection with these issues and there are no reasons why they cannot be readily resolved in the near future.

4.9.9.3 Recommendations for ESAP

The following Table 4.18 summarises the action plan identified for investment by the Company (numbers 1 to 5), together with some additional WAI recommendations for an environmental and social action plan for the mining and concentrator complex.

Table 4.18: Recommendations for Environmental & Social Action Plan

	Action	Priority & Timescale
1.	Reduction in the emission of dust from the TMF, by a combination of maintaining water cover and irrigation sprays.	2015
2.	Reduction in the release of ammonia in treated mine water effluent	2012
3.	Disposal of imported Scorodite in safe containment cell on TMF (benefits to other Company operations)	2011
4.	Reconstruction of stormwater drainage and treatment for Maleevskoye Mine	2011
5.	Improvements to the housekeeping and landscape within the sanitary protection zone of the Zyryanovskoye Plant (49.7ha)	2012
6.	Improvements to ore loading on trucks at Maleevskoye Mine – avoidance of overloading and sheeting of loads, to prevent ore spillage on haul road and dust emission	2011
7.	Characterisation and testing of tailings solids for ARD and metal leaching; risk assessment for long term storage of tailings, including Scorodite.	2012
8.	Prepare integrated closure plans for mine, concentrator and TMF, for physical, biological and social closure, including cost estimates and financial provision.	2013
9.	Characterisation and testing of mine water treatment sludges at Grekhovskoe quarry; risk assessment and groundwater monitoring.	2012

The environmental and social performance of the Zyryanovskiy Mining & Concentrator Complex (ZGOK), including the Maleevskoye mine, is based on a brief site visit and reconnaissance, together with discussions with staff of the geology department and Department of Health Safety & Environment. In the short time available it was only possible to have an overview of the project and the way that the company manages its health, safety, environmental and social obligations.

Whilst WAI believes it has gained sufficient insight into the key issues and performance, there may be additional information that was not seen, or variations in interpretation of the available data that could not be explored further.

5 RIDDER-SOKOLNIY DEPOSIT

5.1 Introduction

Kazzinc own and operate three polymetallic deposits within the Ridder Area: Ridder-Sokolniy, Tishinskiy, Shubinskiy and are planning to develop a further three at Chekmar, Dolinnoe and Obruchevskoe.

5.1.1 Location & Access

All six deposits are located within approximately 35km of the town of Ridder, which itself is situated some 120km north-east of the city of Ust-Kamenogorsk (Ust). The town is located in the north-eastern part of the Ridder basin, which has an undulating relief and absolute elevations between 650m-1,000m above Baltic Mean Sea Level. It is flanked on the southern side by the Prohodny and Ivanovsky mountains which rise to 2,300m, and by several lower mountain ranges on the northern side.

The town of Ridder is linked by metalled roads and by rail, with total driving time to Ust of approximately two hours. Ust is served by regular Air Astana flights to the main Kazakh cities of Astana and Almaty.

The six deposits are located as follows:

- The Ridder-Sokolny mine complex is located on the outskirts of the town of Ridder, with Kazzinc offices located in the town centre;
- The Tishinsky mine is located on the main road between Ridder and Ust, 17km south-west of Ridder and 20 minutes from Ridder-Sokolny Mine;
- The Shubinskiy deposit is located some 14km to the north-east of Ridder via a well-maintained, graded haul road;
- The Chekmar deposit is the most outlying of the Ridder Area deposits and is situated some 46km north of Ridder via a graded road; and
- The Dolinnoe and Obruchevskoe deposits are located 7.5km and 11km respectively to the north of the town of Ridder on a graded gravel road from Ridder to Biysk (located within the Russian Federation).

Figure 5.1 below shows the Ust Region and the Ridder Area deposits.

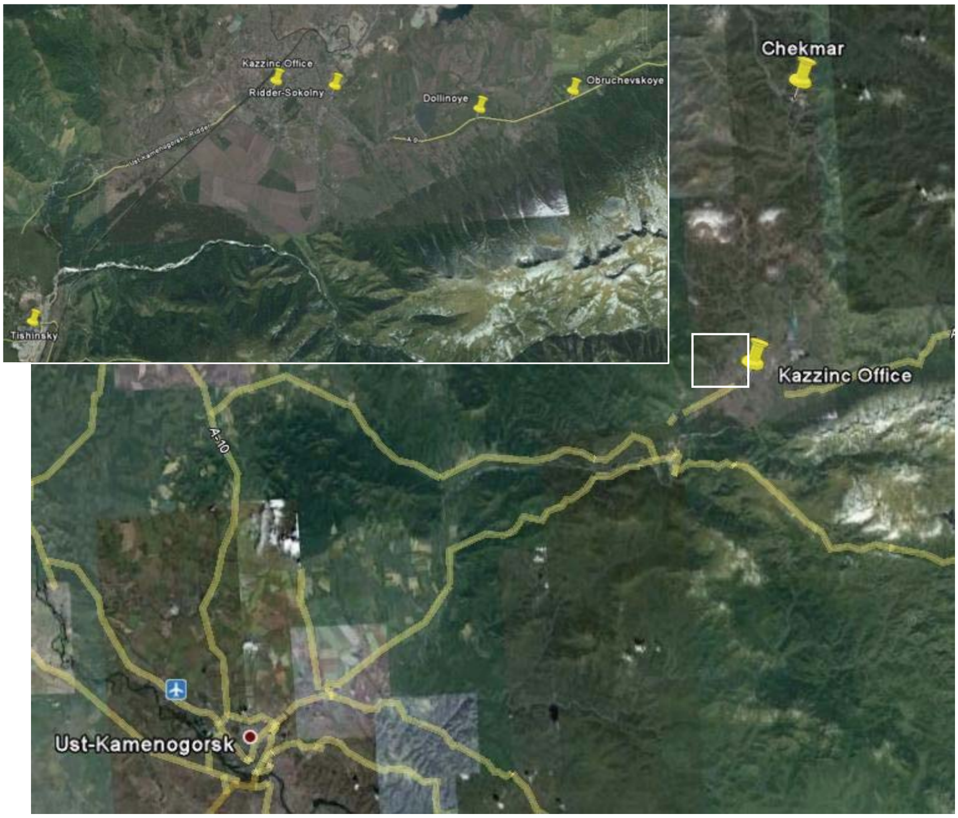


Figure 5.1: Ust Region and (inset) Ridder Area

5.1.2 Topography and Climate

The climate in the region is extreme continental with an average annual temperature of around 2°C, varying between +35°C in July (average of 18°C) and -45°C in December (average of -16°C). Mean annual precipitation is approximately 700mm, of which 75% falls between October and April as snow. Snow cover generally lasts from October to mid-April.

The topography of the region comprises principally undulating relief at absolute elevations between 600-1,200m above Baltic Mean Sea Level, and mountain ranges with peaks of between 1,500-3,000m.

The region has well developed infrastructure.

> ***WAI Comment:*** *In general, access and infrastructure in the Ridder Area are good, and despite harsh winter conditions, present no major obstacles to the continued development of the mines in the area.*

5.1.3 Infrastructure

5.1.3.1 Mineral Rights and Permitting

Kazzinc holds the right to mine gold-polymetallic ore under the terms of the 'Contract for Subsurface Use (MG No.65D)' dated 28 March 1997. The Contract is valid for 25 years from the date of the licence issue and can be extended by mutual agreement between Kazzinc and the issuing authority.

The current mining licence (Ridder-Sokolniy) covers an area of 11.5km^2 and was issued by the Ministry of Energy, Mineral Resources and the Environment of the Republic of Kazakhstan in May 2003, superseding the previous mining lease.

5.1.3.2 Mining Licence for Subsoil Use for Exploitation of Gold-polymetallic Ores (1)

The licence covers an area of 8.588km2 and permits mining to a depth of horizon 18.

The boundaries of the licence (local grid co-ordinates) are defined in Table 5.1 below.

Table 5.1: Mining Licence (1) Boundary Points		
Corner Points	**Y**	**X**
1	80 427	10 465
2	80 572	10 780
3	80 573	11 245
4	80 195	11 535
5	79 750	11 700
6	79 675	12 244
7	80 006	13 800
8	79 880	13 650
9	79 662	13 800
10	79 056	13 696
11	79 530	12 850
12	78 285	12 800
13	78 315	13 532
14	77 952	13 480
15	77 267	12 447
16	76 539	12 437
17	75 680	12 480
18	77 480	11 807
19	77 417	11 294
20	77 044	11 047
21	11 120	10 627
22	77 577	10 450
23	77 407	10 067
24	77 953	9 820
25	78 135	10 300
26	78 500	9 856
27	78 690	10 266
28	78 917	10 575
29	78 548	10 918
30	78 942	10 920
31	79 143	10 646
32	79 440	10 490
33	79 630	10 710
34	80 010	10 833
35	80 172	10 476

5.1.3.3 Mining Licence for subsoil Use for Exploitation of Gold-polymetallic Ores (2)

The licence covers an area of 11.5km^2 and permits mining to a depth of horizon 24.

The boundaries of the licence are defined in Table 5.2 below.

Table 5.2: Mining Licence (2) Boundary Points		
Corner Points	Corner Points Coordinates	
	Latitude (N)	Latitude (E)
1	50°21'38"	83°31'33"
2	50°21'42"	83°31'50"
3	50°21'43"	83°32'12"
4	50°21'44"	83°32'34"
5	50°21'40"	83°34'04"
6	50°21'25"	83°34'28"
7	50°20'45"	83°34'20"
8	50°20'37"	83°33'38"
9	50°20'30"	83°33'37"
10	50°20'31"	83°34'12"
11	50°20'19"	83°34'10"
12	50°19'39"	83°33'13"
13	50°19'38"	83°32'25"
14	50°19'57"	83°32'37"
15	50°20'02"	83°32'16"
16	50°19'49"	83°32'04"
17	50°19'42"	83°31'09"
18	50°20'00"	83°30'45"
19	50°20'18"	83°31'02"
20	50°20'24"	83°31'26"
21	50°20'36"	83°31'04"
22	50°20'42"	83°31'24"
23	50°20'33"	83°31'46"
24	50°20'37"	83°31'57"
25	50°20'50"	83°31'56"
26	50°20'56"	83°31'43"
27	50°21'06"	83°31'35"
28	50°21'14"	83°31'27"
29	50°21'30"	83°31'34"

5.1.3.4 Geological Allotment for Exploration of Gold-Polymetallic Ores on the Flanks of Ridder-Sokolniy

The exploration licence covers an area of 12.1km^2. The boundaries of the licence are defined in Table 5.3 below.

Table 5.3: Exploration Licence Boundary Points		
Corner Points	Corner Points Coordinates	
	Latitude (N)	Latitude (E)
Site #1		
1	50°21'42"	83°31'50"
2	50°21'34"	83°31'00"
3	50°20'53"	83°29'48"
4	50°19'20"	83°30'28"
5	50°19'26"	83°24'50"
6	50°20'20"	83°24'50"
7	50°20'46"	83°33'52"
8	50°20'37"	83°33'38"
9	50°20'30"	83°33'37"
10	50°20'31"	83°34'12"
11	50°20'19"	83°34'10"
12	50°19'39"	83°33'13"
13	50°19'38"	83°32'25"
14	50°19'57"	83°32'37"
15	50°20'02"	83°32'16"
16	50°19'49"	83°32'04"
17	50°19'42"	83°31'09"
18	50°20'00"	83°30'45"
19	50°20'18"	83°31'02"
20	50°20'24"	83°31'26"
21	50°20'36"	83°31'04"
22	50°20'42"	83°31'24"
23	50°20'33"	83°31'46"
24	50°20'37"	83°31'57"
25	50°20'50"	83°31'56"
26	50°20'56"	83°31'43"
27	50°21'06"	83°31'35"
28	50°21'14"	83°31'27"
29	50°21'30"	83°31'34"
30	50°21'38"	83°31'33"
Area – 10.2km^2		
Site #2		
1	50°21'25"	83°34'28"
2	50°20'22"	83°36'02"
3	50°20'22"	83°34'56"
4	50°20'49"	83°34'00"
5	50°20'45"	83°34'20"
Area 1.9km^2		

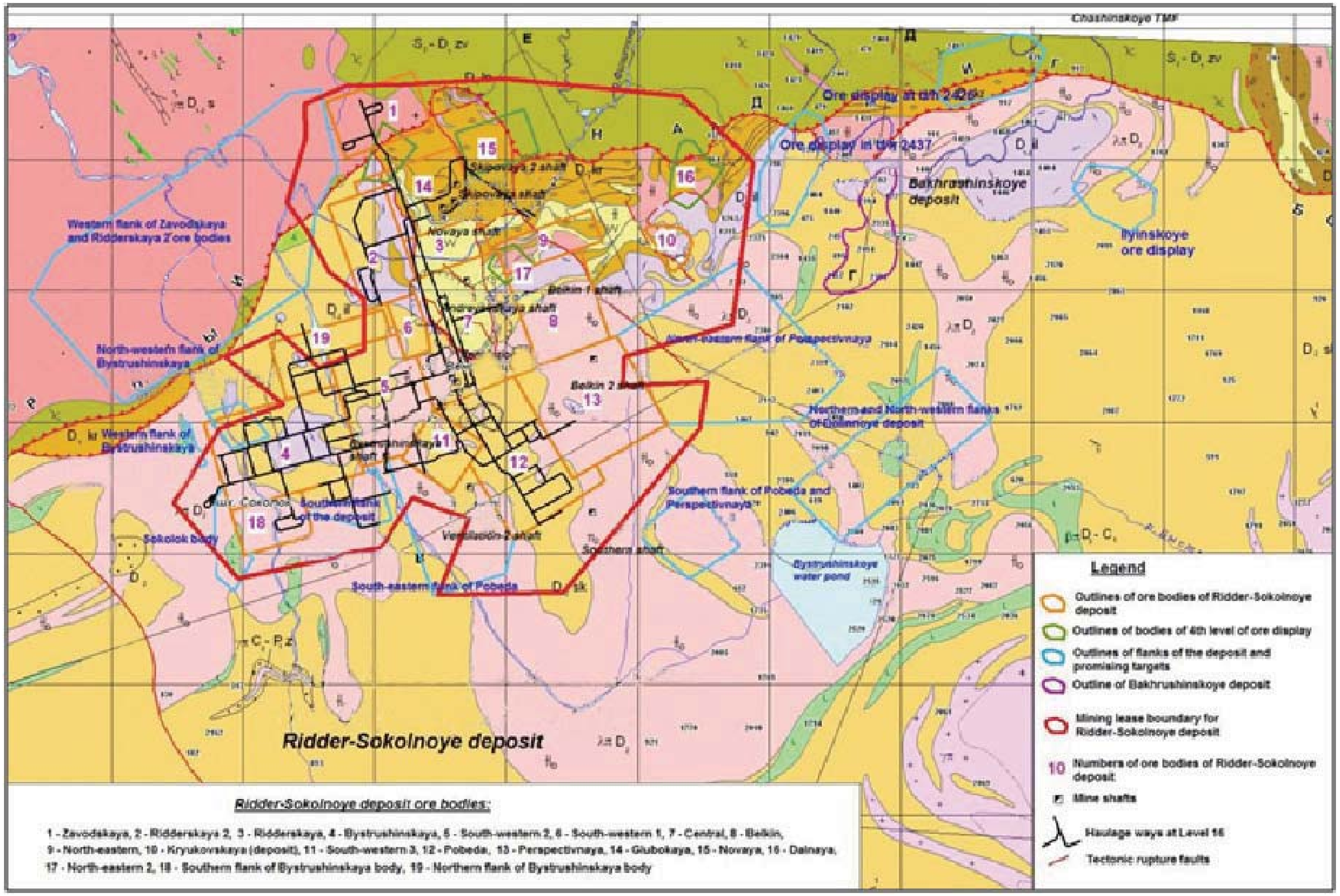


Figure 5.2: Location Plan showing Position of Ridder Licences

5.1.4 Project History

The Ridder-Sokolniy deposit was discovered in 1786 and has been worked since 1789, initially exploiting silver and gold-bearing ore, before producing gold-polymetallic ore from surface and underground working since 1920. The total mine production over the period 1789 to 2008 inclusive is understood to be of the order of 180Mt at 3g/t Au, 29.5g/t Ag, 0.3% (Cu), 0.9% (Pb) and 1.7% Zn. These figures are based on a summary of annual accounting.

The current underground mine layout was developed on the basis of a project called 'Reconstruction of the Mines in the Ridder-Sokolniy Deposit' which was prepared by the Kazgiprotzvetmet Institute in 1984.

Prior to 1994, the deposit was exploited via three separate underground mines (known as Leninogorsk, Ridder and 40th Anniversary). Between 1994 and May 2001 only Ridder and 40th Anniversary were operational, utilising a common haulage and ventilation system, and in May 2001 the three mines were merged into a single operation, subsequently renamed as Ridder-Sokolniy.

5.2 Geology and Mineralisation

5.2.1 Regional Geology

The Ridder-Sokolniy deposit is situated in the northern part of the Ridder mining district in the Rudnyi Altay. Geologically, this district coincides with an unusual graben structure oriented perpendicular to the regional north-west structural trend and preserving Silurian to Middle Devonian volcano-sedimentary sequences almost in the same position as they were laid down. The graben measures 20km in length by up to 8km in width at its centre. It is bounded on all sides by reverse faults. Terrains located immediately to the west and east of these faults display intense folding with steep pervasive north-west striking schistosity.

The graben is filled with four volcano-sedimentary formations comprising, in ascending order, Zavodskaya (S2-D1zv), Leninogorskaya (D1ln), Kriukovskaya (D1kr), Ilinskaya (D1-2il) and Sokolnaya (D2sk) formations (refer to Table 5.4 below).

The Kriukovskaya Formation hosts most of the polymetallic mineralisation known within the confines of the graben structure. It consists of three members:

- Upper member of predominantly fine-grained sedimentary rocks (mainly siltstones of highly variable composition);
- Middle member of acid volcanogenic rocks (agglomerates, tuffs, minor lavas and lava breccias and their reworked derivatives), which attain 350m in thickness in the vicinity of the palaeovolcanic centres (in the central part of the Ridder-Sokolniy deposit and at the Kriukovskaya Lode) and peter out rapidly in the south-western, southern and south-eastern flanks of the Ridder-Sokolniy deposit; and
- Lower member consisting of carbonaceous siltstones and tuffaceous clastic sediments of varying grain size.

Calcareous siltstones at the top of the Kriukovskaya Formation, with their distinctive green-grey or ash-grey colour, serve as the marker horizon throughout the Ridder graben. Isolated reef limestones are also locally known at the same level.

Table 5.4: Stratigraphy of the Ridder Graben

Formation	Symbol	Typical Lithology of Sedimentary and Volcanogenic Origin	Intrusive Rocks
	Q	Alluvial sediments predominated by gravel and pebble size clasts, loam, thickness from several metres to over 100m	
Sokolnaya	D_2sk	Dark-grey to black carbonaceous-pelitic siltstones with intercalations of sandstones, gravelites, calcareous siltstones at the base	Sills of dolerite, porphyritic dolerite and subvolcanic porphyries Subconformable subvolcanic rhyolites ranging in thickness from 130m to 400m with widespread hydrothermal alteration Subconformable tabular extrusive to subvolcanic rhyolites and dacites of variable thickness
Ilinskaya	D1-2il	Variable banded sedimentary and volcano-sedimentary rocks, including tuffs of variable composition, tuffites, tuffaceous gravelites, tuffaceous sandstones, siliceous-mica rocks ("peperite"), sandstones, gravelites, siltstone and rare limestones Basal horizon of coarse tuffogenic-sedimentary rocks with angular clasts of siliceous siltstones, sandstones and quartzites, generally 10-15m in thickness, resting on erosional surface of calcareous-siliceous siltstones of Kriukovska Series	Subvolcanic porphyritic andesites forming subconformable tabular bodies
Kriukovkaya	Dkr	"Hanging wall schists" (carbonaceous siltstones, calcareous-pelitic siltstones, siliceous siltstones, sandstones, gravelites and quartzites) Siltstone member formed of fragmented terrigenous rocks: siltstones, aleuropelite, siliceous siltstones, microquartzite. Volcanic material is transformed into chlorite-sericite-quartz rocks. Volcanomictous gravelite with interlayers of siltstones and silty sandstones; rock fragments consists of liperaite-dacite porphyries; albitophyre, rarely tuffs of acid composition	Subvolcanic porphyritic andesites forming subconformable tabular bodies Subconformable subvolcanic rhyolites ranging in thickness from 130m to 400m with widespread hydrothermal alteration
Leninogorskaya	Dln	Metamorphosed tuffs of acid and intermadiate composition, tuffogenic and volcanomictic sandstones and gravelites Intercalations of siliceous and siliceous-pelitic siltstones	Subconformable subvolcanic rhyolites ranging in thickness from 130m to 400m with widespread hydrothermal alteration
Zavodskaya	S_2-D_1zv	Massive and schistose greenschist facies rocks formed from sandstones, siltstones and pelites (chlorite-epidote-carbonate-albite-quartz schists)	

5.2.2 Local Geology

The Ridder-Sokolniy deposit is a cluster of approximately 20 ore zones of predominantly veinlet-type and disseminated sulphide mineralisation found at the same stratigraphic position within the Middle Devonian volcano-sedimentary Kriukovskaya Formation. Four other ore zones are known from deeper stratigraphic levels. The mineralisation covers an area of approximately 20km^2 and extends down to a depth of at least 700m.

Two NW-trending faults divide the deposit into three structural blocks:

- The Central block is bounded by the Drill holes 50-53 Fault on the south-western side and by the Nikolayevska Shaft Fault on the north-eastern side. It contains the Central ore zone flanked by the Ridder, 2nd Ridder and Zavodskaya ore zones on the north-western side and by Pobeda (Victory) ore zone on the south-eastern side;
- The Western block (south-west of the Drill holes 50-53 Fault) contains three South-Western ore zones (1st, 2nd and 3rd) and Bystrushinskaya and the Southern flank of Bystrushinskaya ore zone; and
- The Eastern block (north-east of the Nikolayevska Shaft Fault) contains the North-Eastern, Philippovskoye, Belkina, Perspectivnaya and Kriukovskaya ore zones.

At depth further mineralisation has been identified and includes the Novaya (New), Glubokaya, Dalnyaya (Far) and the 2nd North-Eastern ore zones.

These ore zones are shown in Figure 5.3 to Figure 5.6 below.

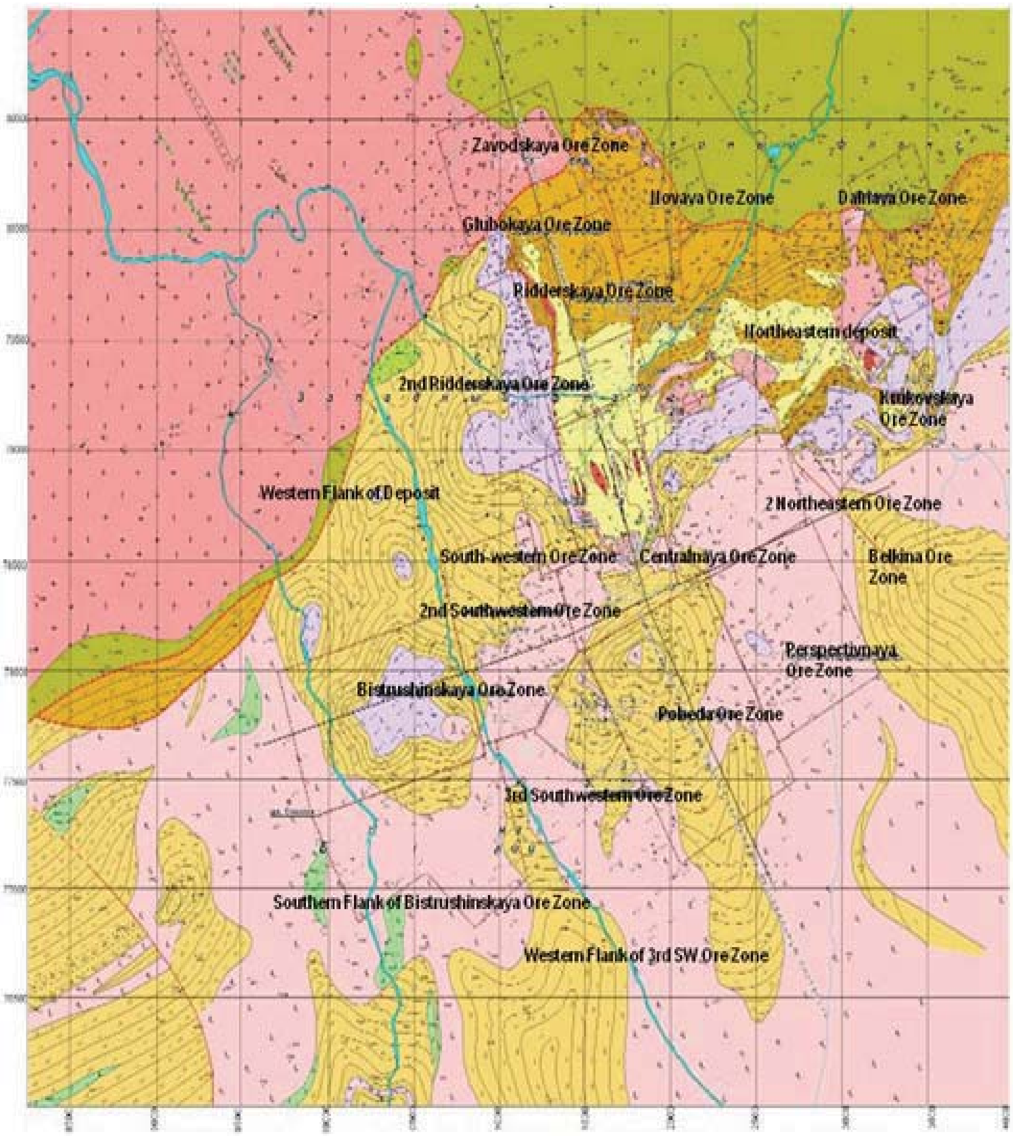


Figure 5.3: Geology Plan showing the Location of the Main Ore Zones

West **East**

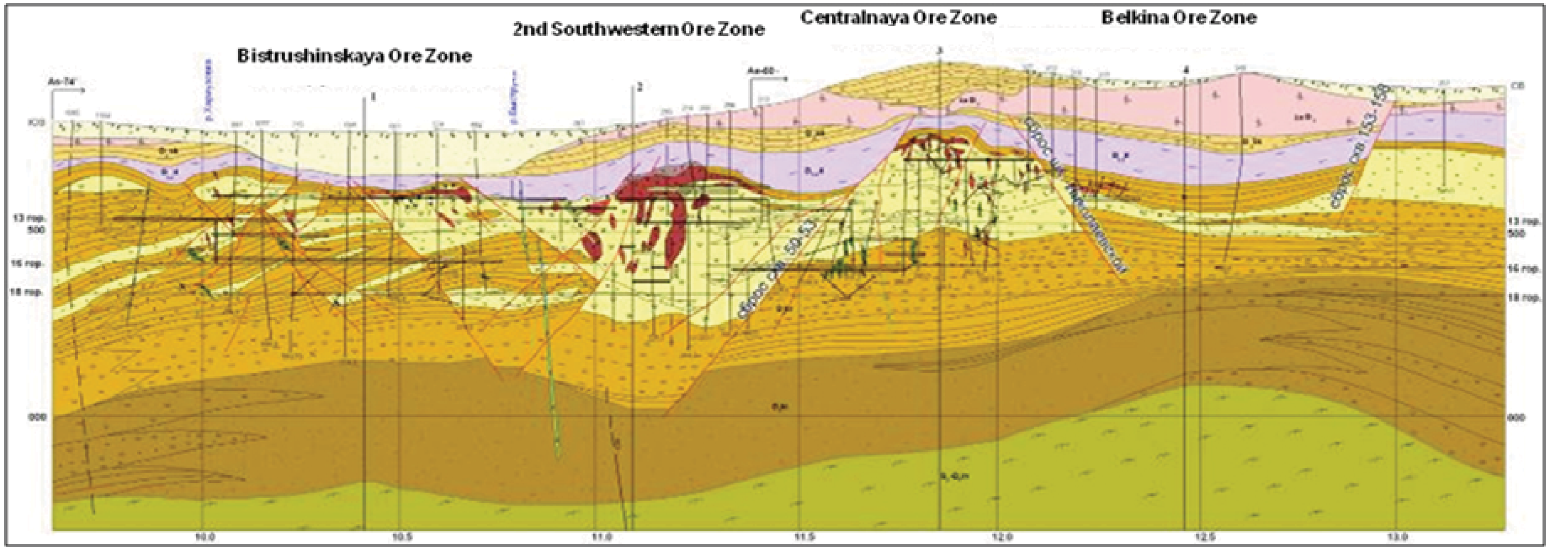


Figure 5.4: West-East Cross Section through the Central Block (main ore zones in red)

North **South**

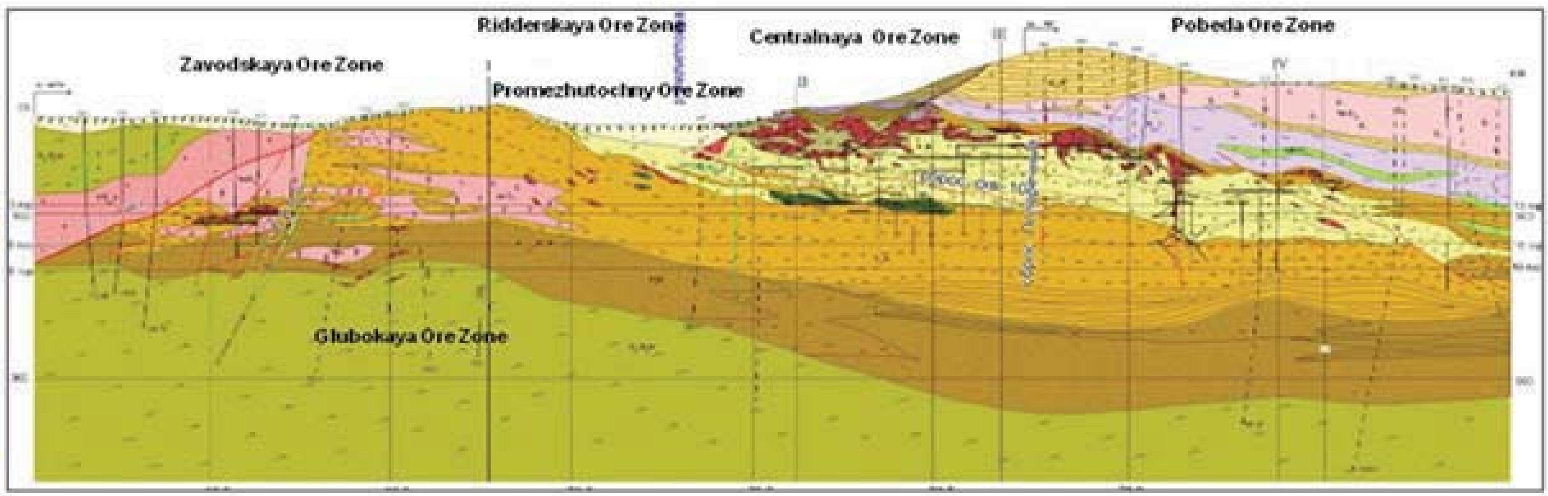


Figure 5.5: North- South Cross Section through the Central Block (main ore zones in red)

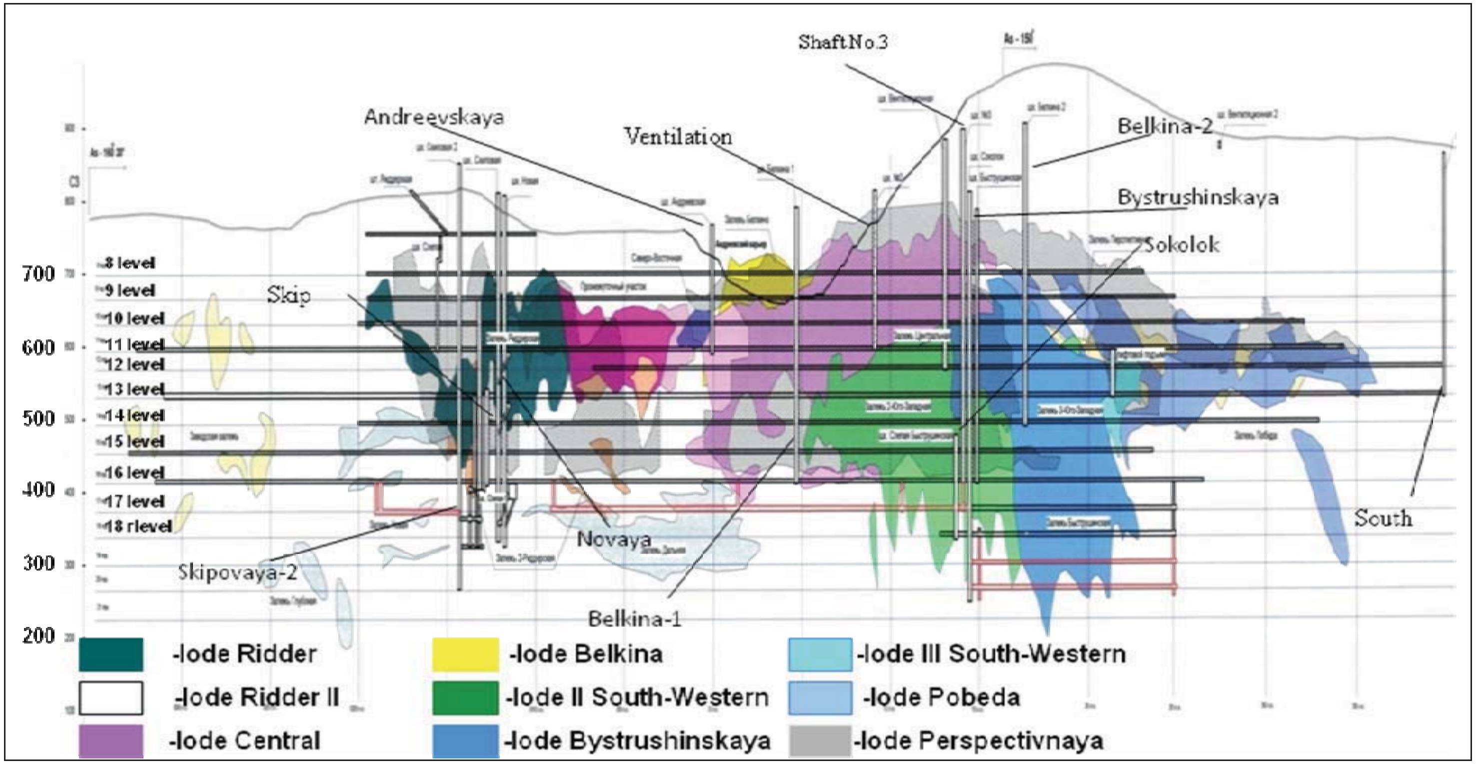


Figure 5.6: Long Section showing the Location of the Main Ore Zones

Horizon I, which is now completely exhausted within the developed mine area, consisted of stratabound mineralisation comprising massive sulphide lenses among veinlet-disseminated mineralisation underlain by veinlet-type and disseminated mineralisation resembling stockwork, which extended to depths of tens of meters. Some veinlet-type and disseminated mineralisation was also found above the massive sulphide lenses. The largest lens (Central) was 1km long (NW), 300-500m wide and up to 100m thick.

The highest grades at this horizon were reported from gold-barite-polymetallic mineralisation, which comprised approximately 15% of the mineralised volume at this horizon but contained approximately 25% of the total contained zinc and lead. Gold grades in this type of mineralisation were in the range of 2g/t Au. Gold-barite-polymetallic mineralisation formed clusters of cupola shaped bodies characteristic for the southern and south-eastern parts of the Ridder-Sokolniy deposit (Central, 2nd South-Western, Perspectivnaya ore zones).

The underlying Horizon II, which is currently in exploitation, contains steeply to moderately dipping veins of zinc-copper mineralisation with variable composition ranging from polymetallic through copper-zinc to copper. The most characteristic feature of this horizon is the predominance of zinc and copper over lead, with zinc being more abundant than copper at Ridder and 2nd South-Western ore zones and copper being more abundant than zinc in Central ore zones. The proportion of copper mineralisation increases with depth, see Figure 5.8.

The other characteristic feature of Horizon II is a higher gold content than at the overlying Horizon I. Gold occurs in sulphide mineralisation and, more importantly, in distinctive narrow veins.

Two types of gold veins within the ore zones are known:

- In Type 1 the gold occurs in association with tennantite-tetrahedrite, galena, sphalerite, ankerite and quartz. These veins, referred to as gold-sulphide veins, are particularly abundant among copper-zinc mineralisation in the 2nd South-Western ore zones. The veins pass upwards into stockwork zones which extend into the gold-barite-polymetallic cupolas; and
- In Type 2, gold occurs in quartz. The vein fill consists predominantly of quartz and dolomite, with minor quantities of unevenly disseminated sulphides. These are particularly abundant in the 2nd South-Western Lode and the Bystrushinskaya ore zones. The veins contain 30-45g/t Au or more, particularly at levels 11-15, and despite small tonnages (typically in the 5,000-10,000t range) may carry as much as 15-20% of the total gold contained in the deposit. In recent years, gold-bearing quartz veins have been worked selectively and stockpiled for processing as high value flux ore.

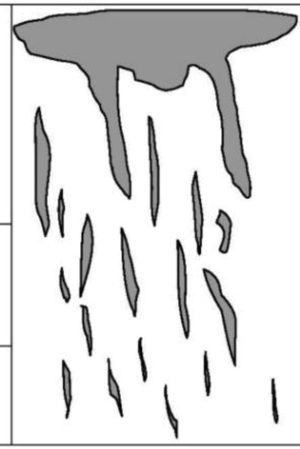
Stratigraphic Column	Mineralisation Horizon	Lodes at Ridder-Sokolny Deposit	Average Grades	Percentage Of Current Resource	MORPHOLOGY OF MINERALISED BODIES	Overall Form. Mining Status
Ilinskaya Fm (20-240m)						
Kriukovskaya Fm (250-650m)	Horizon I (lead-zinc)	Central, Ridder, 2nd Ridder, Zavodskaya, Pobeda, Promezhutochnaya, Belkina, Perspectivnaya, Kriukovskaya, 1st South-Western, 2nd South-Western, 3rd South-Western, Bystrushinskaya, North-Eastern, Filipovskoye prospect	6.2% ZnEq 3.0 g/t Au			Stockwork mineralisation, thick flat tabular and lenticular bodies passing downwards into steeply dipping vein zones. Now practically worked out.
	Horizon II (zinc-copper)	North-Eastern, deep levels of all main lode and flanks of the deposit	4.0% ZnEq 1.8 g/t Au	68		Numerous steeply dipping veins. Developed, currently being mined.
			4.4% ZnEq 1.1 g/t Au	20		Numerous small steeply dipping veins. Undeveloped.
Leninogorskaya Fm (20-350 m)	Horizon III (lead-zinc)	Dalnaya, Glubokaya, Novaya, 2nd North-Eastern	3.7% ZnEq 0.2 g/t Au	12		Small flat tabular bodies and steeply dipping veins. Undeveloped.
Zavodskaya Fm (200-400m)	Horizon IV (polymetallic)					

Figure 5.7: Stratigraphic Position

5.2.3 Description of the Ore Zones

Within the deposit, vein-impregnated, vein, and stockwork-vein ore zones can be distinguished. On the upper levels, flat-dipping hydrothermal-sedimentary, hydrothermal-metasomatic ores are developed, whilst with depth the hydrothermal stockwork-vein formations transform into more typical vein systems.

Each ore zone is characterised by the development of all structural-genetic types of ore at various quantitative ratios. Flat-dipping ores found within the upper levels had a wide aerial distribution over the deposit, but are now mined only on marginal parts of certain ore zones. Today flat-lying ore bodies remain within the Bystrushinskaya, 2nd South-Western and Pobeda ore zones, on the southern flank of Bystrushinskaya ore zone, as well as residual sections in the Perspectivnaya and Belnina lodes.

These ore zones are also characterised by small metasomatised vein ore bodies that often occur on the footwall side of the main flat-dipping ore body and subparallel to it. With distance from the main flat-dipping ore body veins, the flat-dipping ore bodies beneath the footwall become steeply dipping veins.

Currently within the deposit, the steeply dipping vein formations with the ore zones contain the great majority of reserves. These veins are aligned sub-parallel to main shear structures of the deposit and associated fractures.

Rich ores are represented by massive sulphide lenses and gold-bearing quartz veins. Gold-quartz veins with high gold grade (more than 16g/t Au) have been selectively mined and developed particularly in the South-Western and Bystrushinskaya ore zones, although the majority of these have now been mined. In individual cases, such veins occur in the Central and Pobeda ore zones. Gold-bearing quartz veins are typically 0.2-0.8m thick and as a rule occur concordantly with the host lead-zinc and copper ores, their down dip dimensions are up to 40m, and along the strike – 30-50m.

A brief description by ore zone is given below:

Ridderskaya ore zone is mainly represented by vein-like bodies, rarely by columnar and disk shaped ore shoots. These veins are predominantly made up of vein-impregnated sulphide lead-zinc ore, which is altered by complex and oxide variations in the southern part of the lode on the upper levels. In addition, in south-eastern part of the lode, copper ore vein bodies also occur.

2nd Ridderskaya ore zone is represented by sheet-like bodies and beds of hydrothermal sedimentary ores. Ore bodies are predominantly oriented along a strike to the north-west and have a variable south-westerly dip (35-75°). 2nd Ridderskaya ore zone borders on Ridderskaya lode and Intermediate area near "Boreholes 50-53" fault.

Intermediate ore zone area is made up of steeply-dipping vein ore bodies of north-western strike. Lead-zinc ores distributed in the western part of the ore zone dip to the south-west, and copper ores of eastern part of the ore zone dip near-vertically and steeply to the north-east, rarely to the south-west. According to its position, the area is intermediate between the Ridderskaya and Central ore zones. Here, sulphide lead-zinc ores are predominant, with their complex and oxide variations distributed on upper levels, and copper-sulphide ores at the deeper levels.

Centralny ore zone has ore bodies which have a lenticular, sheet-like and vein morphology. On the upper levels, ore bodies are represented by complex dendritic veins and veinlets of various orientation, intersections of multiple-aged veins are observed. At the lower horizons there is a transition from mesh-vein stockwork to oriented vein systems, ore bodies are represented by steeply-dipping veins with banded structure, the banding is conditioned by alternation of variously coloured quartz bands, lenses and spots of sulphides. On the lower levels where copper ores are developed the ore body shape is vein-like or wedge-like. The banded structure veins pinch out rapidly down dip, but along the strike they can be traced for long distances. Veins of the Central ore zone are oriented to the north-west and dip to the south-west along the "boreholes 50-53" fault, but around the fault of Nikolayevskaya shaft - to the north-east, and in the central part of the veins are vertical.

The *Pobeda* ore zone contains both flat-dipping and steeply-dipping ore bodies which can be lenticular, sheet-like, vein-like and disc shaped. The ore zone is made up of sulphide lead-zinc and copper ores. Sheet-like ore bodies consist of vein-impregnated ores. Veins are orientated to the north-west, but in central part near E-W trending veins are also developed.

Perspectivnaya ore zone is characterised by the development of flat-dipping sulphide lead-zinc ore bodies, typically disk-shaped ore bodies predominate, this shape is conditioned by development of short down dip and along the strike fractures filled with quartz and ore minerals. Veins are oriented from north-west to north-south, whilst near Nikolayevskaya shaft fault, ore bodies dip to the north-east, whereas on the north-eastern limb of the ore zone, ores occur horizontally.

Belkina ore zone is mainly represented by flat-dipping ore bodies of sheet-like, band-like and lenticular shape, rarely of disk shape. Only sulphide lead-zinc ores are developed in the ore zone. Ore bodies are made up of thin lenses and streaks and of massive sulfides. On the lower levels rare steeply-dipping thin veins and veinlets with short strike length are developed which are represented by quartz-ore material. Ore bodies are oriented to the north-west and dip to the north-east near Nikolayevskaya shaft fault and to the south-west around the "boreholes 158-153" fault.

The *1st South-Western* ore zone is represented by small vein-like ore bodies. The lode veins are oriented to the north-west and dip to the south-west.

The *2nd South-Western* ore zone is mainly represented by sulphide lead-zinc and copper vein ore bodies, which are typically thin; and have a vein-like shape, rarely columnar. Numerous veins are very thin, however, sometimes these can be traced for considerable distances both down-dip and along the strike. Ore bodies combine one or several veins with numerous sulphide veinlets. Ore bodies typically dip sub-vertically.

The *3rd South-Western* lode mainly consists of sulphide lead-zinc ores transforming to copper ores with depth. Vein-type ore bodies of sub-vertical dip and approximately E-W strike are similar to ore bodies of the 2nd South-Western lode in structural location, mineral content and higher gold grade.

Bystrushinskaya ore zone is mainly represented by steeply-dipping veins and crush zones. Veins and zones are oriented approximately E-W, with crush zones very often overlapping veins and vice versa. Textures are divided into brecciated, brecciform ore bodies filled with quartz, carbonate and sulphide minerals, and brecciated aleuropelites. On the upper levels, minor flat-dipping lenticular ore bodies are developed and represented by vein-impregnated mineralisation in quartz-barite and quartzite. More than half of ore reserves in the lode are represented by copper ores, the rest – by sulphide lead-zinc ores.

Within the *Southern flank of Bystrushinskaya* ore zone a large main flat-dipping ore body has been explored which contains small narrow steeply–dipping ore bodies located on the footwall side of the main body. Steeply–dipping veins are oriented approximately E-W and to the north-east and dip to the south-east.

Lodes of the 4th level of mineralisation (Novaya, the 2nd North-Eastern, Glubokaya, Dalnaya ore zones) are mainly represented by flat-dipping ore bodies occurring sub-concordantly with the contact of the Zavodskaya suite host rocks with Leninogorskaya suite. These lodes are represented by sulphide lead-zinc ores.

Novaya ore zone is represented by a series of small bodies (total 55) of which the main one has an isometric shape and cross-sectional size of 300m with an average thickness of 8.41m. The strike of the ore body is approximately E-W, whose dip is nearly horizontal.

The *2nd North-Eastern* ore zone is represented by a lenticular body 600m long, approximately 300-500m wide and 1 to 19m thick and by 10 small 50-60m lenses occurring nearly horizontally. Ore bodies are oriented to the north-west.

Glubokaya ore zone is represented by one large, some middle-size and some small ore body (a total of 26). The main ore body has lense-sheet-like shape and is isometric about 300m long and average thickness of 16.0m. It is orientated approximately E-W, flat-dipping (15-20°) to the south-west and the west.

Dalnaya ore zone is represented by four quite large flat-dipping ore bodies as well as by 49 small lenses. The main ore body represents two lenses of irregular shape which have a lenticular section. The thickness of the main ore bodies is between 1.0 and 35.0m, average is 13.63m, with dimensions of 600x400m. Ore bodies are oriented approximately E-W, sub-E-W, and nearly horizontal. Dalnaya lode mainly consists of sulphide lead-zinc ores, whilst copper ores are rarely developed. Dalnaya lode is more saturated with mineralisation than other lodes of the 4th level, having a greater content of massive sulphide (totally 30 ore bodies) are observed here, one of which is characterised by higher gold grade.

North-Eastern ore zone is represented by 19 small steeply-dipping ore bodies of vein-impregnated sulphide lead-zinc and copper ores. Ore bodies are elongated to the north-north-nest and dip to the north-east. The lode predominantly consists of sulphide lead-zinc ores, copper ores are of minor significance. Only 3 lenses of massive sulphides with enriched gold are observed in the lode.

Philippovskoye ore occurrence is represented by 11 steeply-dipping small vein oxide and complex copper ore bodies and by 5 small oxide lead-zinc ore bodies with extension to the north-north-west. No rich ore shoots have been discovered.

Krukovskaya ore zone consists of a combination of flat-dipping ore bodies and steeply-dipping vein-stockwork mineralisation of the first level. The ores are represented by oxide and complex lead-zinc variations.

5.2.4 Mineralogy of the Veins

Hydrothermal-sedimentary Ores are represented by brecciated and scaly lead-zinc polymetallic ores known only in the 2nd Ridderskaya ore zone. The breccia-like ores prevail near the set of "faults 50-53", where thick layers (up to 4m) are split up by intercalations of aleuropelites. With distance from the faults, the layer thickness and size of clasts decrease, and at a distance of 30-50m from the fault, the scaly variations prevail. The ore body is represented here by rhythmically alternating intercalations of finely disseminated and microfragmental ores from several centimeters to 15cm thick with thicker layers (up to 0.5m) of aleurolites. On the ore body flanks the aleuropelites contain only spots and blotches of fine-grained pyrite and rarely – inclusions of other sulphides. The scaly lead-zinc ores are fine-grained, formed of a galena-pyrite-sphalerite-dolomitic assemblage.

Hydrothermal-metasomatic Ores and Sedimentation in Cleavage cavities

Massive and vein-impregnated polymetallic ores of the level I are composed of chalcopyrite- galenite-pyrite-sphalerrite mineral assemblage. The main ore mineral is sphalerite; non-metallic minerals are quartz and dolomite. The rare elements in these ores are fahlore, tennantite, gold, silver, arsenopyrite, calaverite, cellular pyrite, stephanite, enargite, hematite. They differ from scaly ores only by their higher copper content.

Hydrothermal, mainly Vein Ores these are zinc-copper ores of the level II, which consist of copper, copper-zinc and polymetallic variations separated by gradational contacts. Unlike the level I ores, these ores are characterised by preponderance of zinc and copper over lead and by their coarse-grained structure resulting from filling of crushed zones by the metal minerals. The veins have a coarse-ribboned, symmetrical, rarely asymmetrical texture. The ores in the crushed zones are spotty, brecciated, rarely impregnated, lense-like-impregnated and massive.

Metamorphogenic-hydrothermal SulphIdic-sericitic Ores of lead-zinc composition are impregnated or massive, of medium-grain size. They consist of pyrite-sphalerite-clinochlore-phengite mineral assemblage. The porphyroblastic textures of ores are typical resulting from cementing of relatively large rounded (15mm) porphyroblasts of dark sphalerite with a phengite and clinochlore assemblage and disseminated small grains of pyrite, chalcopyrite and galena. The ores tend to occur in the zones of later displacement that surround the quartz-barite domes and intersect the bodies of massive and vein-impregnated polymetallic ores.

Oxidized and Mixed Ores are developed in areas lying beneath quaternary deposits and have restrictedly spread. The oxidation zone varies from 30-100m thick. The subzone of the mixed/transitional ores occurs in the lower part of the oxidation zone and its thickness is up to several meters.

5.2.5 Mineralisation

The most distinguishing feature of the Ridder-Sokolniy mineralisation is the dominance of zinc over copper and lead. The average ratio of Cu:Pb:Zn is 0.2:1:2.4, except in copper and zinc-copper veins where copper predominates. The other characteristic feature is the presence of substantial quantities of gold and, to a lesser degree, silver. A list of primary minerals identified at the deposit is given in Table 5.5 below.

Table 5.5: Primary Minerals

Abundance	Metalliferous Minerals	Gangue Minerals
Main	Sphalerite, pyrite, galena	Barite, graphite, dolomite, quartz, sericite, phengite
Abundant	Tennantite-tetrahedrite, gold, silver, electrum, fine-grained pyrite	Albite, ankerite, hydrophengite, calcite, prochlorite, ripidolite, chlorite
Rare	Altayite, hematite, hessite, marcasite, molibdenite, pearcite, rutile	Oligoclase, gypsum, cleavelandite, chlinochlore, corundophyllite, magnesite, muscovite, carbonaceous matter, fluorite
Accessory	Acanthite, arsenopyrite, bismuth, bismuthite, calaverite (?), petzite, stephanite, tellurobismuthite, enargite, bornite	Bleinerite, penninite, siderite, talc

Mineral associations found at Horizons III and IV differs from those at Horizons I and II. Horizon III contains galena-pyrite-sphalerite mineralisation, in which sphalerite and pyrite each form 5-8% by volume, galena forms 1% and chalcopyrite 0.5-1.5%. The gold content is typically in the range of 0.2-0.3g/t Au. Gangue minerals include sericite, quartz, calcite, dolomite, chlorite, potassic feldspar, plagioclase, siderite, barite and fluorite. Horizon IV contains chalcopyrite-galena-sphalerite mineralisation in calcite-quartz veinlets.

5.3 Exploration Works

5.3.1 Sample Collection

Currently mined veins generally do not exceed 50m in strike length. A grid of 25m along strike by 10m to 25m up and down dip is normally required to delineate such veins in order to classify the estimated reserves as C_1 category (under the statutory requirement of mines to report reserves under GKZ (RK) protocols). Detailed pre-production drilling and channel sampling to upgrade the reserves to the B category is generally carried out on a 12.5m by 12.5m or a closer spacing if necessary.

The following sequence has been developed through many years of experience:

- Preliminary exploration by deep diamond core drilling from surface on a 50m by 50m grid (using directional wedging), which is considered optimal to delineate a C_2 category resource; now carried out only to explore the flanks of the deposit;
- Detailed underground exploration with geological mapping and sampling of underground openings and drilling on grids varying from 50m by 25m to 25m by 25m; generally sufficient to delineate C_1 category resource blocks; and
- Additional pre-stoping exploration with channel sampling and drilling to achieve a 12.5m by 12.5m or closer sample spacing required to upgrade the C_1 reserve to the B category.

Two different schemes of underground development and detailed exploration are in use depending on the geometry of mineralised zones. They are:

- Crosscut development scheme with crosscuts across the strike of mineralised structures at 50m spacing and fans of exploration drill holes at 25m intervals along the crosscuts to achieve a 25m by 25m sampling grid; and
- Drive development scheme with drives along the strike of mineralisation and fans of drill holes at 25m intervals. The spacing between the drives varies from 100m to 150m.

All underground openings and drill site locations are surveyed with instrumental methods. Drill hole trajectories are monitored at regular intervals. Placement of drill core markers is checked against grades recorded on roentgen-radioactive logs.

Channel samples are cut along one side of each crosscut exposing sulphide mineralisation. Channel samples are 2m in length or less if required. Drives along the strike are sampled by horizontal channel taken from advancing faces at intervals of 6m or 12.5m. Short crosscuts are developed at 12.5m intervals to intersect the whole width of mineralisation. Raises are sampled by horizontal channels at 4-6m intervals from the same wall or from alternate opposing walls. All channel samples are 10cm wide and 1.5cm deep.

Drill core from underground holes is divided into samples of variable length to account for geological features. The whole core is sent for sample preparation. External drill bit diameters are 46mm or 59mm, producing 42mm or 30mm diameter core respectively.

5.3.2 Sample Preparation

Sample preparation is conducted at the laboratory of Ridder Mining and Concentrating Complex. The scheme is based on the Richard-Czeczott formula $Q=kd^a$, where Q is the minimum sample quantity at a given stage of volume reduction, d is the diameter of the largest fragments defined as the screen size that retains the largest 5% of the mass, k is a coefficient dependent on the distribution irregularity of the mineral of interest and a is a coefficient related to the roundness of mineral grains (generally approximately 2).

The coefficient k is the key parameter. In general terms, the lower coefficient k is, the better it accounts for the erratic distribution of minerals. In this instance, the k parameter of 0.1 has been selected after a series of experiments using schemes with k equal to 0.4, 0.3, 0.2, 0.15 and 0.1.

In accordance with the adopted scheme, samples are crushed to less than 1mm and then divided to obtain an approximately 1kg sub-sample which is pulverised to minus 0.074mm. The scheme is considered to be appropriate for sub-sampling of samples with variable contents of fine-grained gold.

5.3.3 Sample Analysis

Analyses are performed at the laboratory of Ridder Mining and Concentrating Complex and include semi-quantitative multi-element spectral analysis (to select samples for accurate analysis), chemical analyses, phase analyses and fire assaying. External control analyses are performed by CL Evrika (former Central Laboratory of PGO Vostkazgeologia), AO Topaz and VNIItsvetmet, all at Ust-Kamenogorsk. Each laboratory is accredited in Kazakhstan.

5.3.4 Bulk Densities

Resource tonnages are based on tonnage factors calculated from regression formulas which link dry bulk densities to combined copper, lead and zinc grades. Two general regression formulas are in use; one for polymetallic mineralisation and one for copper mineralisation (see Table 5.6 below). Regression coefficients in the general formula for the polymetallic mineralisation are based on 554 bulk density determinations on samples taken from lodes in the Central and Western blocks. Regression coefficients for the general formula for copper mineralisation are based on 140 determinations, also taken from the Central and Western blocks. A separate regression formula has been developed for polymetallic mineralisation of the Zavodskaya Lode.

Volumes are converted to tonnages by applying densities based on the general regression formulas for polymetallic mineralisation and copper mineralisation as given in Table 5.6 below.

Table 5.6: Bulk Density versus Grade (from Geoincentr, 2001)

Deposit, Block, Lode	Number of Samples	Correlation Coefficient	Y axis intercept	Regression coefficients		
				Cu	Pb	Zn
Copper Mineralisation						
Ridder-Sokolniy Deposit	140	0.948	2.692008	0.044040	0.010715	0.012510
Western Block	62	0.874	2.697247	0.035039	0.052520	0.002671
Central Block	78	0.962	2.700849	0.044836	0.058719	0.007287
Lead-Zinc Mineralisation						
Ridder-Sokolniy Deposit	554	0.937	2.669419	0.026710	0.034540	0.018910
Western Block	203	0.927	2.673783	0.035749	0.024483	0.015851
2nd South-Western Lode	135	0.952	2.676467	0.039887	0.022098	0.017107
3rd South-Western Lode	26	0.876	2.670899	0.033712	0.037627	0.009317
Bystrushinskaya Lode	42	0.809	2.670790	0.019446	0.026259	0.015513
Central Block	249	0.951	2.668990	0.028714	0.041528	0.018835
Central Lode	128	0.953	2.670613	0.036217	0.030463	0.017473
Belkina Lode	54	0.978	2.673930	0.053992	0.037482	0.022481
Pobeda Lode	67	0.867	2.670876	0.030090	0.040783	0.019444
Zavodskaya Lode	102	0.934	2.679540	0.019379	0.024205	0.015695
Dalnaya Lode	110		2.732580	0.04389	0.01185	0.011020

5.4 QA/QC Analysis for Ridder-Sokolniy and Satellite Areas

WAI has carried out QAQC analysis on a selected portion of the available data from the Kazzinc assets. The data can be treated as being representative of the sample handling and assaying procedures of the company's Ridder assets and the analysis of these is presented here.

5.4.1 Internal Checks

Internal duplicate checks were carried out on 411 Au-Ag samples from the Solovievskiy licence area during 2010. Statistical analysis has been carried out on the data.

Overall the plots demonstrate that the internal duplicate checks have been carried out to a satisfactory standard. The results for the Ag internal checks give similar results.

For the Cu, Zn and Pb assays 270 samples were selected for internal duplicate tests from a range of assay values from the Solovievskiy licence area during the 2010 period.

5.4.2 External Checks

217 of the Au samples and 209 of the Ag samples submitted for internal analysis were also submitted to Ridder-Sokolniy laboratory for external tests. Figure 5.8 shows scatter plots for these external assays and show excellent correlation.

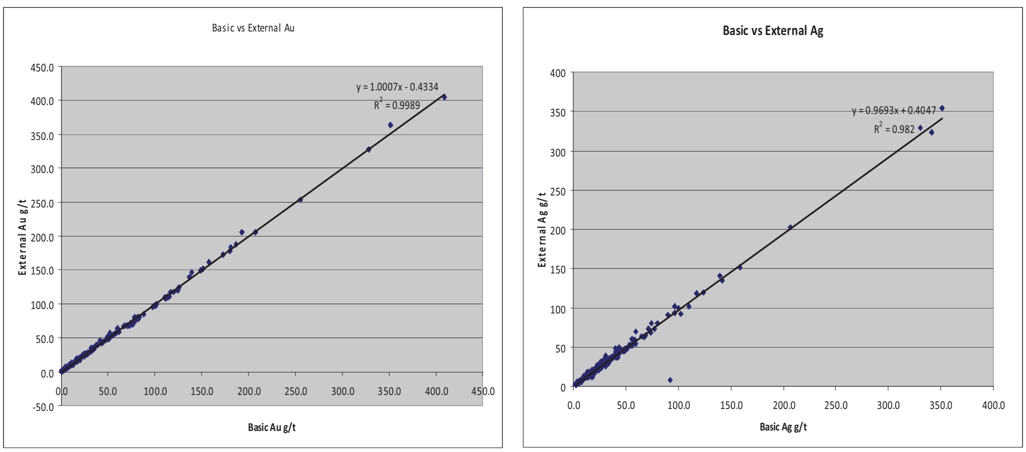


Figure 5.8: Scatter Plots of External Au and Ag Check Assays

Similarly the Q-Q plots for both elements show good linearity.

Overall the results give WAI confidence that the sample preparation and analysis across the Kazzinc assets has been carried out to a high standard.

5.5 Current Mineral Resource Estimates

Mineral Resources for individual Ridder-Sokolniy ore zones have been calculated in accordance with the guidelines of the JORC Code (2004) and are shown in the tables together with a summary below.

5.5.1 Centralny

The Centralny resource model was produced by WAI. A summary of the resource estimate is shown in Table 5.7 below.

Table 5.7: Centralny Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore and 0.6% CuEQV for Cu Ore
(In Accordance with the guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	20.09	1.00	648,768	8.43	5,446,956	0.9	18,020	0.31	61480	0.8	160,800
Indicated	36.21	1.23	1,437,071	8.46	9,847,050	0.44	15,950	0.36	130750	0.85	308,310
Measured + Indicated	56.30	1.15	2,085,839	8.45	15,294,006	0.60	33,970	0.34	192,230	0.83	469,110
Inferred	1.53	0.83	40,763	3.74	183,365	0.45	690	0.15	2320	0.71	10,880

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.2 Bystrushinskoe

The Bystrushinskoe resource model was produced by WAI. A summary of the resource estimate is shown in Table 5.8 below.

Table 5.8: Bystrushinskoe Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore and 0.6% CuEQV for Cu Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
Measured + Indicated	16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
Inferred	0.04	1.53	1,768	3.44	3,922	0.18	63	0.15	53	0.33	118

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.3 North Bystrushinskoe

The North Bystrushinskoe resource model was produced by WAI. A summary of the resource estimate is shown in Table 5.9 below.

Table 5.9: North Bystrushinskoe Resource Estimate (WAI 01.01.11) Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore and 0.6% CuEQV for Cu Ore (In Accordance with the Guidelines of the JORC Code (2004))											
Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
Measured + Indicated	1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
Inferred	0.09	2.46	7,106	6.90	19,997	0.86	776	0.29	264	1.03	925

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.4 Belkina

The Belkina resource model was produced by KMC and audited by WAI. A summary of the resource estimate is shown in Table 5.10 below.

Table 5.10: Belkina Resource Estimate (WAI 01.01.11) Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore (In Accordance with the Guidelines of the JORC Code (2004))											
Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	3.76	1.26	152,331	18.98	2,294,630	0.09	3,384	0.47	17,670	0.98	36,850
Indicated	4.77	1.11	170,126	16.85	2,582,538	0.08	3,814	0.41	19,550	0.85	40,520
Measured + Indicated	8.53	1.18	322,456	17.79	4,877,168	0.08	7,200	0.44	37,220	0.91	77,370
Inferred	0.52	0.88	14,669	23.42	390,393	0.28	1,450	0.32	1,660	0.90	4,670

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.5 Perspectivnaya

The Perspectivnaya resource model was produced by WAI. A summary of the resource estimate is shown in Table 5.11 below.

Table 5.11: Perspectivnaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	4.59	1.55	228,476	16.01	2,359,940	0.13	5,960	0.57	26,130	1.07	49,060
Indicated	3.94	1.36	172,076	16.79	2,124,381	0.13	5,120	0.53	20,860	1.03	40,530
Measured + Indicated	8.52	1.46	400,552	16.37	4,484,321	0.13	11,080	0.55	46,990	1.05	89,590
Inferred	2.03	1.19	77,607	12.08	787,806	0.14	2,840	0.49	9,940	0.95	19,270

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.6 Sokolok

The Sokolok resource model was produced by KMC and audited by WAI. A summary of the resource estimate is shown in Table 5.12 below.

Table 5.12: Sokolok Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
Measured + Indicated	0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
Inferred	0.37	1.40	16,715	15.27	182,315	0.14	520	1.48	5,550	1.98	7,350

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.7 New Glubokaya

The New Glubokaya resource model was produced by KMC and audited by WAI. A summary of the resource estimate is shown in Table 5.13 below.

Table 5.13: New Glubokaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
Measured + Indicated	0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
Inferred	0.07	0.14	314	2.68	6,004	0.31	220	0.74	520	2.10	1,460

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.8 Glubokaya

The Glubokaya resource model was produced by KMC and audited by WAI. A summary of the resource estimate is shown in Table 5.14 below.

Table 5.14: Glubokaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	-	-	-	-	-	-	-	-	-	-	-
Inferred	0.51	0.10	1,640	1.48	24,277	0.08	410	0.83	4,230	1.91	9,740

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.9 Dalnaya

The Dalnaya resource model was produced by KMC and audited by WAI. A summary of the resource estimate is shown in Table 5.15 below.

Table 5.15: Dalnaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	-	-	-	-	-	-	-	-	-	-	-
Indicated	3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
Measured + Indicated	3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
Inferred	1.16	0.18	6,855	3.18	118,984	0.51	5,958	0.40	4,620	2.07	24,083

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

5.5.10 Zavodskaya

The Zavodskaya resource model was produced by KMC and audited by WAI. A summary of the resource estimate is shown in Table 5.16 below.

Table 5.16: Zavodskaya Resource Estimate (WAI 01.01.11)
Total In-Situ Resources At COG of 1.7% ZnEQV for PbZn Ore
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Measured	1.78	0.82	46,922	22.60	1,293,223	0.12	2,111	0.79	14,105	1.53	27,187
Indicated	0.51	0.97	15,816	21.62	352,524	0.10	509	0.67	3,397	1.29	6,523
Measured + Indicated	2.29	0.85	62,739	22.38	1,645,747	0.12	2,620	0.76	17,502	1.48	33,710
Inferred	0.33	1.00	10,567	21.10	222,961	0.14	459	0.60	1,958	1.21	3,991

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

Table 5.17: Kazzinc Mineral Resources Ridder-Sokolniy Mine (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004)

Deposit	Resources	Cut Off Grade	Date	Tonnes (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
					Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Centralniy	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01.01.2011	20.09	1.00	648,768	8.43	5,446,956	0.9	18,020	0.31	6,1480	0.8	160,800
	Indicated			36.21	1.23	1,437,071	8.46	9,847,050	0.44	15,950	0.36	130,750	0.85	308,310
	Measured + Indicated			56.30	1.15	2,085,839	8.45	15,294,006	0.60	33,970	0.34	192,230	0.83	469,110
	Inferred			1.53	0.83	40,763	3.74	183,365	0.45	690	0.15	2,320	0.71	10,880
	Total			57.83	1.14	2,126,602	8.33	15,477,371	0.60	34,660	0.34	194,550	0.83	479,990
Bystrushinskoe	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01.01.2011	-	-	-	-	-	-	-	-	-	-	-
	Indicated			16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
	Measured + Indicated			16.98	2.72	1,486,264	12.29	6,711,147	0.28	48,064	0.42	71,510	0.90	152,812
	Inferred			0.04	1.53	1,768	3.44	3,922	0.18	63	0.15	53	0.33	118
	Total			17.02	2.72	1,488,032	12.28	6,715,069	0.28	48,127	0.42	71,563	0.90	152,930
Bystrushinskoe North Flank	Measured	1.7% ZnEq for PbZn ore and 0.6% CuEq for Cu ore	01.01.2011	-	-	-	-	-	-	-	-	-	-	-
	Indicated			1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
	Measured + Indicated			1.75	1.26	70,796	5.63	317,328	1.02	17,890	0.29	5,140	1.08	18,868
	Inferred			0.09	2.46	7,106	6.90	19,997	0.86	776	0.29	264	1.03	925
	Total			1.84	1.31	77,902	5.69	337,325	1.01	18,666	0.29	5,404	1.07	19,793
Belkina_RSM_WA	Measured	1.7% ZnEq	01.01.2011	3.76	1.26	152,331	18.98	2,294,630	0.09	3,384	0.47	17,670	0.98	36,850
	Indicated			4.77	1.11	170,126	16.85	2,582,538	0.08	3,814	0.41	19,550	0.85	40,520
	Measured + Indicated			8.53	1.18	322,456	17.79	4,877,168	0.08	7,200	0.44	37,220	0.91	77,370
	Inferred			0.52	0.88	14,669	23.42	390,393	0.28	1,450	0.32	1,660	0.90	4,670
	Total			9.05	1.16	337,125	18.11	5,267,561	0.10	8,650	0.43	38,880	0.91	82,040
Perspectivnaya	Measured	1.7% ZnEq	01.01.2011	4.59	1.55	228,476	16.01	2,359,940	0.13	5,960	0.57	26,130	1.07	49,060
	Indicated			3.94	1.36	172,076	16.79	2,124,381	0.13	5,120	0.53	20,860	1.03	40,530
	Measured + Indicated			8.52	1.46	400,552	16.37	4,484,321	0.13	11,080	0.55	46,990	1.05	89,590
	Inferred			2.03	1.19	77,607	12.08	787,806	0.14	2,840	0.49	9,940	0.95	19,270
	Total			10.55	1.41	478,159	15.55	5,272,127	0.13	13,920	0.54	56,930	1.03	108,860

Sokolok	**Measured**	1.7% ZnEq	01.01.2011	-	-	-	-	-	-	-	-	-	-	-
	Indicated			0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
	Measured + Indicated			0.29	1.36	12,839	120.84	1,140,808	0.13	380	1.00	2,940	1.89	5,550
	Inferred			0.37	1.40	16,715	15.27	182,315	0.14	520	1.48	5,500	1.98	7,350
	Total			0.67	1.38	29,554	61.94	1,323,123	0.14	900	1.27	8,440	1.94	12,900
New Glubokaya	**Measured**	1.7% ZnEq	01.01.2011	-	-	-	-	-	-	-	-	-	-	-
	Indicated			0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
	Measured + Indicated			0.32	0.18	1,867	2.48	25,723	0.06	190	0.67	2,160	1.80	5,810
	Inferred			0.07	0.14	314	2.68	6,004	0.31	220	0.74	520	2.10	1,460
	Total			0.39	0.17	2,181	2.51	31,727	0.10	410	0.68	2,680	1.85	7,270
Glubokaya	**Measured**	1.7% ZnEq	01.01.2011	-	-	-	-	-	-	-	-	-	-	-
	Indicated			-	-	-	-	-	-	-	-	-	-	-
	Measured + Indicated			-	-	-	-	-	-	-	-	-	-	-
	Inferred			0.51	0.10	1,640	1.48	24,277	0.08	410	0.83	4,230	1.91	9,740
	Total			0.51	0.10	1,640	1.48	24,277	0.08	410	0.83	4,230	1.91	9,740
Dalnaya	**Measured**	1.7% ZnEq	01.01.2011	-	-	-	-	-	-	-	-	-	-	-
	Indicated			3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
	Measured + Indicated			3.14	0.31	31,104	3.49	352,551	0.68	21,281	0.22	6,975	2.31	72,368
	Inferred			1.16	0.18	6,855	3.18	118,984	0.51	5,958	0.40	4,620	2.07	24,083
	Total			4.30	0.27	37,959	3.41	471,535	0.63	27,239	0.27	11,595	2.24	96,451
Zavodskaya	**Measured**	1.7% ZnEq	01.01.2011	1.78	0.82	46,922	22.60	1,293,223	0.12	2,111	0.79	14,105	1.53	27,187
	Indicated			0.51	0.97	15,816	21.62	352,524	0.10	509	0.67	3,397	1.29	6,523
	Measured + Indicated			2.29	0.85	62,739	22.38	1,645,747	0.12	2,620	0.76	17,502	1.48	33,710
	Inferred			0.33	1.00	10,567	21.10	222,961	0.14	459	0.60	1,958	1.21	3,991
	Total			2.62	0.87	73,305	22.22	1,868,709	0.12	3,079	0.74	19,460	1.44	37,701
Ridder-Sokolniy Mine	**Measured**		01.01.2011	30.21	1.11	1,076,497	11.73	11,394,749	0.64	29,475	0.40	119,385	0.91	273,897
	Indicated			67.91	1.55	3,397,959	10.74	23,454,050	0.38	113,198	0.40	269,542	0.95	645,031
	Measured + Indicated			98.12	1.42	4,474,456	11.05	34,848,799	0.46	142,675	0.40	388,927	0.94	918,928
	Inferred			6.64	0.83	178,004	9.09	1,940,024	0.29	13,386	0.59	39,365	1.12	74,187
	Total			104.76	1.38	4,652,459	10.92	36,788,824	0.45	156,061	0.41	428,292	0.95	993,115

5.6 Mining

5.6.1 Introduction

Ridder-Sokolniy has a long history. The deposit was discovered in 1786, the underground operations first started in 1791. Originally there were three separate mines: '40 Years of VLKSM', 'Ridder' and 'Leninogorskiy'. The three mines were eventually joined together into one operation, as they needed a combined access to ore.

5.6.2 Historical Mining

Approximately 181.6Mt of ore has been mined since the beginning of operations. Some historical highlights are:

- 1811 – Krukovsky deposit was discovered;
- 1817 – Filippovsky deposit was discovered;
- 1820 – Sokolniy deposit was discovered;
- 1900 – The Austrian concession "Tury-Taksisa" gained the deposit development rights;
- 1914 – The English firm of L. Urkwart gained the deposit development rights;
- 1918 - The concession was terminated, mines were flooded;
- 1925 - Reconstruction and exploration of Ridder deposits was started; and
- 1930 – Intensive development of Sokolniy deposit.

Mining operations have been conducted in the 17 major ore zones till present day. The future plans are to develop the eastern flank of the Perspectivnaya Lode at Bakhrushinskoy deposit.

5.6.3 GKZ Resources and Reserves

The total resources of approximately 23Mt are currently accounted for by State balance (as of 01 January 2010). A detailed resource statement, approved by GKZ (RK) is given in Table 5.18 below.

Table 5.18: Ridder-Sokolniy GKZ-Approved Resources & Reserves as of 01.01.2010			
	Category	**On-Balance**	**Off-Balance**
Gold-Polymetalic Resources	B	3,643.3	603.0
		6,338.9	353.3
	C1	19,339.6	15,513.7
		24,704.4	11,081.1
1.35	B+C1	22,982.9	16,116.7
		31,043.3	11,434.4
0.62	C2	20,729.1	11,645.4
		12,837.4	12,533.4
Inclusive of Undeveloped Resources	B	-	-
	C1	4,007.6	938.2
0.25		997.0	222.3
	B+C1	4,007.6	938.2
0.28		997.0	222.3
	C2	15,650.3	8,661.3
		4,334.9	10,290.4
Inclusive of Developed Resources	B	3,643.3	603.0
		6,338.9	353.3
	C1	15,332.0	14,575.5
1.58		23,707.4	10,858.8
	B+C1	18,975.3	15,178.5
1.67		30,046.3	11,212.1
	C2	5,078.8	2,984.1
		8,502.5	2,243.0
Included into Developed Resources: Cu ore	B	328.4	34.4
		1,172.9	27.0
	C1	3,143.8	1,486.0
1.14		2,786.1	618.8
	B+C1	3,472.2	1,520.4
0.94		3,959.0	645.8
	C2	478.2	255.7
		447.3	106.1
Included into Developed Resources: Pb-Zn ore	B	3,314.9	568.6
		5,166.0	326.3
	C1	12,188.2	13,089.5
1.68		20,921.3	10,240.0
	B+C1	15,503.1	13,658.1
1.75		26,087.3	10,566.3
	C2	4,600.6	2,728.4
		8,055.2	2,136.9

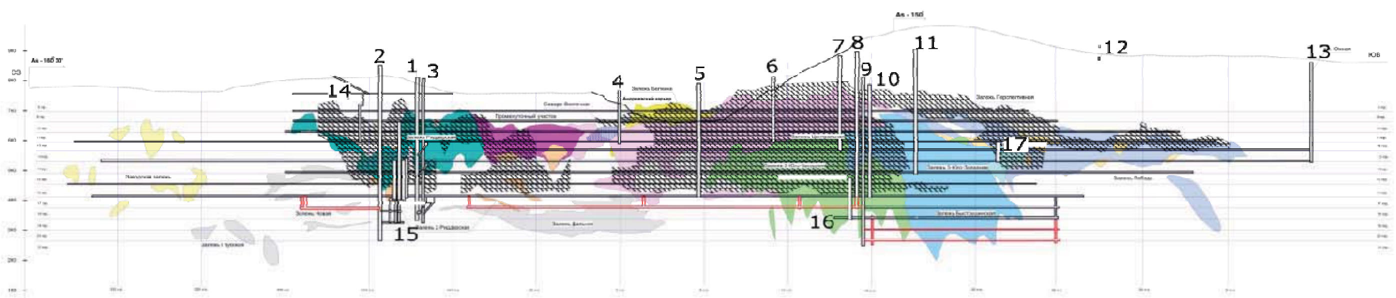
Historically, the mine has increased its resources by the same amount as mined by exploring adjacent areas. This programme remains in place, but the resource grade is progressively lower because the resource increase is less than material mined.

5.6.4 Mine Design and Current Mining Activities

Originally, the operations were worked as three separate units. The mine has an extensively developed underground infrastructure. Mining activities take place on 11 haulage levels and in 11 different ore zones, which are accessed by 12 shafts (10 in operation), to a maximum depth of 460m, hoisting ore and waste, men and materials.

The main hoisting shafts are Skipovaya (for Pb-Zn ore) and Novaya (for Cu ore and development waste). The Skipovaya shaft is equipped with two 7.5m^3 skips, which can be loaded from 11, 13 and 16 levels with a maximum weight capacity of 13,300kg. The Novaya shaft has two 4.5m^3 skips, which has loading facilities on the same levels.

Bystrushinskaya, Novaya and Andreevskaya are shafts used for men hoisting. A long section showing the mined out areas (hatched), shafts and ore zones (differentiated by colour) is given in Figure 5.9 below.



Ridderskaya	Perspectivnaya
2nd Ridderskaya	2nd South-West
Promezhutochnaya	Bystrushinskaya
Tsentralnaya	Zavodskaya
Belkina	3rd South-West
Pobeda	Unexploited Deposits (Novaya, Dalnaya, Glubokaya, North-East)

Mined Out Areas are Shown Hatched.

Shafts: 1 – Skipovaya-1; 2 – Skipovaya-2; 3 –Novaya; 4 – Andreevskaya; 5 – Belkina-1; 6 – Shaft No2; 7 – Ventilation; 8 – Shaft No3; 9 – Sokol; 10 – Bystrushinskaya; 11 – Belkina-2; 12 – Ventilation No2; 13 – Yuzhnaya; 14 – Slepaya; 15 – Slepaya-1; 16 – Slepaya Bystrushinskaya; 17 - Cage

Figure 5.9: Long Section of Ridder-Sokolniy

Figure 5.9 also show the future development in red. This development work targets to access levels 18-20. A detailed design for mining of the bottom levels is now being undertaken. Some preliminary assessment has shown that a conveyor decline, delivering ore to main haulage level 16 provides better economics in comparison to trackless haulage. The conveyor option is considered as preferred.

Currently, main mining activities are concentrated on levels 11 to 16, with 16 level being the major production haulage level. Approximately 70% of the ore is hoisted from 16 level; 20% from 11 level; the remaining 10% from 13 level. The main mining area is currently focused on 14 and 15 levels.

5.6.5 Mining Schedule

The mine schedule is based on short term (monthly, annually), medium term (3 years) and long term mine planning (up until year 2035). WAI has been provided and reviewed both short and long term schedules, highlighting areas where future production is going to take place.

The annual production schedule is designed at the end of each year and highlights exactly which blocks are to be mined and the amount of ore expected from these blocks together with metal grades. The mining schedule also contains information on development and stope preparation requirements for the year. WAI has reviewed the mining schedule for 2011 which is summarised in Table 5.19 below.

Table 5.19: Summary Mine Schedule for 2011

Deposit	Ore	Mining Method* (% of total ore)			Stope Access Works		Stope Preparation Works		Exploration		Blasthole Drilling	Subcontractors Stope Access Works		Subcontractors Stope Preparation Works	
	(kt)	S1	B	S2	m	m^3	m	m^3	m	m^3	m	m	m^3	m	m^3
Tsentralnaya	160	53	47	-	810	4,900	1,200	7,200	-	-	75,080	125	500	675	3,700
Ridder	96	-	100	-	125	500	630	4,380	-	-	21,100	-	-	-	-
Pobeda	378	100	-		1,240	8,260	2,000	13,780	600	2,400	62,500	1,455	7,310	1,720	9,100
Belkina	76	58	-	42	125	500	580	4,100	-	-	11,000	-	-	-	-
Perspectivnaya	102	100	-	-	325	1,600	930	5,900	150	600	18,500	50	300	315	1,200
2ndSouth-West	254	58	32	9	2,975	16,900	5,350	34,600	385	1,540	87,000	1,150	4,700	1,500	6,790
3rdSouth-West	127	100	-	-	770	4,750	1,710	14,600	705	2,820	41,800	-	-	-	-
Bystrushinskaya	821	-	100	-	4,780	44,410	6,650	59,320	3,660	14,640	80,090	-	-	-	-
TOTAL	2,125	44	53	3	11,150	81,820	19,050	143,880	5,500	22,000	397,070	2,780	12,810	4,210	20,790

The schedule indicates that 53% of the ore will be mined using sublevel open stoping with backfill). A description of the employed mining methods is given below.

The long-term mining plan has been designed to 2035 due to the active exploration programmes in place, this will be subject to further changes and therefore not considered within this report.

5.6.6 Mining Methods

Most of the production is performed using hand-held production equipment and scrapers. The main haulage equipment is tracked transport. Trackless methods have been recently introduced, and are focused in the Pobeda ore zone which contains high-grade thin veined gold zones in the southern part of the mine. It is the company plan to implement 30% trackless mining machinery (especially in development and haulage) in 2011 and to increase trackless machinery utilisation to 50% of overall production by 2013.

5.6.6.1 Sublevel Open Stoping

The majority of the ore is extracted using the Sub-Level Open Stoping (SLOS) method or Open Stoping method, which are relatively similar. Figure 5.10 below illustrates SLOS system. SLOS is used in steeply dipping orebodies of more than 50° and where the thickness varies from 3-15m. Ore in the blocks have a medium to high grade. The average block parameters are as follows:

- Length – 50m;
- Width – defined by orebody thickness; 6.1 on average; and
- Height – 40m.

Open stoping is used for thicker orebodies, where average thickness reaches 12.4m.

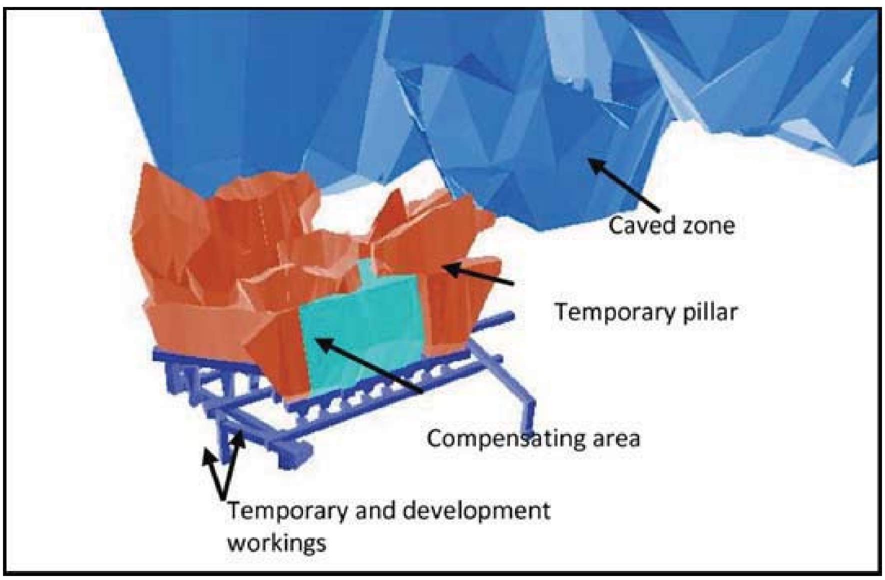


Figure 5.10: Sub-Level Open Stoping: 3D Image

The stopes are split into two sublevels, each of them having a haulage drive and access crosscut. There is also a scraping drift located adjacent to the footwall, ventilation and ladderway raises, and orepasses arranged at each sub-level. Blasthole fans are drilled from a dedicated drift. After blasting, ore is gravity fed into chutes which have been installed along the length of the orebody. A longitudinal retreat method of extraction is used.

The top sublevels are extracted 10-15m in advance of the bottom sublevels. Blasthole fans are charged and fired after all of the ore from previous cycle has been mucked. Kazzincmash drill rigs LPS-3U are used for the blastholes and they are charged using a ZP-12 pneumatic charger. Ore is mucked by 55kW scrapers 55LS-2SM.

After a stope has been mined, temporary pillars are removed and the stope is left unfilled. This leads to caving of rock above the stope into it, caving in many cases accretes to surface. Prior to mining, any area, potentially affected by caving, is thoroughly assessed and checked for the presence of buildings or constructions and appropriate safety pillars are designed.

5.6.6.2 Sublevel Open Stoping with Backfill

The method is similar to SLOS as shown Figure 5.11 below, but uses backfill. The ore is mined in same manner as in the SLOS method, except that an additional drive is developed for backfilling purposes, or the exhausted stope is backfilled via a borehole. Temporarily pillars are left between sections of stope, and removed after the backfill has set.

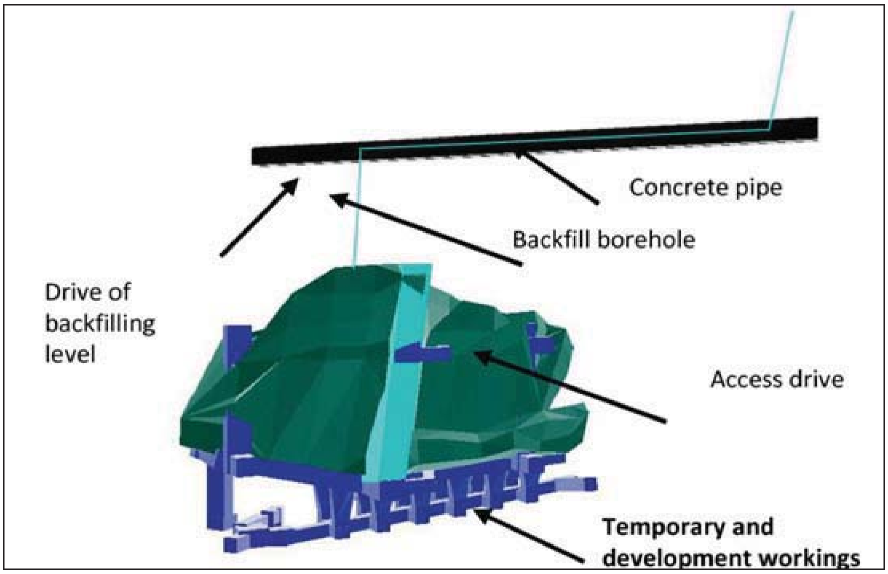


Figure 5.11: Sublevel Open Stoping with Backfill

The hardness and stability of the backfill is monitored and controlled via boreholes.

The backfill is used in areas of the mine where the caving of empty stopes is dangerous or could potentially damage buildings on surface. Backfill is also used where there is potential for a slide or collapse contour reaching the shaft protection pillars.

5.6.6.3 Shrinkage Stoping

Shrinkage stoping is applied in thin steeply dipping ore bodies with a thickness varying from 0.2-1.5m, in medium to stable rock. This method uses the broken ore as a platform for drilling the next lift and installing ground support.

After blasting, ore is taken from the stopes until there is sufficient space to drill the next lift. The ore is fed into rail wagons from chutes and draw points. The miners access the faces via timber ladder-ways which are surrounded by crushed rock but kept open whilst the stope is in production. As well as drilling and blasting, the miners conduct ground support by installing rock bolts.

Shrinkage stoping is used for extracting approximately 3-5% of overall tonnage. Typical shrinkage stope parameters are as follows:

- Length – 50m;
- Width – defined by orebody thickness;
- Height – 40m.

5.6.6.4 Unmechanised Sublevel Stoping

This method is similar to shrinkage stoping and is applied in thin (0.2-1.5m) steeply dipping ore bodies in medium to stable rock. The difference with shrinkage stoping is that this method allows for an increased production rate by developing several faces simultaneously and the lag time between ore being blast and that ore being hauled to the shaft is minimal. The other difference is this method requires more development as shown in Figure 5.12 below.

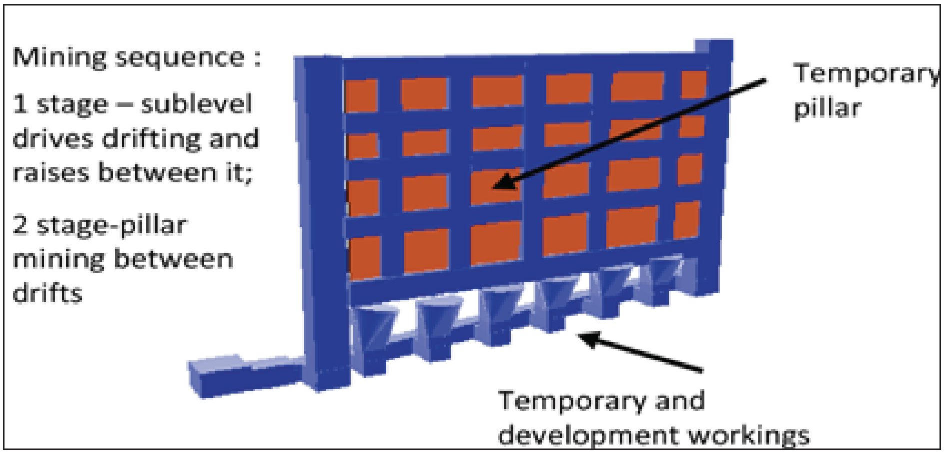


Figure 5.12: Unmechanised Sublevel Stoping

This method is rarely used, as the majority of orebodies could be mined employing more productive and less labour-intensive methods. The typical stope parameters are as follows:

- Length – 50m;
- Width – defined by orebody thickness;
- Height – 40m.

WAI Comment: *The increase of trackless equipment on the mine will improve the development advance rate and open up access to multiple orebodies quicker. As the number of large orebodies suitable to bulk mining is decreasing; the trackless mining methods will ensure that the target production rate is still met.*

5.6.7 Block Grade Evaluation

The mining stopes are designed once the mineralised cut-off grade of the orebody has been outlined. Usually more than one option is considered in terms of ore block access and extraction. All options considered are compared by the amount of preparation required, estimated losses and dilution, final metal content, and proposed value of extracted metals. As a result every mining block is assigned its Net Smelter Return (NSR) value and all the values are compared. At the time of the WAI visit, the cut-off NSR was US$36/t.

The NSR value is dependent upon market prices for metals and other external factors. The actual block NSR value is monitored during mining, and if it is below the set limit for current month, some measures are put in place.

> ***WAI Comment:*** *It is an advantage of the mine that ore tonnages and grades can be tracked for every minimal mining unit using weighing and sampling system of every loaded wagon. Evaluation of block NSR is a good practice, which improves efficiency of mining, and which is not common in former Soviet Union countries.*

5.6.8 Rock Properties and Geotechnical Conditions

The rock properties are generally medium stable and medium hard. The hardness factor varies from 6 to 14 for ores and 9 to 13 for barren rock. The drillability factor is in a region of 15. Loosening coefficient is normally from 1.24 to 1.88 with average of 1.56 (used in calculations). Average density 2.8t/m3 for ore; 2.7t/m^3 for waste. Ore is hazardous in terms of silicosis possibility.

5.6.9 Drilling and Blasting

Most of the production blastholes are drilled using LPS-3U pneumatic drillrigs, as shown in Photo 5.1 below, produced locally by Kazzincmash in Ridder town. The blasthole sizes are either 110mm or 130mm in diameter depending on the rock properties. There is a tendency to use 110mm diameter, as it helps to reduce dilution, although requires higher density of the blasthole grid.

The mining of the Pobeda deposit will be with trackless equipment in both development and production. Atlas Copco Rocket Boomer 104 and Boomer 282 jumbos will be used for the mine development.

Photo 5.1: LPS-3U Pneumatic Drillrig in Operation

Both electric and non-electric initiation systems are used in development blasting. Detonating cords are used to initiate production blasts in the stope. Electric blasting is used in vertical raises and a combination of electric and non-electric initiation is used in mass blasts.

The high explosives are mainly granulated explosives, Granulit-A6, Granulit-AS8 and Igdanit, to maintain ore fragmentation. Packaged Ammonit-6ZhV in 90mm diameter cartridges are used for wet blast holes and Ammonal-200 in 32mm diameter cartridges are used as a booster.

5.6.10 Losses and Dilution

Mining activities take place on a large number of ore bodies which exhibit considerable variation in both morphology and grade, therefore the dilution and loss parameters are variable. The difference can also explained by the large number of ore bodies which are mined at a time. The monthly production reports containing the detailed data on actual dilution and losses were provided to WAI at the site. The monthly results show that expected losses are 3-6% for sub-level stoping; 3-8% for backfilled methods, 25-35% for sub-level open stoping and 20-40% for backfilled stopes.

5.6.11 Ore and Waste Transportation

Ore and waste is drawn from chutes and other drawpoints into rail wagons and transported by K-10, K-14 and EL 5/04 electric locomotives to the shaft. Each locomotive is capable of towing up to 11 wagons. The capacity of the wagons is 0.8, 2.2, 2.5 and 4.5m^3 depending on the level where it is operated. The wagons are discharged by rotary wagon tippers OK-2.8-2.96, OKE-4-410 or by wagon pushers installed at the shaft stations.

The average tramming distance on the 11, 13 and 16 haulage levels is normally around 10km, and an example for 16 level is shown in Figure 5.13 below, and the standard locomotive travel speed is 7-12km/h.

Ore and waste from 18 level is hoisted in 2.2m^3 wagons to 15 level at Slepaya-Bistrushinskaya Shaft. From 15 level, it is moved to the main haulage level on 16 level and then delivered to skips at shafts Skipovaya (Pb-Zn ore) and Novaya (Cu ore and waste).

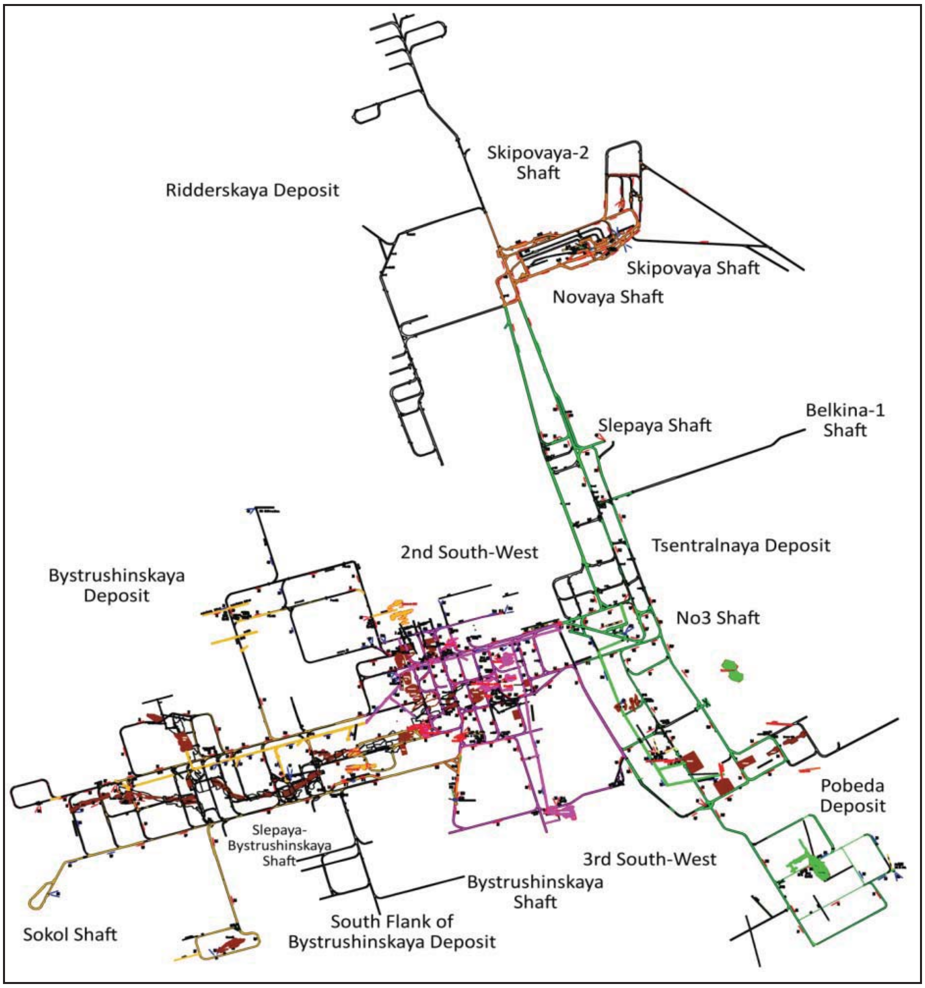


Figure 5.13: Plan of Horizon 16

5.6.12 Hoisting Facilities

Skipovaya shaft is equipped with a 2C-5x2.3 winder (drum diameter 5m, width 2.3m) which is used for (on levels 13 and 16) Pb-Zn ore hoisting (7.5m^3 skips). Novaya shaft has two winders, skip type and cage type. The skip hoisting is operating with 4.8m^3 skips hoisting Cu ore from level 16. Cage hoisting is performed by a CR5-3.0/0.6 winder working with double deck cage and counter weight.

Andreyevskaya Shaft is used for man riding to 11 level and is equipped with a CR-2.2AR cage type winder. Bystrushinskaya shaft is equipped with a ShPM2x4x1.7 winder operating with a double decked cage and counterweight.

Belkina-2 Shaft is used as an emergency egress for the miners, employing a 2BM-3000/1520 winder and cage. Slepaya-Bystrushinskaya shaft is equipped with a 2x3x1.5 U4P single cage winder. This shaft is used for hoisting men, materials and equipment and ore from 17 and 18 levels to 15 level.

5.7 Dewatering System

There are 4 major pumping stations at Ridder-Sokolniy mine, located on 11, 13, 16 and 18 levels as shown in Figure 5.14 below. Each station works has two pipelines, one operational and another is a reserve. All of the pipelines are installed within Novaya shaft.

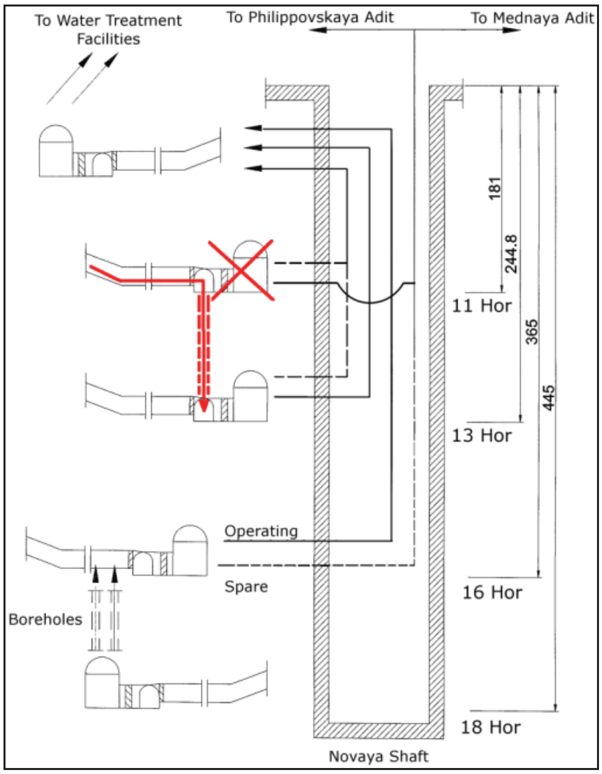


Figure 5.14: Dewatering Scheme (planned layout changes are shown in red)

Water from the lower levels is collected in the sumps on 18 level and then pumped, via 100mm diameter pipes installed in boreholes, to the next pumping station located at the 16 level sumps. Detailed data on pumping facilities is given in Table 5.20 below. From level 16, water is pumped to the adit level via a 400mm diameter pipe which is then used at the ore treatment facilities. After the water is cleaned, it is discharged into Philippovka River. The pumping station on the 13 level works in line with the same treatment facilities.

Water from the 11 level sumps is discharged to Phillipovka River via the Mednaya adit without treatment.

Table 5.20: Major Dewatering Facilities					
Unit	**Adit Station**	**11 Horizon Station**	**13 Horizon Station**	**16 Horizon Station**	**18 Horizon Station**
m^3/h	1280	734	660	620	102
pcs	1	3	3	2	2
m^3	1600	6 200	6 200	3 700	1830
	1D1250/125	TsN 1000/180	14M12-4	14M8-4	TsNS180/170
m^3/h	1250	1000	900	600	180
m	125	180	310	380	170
pcs	3	5	5	5	3
pcs	1	2	2	2	1
pcs	1	2	2	2	1
pcs	1	1	1	1	1

The mine is intending to upgrade the dewatering scheme by installing new pumps on 13 level and removing the existing 11 level pumping station. Boreholes will be used to move water from the 11 level pumping station to 13 level, where higher capacity pumps will be installed in order to deal with the increase water quantities.

5.7.1 Ventilation

Fresh air is supplied to the workings from Novaya, Skipovaya-2, Andreyevskaya, Belkina-1, Bystrushinskaya, Slepaya-Bystrushinskaya, Sokolok and Yuzhnaya shafts.

Foul air is discharged via Ventilation, Shaft No.3, Belkina-2 and Severniy shafts by means of exhaust ventilation fans. In order to minimise ventilation requirements, mined out areas, and collapsed areas connected to surface are barricaded off by concrete and timber constructions.

There are a number of ventilation fans installed both on surface and underground. The surface ventilation fans are installed as follows:

GVhV- 40 – 2200 at Ventilation Shaft, reversible by system of by-passes;
VUPD-2.8 – Shaft No.3, reversible by changing rotation direction of the engine;
VUPD-2.8 – Shaft Belkina-2, reversible by changing rotation direction of the engine;
VOKD-1.8 – installed in ventilation raise, not used in reverse mode;
VOD-30 – Shaft Andreevskaya, not used in reverse mode;

The main underground fans are installed as follows:

VOD-21 – on level 14 of 2-South-West ore zone, reversible by changing rotation direction of the engine which should be performed together with reverse of GVhV - 40 - 2200;
VOD-21 – on level 16 of 2-South-West ore zone, not used in reverse mode;
VOD-21 – on level 11 of Bystrushinskaya ore zone; reversible by changing rotation direction of the engine which should be performed together with the reverse of GVhV - 40 - 2200;
VM-12 – Yuzhnaya shaft, level 14 of Pobeda ore zone, not used in reverse mode;

An additional fan is expected to be installed at surface on Belkina-2 shaft by the end of 2010.

Local mobile ventilation fans are also located in the workings to supply fresh air to underground sites. Types and quantities of the fans are specified below.

VME-5 – 17 units;
VME-6 – 58 units;
Korfman ESN9-450 – 5 units.

> ***WAI Comment:*** *It was noticed by WAI during the underground visit that the ventilation was operating efficiently, providing a good quantity of air to the remote locations of the mine as required.*

5.7.2 Mine Personnel

The total number of people employed by Kazzinc at Ridder-Sokolniy is approximately 1,600, with a further 700 contractors. The mine has a developed structure of departments, each of the departments has specific designated areas and scope of works. A list of departments, together with the number of staff employed is given in Table 5.21 below.

Table 5.21: Kazzinc Personnel

Administration	9
Health and Safety Service	9
Dispatcher department	8
Production management	2
Surveyor Department	23
Geology Department	24
Mine Design and Planning Department	8
Underground Mining Brigade No1	97
Underground Mining Brigade No2	98
Underground Mining Brigade No3	91
Underground Mining Brigade No4	69
Underground Mining Brigade No5	50
Underground Development Works Brigade No14	52
Underground Development Works Brigade No15	64
Underground Development Works Brigade No17	89
Underground Development Works Brigade No18	113
Underground Drilling Works Brigade No6	80
Underground Exploration Works Brigade No11	58
Underground Blasting Works Brigade No7	73
Underground Transportation Brigade No8	235
Underground Materials Supply Brigade No12	64
Underground Installations Maintenance Brigade No9	175
Underground Ventilation Brigade	37
Underground Backfill Brigade No10	52
Total for the mine	**1,580**

It was noticed by WAI on the site visit that the mine has high standards of health and safety in place, particularly the use of PPE and minimising risks in underground environs, as well as paying a lot of attention to social aspects. Sports and recreational facilities, medical services are available to employees and their families.

> ***WAI Comment:*** *WAI considers that there are no major employment issues for the mine. The mine is located in close proximity to a town with a population of approximately 60,000 people. Historically mining has been one of the main business activities in the area, and WAI understands that there is a large number qualified personnel available in Ust-Kamenogorsk (located approximately 80km north-west from Ridder.*

5.7.3 Future Mine Development

Most of the mine development work will be centered at the Bystrushinskaya and Pobeda areas next year. It is proposed to develop levels here below 16 level using trackless equipment. There are plans to develop a 2,225m decline from 13 to 20 level in the north-west part of the mine, accessing the south flank of the Bystrushinskaya deposit. This decline will be connected with a 980m conveyor, with a 10◦ inclination, running from 16 to 20 level. The conveyor will move ore from the bottom levels to the main haulage on 16 level.

Exploration is currently taking place on the flanks of the ore zones and it is anticipated that these new resources will be incorporated into future production plans.

5.7.4 Conclusions

Ridder-Sokolniy mine is an operation with a long history and an expected long mine life. The developed reserves and exploration will ensure the availability of a reserve up 2030 and beyond. The mine is operated to a high standard both in management and actual mining activities. The employment of qualified and skilled personnel provides a stable workforce and assists in meeting the desired production rate. The existing underground infrastructure has some spare capacity to take up the planned increase in production rate. Despite most of mining requiring backfill and employing manual mining methods, reasonable mining costs have been achieved. This has also been helped by the availability of company owned machinery manufacturing facilities for locomotives, wagons and drill rigs and the availability of power.

5.8 Process

5.8.1 Introduction

The Ridder Mining and Metallurgical Complex (RMCC) which is owned and operated by Kazzinc is situated approximately 120km due east of Ust-Kamenogorsk in the north eastern region of Kazakhstan. Various sulphide and precious metal bearing ores are mined in the Ridder area and processed in the nearby metallurgical facilities. Zinc ore from Kazzinc's Shaimerden mining operation 200km south west of the city of Kostanay is also transported to Ridder for processing.

The ores that are processed at Ridder are produced from the following mine resources:

- Ridder-Sokolniy mine;
- Tishinskiy mine;
- Shubinskiy mine;
- Staroye tailings dam sands;
- Aged slimes from crushing;
- Fresh slimes from crushing; and
- Shaimerden zinc oxide and other oxide ores.

All but the zinc oxide ores are initially sent for crushing at one of three crushing facilities. The Shaimerden ore is transported by rail directly to the zinc smelter. Of the ores sent for crushing one ore type, a high gold grading silica rich ore from the Ridder-Sokolniy mine is dispatched to Ust-Kamenogorsk directly after crushing. The remaining ores are further treated by crushing, milling, gravity and froth flotation in one of two concentrator buildings.

The target mine ore production/ore treatment tonnages for 2010, excluding oxide ores, are given in Table 5.22 below.

Table 5.22: Ridder Concentrators Target Ore Production and Treatment for 2010

Ridder	Ore Type	Tonnes (kt)	Zn (%)	Pb (%)	Cu (%)	Au (g/t)	Ag (g/t)
Ridder-Sokolniy	Pb/Zn	1,952	0.54	0.31	0.15	1.75	13.37
Ridder-Sokolniy	Cu	270	0.22	0.1	1.25	0.4	5.3
Ridder-Sokolniy	Au Flux	30	0.50	0.28	0.28	1.83	12.26
Tishinskiy	Polymet	1,187	5.23	0.66	0.4	0.83	9.84
Subinskoye	Cu/Zn	190	1.52	0.16	1.17	0.26	11.02
Staroye Tailings	Au/Zn	1,100	0.63	0.28	0.04	1.04	12.38
DMS Slimes	Polymet	142	2.68	0.38	0.33	0.36	6.69
Old DMS Slimes	Polymet	197	2.2	0.54	0.22	0.43	8.1

From the inputted ore given above, the metal production forecast is shown in Table 5.23 below.

Table 5.23: Ridder Concentrators Target Production for 2010

Metal	Form	Tonnes (t)	Comment
Cu	In concentrates	10,591	Sold to Chinese Smelters
Pb	In concentrates	8,256	Transported to Kazzinc Smelter in Ust-Kam
Zn	In concentrates	73,642	Treated at Ridder Zinc smelter
Au	In ore/concentrates	3,392	To further processing in Ust-Kam
Ag	In ore/concentrates	17,501	To further processing in Ust-Kam

The copper concentrate is sold to a Chinese smelter and the lead concentrate is dispatched to the Kazzinc lead smelter in Ust-Kamenogorsk. The gold concentrate is also dispatched to Ust-Kamenogorsk where it is fed into the lead smelter.

The zinc concentrate is transported by truck a few kilometres to the Kazzinc zinc smelter that is also located on the edge of the town of Ridder. Several other ores and concentrates, including the Shaimerden and other oxide ores are treated at the zinc smelter as shown in Table 5.24 below.

Table 5.24: Feed to Ridder Zinc Smelter (2010)

Source	Tonnes Ore/Concentrate		Tonnes Zn	
	Planned 2010	Actual 2010	Planned 2010	Actual 2010
Combined Zn Concentrate, Ridder Concentrator	137,066	132,348	73,643	69,133
Zyryanovskiy Concentrate	0	8,480	0	4,703
Akzhalskiy Concentrate	0	0	0	0
Artemievskiy Concentrate	8,641	15,761	4,407	8,093
Zheskentskiy Concentrate	10,121	2,278	4,549	1,051
Ust-Talovskiy Concentrate	1,421	8,554	537	3,608
Berezovskiy Concentrate	5,364	2,733	2,361	1,211
Karagailinskiy Concentrate	8,548	4,409	2,821	1,782
Belousovskiy Concentrate	9,564	4,382	3,826	1,620
Balquash Concentrate (Cu-Cd)	4,200	3,638	1,962	1,674
Shaimerden Ore	225,719	235,746	47,401	48,726
Zn cake ex Ust-Kam Metal Complex	0	4,202	0	841
Iron Concentrate	16,370	7,188		
Zinc cake ex Ridder zinc plant	54,230	80,138	11,774	17,480
Zn white bleach	0	0	0	0
Slimes from Purification Facilities	0	0	0	0
Slimes from Ust-Kam	0	0	0	0

There are two types of smelting operation in the zinc smelter. There are Waelz kilns for treating the oxide ores that produce a precipitated zinc oxide fume product, and there are fluidised bed roasters that treat the zinc sulphides and produce a zinc calcine. Both products are then acid leached using acid made from the roaster off-gas in the on-site acid plant to produce a zinc-rich solution. This solution is purified and electro-won to produce the final zinc metal product. Cadmium and excess acid are also produced in this facility along with several other minor by-products. Table 5.25 below shows the planned smelter production for 2010.

Table 5.25: Ridder Zinc Smelter Target Production for 2010

Metal	Form	Tonnes (t)	Comment
Cu	In Cu cake	878	To Zyrianovskiy plant for further processing
Pb	In by product	1,747	Transported to Kazzinc Smelter in Ust-Kam
Zn	Metal	110,829	Sold to final customer
Au	In by product	0.0285	To further processing in Ust-Kam
Ag	In by product	1.80	To further processing in Ust-Kam
Cd	Metal	378	Sold to final customer
H_2SO_4	93 - 95% concentrate	146,464	Sold to final customer

5.8.2 Services

Power to the Ridder mines and ore processing facilities is supplied by overhead power cables from the Bukhtarminskoya Hydro-power station some 140km away to the west. The whole site consumes 519,026MW per annum.

5.8.2.1 Industrial Water Supply

Two systems are applied for industrial water supply to RMCC facilities:

- The fresh industrial water pipeline with water intake from surface sources; and
- The process water circulation system of the Concentrator.

The sources of the industrial water circulation system are as follows:

- Verkhne-Khariuzovskiy water dam. Water from the Gromotukha River flows by gravity through a reinforced concrete flume to the "zero chamber" and from there goes through two water pipes to a 1,000m^3 distribution tank and further on flows by gravity through two 1,000mm water pipes to the RMCC industrial water supply system; and
- Bystrushinskiy water dam. Water from this dam goes through a 900mm pipeline to the general production water supply system. (This water dam is a backup dam that is to be used in case of a shortage of water from the Verkhne-Khariuzovskiy dam. The annual water consumption from these facilities is 153Mm^3.

Concentrator Process Water Circulation System

This system is intended for water supply for crushing and processing and it operates as follows. Slurry with a solid content of 6 to 11%, and pH = 8.8 is pumped sequentially by pumping stations Nos.1, 2 and 5 from the concentrator to the Talovskoye tailings dam where the first settling stage is performed. The settled water from Talovskoye is then pumped through pumping station No.4 into Chashinskoye tailings dam where water is further settled and becomes suitable for application in the production process. Clarified water with a suspended solids content of 4.3mg/l and pH =7.6 from Chashinskoye tailings dam is pumped through the recycling water pumping station No.6 into the Concentrator process water tank. The volume of water that is recycled in this way is 28.8Mm^3 per year.

5.8.3 Ore Crushing Facilities

There are three crushing installations at Ridder, two of these are located adjacent to the Ridder–Sokolniy mine and Concentrator buildings Nos.2 and 3, while the third installation is located at the Tishinskiy mine site where it crushes ore from the Tishinskiy mine to be used as the feed to a dense media separation facility.

5.8.3.1 Crushing Section No.2

Located in an elevated position behind Concentrator building No.2 is the crushing Section No.2 facility which consists of two almost identical crushing circuits. The first of these circuits, Processing Chain No.1, is used mainly for crushing the Tishinskiy heavy ore product from the DMS beneficiation plant. This ore is transported from the mine by rail and stockpiled close to the crusher and from there it is moved by trucks to a 1,000t primary crusher feed bin.

The primary crusher is an old McCool CCD 500 gyratory cone crusher with an installed power of 75kW. Ore from the feed bin is conveyed to a 150mm vibrating screen from which the coarse fraction feeds the primary crusher. The crushed product then joins the screen fine fraction and is conveyed to a 12mm vibrating screen from which the coarse product feeds the Russian Uralmesh CSD 1750B secondary cone crusher. The fine product from this screen however is fed to a spiral classifier. Note that copious amounts of spray water are used throughout the crushing operation here and in all of the crushing facilities, in order to facilitate the transport of ore through transfer points, to reduce sticking of fine fractions to operating parts of crushers and to keep dust levels low which would otherwise pose a health risk. Classifier overflow is pumped to Concentrator building No.2 for processing.

The secondary crushed product is conveyed to a third and final 16mm vibrating screen from which the coarse product feeds the Russian Uralmesh CMD 2200 tertiary crusher. The final tertiary crushed -16mm product joins the tertiary vibrating screen underflow and together with the second stage classifier sands, is conveyed to the Concentrator building No.2 mill feed bins for further processing.

Processing Chain No.1 of Crushing Section No.2 is also used to crush the high grade gold "Flux" ore from the Ridder-Solkolniy mine for intermittent short periods. And there is facility to remove this from the crushing circuit after secondary crushing and to load it out in trucks to Ust-Kamenogorsk.

Processing Chain No.2 of Crushing Section No.2 is used to treat the Ridder-Sokolniy Copper Ore and the Shubinskiy mine ore. The circuit is essentially a copy of Processing Chain No.1 with the exeption that the primary jaw cursher CMD-110 has an installed power of 110kW. There is also some flexibility built in to this crushing chain enabling the circuit feed or final product to be transferred over to Processing Chain No.1. Both the Ridder-Sokolniy copper and the Shubinskiy mine ore also have their own ore bin and feeding systems to introduce each ores into the crushing circuit. Ores crushed in this circuit are conveyed to the Concentrator building No.3 mill feed bins for further processing.

5.8.3.2 Crushing Section No.3

Located close to Crushing Section No.2 and Concentrator Building No.3, Crushing Section No.3 is comprised of a single line, three stage crushing facility that is normally dedicated to treating Pb/Zn ore from the Ridder-Sokolniy mine which lies directly beneath it.

The ore is hoisted up a vertical shaft to the surface in skips to a 500t bin. The ore is then fed on a steel plate apron feeder to the 150mm grizzly screen from which the coarse oversize ore goes for primary crushing in a Kennedy CCD 900 cone crusher (manufactured in 1929) with 184kW power. The screen unders and the primary crushed ore are conveyed to one of two 60mm grizzly screens. The oversize product from the screen is fed to the secondary Uralmash CSDT 2,200 crusher while the screen undersize is fed to a 12mm grizzly screen from which the oversize joins the secondary crusher product and the undersize is fed to a spiral classifier. Note that as for the other crusher circuits, copious amounts of water are added to the circuit. The coarse fraction from the classifier joins the final tertiary crushed product while the overflow is pumped to Concentrator building No.3 for further processing.

The secondary crusher product in Crushing Section No.3 is fed by conveyor to a series of 13 16mm vibrating screens from which a complex system of conveyors carries the coarse product to four installed tertiary crushers. Two of the four tertiary crushers are Russian Uralmesh CMDT 2,200 cone crushers while the other two are newly installed Metso/Nordberg HP 400 and HP 200 cone crushers. The undersize from the vibrating screens form the final -16mm crushed ore is conveyed to the feed bins of the Concentrator building No.3.

> ***WAI Comment:*** *While most of the facilities in Crushing Section Nos.2 and 3 are old they appear to be well maintained and fit for purpose. Kazzinc management have replaced two of the worst crushers in Crusher Section No.3 with modern Nordberg units from Metso which it is claimed has increased the crushing capacity of the circuit. Further renewal and replacement in the Crusher Sections now forms part of a future plan to increase the overall milling capacity of Concentrator building No.3.*
>
> *The strategy of upgrading the crushing sections appears to revolve around the replacement of individual units of equipment with more modern and efficient units. There are several areas in the crushing circuits as a whole however that do not conform to modern practice and where improvements could be made. For example, the number of standby units is inefficient as is the number of vibrating screens in each circuit. It is also inefficient to pass the classifier coarse product directly to the final tertiary crushed product as appears on both flow sheets in Section No.2. A modern circuit*

might be comprised of a jaw crusher, a standard cone crusher and a short head cone crusher, arranged in circuit with a double deck screen.

The screens (13 units) in Section No.3 installed upstream the fine crushers are in continuous operation. This scheme is arranged in such a way to allow ore stockpiling in the Crushing section for the period of ore supply interruption from the mine. Design and survey works are being carried out by TOMS Institute to select crushing and equipment flowsheet for Crushing Section No.2, particularly, for fine crushing and ore fines removal.

It is suggested that Kazzinc put a temporary halt to the acquisition of new crushing equipment and take a step back in order to take a higher level look at their current and future crushing needs and what the most efficient means of achieving this might be.

5.8.3.3 Tishinskiy Crusher

Ore from the Tishinskiy mine is hoisted to the surface and tipped into one of two Run-of-Mine ore bins. There is also a third ore bin into which ore from outside sources can be introduced into the crushing circuit. From these bins the ore is fed by conveyor on to a 100mm grizzley screen from which the coarse +100mm fraction is fed to a UZTM (A Russian manufacturer based in the Urals) model CRD 700/75 primary cone crusher with an installed power of 250kW and a discharge setting of 150mm. This primary discharge then joins the -100mm screen fine fraction and is conveyed to a 50mm vibrating screen. Coarse product from this screen, feeds a secondary UZTM CSD 2200T cone crusher with an installed power of 250kW producing a -80mm product. The crushed product then joins the -50mm screen underflow and is conveyed to a 2,200t crushed ore bin from which it is fed into the dense media separation (DMS) plant.

There is a third, tertiary crusher in the Tashinskiy DMS beneficiation plant building that operates on the DMS light (waste) ore fraction. After screening and washing to remove and reclaim the dense media, the light ore fraction is fed by conveyor to a UZTM CMD 2,200T cone crusher with an installed power of 250kW from which the discharge is conveyed to a 25mm vibrating screen. The +25mm fraction from the screen is conveyed back to the tertiary crusher feed. The -25mm fine screen product is fed by conveyor to a stockpile. Further on, the crushed light fraction is used for preparation of a concrete fill mixture for mine back fill. The concrete back fill preparation plant and Tishinskiy ore DMS process are discussed elsewhere in this report.

5.8.4 Concentration Facilities

There are three main concentrators at and around the Ridder mining complex. These are the DMS plant situated at the Tishinskiy mine site, and Concentrator buildings No.2 and No.3, which are located adjacent to the town of Ridder itself.

5.8.4.1 DMS Plant

The crushing section of the Tishinskiy DMS plant as described earlier produces a minus 50mm feed to the dense media separation facility. Prior to separation in the DMS drum however, the ore is split on a double deck vibrating screen into +5mm, +2mm and -2mm fractions. The latter fines fraction of the ore is passed through a bank of two spiral classifiers to recover coarse and heavy material which joins the 5mm product. The fine and lighter material from the classifiers is thickened and filtered in a plate and frame filter press. It is then fed by conveyor to a temporary stockpile before being transported by rail to the concentrator. It is then re-pulped and pumped to the Main Building No.2 concentrator for processing.

The coarse -50mm +5mm fraction from the double deck screen enters the Russian made DMS SB-200 drum separator which contains a dense media made up of water and a ferrosilicon: magnetite (70:30) powder. The operating volume of the drum is 1,400m^3 with a design throughput of 200tph. The density of the media is controlled at 2.75g/cm^3 such that the lighter gangue material will float and can be easily removed from the drum, while the heavy, base metal sulphide fraction of the ore will sink and can be collected separately from the bottom of the drum. To prevent the build-up of fine ore particles in the heavy media suspension, the overflow DMS plus that washed from the light and heavy fractions during screening is cleaned using wet magnetic roll separators. The dense medium solids cleaned off ore slimes and excess water is used again for suspension preparation.

Photo 5.2: DMS Drum and Magnetic Roll Separators

As described earlier, the light floating fraction from the DMS is washed on a screen and further crushed to -25mm for use as backfill in the Tishinskiy mine. The heavy sinking fraction is similarly washed on a vibrating screen after which it joins the - 5mm ore fraction and is conveyed to the heavy fraction storage bin. From here it is loaded into dump cars and transported by rail for further crushing at Crusher Section No.2 and further processing.

Including engineering and management, the Tishinskiy DMS plant employs 66 full time staff. It operates continuously. Periodic shutdowns of the DMS processing chain is allowed when rich ore with zinc content over 8% is delivered from the mine but it happens quite rarely. Ore crushing only is performed at the plant in such cases. The crushing and processing plant is designed for removal of waste rock and backfill concrete at the initial processing stage in order to increase the qualitative indicators of ore processing. On average the DMS plant rejects 15.6% of the ore that is mined.

> ***WAI Comment:*** *An alternative to DMS would be to develop a mine planning strategy to yield a less variable Run of Mine grade and/or a simple blending facility might be a more efficient approach and a separate study into this possibility is recommended.*

5.8.4.2 Concentrator No.2 and No.3

Concentrator buildings Nos.2 and 3 process all the sulphide ores that are mined in the Ridder area. These are currently the Tishinskiy polymetallic DMS heavy ore fraction, and the slimes from both the Tishinskiy DMS plant and from the Crushing Sections Nos.2 and 3, which are treated in the No.2 Concentrator, and the Ridder-Sokolniy Cu ore, the Ridder-Sokolniy Pb/Zn ore, the Shubinskiy Cu/Zn ore and the Staroye tailings dam sands, which are treated in the No.3 Concentrator. Each of the ore types requires a different treatment to efficiently recover the contained metals and therefore as the figures below show, the overall flow sheet for each concentrator is quite complicated. For this reason, the flow sheets as applied to the individual ore types will be discussed individually in later sections of this report.

In general however, each of the concentrators is comprised of milling, froth flotation and gravity separation. The primary mills are generally grate discharge overflow ball mills with manganese steel liners, while the secondary mills are open discharge overflow ball mills with rubber liners. All the mills have an installed power of 400kW and were originally 2.7m in diameter by 3.6m in length. More recently however, certain of the secondary mills in Concentrator No.3 have been lengthened to 4.5m. There is also a single mill in Concentrator No.2 that is 4.5m long but this is currently mothballed. All of the mills have been manufactured and modified on site in Kazzinc's own Engineering workshop. All of the mills are in closed circuit with either a spiral classifier (primary mills) or a hydrocyclone classifier (secondary mills).

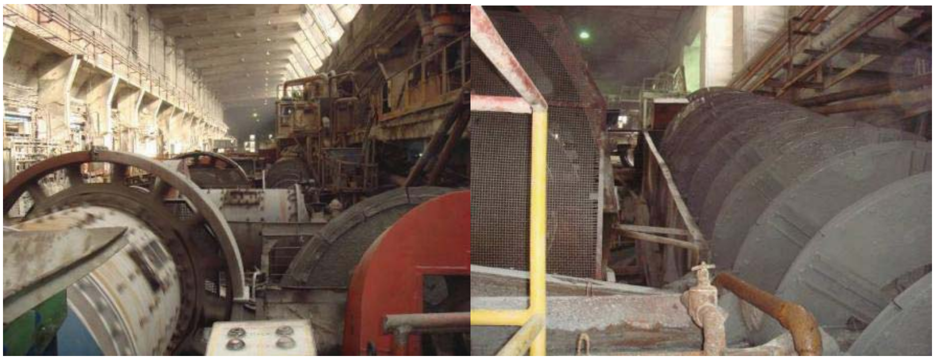


Photo 5.3: Concentrator Building No.3 and Spiral Classifier

Gravity separation is achieved with jigs, shaking tables and centrifugal concentrators. The jigs are Russian MOD-2M diaphragm jigs which are positioned on the primary and secondary mill discharge streams. These machines are to be gradually replaced with Knelson centrifugal concentrators. The gravity concentrates produced by the jigs and Knelson concentrators are further upgraded in two stages using Russian SKO 15m^2 and 7.5m^2 shaking tables.

Photo 5.4: Jig on Mill Discharge (centre left) and New Knelson Concentrator

Froth flotation is conducted in Russian forced air RIF 16m^3 rougher cells and RIF 8.5m^3 or FM-6.3m^3 cleaner cells. There is a plan underway in both concentrators however to replace these cells with 14.15m^3 and 4.25m^3 Wemco induced air flotation cells.

Photo 5.5: RIF Flotation Cells and Shaking Table

> ***WAI Comment:*** *The upgrading of the existing flotation machines and the introduction of Knelson concentrators in place of jigs is warranted and in line with modern mill practice. There is however, very little free gold in any of the ores and even after two stages of concentration with shaking tables this results in a low grade gravity concentrate of around 120g/t Au. It is questionable therefore whether shaking tables are the correct choice of equipment in these circumstances or even whether gravity separation is warranted at all, given that the gold reporting to the gravity concentrate may be picked up during flotation. A review of the gravity circuit is therefore warranted.*

All concentrates are dewatered with conventional rake thickeners and ceramic disc filters prior to being shipped for further processing.

Photo 5.6: Thickening and Filtration

5.8.4.3 Planned Changes to Concentrator Capacities

Current and planned annual tonnage throughputs for the Concentrators Nos.2 and 3 are shown in Table 5.26 and Table 5.27.

Table 5.26: Concentrator Building No.2 Annual Tonnage		
	Current (t)	**Planned (t)**
Slimes Polymet	500,000	500,000
Tishinskiy Polymet (heavy)	979,090	979,090
Shubinskiy Cu/Zn	0	190,000
Total	1,479,090	1,669,090
Planned Increase	190,000	
% Increase	13	

Table 5.27: Concentrator Building No.3 Annual Tonnage		
	Current (t)	**Planned (t)**
Ridder-Sokolniy Cu	180,000	290,000
Ridder-Sokolniy Pb/Zn	1,900,000	2,300,000
Shubinskiy Cu/Zn	190,000	0
Staroye tailings dam Au/Zn	1,100,000	1,100,000
Total	3,370,000	3,690,000
Planned Increase	320,000	
% Increase	9	

In the case of the Concentrator building No.2, plans are in place to increase the flotation capacity by the introduction of five Outotec 50m^3 tank cells in order to accommodate the extra tonnage. The ongoing introduction of new and more efficient crushing equipment to Crushing Section No.2 will to some degree offset the increased throughput; there is however, no plan to increase the milling capacity as it is claimed there is sufficient excess capacity in the current circuit.

In the Concentrator building No.3 the strategy for handling the planned extra throughput is to increase the milling capacity. This will be achieved by replacing the two primary 2.7x3.6m primary, plus the 2.7x4.5m secondary ball mills with single 4.5m x 5.0m primary and a single 4.5x5.0m secondary ball mills (section 5). This will be aided by a finer crush size from the planned new crushers, and by adopting a smaller steel ball size in the secondary mill, it is believed that this will not only accommodate the increased tonnage but it will also produce a finer grind. There is however, no plan to modify the existing froth flotation and gravity sections of the concentrator as it is claimed that sufficient capacity already exists.

> ***WAI Comment:*** *There is no cause to doubt the calculations and conclusions of the Kazzinc management regarding the planned upgrades. Given the complexity of operations in both of Concentrators Nos.2 and 3 and the potential cost of any mistake however, a full review of the planned upgrades is suggested with the joint objectives of identifying any possible bottle necks and confirming the planned throughput and grind. It is recommended that a "fresh pair of eyes" undertake this review.*
>
> ***General Comment on Facilities:*** *All of the plants visited – the DMS facility, the Crusher Section Nos.1 & 2 and the Concentrator buildings Nos.2 & 3 were well maintained and well managed. Some of the equipment was very old and in need of replacement with either new equipment or alternative equipment but the management were aware of this and plans were in place to rectify these matters. All of the management and operational staff that were interviewed or engaged in less formal conversation were knowledgeable and helpful.*
>
> *If there is one negative it is the condition of certain stairways, walkways and access ways within the facilities which were not entirely up to modern European standards.*

5.8.5 *Ore Types and Treatment*

The various ore types that are mined in the Ridder area and processed at the Ridder concentrators and their nominal grades are summarised in Table 5.28. The oxide ore from Shaimerden and its subsequent smelting and refining will be discussed later in the report.

Table 5.28: Summary of Nominal Grades

Ore	Au (g/t)	Ag (g/t)	Cu %	Pb %	Zn %
Tishinskiy Polymet Ore (heavy fraction)	0.83	9.84	0.40	0.66	5.23
Ridder-Sokolniy Lead Zinc Ore	1.80	13.23	0.15	0.31	0.54
Ridder-Sokolniy Copper Ore	0.40	5.30	1.25	0.10	0.22
Shubinskiy Copper Zinc Ore	0.26	11.02	1.17	0.16	1.52
Staroye Tailings Dam Gold Zinc Sands	1.04	12.38	0.04	0.28	0.63
Polymet Slimes from DMS – Old and New	0.40	7.51	0.27	0.47	2.40

Each of the above ores is different to the other and as such each ore must be treated differently in order to efficiently recover the contained metals.

5.8.5.1 Tishinskiy Polymetallic Ore

The preliminary upgrading of the Tishinskiy ore using dense media separation and its subsequent crushing to -16mm has been described earlier. This pre-treated ore is then delivered to Concentrator building No.2 where it is subjected to two stages of milling. Jigs are installed at the primary and secondary mills' discharge and the jig concentrate is reground in a 2.7m x 3.6m rubber lined overflow ball mill and then is concentrated further on shaking tables. The final gravity concentrate is pumped to the dewatering container SK-2.5m^3 which has a porous base, overlain with filter cloth. When full the dewatering container is removed from the circuit and allowed to drain naturally. After dewatering the concentrate is transferred to a standard container of the same size but without a porous base. The SK-2.5 container dimensions are: height 1,760mm, base diameter 1,057mm, top diameter 1,495mm.

The gravity tailings from the cleaning circuit are recycled back to the primary milling circuit.

After two stages of milling, the final milled product (less the gravity concentrate), has been reduced in size to a P80 of 74 microns and is now ready for the first stage of what is quite a complicated process of differential froth flotation.

In essence however, a first stage of flotation involving roughing scavenging and three stages of concentrate cleaning are employed, using a mixture of n-butyl xanthate and proprietary C7 dithiophosphate collectors to produce a mixed Cu/Pb concentrate. Zinc and pyrite are depressed during this float by the addition of zinc sulphate, sodium sulphide and very small amounts of sodium cyanide. The lead is then floated from the Pb/Cu concentrate using further xanthate and C7 additions while depressing the copper with the addition of lime and more sodium cyanide. This final concentrate is pumped to the lead thickener for dewatering. Tailings from this Pb float then pass to copper flotation where the copper sulphide minerals are activated by the addition of sulphuric acid and floated into a final concentrate by the addition of more of the xanthate/C7 collector combination. Sodium sulphite is also added to the copper float to depress any traces of Pb, Zn or pyrite that are still present. The final copper concentrate is pumped to the copper thickener for de-watering. The tails from copper flotation are reground in a 2.7x3.6m open discharge rubber lined overflow ball mill before being floated in a middling flotation circuit producing Cu rich and Zn rich products that are recycled back into those respective circuits.

Tailings from the primary Cu/Pb float are then subjected to zinc flotation after activation with copper sulphate and the addition of xanthate and C7 collectors. Aerofloat promoter is also used in the scavenger part of this float and lime is used to suppress pyrite. The resulting zinc flotation concentrate is then pumped to the zinc concentrate thickener for de-watering.

5.8.5.2 Shubinskiy Cu/Zn Ore

After crushing to -16mm in Crusher Section No.2 – Chain No.2, the Shubinskiy ore is conveyed to the Concentrator building No.3 where it is milled in two stages to produce a P80 -85micron flotation feed. There are jigs on the discharge of each of the two primary mills, as well as on the secondary mill discharge. The Shubinskiy ore has a tendency to slime during milling however, which produces a high viscosity pulp that makes the jigs operate inefficiently. Due to this and the low gold content of the Shubinskiy ore therefore, these jigs do not normally operate while this ore is being processed. Note however, that this same processing circuit. – Section No.2 – is also used to treat the copper ore from the Ridder-Sokolniy mine, the two ore types being treated in separate batches. The jigs would be in operation while treating the Ridder-Sokolniy copper ore.

After milling the Shubinkiy ore is subjected to two stages of differential froth flotation to produce separate copper and zinc concentrates. The first stage of the differential float involves roughing and scavenging with three stages of concentrate cleaning including the usual middling recycle streams to produce a final copper concentrate. A 60:40 blend of butyl xanthate and sodium Aerofloat are used to float the copper with the addition of T-66 frother. Zinc sulphide and other unwanted sulphide minerals are depressed during this stage of flotation by the addition of zinc sulphate, sodium sulphite and small quantities of sodium cyanide. The resulting copper concentrate is pumped to the copper thickener for dewatering.

In the second stage of the Shubinskiy differential float, the zinc sulphide is activated by the addition of copper sulphate to the pulp and then floated with further additions of the xanthate/aerofloat collector blend and T-66 frother. In the process of zinc flotation, lime is added to raise the pulp pH and depress unwanted sulphides such as pyrite. The zinc flotation stage comprised of roughing, scavenging and three stages of concentrate cleaning. The final concentrate is pumped to the zinc concentrate thickener for dewatering.

5.8.5.3 Ridder-Sokolniy Cu Ore

The copper ore from the Ridder-Sokolniy mine is crushed to -16mm and subjected to milling and froth flotation in the same circuit as the Shubinskiy ore described above. In this instance however, the jigs in the milling circuit are in operation and the concentrate that they produce is cleaned on two stages of shaking tables. There is also a 2.1x3.0m open discharge rubber lined overflow ball mill in the gravity cleaning circuit.

Copper flotation is carried out using roughing, scavenging and three stages of cleaner flotation. The flotation machines used in the zinc circuit for treating Shubinskiy ore are used here for scavenging copper from the flotation tailings during Ridder-Sokolniy copper ore processing. During copper flotation zinc sulfides that are present in the ore in small quantities are depressed by the addition of zinc sulfate. 60:40 butyl xanthate and Aerofloat mixture is used as copper collector with addition of the frother T-66.

5.8.5.4 Ridder-Sokolniy Pb/Zn Ore

This Pb/Zn ore is crushed to -16mm in Crusher Section No.2 and conveyed to Concentrator building No.3. Further processing follows essentially the same route as described above for the other ores with two stages of milling, gravity concentration and froth flotation. Gravity concentrates from the jigs report directly to a regrind circuit incorporating a 2.7x3.6m open discharge rubber lined overflow ball mill before being upgraded in the two stages of shaking tables. This is a recent change to the gravity circuit which has reportedly led to a doubling of the gold recovery from the jig concentrate as well as producing a significant increase in the grade of the final gravity concentrate from 100 to 120g/t Au. The final gravity concentrate is dewatered as described earlier and stored in bins for transportation to Ust-Kamenogorsk for further treatment. Gravity tailings from the cleaning circuit are recycled back to the secondary milling circuit.

The final milled product, less the gravity concentrate, at a grind size P70 of 74 micron, is fed to froth flotation. The flotation circuit includes both bulk flotation and selective flotation procedures. A bulk sulphide concentrate and tailings are produced in the bulk flotation circuit which includes roughing, scavenging and two stages of cleaning flotation. Butyl xanthate and sodium aerofloat are added in the ratio 60:40 along with T-66 frother as required.

The tailings from the bulk stage of flotation are pumped to the tailings dam. The bulk concentrate is further condition with sodium sulphide in order to desorb the flotation reagents and then thickened and pumped to a 2.1x3.0m open discharge rubber lined overflow ball mill. The regrinding discharge containing 94% passing 74 microns goes to selective circuit. The selective flotation circuit includes copper-lead flotation stage, zinc flotation stage, a stage of gold re-flotation from the zinc stage tailings and a copper-lead concentrate separation stage. The copper-lead flotation circuit consists of roughing scavenging and four cleaners. Flotation is carried out in a weakly alkaline media with depression of zinc minerals by the addition of zinc sulphate and some small quantity of sodium cyanide. Butyl xanthate and sodium aerofloat are added in the ratio 60:40 along with T-66 frother as collectors. A bulk copper-lead concentrate with a high gold grade and copper-lead flotation tailings are produced in the copper-lead circuit.

Copper-lead concentrate splitting into copper and lead concentrates was introduced in 2010. Copper is floated in acidic conditions using sodium thiosulphate and ferrous sulphate to depress lead and zinc minerals. The copper circuit includes rougher, scavenger and three cleaner stages. Copper concentrate is produced in copper circuit in form of the cell concentrate and lead concentrate in form of the cell tailings product. The copper concentrate reports to a copper concentrate thickener for dewatering and the lead concentrate reports to the gold-bearing concentrate thickener for dewatering.

The tails from Cu-Pb flotation then reports to zinc flotation. The zinc flotation circuit includes rougher, scavenger and three cleaner stages. Zinc minerals are firstly promoted by copper sulphate, then subject to flotation using 60:40 blend of butyl xanthate aerofloat with addition of T-66 frother. The zinc concentrate is pumped to the zinc thickener for dewatering.

The tailings from zinc flotation are then subject to a further stage of froth flotation with the addition of 60:40 blend of butyl xanthate: aerofloat and T-66 frother. A low grade gold/pyrite concentrate is produced in flotation circuit. This concentrate is mixed with the copper concentrate. Note that the copper concentrate sales contract with China allows for a minimum Cu content of 18% and the addition of the pyrite does not dilute the Cu level below this. The advantage to Kazzinc of adding the pyrite is the extra gold credit.

5.8.5.5 Staroye Tailings

The Staroye tailings dam operated from the start-up of the first concentrator in 1926 through to 1953 at which time tailings disposal was moved to the Chashinskoye site. This material is now being reclaimed and processed for its gold and zinc values. The sands are moved by means of trucks and front end loaders to a temporary stockpile before being fed through grizzly bars into a feed hopper which in turn feeds onto a conveyor belt and into a rotating scrubber. Water is added to the scrubber which together with the sands forms a pulp which discharges into a trommel screen on the end of the scrubber. The pulp passes through the trommel and is pumped to the Concentrator building No.3 for processing. The coarse material in the tailings which is essentially barren waste, discharges at the end of the trommel and is discarded.

The pulped tailings reports to a standalone milling circuit made up of one primary grate discharge and two secondary centre discharge rubber lined overflow ball mill, each mill being 2.7x3.6m with a 400kW motor. The mills operate in closed circuit with a hydrocyclone. There is a jig on each of the mill discharges and the collective rougher gravity concentrate from these jigs is pumped to a 2.1x3.0m open discharge rubber lined overflow ball mill for regrinding before it is further concentrated on two stages of shaking tables to produce a final gravity concentrate. This concentrate is then dewatered as described earlier and stored in bins for transportation to Ust-Kamenogorsk for further treatment. Gravity tailings from the cleaning circuit are thickened and recycled back to the secondary milling circuit.

The final milled product less the gravity concentrate has now been reduced in size to 95% passing 74 microns and is conditioned with a 60:40 blend of butyl xanthate and aerofloat collectors plus T-66 frother before being pumped to a bank of rougher and scavenger flotation machines to produce a bulk Au/Zn concentrate. The tailings from the bulk flotation are pumped away to the operating Talovskoye tailings dam.

Sodium sulphide is then added to the bulk Au/Zn concentrate to partially desorb the flotation reagents from the particles and it is then thickened and sent for differential flotation using roughers, scavengers and three stages of concentrate cleaning. A gold concentrate is produced here by the addition of more of the xanthate/aerofloat collector and T-66 frother. Zinc sulphate, sodium cyanide and activated carbon are also added in order to depress zinc sulphides and pyrite. The resulting gold concentrate is sent for thickening with the gold concentrate that is produced from the Ridder-Sokolniy Pb/Zn ore.

Tails from the gold float are then subject to further roughing, scavenging and three stages of cleaning to produce a zinc concentrate. This is achieved with the addition of copper sulphate to activate the zinc sulphide and further additions of the collectors and the frother. Lime is also added to depress pyrite.

Tails from the zinc float then report for pyrite flotation. This again consists of roughing, scavenging and three cleaners and is affected by the addition of more of the collector and frother. The pyrite concentrate contains gold and is added to the copper concentrate that is sold to China. The tailings from this final stage of flotation are considered waste and pumped out to the Talovskoye tailings dam.

5.8.5.6 Slimes

Slimes originate from Tishinskiy crushing circuit (current slimes) as well as being reclaimed from the slimes collectors (aged slimes) at the Tashinskoye crushing and concentrating (DMS) plant. Slimes are delivered to temporary stockpiles at the Concentrator and then repulped in a separate scrubber and fed to Concentrator of building No.2 in form of pulp.

Slimes are milled to 90% passing 74 micron in a 2.7x3.6m open discharge rubber lined overflow ball mill. There are no jigs in this circuit. Copper and lead are floated into a combined concentrate using roughing, scavenging and three cleaning stages using C7 collector and T-66 frother. Zinc and pyrite are depressed by the addition of zinc sulphate and cyanide.

The Cu/Pb concentrate then passes to differential flotation, again with roughing scavenging and three cleaners where only activated carbon, zinc sulphate and T-66 frother are added to generate a final Pb concentrate which is pumped to the Pb thickener for dewatering. Tailings from the Pb float then report for Cu flotation in the standard rougher, scavenger, three cleaner circuit where the copper sulphides are activated by the addition of sulphuric acid to give a pulp pH of 5.0. C7 collector and T-66 frother are added along with zinc sulphate to depress the zinc. The final Cu concentrate that is produced is pumped to the copper thickener for dewatering.

The tailings from Cu flotation, along with the first cleaner tails from the initial Cu/Pb combined float, are then reground in a 2.1x3.0m open discharge rubber lined overflow ball mill to a size of 98% passing 74 microns. Roughing and scavenging flotation are then employed to recover any remaining Cu and Pb, with the use of the C7 collector and T-66 frother. Zinc sulphate and sodium cyanide are also added to depress zinc and pyrite. The resulting low grade Cu/Pb concentrate is recycled back to the head of the initial combined Cu/Pb float.

The tails from this scavenging operation still contain some zinc and they go to join the main tailings stream from the Cu/Pb float where rougher and scavenger flotation is used to produce a low grade zinc concentrate using C7 collector and T-66 frother. The tailings from this stage are now essentially barren and are pumped out to the Talovskoye tailings dam. The low grade zinc concentrate is forwarded to a final rougher scavenger and 3 cleaner circuit where further C7 and T-66 additions are used to produce a final zinc concentrate. Pyrite is depressed during this float by the addition of lime and the tails from this stage are pumped back to the head of the zinc low grade rougher scavenger float. The final zinc concentrate is pumped to the zinc thickener for dewatering.

WAI Comment: *While the basic processes and reagents that are in use in the concentrators are standard, the complexity of the flow sheet and the seemingly smooth running of the operation is a testament to the time and effort that has gone into the developing the process over many years.*

5.8.6 Production

The following two tables summarise the tonnes treated and the concentrate production from the various ores for the period 2007 to 2010. Both planned and actual data is given for 2010.

5.8.6.1 Tishinskiy

Table 5.29: Tishinskiy Polymetallic Ore					
	2007	**2008**	**2009**	**2010 Planned**	**2010 Actual**
Tonnes Treated	1,388,373	1,061,843	1,234,778	1,187,500	1,250,683
Au g/t	0.82	0.8	0.88	0.83	0.85
Ag g/t	11.1	10.02	10.92	9.84	10
Cu %	0.39	0.4	0.43	0.04	0.42
Pb %	0.71	0.72	0.8	0.66	0.66
Zn %	5.06	5	5.28	5.23	4.93
Cu % Recovery	78.04	84.76	81.69	82.00	79.27
% Cu concentrate Grade	29.17	29.43	29.06	28.50	28.75
Pb % Recovery	71.44	71.44	68.81	68.00	65.63
% Pb concentrate Grade	73	72.55	71.07	72.00	67.45
Zn % Recovery	91.2	91.91	92.93	93.00	91.28
% Zn concentrate Grad	56.49	56.37	55.82	56.00	54.74
Gravity Au % Recovery	3.69	2.46	5.88	13.76	2.22
Gravity Au g/t Grade	63.01	58.57	51.16	75.00	56.26
Total Au % Recovery	65.82	65.32	66.39	72.76	61.67

WAI Comment: *The actual tonnage for 2010 has met and even slightly exceeded the plan, although zinc and lead grades were slightly down. Generally however, the planned and actual production is in reasonable agreement. The one oddity in this is the gold grade and recovery which is in line with previous years, but which for some reason were predicted to rise.*

Production figures were expected to increase as a result of an upgrading of the ore gravity concentration circuit but due to delays in equipment delivery it was not completed in time.

5.8.6.2 Ridder-Sokolniy Pb/Zn Ore

Table 5.30: Ridder-Sokolniy Pb/Zn Ore					
	2007	**2008**	**2009**	**2010 Planned**	**2010 Actual**
Tonnes Treated	1,847,190	1,949,921	1,762,822	1,952,200	1,810,443
Au g/t	1.72	1.68	1.61	1.75	1.77
Ag g/t	10.55	10.68	7.96	13.37	13.46
Cu %	0.16	0.17	0.23	0.15	0.22
Pb %	0.25	0.26	0.27	0.31	0.28
Zn %	0.52	0.53	0.50	0.54	0.53
Gravity Au %Recovery	27.78	25.66	35.25	34.00	26.60
Gravity Au g/t Grade	106.59	117.35	162.36	140.00	138.56
Float Au % Recovery	55.8	53.55	36.97		39.45
Float Au g/t Grade	72.88	73.23	54.32		54.51
Cu %Recovery	-	-	1.82	73.00	27.85
%Cu concentrate Grade	-	-	15.60	17.19	6.41
Pb %Recovery	-	-	-	70.00	0.00
%Pb concentrate Grade	-	-	-	31.00	0.00
Zn %Recovery	63.77	57.15	52.45	55.00	50.00
%Zn concentrate Grade	57.10	56.15	54.31	54.00	54.41
Pyrite Au % Recovery	-	2.97	7.28	4.00	0.77
Pyrite Au g/t Grade	-	9.02	11.21	10.00	12.28
Total Au % Recovery	85.17	84.42	78.82	85.90	83.33

WAI Comment: *Based on the above figures, the 2010 tonnage of Ridder-Sokolniy Pb/Zn Ore were approximately on target both for tonnage and grade.*

5.8.6.3 Ridder-Sokolniy Cu Ore

Table 5.31: Ridder-Sokolniy Cu Ore					
	2007	**2008**	**2009**	**2010 Planned**	**2010 Actual**
Tonnes Treated	169,671	132,386	262,886	270,000	160.091
Au g/t	0.40	0.43	0.65	0.40	0.67
Ag g/t	5.88	5.61	5.22	5.30	6.68
Cu %	1.31	1.06	1.04	1.25	1.05
Pb %	0.06	0.07	0.09	0.10	0.15
Zn %	0.18	0.19	0.23	0.22	0.33
Gravity Au % Recovery	4.25	0.87	13.09	12.00	6.21
Gravity Au g/t Grade	89.27	11.55	156.14	100.00	63.49
Cu % Recovery	95.88	96.17	95.17	62.00	95.84
%Cu concentrate Grade	24.23	24.64	23.88	25.00	22.56
Total Au % Recovery	66.37	63.56	77.15	74.00	67.49

WAI Comment: *The tonnage treated in 2010 shows a substation shortfall in production. Gold recovery and grade are also approximately half of plan. Both of these issues warrant further investigation.*

5.8.6.4 Ridder-Sokolniy Ore Flux

Table 5.32: Ridder-Sokolniy Ore Flux				
	2007	**2008**	**2009**	**2010 Actual**
Tonnes Treated	38,218	44,470	60,316	59,820
Au g/t	15.63	19.8	26.8	19.71
Ag g/t	12.85	9.7	13.8	19.6
Cu %	0.35	0.3	0.5	0.62
Pb %	0.66	0.3	0.2	0.37
Zn %	1.29	0.8	0.6	0.67
Total Au % Recovery	100	100	100	100

5.8.6.5 Shubinskiy

Table 5.33: Shubinskiy Cu/Zn					
	2007	**2008**	**2009**	**2010 Planned**	**2010 Actual**
Tonnes Treated	186,789	179,457	211,650	190,000	194,414
Au g/t	0.46	0.54	0.72	0.26	0.55
Ag g/t	13.95	12.49	13.94	11.02	11.51
Cu %	0.94	1.15	1.37	1.17	0.91
Pb %	0.24	0.19	0.23	0.16	0.22
Zn %	1.64	1.66	1.74	1.52	1.65
Cu % Recovery	83.14	88.21	87.41	87.00	83.56
%Cu concentrate Grade	22.52	22.53	22.38	22.80	22.37
Zn % Recovery	72.89	78.09	78.02	78.00	75.92
%Zn concentrate Grade	46.12	47.16	45.39	45.00	42.59
Total Au % Recovery	42.35	35.11	35.06	32.00	28.33

WAI Comment: *Production for 2010 was on target although the gold grades in the ore are substantially higher than planned while gold recovery is lower than plan and lower still than the previous year. Together these two factors constitute a significant loss that warrants further investigation.*

5.8.6.6 Staroye Tailings

Table 5.34: Staroye Tailings					
	2007	**2008**	**2009**	**2010 Planned**	**2010 Actual**
Tonnes Treated	346,232	421,885	936,966	1,100,000	414,636
Au g/t	1.73	1.64	1.76	1.04	1.12
Ag g/t	16.79	16.56	14.99	12.38	11.16
Cu %	0.07	0.07	0.15	0.04	0.06
Pb %	0.44	0.37	0.35	0.28	0.26
Zn %	1.18	0.92	0.97	0.63	0.72
Float Au % Recovery	52.83	56.63	45.31	41.00	37.78
Float Au g/t Grade	44.06	38.37	38.85	37.00	29.18
Gravity Au % Recovery	10.98	6.69	8.60	8.00	7.19
Gravity Au g/t Grade	76.05	52.82	91.24	75.00	49.96
Zn % Recovery	27.10	19.94	23.95	25.00	11.84
%Zn concentrate Grade	39.15	35.20	31.84	40.00	21.65
Total Au % Recovery	72.40	69.26	64.06	56.70	56.68

WAI Comment: *The actual tonnage is very low and under one half of the planned tonnage for 2010. Planned recoveries and/or grades for gold and zinc are low compared to previous years, presumably because of the lower grades in the ore.*

The reason for the sands tonnage decrease in 2010 is that sands had not been processed from February to May inclusive. This was due to an emergency situation at the existing tailings dam and the Staroye tailings dam quarry from where sands were transported to the Concentrator, was flooded.

5.8.6.7 Tishinskiy Slimes

Table 5.35: Current Tishinskiy Slimes

	2007	2008	2009	2010 Planned	2010 Actual
Tonnes Treated	0	141,649	27,732	142,500	225,803
Au g/t	-	0.59	0.52	0.36	0.61
Ag g/t	-	7.97	6.85	6.69	7.96
Cu g/t	-	0.32	0.34	40.33	0.34
Pb g/t	-	0.45	0.49	0.38	0.48
Zn g/t	-	2.78	2.86	2.68	2.8
Cu % Recovery	-	27.26	22.41	32.20	20.12
%Cu concentrate Grade	-	18.69	21.20	20.97	18.81
Pb % Recovery	-	23.48	24.35	30.00	27.54
%Pb concentrate Grade	-	36.49	33.47	41.00	40.35
Zn %Recovery	-	73.45	76.29	74.16	73.21
%Zn concentrate Grade	-	46.20	42.05	44.00	42.6
Total Au % Recovery	-	55.63	73.36	58.08	71.55

***WAI Comment:** Actual production of Tishinskiy slimes in 2010 is much greater in terms of tonnage and gold grade than planned. This may be a result of a backlog from 2009 when only approximately 28kt was treated.*

5.8.6.8 Old Tishinskiy Slimes

Table 5.36: Old Tishinskiy Slimes

	2007	2008	2009	2010 Planned	2010 Actual
Tonnes Treated	207,162	149,609	0	197,500	144,168
Au g/t	0.54	0.52	-	0.43	0.55
Ag g/t	8.44	8.10	-	8.10	7.58
Cu g/t	0.25	0.29	-	0.22	0.31
Pb g/t	0.57	0.49	-	0.54	0.45
Zn g/t	2.66	2.49	-	2.20	2.21
Cu % Recovery	9.36	32.23	-	32.20	12.22
%Cu concentrate Grade	20.03	20.97	-	20.97	16.01
Pb % Recovery	25.83	24.02	-	30.00	16.07
%Pb concentrate Grade	33.44	41.12	-	41.00	25.06
Zn % Recovery	70.06	74.16	-	74.16	56.03
%Zn concentrate Grade	45.01	45.55	-	44.00	31.05
Total Au % Recovery	49.66	52.70		58.08	62.25

***WAI Comment:** Tonnages are significantly down on plan for 2010, which is presumably due to the excess of the current Tishinskiy slimes that have been treated, as both these materials are treated through the same circuit. Aside from this all the metals with the exception of gold are yielding poor recoveries or concentrate grades or both as compared to the planned production*

Cu losses in zinc and lead concentrates resulting from the presence of secondary (up to relative 35%) and oxidized (up to relative 30%) copper forms in the old slimes, were poorly depressed in the copper-lead and lead flotation circuits.

Pb losses in zinc concentrate and discard tailings resulted from the presence of oxidized forms of lead minerals in the old slimes (up to relative 70%).

Zn losses in discard tailings resulted from presence of non-recoverable oxidized zinc forms in the old slimes (relative 40%).

5.8.6.9 General

Table 5.37 shows the overall performance for 2010.

Table 5.37: Overall Performance			
	2010 Planned	**12 Month 2010 Actual**	**Shortfall**
Tonnes	5,039,700	4,200,238	839,462

WAI Comment: *The above table shows shortfall in tonnage mined and treated of 839kt. WAI personnel were unaware of this situation at the time of the site visit and as such there has been no opportunity for discussion.*

An examination of the individual production profiles for each of the ore types reveals some large deviations between the planned 2010 production and actual. Leaving the issue of tonnes aside, there are still significant variations in the planned grades and recoveries of each of the metals, especially for gold. This may be more a planning problem than a processing problem, or it might be both, but the poor reconciliation between the expected and the actual does need to be investigated.

5.8.7 Zinc Smelter

WAI Comment: *The recovery of gold from the various concentrates that are treated at the zinc smelter appears to be very poor. In 2009 for example some 524kg of gold was sent to the smelter from Ridder-Sokolniy and a further 297kg was brought in from other sources, such that the total gold in the smelter feed is 819.831kg. According to the figures supplied by the smelter, only 28kg of this gold is recovered. Most of this gold arrives in the zinc sulphides and may be recoverable. The current process as described above involves the roasting and acid leaching of the concentrates to produce a zinc rich solution for further treatment. The leach residue which will still contain the gold, is then fed into the Waelz kilns to recover the unleached zinc. The Waelz kiln residues are stockpiled in waste dumps behind the plant in Ridder City.*

It is recommended that the recovery of gold from the zinc concentrates be investigated. This might be achieved for example by the inclusion of a small CIL plant after the acid leaching of the zinc calcine and before adding the leach residue to the Waelz kilns.

If this suggestion were to be found viable, then there may also be the opportunity to treat the gold containing pyrite concentrate in the roasters rather than adding it to the copper concentrate and sending it to China. This will produce extra acid that can either be sold or used for Kazzinc's own future projects as briefly suggested elsewhere in this report. The gold remaining in the calcined pyrite concentrate could then be cyanide leached along with the zinc calcine.

5.8.8 Manpower

Manpower for the Ore beneficiation facilities is given in Table 5.38 and the zinc smelting facilities in Table 5.39 below.

Table 5.38: All Crushing, Concentration and Related Activities		
	Plan	**Current**
Tishinskiy Crushing & DMS Plant	78	73
Reaearch Laboratory	29	28
Crushing Section Nos.2 & 3	98	95
Concentrator Building No.2	92	87
Concentrator Building No.3	151	149
Tails and Slimes processing	47	42
Product Thickening, Filtration and Transport	38	39
Reagent Area	21	21
Auxiliary (Storeman)	1	1
TMF	43	43
Management	7	7
Operation Support Service	4	4
Safety	2	1
Security	5	5
Total:	616	595

Table 5.39: Zinc Smelting and all Related Activities						
Shop, division	**Number of personnel**					
	Total	***including***				
		Workers	**Employees**			
			Total	***including***		
				Managers	**Specialists**	**Clerks**
Shop 1	160	144	16	15		1
Shop 2	137	127	10	8	1	1
Electrolysis Shop	224	214	10	8	1	1
Hydrometallurgical Shop	90	82	8	6	1	1
Waelz Shop	296	278	18	16	1	1
Lead Shop	7	6	1	1	-	-
Maintenance Shop	99	90	9	8	-	1
Technical Support Service	10	-	10	1	9	-
HSE Service	6	-	6	1	5	-
QA/QC Service	78	64	14	4	10	-
Including: Service	8		8	-	8	-
Qa/Qc	55	52	3	3	-	-
Research	11	9	2	1	1	-
Dust-Gas Sampling	4	3	1	-	1	-
Personnel Services	10	-	10	2	7	1
Including: Service	6	-	6	1	5	-
Leading Legal Advisor	1	-	1	-	1	-
G&A Service	3	-	3	1	1	1
Management	1	-	1	1	-	-
Budgeting Dept	9	-	9	1	8	-
HR Service	1	-	1	-	1	-

5.8.9 Tailings Dams

From the commencement of operations at the first concentrator in 1926, the concentrator tailings were pumped to the Staroye tailings dam. This dam remained in continuous use until 1953 by which time some 13Mt of tails had been deposited there. Subsequent to 1953 the tailings deposition was switched to the Chashinskoye tailings dam and a further 95Mt of tailings were pumped here up until 1978. From this date until the present day all the concentrator tailings have been and continue to be pumped to the Talovskoye tailings dam and to date some 100Mt have been deposited here.

As described earlier, the tailings contained in the Staroye dam are being retreated at a rate of 1.1Mt. The Chasinskoye tailings are also considered to be a resource and it is proposed to begin treating this material to recover its gold and silver content in 2013. Drilling has shown the dam to contain a potential resource of approximately 88Mt of tailings with 1.7Moz of gold and 14.0Moz of silver.

Although there are no firm plans in place to retreat the Talovskoye tailings dam, it is recognised that there is the potential to do this at a future date although this dam will likely remain in use for a further 20 years.

Photo 5.7: Talovskoye and Chashinskoye Tailings Dams

5.8.10 Laboratory

The assay laboratory function at Ridder is essentially split into two entities which both operate under the umbrella of the same top management. These two divisions can be classed as being "crushing and concentration" and the "zinc smelter". Each division operates a main assay facility close to the operations that it services plus smaller satellite laboratories, the main purpose of which is to provide express analysis for plant control.

Aside from the process control and metallurgical accounting functions of the laboratory, it is also responsible for the environmental monitoring of all gas, liquid and solid emissions from the Ridder Complex. The laboratory facilities employ a total of 170 staff and operate around the clock.

The Crushing and Concentration laboratory mainly uses Atomic Adsorption Spectroscopy, X-Ray Florescence Spectroscopy and Fire Assay with a gravity finish, as well as several of the classical wet chemical methods. There is also an on-line XRF analyser that analyses the more important pulp streams from both concentrator buildings every hour and so provides an important control function.

The Zinc Refinery laboratory uses the following spectrometric methods of analysis:

- Atomic Adsorption;
- Atomic Emission;
- Inductively Coupled Plasma;
- Spark Emission;
- Infra Red; and
- X-Ray Florescence.

WAI Comment: *The laboratory equipment is modern and the facilities are well maintained and well managed.*

5.8.11 Engineering Workshop

Kazzincmash is a separate organisation owned by Kazzinc and operating within the Ridder metallurgical complex. Its main activities are the manufacture and repair of mining, concentrating and metallurgical equipment and the manufacture of spare parts. As such most of the equipment that is required on the Ridder complex is manufactured here and some 50% of Kazzincmash's output is for external clients

The workshops employ 1,435 people and the capability of the operation is wide ranging; from mill shells and gears, crushers and pumps to drill rigs, mine cages and rail cars, locos and loader buckets and parts. Kazzincmash are also in partnership with international mining equipment supplier such as Atlas Copco, Caterpillar, the SMS group and Weir for maintenance and manufacturing.

5.9 WAI Ore Reserve Estimate

5.9.1 Introduction

WAI has carried out stope design, and produced JORC Code (2004) compliant Ore Reserves for the various ore bodies at Ridder-Sokolny, based upon the most recent Mineral Resource Block Models. WAI has used GijimaAST Mine2-4D® software to prepare the designs, and Microsoft Excel to prepare production schedules.

5.9.2 Mine 2-4D Software

Mine2-4D is an automated mining software package developed in Australia and currently marketed by GijimaAST. It allows the user to accurately design mine excavations such as development and stoping, and then allows the operator to apply time-dependent mining activities such as backfilling and cable bolting in a fully three-dimensional graphical environment. Following the design of excavations and associated activities (i.e. bolting or backfilling), mining activities can be sequenced, with time delays built into the sequence where appropriate.

Typically, following the completion of the design is complete and the associated activities applied, the mine design is exported into the Enhanced Production Scheduler (EPS) which is a direct representation of the 3D mine design in a Gantt chart format with the sequence and delays built into the model in Mine2-4D are also exported to EPS. However, due to time constraints, it has not been possible to utilise EPS for production of the Ridder-Sokoliny deposit, and as such the production schedules have been produced using Microsoft Excel.

5.9.3 Mining Parameters

The stope blocks for the Ridder-Sokolny ore bodies have been designed to a minimal average block grade of 2% Zinc Equivalent (Zn_{EQV}).

5.9.4 Mine Layouts

Mining areas at the Ridder-Sokolny mine have been laid out following the proposed mining methods laid out in Section 5.6 'Mining'.

5.9.4.1 Centralny Orebody

The Centralny orebody is one of the principal orebodies within the Ridder-Sokoliny deposit, and mining is currently on-going within the ore body. Principal mining levels are currently present at 20-40m intervals, with sub-levels to allow access to additional mining levels.

Figure 5.15 shows the Centralny orebody, highlighting mined areas of the orebody (orange/red) and non-mined areas (purple).

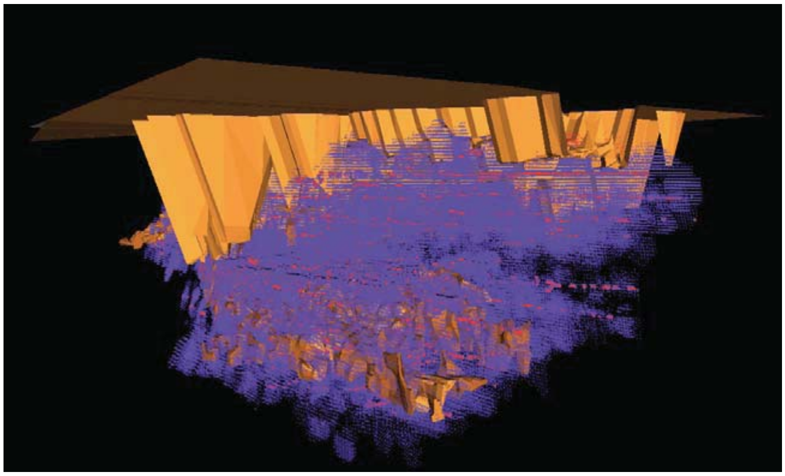


Figure 5.15: Centralny Orebody

WAI has designed stope blocks at 20-40m vertical intervals from the 375m to 775m Levels, utilising existing development to access stopes where feasible, with additional development required where existing infrastructure is not present.

Figure 5.16 below shows the proposed and existing stopes for Centralny.

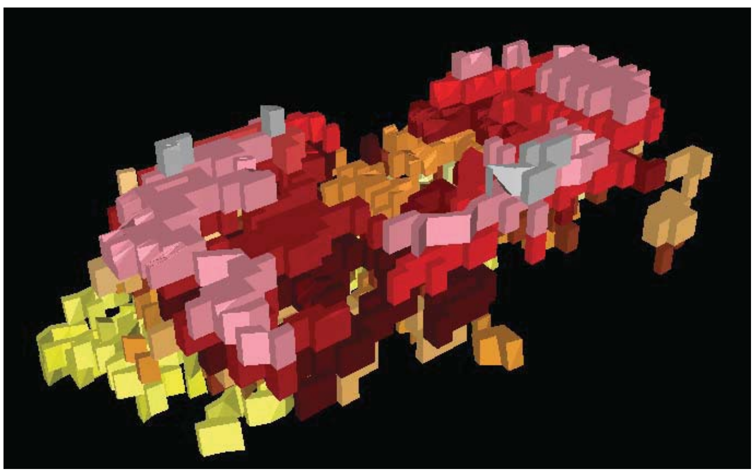


Figure 5.16: Centralny Stope Blocks

5.9.4.2 Belkina Orebody

The Belkina orebody is one of the larger orebodies within the Ridder-Sokolniy deposit, and mining is currently on-going. Principal mining levels are currently present on the 530m, 565m, 590m, 625m, 665m elevations, with sub-levels to allow access to additional mining levels.

Figure 5.17 shows the Belkina orebody, highlighting mined and non-mined areas.

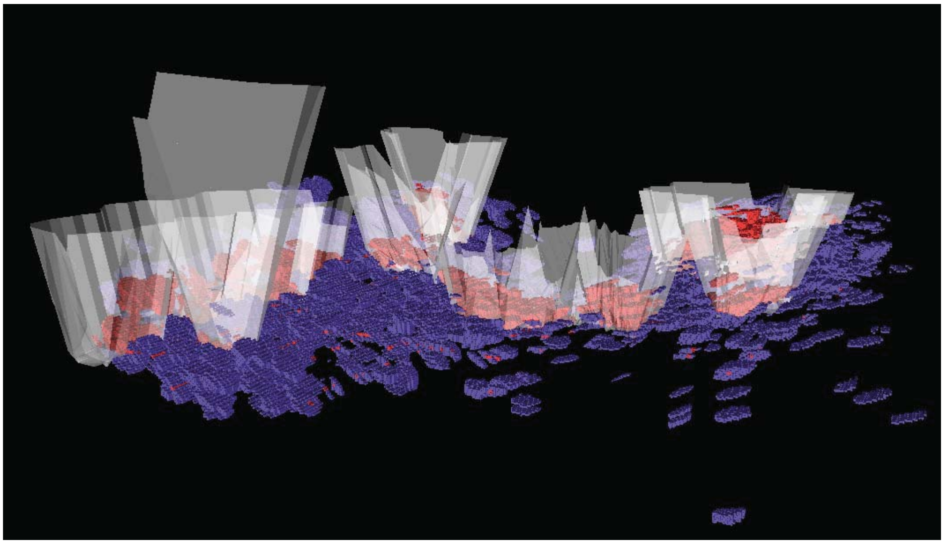


Figure 5.17: Belkina Orebody
Showing mined (red) and unmined (purple) areas, with mined-out envelopes (white)

WAI has designed stopes at 20m vertical intervals from the 535m to 755m Levels, utilising existing development to access stopes where feasible, with additional development required where existing infrastructure is not present.

Figure 5.18 below shows the proposed and existing stopes for Belkina.

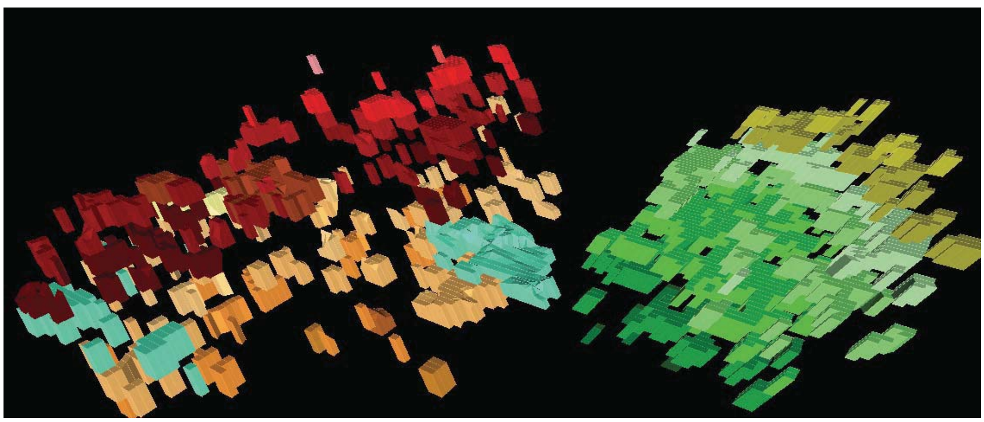


Figure 5.18: Belkina Stope Blocks

5.9.4.3 Perspectivnaya Orebody

The Perspectivnaya orebody is similar in size to the Belkina orebody. The orebody forms part of the Ridder-Sokoliny deposit with mining currently on-going. Principal mining levels are currently present on the 490m, 530m, 560m, 600m and 630m elevations, with sub-levels to allow access to additional mining levels.

Figure 5.19 shows the Perspectivnaya orebody, highlighting mined and non-mined areas.

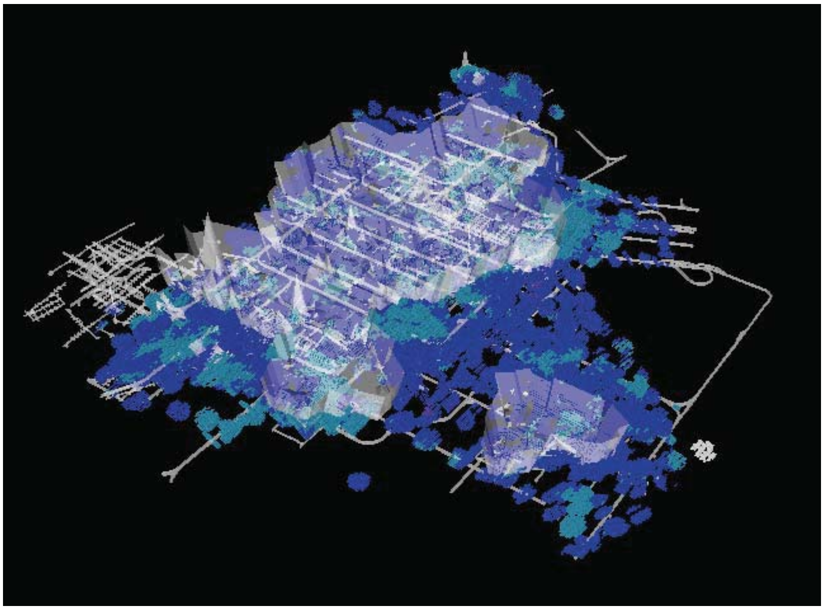


Figure 5.19: Perspectivnaya Orebody
Showing un-mined (blue) areas, with mined-out envelopes (white)

WAI has designed stope blocks at 30m vertical intervals from the 450m to 660m Levels, utilising existing development to access stopes where feasible, with additional development required where existing infrastructure is not present.

Figure 5.20 below shows the proposed stopes for Centralny.

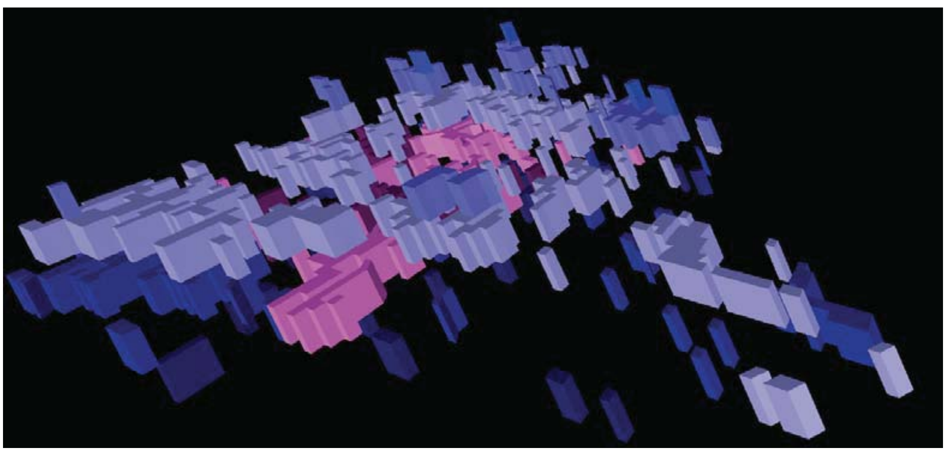


Figure 5.20: Perspectivnaya Stope Blocks

5.9.4.4 Zavodskaya Orebody

To date, the Zavodskaya orebody has been largely mined out, however a number of areas remain within the orebody suitable for mining. Principal mining levels are currently present on the 405m, 445m, 525m and 585m elevations, with sub-levels to allow access to additional mining levels.

Figure 5.21 shows the extent of existing extraction within Zavodskaya, relative to the geological block model.

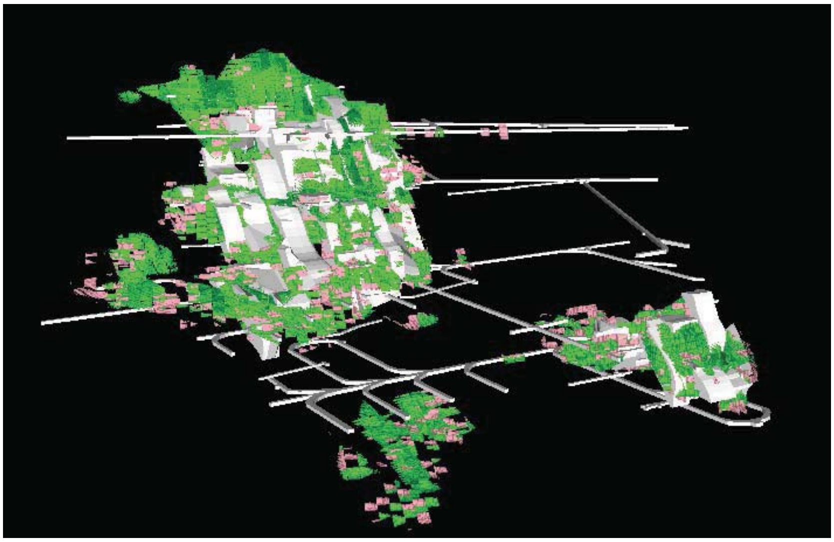


Figure 5.21: Existing Development at Zavodskaya

WAI has designed stopes at 20m vertical intervals from the 565m to 645m Levels, utilising existing development to access stopes where feasible, with additional development designed as required. It should be noted that much of the resource below the 565m Level has been sterilised by existing stope blocks.

Figure 5.22 below shows the proposed and existing development for Zavodskaya.

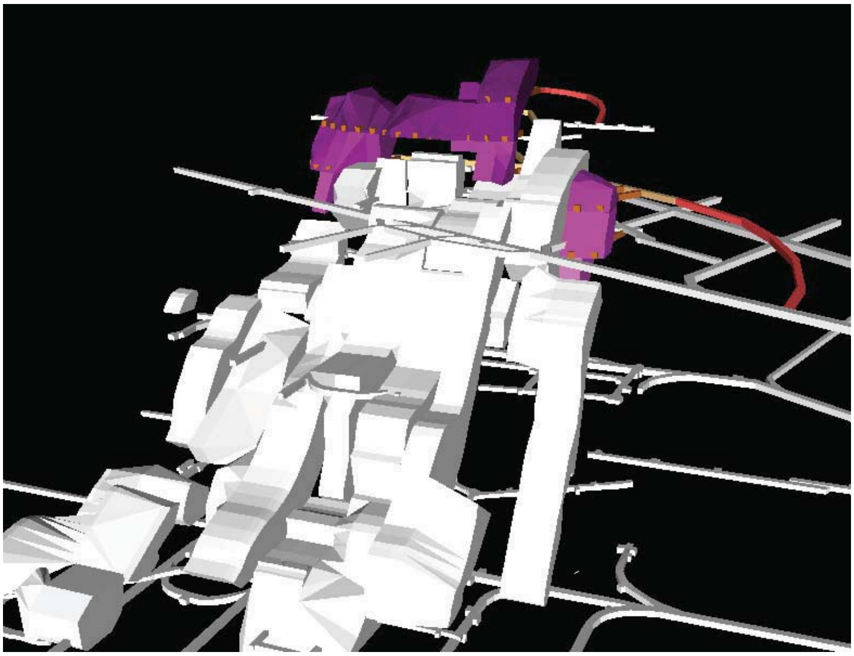


Figure 5.22: Zavodskaya Current (white) and planned (coloured) Stope Blocks

5.9.4.5 North Bestrushinskoye

The North Bestrushinskoye ore body has been accessed on the 400m Level, and whilst mining has been carried out within the ore body, areas for further extraction remain.

WAI has designed stopes in un-mined areas at 10-20m vertical intervals from the 380m to 570m Levels, utilising existing development to access stopes where feasible, with additional development required where existing infrastructure is not present.

Figure 5.23 below shows the proposed stoping blocks for North Bestrusinskoye.

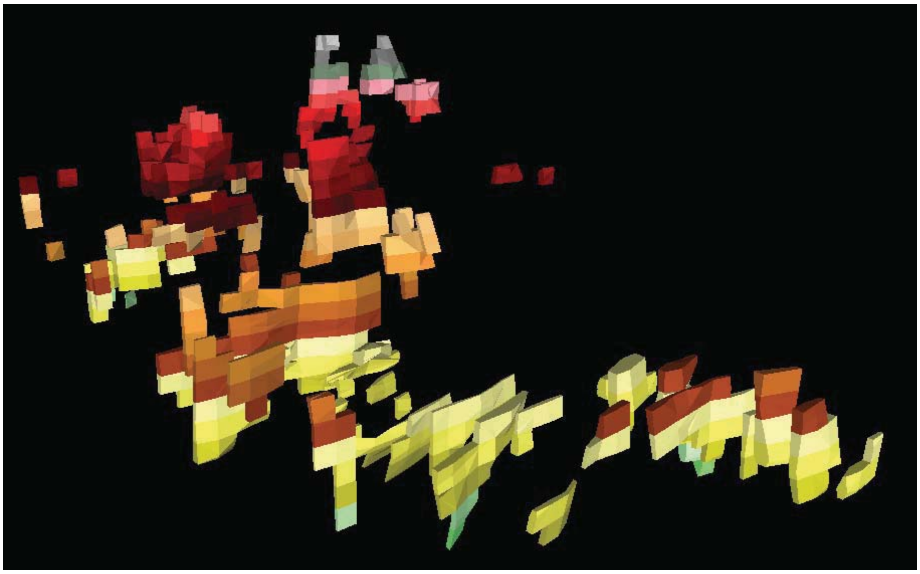


Figure 5.23: North Bestrushinskoye Stopes

5.9.5 Ore Reserves

Ore Reserves for the Ridder-Sokolny ore bodies have been calculated in accordance with the guidelines of the JORC Code (2004). A summary of these JORC compliant Ore Reserves is presented in Table 5.40.

Table 5.40 Ridder-Sokolniy Ore Reserve Estimate
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Deposit	Reserves	Ore (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (Pb)		Zinc (Zn)	
			Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Centralny	Proven	5.02	0.71	114,830	5.19	837,614	0.69	34,566	0.18	9,076	0.52	26,208
	Probable	9.17	1.03	303,446	5.95	1,756,434	0.26	23,699	0.34	31,365	0.64	58,683
	Total	**14.19**	**0.92**	**418,276**	**5.69**	**2,594,048**	**0.41**	**58,264**	**0.28**	**40,440**	**0.60**	**84,891**
Belkina	Proven	2.24	0.98	70,492	25.38	1,826,106	0.07	1,541	0.45	9,960	0.93	20,814
	Probable	1.22	0.82	31,972	13.16	514,297	0.06	672	0.29	3,499	0.61	7,369
	Total	**3.46**	**0.92**	**102,463**	**21.08**	**2,340,403**	**0.06**	**2,213**	**0.39**	**13,459**	**0.82**	**28,183**
Perspectivnaya	Proven	1.45	1.54	71,946	18.55	864,235	0.11	1,632	0.52	7,501	0.97	14,126
	Probable	0.83	1.33	35,682	13.82	369,402	0.09	785	0.37	3,086	0.69	5,711
	Total	**2.28**	**1.47**	**107,628**	**16.82**	**1,233,637**	**0.11**	**2,416**	**0.46**	**10,587**	**0.87**	**19,837**
Zavodskaya	Proven	0.24	0.72	5,511	21.73	165,757	0.13	299	0.86	2,050	1.71	4,047
	Probable	0.03	1.56	1,278	19.65	16,120	0.10	26	0.70	179	1.28	326
	Total	**0.27**	**0.80**	**6,789**	**21.53**	**181,876**	**0.12**	**324**	**0.85**	**2,228**	**1.66**	**4,373**
North Bestrushinskoye	Proven	-	-	-	-	-	-	-	-	-	-	-
	Probable	0.80	1.05	26,971	23.76	612,977	0.12	971	0.85	6,791	1.54	12,387
	Total	**0.80**	**1.05**	**26,971**	**23.76**	**612,977**	**0.12**	**971**	**0.85**	**6,791**	**1.54**	**12,387**
Total	***Proven***	*8.95*	*0.91*	*262,779*	*12.85*	*3,693,712*	*0.43*	*38,038*	*0.32*	*28,587*	*0.73*	*65,195*
	Probable	*12.05*	*1.03*	*399,349*	*8.44*	*3,269,230*	*0.22*	*26,153*	*0.37*	*44,920*	*0.70*	*84,476*
	Total	***21.00***	***0.98***	***662,127***	***10.32***	***6,962,941***	***0.31***	***64,188***	***0.35***	***73,505***	***0.71***	***149,671***

5.10 Environmental Issues

5.10.1 Introduction

WAI was commissioned by Kazzinc to carry out a review of environmental and social issues associated with the Ridder-Sokolniy Mining Complex in line with national and international requirements.

5.10.2 Environmental & Social Setting and Context

5.10.2.1 Landscape, Topography

The Ridder basin has a predominantly gently undulating topography with hills varying from 650m to 1,000m in height. The basin is surrounded by the Prohodny ridges to the south, up to 1,800m in height. To the north, the Ivanovsky ridges rise to 2,300m, with other lower mountain ranges also present. The region is seismically active with earthquakes up to 4 on the Richter Scale being reported in the Ridder area in the last 10 years.

5.10.2.2 Climate

The climate is sharply continental with hot summers reaching temperatures in excess of +40°C and cold winters dropping to -47°C. Snow cover is present from November to April. Average rainfall is 710mmpa. The prevailing wind is from the north-west. Average wind speed is 5.7m/s in the winter, and 3.5m/s in the summer.

5.10.2.3 Land Use and Land Cover

Soil horizons are expected to be thin and lacking in fertile topsoil – no information on soil characteristics has been provided. Flora is defined by several landscape zones, affected by site geography, altitude and terrain. There are numerous tree and grass species, in a mosaic habitat, with coniferous species predominantly found at higher altitudes and pine and mixed forest, including birch found in the lower foothills. There are flat areas of scrubby grassland, with apparent species abundance being somewhat limited by climatic conditions. There are 94 bird species in the area, the majority being non-migratory, and 90 animal species, including bear, mink, deer, wild cats and numerous rodent species. Two of the bird species are reported to be rare raptors.

5.10.2.4 Water Resources

Four rivers are present within the mining licence, namely Filipovka, Bystruka, Harivzovka and Bolshoy Talovka.

It is reported that two main aquifers are present beneath the site. The upper aquifer is contained within superficial Quaternary alluvial deposits which occur approximately 20m below surface, and locally extend to 120m in thickness. The lower aquifer is a fracture and fault controlled groundwater body within the Paleozoic sedimentary and sedimentary volcanogenic rocks. A highly fractured zone is reported to occur at approximately 800m.

5.10.2.5 Communities and Livelihoods

The mine is located in Ridder city, and a number of small villages are also present in the environs, such as Talovka village.

5.10.2.6 Infrastructure & Communications

The Ridder-Sokolniy Mining Complex is situated in a topographic basin 120km north-east of Ust-Kamenogorsk, connected by a metalled road, and freight and passenger railway lines. Other than the mine and associated infrastructure, there are no other significant industrial emissions sources in the Ridder area, and air quality is therefore considered to be good.

5.10.3 Project Status, Activities, Effects, Releases & Controls

5.10.3.1 Project Description & Activities

The Ridder-Sokolniy deposit was discovered in 1786.

Mined ore is treated at the Ridder concentrator, which has separate circuits for polymetallic ore and copper/gold ore from Ridder and other nearby Kazzinc operations (Tishinskiy and Shubinskiy ore and Tishinskiy and Staroye tailings). Zinc concentrate produced from polymetallic ore is treated at the Ridder zinc refinery, whilst lead and gold concentrates are sent to the Ust-Kamenogorsk lead smelter for pyrometallurgical treatment. Copper concentrates are sold to third parties, however a Kazzinc copper refinery is planned, and refurbishment of the lead smelter is planned (the old smelter is currently mothballed). The operational site also contains redundant surface workings, old waste dumps and Tailings Management Facilities (TMF). Tailings from the Staroye TMF are currently reprocessed in the Ridder concentrator, with plans to reprocess Chashinskoye TMF in the near future. Current process tails are sent to the Talovskoye TMF, located approximately 5km from the concentrator.

The underground minewater is neutralised and pumped 3.5km overland to a waste water treatment facility. An acid neutralisation facility (gypsum plant) and two paste backfill plants are also located within the project area. In residential areas, it is understood that historic underground workings have been backfilled with waste rock material, and that current operations in these areas are backfilled with paste tails. WAI understands that in areas where no surface infrastructure is present, mining subsidence is permitted to occur.

5.10.3.2 Land Ownership and Tenure

Kazzinc has the right to mine gold-polymetallic ores under the terms of the Contract for Subsurface Use (1997), which is valid for 25 years. The mining licence can be extended in duration by mutual agreement between Kazzinc and the regulatory authority. The current mining lease (2003) covers an area of approximately 12km^2. The mining licence also includes areas of residential development, and outlying areas that have not been disturbed by surface mining activity. The land allotment area encompasses current mining and processing infrastructure. Ancillary facilities, historic surface workings and storage facilities, and peripheral undisturbed areas.

> ***WAI Comment:*** *The WAI team has viewed a number of mine licences and permits and considers that the mine licences have been obtained in line with State requirements. It is understood that the future expansion of mining activity would not necessitate relocation of communities within the current licence boundary.*

5.10.3.3 Energy Consumption & Source

Power is supplied from two main lines from Ust-Kamenogorsk at 132kV and 113kV respectively. The Company owns a Hydroelectric Power Plant at Bukhtarma which feeds the Ust-Kamenogorsk electricity grid. No Polychlorinated Biphenyls (PCBs) are believed to be present. Energy consumption for 2009 was reported to be 528,990,778kWh at the Ridder-Sokolniy complex.

> ***WAI Comment:*** *At the time of the visit, the transformer station was well managed and maintained, and energy usage is reviewed as part of the Company-wide resource minimisation scheme. The 5% target is thought to be met in the energy department, apart from with regard to the heating system.*

Rational use of energy is assessed on a continuous basis to reduce costs, and plans to save energy are developed and implemented.

5.10.3.4 Mine Wastes – Rock

Waste rock is used for void backfilling underground. There is a government owned waste dump behind the concentrator, and Kazzinc has no responsibility for it.

Settled solids from the sulphuric acid neutralisation facility ('gypsum' plant) are piped to the aforementioned unlined Kryukovskoye mined-out open pit for evaporation of liquid fraction. The solids are allowed to accumulate in situ and the facility will eventually be capped and revegetated.

Waste rock is deposited in the mining subsidence area. Surface movement related to subsidence is monitored once a year. Precipitation entering this subsidence zone percolates through the rock mass and into the mine drainage system. Kazzinc considers that distribution of contaminants from this drainage water is localised within the subsidence zone.

The geochemical composition of solids from the sulphuric acid neturalisation facility was studied, and testwork on its further treatment using Russian Technology was performed. During the period of disposal of solids in the open pit, run-off was pumped to the mine water treatment facilities. In recent years the acid has not been neutralised, but sold.

5.10.3.5 Mine Wastes - Tailings

The Ridder Concentrator processes Pb-Zn, Cu and Au ore from Staroye tailings, Shubinskiy, Ridder-Sokolniy and Tishinskiy mines. After leaving the concentrator complex, the tails are piped in 2 separate lines. Within a short distance the tailings pipelines enter a transfer station where the 15% of the Ridder-Sokolniy and Staroye tails are diverted to one of the Paste Backfill Plants.

There are two Paste Backfill Plants at Ridder-Sokolniy complex, which process diverted Ridder-Sokolniy and Staroye tails for void filling underground. Excess diluted tailings are sent to the Talovskoye TMF via the main pipeline. The Talovskoye TMF is a valley impoundment located in the Bolshoy Talovka river valley. Tailings are discharged via a distribution pipeline. The TMF has a capacity of 120Mt, with a current volume of 70Mt.

The tailings are piped approximately 5km from the concentrator to Talovskoye TMF, via 3 pumping stations, two of which have emergency discharge ponds. At the first pump station all tailings are mixed and from this point onwards pumped in single pipeline with a standby line running parallel. There is no secondary containment below the pipeline, however 2 emergency discharge stations are present along the length of the route, which crosses the Filipovka River in two locations. Flow rate and pressure is measured at each pumping station, with an average flow of 4,000m^3/hour, translating to an annual average of approximately 4Mm^3/yr. The pipeline is inspected daily for corrosion, and some sections are currently being replaced.

Supernatant water from Talovskoye TMF is piped back to the concentrator, for use as technical water via the old Chashinskoye (State owned) TMF. None of the facilities are lined. The base of the tailings dam is formed by clay deposits of known quality and thickness which serve as an impermeable membrane and prevent water percolation to underground aquifer. Drainage facilities are constructed downstream of Talovskoye tailings dam by means of which drainage is intercepted and sent to the tailings pond.

Sulphuric acid from the zinc refinery can be neutralised if necessary, with crushed limestone and the slurry is pumped to two storage ponds ('Kryukovskie mined out open pits') for settlement. When full, decant water from these ponds is pumped to the Mine Water Treatment Plant. This area is not currently in operation since all the sulphuric acid produced is sent to the consumer.

Additionally, the Staroye old TMF (State owned) is being reprocessed by the Company. The dry tailings material is excavated and transported via dump trucks to a mixing station at the site, where water is added, and the resultant slurry is pumped to the concentrator for reprocessing. The Staroye TMF is expected to be exhausted by 2014. The same process is planned for the old Chashinskoye TMF, however this has not yet commenced.

***WAI Comment:** No engineering designs or geotechnical monitoring information has been reviewed for this structure. It is assumed that it has been designed in line with State requirements, and to withstand seismic tremors and seasonal meltwater intake, however WAI cannot comment on these aspects.*

Kazzinc reports that the stability of the TMFs is guaranteed by the project design which was approved by the State authorities. The Mechanobr Engineering Institute (Russia) who designed the Talovskoye tailings dam carries out continuous monitoring of TMF's operation. A grid of monitoring holes and movement pins are used to assess the dam stability. The company also monitors the dam levels, condition and compliance with design parameters. When the mine closure plan is developed, the environmental section will include an assessment of environmental pollution and the required amount of reclamation.

It is understood that the Concentrator's recycling water supply through Chashinskoye TMF circuit will be made redundant in the future. This would make the system less complicated, increase efficiency and reduce the risk to the environment.

Secondary containment should be developed for the length of the tailings pipelines, and at river crossings. A pumping station is present on the pipeline to receive tailings pipeline run-off when pumping is stopped.

5.10.3.6 Water Management & Effluents

WAI considers the following activities could result in surface and groundwater pollution:

- Transportation, surface stockpiling and loading/unloading of Tishinskiy and Shubinskiy fine fraction ore materials;
- Excavation, loading, haulage and deposition of Staroye tailings material;
- Dust blow from TMF surfaces;
- Dust blow from road transportation;
- Spills and leaks of hazardous materials and chemicals (e.g. reagents and tailings);
- Surface runoff transporting potential contaminants including heavy metals, hydrocarbons and reagents (e.g. waste rock and ore stockpiles);
- Breaches of TMF and settlement facilities containing contaminative materials;
- Seismic activity compromising containment facilities;
- Exhaust gases from underground mobile fleet and combustion processes;
- Gases from explosives usage;
- Localised methane emissions from microbial sewage treatment and landfill facility;
- Localised chemical and solvent volatilisation;
- Tailings delivery across rivers with no secondary containment;
- Unplanned discharge of untreated minewaters;
- Unsegregated deposition to the subsidence area, all waters will report to the mine water treatment facilities;
- TMF and storage/settling pond percolation;
- Leaching from backfilled rock/paste backfill in underground working, to be treated by the mine water treatment facility until clean;
- Absorption/dissolution of explosives residues and gases;
- Hydrocarbons from the mobile fleet;
- Leaching from contaminated soils; and
- Rebound of contaminated minewater at cessation of/or interruption to pumping. However, water treatment is envisaged until the water is clean.

***WAI Comment:** Surface storm run-off collection and discharge is managed at the mine site and surface diversion trenches are also present at the site. A new area for dry tails storage has been developed. When dry tails are transported, wagons are covered with canvas to avoid dust generation.*

A hydrogeological assessment was prepared during the mine development phase, and research into migration of material migration with groundwater and hydraulic connection with surface waters has also been studied. Areas of potential underground and surface water use by the population monitored.

It should be ensured that where sub-contractors are used, they adhere to company policies to minimise the potential for environmental pollution.

There are several discharge points to surface water receptors within the mining licence. The outlets discharge to the Filipovka and Bystruka rivers. Discharges to surface are permitted in accordance with national legislative requirements.

Discharge streams include industrial effluent from the compressor house, domestic effluent treated via microbial action and chlorination, and underground mine water neutralised by mixing with lime milk and settling in the Waste Water Treatment Plant and supplementary containment pond. Approximately 2000m^3 an hour of underground water is pumped to surface, treated and discharged in this manner.

Technical water for the Concentrator and ancillary operations (compressor stations laboratory) at Ridder Complex is sourced from the Bystrushinsky water reservoir, under the provisions of an abstraction licence.

Other than the diversion of the Bolshoy Talovka River, no surface runoff water management systems are present. Potable water is sourced from the Ridder city (Gramotuka watershed) mains water supply. The Company also has an abstraction permit for use of water for technical and potable supply. In 2009, 257,000m^3 were used by the complex, at a rate of 350,400m^3/day for potable supply and domestic needs. No technical water was obtained from this source.

There is an internal initiative for 2011 to reduce water and energy consumption by 5% with results reportedly favourable for water consumption and discharge rates.

> ***WAI Comment:*** *Although there is clearly adequate water supply to meet current operational needs, the Company assessed the possibility of recycling minewater for use at the Zinc Smelter and the Concentrator. Mine water treatment facilities are being expanded for this purpose with completion anticipated in 2012. This would potentially negate the need for river water abstraction and would reduce the potential for contamination of surface water features. WAI is encouraged by the Company's resource efficiency initiatives.*

A mine domestic sewage treatment plant is located within the mine site. Domestic sewage is delivered to the facility into two agitator tanks, where microbial degradation occurs. Treated water is chlorinated and discharged to the Bystruka River. Sewage solids collect in a cess pit and are periodically collected and disposed of off site by a specialist contractor.

Minewater is mixed underground with chemical grade lime milk to neutralise acidity, and is then pumped 3.5km at surface to the waste water treatment facility at a rate of 2,000m^3/hr. The water enters four settlement cells to remove suspended solids prior to discharge into a clay-lined secondary pond. Prior to discharge, the pH of treated waters is monitored at the waste water facility every two hours. During the site visit, pH was noted to range from 8.5-10.5. During the site visit, an odour was noted which may have resulted from the use of explosives underground. Water is finally discharged from this facility to the Zurhardt stream, and ultimately to the Filipovka River

Mine water is delivered to the surface from underground water sumps. Availability of water sumps at Levels 11, 13, 16 and 18 make it possible to collect water underground and periodically feed it to surface water treatment facilities. The design of the mine water drainage system eliminates any potential overtopping of the settlement pond at surface. A containment pond is provided to avoid discharge of untreated mine waters to the river.

***WAI Comment:** Refurbishment of water treatment facilities is currently being carried out with the installation of a lime liquor preparation and supply system instead of mixing underground in the mine workings. This should regulate pH during treatment. The mine closure plan includes long-term operation of water treatment facilities once mine workings are flooded, to ensure the quality of discharge water.*

5.10.3.7 Emissions to Air

Particulate emissions could result from the following:

- Transportation, surface stockpiling and loading/unloading of Tishinskiy and Shubinskiy fine fraction ore materials;
- Excavation, loading, haulage and deposition of Staroye tailings material;
- Dust blow from TMF surfaces; and
- Dust blow from road transportation.

In addition to the particulate emission sources outlined above, air pollution at Ridder-Sokolniy could result from the following processes:

- Exhaust gases from mobile fleet, railway, and combustion processes;
- Gases from explosives usage;
- Localised methane emissions from microbial sewage treatment and landfill facility; and
- Localised chemical and solvent volatilisation.

Sources of air emissions have been included in a draft emissions inventory. In line with national requirements, Kazzinc has plans to minimise volumes of emissions and discharges as per best practice.

5.10.3.8 Discharges to Soil

Potential soil contamination could arise at the operations via runoff from temporary stockpile areas, deposition of contaminated dusts, reagent/process spills, and emergency situations, such as a chemical spill or containment breach. At Ridder-Sokolniy complex, there are several issues which may lead to the above, including:

- Transportation, surface stockpiling and loading/unloading of Tishinskiy and Shubinskiy fine fraction ore materials;
- Excavation, loading, haulage and deposition of Staroye tailings material;
- Dust blow from TMF surfaces;
- Dust blow from road transportation;
- Spills and leaks of hazardous materials and chemicals (e.g. reagents and tailings);
- Surface runoff transporting potential contaminants including heavy metals, hydrocarbons and reagents (e.g. waste rock and ore stockpiles);
- Breaches of TMF and settlement facilities containing contaminative materials; and
- Seismic activity compromising containment facilities.

5.10.3.9 Waste Management – General

Limited wastes are stored at the site. Scrap metal and any timber is stored in metal containers at each area of the facility, and periodically removed by contractors. Domestic wastes, poor quality industrial wastes, including reagent containers, and other miscellaneous items are disposed of in an unlined subsidence zone resulting from mining, with this material being used as backfill. Reagent drums are neutralised and pressed before disposal to avoid contamination and recycling.

The transport fleet, fuel delivery and maintenance facilities are subcontracted out, and hence the Company does not handle any materials associated with these processes. Old oil from the substation transformers is removed by contractors and reused for combustion purposes.

Waste material is used for reclamation of the mining subsidence zone. It is reported that all percolation waters are intercepted by the mine water collection system, and treated prior to discharge. Although this system has been approved locally, it is not considered to meet best practice and should be further assessed in line with international standards.

5.10.3.10 Hazardous Materials Storage & Handling

Fuel delivery and storage is subcontracted to the city facilities, which have a storage area in the vicinity of the mine. Therefore, no fuels are stored at the sites.

Explosives are delivered by rail to an explosives magazine at a remote site to the north of the Zn refinery. From here, the explosives required for underground use are transported by secure rail wagon to Ridder mine, lowered in containers in the shafts and stored in intermediate underground areas prior to use. The average volume of explosives stored in the main facility equates to 120t. This includes ANFO, for which there is a manufacturing area (not visited by WAI) where the product is transported in 40kg bags. In underground areas the maximum volume of explosives stored is 2t in any one location.

The main railway line from Ust-Kamenogorsk crosses two rivers, the Ulba and Gramotuka on the way to Ridder. Freight on this line includes metal concentrates, chemicals including cyanide and sulphuric acid, together with clinker and explosives. The Tishinskiy concentrate is transported in uncovered wagons over a 24 hour period. Approximately, 140kt/month of ore is transported to Ust-Kamenogorsk.

Chemical reagents are delivered to the Company rail depot, and diverted onwards by rail to the Zn concentrator. There is an intermediate storage area to the north of the Zn refinery, in a remote site. These reagents include chemical grade lime, sulphuric acid, sodium cyanide, and various other process reagents. For hazardous chemicals, such as sodium cyanide, the trains have a police escort, and are equipped with neutralisation reagents in the event of a spill. Cyanide is delivered as granules, in metal drums, which are offloaded manually, and transported to a ventilated, alarmed storage area. In this area, the drums are loaded into a sealed dosing facility, which opens the drums, mixes the cyanide with water, and rinses the containers, which are then crushed. The cyanide liquid is then piped directly into the plant. Personnel working with hazardous substances had gloves, goggles and respirators, however full chemical suits are not worn. Emergency wash facilities exist.

Other chemicals are stored in separate areas in the reagent unit, however, these are not bunded, but it was reported that a drainage system for all effluent conveys any spills back into the process.

Emergency response plans for dealing with spills exist, and spills are either neutralised or diluted and washed with water. However, no spill containment kits are used.

> ***WAI Comment:*** *WAI considers that in general, storage areas are appropriate and well maintained; all storage facilities correspond with national safety requirements although WAI recommends acquiring spill containment kits for emergencies.*
>
> *WAI considers that the Company would benefit from demonstrating a commitment to compliance with the International Cyanide Management Code (ICMC). Transportation, handling and storage practices appear adequate.*

Where contractors are used, they are conversant with Company handling, storage and disposal protocols. Clear lines of responsibility, in the event of an incident, such as a cyanide or fuel spill, are established. Spill kits should be provided for containment of spills. The spill management plans should be extended to include disposal of containment spill mats, and these plans should be regularly tested via drills.

5.10.3.11 General Housekeeping

General housekeeping at the site appeared generally good, with most areas being maintained in a tidy manner.

> ***WAI Comment:*** *WAI considers that housekeeping at the site is good.*

5.10.3.12 Fire Safety

Fire extinguishers and fire safety stations are located at various points across the mine and plant site. All staff and contractors are given training in fire safety plans, emergency evacuation routes and fire fighting protocols, although these are not tested via actual drills. There is also a training log, outlining training received. The State mine rescue team, located in Ridder city, is responsible for underground fire management. A designated person in each area is responsible for inspection and maintenance of fire extinguishers.

> ***WAI Comment:*** *The fire management systems appear appropriate to the size of the operations, and WAI considers that these issues are being well managed by the health and safety team. WAI understands that Kazzinc intends to introduce more drill training systems, and recommends that this should be implemented as a priority.*

5.10.3.13 Security

Security control points are present at the entrance to the site, and control points are also included for all operational areas. These control points are permanently manned (two 12-hour shifts), and access is controlled via a 'pass' system.

> ***WAI Comment:*** *Whilst access to the main areas of infrastructure is appropriately controlled, the scale of the site means that it is not possible to prevent access to all areas. Additional hazard signage could be employed at dangerous areas, such as TMFs, storage ponds and old surface workings/ground collapse areas.*

5.10.4 Permitting

5.10.4.1 ESIA/OVOS

An OVOS (Russian equivalent to Environmental Impact Assessment (EIA)) produced in 2007 by Vostokvodoochistka is in place for the Ridder Mining Complex (including Tishinskiy mine and processing facilities), which covers extraction and processing. The Ecological Expertise for the 2007 OVOS concluded that the project would not have any negative environmental impacts, subject to the described mitigation measures being implemented.

WAI has reviewed the environmental monitoring and mitigation programmes which were approved for 2009-2010. These reports have to be prepared by an external registered contractor. Regular audits other than for ISO 14001 compliance purposes are not undertaken by State authorities.

> ***WAI Comment:*** *WAI considers that the Company is well aware of its obligations with regard to meeting national requirements, and seems to be performing the appropriate environmental studies in this regard.*
>
> *However, analysis of OVOS compliance with international standards should be performed to highlight any shortfalls. As required under the national Subsoil Use Contract, a mine closure plan has been developed with an associated closure fund held in a special Kazzinc account. A detailed closure plan will be developed 3 years before mine closure, as per national standards. This differs from international best practice requirements. An independent audit of all facilities at Ridder-Sokolniy mine was commissioned by Kazzinc in 2010 and performed by Ecoterra.*

5.10.4.2 Environmental Permits and Licenses

WAI considers that Kazzinc is in possession of all the relevant environmental permits as dictated by Kazakh legislation.

5.10.5 Environmental Management

5.10.5.1 Environmental Policy and Company Approach

According to the environmental activities plan, construction of a new site for handling Tishinskiy tails, located further from the river was completed, and the former site was closed. All company training materials are prepared by company Environmental specialists. The environmental budget is prepared on an annual basis by the environmental manager, based on the cost of permitting requirements and expenditure. For 2010, the environmental plan included measures for the following:

- Air – dust suppression on roads;
- Water – prevention of water pollution;
- Flora and Fauna – protection of species and tree planting;
- Production waste management;
- Radiation research and protection; and
- Environmental publicity programmes.

The sum allocated to the above for 2010 was 13.7 Million Tenge (approximately US$1M). This sum does not include any contingency for unexpected costs, such as pollution associated with e.g. a breach of Talovskoye TMF or unplanned closure of a facility.

> ***WAI Comment:*** *Expenditure for accident elimination or mine closure related contingencies are included as part of the mine closure fund and environmental insurance contract. Risk assessments are performed to identify priority environmental issues. Monies are then allocated to address these.*
>
> *WAI also considers that Kazzinc is proactive with regard to dissemination of information at key stages of project development. The Company is meeting national obligations, however WAI suggests that current practice should be specifically compared with international standards for consultation and disclosure. The Company relationship with the local community is very good.*

5.10.5.2 Environmental Management Staff & Resources

There are two Environmental Managers at Ridder Mining and Concentrating Complex, responsible for the Ridder operations, and other outlying sites in the area. The corporate Environmental Manager is based at the Company head office in Ust-Kamenogorsk. Environmental monitoring and sampling is carried out by the Company environmental laboratory, supported where necessary by external contractors. Corporate Environmental training is also provided to managers. Environmental training aspects are included within a training program provided by a special personnel training department. The Company has special procedures for managing sub contractors both in terms of environmental and health and safety performance. Health and Safety Training is expanded further in the Health and Safety section. Top tier managers receive external training, and in 2005 completed training in ISO 14001 EMS.

5.10.5.3 Systems and Work Procedures

The Company is accredited under the following international management systems:

- ISO 9001 – Quality Management (achieved 2004);
- ISO 14001 – Environmental Management (achieved 2006); and
- OHSAS 18001 – Occupational Health and Safety (achieved 2006).

The independent accrediting Company for the above standards performs annual audits. A general assessment of environmental conditions is made during the annual monitoring visits. Six ISO 14001 audits were performed internally in 2009, and seven non compliances were identified and measures were implemented in January 2010 to rectify these.

The Company has developed a number of long term corporate global programmes. These include: the reduction of sulphur dioxide emissions, the reduction of Pb particulates, the recycling of water at reprocessing of industrial wastes and technogenic mineral formations.

Kazzinc operates a transparent policy with regard to dissemination of information on their operations. Formal public hearings are held when a new development occurs at the mine, and regular annual meetings are also held.

> ***WAI Comment:*** *WAI considers that the environmental management at the mine, with regard to meeting national requirements is good. The Environmental Managers appear well aware of their duties, efficient and proactive, and aware of national legislative requirements. WAI is encouraged that the Company has obtained accreditation under international management systems, and is aware of the requirements to maintain these. Development of these systems is annually checked by the international certification authority, and their improvement is being noted.*

5.10.5.4 Environmental Monitoring, Compliance & Reporting

Environmental Monitoring plans, prepared by a registered licensed contractor (as stipulated by State requirements) are approved annually. Discharge and emission limits were set for 2010. The monitoring plan includes information on location of sampling, frequency and determinands for groundwater, surface water, discharge points, snow, air, dust and soils. Monitoring reports, based on the findings of the monitoring plan, are sent to the Environmental Ministry.

Groundwater is sampled via 8 boreholes in 4 locations across the project area. Surface water is sampled in 7 locations from the Filipovka, Bystruka, and the Bolshaya Talovka Rivers. When the concentrations are exceeded, mitigation measures to rectify the situation are implemented, to ensure that all site runoff is treated.

According to the monitoring programme, there are 13 monitored discharge points in 7 main areas which are sampled monthly for pH, copper, iron, lead, zinc, cadmium, manganese, ammonium salts, nitrate, and nitrite, and quarterly for the additional determinands of sulphates, suspended solids, dry residue, and oils. At the waste water treatment facility, the pH of the neutralised minewater is monitored by resident staff every two hours on a daily basis.

According to the monitoring plan, air is monitored in 9 locations at the operational mine site for dust and gas emissions. In addition one air monitoring point is located in a residential area on the periphery of the Sanitary Protection (buffer) Zone. Total dust is monitored in this location. The Sanitary Protection Zone (SPZ) for Ridder-Sokolniy Concentrators buildings is 1000m, and for Talovskoye TMF is also 1000m.

According to national legislation, maximum permissible discharges from mine site are established which are specified in the permit for substance emissions.

Continuous gas monitoring occurs in hazardous chemical storage areas, and in the plant, and alarms sound if limits are exceeded. Greenhouse Gas (GHG) monitoring does not currently occur, since GHG emission legislation is not currently well defined under Kazakh law. GHG sources at the site would potentially include emissions from fixed and mobile plant, the acid neutralisation areas, and lime preparation facilities and from the sewage treatment works. In 2010 an inventory of greenhouse gas emissions sources was produced and approved by the State environmental authority.

Noise is monitored at point sources, on an annual basis. Noise levels at potentially sensitive community receptors are not controlled. A working level of 80dBA is permitted, and where this is exceeded, Personal Protective Equipment (PPE) is provided, or mechanical improvements are made.

Soils are monitored in 6 locations on an annual basis.

Process tailings composition from all concentrator circuits is continuously assessed as part of the process flow in the plant. Tailings recycled water composition is analysed at Talovskoye and Chashinskoye TMFs.

Radioactive point sources are monitored at over 1000 locations, on a quarterly basis, using either mobile or automatic detection equipment. Radon exposure is also monitored twice per year in working areas, including the underground mine, and at planned construction sites together with scrap metal if required. The threshold level of 310Bcq/m^3 has not been exceeded at any locations where radon is measured.

Monitoring for air and water quality is performed by the Company environmental laboratory, with samples analysed at the Analytical Laboratory, which is nationally accredited. Groundwater and soils are monitored by specialist contractors. It is reported that full tailings analysis and recycling water analysis is performed by the Company Analytical Laboratory, however WAI has not reviewed the results.

Environmental test results are reviewed and reported monthly by the Company Environmental Managers. These reports include a comparison against both State and site-specific thresholds.

These reports are prepared and submitted to the Ministry of Environmental Protection to report on compliance. Furthermore, annual State environmental inspections are carried out to assess compliance and environmental management practices. Non compliances can either result from exceedance of thresholds or quantities, or inadequate management/mitigation measures, resulting in discretionary fines.

In 2009, the total fines payable by Kazzinc amounted to 93,000 Tenge (approximately US$600). This largely related to slightly elevated metal concentrations in discharge waters.

> ***WAI Comment:*** *The scope of the environmental monitoring programme has been developed in line with national legislative requirements, as testified by the limited penalties due to non compliances. The Kazzinc Environmental Managers, Laboratories and third party specialists are aware of compliance requirements and the analytical and sampling methodologies required to assess these. However, the monitoring programme should be compared with International standards and recognised best practices.*

A hydrogeological assessment of the mine operations was performed when the deposit reserves were calculated. Ongoing hydrogeological studies are being carried out as part of the operational phase by the RMCC hydrogeology service. Monitoring holes are located at specific points ensuring control of hydrochemical processes. This is especially important, given the sensitivity of groundwater supplies in the area. An OVOS was developed for the deposit development and approved by the State environmental expertise, which included baseline data characterisation.

Groundwater testing should also include chemical reagents used in the concentrator (e.g. cyanide and microbial parameters. When limits are exceeded, mitigation measures are implemented, to ensure adequate treatment.

With regard to surface water, WAI considers that the distribution and number of river sample points is good. Surface water sampling locations are considered adequate to assess the quality of river water and the potential influence of industrial discharges across the site. Ochre coloured discharge was noted at the Chashinskoye water discharge point, owned by the State which enters the Filipovka River catchment. This indicates high sulphate-ions levels in monitoring results, although when assessed in the context of many years of monitoring results, Kazzinc states that no high acidity has been recorded in run-off, and the level of contaminants is in line with State approved levels.

The quantity and location of point source air quality monitoring sites is considered adequate. Additional noise monitoring should be undertaken at the location of sensitive residential receptors, either within or outside the licence area.

WAI appreciates that the history of mining in the area has already resulted in previously disturbed and depleted soil formations. The State approved Environmental monitoring program includes monitoring of soil contamination within the zone of influence of surface facilities. A survey, financed by the government was performed to assess contamination in the area of the mine site on a 100x100m grid.

Ore rock waste is not stockpiled at the site. Mine ore is directly delivered to the Concentrator without stockpiling. Mine wastes are used for reclamation of the subsidence zone that has resulted from mining of the Ridder lode. Drainage water from this zone enters the mine drainage system. Periodic control shows the mine run-off pH is 7 or higher and the pH of flotation tailings is higher than 9.

The composition of the tailings liquid and solid phase is monitored each shift on a daily basis to prevent metal losses, or high level reagent discharge and to assess run-off and tailings acidity. The assessment should include a full analysis for tailings solids.

WAI considers that the detection limits for some determinant testing, such as sulphate, arsenic and mercury, is not sufficiently low to assess potential contamination. However these methods and detection limits are established in the Kazakh State register and therefore Kazzinc follows them.

WAI considers that the radioactivity monitoring performed is appropriate to the size of the operations.

Overall compliance with national standards appears to be good.

5.10.5.5 Emergency Preparedness & Response

There are Emergency Preparedness and Response Plans for various scenarios, such as a TMF breach, outlining responsibilities, actions and reporting requirements. However, these plans are not tested via actual drills, except in areas considered as more hazardous, such as the Talovskoye TMF and reagents storage area.

There are chemicals for spill neutralisation for cyanide and sulphuric acid, and other spills are flushed with water. There are no spill kits for containing spills. All spills are in the Concentrator and reagent storage area is collected by the surface drainage system, which reports to Talovskoye TMF.

5.10.6 Social and Community Management

5.10.6.1 Stakeholder Dialogue and Grievance Mechanisms

Internally, the main method for internal communication of environmental issues is via training, whilst health and safety issues are also communicated via signs and notice boards. An internal Kazzinc newspaper is also produced and distributed to employees. Externally, public hearings, as required by law, are advertised in the local paper twenty days in advance, when there is a new development, or an expansion to existing facilities. Additionally, quarterly meetings are held with community members. Furthermore, an annual programme of environmental presentations is developed and delivered to invited representatives. This is also reported in the local newspaper and on television. Tours of the mine facility for pupils, students and public organisations also promote transparency, and local awareness of the operations.

Internal occupational grievances can be raised during quarterly meetings, attended by workers and sector managers. Occupational related issues can also be communicated in writing to sector managers, via the health and safety manager for that area. There are also comment boxes for anonymous complaints, and a comment book for suggestions to improve working conditions. Additionally, managers allocate times during which personal complaints can be discussed.

External grievances can be made either verbally or formally in writing. Grievances are either addressed to the Company director, or the relevant sector head. Grievances are recorded and investigated by a team including a Company lawyer, and reported back to the complainant. The Company director is informed of the outcome together with an estimation of any compensation considered necessary. Compensation measures are discretionary and the final decision is formally reported by the director or relevant sector head. Environmental complaints resulting in Company action have been infrequent. In 2007 in response to a complaint about haulage dusts, an alternative haulage route was devised.

> ***WAI Comment:*** *WAI considers that the internal stakeholder dialogue and grievance mechanism is appropriate to the size and nature of the operations.*

5.10.6.2 Social Initiatives and Community Development

Ridder city has a population of approximately 58,000 people, with an additional 3,000 living on the outskirts or in villages. Part of Ridder city lies with the western boundary of the mining licence. Additionally, the villages of Kanalovka and Talovka lie within the mining licence, the latter being situated to the south of the Talovskoye TMF. In 2009, the total number of employees at Kazzinc was 3,518 with the vast majority living locally.

In line with Company policy, the following financial sums have been allocated in 2010:

- 1.5M Tenge for wedding or funeral costs;
- 1.5M Tenge for maternity pay;
- 10M for unused leave on resignation;
- 2000 Tenge per pensioner (the 3010 received a total figure of 7.7 M Tenge);
- Optional salary contribution to local Kazzinc orphanage – donations totalled 797,115 Tenge;
- 500,000 Tenge for city improvements/repairs; and
- 2M Tenge for additional community assistance.

In addition to the above, Company employees also receive other employment benefits, such as medical treatment, bereavement support, presents and parties for employees' children, a profit share bonus scheme, long service recognition, optional and membership of trade unions. Additionally, the Company has a scheme for teaching English to managers, with the opportunity on successful completion of the course to work overseas for up to two years, at the Company's expense. The Company also funds a technical university in Ridder, open to current employees, to enable them to further their training, and specialism. External experts are invited to provide guest lectures.

In addition to the above, the Company has undertaken a number of social initiatives to benefit the local community. These include:

- Maintenance of public roads;
- Provision of equipment for waste collection;
- Repairs to the hospital;
- Rebuilding a local church;
- Financial assistance and equipment to schools and kindergarten;
- Development of a local orphanage for 0-3 years;
- Scheme to find adoptive parents for local orphans in Ridder and Ust-Kamenogorsk; and
- Educational excursions to mine and processing facilities.

With regard to land take, in line with national policy, if acquisition or resettlement is required, the Company lawyer assesses the condition of the land/property that would be affected, and the Company compensates the individuals accordingly.

> ***WAI Comment:*** *WAI considers that the Company is working well with the local community, and is encouraged that a social development plan and supporting funds have been developed.*

5.10.7 Health & Safety

5.10.7.1 Health & Safety Management Arrangements

The Company has a central health and safety (H&S) management team in Ust-Kamenogorsk, with engineers designated for different sections of the operations. At Ridder, there are two corporate Health and Safety managers, who are responsible for a team of engineers, responsible for different areas. Additional inspectors are also present to undertake regular assessment of the facilities. The Company is also accredited for H&S management under the OHSAS 18001 scheme.

There is a comprehensive training programme, which varies in the level of detail, depending on the target audience. The environmental training covers issues such as dealing with spills, waste management, water and power consumption and how to manage and minimise environmental risks. Managers also receive training every 3 years, from State authorities. WAI viewed a number of personnel training logs.

There is a medical point in the concentrator, and all employees are first aid trained. In chemical storage areas there is also an emergency wash facility. The Company also has a special clinic in Ridder, where all employees receive annual medical checks. The Company has a contract with the State ambulance provider and the nearest hospital with an Accident and Emergency (A&E) department is in Ridder city, a few kilometres away. Spot tests for drugs and alcohol are also performed.

Noise, dust and gases are monitored in working areas, and alarms sound if safe conditions are exceeded. Additionally gas monitors are present in reagent storage areas, and respirators are also worn in these areas. PPE is provided to all employees, and additional PPE is provided when hazardous substances are handled.

A CCTV system is located in the main office, with screens monitoring various areas across the site.

> ***WAI Comment:*** *WAI considers that health and safety is very well managed at the site, with appropriate training being provided, and good response systems in place for personnel. PPE was worn in all areas, and H&S managers were proactive in their management of H&S issues, both with regard to national and international requirements. WAI would recommend implementing drills to test Emergency Response Plans, to assess their adequacy. Hazard suits should be provided to workers handling and unloading hazardous substances. Another minor recommendation is that alternative dust masks should be sourced, since the current design is cumbersome, and could result in cases of masks not being used.*

5.10.7.2 Performance and Accident Records

The Company generally has a good health and safety record, with 102 days lost to injury for the concentrator (total 607 staff) in 2009. Two fatalities occurred in 2008, and there were 11 injuries overall in 2009. A full investigation into fatalities was performed with an assessment of further actions required to avoid repeat incidents. Another fatality occurred on 11 November, 2010, due to an explosion in a lime preparation area. Up to this point, accidents had been infrequent, with the majority being assigned to negligence. The Company had an aim to reduce the number of accidents by half in 2010. Accident logs are maintained, and in the event of an accident, a full inspection and investigation, with remedial action, takes place. Daily informal inspections are performed, and there are also annual inspections from various State departments including: Sanitary Epidemiological, Police, Safety and Emergency Response units. Formal internal inspections are performed three times per year.

There are clear lines of responsibility in case of an accident, and disciplinary and re-training procedures exist if unsafe working practices are noted by H&S managers. The managers can also stop production and issue warnings if breaches are noted. These are then recorded, and remedial measures, with a given timescale for implementation are devised. The progress on implementation is then inspected by the H&S manager. Health and safety signs are displayed across the site, and appropriate signs were in place at all hazardous areas.

WAI Comment: *WAI considers that the health and safety management is very good at the site, and that the appropriate record keeping and improvement procedures are in place to deal with incidents. Overall the safety records are good, and all personnel seem committed to improving safety management.*

5.10.8 Mine Closure & Rehabilitation

5.10.8.1 Mine Closure Plans

It was reported to WAI that for the Ridder-Sokolniy mine and processing facilities, an initial estimate of potential closures costs was made, and a sum of money was deposited in a protected Company fund, for this purpose. This figure amounted to US$3.7M. In 2008, this figure was revised to US$4.1M, based on outline closure concepts. This sum does not include the Chashinskoye and Staroye TMF, which are the subject of a separate closure fund of US$4.8M established in 2000 and due for revision in 2011. Typical closure measures include demolition of surface structures, stripping of utilities, stabilisation of mining related features, earthworks and revegetation. It is reported that standard post-closure monitoring includes: rivers and groundwater, minewater, dust, soils, flora and fauna. It is also reported that post-closure aftercare is likely to include the continuation of minewater treatment and pumping. At Tishinskiy, this is envisaged for 12 years, before the workings are allowed to flood and groundwater rebound occurs. This period has not been estimated for Ridder-Sokolniy.

5.10.8.2 Financial Provision for Closure

It is reported that the closure fund will periodically be updated during the mine life, with a final detailed closure plan to be prepared 2 years before actual closure, in line with State requirements. The Company provides training in both mining and non-mining related sectors, and aims to provide employment in alternative jobs, once closure has occurred. Progressive reclamation is carried out, e.g., demolition of old buildings, and rehabilitation of the land.

Kazzinc closure plans are understood to be in line with the national requirements but are not compliant with international standards, since it is considered best international practice to develop a final closure plan prior to operations commencing, and consequently the Kazzinc closure plan and funds should be regularly reviewed, and amended to reflect current and future obligations. Systems should also be put in place in the event of unplanned company closure, as required by international best practice.

5.10.9 Conclusions

5.10.9.1 Environmental and Social Liabilities & Risks

There has been mining activity in the Ridder area since 1786, via surface and underground workings, with concentration facilities since the 1920s. Prior to 1991, mining and processing at Ridder-Sokolniy was State owned and operated. Kazzinc started operating at Ridder in 1997. The mineral rights for Ridder-Sokolniy mine remain State owned, and operations take place under the Subsoil Use Contract with the government. Additionally, Kazzinc leases Staroye and Chashinskoye old TMFs for reprocessing purposes. The Company also uses the Kryukovskoye old open pits for temporary disposal of gypsum slurry produced from RMV sulphuric acid neutralisation. The Company is responsible only for liabilities generated via their current operations.

The Company has environmental insurance, to cover environmental damage up to a maximum value of 22 Million Tenge (US$146,880).

As previously stated, under the mining agreement, if no surface infrastructure is present, it is permitted to let subsidence occur. As a result, there are areas within the mining licence which have subsided, including the site currently used as the Company waste disposal facility. It is the intention of the Company to prevent subsidence in residential areas, or areas with surface infrastructure, by using rock waste and paste backfill to stabilise underground workings. Paste backfill is pumped into underground voids via a series of fixed manifold outlets in a single pour. The progress of the pour is monitored from surface using a dip meter equivalent, to ensure that the material reaches the void roof. For every 1,000m^3 of paste backfill produced, three sub samples are prepared and tested over a three month period, to assess density and uni-axial strength throughout the solidification process. It is also reported that core samples of in situ paste backfill undergo the same testing processes after setting has occurred.

The Company does not have any liabilities associated with contaminations prior to 1997 when it was established. The Company prepared an OVOS for the waste dumps and determined boundaries for State responsibility. Similarly as for Staroye and Chashinskoye tailings dams, liability for water run-off and soil is delimited between the parties in the RMCC OVOS. Liability in the event of a claim, particularly with regard to subsidence, which may be due to historic workings, is determined by the national legislation.

Underground voids are assessed by underground surveys, and measures have been taken to prevent collapse due to incomplete backfilling.

The process of voids backfilling, material composition and backfill properties are controlled by RMCC backfilling laboratory.

5.10.9.2 Compliance with Local and International Standards and Expectations

The following is a list of WAI's main conclusions and recommendations with regard to the Ridder-Sokolniy mine and processing operations.

- Current environmental management is performed in line with national requirements, and WAI is encouraged by the Company's accreditation under environmental, quality and health and safety management systems. The implementation of these systems is translated to practical environmental improvement and sustainability across all management tiers and operational sectors;
- Environmental risks, associated targets and an action program were assessed during due diligence carried out in 2010. The use of Key Performance Indicators (KPIs) and risk assessments should be considered to achieve this;
- The approved OVOS has been prepared in line with national legislation, but it is recommended to renew the OVOS for compliance with the international best practice;
- The environmental monitoring programme meets national standards, but comparison with international standard requirements should be performed, and amendments made as required;
- The responsibility for environmental monitoring is centralised, such that monitoring performed by different departments and contractors is fully incorporated in the environmental monitoring programme, and as such all contaminative sources can be addressed. The results are recorded in a central database, in order to identify trends and point sources/plumes, for each determinant across all media;
- Enhanced environmental protection along the tailings pipeline should be considered, especially at river crossings. Consideration should also be given to using all emergency discharge points;
- The current landfill facility should be assessed in line with international standards;
- Consideration should be given to adherence with the ICMC;
- Health and safety is generally well managed, but could be improved by the introduction of practice drills. Apart from the recent fatality, health and safety performance to date has been good;
- Site security is good, however extra signage could be introduced to prevent access to dangerous areas;
- Progressive rehabilitation of sterile areas of the mine site should be implemented for stabilisation and environmental protection purposes;

- The current closure estimate for Ridder-Sokolniy, is in compliance with national legislation but should be reviewed against international best practice, and include provisions for unplanned company closure;
- Internal and community dialogue processes are developed in line with national requirements; and
- The Company demonstrates a good working relationship with the local community and has a well developed social plan and fund.

5.10.9.3 Recommendations for ESAP

WAI considers that given the scale of operations at Ridder Mining Complex, a gap analysis should be prepared by Kazzinc, to inform the production of an action plan, which should also consider the above recommendations.

6 TISHINSKIY DEPOSIT

6.1 Introduction

The Tishinskiy deposit is situated on the south-eastern spurs of the Ulbinsky Ridge, 200-400m east of the Ulba River and 18km south-west of Ridder. The collar of the main shaft (Tishinskiy Shaft) is at an elevation of 644.3m and the neighbouring mountains rise to 1,000-1,200m above sea level.

The Tishinskiy mine is linked by rail and asphalt road, which lies adjacent to the mine with Ridder and Ust-Kamenogorsk.

6.1.1.1 Mineral Rights and Permitting

Kazzinc holds the right to mine pyritic polymetallic ores under the terms of the 'Contract for Subsurface Use' dated 21 May 1997. The Contract is valid for 25 years from the date of the licence issue (i.e. 21 May 2022) and can be extended by mutual agreement between Kazzinc and the issuing authority.

The current mining lease covers an area of 3.8km^2 and was issued by the Ministry of Energy, Mineral Resources and the Environment of the Republic of Kazakhstan in November 2002, superseding the previous mining lease.

The lease permits mining to a depth of -590m relative to Baltic Mean Sea Level. The deepest mining level is currently at an elevation of approximately -230m, the deepest shaft reaches -345m and mineralisation locally extends down to an elevation of -570m (mineralisation has not been studied below the depth level of -570m).

The boundaries of the lease are defined as detailed in Table 6.1 below.

Table 6.1: Lease Boundary Points

Boundary Points	Geographical Coordinates	
	Latitude N	Longitude E
1	50°15'46.6"	83°21'58.2"
2	50°16'31.3"	83°20'44.1"
3	50°17'00.9"	83°20'43.5"
4	50°17'16.3"	83°21'14.1"
5	50°16'51.2"	83°22'32.5"
6	50°16'24.5"	83°22'32.8"

6.1.2 Project History

The Tishinskiy deposit was discovered during prospecting of the Butachihinsko-Kedrovska shear zone in 1958. Preliminary and detailed exploration of the new discovery was completed in mid-1963. Up to August 1961 the deposit had been explored, principally by surface drilling, to a depth of 450m from surface. By the end of June 1963 the central part of the Main Lode had been delineated to a depth of 750m and the flanks of the deposit to a depth of 450m. Mine development and detailed underground exploration began in July 1963. Whilst underground work was in progress, drilling from surface continued until the end of 1983 focusing primarily on deep levels of the deposit. The central part of the Main Lode and its eastern flank were delineated to depth of 1,200m and the western flank to a depth of 900m (Tishinskiy Shaft collar being at 644m).

Exploration was initially (to 1957) conducted by Altay Base Metals Exploration (ACMR), then jointly by Leninogorsk Geological Exploration Expedition (Leninogorsk GRE) and Leninogorsk Polymetallic Complex (LPC) and, from 1997, by Kazzinc (successor to LPC).

A summary of the exploration conducted up to June 1999 is given in Table 6.2 below.

Table 6.2: Exploration Summary (1954 – 1999)				
Activity	Unit	Exploration phase		
		1954 - June 1963	July 1963 - 1983	1984 - June 1999
Geological mapping 1:10,000 scale	km^2	59		
Prospecting traverses				
1:25,000 scale	km	756		
1:10,000 scale	km	*1,243*		
Geological mapping 1:2000 scale	km^2	8		
Cartographic drilling	m	51,300		
Pitting	m	18,350		
Trenching	m	2,810		
Prospecting and structural drilling	m	20,000	20,693	
Exploration drilling				
from surface	m	60,900	78,505	
underground	m	1,000	51,062	34,018
Exploratory underground workings	m	1,332	9277,0	10,794

Mining commenced in 1964 from an open pit sited in the centre of the deposit. Open pit production ceased in 1978, overlapping with underground mining, which began in 1969, initially Level 6 reserves were mined employing sub-level stoping with backfill using hand-held equipment, and from 1976, mechanised sublevel stoping with backfill. At that time (1976) the mining of open pit's west wall reserves was started above Level 5 employing sub-level caving method. The current underground mine layout was developed on the basis of a project prepared by the Kazgiprotzvetmet Institute in 1991.

6.2 Geology and Mineralisation

6.2.1 Regional Geology

The Tishinskiy deposit is situated in the central portion of the Butachihinsko-Kedrova shear zone, which adjoins the south-western flank of the Ridder graben. In contrast to the Ridder graben, in which Silurian to Middle Devonian volcano-sedimentary sequences are found almost in the same position as they were laid down, Palaeozoic formations in the Butachihinsko-Kedrova shear zone have undergone strong polyphase folding and intense shearing. The deformed formations comprise high grade metamorphic Ordovician basement rocks and weakly metamorphosed volcanogenic-sedimentary Devonian-Upper Carboniferous rocks.

The structure of the Butachihinsko-Kedrova shear zone is characterised by major WNW to NW trending folds with steeply dipping limbs and pronounced axial plane schistosity. As a result of Hercynian tectogenesis, Middle-and-Upper Devonian formations of Kedrovsko-Butachilkhinsky zone were formed as NW-trending folds, the major of which are the Siniushinsky anticlinorium and the Bystrushinsky and Beloubinshy synclinoriums. The Thishinsky deposit area covers the structures of south-west limb of Siniushinsky anticlinorium and north-east limb of Bystrushinsky synclnorium and is situated in the central part of Kedrovsko-Butachikhinsky shear zone which adjoins the south-east flank of the Ridder graben.

The structure of the Butachihinsko-Kedrova shear zone is characterised by major WNW to NW trending folds with steeply dipping limbs and pronounced axial plane schistosity.

6.2.2 Local Geology

Tishinskiy is a stratabound deposit developed along the E-W trending subvertical contact between the Ilinskaya and Sokolnaya Formations, both of Middle Devonian (Eifelian) age. The upper sections of the stratigraphically older Ilinskaya Formation (D2e1il) underlie the northern part of the deposit area. They consist of dark green chloritised tuffs and lavas of andesitic to basaltic composition, with intercalations of reworked tuffs, siltstones and fine grained sandstones. The basal part of the younger Sokolnaya Formation (D2e2sk), underlies the southern part of the deposit area. It consists predominantly of dark grey to black carbonate-pelitic siltstones. Both formations are cut by rhyolite porphyries and felsites which have been variously interpreted as sub-volcanic bodies associated with Devonian volcanic activity, or late Palaeozoic intrusives.

At the deposit, the contact zone between the Ilinskaya and the Sokolnaya Formations has been subjected to intense shearing and hydrothermal alteration over a width of around 400m. The alteration displays zoning with quartz-sericite and sericite-quartz in the centre through quartz-carbonate-chlorite-sericite in the intermediate zone to quartz-carbonate-chlorite in the outer zone.

A major proportion of the resource is contained in three mineralised bodies within the Main Lode, which is elongated in an east-west direction, with a near vertical northerly dip.

Main Orebody, also known as Orebody No.1 extends over a strike length of 1,250m from surface to a depth of 1,270m corresponding to level 22 at the absolute elevation of -590m. The central part of this body, about 500m in strike length and up to 60m in width, contains three subvertical lenses of massive sulphides (Western, Central and Eastern) with a combined strike length of about 200m, enveloped in disseminated mineralisation. The Western and Central lenses peter out downwards on levels 14 to 18 at the absolute elevations of -110m and -350m respectively and the Eastern lens pinches out at about the zero datum (10m below level 12). Average widths range from 6.5m to 17m. Lower grade disseminated mineralisation with much reduced widths (generally less than 10m) occurs in the Western Shaft section of the mine on the western flank of the deposit above level 8 and on the eastern flank, where it peters out rapidly towards a major transverse fault.

Orebody No.67 has a strike length of 1,000m, dip extent of 700m and an average width of 3.7m. Orebody No.1011 has a strike length of 550m, dip extent of 400m and an average width of 1.7m. On its flanks, the Main orebody divides into a number of parallel tapering branches to the east and west and gradually fades out.

Lenses and small vein-like bodies on the flanks of the deposit range from 25m to 50m in strike length and from 40m to 150m in down-dip extent.

Based on surface drill holes, the deepest parts of the deposit have been interpreted as lenses of disseminated mineralisation, which are open at depth. Underground holes intercepted lenses of massive sulphides among disseminated mineralisation on the western flank of the deposit indicating that massive sulphide mineralisation is also present.

Host rocks are strongly foliated carbonate-sericite-quartz schists and massive metasediments of the Sokolnaya Formation and, locally, carbonate-chlorite-sericite-quartz schists derived from volcanogenic rocks of the Ilinskaya Formation.

The southern contact (hanging wall) of Orebody No.1 coincides with a series of closely spaced faults. Lateral movements along these faults have produced two major gouge zones, which extend along the Western and Eastern massive sulphide lenses over a strike length of up to 170m and down to the bottom of the deposit. When disturbed, the gouge behaves like quicksand. Ground conditions are also very difficult along the northern contact of the orebody, which is strongly foliated along with the host sericite-carbonate-chlorite-quartz schists. The ore itself is foliated and cut by flat joints. This combination renders underground workings susceptible to roof failures and seriously impedes stope cleaning.

The available geological information suggests that Tishinskiy was originally a stratabound volcanic hosted deposit that has subsequently been subjected to intensive dynamic metamorphism and shearing producing plastic flow folds, boudinage structures, and lit-par-lit crystallisation schistosity. The deformation has redistributed primary sulphides and consequently modified the morphology of the deposit.

6.2.3 Mineralisation

Four primary and two secondary mineral associations have been recognised at the Tishinskiy deposit. The prevalent associations are: (1) A polymetallic (chalcopyrite-galena-pyrite-sphalerite) association, and (2) A copper-zinc (chalcopyrite-sphalerite-pyrite) association. Minerals representing all associations are listed in Table 6.3 below.

Table 6.3: Mineral Associations

	Metalliferous Minerals			Gangue Minerals		
Mineral Association	**Main**	**Abundant**	**Rare**	**Abundant**	**Main**	**Rare**
Oxidised Zone	Smithsonite, Cerussite, Goethite, Hydrogoethite, Beaverite	Malachite, Jarosite, Anglesite, Azurite	Native copper, Cuprite, Manganese hydroxides, Calamine, Aluminosilicate, Chrysocolla		Quartz	Halloysite
Secondary Sulphide Enrichment Zone	Chalcocite	Covellite	Bornite			
Pyritic	Pyrite		Rutile, Melnikovite, Pyrrhotite, Cobaltite	Quartz	Dolomite, Prochlorite	corundophyllite
Copper-pyrite	Chalcopyrite, Pyrite		Bismithite	Quartz, Dolomite	Prochlorite	Breinerite, Mesitite (Ferroan magnesite)
Copper-zinc	Chalcopyrite, Sphalerite	Pyrite	Galena, Altayite	Quartz	Dolomite, Calcite	Magnetite, Maghemite, Graphite
Polymetallic	Sphalerite, Galena, Chalcopyrite	pyrite	Tetrahedrite, Tennantite, Native gold, Native mercury, Hessite, Tellurobismuthite, Cavalerite (?), Claustthallite (?), Ilmenite, Arsenopyrite	Quartz, dolomite	Prochlorite, Phengite	Calcite, Albite, Gypsum

Disseminated veinlet-type polymetallic mineralisation predominates, but massive sulphide lenses of various sizes occur in the central parts of mineralised bodies.

The average ratio of pyrite and chalcopyrite to sphalerite and galena in disseminated mineralisation (which comprises about 90% of all mineralisation) is 1.5:1. The same ratio in massive sulphides is 0.6:1. The average Cu:Pb:Zn ratio to a depth of 1,000m is 0.46:1:5.4. The lead content decreases with depth and eventually, at a depth exceeding 1,000m, is lower than the copper content. The gangue composition also changes with depth, with magnetite, maghemite and graphite appearing in copper-zinc mineralisation at depths exceeding 1,000m. There is also a weak lateral metal zoning marked by a gradual eastward increase in the copper content and a pronounced drop in the gold content on the western flank. The pyrite content is less than 5.0%.

6.3 Exploration Works

6.3.1 Sample Collection

The current resource estimates are based on results of diamond core drilling from surface, underground mapping and sampling and underground diamond core drilling.

According to the Soviet resource classification system, which is still utilised by GKZ (Republic of Kazakhstan), the Tishinsky deposit was classified as being of Group II category, in the four-tier classification of mineral deposits with respect to geological complexity. Group II comprise of mineral deposits of considerable overall dimensions and moderate complexity.

According to specific instructions supplementing the Soviet resource classification, (under which the mines in Kazakhstan have a statutory requirement to report their reserves under to GKZ (RK,). a drilling grid of 100m by 100m was required for the delineation of a C_1 category resources for mineral deposits of that complexity group. The required sampling density has been achieved in the Main orebody on the currently developed deeper levels 11 to 16 and partially on the undeveloped levels 17 and 18. The same instructions defined a drilling density of 200m by 200m for the delineation of a C_2 category resource.

Detailed pre-production drilling and channel sampling was carried out by Kazzinc on the mining areas on grids ranging from 12.5x10-20m to 25x25-30m. Kazzinc considers this grid density sufficient for a resource upgrade from the C_1 category to the B category.

6.3.1.1 Surface Diamond Core Drilling

The main drilling campaigns were carried out by Leninogorsk GRE during 1958-1963. Surface drilling continued to 1983. Drill holes were sited on the northern side (in the footwall) of the Main orebody on profile lines bearing 180° except for some early holes, which were collared on profile lines bearing 12°.

Early holes were angled 80-88° to the south and drilled at 150-130mm external diameter, which were reduced to 110mm diameter (or occasionally to 91mm) at depth. Directional drilling with wedges was introduced in 1964. Holes were drilled vertical to depths of 600-800m and then wedged to deflect upwards. External bit diameters were 112mm to a depth of 25-30m, 93mm to 70-80m depth, 76mm to 600-800m depth and then 59mm at greater depths. Most holes naturally deviated to azimuths of 160-175°.

Drill collar locations were surveyed with instrumental methods. Down hole surveys were carried out at approximately 20m intervals using a variety of downhole survey instruments. Inclinations of the deepest portions of drill holes (final 150-200m) were generally determined using only acid-etch tubes with bearing directions extrapolated from previous measurements.

Kazzinc reported that core recoveries from mineralised intervals were generally better than 70%.

6.3.1.2 Underground Exploration and Resource Delineation Methods

Underground exploratory workings in the central part of the Main orebody consist of drives, which are generally oriented on a bearing of 90°, and crosscuts on a bearing of 180°. Crosscuts on the western flank of the Main orebody, where the overall strike changes to WNW, are oriented on a bearing of 228°. The average distance between crosscuts is 100m. With levels being developed 60m apart, this layout of exploratory workings forms a regular three dimensional grid of 100m along strike by 60m down dip. This layout is appropriate to the overall structural orientation. Historically, crosscuts on levels 3, 5, 6 and 7 were driven across the whole width of the Main orebody at 50m intervals.

Superimposed on the exploratory workings are extraction drives, crosscuts, sublevels, service raises and other underground openings developed in the process of stope preparation. The density of such workings varies from 10m to 20m along the strike and from 12m to 20m along the dip.

Geological documentation for each opening consists of 1:100 scale mapping of both walls and, in areas with more complex geology, also mapping of the backs, with annotations and descriptions.

Direct underground exploration is augmented with underground diamond core drilling. Underground holes are collared along drives at 12.5-25m intervals and drilled as fans oriented either north or south. On levels 11 and 13 in the central part of the Main orebody (crosscut lines 25-29), where the lode displays a flexural bend, changing its orientation from 90° to 45-30°, some exploration drill holes were oriented 265-275°. Underground drill holes on the western flank of the deposit (profiles 6 west to 5 west) were oriented on a bearing of 228° consistent with the change of strike in that area.

Most holes were drilled at 59mm diameter, but some horizontal holes were started and drilled up to 120m at 76mm diameter. A few deeper holes were completed in 46mm.

As with surface drilling, good core recoveries were difficult to achieve due to the presence of pervasive schistosity and strong localised fracturing and faulting. The overall core recovery in underground drill holes completed during the period 1984 to 1999 (in horizontal holes) was reported as 69.4%, whilst the overall core recovery from mineralised intersections was reported as 71.9%. The current situation has not changed.

Drill hole collar positions were surveyed with instrumental methods. Hole deviation surveys were carried out in approximately 55-60% of underground drill holes completed during the period 1963-1999. Changes in drill hole directions between the neighbouring survey points (at 20m spacing) were random and did not exceed 2°, which is less than the precision of survey instruments. As shown by control surveys of the actual piercing points in underground workings, the accuracy of the determination of X, Y and Z coordinates of the piercing points ranged from 0.7-1.1m.

Recorded drill hole depths and locations of mineralised intercepts were verified using grades recorded on roentgen-radioactive logs. Currently in-fill drilling is conducted with a Diamec 252 rig.

6.3.1.3 Sampling Techniques and Quality

Drill core recovered from mineralised intercepts is divided into 1m long samples or slightly shorter/longer lengths which are determined by geological factors: lithologic contacts or qualitative changes in mineralisation. Low grade sulphide mineralisation elsewhere is sampled over core lengths of 1-2m. Only core recovered from 76mm diameter holes is sawn along the core axis. Core from 59mm and 46mm diameter holes has been sampled in its entirety.

A comparison of analytical results on drill core samples with results of roentgen-radiometric logging (RRK) did not reveal any systematic errors.

Underground workings were sampled by cutting channels along horizontal lines over mineralised intervals exposed in crosscuts and from wall rocks stepping out 3-5m from visible limits of sulphide mineralisation. Channels are typically 10cm wide, 3cm deep and generally 1m in length. Only one side wall was sampled. Drives through mineralised bodies were sampled by taking horizontal channels from faces exposed at intervals ranging from 3m to 25m as the drives were developed.

Starting from 1982, channel samples were collected using pneumatic hammers and diamond rock cutting saw. Earlier samples were taken using chisel and hammer. The channel sampling quality was monitored by taking control channel samples immediately above and adjoining the existing channels. As reported by Kazzinc, differences between results on routine and control samples did not indicate any systematic bias.

6.3.2 Sample Preparation

The sample preparation scheme used at all exploration stages was based on the Richard-Czeczott formula Q=kda, where Q is the minimum sample quantity at a given stage of volume reduction, d is the diameter of the largest fragments defined as the screen size that retains the largest 5% of the mass, k is a coefficient dependent on the distribution irregularity of the mineral of interest and a is a coefficient related to the roundness of mineral grains (generally approximately 2).

The coefficient k is the key parameter. In general terms, the lower the coefficient k is, the better it accounts for the erratic distribution of minerals. In this instance, the k parameter of 0.16 has been selected after a series of experiments using schemes with varying k coefficients.

The same sample preparation scheme is used at the Ridder sample preparation facility today. In accordance with the adopted scheme, samples are crushed in three stages to less than 2mm and then divided to obtain approximately 3kg sub-sample. A duplicate is retained for mineral processing and other purposes, whilst the remainder is ground in roll grinder, sieved through a 1mm screen and split. A 1.5kg sub-sample is pulverised and sieved to pass 0.074mm screen. After mixing, sub-samples are collected for Cu, Pb and Zn analyses (50g), gold and silver analyses (300g), internal and external control analyses (500g). A 650g duplicate is retained for reference purposes.

The scheme is considered to be appropriate for the determination of base metal grades, but Kazzinc are aware that a scheme based where k=1, in which the first split is done at 1mm grain size, would be more appropriate for precious metals. Control analyses on duplicates representing different types of mineralisation and different grade classes do not reveal any significant discrepancies (see QA/QC section).

6.3.3 Sample Analysis

Analyses are currently performed at the Ridder MCC laboratory, but were in the past also performed at the Leninogorsk GRE and PGO Vostkazgeologia laboratories. Past and current procedures and methods are essentially the same. Internal and external control analyses on duplicates of routine samples have not revealed any significant random or systematic errors.

6.4 Proposed Exploration Drilling for Deep Horizons and Flanks

Kazzinc has proposed an extensive exploration programme of deep drilling at Tishinskiy in 2011. These holes envisage intersecting the main ore zone at depth for extension of the mine and on the flanks to

The programme consists of:

- Drilling of x17-18 drill holes in the center and on the western flanks (and to some extent on the eastern margin) of the Main ore zone between the -410 and -470m elevations at 18 to 21 levels; and
- Drilling x4 or 5 very deep directional drill holes to intersect the Main ore zone at the -600m elevation which lies just below -22 level. These holes will be from 1,500-1,600m long.

Exploration drilling is to be continued through 2011. The exploration budget and programme is given in Table 6.4 below.

> ***WAI Comment:*** *WAI has reviewed the current drilling practices on site and has found them to be well implemented. The recording of drill hole data (logging and surveying), sampling procedures and drilling practices were of very good standard.*
>
> *WAI has reviewed the proposed exploration programme and budget for 2011 and has no reason to believe it is not fit for purpose.*

Table 6.4: Planned Exploration Works For 2011 Proposed by Kazzinc

Object	Units	Total 2011	Same by Quarters			
			1	2	3	4
1. NW Flank of Bystrushinskaya Lode	m	5,600	1,400	1,400	1,400	1,400
	US$	685,714	171,429	171,429	171,429	171,429
	k tenge	100,800	25,200	25,200	25,200	25,200
2.Western Flank of Zavodskaya Lode	m	4,000	1,000	1,000	1,000	1,000
	US$	489,796	122,449	122,449	122,449	122,449
	k tenge	72,000	18,000	18,000	18,000	18,000
3.Western Flank of 2nd Ridderskaya Lode	m	8,000	2,000	2,000	2,000	2,000
	US$	979,592	244,898	244,898	244,898	244,898
	k tenge	144,000	36,000	36,000	36,000	36,000
4. Deep Horizons of Belkina Lode	m	20,000	5,000	5,000	5,000	5,000
	US$	2,448,980	612,245	612,245	612,245	612,245
	k tenge	360,000	90,000	90,000	90,000	90,000
5. South-East Flank of Pobeda Lode	m	11,000	3,000	3,000	2,500	2,500
	US$	1,346,939	367,347	367,347	306,122	306,122
	k tenge	198,000	54,000	54,000	45,000	45,000
6. Ore Occurence of Borehole No. 2426	m	10,400	2,600	2,600	2,600	2,600
	US$	1,273,469	318,367	318,367	318,367	318,367
	k tenge	187,200	46,800	46,800	46,800	46,800
7. Ore Occurence of Borehole No. 2437	m	10,400	2,600	2,600	2,600	2,600
	US$	1,273,469	318,367	318,367	318,367	318,367
	k tenge	187,200	46,800	46,800	46,800	46,800
8. Ilinskoye Ore Occurence	m	4,800	1,200	1,200	1,200	1,200
	US$	587,755	146,939	146,939	146,939	146,939
	k tenge	86,400	21,600	21,600	21,600	21,600
9. Bakhrushinskoye Deposit	m	16,000	4,000	4,000	4,000	4,000
	US$	1,959,184	489,796	489,796	489,796	489,796
	k tenge	288,000	72,000	72,000	72,000	72,000
10. Deep horizons of Tishinskiy Mine	US$	897,959	224,490	224,490	224,490	224,490
	k tenge	132,000	33,000	33,000	33,000	33,000
	m	96,200	24,300	24,300	23,800	23,800
Total Exploarion Works of Ridder Mining and Processing Complex	US$	11,942,857	3,016,327	3,016,327	2,955,102	2,955,102
	k tenge	1,755,600	443,400	443,400	434,400	434,400
	US$	897,959	224,490	224,490	224,490	224,490

6.5 Current Mineral Resources Estimate

6.5.1 Topography and Underground Survey

Underground survey data has been provided by the client as wireframes in Micromine format, an isometric view of these, with just the lower levels of stopes displayed for clarity, is shown below in Figure 6.1. The survey upon which the wireframes are based is understood to be up to date as of January 2011.

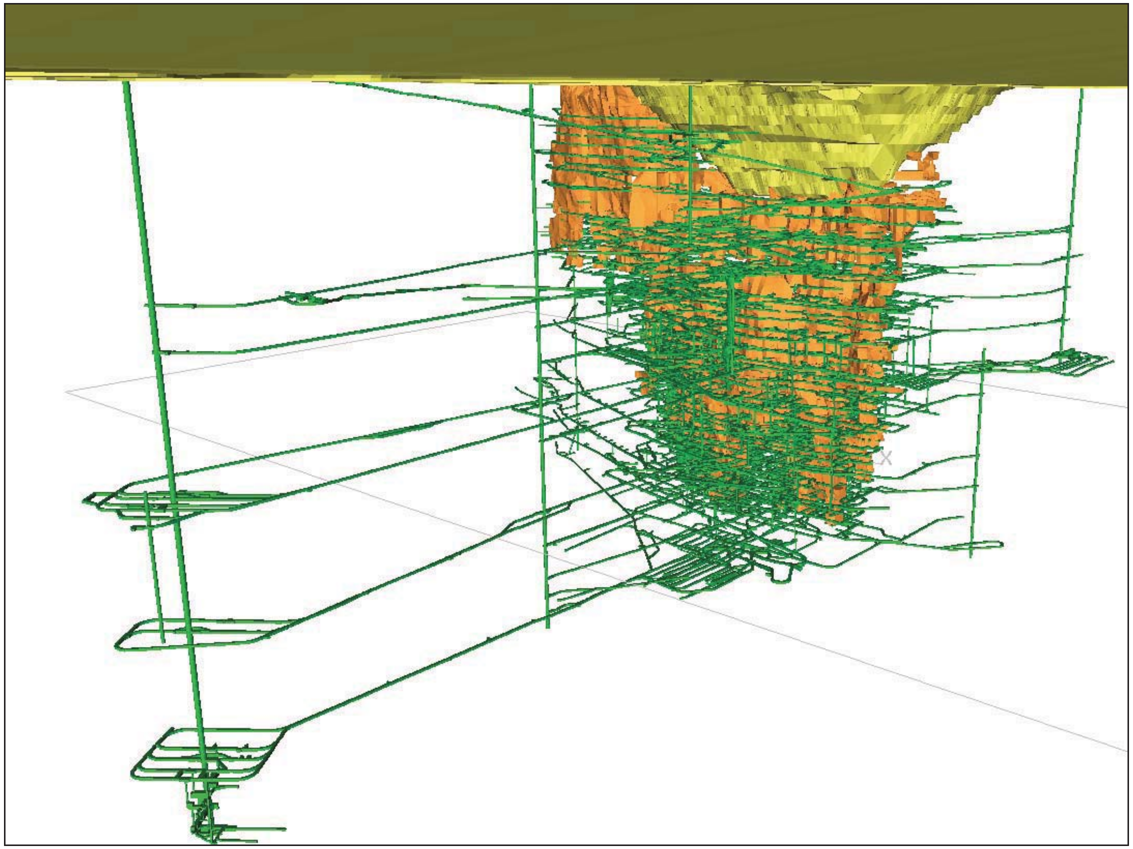


Figure 6.1: Isometric View of Underground Survey Wireframes

6.5.2 Database Compilation

6.5.2.1 Database

The sample database was supplied as a single desurveyed file. Verification was carried out to ensure there were no duplicate or overlapping samples. The database consists of a mixture of channel samples and underground drilling but is not coded as such. A summary of the database is shown in Table 6.5.

Table 6.5: Sample Database Summary

Total Holes/Channels	Total Length (m)	Number of Assays
10,486	359,848	287,740

6.5.2.2 Drillhole Sections

Drilling at Tishinskiy is comprehensive and has been undertaken from both surface and underground. Drilling is generally orientated in a north south direction, at varying dip angles to cut across the strike of the deposit. The profile sections are spaced approximately 12.5m apart and are shown below in Figure 6.2 in plan view.

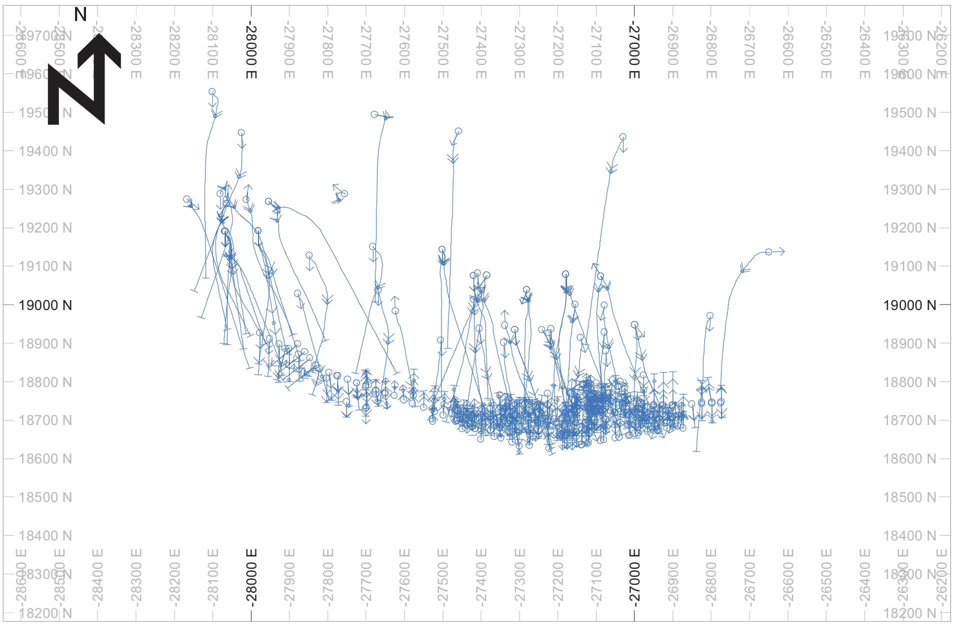


Figure 6.2: 100m Level Plan of Drillholes

6.5.3 Domaining

6.5.3.1 Equivalent Grades

Tishinskiy is a polymetallic deposit and equivalent grades are calculated for Zinc. For the Tishinskiy deposit the equivalent grade conversion factors are listed in Table 6.6. The minimum grades required for conversion are listed in Table 6.7.

Table 6.6: Conversion Factors for Equivalent Grades	
Zn	1
Pb	0.84
Cu	2.64
Au	0.96
Ag	0.01

Table 6.7: Minimum Grades for Conversion to Equivalent Grades	
Zn	0.4
Pb	0.1
Cu	0.06

6.5.3.2 Statistical Analysis

Domaining was carried out based on equivalent grades. A statistical analysis of grades has been carried out to assess grade levels used to determine areas of mineralisation.

A Histogram and Log Probability Plot for Zn indicate a break in population at approximately 25% Zn. Zn samples were coded as belonging to the low grade (<25% Zn) or high grade (>25% Zn) population and these groups were assessed separately for estimation purposes.

6.5.3.3 Zone Modelling

The subsequent interpretation and modelling work was based on cut off grades of 2.2 % Zn equivalent. A separate assessment was made based on a low and higher grade Zn domain with a cut off of 25% Zn. The steps used in the mineralised zone interpretation and modelling are summarised below:

1) An additional 'mineralisation limit' wireframe model was also defined. This defined an approximate limit to mineralisation;

2) Existing cross-sections for the deposit, alongside existing stope outlines, were used, to define directional strings down the centre of the principal mineralised structures. Two sets of strings were defined; along strike directional strings and across strike dip strings;

3) The sample data was converted into 5m composites;

4) The composites were then flagged if they were either above the 2.2% Zn equivalent cut-off level. A separate tag was made indicating if a sample was above or below the 25% Zn cut off. A minimum of 1m above cut off were required for a composite to be included for modelling. A maximum of 3m internal waste was allowed for inclusion within multiple composites; and

5) Based on the flagged composites, and the directional strings for dip and strike, an indicator estimation was carried out using a 50m x 50m x 10m search ellipse (along strike x down dip x across strike) to estimate those areas inside the approximate mineralisation limit that are above cut off grade for Zn equivalent. A separate estimation was carried out to estimate if blocks were above or below the 25% Zn cut off. A typical cross-section of the resultant interpretation is shown in Figure 6.3. The estimation, using nearest neighbour method, was carried out in to a block model with block sizes of 10m x 1m x 10m (across strike x along strike x down dip).

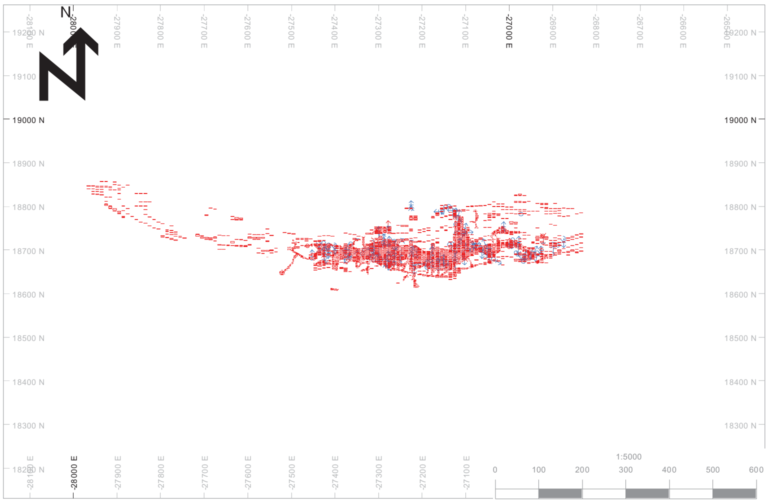


Figure 6.3: Example Plan section 220m of Mineralised Zone Interpretation with Composites above Cut-Off Grade

6.5.4 Sample Data Processing

Samples were coded based on their locations relative to the mineralised domains defined as described above. All samples falling within the defined mineralised zones based on the Zn equivalent cut off grade were selected for further processing.

6.5.4.1 Statistical Analysis

Statistical analysis for Pb and Cu has been carried out on the samples to identify any potential bias that may be present within the data. This indicates an approximately log normal population of samples for these elements.

Both Zn domains show a roughly log-normal distribution of grades.

The basic statistics for each element are listed in Table 6.8.

Table 6.8: Basic Statistics by Element

FIELD	Minimum	Maximum	Mean	Variance	Standard Deviation	Log Estimate of Mean	Coefficient of Variation
Zn Low	0	53.11	5.15	31.42	5.60	6.80	1.09
Zn High	0	59.79	31.10	94.13	9.70	35.75	0.31
Pb	0	28.3	1.44	8.36	2.89	1.62	2.01
Cu	0	55	0.63	0.90	0.95	0.78	1.51
Au	0	795.2	0.70	17.01	4.12	0.72	5.88
Ag	0	3973	13.20	1308.94	36.18	16.20	2.74

6.5.4.2 Removal of Outlier Grades

To identify the need for top cuts, the log probability plots and quantile distribution for each element were studied to identify the presence of any outlier values. Following statistical analysis and the splitting of the two Zn populations it was decided that top cutting was not necessary.

6.5.4.3 Compositing

The overwhelming majority of samples are 1m in length and so a composite length of 5m was chosen to correspond with block model sizing. This allowed further processing of samples giving a consistent level of support for geostatistical analysis.

6.5.4.4 Data Processing Summary

The statistical analysis of the Tishinskiy sample database is summarised below:

- No significant bias is shown in the Cu, Pb, Au and Ag assays;
- A slight difference in Zn population distribution is shown above a cut off grade of 25% Zn. Zn will be modelled separately as a lower grade and higher grade domain based on this cut off value; and
- A 5m composite interval has been applied to standardise sample length for geostatistical interpretation.

6.5.5 Variography

6.5.5.1 Introduction

Variographic analysis was performed using Datamine Studio v3 software. Absolute, as well as relative variograms were generated, with the spherical scheme model being used for modelling purposes. Variography was carried out on the 5m composited data.

6.5.5.2 Variogram Parameters

Directional semi-variograms for Pb, Cu, Au and Ag were generated for the along strike, across strike and down-dip directions using the composited and top-cut data. Similar variograms were generated for Zn, but separated in to the high grade and low grade domains described above.

The experimental variograms and the fitted variogram models for Zn shown below in Figure 6.4.

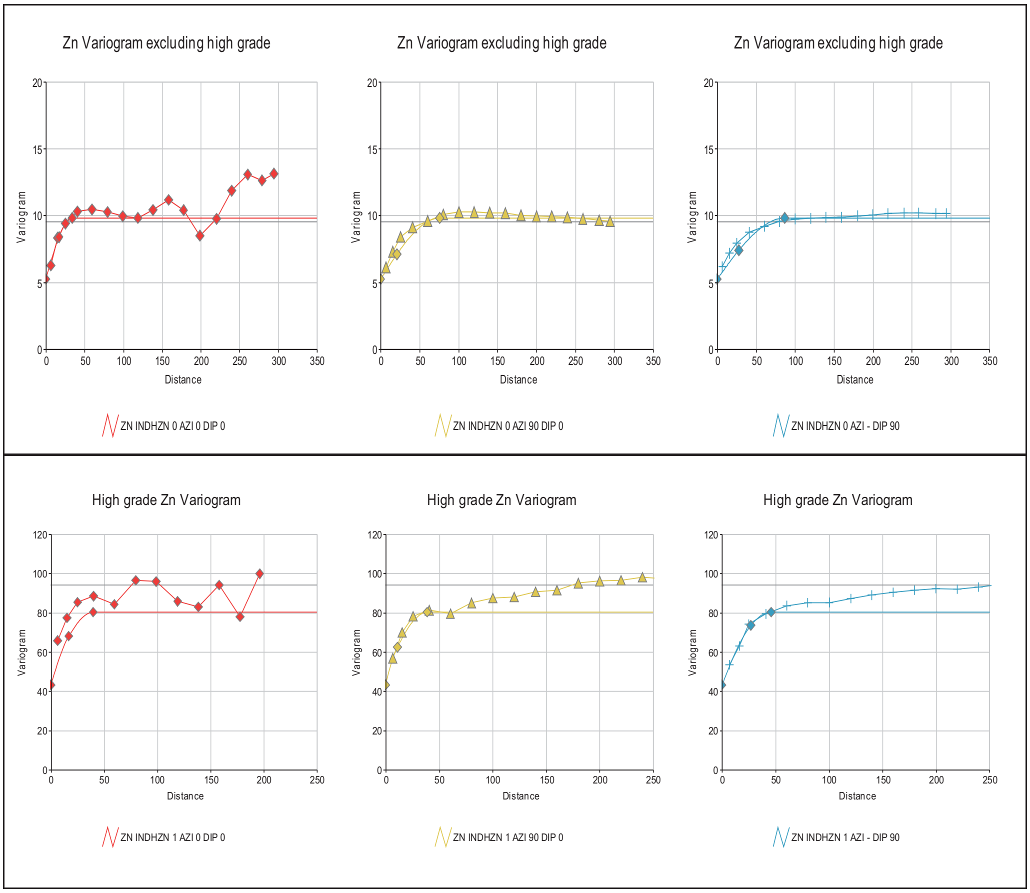


Figure 6.4: Variograms for High grade and Low grade Zn Populations

6.5.5.3 Variogram Interpretation

The principal direction of continuity for each group was selected from the generated experimental semi-variograms and modelled with three-structure spherical models. The three orthogonal orientations represented the predominant along-strike, down-dip and cross-strike directions. Overall the semi-variograms generated are considered to be well structured and interpretable.

6.5.6 Block Modelling

A summary of the block model prototype used is shown in Table 6.9. A cell size of 10m x 5m x 12.5m (across strike x along strike x down dip) was used for the final block model cell size.

Table 6.9: Summary of Block Model Parameters		
Property	Direction	Metres (m)
Model Origin	X	-28100
	Y	18600
	Z	-670
Parent Cell Size	X	10
	Y	5
	Z	10
No. of Cells	X	135
	Y	90
	Z	141

6.5.7 Density

Density values were assigned by ore type based on metal content using the equation listed in Table 6.10 below.

Table 6.10: Density Calculation
2.82 + (0.0254 x (Cu + Pb + Zn))

6.5.8 Grade Estimation

6.5.8.1 Introduction

Grade estimation was carried out using Ordinary Kriging (OK) as the principal interpolation method. Inverse Power of Distance Squared (IDW^2) and Nearest Neighbour (NN) were also used for comparative purposes for each element. The OK method used estimation parameters defined by the variography. The estimation was performed on mineralised material domains defined during the domaining process described above and only drillhole composites contained within a domain were used in the grade estimation of that domain.

6.5.8.2 Kriging Plan

For the mineralised zones the OK estimation was run in a three pass kriging plan, the second and third passes using progressively larger search radii to enable the estimation of blocks unestimated on the previous pass. The search parameters were derived from the variographic analysis, with the first search distances corresponding to the distance at 2/3rds of the variogram sill value and the second search distance approximating the variogram range.

Sample weighting during grade estimation was determined by variogram model parameters for the OK method. Block discretisation was set to 3x3x3 to estimate block grades. Sub cells were estimated individually.

The directional control settings defining the local variation in the strike and dip of the mineralised zones that were defined during the domaining process were used during estimation. The dip and dip directions were used as vectors to interpolate dip directions and dip values into the block model. These orientations were subsequently used during grade estimation to orient the search ellipses independently for each block. This dynamic anisotropy procedure gives a more realistic reflection of the local variations in the strike and dip of the deposit. A summary of the kriging plan is shown in Table 6.11.

Table 6.11: Summary of Kriging Plan

Search Ellipse	Parameter	Pb	Cu	Au	Ag	Zn Low Grade Domain	Zn High Grade Domain
1st	Search Radii 1 (m) – across strike	67	67	53	60	50	26
	Search Radii 2 (m) – along strike	37	53	40	67	22	26
	Search Radii 3 (m) – down dip	67	97	63	60	58	30
	Minimum Composites	6	6	6	6	6	6
	Maximum Composites	15	15	15	15	15	15
	Minimum Octants to be Filled	3	3	3	3	3	3
	Minimum Composites Per Octant	1	1	1	1	1	1
	Maximum Composites Per Octant	3	3	3	3	3	3
2nd	Search Radii 1 (m) – across strike	100	100	80	90	75	39
	Search Radii 2 (m) – along strike	58	80	60	100	33	39
	Search Radii 3 (m) – down dip	100	145	32	90	87	45
	Minimum Composites	4	4	4	4	4	4
	Maximum Composites	15	15	15	15	15	15
3rd	Search Radii 1 (m) – across strike	201	201	159	180	150	78
	Search Radii 2 (m) – along strike	111	159	120	201	66	78
	Search Radii 3 (m) – down dip	201	291	189	180	174	90
	Minimum Composites	1	1	1	1	1	1
	Maximum Composites	15	15	15	15	15	15

6.5.9 Validation

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

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

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

6.5.9.1 Visual Assessment of Grade Validation

A visual comparison of composite sample grade and block grade was conducted in plan view and cross section. Visually the model was generally considered to spatially reflect the composite grades.

6.5.9.2 Global Statistical Grade Validation

Statistical analysis of the block model was carried out for comparison against the samples and composited drillhole data. This analysis provides a check on the reproduction of the mean grade of the composite data against the model over the global domain. Typically the mean grade of the block model should not be significantly greater than that of the samples from which it has been derived. The mean block model grade and its corresponding mean sample and composite grades are shown in Table 6.12.

Table 6.12: Global Statistical Grade Validation				
	Mean Composite Grade	Model Block Grades		
		OK	ID	NN
ZN Low	5.5	5.1	5.2	5.0
ZN High	22.4	23.0	23.4	23.8
PB	1.7	1.7	1.7	1.6
CU	0.7	0.6	0.6	0.6
AU	0.8	0.6	0.7	0.6
AG	15.5	13.8	13.8	13.4
NB –				
1) No cut off grade applied				
2) OK (ordinary kriging), NN (nearest neighbour), ID (inverse distance weighting squared)				

6.5.9.3 Swath Analysis

Swath plots have been generated from the model by averaging both the composites and blocks along northings, eastings and vertically. The dimensions of each panel are controlled by the dimensions of the block size. Swath plots were generated for all block model estimation methods. Each estimated grade should exhibit a close relationship to the composite data upon which the estimation is based.

6.5.9.4 Validation Summary

Globally no indications of significant over or under estimation are apparent in the model nor were any obvious interpolation issues identified. From the perspective of conformance of the average model grade to the input data, WAI considers the model to be a satisfactory representation of the 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 OK 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 OK 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.

6.5.10 Resource Classification

The resource classification for the Tishinskiy deposit is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves JORC Code (2004). Criteria for defining resource categories were also derived from the geostatistical studies.

Key drillhole spacing for the allocation of resources can be summarised as:

- ***Measured*** resources - at least 50m x 58m (along-strike x down-dip);
- ***Indicated*** resources - at least 100m x 116m (along strike and down dip); and
- ***Inferred*** resources - all other blocks within defined mineralised zones.

6.5.11 Depletion

The block model was depleted against stope and other developments. This information is believed to be accurate to 01 January 2011.

6.5.12 Resource Evaluation

The resource classification for the Tishinskiy deposit is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves JORC Code (2004). The grades in the final resource model were derived from the Ordinary Kriging estimation method for all elements. A complete block model was built containing both the remaining in-situ material as well as previously mined material. An evaluation of the remaining unmined in-situ material is shown below in Table 6.13 reported, for consistency with the domaining method, to cut off grades of 2.2% Zn equivalent.

Table 6.13: Tishinskiy Resource Estimate - At COG of 2.2% ZnEQV
(WAI, 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnage (kt)	Density	Zn		Pb		Cu		Au		Ag		ZNEQV
			%	Kt	%	kt	%	kt	g/t	oz	g/t	oz	%
Measured	21,225	3	4.7	1,000	1.0	212	0.6	125	0.60	412,000	9.02	6,154,000	7.4
Indicated	7,010	3	4.3	305	0.9	66	0.4	31	0.46	104,000	9.75	2,072,000	6.3
Inferred	5,185	3	4.5	231	1.4	70	0.6	29	0.33	55,000	11.94	2,380,000	9.4

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

WAI has also prepared grade-tonnage curves for combined measured and indicated resources (Figure 6.5) which demonstrate the relative values of the model and resource estimate. The grade-tonnage curve is generated by averaging nodes into block grades and accumulating the respective tonnages above a series of cut-off grades. The average grades are then plotted against the tonnages at each cut-off.

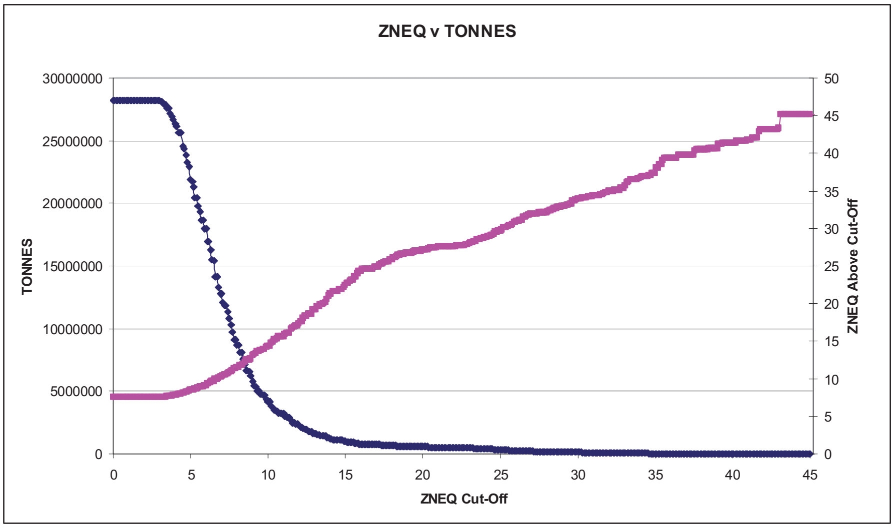


Figure 6.5: Grade Tonnage Curve for Measured and Indicated Resources

6.6 Mining

6.6.1 Introduction

Operations at the Tishinskiy mine started in 1975 using both surface and underground mining methods. To date, approximately 49Mt of ore has been extracted (including open pit ore). The open pit ceased operations in the mid-1990s due to the exhaustion of the reserves and has remained inactive since that time. A graph showing the annual underground production rate between 1975 and 2006 is presented in Figure 6.6 below.

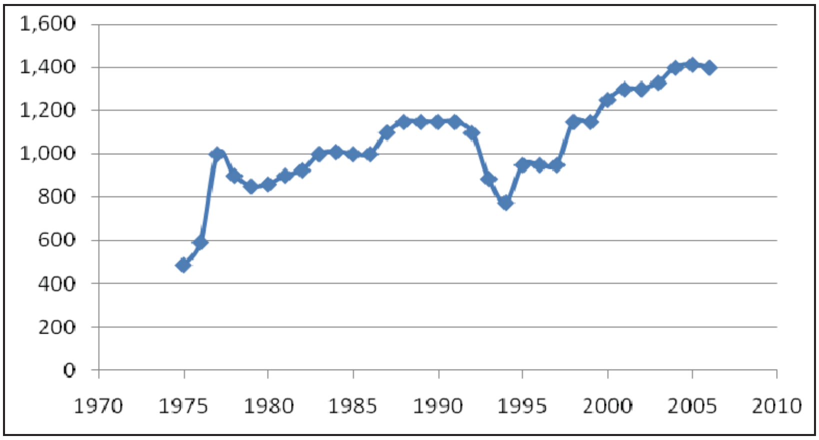


Figure 6.6: Annual Underground Production (ktpa)

Historically, the main mining method employed in the underground mine has been underhand open stoping with hydraulic cemented backfill. Trackless mining equipment is employed for the development and stoping activities but a track haulage system is used to transport ore and waste from drawpoints and ore passes to the main hoisting shaft.

The current ore production rate is between 1,300-1,400ktpa.

6.6.2 Mine Design and Current Mining Activities

Considering the time in operation, Tishinskiy mine has a very developed infrastructure. Currently the Main orebody is accessed via 5 shafts: Ulbinskaya Shaft, Slepaya-Ulbinskaya Shaft, RESh Shaft, Ventilation Shaft and Tishinskaya Shaft. Additional access is provided via decline. A 3D layout of the mine is shown on Figure 6.7 below.

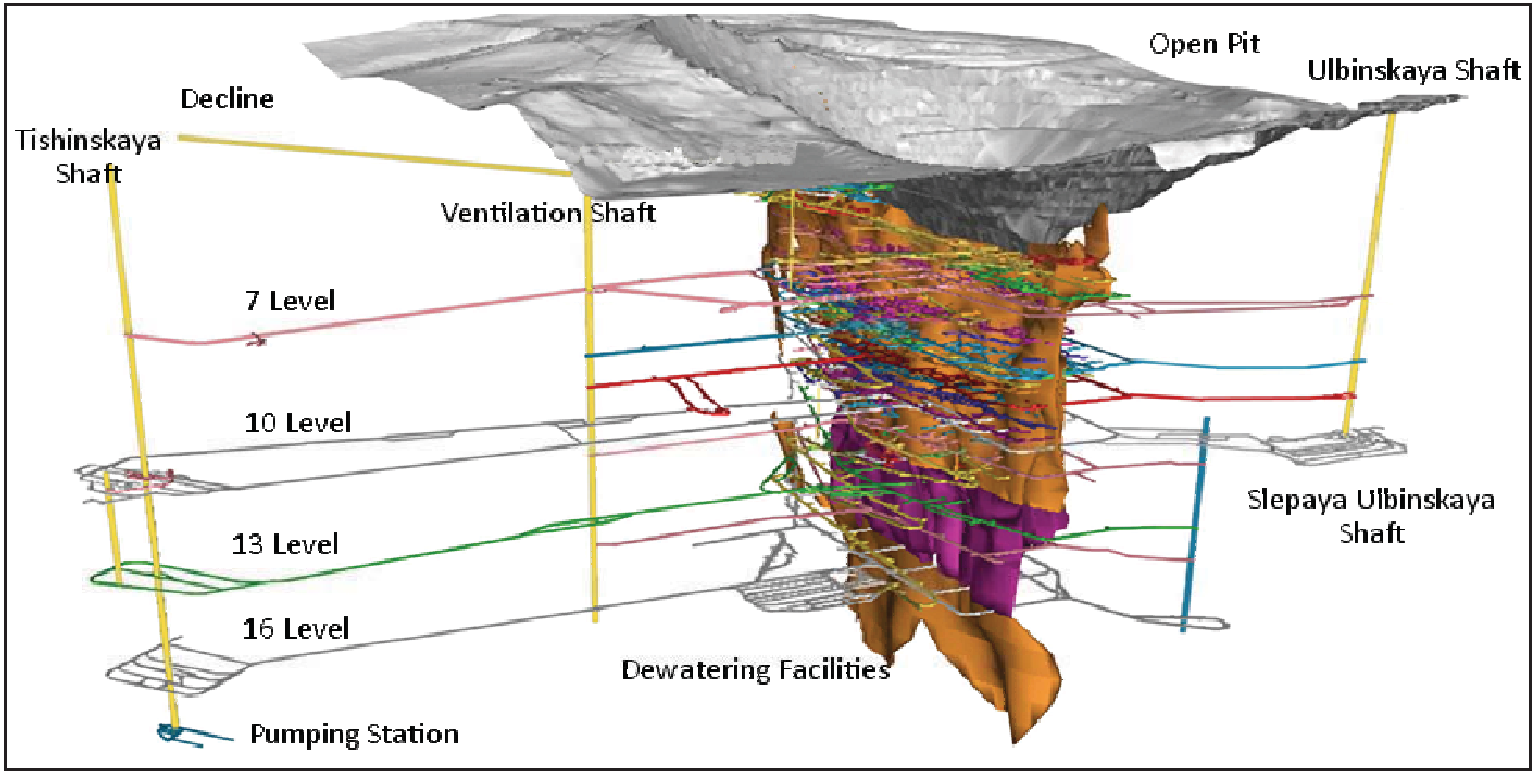


Figure 6.7: 3D Layout of Tishinskiy Mine

The major ore haulage areas are 7, 10, 13 and 16 levels. Current production is focused between 12-16 levels, with the majority of ore coming from 15 level. The distribution of the planned production between the levels in 2011 is given in Table 6.14 below.

Table 6.14: Distribution of Production between Levels

Level	Ore to be Mined (kt)	% of Total
12	61.6	5
13	202.9	15
14	357.7	27
15	553.1	42
16	154.7	12

The deposit was developed from top to the bottom and from the centre to flanks. The ore is drilled, blasted and mucked by trackless equipment, and hauled by electric locomotives at one of the major hauling levels. The Main orebody is steeply dipping, with an average thickness ranging from 6.5-17m and is relatively consistent, which altogether provides highly favourable conditions for bulk mining. Unusually, all mining takes place beneath backfill but very few stability problems have been reported.

Compressed air is supplied from a centralised compressor station consisting of 6 turbo compressors K-250-61-2 with a capacity of 250m^3/min each. The air is pumped via the Ventilation shaft by means of a 426mm diameter pipeline.

6.6.3 Mining Schedule

The mine production rate target for 2011 is 1,330kt as shown in Table 6.15, which is equally distributed between the 12 months. The mine operates 6 days week, with 3 shifts per day (6 hours per shift for the underground works. The mine planning is based on mining blocks of 100m in length and 60m in height, which cover the majority of ore bearing areas. The blocks are split into 20m sublevels, forming a basement area for stope preparation development and ore extraction works. The minimal unit, considered in mining scheduling, is the stope. The stopes are mined from top to the bottom, with two or three levels active at a time. This provides additional flexibility by performing backfill, extraction and stope preparation works at various areas simultaneously.

Table 6.15: Summary of Mining Schedule for 2011

	Unit	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Total Ore	kt	**109.3**	**105.5**	**116.8**	**113.,0**	**109.3**	**105.5**	**109.3**	**109.3**	**113.03**	**116.8**	**113.0**	**109.3**	**1,330.0**
Zn	%	4.81	4.81	4.81	4.81	4.81	4.81	4.81	4.81	4.81	4.81	4.81	4.81	4.81
	t	5,256	5,075	5,617	5,437	5,256	5,075	5,256	5,256	5,437	5,618	5,437	5,253	63,973
Pb	%	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.54	0.55	0.54	0.54	0.55
	t	596	575	637	617	596	576	596	596	616	637	616	595	7,253
Cu	%	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38
	t	410	396	438	424	410	396	410	410	424	438	424	410	4,990
Au	g/ t	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78
	kg	85.2	82.3	91.1	88.2	85.2	82.3	85.2	85.2	88.2	91.1	88.2	85.2	1,037.4
Ag	g/ t	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	8.99	9.00
	kg	983	949	1,051	1,017	983	949	983	983	1,017	1,051	1,017	982	11,965
Development	m	890	865	970	940	895	875	895	900	935	965	935	895	10,960
	m^3	**9,535**	**9,230**	**10,180**	**9,885**	**9,545**	**9,255**	**9,555**	**9,565**	**9,880**	**10,215**	**9,865**	**9,555**	**116,265**
Drilling Works	**m**	**15,430**	**16,100**	**15,440**	**17,450**	**16,100**	**17,440**	**17,385**	**16,815**	**17,445**	**17,435**	**16,780**	**16,770**	**200,590**
Backfilling	m^3	**31,570**	**32,840**	**31,620**	**35,650**	**32,900**	**35,645**	**34,290**	**35,660**	**35,660**	**35,635**	**34,265**	**34,265**	**410,000**

The expected dilution and mining losses for 2011 are 14-15% and 5-6% respectively. The majority of the ore in 2011 will be mined via underhand blasthole open stoping with hydraulic cemented backfill. A description of the mining method is given below.

6.6.4 Mining Method

6.6.4.1 Introduction

Underhand sub-level blasthole open stoping with cemented backfill is the main mechanised mining method. This method utilises jumbo development, electrico-hydraulic production drilling and diesel load haul dump (LHD) loaders. The production mining fleet typically consists of:

- Development drills - Sandvik Minimatic 265 and Atlas Copco Boomer 282 multi boom jumbos, heading sizes of approximately $12m^2$;
- Production drills - Sandvik Solo 5/7F and Sandvik Solomatic 720 drilling holes up to 102mm diameter; and
- CAT R1300G LHD and Sandvik Toro 6M, both with a bucket size of approximately $3m^3$ and remote control capability.

6.6.4.2 Underhand Blasthole Sub-level Open Stoping with Backfill

The primary transverse stope dimensions are 40m in length (N-S orientation) by 10m width (E-W orientation) and 20m in height, extracted as a continuous panel sequence along strike. The secondary stopes are 25m in length (E-W), 10m in width (N-S), and 20m in height, extracted from hangingwall to footwall in a continuous panel sequence across strike.

Normally, 4 stopes are in production, 2 are being backfilled and 5 being drilled at any one time. As well as these, approximately 6 will be in development or preparation stage. The stopes cycles are usually 4 months, including development, but can vary from 1.5 months to 8 months depending on the actual size of the block.

6.6.4.3 Backfill

The mine uses cemented hydraulic backfill, which consists of pre-processed tailings and smelter slag. The composition of the backfill is detailed below:

- Light Tailings Fraction – $647kg/m^3$;
- Smelter Slag – $300kg/m^3$;
- Glanulated Slag – $270kg/m^3$;
- Mill Tailings – $116kg/m^3$; and
- Cement – $64kg/m^3$.

The inclusion of smelter slag and granulated slag into backfill resulted in significant reduction in the cement consumption (from 220kg/m to $64kg/m^3$). There are three types of backfill produced M25, M35 and M40. Each type has a specific purpose:

- M25 is used for stope backfill (general use);
- M35 for 0.7-0.8m roof pillar forming underneath transport and drilling drives; and
- M40 – for 4.0-4.5m floor pillar forming if combined cemented and waste backfill is used.

The strength characteristics of the three types of backfill are described in Table 6.16 below.

Table 6.16: Compressive Strength Properties of Various Types of Backfill (MPa)						
Type of backfill/Age (Days)	**14**	**28**	**60**	**90**	**180**	**360**
M25	0.9/0.9	1.7/1.4	2.0/2.1	2.5/2.6	2.7/2.8	3.5/3.5
M35	1.4/1.4	2.5/2.2	3.0/2.9	3.5/3.6	4.0/4.2	4.7/5.0
M40	1.8/1.9	2.6/3.0	3.4/3.5	4.2/4.0	4.5/4.5	5.6/5.5

The mine has its own backfill preparation plant, where backfill is mixed and pumped to a pipeline connected to the underground infrastructure. From this pipeline the backfill is distributed to stopes via backfill boreholes (normally one for the backfill, and an additional borehole for control) and pipeline extensions.

6.6.5 Rock Properties and Geotechnical Conditions

The rock properties of the deposit are generally medium to low stability. The areas between the ore and the host rock within 5-7m from the contact area have reduced stability. It is has also been influenced by tectonic faults.

There are four main regions which have delineated by their rock properties:

- Main orebody within lines (13-32) across strike;
- Northern orebodies within lines (10-33) across strike and thin orebodies of Main orebody to the east of line 33 across strike;
- Main orebody to the west of line 13 across strike; and
- Southern contact of Main orebody represented by unstable mineralised and barren rocks.

The average mine rock stability is as follows:

- 15% is stable;
- 50% is medium stable;
- 30% is unstable, and
- 5% is strongly unstable.

The average rock hardness coefficient is 8-10 (Russian Protodyakonov's scale), with values varying from 4 to 16. The bulking factor is 1.4-1.5. The ore is classified as hazardous in terms of risk of silicosis.

6.6.6 Ore and Waste Transportation

There are three main haulage horizons at 10, 13 and 16 level. At the intermediate levels of 11, 12, 14 and 15 level, the ore is transported by trackless equipment to the nearest orepass, where it drops to one of the haulage levels as shown in Figure 6.8 below.

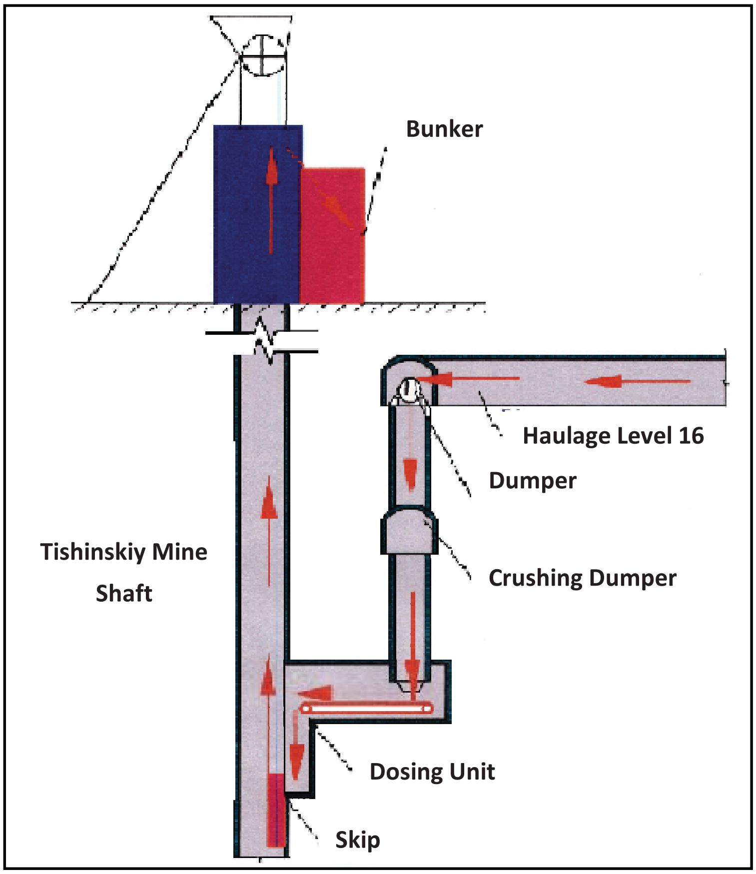


Figure 6.8: Ore Haulage Schematic

Ore is mucked by a TORO-301D LHDs (3 units), CAT R1300 (4 units), TORO-151 (1 unit) and TORO-006 (1 unit). A number of remote controlled LHD are used to extract ore from the stopes located under active levels, which minimises risk of accidents.

During stope preparation, rock is transported by TORO-301D (5 units), TORO-301HL – (1 unit), TORO-300 (1 unit) and LK-1 (1 unit).

K14M type locomotives are used for hauling the 4.5m^3 wagons of ore, with normally 10 wagons per train, to the shaft. There are 4-5 locomotives operating at each level with the exception of 10 level, where the number of locomotives is reduced to one. The total number of locomotives is 16.

The wagons of ore are emptied into a bunker via rotary wagon tippers, and then crushed by a jaw crusher. The crushed ore is loaded into skips by means of belt conveyor. The skip loading point has both weighing and volume control facilities. The amount of ore in the skip is limited by tonnage (13t) ore, total volume (9.5m^3).

6.6.6.1 Hoisting Facilities

The main hoisting shaft is Tishinskaya Shaft. The shaft is equipped with 3 winders:

- A single cage winder;

- A single-skip hoisting winder, and
- A double-skip winder.

The cage hosting is performed by a BTsK8/4.5x2.25 machine, working with a double decked cage measuring 4.5x1.5m and a 14.8t counter weight. This cage is used for hoisting men, materials and equipment. The double-skip hoisting system, TsR6-6.4/1.8, provides skip speeds up to 8.5m/s, which ensures an hourly performance of 18 skips per hour required for ore production.

The single-skip hoisting system, BTsK8/4.5x2.25, operates with a 5.5m^3 skip and 12.8t counterweight. It is used for hoisting waste at a maximum speed of 8m/s. It is capable of 8 skips per hour production rate.

In addition to the Tishinskaya Shaft, the other operating shafts at Tishinskiy mine are:

- The Ventilation Shaft which is equipped with a TsSh2.25x4 winder, a 4.5x1.5m cage and a 9.4t counter weight. This shaft is used for ventilation and man riding;
- The Ulbinskaya Shaft is a single-cage shaft currently not in operation;
- The West-Ventilation shaft has no winder installed and is used for ventilation only, and
- The Slepaya-Ulbinskaya Shaft is used for emergency egress.

In addition to the above hoisting facilities, the mine has a decline sunk down to the 16 level, which is used for access.

6.6.7 Dewatering System

The mine has a two-stage dewatering system. At the first stage, heavy fractions are removed and pumped to the surface for further processing. At the second stage, most of the water is pumped to the surface.

The first stage of dewatering is performed by Feluwa pumps made in Germany, whose specifications are given in Table 6.17 below. These pumps are installed at 16 level, close to Tishinskaya Shaft. The main purpose of the first stage pumps is to increase the life of the Sulzer pumps by removing the heavy fractions and to reduce environmental impact.

The second stage dewatering takes the rest of the underground water to surface. It is performed by x4 centrifugal Sulzer pumps, whose specifications are given in Table 6.18 below.

Table 6.17: First Stage Pumps

Type	Feluwa ZGL 100/200 – K 130 – 2 SM 350 HD
Head	1100m
Capacity	8-10m^3/hour
No of Units installed	4

Table 6.18: Second Stage Pumps

Type	Sulzer HPH 58-25-27- 8
Head, meters of water column	960m
Capacity	650m^3/hour
No of Units Installed	4

6.6.8 Ventilation

The mine has a complex ventilation system, where the majority of the fresh air is supplied from the extremeties of the mine, and exhaust air is discharged at the centre. The main ventilation facilities are described in Table 6.19 below.

Table 6.19: Main Ventilation Facilities of the Mine	
Main Ventilation Fan at Ventilation Shaft	
Main Fan	VOD-40 reversible
Engine Capacity	1,600kW
Voltage	6kV
Nominal Capacity	$275m^3/s$
Depression	143mm
Ventilation Fan at the Decline	
Main Fan	VM12(2 units)
Engine Capacity	110kW
Voltage	0,4kV
Nominal Capacity	$46\ m^3/s$
Depression	43mm
Ventilation Fan at Ulbinskaya Shaft	
Main Fan	VOD 30
Engine Capacity	800kW
Voltage	6kV
Nominal Capacity	$131m^3/s$
Depression	54mm
Ventilation Fan at Zapadnaya Shaft	
Main Fan	VTsD-31.5M reversible
Engine Capacity	1600kW
Voltage	6kV
Nominal Capacity	$228m^3/s$
Depression	121mm

Remote sites, such as development faces, extraction stopes etc., are supplied with fresh air by means of portable local ventilation fans, typically these fans are Russian-made VME-5, VME-6 and VME-8.

6.6.9 Mine Personnel

Approximately 765 personnel are employed at Tishinskiy mine. The mine has a well developed departmental structure of management. Each department has a specific designated area of responsibility and scope of work. A list of departments together with the number of staff employed in each is shown in Table 6.20 below.

Table 6.20: Mine Personnel by Department	
Administration	19
Ventilation Service	2
Design and Engineering Department	4
Surveyor Department	12
Geology Service	14
UG Mining and Loading Brigade No1	92
UG Development Brigade No2	86
UG Reinforcement Brigade No3	91
UG Drilling and Blasting Works Brigade No4	106
UG Backfilling Brigade	53
UG Transportation Brigade	91
UG Stationery Equipment Maintenance Brigade	155
incl. Hoisting	118
incl. Dewatering (General)	23
incl. Dewatering (Main)	14
Auxiliary and Open Pit Mining Works Brigade	40
Total Administrative	*138*
Total Non-administrative	*627*
Total Mine	**765**

As at Ridder-Sokolniy mine, WAI notes that the mine has high standards of health and safety in place, particularly the use of PPE and minimising risks in underground works, as well as paying a lot of attention to social aspects. Sports and recreational facilities, medical services are available to employees and their families.

WAI Comment: *WAI considers that the mine has no significant employment issues.*

6.6.10 Future Mine Development

With depletion of the ore reserves from the central part of the deposit, the mine will be developed towards its flanks, and towards to the bottom. Current exploration programmes are targeting to bring into production the horizons between 18 and 22 levels. An additional resource estimation study is currently being conducted for the eastern flank of the deposit, with production from this area due to commence in 2012.

For the medium term plan, the mine is going to operate at its current rate of 1,300-1,400ktpa.

6.6.11 WAI Ore Reserve Estimation

WAI has performed an evaluation of the Tishinskiy mine Ore Reserves in accordance with the requirements of the JORC Code (2004). The evaluation has been performed using a geological block model produced by WAI dated 01.01.2011 and considers modifying factors, such as dilution and losses, with respect to the current mining operations and methods.

The stope boundaries were generated to correspond with the standardised dimensions of the existing stopes and development in the mine and also to be above the minimum cut-off grade for exploitation. A summary of typical stope parameters is given in Table 6.21 below.

Table 6.21: Typical Stope Parameters

Length (m)	20
Width (m)	10
Height (m)	20
Minimal Zn_{Eq} Average Grade (%) For a MU	3.0
Dip Angle (Deg)	90

Datamine® Mineable Stope Optimiser® (MSO) software was used to assess whether mineable stopes can be generated for certain parts of the deposit. MSO computes the optimal size, shape and location of stopes for an underground mine using an input block model containing grades or values. $Zn_{Equivalent}$ grades were used for Tishinskiy evaluation.

MSO searches for the optimal mineable shapes taking into account the orebody geometry. The stope shape is parameterised, the programme generates outlines on adjacent sections, and links these to create a wireframe shape for evaluation against the block model. The sectional outlines are defined by four points on the roof and floor. Constraints are applied on the dip and strike of the final stope shape.

The stopes produced by MSO are then evaluated against the geological block model in order to quantify average grades of the economic minerals and the metal content, together with ore and waste tonnages.

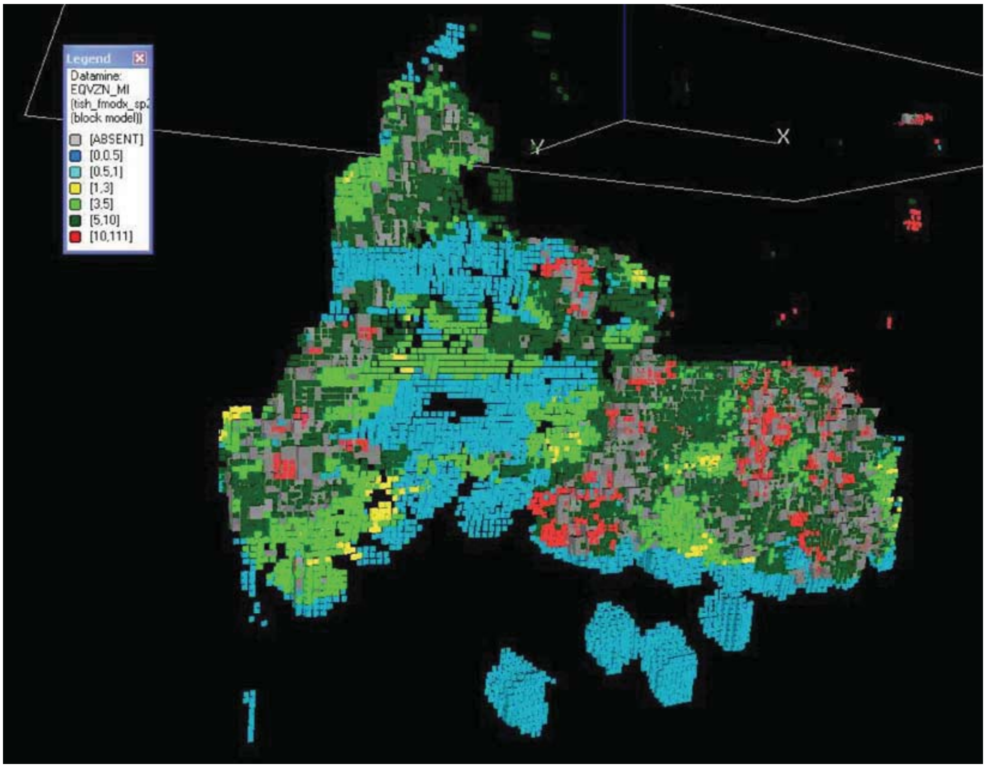


Figure 6.9: Stope Layout for Tishinskiy

(Blue – low grade or Inferred material; grey – MSO stopes; light blue, yellow, green, dark green and red – model blocks with 0.5-1, 1–3, 3–5, 5-10 and >10 % of ZnEq in Measured and Indicated categories)

A total of 2,725 stopes were produced for the ore reserves. The greatest part of the reserve is contained in the central part of the deposit, but there is a significant portion of reserves located in the western part of the deposit. A summary of ore reserves as calculated in accordance with teh JORC Code (2004) is given in Table 6.22 below.

Table 6.22: Tishinskiy Ore Reserves Estimate
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

	Ore	**Zinc**		**Copper**		**Lead**		**Gold**		**Silver**	
	kt	%	kt	%	kt	%	kt	g/t	kg	g/t	kg
Proven	18,886	4.22	797	0.52	98	0.91	172	0.54	10,198	8.12	153,353
Probable	4,928	4.13	204	0.4	20	0.88	43	0.47	2,316	9.36	46,124
Total Proven and Probable	***23,814***	***4.20***	***1,001***	***0.50***	***118***	***0.90***	***215***	***0.53***	***12,514***	***8.38***	***199,477***
Inferred material within stopes	*156*	*11.17*	*17*	*0.45*	*1*	*1.2*	*2*	*0.45*	*70*	*11.05*	*1,728*

The layout of the generated stopes is relatively dense, therefore, a simplified approach has been used in order to access development requirements. It was assumed that the ore will be hoisted via Tishinskaya shaft, in-line with the present operations, and further blocks below level 16 will require haulage up to level 16 providing access to the most remote stopes created by MSO. The length of such haulage levels was then added to the overall capital development requirements. An additional allowance was made for ramp construction below Tishinskaya shaft bottom, which will be used for both access and haulage purposes.

Stope preparation requirements were estimated on the basis of historical production figures. A summary of development requirements is given in Table 6.23 below.

Table 6.23: Summary of Development Requirement

Item	Total Length (m)	Total Volume (m^3)
Haulage Drives at Main Levels	2,813	45,000
Haulage Drives at Sublevels	8,438	84,375
Ramps	600	9,600
Stope Preparation	NA	2,500,000

It should be stressed that the estimated development requirements used in the ore reserve estimation are to ensure the economic viability of the reserve only and do not represent the actual development schedule proposed by Kazzinc.

6.6.12 Conclusions

Tishinskiy mine is a mature operation. Despite relatively unfavourable rock conditions, resulting in the requirement for additional ground support, mining is performed to a high standard. The utilisation of modern western underground mining equipment and qualified personnel provides confidence to WAI that production targets are being met. The remaining Ore Reserve together with available Mineral Resources estimated by both internal and independent sources, show long-term potential for this operation.

6.7 Environmental

6.7.1 Introduction

The Tishinskiy Mine forms part of the Ridder Mining Complex (RMC). The mine is approximately 18km west of the Ridder complex and is 120km north-east of Ust-Kamenogorsk, connected by a metalled road, and freight and passenger railway lines.

The site boundary was delineated by the Tishinskiy mining licence, see section 6.7.4 for details.

The first horizon is at 670m and the lowest level at -230m, the upper horizons have all been mined out (by open pit, the western flank – by underground method) when the mine was operated by the State. No processing of ore takes place at Tishinskiy Mine, with ore sent to RMC concentrator by rail.

There has been mining activity in the Ridder area since 1786, via both surface and underground workings and with concentrator facilities since the 1920s. Mining at Tishinskiy commenced in the 1960's when the mine was owned and operated by the State. Kazzinc`s involvement commenced in 1997 when the Company was established.

6.7.2 Environmental & Social Setting and Context

6.7.2.1 Landscape, Topography

The Tishinskiy Mine area is predominantly gently undulating topography with hills varying from 650m to 1,000m in height.

6.7.2.2 Climate

The climate is sharply continental with hot summers reaching temperatures in excess of 40°C and cold winters dropping to -47°C. Snow cover is present from November to April. Average rainfall is 710mm/yr. The prevailing wind is from the north-west. Average wind speed is 5.7m/s in the winter, and 3.5m/s in the summer.

6.7.2.3 Land Use and Land Cover

Soil horizons are expected to be thin and lacking in fertile topsoil – no information on soil characteristics has been provided. Flora is defined by several landscape zones, affected by site geography, altitude and terrain. There are numerous trees and grass species, in a mosaic habitat, with coniferous species predominantly found at higher altitudes and pine and mixed forest, including birch found in the lower foothills. There are flat areas of scrubby grassland, with apparent species abundance being somewhat limited by climatic conditions. 94 bird species are reported in the area, the majority being non-migratory, and 90 animal species, including bear, mink, deer, wild cats and numerous rodent species. Two of the bird species are reported to be rare raptors.

6.7.2.4 Water Resources

Two Rivers are present within the Tishinskiy mining license area, the River Ulba to the east, the River Poznopalovka a tributary of the Ulba which enters the license area in the north of the site. The Poznopalovka River was diverted from its natural course to the River Ulba and is now re-directed via a water diversion channel, 500m upstream from its natural outfall.

Aquifers in the area are of lower quaternary alluvial sediments (35m thick and 300-500m wide) represented by river gravel/boulder, sandy gravel in a wide band of 800-1200m.

An underground abstraction point is located at the northern end of the site this water is used for the boiler house, mine offices and as potable water for the nearby communities. The intake is managed, and water is transported by L-TVK Ltd. (a contractor). All other on site processes e.g. Batching plant use treated mine water.

6.7.2.5 Communities and Livelihoods

A village is located c4km from the site: this consists of mainly holiday homes and a few permanent dwellings. The village developed after the mine in c1960s and was purposely built in the area on land where exploration had proved there were no underlying ore reserves thus reducing the risk of subsidence.

6.7.2.6 Infrastructure & Communications

Occupational health monitoring is carried out at the plant, a set of monitoring results were presented to WAI and found to be compliant with national standards. No vibration of noise monitoring is undertaken in the plant. No water is discharged from this plant. Operatives working on the day of the visit wore limited PPE, however it was not considered fully compliant with best practice. All fire extinguishers checked were in date and pressures were compliant at 2psi.

> ***WAI Comment:*** *The batching plant is well managed and standards of housekeeping are very good, however, there was a lack of H&S signage and PPE (high-vis vest should be worn).*

6.7.3 Project Status, Activities, Effects, Releases & Controls

6.7.3.1 Project Description & Activities

Neutralisation plant

A neutralisation process plant to treat mine water was constructed in 1968 and has been operating for 42 years. The mine water treatment is undertaken in two stages, below and above ground. The above ground tanks are constructed of concrete. An overhead crane with grabber removes the sludge after each water treatment cycle. The sludge are removed and deposited in the subsidence area, (general site waste landfill). The tank cleaning process takes approximately 3 days.

The mine water is collected underground at the 16th horizon into 4 settling/sludge tanks to begin initial mine water treatment. The residence time in these settling tanks is several days after which the mine water is pumped to the surface. Underground slimes are pumped to the surface and, together with ore, are treated in the crushing and DMS sections. The water enters the above ground tanks at pH 8.0-5.5 with a flow inlet rate of 730m^3/hr, where 'lime milk' is added to increase the pH to 9.5-10.5. There are 8 above ground settling tanks each 5m deep, the water in each tank is monitored daily.

Following treatment of the mine water, approximately 25% to 30% is piped to the Dense Media Separation Plant (DMS) and batching plant, whilst the remaining water is discharged to the River Ulba at a flow rate of 16-17,000m^3 per day. Water Monitoring is undertaken at the discharge point and also 500m upstream and downstream of this point. NO_3 (Point 10) is elevated due to blasting in the underground mine.

> ***WAI Comment:*** *The discharge monitoring results did show exceedances for NO_3 at the discharge point, however the exceedances are still within the state limits for this type of discharge and NO_3 content in the river water decreases downstream.*

The neutralisation plant is very well organised and appears to be operating efficiently. Results from monitoring the inlet following stage 1 to discharge to River Ulba were presented for 2009-2010 and were acceptable. Given the age of the neutralisation plant, the condition of concrete tanks is inspected on a systematic basis by a special Buildings and Structures Monitoring Service; and repair plans are prepared and implemented yearly.

Batching Plant

Tishinskiy Mine has its own batching plant which produces an average of 380,000m^3 of cementitious material on average per year. This material is used as underground backfill for void filling. The building was in a reasonable state of repair and appears to be well maintained with a good standard of housekeeping. The plant only uses re-processed water for the production. The ball mills are fed with waste rock by conveyors. Except for cement, only waste materials are used in the production of back fill.

Quality Control testing of the mixture is undertaken every 1000m^3 of backfill.

Rail link

The Railway and rail yard is operated by 'Kazzinc Logistics'.

Dust/air quality monitoring is undertaken in the area three times over during the summer months (given the moisture content of the material) and monitoring also occurs 500m from the SPZ boundary on a monthly basis, as approved by State authorities.

It was noted that not all the railway operatives wore high-vis PPE. The rail depot is approximately 500m from the river Ulba, and it is understood that the River water is monitored upstream and downstream of this facility. Copies of the 2009 and 2010 monitoring results were presented to WAI.

> ***WAI Comment:*** *In general, the area is reasonably well maintained and appears to be operated in a satisfactory manner. Although this area is outside of the area of the mine control, the following comments are relevant. Best practice requires that all operatives working on and in the vicinity of a railway should wear high-vis work wear at all times for safety reasons. It was noted that several workers did not have appropriate PPE and the explanation given for this was unacceptable. It should be mandatory that PPE is worn in this area.*

Stockpiling of ore at the rail depot is adjacent to the rail line. The material is stored in a designated concrete lined area to prevent ore from getting onto the rail line.

All the material (ore and slime) loading areas are either concrete or clay lined to prevent soil and groundwater contamination. The OVOS draft has been approved with the State Expertise Assessment Department. As part of Kazzinc's corporate social responsibility programme, they neutralise contaminated water discharge from the State-owned dump to the Ulba river.

Dense Media Separation Plant (DMS)

In the DMS plant ore is crushed and pre-concentrated and then sent to the Ridder Concentrator. Tishinskiy ore pre-concentration tailings (slimes) are sent to the Concentrator also after de-watering in the filter-press which is installed in the DMS plant. The flowsheet was implemented in 2002 to exclude slimes collection. The slimes collected over the 1997-2001 period are currently removed from the slime pond and sent to the Concentrator for additional metal recovery. The slimes pond will be rehabilitated in 2012 when the slimes have been removed. Filter-press filtrate (Crushing and Concentration plant overflow) of pH 9-11 is sent to neutralise acid flow (pH 4-5) from the state-owned dump number 2. After mixing and settling, the flow of pH 7 (free from metal hydroxides) goes to the Ulba river.

Water sampling is undertaken before and after treatment and also at the discharge point on the River. The rate of discharge agreed with the state is 100m3/hr, however it is understood that the discharge is spasmodic, and only takes place during the filter-press operation period, and does not exceed this discharge rate.

> ***WAI Comment:*** *In general the dewatering of the slimes via the DMS plant is acceptable, and the plant appears to be efficiently managed.*

Vehicle Work Shops and Plant Maintenance

There are two areas where vehicle and plant maintenance is undertaken, both of which are housed in buildings dating back to the 1960's. Both buildings are maintained and kept in a reasonable state of repair; however they would benefit from some further improvements, nonetheless the standard of housekeeping within the building is good. Facilities to deal with grease and oil spills were in place, and also personal washing facilities were checked and found to be acceptable. All fire extinguishers were up to date with checks and pressures were at 2psi.

A waste management regime was in place in both workshops. There were no vehicle washing facilities evident at either location, therefore limited run off is expected. Only limited H&S signs were present in all areas.

> ***WAI Comment:*** *In general the workshop and maintenance plant were both well managed and housekeeping was of a good standard. It is noted that procedures are in place to deal with waste and spillages. The H&S department presented the 'Safe work procedures' for these installations on request, and these are considered adequate.*

Open Pit Area

The former open pit area is located in the centre of the mine site, to the east of the River Ulba. The pit is understood to be 300m deep and below this at 400m depth is the underground mine. The open pit remains the possession of the state, therefore Kazzinc have no liabilities associated with it. Due to the excavation and de watering of this pit a cone of depression of 5km^2 has occurred. De-watering is undertaken via two lines of boreholes, and this water is pumped to the neutralisation zone for treatment. Groundwater regime is monitored monthly by the Hydrogeological Dept via 14 drillholes; inflows to the mine are also monitored. Interim and final hydrogeological reports including the depression-cone induced. Process development forecasts are issues at the end of the quarter and year respectively.

> ***WAI Comment:*** *Hydrogeological observations ensure good control. Kazzinc's obligations under the Subsoil Use Contract are met in full.*

Ventilation Shafts

There are 3 ventilation shafts within the sit area comprising of 1 inlet and 2 outlets. These shafts have their own individual testing regime.

> ***WAI Comment:*** *All the results presented were acceptable with regard to frequency of testing.*

Miscellaneous Buildings

Several other buildings are occupied on site, which although owned by Kazzinc are rented and operated by a third party company, e.g.:

- Boiler House rented and operated by L-TVK LLP;
- Coal Store Rented and operated by L-TVK LLP; and
- Fuel store Kazzinc building operated by Kazzinc procurement and logistics.

These buildings do not form part of the remit of this report. However water supplied to the boiler house comes from the neutralisation plant. The overhead pipeline showed no evidence of erosion or leaks.

6.7.3.2 Land Ownership and Tenure

Kazzinc holds the right to mine polymetallic ores under the terms of the Contract for Subsurface Use (1997), which is valid for 25 years. The mining license can be extended in duration by mutual agreement between Kazzinc and the regulatory authority. The current mining lease (2003) covers an area of approximately 12km^2.

> ***WAI Comment:*** *WAI has viewed the mine licence and permit and considers that these have been obtained in line with State requirements. It is understood that the future expansion of mining activity would not necessitate relocation of communities within the current licence boundary. It is understood from discussions that the local village developed after the mine was opened in an area with no underlying ore reserves to minimise the risk of subsidence.*

6.7.3.3 Energy Consumption & Source

Power is supplied from two main lines from Ust-Kamenogorsk at 132kV and 113kV respectively. The Company owns a Hydroelectric Power Plant at Bukhtarma which supplies the national electricity grid. Power for the mine is supplied by the national grid.

6.7.3.4 Mine Wastes – Rock

A total of 6 waste rock/overburden dumps are located at Tishinskiy mine area, these are of various sizes, namely waste dumps 1,2,3,4 7 and 8. Waste dumps No 5 and 6 are no longer in existence. The waste dumps date back to c1964 and resulted from the former open pit (excavated to level 5) when operational.

The open pit was closed in the 1970's. The waste dumps belong to the state although they are within the Tishinskiy mine area. WAI understands that Kazzinc does not have any liability for these dumps. Therefore, no waste dump characterisation information has been undertaken.

Waste Dump No 2 (deposition 1965-1977) is at the southern end of the mine site and is by far the largest of all the dumps. Prior to Kazzinc leasing the mine area Slime Pond 1 (SP1) was developed within the dump. This remains state owned, however Kazzinc have permission to excavate these slimes and send them for re-processing, which should commence in 2011. It is understood from discussions that this slime pond was constructed as an engineered clay lined containment, pre 1992. For SP1 slimes processing in 2011 under the Subsoil Use Contract the slime pond has been dewatered.

Waste Dump 2 is infiltrated by rain, and this water discharges as springs appearing along the contact of dump footage and the underlying loose sediments. Flow is 153-462m^3/hr at annual average value over 236m^3/hr. To neutralise the runoff from the state-owned dump Kazzinc treats it free of charge using DMS plant overflow as a neutraliser.

Slimes Pond 2 is also located within Waste dump No 2 and Kazzinc is currently excavating these slimes, which are taken to the railway stockpile to be loaded into railway wagons and transported to Ridder concentrator for re-processing. It is understood that SP2 is lined with compacted clay. Kazzinc has been processing SP2 slimes since 2003.

The water runoff and seepage from Slime Pond 2 and Waste Dump 2 migrates in a southerly direction and is captured in 2 trenches developed by Kazzinc before reaching the River Ulba. The seepage water when mixed with waste dump 2 water has a pH level of 5 (acidic), although after mixing with alkali flow from the DMS plant in the trench, pH 7 is recorded. Following treatment, although water is discharged to the River Ulba. The residual sludge is excavated and taken to the Concentrator for re-processing. It is unclear how the trenches were designed and engineered.

> ***WAI Comment:*** *The trenches are used to catch underground acidic flow from the SP2 and mix it with alkali flow from Crushing and Concentration plant before discharging to the Ulba river which lowers the impact of the state-owned dump 2 on the Ulba river.*

Kazzinc controls the efficiency of dump runoff interception in drainage trenches via 4 boreholes monitored once a month. Kazzinc produced an OVOS for waste dump 2, an OVOS for water interception trenches and mixing of runoffs from the dump and Crushing and Concentration plant, and an OVOS for SP1 and SP2. Gidrotsvetmet Institute (Russia) and Kazmekhanobr Institute (Kazakhstan) conducted scientific researches for a long period of time and provided a detailed assessment of contaminants leaching from the dump; based on this research, the current runoff neutralisation system was implemented. It is understood from the monitoring report that soils/slimes from waste dump 2 and slime pond 2 are monitored once a year, for total, mobile and solubility for Cu, Pb, Zn, Cd, Co, Cr, Ni, Hg, As, Mn, F, Be and V and of these it appears that Cu, Zn, Co, Cr Ni, and F are soluble.

St Petersburg Mining Institute (Russia) carried out a 3-year integrated study of the environmental condition of Tishinsky mine including dump 2 (covering runoff, land cover, soils atmospheric air, etc).

Tishinsky mine waste rock is not used for the road construction as it is mainly shale which turns into clay when affected by atmospheric precipitation. The scope of liability for Kazzinc and the state was clearly defined by law in 2002. All the mine dumps belong to the state. As requested by the state authorities, Kazzinc prepared an OVOS for each dump separately and recorded the level of environmental impact. The nature and level of the dumps environmental impact was recognised by the state authorities in the environmental impact statements. The impact of Kazzinc's operations has been identified and controlled. Meanwhile Kazzinc is implementing mitigation measures for the state-owned facilities located within the mine territory, and the treatment of water is a good will gesture on the part of Kazzinc, to avoid contamination of the river.

6.7.3.5 Mine Wastes – Tailings

Ore is pre-concentrated in DMS plant. The former tailings (slimes) are discussed in section 6.7.3.4.

6.7.3.6 Water Management & Effluents

There are two discharge points to the river Ulba from within the mining licence area. Discharges to surface waters are permitted in accordance with national legislative requirements. Water balance calculations were presented for 2009 and 2010 and these are updated annually.

Water is abstracted, under permit No 03YK-183 issued by the Water Resource committee.

> ***WAI Comment:*** *There is clearly an adequate underground water supply to meet current operational requirements. Water consumption and drainage was audited as part of the draft discharge limits in 2009. The recommended measures regarding water recirculation in the mine boiler house and increased mine water consumption in DMS plant are being implemented.*

Potential soil contamination could arise via runoff from temporary stockpile and dumps, deposition of contaminated dusts, oil/grease spills, and emergency situations, such as a chemical spill or containment breach.

- Excavation, loading, haulage of tailings material from slime ponds;
- Dust blown from waste rock dumps ore stockpiles, and filter cake;
- Dust blown from road and rail transportation;
- Spills and leaks of hazardous materials and chemicals (e.g. fuels, oils, waste oils);
- Surface runoff from roads constructed from waste rock material; and
- The subsidence zone is used a landfill – Kazzinc states that this is to reclaim disturbed areas, and that soil contamination in this area does not occur.

Pollution of surface waters could result from:

- Sludge catchment trenches overtopping; and
- Unplanned discharge of untreated minewaters.

Additional processes which may result in groundwater contamination include:

- Unsegregated deposition of waste to landfill; waste is deposited in the mine subsidence zone and runoff goes to the mine water draingage system for further treatment;
- Leaching from backfill in underground working; it is stated that leaching from backfill, runoff is collected and treated by the mine drainage system
- Absorption/dissolution of explosives residues and gases;
- Hydrocarbons from the mobile fleet;
- Leaching from contaminated soils; and
- Treatment of waters - to continue until waters are clean, which is anticipated to take 12 years.

In addition to the emission sources outlined above, air pollution at Tishinskiy could also result from the following processes:

- Exhaust gases from mobile fleet, railway, and combustion processes; and
- Gases from explosives usage.

> ***WAI Comment:*** *Surface water runoff is reasonably well managed although the site could be improved by the renovation and introduction of lined ditches/interceptor trenches.*

Although there is no Kazzinc liability for the other stockpiles/waste dumps on site, leached water from these may enter the water courses and cause contamination which may affect the general site environment. As requested by the state authorities and as part of the OVOS, in 2001, Kazzinc assessed the environmental impact of state-owned dumps and identified all sources and nature of potential contamination. As part of their CSR initiatives, Kazzinc treats dump runoff via the mine drainage system, to neutralise runoff from the State owned dump 2. Kazzinc has also rehabilitated waste dumps 1 and 8.

Kazgiprozem Institute conducted an assessment and provided recommendations for reclamation of the State-owned dumps, which has been followed. Currently dust is monitored 2-3 times a year, during dry periods. The depth of soil freezing in the region is 3m and dust generation does not occur during the winter.

6.7.3.7 Emissions to Air

The air quality in the Tishinskiy area is considered to be good and there are no other significant industrial outlets in the locality. Air quality monitoring is undertaken on the border of the sanitary protection zone annually for dust, sulphur dioxide and nitrogen dioxide. Results were presented to WAI for review and found to be acceptable. These locations are not plotted on a plan; however one point is measured in close proximity to the rail line and stockpiles.

An emission stack associated with the batching plant is included in the monitoring regime; this is monitored for dust 3 times a year. All results presented in the 2009 report were acceptable. Air quality is also monitored at the ventilation shafts - see section 6.7.2.6 for details.

> ***WAI Comment:*** *Monitoring frequency and methods comply with the national environmental requirements and are approved yearly by the state authorities in the environmental monitoring programme. It is recommended that national requirements should be compared with the international standards to assess control coverage, and be amended as appropriate.*

6.7.3.8 Waste Management – General

Sewage and domestic waste water is removed from site via the main sewer system, operated by a specialist contractor. All the wastes are classified, as agreed with the state authorities, as backfilling material for reclamation of subsidence zone developed on the western side slope of the former open pit towards Tishinskiy Hill.

All the runoff from the area enters the mine drainage system and is treated at the neutralisation station.

On completion of infilling of the subsidence zone (estimated to be around 2030) the area will be remediated, by installing a clay cap and then revegetated. Ultimately this will need to be approved by the state.

Domestic (kitchen) and clinical (centre) waste is removed from site by the local authority and disposed of at the municipal landfill in Ridder.

Waste oil and greases are taken to the Ridder testing laboratory for analysis. Each batch is tested twice and the oils are either sent for recycling or disposal at a state landfill. Oily and greasy rags are collected in appropriate containers and burned on site near the subsidence zone, with the ashes disposed of in this zone. Contaminated overalls are either burned or sent to a commercial laundry for washing. Waste tyres are sold to a third party for recycling. Scrap metal from the vehicle workshops and mine is collected and stored close to the vehicle maintenance area; and is then taken off site by a third party contractor.

> ***WAI Comment:*** *In general there is a reasonably efficient waste management system in place whereby waste is sorted for either recycling, or disposal, and where putresible waste is sent off site to the state operated landfill.*

Potential rock displacement is surveyed and controlled using movement pins on an annual basis, although no movement has been noted for a long period. Water resources are monitored by the Hydrogeological Department.

PH control is performed on a regular basis; over the decades of deposit mining pH level has never been lower than 7, which is typical for ore hosting metamorphous shale. The deposit's rock and ore composition is reported not to result in ARD generation.

6.7.3.9 Hazardous Materials Storage & Handling

Fuel delivery is subcontracted. Explosives are delivered by rail to an explosives magazine at a remote part of the site. From here, the explosives required for underground use are transported in sealed containers and, lowered in by shaft to the underground storage area prior to use.

> ***WAI Comment:*** *WAI considers that in general, storage areas are appropriate and well maintained, although some of the buildings require upgrading maintenance. Where contractors are used, it should be ensured that they are fully conversant with Company handling, storage and disposal protocols.*

It was noted that spillage/cleanup protocols and spill kits were available in the vehicle maintenance plant areas for oil spills. It was demonstrated and checked that the spill kits were adequately stocked and that staff were fully conversant with the procedures for spills.

6.7.3.10 General Housekeeping

General housekeeping at Tishinskiy Mine is good, with most open areas and operational buildings being maintained in a tidy and orderly manner.

> ***WAI Comment:*** *WAI considers that housekeeping at site is good and it was noted by WAI that the inside of the operational buildings were well maintained and appeared to be working efficiently.*

6.7.3.11 Fire Safety

A fire protection plan has been developed for Tishinskiy Mine. This would appear to be very comprehensive for both above and below ground facilities.

All staff and contractors are given training in fire safety procedures, advised of emergency evacuation routes and fire fighting protocols, as part of the induction records kept. A site induction programme is given to each employee which includes fire protection details.

Fire drills are undertaken twice a year. A copy of the latest fire drill report (9 July 2010) was presented to WAI. The State mine rescue team, located in Ridder city, is responsible for underground fire management.

A designated person in each area is responsible for inspection and maintenance of fire extinguishers. During the site visit WAI staff randomly checked the extinguishers, for pressure and calibration date, and all extinguishers checked were acceptable.

> ***WAI Comment:*** *The fire management systems appear appropriate to the size of the operations, and WAI considers that these issues are being well managed by the health and safety team. WAI understands that Kazzinc intends to introduce more drill training systems, and recommends that this should be implemented as a priority.*

6.7.3.12 Security

Security for Tishkinskiy mine is provided by Group 4 Security (G4S), a world renowned company. Security is in place on all of the tracks passed through during the visit and a vehicle check of passengers was undertaken on entering and leaving the mine site. The security service is provided on a continuous basis.

The only incident to occur over the last two years was of trespassing and petty theft from employee's personal effects. No mine equipment or assets were reported to have been taken.

> ***WAI Comment:*** *It was noted that security was in attendance at site. It would also be a suggestion that the mine gate should be a lifting barrier instead of a rope/chain barrier.*

6.7.4 Permitting

6.7.4.1 ESIA/OVOS

An OVOS has been produced which includes the Tishinskiy Mine. There were no fundamentally different conclusions/impacts contained within the OVOS appertaining to Tishinskiy Mine.

6.7.4.2 Environmental Permits and Licenses

Kazzinc holds the right to mine polymetallic ores under the terms of the Contract for Subsurface Use (1994), which is valid for 25 years.

The tenure lease for Tishinskiy mine lies with the state of Kazakhstan. The mining permit is defined by the "subsoil use" contract based on an agreed tonnage with the state authorities. A water abstraction licence for potable water and discharge is in place.

The environmental emissions permit prepared by Kazzinc and agreed with the Ministry of Environmental Protection (RK), permit No.0056107 (2010), has been granted to Ridder Mining and Concentrating Complex (RMCC). This permit regulates water, waste and air, emissions at Tishinskiy.

> ***WAI Comment:*** *The WAI team has viewed a number of mine licences and permits and considers the mine licences to have been obtained in line with State requirements. It is understood that the future expansion of mining activity would not necessitate relocation of communities within the current licence boundary. The Company has environmental insurance to cover damage to third parties. When issuing air emission permits, state environmental authorities approve medium-term and annual plans for emissions and discharged minimisation which is a mandatory requirement of the national legislation.*

6.7.5 Environmental Management

6.7.5.1 Environmental Policy and Company Approach

There are no specific additional initiatives implemented by Tishinskiy Mine separate to Kazzinc's company plan.

6.7.6 Environmental Management Staff & Resources

6.7.6.1 Systems and Work Procedures

The Environmental Monitoring Programme has been agreed and endorsed by the Ministry of Environmental Protection Republic of Kazakhstan (MoE RK) and is renewable each year.

The monitoring plan includes information on location of sampling, frequency and determinands for groundwater, surface water, discharge points, snow, air, dust and soils.

6.7.6.2 Environmental Monitoring, Compliance & Reporting

Groundwater is sampled via 4 boreholes in the south of the project area. Samples are collected quarterly and analysed for pH, hardness, mineralisation, dry residue, nitrates, nitrites, sulphates, total iron, arsenic, cadmium, selenium, manganese, ammonium, copper, lead and zinc. Surface water is sampled in 2 locations from the Ulba River. Samples are collected monthly and analysed for pH, copper, lead, zinc, cadmium, iron, manganese, ammonium salts, nitrate, and nitrite, and quarterly for the additional determinands of sulphates, suspended solids, dry residue, oils and overall toxicity. The results are compared with Maximum Allowable Concentrations (MACs), and are compared with potable water levels.

2 monitored discharge points are monitored which are sampled monthly for pH, copper, iron, lead, zinc, cadmium, manganese, ammonium salts, nitrate, and nitrite, and quarterly for the additional determinands of sulphates, suspended solids, dry residue, and oils. At the neutralisation treatment facility, the pH of the neutralised minewater is monitored by resident staff every two hours on a daily basis, as a quarterly control check.

Air is monitored at 6 locations at the operational mine site batching plant, slime ponds and rail loading yard for dust and gas emissions. Several point sources may be present at each location, and are sampled 2-3 times per year. The determinants include: nitrogen dioxide and carbon monoxide, and total dust is also measured. In addition one air monitoring point is located in a residential area on the periphery of the Sanitary Protection (buffer) Zone on the far side of the Ulba River. Total dust is monitored in this location.

Greenhouse Gas (GHG) monitoring does not currently occur. GHG sources at the site would potentially include emissions from fixed and mobile plant, the acid neutralisation areas, and the lime preparation facilities.

Noise is monitored at point sources, on an annual basis. Noise levels at potentially sensitive receptors are not controlled. A working level of 80 dB is permitted, and where this is exceeded, Personal Protective Equipment (PPE) is provided, or mechanical improvements are made.

Soils are monitored at 2 locations on an annual basis and are analysed for the following lead, cadmium, copper, zinc, iron, bismuth, antimony, arsenic, mercury, fluorine, nickel and cobalt. Mobile forms of cobalt, copper, nickel, fluorine and chromium are also analysed together with soluble forms of copper, lead, zinc, cadmium, arsenic, fluorides, sulphates, manganese, nickel, cobalt and chromium.

Radioactive point sources are monitored, on a quarterly basis, using either mobile or automatic detection equipment. Radon exposure is also monitored twice per year in working areas, including the underground mine, and at planned construction sites together with scrap metal if required.

Collecting of air quality, tailings/slimes analysis and supernatant surface and discharge waters is undertaken by Kazzinc environmental laboratory, and samples are analysed at Kazzinc analytical laboratory, which is nationally accredited and located at the RMC. Groundwater and soils are monitored by specialist contractors.

Results are reviewed and reported monthly by the Company Environmental Managers.

> ***WAI Comment:*** *The scope of the environmental monitoring programme has been developed in line with national legislative requirements. The Kazzinc Environmental Managers, Laboratories and third party specialists are aware of compliance requirements and analytical and sampling methodologies required to assess these. WAI therefore considers that the monitoring programmes are fully compliant with national standards, however it is recommended to make a comparison against international best practice, and amend as necessary.*

It is considered that the quantity of river sample points (3) is sufficient (upstream from the mine, after the mine water discharge point and downstream of the mine). Ochre coloured discharge was noted at the downstream water discharge point on the Ulba River. This is suggestive of the influence of the rock dumps and embankment formed by the open pit overburden, all of which are property of the state and outside Kazzinc's responsibility.

The quantity and location of point source air quality monitoring sites is considered adequate.

Dust pollution monitoring is undertaken in and around slime ponds and the rail loading yard 2-3 times a year. 2009 results were presented to WAI, and found to be compliant with national standards.

The stacks on the DMS plant (total of 9 stacks Nos 125-133) are monitored 2-3 times a year. 2009/2010 testing results were presented to WAI, and found to be compliant with national standards.

6.7.6.3 Emergency Preparedness & Response

A total of 34 Emergency Preparedness and Response Plans covering various possible scenarios has been developed. These outline responsibilities, actions and reporting requirements should an incident occur.

> ***WAI Comment:*** *The plans were comprehensive, however these plans have not been tested via any actual drills. Out of the 34 plans, WAI randomly selected Response Plan No 1, to review as an example, and this was presented to WAI together with a discussion on how the plan works. The plan appeared adequate, but plan testing via drills should be implemented.*

6.7.6.4 Training

There are no specific additional initiatives implemented by Tishinskiy Mine separate to Kazzinc's company plan.

6.7.7 Social and Community Management

There are no specific additional initiatives implemented by Tishinskiy Mine separate to Kazzinc's company plan.

6.7.7.1 Stakeholder dialogue and grievance mechanisms

There are no specific additional initiatives implemented by Tishinskiy Mine separate to Kazzinc's company plan.

6.7.7.2 Social Initiatives and Community Development

There are no specific additional initiatives implemented by Tishinskiy Mine separate to Kazzinc's company plan.

6.7.8 Health & Safety

6.7.8.1 Health & Safety Management Arrangements

Tishinskiy mine has a H&S Manager, who is responsible for the H&S of both above and below ground operations. A total of 51 ''Safe Work Procedures' are in existence covering each aspect of work undertaken at the mine. The H&S Manager has a team of 5 inspectors per shift which cover both above and underground works. During each shift these persons randomly check the working areas. The Company is also accredited for H&S management under the OHSAS 18001 scheme.

The environmental training covers issues such as dealing with spills, waste management, water and power consumption and how to manage and minimise environmental risks. Managers also receive training every 3 years, from State authorities. WAI viewed a number of personnel training logs.

The Company also has a special clinic in Ridder, where all employees receive annual medical checks. The Company has a contract with the State ambulance provider and the nearest hospital with an Accident and Emergency (A&E) department is in Ridder city, a few kilometres away. Spot tests for drugs and alcohol are also performed.

Noise, dust and gases are monitored in working areas, on a continuous programme. PPE is provided to all employees, and additional PPE is provided where necessary eg ear defenders.

> ***WAI Comment:*** *WAI considers that health and safety is very well managed at the site, with appropriate training being provided, and good response systems in place for personnel. PPE was worn in all areas to some degree, however it was noted that on the day of the site visit, some staff did not*

have hi-vis PPE on or hard hats, this was especially noted in the area of the rail sidings and landfill/subsidence area. It should be noted that heavy plant was in use in these areas. In general H&S managers and their teams are proactive in their management of H&S issues, both with regard to national and international requirements. WAI would recommend implementing drills to test Emergency Response Plans, to assess their adequacy.

It was noted that there were areas which would benefit from additional warning/instruction signs, to comply with international standards. WAI would suggest that although there is obviously a road infrastructure on site, there was a lack of signage indicating ''give way' and 'stop' and 'one way' etc. It was also noted that a speed limit was displayed on some roads; however it is the opinion of WAI that this was not being adhered to on site.

6.7.8.2 Performance and Accident Records

Tishinskiy mine generally has a good health and safety record. The last time an accident happened was in 2009, where an employee was off for 3 months.

There are clear lines of responsibility in case of an accident, and disciplinary and re-training procedures exist if unsafe working practices are noted by H&S managers. The managers can also stop production and issue warnings if breaches are noted. These are then recorded, and remedial measures, with a given timescale for implementation are devised. The progress on implementation is then inspected by the H&S manager.

WAI Comment: *WAI considers that the health and safety management is very good at the site, and that the appropriate record keeping and improvement procedures are in place to deal with incidents. Overall the safety records are considered good, and all personnel seem committed to improving safety management.*

6.7.9 Mine Closure & Rehabilitation

6.7.9.1 Mine Closure Plans

In accordance with state legislation the mine closure plans are not required to be prepared until 2 years prior to closure.

WAI Comment: *WAI recognises national closure requirements are being met, and there is a closure program developed at the mine with specified types and volumes of works to be performed and closure fund established and kept on the Kazzinc's account. Detailed project specifying closure costs will be developed 2 years prior to the mine closure, however this should be compared and updated in line with international standards and should cover unplanned company closure.*

6.7.9.2 Financial Provision for Closure

Tishinskiy Mine initial potential closure cost is estimated at US$1,000,000, and this sum was deposited in a protected Company fund. This has been increased to US$4,000,000. Typical closure measures include demolition of surface structures, stripping of utilities, and stabilisation of mining related features, earthworks and re-vegetation. It is reported that standard post-closure monitoring includes: rivers and groundwater, mine water, dust, soils, flora and fauna. It is also reported that post-closure aftercare will include the continuation of mine water treatment after flooding of the mine workings. At Tishinskiy, this is envisaged for 12 years, before the workings are allowed to flood and groundwater rebound occurs.

It is reported that the closure fund will periodically be updated during the mine life, with a final detailed closure plan to be prepared 2 years before actual closure, in line with State requirements.

WAI Comment: *The current closure fund estimate is revised on an annual basis and will be itemised during closure project development. It is recommended that funds include sums for unplanned* *company closure.* The environmental budget is prepared on an annual basis by the environmental managers and this is based on the continuous environmental improvement strategy of the mine.

6.7.10 Conclusions

Kazzinc has acquired a number of historic mining features at Tishinskiy Mine. SP1 and waste dumps remain the property of the State therefore liability remains with the State.

WAI considers that Company environmental management and monitoring is performed in line with national requirements, and Kazzinc has achieved accreditation under international environmental, quality and health and safety management systems (ISO 14001 and OHSAS 18001). The Environmental management system has been implemented, with findings of annual inspections speaking for its continuous improvement.

A key area for attention is water recycling and management, together with mine closure provisions.

The Company is not held liable to the government for any historical contamination, and considers that liabilities for current operations are clearly defined.

In 2010 an independent environmental audit was conducted at Kazzinc which included Tishinsky Mine. During the audit, the adequacy of all environmental aspects, risks, targets and environmental management programmes used at the operations were assessed.

7 SHUBINSKIY DEPOSIT

7.1 Introduction

The deposit was discovered in 1846 and initially named Priisk-1. Three years later two silver-bearing mineralised bodies were exposed in shallow underground exploratory workings. However, it appears that no further prospecting or mining was undertaken and the occurrence was overlooked until its rediscovery during regional prospecting and prospecting-assessment programmes in 1954-1955.

Trial underground production began in 1990/1991 and continued intermittently and at low output rates for over six years. In 1999 the production exceeded 100kt and has since risen to reach its full capacity of 200ktpy in 2009. During the period 1992-2009 inclusive, the mine produced 1.6Mt of ore grading 0.66g/t Au, 17.9g/t Ag, 1.74% Cu, 0.28% Pb and 2.25% Zn. The mine is operated by TOO Shubinskiy, who had worked as Kazzinc's mining contractors prior to April 2009.

The mine development consists of five underground levels connected by two vertical shafts located on the north-western and south-eastern flanks of the deposit. Level 1 is at an absolute elevation of 980m (at 75-100m depth). Level 2 is developed 50m below level 1 and 60m above level 3. Levels 3, 4 and 5 are 60m apart. The shaft on the north-western flank is 319m deep and is used for personnel, equipment and ore and waste hoisting.

7.1.1 Location and Access

The Shubinskiy mine is situated in the Glubokovskoe region of East Kazakhstan province 14km north-east of Ridder from which it is readily accessible by an all weather graded road (Figure 7.1).

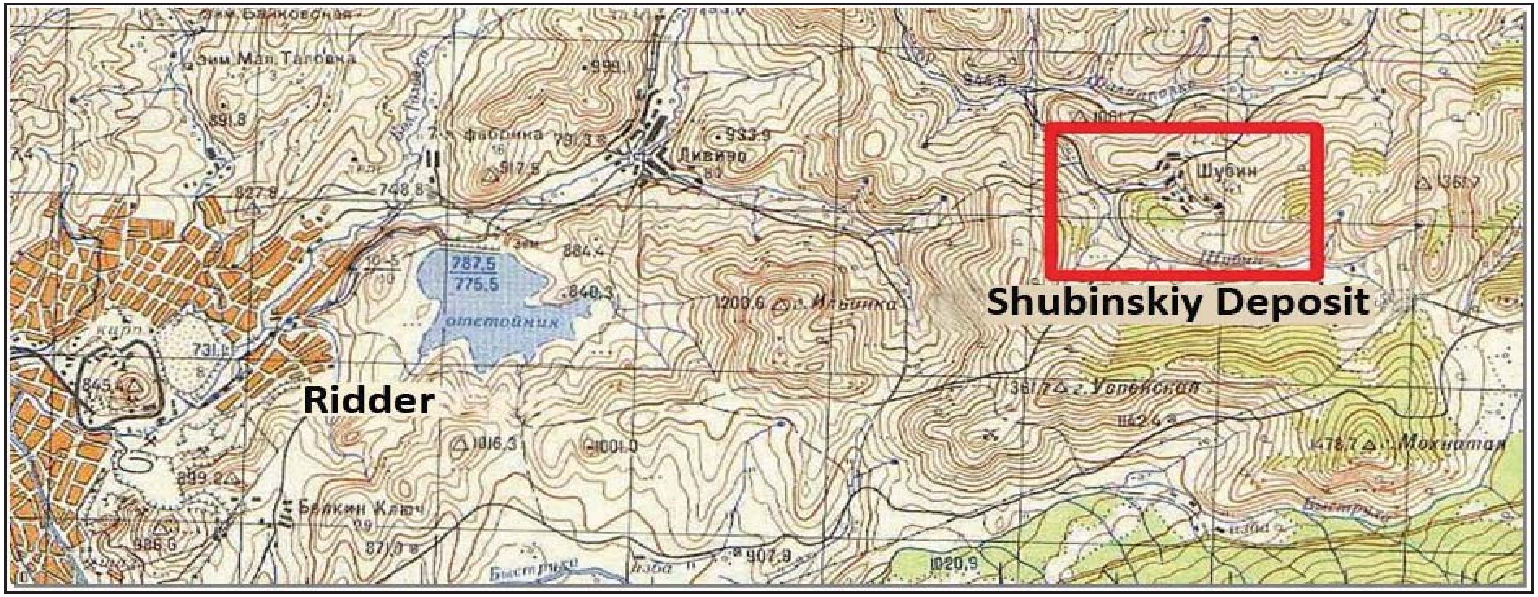


Figure 7.1: Location of Shubinskiy Mine in Relation to Ridder
(Scale – Each Division± 2km)

WAI Comment: *The Shubinskiy site is favourably located in relation to a major centre of industry and population at Ridder, providing a significant advantage to the project.*

7.1.2 Topography and Climate

Topographically the area comprises upland with strong relief, the deposit itself being located on an undulating plateau, with ground elevations ranging from 1,000-1,100m above sea level; the plateau is bounded to the north and south by the deeply dissected valleys of the Martyn and Shubin tributaries of the Ulba river the waters of which are potable and suitable for technical purposes.

Climate is extreme continental with the average annual temperature of 1.6°C and seasonal variations from +34°C in July to -45°C in December. Mean annual precipitation is 667mm, of which the maximum falls as rain in summer months. Snow cover lasts for 160 days from November to April. The ground freezes to depths of up to 1.5-2.0m.

The entire region (with exception of the higher mountain region) is forest-covered, predominately with coniferous species.

7.1.3 Infrastructure

The basis of the local economy is the production of non-ferrous metals at the Kazzinc Ridder Concentrator and Metallurgical Complex from production of Ridder-Sokolniy, Tishinskiy (underground and slimes), Shubinskiy mines and sands from the Staroye tailings dam. Electric power is provided by the Bukhtarminskoy power station, supplemented by the Ridder heat and power plant. An adequate labour supply is available in Ridder.

Building materials, apart from timber from the abundant forests, include the various local volcano-sedimentary lithologies, gravel- pebble deposits, clays and cement.

Other local industries comprise forest management, a meat-packing plant and a clothing factory amongst others but agriculture is poorly developed, as a consequence of the dissected relief.

7.1.4 Mineral Rights and Permitting

Kazzinc holds its tenure rights to mine polymetallic ores from the Shubinskiy deposit through its wholly owned subsidiary TOO Shubinskoe. Contract No.1296 for Subsurface Use was initially granted to AO Leninogorsk Polymetallic Complex (LPC) on 30 December 2003 and then transferred to TOO Shubinskoe on 8 April 2009. The contract is valid until 7 April 2015 and can be extended by mutual agreement in writing between the tenure holder and the issuing authority; it supersedes Licence Series MG No 68 granted to AO Leninogorsk Polymetallic Complex on 7 April 1995.

The Shubinskiy mining lease covers an area of 97ha (0.97km^2) defined by four boundary points (Table 7.1).

The legal depth limit of mining defined in the mining licence is level 9 at an elevation of 510m above Baltic Sea datum. The mine is developed to level 5 at the elevation of 750m. The deposit as currently delineated, peters out between the elevations 570m to 510m.

Table 7.1: Mining Licence Boundaries

Boundary Points	Local Interim Co-ordinates*[1]	
	X	Y
1	30,590	-4,760
2	31,340	-4,140
3	30,710	-3,365
4	29,960	-3,995

***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.*

7.2 Geology and Mineralisation

7.2.1 Regional Geology

The Shubinskiy deposit is situated at the boundary of the Uspensko-Karelinsky and Shubinskiy structural blocks within the Ulbinskaya branch of the Uspensko-Karekinska shear zone.

The Uspensko-Karelinsky block underlies the southern part of the deposit area. It is composed of Lower-Middle Devonian volcano-sedimentary rocks of the Leninogorskaya, Kriukovskaya, Ulbinskaya and Uspenskaya formations. The Shubinskiy block containing terrigenous sediments of the Middle-Upper Devonian Beloubinskaya Formation underlies the northern part of the deposit area.

The contact between the two blocks strikes 315-330° and dips 65-85° NE. Rocks on either side have been subjected to intense polyphase deformation, which has resulted in complex folded structures and pervasive schistosity. The contact itself is a shear zone.

7.2.2 Deposit Geology

7.2.2.1 Stratigraphy and Lithology

The deposit covers an area of 600x110m extending from north-west to south-east along the contact of the Uspenskaya Formation (D_2ef-qv us) and the Beloubinskaya Formation (D_2qv_2-D_3fr ЬІ). It has been delineated by diamond core drilling from surface to a depth of 730m. The dip is concordant with schistosity, generally within a range of 65-85°NE.

The deposit is hosted by volcanogenic-sedimentary rocks of the Uspenskaya Formation in the contact zone with the sediments of the Beloubinskaya Formation as shown in Figure 7.2. The host rocks comprise chlorite-sericite-quartz schists and sericite quartzites derived primarily from acid tuffs and their reworked derivatives.

Rocks of the Beloubinskaya Formation, which form the hanging wall of the deposit, comprise carbonaceous-pelitic and pelitic siltstones, silty sandstones with intercalations of sandstones and occasional small lenses of acid tuffs in the immediate vicinity of the contact with the Uspenskaya Formation.

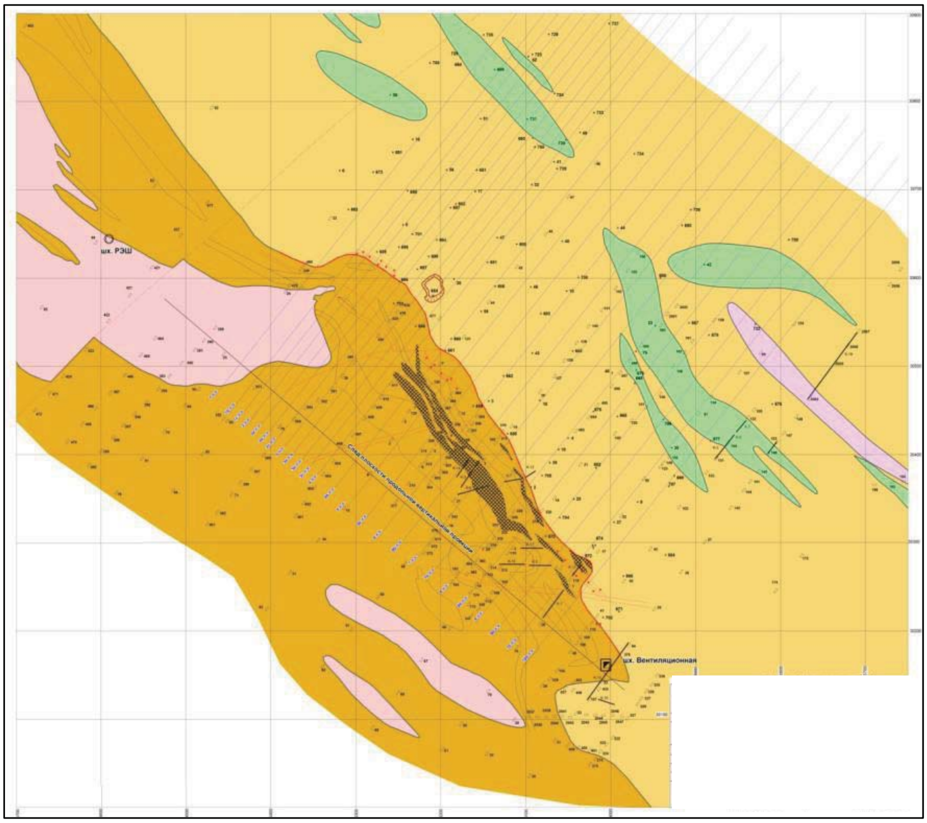


Figure 7.2: Geology Showing the Mineralised Bodies and Lenses (Hatched) on the Contact between Uspenskaya Formation (Yellow) and the Beloubinskaya Formation (Pale Yellow) with Magmatic Rocks (Green and Mauve)
(Scale: Grid 100m)

7.2.3 Ore Body Morphology

The deposit consists of two clusters of sub-parallel, tabular and lenticular mineralised bodies, one on the north-western flank of the deposit and another on the south-eastern flank, referred to as the North-Western Lode and the South-Eastern Lode respectively as shown in Figure 7.3 below. In the 1989 report, the ore bodies were numbered in sequence from hanging wall to footwall as shown in Table 7.2 below.

Orebodies No.1 and 2 host the bulk of the mineralisation. Orebody No.1 extends throughout the deposit as a string of narrow boudinaged lenses and widens into two sub-vertical shoots at both ends. In the South-Eastern Lode area, the shoot extends for 580m down dip (from level 1 to level 9) and attains 210m in strike length on level 5. The smaller shoot in the North-Western Lode area breaks up into separate lenses which in aggregate extend for 550m down dip. The combined strike length in this area is 100m.

Orebody No 2 also extends throughout the deposit as a narrow pinch-and-swell structure and opens up into a large shoot in the North-Western Lode area. This shoot extends down dip for 725m (from level 1 to level 9), attains 220m in strike length and 28m in width. A smaller shoot on the South-Eastern Lode area extends down for 420m and has a strike length of 170m.

The ore bodies and lenses themselves, dip 65-80° NE and strike 320-340°. They vary from 100-130m to 200-220m in length, extend down dip from 190m to 500-725m with widths between 0.26-0.40m to 21-28m and average values from 1.6-9.0m. The morphological variations include columnar, linear and lenticular, and the contacts are irregular as a consequence of tectonic disturbances; apophyses also occur. The main parameters are summarised in Table 7.2.

Five ore bodies and 24 lenses of pyritic polymetallic mineralisation have been identified at Shubinskiy. The majority of the reserves (78.7%) are concentrated in the No.1 and No.2 ore bodies, and 63.9% in these bodies in the NW Lode where the grades are higher. Only 0.9% of the reserves are contained in the 24 small lenses.

The ore zone hangingwall contact coincides with the interface of the Beloubinskaya and Uspenskaya formations whilst on the south west the boundary is diffuse and is defined by the progressive disappearance of polymetallic and pyritic bodies. The zone has a strike length of 550m and extends down dip from 450-580m; below 500m the ore bodies decrease in size and the mineralisation becomes discontinuous.

The North-Western Lode has almost been exhausted from level 1 to level 4 and is now mined on level 5. The South-Eastern Lode is mined on level 2, whilst stopes are being prepared on level 3. Mixed sulphide-oxide mineralisation in the upper part of both lodes remains largely intact on and above level 1.

Table 7.2: Characteristics, Morphology and Parameters of Ore Bodies

No of Ore Body	Ore Body Location			Drilling Grid From-To/Average		Ore Body Dimensions(m)			Grade (Range) average, %			BeddingDip/ Azimuth	Morphology of the Ore Bodies
	Lode	Horizon	X-Sections	C_1	C_2	Strike Extent/ Continuous	Dip Extent/ Continuous	Thickness Range/ Average	Copper	Lead	Zinc		
1	NW	1-9	2B-4B	15-25*50 25	40-60 50	100 100	550 150	0.68-7.20 1.89	0.02-5.21 1.72	0.08-3.19 1.20	0.09-10.48 5.55	35-60° 50-85°	Linear
	SE	1-9	5B-11B	10-25*60 31	25-40*80 60	210 210	580 580	0.4-11.0 3.11	0.02-11.55 0.47	0.04-3.85 0.55	0.09-21.3 3.08	10-90° 50-85°	Linear
2	NW	1-9	0-5B	15-25*75 42	25-75 40	220 220	725 725	0.83-28.2 9.0	0.02-6.18 2.78	0.08-4.85 0.62	0.09-14.22 4.68	15-75° 53-85°	Columnar Linear
	SE	1-9	6-10B	15-25*60 25	20-75 53	170 140	420 420	0.58-7.40 2.39	0.02-16.35 1.20	0.02-2.56 0.34	0.09-10.5 2.57	48-70° 46-75°	Linear
3	NW	1-8	1B-5B	10-25*70 35	25-100 70	140 85	500 340	0.26-14.8 3.7	0.02-7.4 1.59	0.08-2.08 0.38	0.2-9.91 2.35	35-57° 52-83°	Columnar Linear
	SE	1-7	7B-10B	10-25*60 33	10-60 25	130 130	340 340	0.37-10.5 3.43	0.08-15.22 1.57	0.02-7.47 0.41	0.05-23.35 2.10	54-120° 40-90°	Linear
4	NW	1-6	2-5	10-60 30	25-70 65	130 80	540 300	0.52-21.3 4.5	0.10-4.12 1.73	0.02-0.25 0.22	0.06-4.41 1.99	30-60° 60-77°	Linear
	SE	2-3	7B-9B	10-50*50 36	40-60 40	130 75	190 120	0.75-6.2 3.21	0.02-12.10 1.58	0.02-1.9 0.07	0.06-1.85 0.69	30-155° 65-90°	Linear
5	NW	3-7	3B-5B	-	25-60 60	50 50	185 185	0.78-3.17 1.6	0.08-2.08 1.0	0.02-0.75 0.16	0.02-6.2 2.05	35-42° 68-70°	Lenticular
	SE	3	9B	-	15-50	25	11	1.72	5.53	0.73	2.84	36-142° 70-80°	

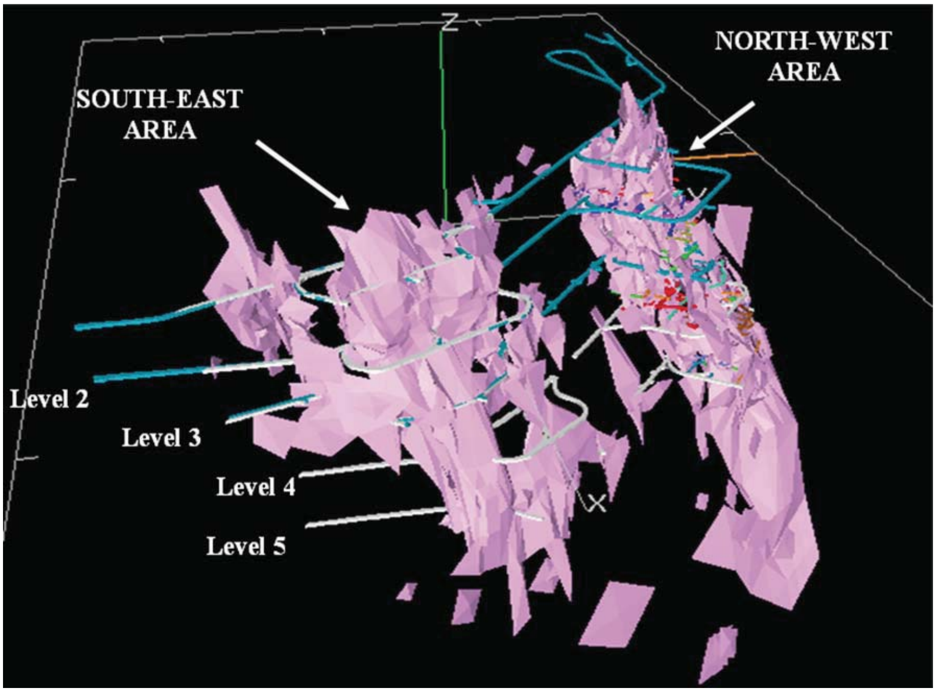


Figure 7.3: Modelled North Western and South Eastern Lodes

7.2.3.1 Structure

The tectonic structure of the Shubinskiy deposit is complex; the deposit is located on the south western flank of the NW striking Listvyazhnaaya syncline, where the Shubinskoya syncline and its small transverse plications, thought to be related to the mineralisation phase, complicate the structure.

Ore bodies and lenses are metamorphosed and sheared, the local shear zones range from 10-20m in width and 50-550m in length (Figure 7.4). Fracture dislocations and faulting, believed to be subsequent to the mineralisation processes are characterised by crushing and shearing; these disruptive features, one of which occurs along the contact of the Beloubinskaya and Uspenskaya formations, extending over the length of the deposit, strikes NW-SE and dips at 60-80°NE (Figure 7.5). The smaller crush zones have been observed in the metasomatically altered and sheared acid tuffs and tuffites.

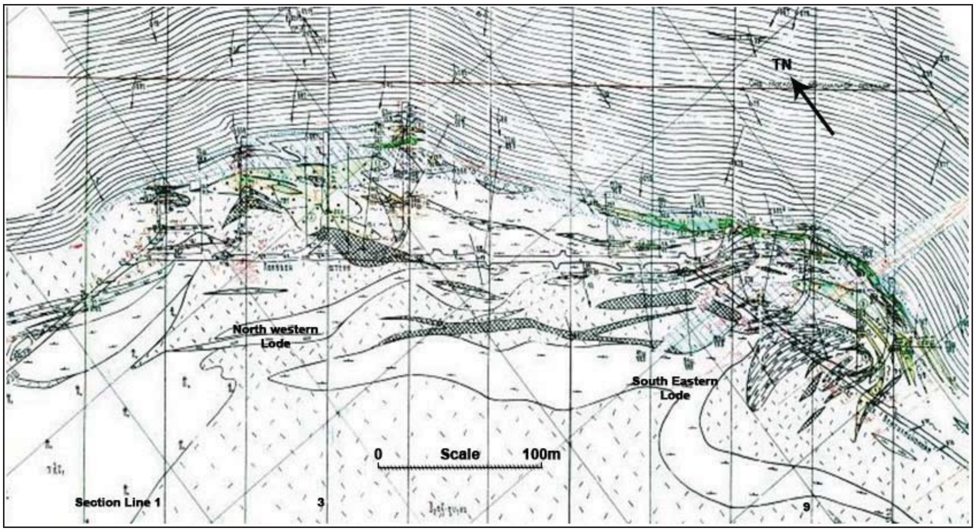


Figure 7.4: Plan of Level No.3 (870m) showing the NW and SE Lodes and the Dislocated and Sheared Ore Bodies

(Section Lines Indicated by 1, 3 and 9)

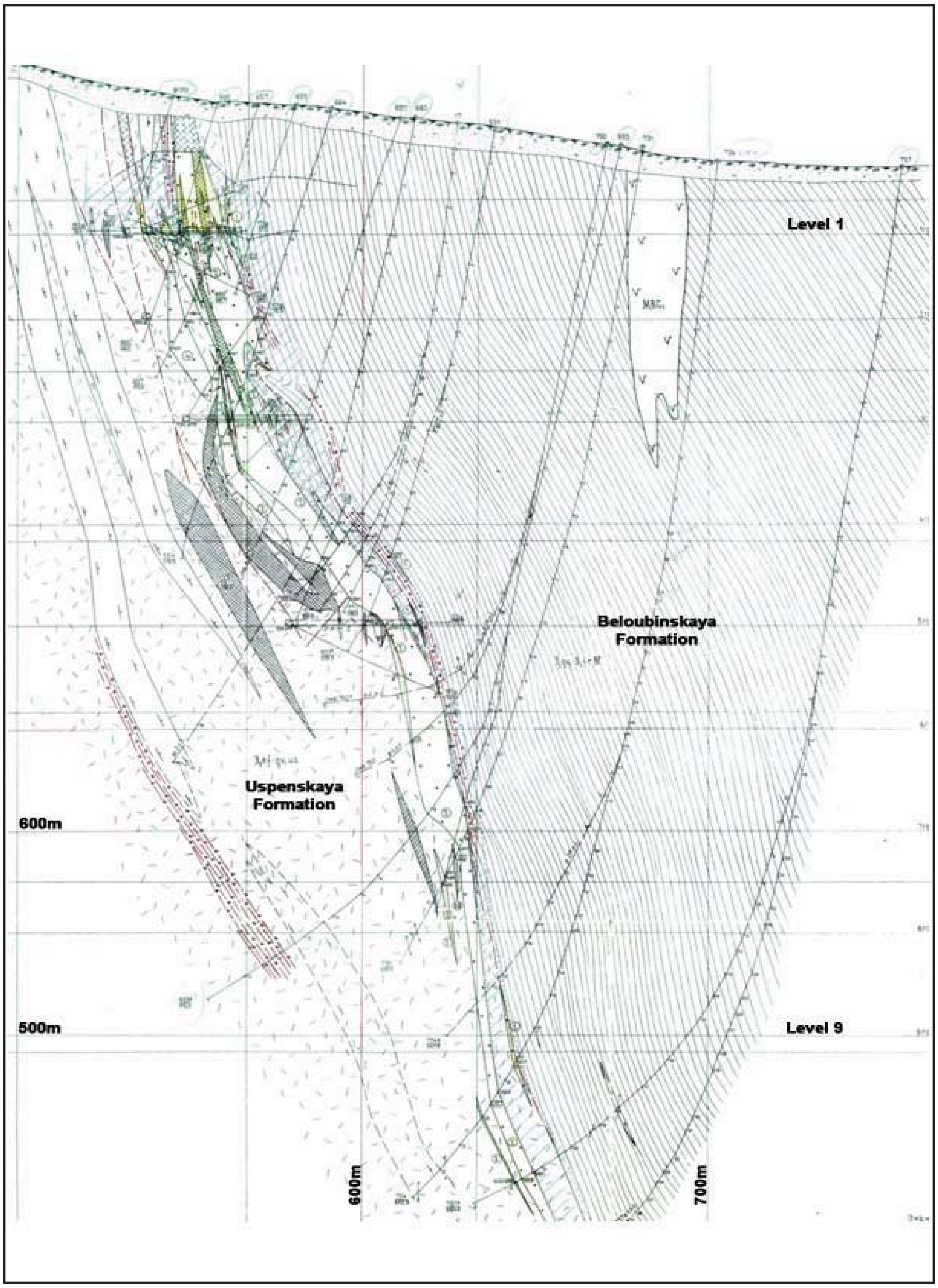


Figure 7.5: Cross Section No 3 through the North Western Lode
(Refer to Horizontal Section of Level 3 in Figure 7.4 for Position of Section Line)

7.2.4 Mineralogy

The primary sulphide and mixed zones are defined by the degree of oxidation, the former comprising <12% copper, <20% lead and <10% zinc, oxidised minerals. The corresponding figures for the mixed zone are Cu 12-50%, lead 20-50% and Zn 10-30%.

Primary sulphide mineralisation comprises four major mineral associations:

- Pyritic;
- Chalcopyrite (chalcopyrite- pyritic). Cu≥(Pb + Zn);
- Pyritic-copper- zinc (chalcopyrite-sphalerite- pyrite) Cu≥ Pb; and
- Polymetallic, substantially lead-zinc (pyrites-galena-sphalerite). Cu ≤ (Pb + Zn).

Veinlet-type and disseminated mineralisation predominate. Massive sulphides occur as isolated lenses and altogether make up approximately 11% of the original 1989 resource. Table 7.3 lists primary minerals identified at the deposit.

Table 7.3: Primary Minerals

Abundance	Metalliferous Minerals	Gangue Minerals
Main	Pyrite, sphalerite, chalcopyrite, galena	Quartz, sericite, chlorite
Abundant	Arsenopyrite, tennantite-tetrahedrite, marcasite, melnikovite (layered) pyrite	Rutile, calcite, siderite, leucoxene
Accessory	Pyrrhotite, hessite, tellurobismuthite, native gold, native silver	

The depth of oxidation varies from 25-35m and is defined as the boundary above which oxidised minerals of copper, lead and zinc exceed 50%, 50% and 30% respectively.

Pyritic and copper-pyrite sulphide mineralisation, being more susceptible to surface weathering, has been converted to a porous mixture of limonite, goethite, lepidocrocite and halloysite. Secondary copper minerals include malachite, azurite, tenorite, cuprite, native copper and chrysocolla. Base metal grades are low and for this reason oxidised mineralisation is not included in the reserves. Gold and silver grades have not been determined in the oxidised zone.

Mineralisation containing oxidised forms of copper, lead and zinc in the range of 12-50%, 20-50% and 10-30% respectively, is classified as mixed and extends from the base of oxidation approximately down to level 1 at an elevation of 980m. This is a supergene sulphide enrichment zone containing inter alia chalcocite, covellite, bornite, goslarite (hydrous zinc sulphate) and chalcanthite (hydrous copper sulphate). Copper enrichment is approximately 1.5-1.9 times in the North-Western Lode and 4-4.2 times in the South-Eastern Lode. Zinc enrichment is 1.2-1.5 times. The enrichment is defined as a ratio of the average grade of mixed mineralisation to the average grade of primary sulphide mineralisation.

Horizontal zoning is reflected in the predominance of lead and zinc near the hanging wall and a gradual decrease towards the footwall, where the copper content increases. In continuous and disseminated sulphide bodies the lower horizons are pyritic. The NW section has a higher copper content than the south east section; vertical zoning is manifest by the more rapid decrease in copper than lead and zinc.

Increased concentrations of tellurium and selenium are characteristic of the chalcopyrite and lead-zinc ores and increased grades of cadmium (up to 0.11%) and silver (up to 270g/t) occur in continuous pyritic-polymetallic ores. Gold grade is usually low (average 0.2-0.6g/t), although higher values have been recorded occasionally, in the upper part of the deposit.

Harmful impurities include silicon dioxide, iron and arsenic. Iron as pyrites is up to 37% in chalcopyrite ores which are characterised by increased arsenic (up to 0.3%), also in the upper parts of the deposit. The maximum content of silicon dioxide in copper-zinc ore is 81.8%.

The chemical composition of the Shubinskiy ores is given in Table 7.4.

Table 7.4: Chemical Composition of Ores

	Contents %
Na_2O	0.01-0.20
MgO	0.07-1.50
Al_2O_3	0.39-16.84
SiO_2	2.11-93.6
P_2O_5	0.008-0.168
K_2O	0.04-5.69
CaO	0.07-0.36
TiO_2	0.008-0.23
MnO	0.02-3.71
$Fe_2O_{3\ total.}$	0.04-64.06
S_{total}	1.05-49.01
BaO	0.03-0.50
PbO	0.05-4.48
CuO	0.05-9.59
ZnO	0.05-21.6

7.3 Exploration

7.3.1 Historical Work

The deposit was discovered in 1846 and initially named Priisk-1 and was first properly investigated during regional prospecting and prospecting-assessment programmes in 1954-1955.

Subsequent exploration was carried out in three stages:

- Preliminary exploration conducted by Leninogorsk GRE in 1954-1960;
- Preliminary exploration conducted by Leninogorsk GRE in 1973-1975, targeting deeper levels of the deposit and its flanks; and
- Detailed exploration and simultaneous mine development by LPC in 1976-1989.

A summary of work accomplished during the period 1954 to 1989 inclusive is given in Table 7.5.

Table 7.5: Exploration Summary					
Exploration Works Conducted	Unit	Prospecting Assessment Date	Preliminary Exploration		Detailed Exploration and Mine Development
		1954-1955	1954-1960	1973-1796	1986-1989
Geological Mapping 1:10,000	km^2	50			
Prospecting Traverses	km^2	92			
Geological Mapping 1:2,000	km^2	2			
Cartographic Drilling	m	13,968			
Trenching	m^3	1,800			
Pitting	m	3,391			
Diamond Core Drilling					
Surface Drilling	m		17,616	33,256	
	number		60	86	
Underground Drilling	m				9,575
	number				181
Drill Core Samples	number		2,590	3,120	9,421
Underground Openings					
Vertical Workings	m				651
Horizontal Workings	m				5,440
U/g Channel Samples	number				3,782

7.3.2 Drilling

7.3.2.1 Surface Diamond Core Drilling

Surface drilling was carried out by the Leninogorsk GRE. Drill holes were sited on profile lines, bearing 219° and inclined south-west. Initial inclinations ranged from 73-86° and subsequently holes were deflected to lower angles either with or without the use of directional wedges. The initial drilling diameter, either 112mm or 132mm, was reduced to 76mm and 59mm below the oxidised zone.

Drill collar locations were surveyed by instrumental methods. Downhole surveys were carried out at 20m intervals, or more frequently around deflection points. Control depth measurements were taken at intervals of 100-150m.

The average core recovery over mineralised intercepts for surface drillholes was 75.6%.

> ***WAI Comment:*** *the core recovery for the mineralised intercepts would be considered unsatisfactory by current drilling standards, but is probably explained by the complex disruptive fracturing which characterises the ore bodies; no core was available for inspection.*

7.3.2.2 Underground Exploration

Kazzinc obtained the subsurface use right in 2004. Previously this right was owned by HSC Leninogorsk Polymetallic Complex. For the period 1997-2003 inclusive, only 4,231m of underground workings were drifted through infill drilling and 1,403m of channel samples were taken from mine workings.

After Kazzinc obtained the subsoil use right it intensified underground infill drilling using a modern Diamec 232 drill rig. For the period from 2004 till 2010, 36,397m of prospecting holes were drilled and 982m of channel samples were taken from mine workings (see Table 7.6 below).

Table 7.6: Exploration Works 2004-2010					
Year		Core			
	Samples Taken	BSK	Diamec	Sludge	Trench
1997	1,232	0	0	1,229	0
1998	1,078	46	0	396	563
1999	1,500	142	0	1,077	276
2000	555	0	0	457	86
2001	585	0	0	278	307
2002	297	0	0	174	115
2003	488	0	0	432	56
2004	606	0	0	248	355
2005	2,015	1,278	0	405	323
2006	3,287	2,044	0	786	304
2007	7,477	773	5,250	1,379	0
2008	7,240	93	6,917	230	0
2009	8,700	0	8,306	348	0
2010	8,096	0	7,660	432	0
Total 2004-2010	38,027	4,188	28,133	40,76	982

Based on such works, the density of the exploration grid within levels 1-5 has improved to 10-15mx12.5m (explored along strike by either drill holes or mine workings on 12.5m centres, and down dip from 10-15m). Level 6 reserves were investigated on a spacing of 25m. This exploration grid has greatly improved the confidence in estimates and reserve category accordingly.

Underground exploratory workings prior to the commencement of mining were developed on levels 1, 3 and 5 at vertical intervals of 110-120m. Horizontal workings comprised crosscuts on a bearing of 39° at 50m intervals across the entire width of the deposit plus several drives in the North-Western and South-Eastern lodes to confirm the lateral continuity of the mineralisation, and drill cuddies.

Superimposed on the exploratory workings, haulage and extraction drives, crosscuts, sublevels, service raises and other underground openings were subsequently developed in the process of stope preparation. The frequency of such workings varies from 10-20m along strike and 12-20m down dip.

Geological documentation for each development consists of 1:100 scale sections of both side walls and, in areas with more complex geology, the backs, with annotations and descriptions.

As a general observation, drawings and descriptions adequately address forms and types of mineralisation but are short on structural details. Some workings have photographic documentation.

Direct underground exploration was augmented with underground diamond core drilling. Underground holes are collared on profiles at 25m intervals and fan drilled on an orientation from north-east (39°) and south-west (219°). Some drillholes sited on the south-eastern flank of the deposit were drilled on different bearings to define a subsidiary fold structure developed in this area.

Upside-facing and horizontal holes were initiated with 59mm external diameter drill bits and downward-facing holes with 76mm external diameter bits. The hole sizes were subsequently reduced to 46mm and 59mm respectively. Out of the total 9,575m of drilling, 812m were drilled using double core barrels.

Drillhole collar positions were surveyed by instrumental methods. Downhole surveys, which were routinely conducted in virtually every borehole, did not reveal any significant deviations.

The overall core recovery in underground drillholes completed to 1989 was reported as 70.5%, whilst the overall core recovery from mineralised intersections was reported as 73.2%. Some holes with low core recovery were redrilled (a total of 337m). Roentgen-radiometric logging (RRK) was conducted in most boreholes and the results were substituted for chemical analyses in 9 holes with low core recoveries.

A statistical analysis based on hydrostatic weighing of 1,000 drill core samples from the detailed exploration stage revealed that core recoveries by weight were on average approximately 8% lower than linear recoveries recorded in drill logs.

> ***WAI Comment:*** *The core recovery for the mineralised intercepts would be considered unsatisfactory by current drilling standards, and confidence is further reduced by the discrepancy between linear measurements (on which the recoveries are assumed to be based) and hydrostatic weighing.*

7.3.3 Sampling

7.3.3.1 Sampling Techniques

Drill core samples were collected from all intervals with visible polymetallic sulphide mineralisation and from lithologies on either side of each mineralised interval. Most intervals with hydrothermal alteration and disseminated pyrite were also sampled.

Drill core samples from the 1955-1961 and 1974-1975 drilling programmes were split in a mechanical core splitter. Sample intervals ranged from 0.2m to 2m in length, but did not exceed 1m over mineralised intervals.

Relatively small diameter underground drill core was not split, except for intervals subjected to control duplicate sampling. Sludge samples were collected in addition to drill core over intervals with low core recoveries.

Geochemical chip samples were collected from barren intervals.

The variability of drill core sampling was assessed by comparing analytical results from two corresponding halves of drill core. Relative differences did not exceed 12% and had no negative or positive bias.

RRK readings reportedly showed no correlation with drill core recovery at recoveries in excess of 70%. However, a comparison of results from 9 pairs of drill intercepts located in close proximity to each other showed that, on average, copper and zinc grades were higher at lower core recoveries, whilst lead grades were lower. No explanation was given.

Underground workings were sampled by cutting horizontal channels along both walls of crosscuts through mineralised intervals, alteration zones with disseminated pyrite and wall rocks adjoining sulphide mineralisation. Channel dimensions were generally 0.10x0.03x1.0m.

Drives through mineralised bodies were sampled by taking horizontal channels from faces exposed progressively at intervals ranging from 4-5m, as drives were being developed.

Drives through barren rocks were sampled at intervals of 17-23m

Channel samples were collected using either pneumatic hammers or rock cutting saw.

The channel sampling quality was monitored by taking control channel samples immediately above and adjoining the existing channels. In total 16.8% of channel samples were "duplicated" with such control samples. Average of relative paired differences between twinned channel samples did not exceed 7% for copper and zinc. Higher average relative paired differences were reported for lead: 5% for concentrations <0.5% Pb; 17.6% for 0.5-1% Pb and 13.2% for concentrations >1% Pb. These variations indicate an erratic distribution of galena, although the overall impact on the reserves is not significant since these last two grade classes combined do not exceed 9% of the total number of channel samples.

7.3.3.2 Sample Preparation

Two different sample preparation schemes were used during the exploration period. Both schemes were based on the Richard-Czeczott formula Q=kda, where Q is the minimum sample weight at a given stage of volume reduction, d is the diameter of the largest fragments defined as the screen size that retains the largest 5% of the mass, k is a coefficient dependent on the distribution irregularity of the mineral of interest and is a coefficient related to the roundness of mineral grains (generally approximately 2).

The coefficient k is the key parameter. In general terms, the lower the coefficient k is, the better it accounts for the erratic distribution of minerals.

All samples from the Shubinskiy exploration programmes were prepared at k=0.16. Samples were jaw crushed twice to pass a 2mm sieve size, reduced, ground in a roll grinder, sieved through a 1mm screen, reduced again and pulverised to 0.074mm nominal particle size. Pulverised analytical samples weighed approximately 300-350g.

During the first phase of the preliminary exploration (1955-1961), samples were prepared at the sample preparation facility of the Karelinska geological exploration party. During the second drilling campaign (1974-1975), samples were prepared at the Leninogorsk GRE laboratory. During the detailed exploration phase, samples were prepared at the laboratory of the Leninogorsk Processing Plant Complex (Kazzinc's predecessor).

7.3.3.3 Analyses and QA/QC

Prior to 1976 analytical work was performed by the chemical laboratory of Leninogorsk GRE and the Central Laboratory of PGO Vostkazgeologia. Leninogorsk GRE was the principal laboratory and conducted semi-quantitative analyses for 16 elements on drill core and geochemical samples and chemical analyses for copper, lead and zinc. Fire assaying for gold and silver were carried out by Leninogorsk GRE (up to 1964) and then by Vozkazgeologia. Phase analyses were conducted by Voskazgeologia and Leninogorsk Polymetallic Complex.

VNIItsvetmet performed external control analyses on routine samples analysed by Leninogorsk GRE and Youzhkazgeologia completed control analyses on samples analysed by Vostkazgeologia.

The chemical laboratory of Leninogorsk Polymetallic Complex served as the principal laboratory during the detailed exploration stage. Samples were initially submitted for semi- quantitative multi-element spectral analysts to screen out the samples for accurate analyses. Samples with spectral copper, lead and zinc exceeding 0.2% were then sent for chemical analysis for copper, lead and zinc and fire assaying. External control analyses were done by Vostkazgeologia.

Durnev et al (1989) also reported that earlier quality control results (for the periods 1954-1960 and 1973-1975) were audited by GKZ USSR and since the audit did not reveal any problems the analytical work completed during those periods was considered acceptable.

> ***WAI Comment:*** *Drilling methods, sampling and, sample preparation appear to be satisfactory although the core recoveries and linear/weight discrepancies give cause for concern; based on historical reports the QA/QC (internal and external control analyses) did not appear to reveal any significant random or systematic error.*

7.4 Mining

7.4.1 Introduction

Underground production began at Shubinskiy with trial mining in 1990/1991 and continued intermittently at low output rates for over six years. In 1999 the production exceeded 100,000t and has since risen to reach its full capacity of 200,000tpa in 2009. During the period 1992-2009, the mine produced 1.6 Mt of ore grading 0.66g/t Au, 17.9g/t Ag, 1.74% Cu, 0.28% Pb and 2.25% Zn. The mine is operated by TOO Shubinskiy, who had worked as Kazzinc's mining contractors prior to April 2009, and operates for 305 days on a 3 six hour shift basis.

Ore is extracted from the stopes on the 1-5 levels, it is loaded in rail wagons and hauled by electric locomotives to the hoisting shaft. The rail wagons are hoisted to surface in the cage and taken to a discharge point adjacent to the shaft. The ore is hauled by road trucks to the Ridder processing plant, where it is stockpiled for separate treatment.

The Shubinskiy ore is processed approximately once a month or when the other concentrator circuits are shut down for a day or longer to allow for the separate treatment of copper-zinc ores. Zinc, copper and gravity concentrates are produced.

7.4.2 Mine Design

The current mine layout consists of five underground levels connected by two vertical shafts located on the north-western and south-eastern flanks of the deposit. 1 level is at an elevation of 980m (at 75-100m depth). Level 2 is developed 50m below 1 level and 60m above 3 Level. Levels 3, 4 and 5 are 60m apart.

The key infrastructure consists of two surface shafts and five production haulage levels, as shown in Figure 7.6. The ventilation shaft is located to the south east and the main Shubinskiy ore-waste hoist shaft to the north-west. The Shubinskiy shaft is 319m deep and is used for personnel, equipment and ore and waste hoisting. The Shubinskiy Shaft adit is located at approximately +1035m elevation with the resources extending to 12 level, approximately 700 metres below the Shubinskiy Shaft adit. The extraction sequence for each level commences in the centre of each lode and extends top-down and towards the flanks.

The mine has been designed to extract ore to 5 level using the existing infrastructure and mining method. The ore below 5 level is to be accessed via a decline. Ore will hauled from the lower levels using 20t capacity trucks and delivered to the ore pass system on 4 and 5 level.

Kazzinc has stated that the current ventilation infrastructure and pumping capacity from 5 level to surface will be adequate for production below 5 level using a diesel truck fleet as no significant development infrastructure upgrade required.

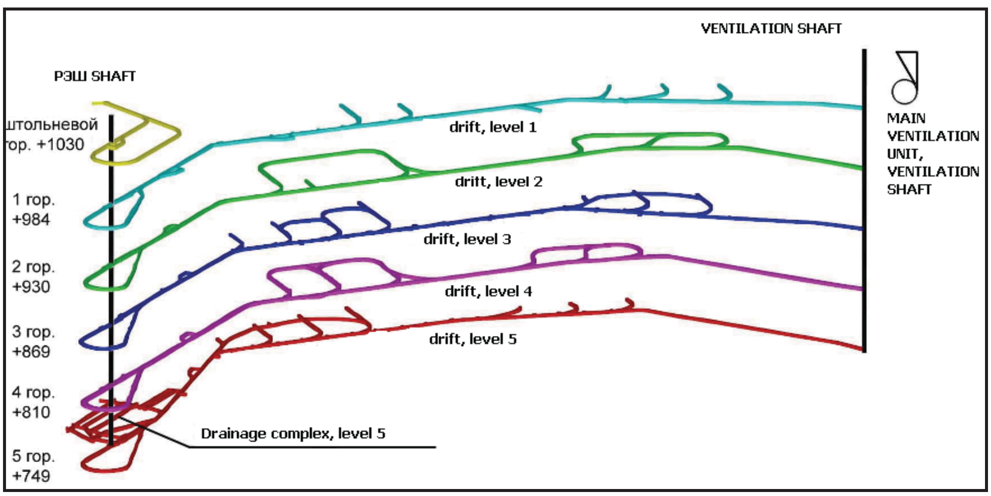


Figure 7.6: Schematic Underground Mine Layout

7.4.3 Mine Optimisation

The Shubinskiy mine uses a zinc equivalent (ZnEq) cut-off for the polymetallic orebodies. The ZnEq grade is calculated as Zn% + 0.4Pb% + 1.7Cu%.

Waste is termed as material with a grade less than 1.0% ZnEq, and although contains mineralisation grades less than 0.4% ZnEq cannot be treated by the current Kazzinc concentrator.

The cut-off grades are based on historic metal prices used for the 1989 TEO for the Shubinskiy project and are therefore unlikely to be optimal for 2010 costs, metal prices and process terms.

The mineralisation identified in the lower levels (south-eastern deposit) are insignificant and deemed to be uneconomical. Additional ore that is expected to extend the life of the mine will be sourced from the north-eastern lower levels of the Shubinskiy deposit.

7.4.4 Mining Reserve

The mining reserves currently used at the Shubinskiy mine site have been estimated using conventional Soviet methods. At present no Mineral Resources or Ore Reserves have been prepared for Shubinskiy in accordance with the guidelines of the JORC Code (2004).

7.4.5 Recovery and Losses

Kazzinc currently uses Chamber Stoping with Backfill as the mining method and proposes to use this same method for extracting the remaining mineable ore. The total primary and secondary losses are typically between 1% and 6%, with the overall average being 2.5%.

7.4.6 Production Schedule

During the period 1992-2009, the mine produced 1.6Mt of ore grading 0.66g/t Au, 17.9g/t Ag, 1.74% Cu, 0.28% Pb and 2.25% Zn as shown in Table 7.7. The Shubinskiy mine is expected to continue at a production rate of 190ktpa for another 12 years, finishing in 2022. The production for 2010 was planned to be 190kt at 0.28g/t Au, 11.02g/t Ag, 1.17% Cu, 0.16% Pb and 1.52% Zn.

In 2009, 7,810m^3 of development was achieved and 9,520m^3 is planned for 2010; in the stopes 20,704m of drilling was achieved in 2009 and 19,600m is expected in 2010.

Table 7.7: Mine Production						
Year	**Tonnes (kt)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
1992	8	0.88	25.00	2.50	0.13	3.75
1993						
1994	31	0.52	22.88	1.63	0.20	2.61
1995	68	0.83	15.18	0.93	0.23	0.92
1996	22	1.14	22.73	1.82	0.45	2.27
1997	2	0.77	30.68	3.50	0.38	3.16
1998	97	0.91	19.97	2.20	0.27	2.51
1999	89	0.81	28.42	2.62	0.44	3.83
2000	100	0.89	22.24	2.80	0.28	3.03
2001	103	0.75	21.47	2.75	0.28	3.47
2002	115	0.77	28.70	2.69	0.50	2.87
2003	94	0.64	18.94	2.13	0.30	3.20
2004	95	0.54	14.91	1.60	0.23	1.61
2005	94	0.53	16.64	1.48	0.29	2.00
2006	125	0.4	13.95	1.17	0.26	1.81
2007	184	0.46	13.97	0.93	0.24	1.63
2008	190	0.55	12.41	1.13	0.19	1.64
2009	200	0.73	14.17	1.41	0.24	1.77
Total	1,619	0.66	17.91	1.74	0.28	2.25

7.4.7 Geotechnical

The maximum mine output is expected to be 190ktpa, further exploration will permit a technical and economic evaluation for development and mining of the deposit below the 5 Level. The lower level material is not expected to increase the production rate but to extend the life of the mine.

The ores hardness coefficient ranges 10-13, host rock hardness coefficient ranges 8-12, internal wastes hardness coefficient ranges from 6-10. Therefore the stability rating of the ore and host rock is from 'average' to 'stable', as shown in Table 7.8. Localised areas of shale may be unstable, with a thickness from 3-5m, and are confined to the hanging wall.

The rock densities range from 2.5-2.8t/m^3, ore density depends on the degree of mineralisation and varies from 2.9-4.4t/m^3. The swell ratio is 1.48 for waste and 1.50 for ore.

The hydrogeological conditions of the deposit are characterized as ordinary. The predicted inflow of fracture water to 8 Level ranges from up 69-104m^3/h.

Table 7.8: Rock Stability Classification

The stability category	The stability degree	The names of ores and rocks	The solidity factor by M.M. Protodiyakonov	The duration of the ores stable condition in mine workings	The rocks corruption intensity	The mine workings maintenance conditions
II	Stable	Massive pyritic copper-zink ore Copper-pyritic ore Quartz albitophyres Quartzits	11-15 10-15 8-10 10	Up to 6 month	Sloughing occurs only at the particular areas The structural weakening factor 0.4-0.6	With the long duration condition demands reinforcing in particular places. Acceptable baring.
III	Moderately stable	Siliceous siltstones	5-8		Local and separate corruptions with depth up to 1 meter. The structural weakening factor 0/3-0/4	Demands reinforcing. Fastenings heaving off the boreholes is acceptable up to 100 m^2
IV	Unstable	Sericite-quartz, chlorite-sericite-quartz slates	3-5	Up to one day and night	The corruption spreads to the major portion of the workings contour and develops more than to 1 meter. The structuring weakening factor 1.2-0.3	The reinforcing is demanded along with the roadheading. Barings are acceptable not more than to 10m^2
V	Very unstable	Chlorite-sericite-quartz rocks and slates in the zones of fractures. Intensively dragfolded.	1-2	Corrupts following the roadheading	Significant rock masses begins to move. The structuring weakening factor 0.2	Advanced strengthening or strengthening close to the bottom hole is demanded. The roadheading without strengthening is excluded.

7.4.8 Mining Method

The mining method is chamber mining with backfill and is used down to 5 Level. It employs conventional handheld pneumatic machine drift development and stope production drilling, electric scraper mucking and electric locomotive haulage. Drifting is carried out with handheld portable rock drills using compressed air. In the stoping operations, the ore from the drifts and stopes is mucked using scraper winches.

The chamber mining (with backfill) method is also known as sublevel open stoping with classified cemented tailings backfill. The extraction sequence is top-down (under hand), working from the centre of each lode to the strike flanks. The sequencing of the stopes is dependent on extracting the primary stopes first and backfilling them prior to extracting the secondary stopes. All stopes are backfilled to avoid damage to stoping areas on adjacent working levels.

The stope dimensions are typically 60x15x20m with the average stope producing 25kt and extracted as a continuous panel sequence along strike. Given the equipment currently utilised the minimum mining width within scraper stopes is typically 1.2-1.8m. The level spacing is 60m with 20m sublevels.

The mining method below 5 Level is proposed to be mechanised with ramp development access to the levels and sublevels. It will employ mechanised development, semi-mechanised production drilling, diesel loaders and diesel trucks.

The rock handling management appears to be appropriate without material losses (spillage) or dilution. The waste produced from the development below 5 level will need to be management carefully to prevent cross-tramming issues.

> ***WAI Comment:*** *WAI believes that the mining method is technically sound and cost effective for the geometry of the orebody.*

7.4.8.1 Drilling and Blasting

Handheld pneumatic drills are used for drilling the mine development with blastholes usually 38mm in diameter. The main explosives are Granulite A6 primed with a small diameter (32mm) cartridge (Ammonal, ammonite, powergel) in dry holes. In wet holes, packaged explosives are used. Blasting operations are performed using nonelectric blasting agents.

Pneumatic handheld production drills capable of drilling 110mm and 130mm diameter holes up to 20m in length are used in the stopes. The explosives used in stoping are similar to development, except that larger diameter cartridges are loaded into the holes.

> ***WAI Comment:*** *Shubinskiy is susceptible to sulphide dust explosions due to the nature of the ore. Precautions taken during blasting, involve washing of walls and use of inert material in the charging column to retard the blast flame, all which is acceptable and standard practice for most mines with this type of ore.*

7.4.8.2 Loading and Hauling

Ore is mucked in the stopes along the production drive using wire rope scrapers. The ore is scraped to and drops into timber boxholes (ore chute) which load directly into rail wagons. The scraper winches are rated at 17kW, 30kW and 55kW and are capable of moving ore up a scraping distance to 60m.

Rail wagons are used to deliver ore from the stopes to the Shubinskiy hoist shaft. The wagons have a capacity of 4-7t, 7 wagons make up a train which is hauled to the shaft by the electric locomotives.

The loaded rail wagons are hoisted to the surface via the shaft cage (with typically 210 wagon loads per day each way). The cars are individually hauled to the Shubinskiy adit where they are tipped and the ore is discharged onto a run of mine stockpile.

From there, the empty rail car is returned underground. The ore on the surface stockpile is loaded and hauled to the Ridder process plant by road trucks.

7.4.8.3 Mining Fleet

Table 7.9 below summarises the current mining fleet at Shubinskiy.

Table 7.9: Mining Equipment Fleet	
Model	**Quantity, pcs.**
LPS-3U Drill Rig	6
PT-48A Handheld Drill Rig	8
PP-54B2 Handheld Drill Rig	11
ЗП-2 Charging Machine	2
БП Concrete Mixing Machine	1
СП-8 Manual Pneumatic Drill Rig	4
Ulba-150I Mixing and Pumping Machine	1
ShVA-710 Secondary Scraper Winch	4
ShVA-18000 Secondary Scraper Winch	5
PPN-1 Pneumatic Loader	1
PKU Universal Bucket Loader	1
Injectors	7
TORO -151 LHD	1
30LS 2CM Scraper Winch	4
55LS 2CM Scraper Winch	7
17LS 2CM Scraper Winch	4
K-10M Electric Locomotives	6
VG-2,2 Railway Car	55
UVB-1,7 Railway Car	15
Diamec 232 Exploration Drill Rig	4

7.4.8.4 Backfilling

All of the stopes at Shubinskiy mine are backfilled. Tailings from the Talovskoye tailings dam are used as a stope backfill material. The surface trucks delivering ore to the Ridder processing plant bring the tailings back to Shubinskiy mine site on the return journey. In 2009, the backfill plant sent 41,450m^3 underground and it has been scheduled to place 76,000m^3 of backfill in 2010.

The backfilling strategy is to fill the primary stopes with cemented backfill. The cement content ranges from 6.1-8.7% depending on the location of the fill. The higher percentage cemented fill is placed in the bottom of the stope, creating a 5m thick sill pillar, as a higher weight bearing capacity is required. As the primary stope is filled, the percentage of cement is reduced.

The cemented backfill is left for 9 months before the stopes on other levels or the secondary stopes located nearby can be extracted. The secondary stopes are also backfilled but the only cemented fill used is to create the 5m sill pillar at the base; the remainder of the stope is filled with hydraulic or uncemented backfill.

The normal blasting practice when removing ore adjacent to backfill is to drill the blast holes 1m short of the backfill contact. This will prevent blast damage to the backfill surface and keep grade dilution to a minimum.

7.4.9 Dewatering

During 1985-1989, the total water inflow rate was 50m^3/h and the maximum water inflow rate reached 77.5m^3/h. Approximately 7-24% of this water was used for processing. In 1999-2000 the total water inflow reached 31-46m^3/h and 60% of this water, approximately 18m^3/h, was used for process water.

The present pumping station is situated on 5 level adjacent to the mine shaft and is equipped with three pumps with the capacity to move water back to the surface. Mine water is collected in the water channel located in the main drive on all working levels. It flows towards the shaft and down to 5 level via drainage holes where it is collected in an underground sump with a capacity of 3,042m^3 and is then pumped to the surface treatment facilities.

The anticipated water inflow when the mine reaches 9 Level will be 69-142m^3/h and will need to be studied carefully to ensure that dewatering can occur efficiently. A dewatering complex is planned on 7 or 8 Level for pumping of mine water to the surface and capable of dealing with a maximum water inflow of 150m^3/h.

7.4.10 Future Development

It is intended that mining below 5 level will be via a decline which will be developed using electric over hydraulic jumbo drill rigs and the broken material will be mucked by diesel load haul dump (LHD) loaders. The material will be loaded into 20t capacity trucks and hauled back to 5 level and stockpiled on the level. This material is then loaded into the rail wagons, transported to the Shubinskiy shaft and hoisted to surface.

Kazzinc has stated that the current ventilation infrastructure from 5 level to surface and the pumping capacity will be adequate for production below 5 level using a rubber tyred mining fleet.

The rubber tyred mining production fleet indicated by Kazzinc will typically consist of:

- Twin boom electric-hydraulic jumbos development drills, excavating heading sizes typically 12m^2;
- Electric hydraulic production drills capable of drilling 102mm diameter holes to a length of at least 30m;
- Diesel LHDs with a bucket capacity of 3m^3 with remote control capability; and
- Diesel underground trucks with a capacity of 20t.

7.5 Infrastructure

7.5.1 Introduction

The Shubinskiy mine infrastructure consists of the mine offices, workshop, backfill plant, compressor house, warehouse and stores which are located adjacent to one another. Other facilities on the mine site include the surface ventilation, boiler house, and the ore handler system.

7.5.2 Offices

The mine offices are located adjacent to the Shubinskiy shaft. These buildings provide office space for the senior management, financial, human resources, mining, geology, surveying, I.T. and engineering.

7.5.3 Maintenance Workshop

The workshop is divided into several sections depending on the size of the vehicle/equipment to be serviced or repaired, from light vehicles and buses to heavy plant equipment.

7.5.4 Warehouse and Stores

The warehouse and stores area is occupied by the purchasing and stores departments where the management, issuing and distribution of materials and parts around the mine site take place.

7.5.5 Ore Loading Facility

Ore hoisted to the surface is dumped on a stockpile adjacent to the surface adit before being loaded onto road trucks and hauled to the Ridder processing plant.

7.5.6 Ventilation

The Shubinskiy ventilation is an exhaust system with the primary (VPC-16) fan mounted at the top of the ventilation shaft. The mine workings and primary ventilation complex were inspected in April 2000 by

Kazennoe enterprise "*Paramilitary mine-rescuing crew of the Leninogorsk*". The results were documented in the "Mine workings ventilation and primary ventilation complex functioning of the Shubinskiy deposit observation report", April 2000.

The results showed that:

- The fan productivity was 25.4m^3/s;
- The total fan pressure was 90.4kg/m^2; and
- The total resistance was 0.14.

The characteristics of the ventilation and mine network displayed in the report showed that the fan was operating inefficiently. This was because of the oxidation in the mine workings where more ventilation is required to flush the low oxygen zones. The actual efficiency of the fan suggests that the air flow is 24.9m^3/s with the pressure of 88.0kg/m^2.

During the cold season air is preheated by an air heater (surface area 68.01m^2) mounted jointly with a VC-11 fan which provides the preheated air to the adit through metal ducting.

When mine development is taking place, or the stoping blocks are being prepared, ventilation is reversed. The reversing of the air flow is performed by adjusting the blades in the fan which are controlled remotely.

The estimated air required for ventilation is 49.96m^3/s, which is currently above capacity and the necessary air will need to be replaced with either:

- the VPC-16 fan working simultaneously with the VM-12 fan; or
- two VM-12 fans working together.

The second option of two VM-12 fans working together is the preferred option as the expected airflow is 45.6m^3/s with the pressure of 291.1kg/m^2.

The development of the mine below 5 Level will require the installation of an underground VME-12 type air fan on 5 Level for the ventilation of the decline whilst being driven. An existing main fan unit (VOD – 21) with the capacity of 120m^3/s will provide the required fresh air volume, and its distribution in the ventilation system of the mine, when extraction of ore from the lower levels commences.

7.5.7 Backfill Plant

The backfill plant which is located on the mine surface includes:

- Cement silos;
- Tailing stockpile bunker;
- Tailings preparation; and
- Mixing facility.

The inert aggregate or tailings are delivered to the plant by road trucks from the tailing dam. The tailings are tipped into a receiving bunker and fed into the tailings preparation process by a winch and wire rope scraper.

From the receiving bunker, the tailings are dropped onto a conveyor through a grizzley to remove any mine rubbish. It then passes over a vibrating screen to remove the oversize material greater than 30mm. The oversize material is sent to a dump and the undersized aggregate is delivered by another conveyor into the mixer.

Cement is delivered to the backfill plant by bulk-cement transport units and unloaded into the two cement silos (60t each). After that the cement is delivered to an active cement storage silo with a capacity of 7t and into the cement batch type mixer with the tailings. Water is added and then the components agitated.

The finished backfill mixture is delivered to the well which is connected to 1 Level with a 108mm diameter pipe and then transported to the stopes by 159mm diameter pipelines. The pipeline is washed out with water from 5m^3 special tanks to ensure the pipeline is clear of blockages as soon as the batch of backfill has been placed.

> ***WAI Comment:*** *The condition of the walkways, ladder and access ways within the backfill plant facilities were not up to modern European standards.*

7.5.8 Power Supply

The current power supply to the Shubinskiy mine is from two 6kW power lines connected to the Talovka power substation (distance 8.5km). In order to maintain power supply to the Shubinskiy mine, at increased ore output, it will be necessary to overhaul the power lines by replacing the wood poles with ferroconcrete ones. In the longer term it may be necessary to install two more 6kW power lines to ensure supply to deeper levels in the mine.

Also it will be necessary to increase the cross section of the wires to 240mm^2 by using two AC-120 power lines which will prevent the danger of voltage drop on the lines.

The electricity receivers have a rated power capacity of 4,750kW (including 1,930kW by 0.4kW). The electricity consumption is 5,000 – 5,400MW/h/year. The increase in productivity will increase to 110kW at the mine, boiler to 120kW, the crushing to 450kW and 200kW for the other consumers (total 880kw), and the approximate rated power capacity will be 5,630kW (including 2,520KW by 0.4kW). The approximate annual power supply will be 6,800MWh/h. The average load current is expected to be 120-140A whilst the maximum load current is expected to 280A.

7.5.9 Water Supply

The water supply to the underground mining workings is delivered by a 100mm diameter pipeline, which runs down the Shubinskiy shaft and into hydro-reducers on each level.

> ***WAI Comment:*** *WAI assumes that the water is supplied from a fresh water dam located near to the site or a central dam for the Ridder mines.*

7.5.10 Air Supply

The compressor station consists of 3 compressors; 6VV-30/8, 305VP-30/8 and 2VM-10-63/9, with an efficiency per unit of 58m^3/min. A 250mm diameter compressed air pipeline runs down the Shubinskiy shaft, and divides into each level.

One of the compressors (2VM-10-63/9) needs a major overhaul to achieve its maximum efficiency and the out-of-date compressors (6VV-30/8 and 305VP-30/8) are to be replaced with two new (ZT 250-10-50) compressors that each have a capacity of 36m^3/min each.

The total efficiency of the compressor station unit must not be less than 135m^3/min.

7.5.11 Heat Supply

The JSC Energotsvetmet production boilers have the maximum efficiency of 1.3Gcal/h. In order to provide the underground workings with heated air at a flow rate of 22.8m^3/s, a heating load of 1.65 Gcal/hour is needed. This means that with the boiler temperature chart showing 95/70°C, the net water consumption is about 66t/h.

As the mine gets deeper it will be necessary to reduce the net water consumption and still provide the mine with heat energy. The temperature chart of 150/70°C will be required which will reduce the net water consumption to 21.5t/h.

The most appropriate boiler will be a KV-TC-4-150 with a maximum efficiency of 4Gcal/h, coal consumption of 2000t/year. As a result of installing a new heat system some of the heating network may need to be changed to ensure the efficient distribution of heat.

7.6 Environmental

7.6.1 Environmental & Social Setting and Context

7.6.1.1 Landscape, Topography

The Shubinskiy Mine area is a predominantly gentle undulating topography with hills varying from 650m to 1,000m in height, situated in the Lenningorskoye valley on the northern side of the Shubin and Martyn Klutch brook which converge to from the Fillipovka River in the Ulba valley. Both brooks pass in close proximity to the mine, flowing south. The Martyn Klutch brook is used by the mine for water discharge purposes.

The region is seismically active with earthquakes up to 4 on the Richter scale being reported in the Ridder area in the last 10 years.

The air quality in the Shubinskiy Mine area is considered to be good and there are no other industrial outlets in the locality. Soil horizons are expected to be thin and lacking in fertile topsoil – no information of soil characteristics has been provided. Flora is defined by several landscape zones, affected by site geography, altitude and terrain. There are numerous trees and grass species, in a mosaic habitat, with coniferous species predominantly found at higher altitudes and pine and mixed forest, including birch found in the lower foothills. There are flat areas of scrubby grassland, with species abundance being somewhat limited by climatic conditions. 94 bird species are reported in the area, the majority being non-migratory, and 90 animal species, including bear, mink, deer, wild cats and numerous rodent species. Two of the bird species are reported to be rare raptors (birds of prey).

7.6.1.2 Land Use and Land Cover

The deposit was first discovered in 1846, mined for some years and later abandoned. In 1954-1960 preliminary shallow exploration re-commenced with deeper exploration undertaken in 1973-1975 and mine development 1976-1989. At this time it was State owned and operated.

It is a polymetallic deposit primarily zinc with a secondary copper and some lesser minerals. Approximately 190ktpa of ore is produced. No processing of ore takes place at Shubinskiy Mine, it is all sent to Ridder, road transport provided by Kazzinc logistics.

The site has several occupied buildings mostly constructed in 1970s, these are discussed in section 7.7.1.5 below. A series of hardcore roads are linking the infrastructure at site.

7.6.1.3 Water Resources

Underground water from fractures, rock and ore contacts, veins and fracture veins enter the mine workings at horizons of 62m and 150-200m. Clay (25m thick) beneath the Martyn Klutch brook prevents in-flow of water. Water losses from the Martyn Kluch are not recorded to confirm this assumption. The cone of water depression is 1.1km^2.

A water abstraction point is located on the Martyn Klutch brook, upstream of the mine. These water supplies the domestic water for the site. All water used by the batching plant is sourced from the compressor house. Water balance calculations were presented for 2009 and 2010, these are updated annually.

Hydrogeological assessment of the mining operation was conducted as a special assessment during the reserve estimation and mine design phase. The hydrogeological processes are continually assessed by the Ridder mining complex geological department.

7.6.1.4 Communities and Livelihoods

The nearest habitation is 21km from site, with the main workforce living in Ridder.

7.6.1.5 Infrastructure & Communications

Neutralisation Plant

A neutralization process plant to treat mine water was constructed in c1970s and upgraded in 2006-2007 (expansion of settlers and reagent section, repair of the settlers and buildings).

The mine water treatment is undertaken in one stage only, however water is collected below ground in a single tank and pumped to surface at a rate of 900m^3/day. A single concrete tank with 3 wooden segregation filtration boards installed is on a slight gradient so the water flows very gently through; this is a continuous process, with the retention time being approximately 5 days. The filtration boards are cleaned once a year, the waste sludge from the process is taken to the dumps, or used to fill the subsidence area, or to backfill the mine.

Following treatment the water flows by gravity to the Martyn Klutch brook for discharge. Monitoring is carried out on the brook both upstream and downstream of the discharge points and quality control results were presented to WAI (September and October 2010) respectively and all results were found to be acceptable.

Concrete corrosion assessment is performed annually, and repairs are planned and carried out based on the corrosion monitoring results.

Batching Plant

Quality Control testing of the mixture is undertaken every 1000m3 of backfill. Testing of the backfill cement after deposition is undertaken at 7 and 28 days using core sampling and sent to the Geotechnical laboratory at Ridder. Following the initial testing, inclined drilling of the chamber is undertaken after 3 months using a radial configuration; these cores are recovered and tested every metre. If settlement has occurred a further injection of backfill cement is carried out to ensure the chamber is completely full.

Ventilation Shafts

Emissions control is performed by the State Mine Technical Department once per quarter.

Pump House

A pump house located at the north of the site, is used for water extraction from the Martyn Klutch brook, and is in the process of being replaced with another building and pump installation.

Surface Water Channel

A surface run-off channel system has been constructed across the site. Debris is cleaned from the channel and concrete damage is repaired as routine maintenance, and it was evident that this is done. The water channel goes to the neutralization plant for treatment before being discharged into the Martyn Klutch brook. Monitoring of the brook in channels is undertaken at 2 intermediate points and at this outlet point.

7.6.2 Project Status, Activities, Effects, Releases & Controls

7.6.2.1 Project Description & Activities

Potential soil contamination could arise via runoff from temporary stockpile and dumps, deposition of contaminated dusts, oil/grease spills, and emergency situations such as chemical spill or containment breach.

Although there is no liability by Kazzinc with regard to the waste dumps on site, leached water from these may enter the water course and cause contamination which may impact on the general environment of the site. In 1995 Russian Institute GidroTsvetMet studied leaching of contaminants with drainage waters coming from the mine stockpiles. Kazzinc's response was to cover the stockpile owned by the State with a water resistant film to protect it from atmospheric precipitation.

In 2004, when the mine was transferred to Kazzinc (as Shubinskiy Ltd), in order to assess the mine's environmental condition an audit was carried out in co-operation with the State environmental authorities and in accordance with national legislation. The degree of site contamination, and division of responsibility between Kazzinc and the State, was documented by the relevant Certificate.

7.6.2.2 Land Ownership and Tenure

Kazzinc holds the tenure rights to mine polymetallic ores from Shubinskiy deposit, held in the name of TOO Shubinskiy.

The Company is required to have environmental insurance, to cover environmental damage up to a maximum value of 22,032,000 Tenge (US$146,880).

> ***WAI Comment:*** *WAI has viewed the mine licence and permit and considers that these have been obtained in line with State requirements. It is understood that the future expansion of mining activity would not necessitate relocation of communities within the current licence boundary. The nearest residential dwelling to the mine is 21km. The environmental insurance coverage is assessed on a site specific basis to ensure that liabilities have been identified and characterised and that adequate insurance cover is in place.*

7.6.2.3 Mine Wastes – Rock

A historic waste rock dump is located on the site, this is believed to date back to the 1970's and as such is not the responsibility of Kazzinc. Adjacent to this is a concrete-base temporary storage house for ore rock, awaiting transportation to Ridder.

A surface water run-off channel surrounds both the waste dump and the ore stockpile; this channel is constructed of concrete and installed to a depth of 0.3m ground surface. This channel forms part of a larger surface water collection system on site.

> ***WAI Comment:*** *Based on the audit report performed at acquisition, and subsequent studies, the Company decided to cover the dump with an HDPE liner. The dump appears to be well contained although the liner had slipped downwards. The Company considers that environmental liabilities have been adequately identified, and the chemical composition, Acid Rock Drainage (ARD) and leaching potential of this material was characterised.*

Run-off from the ore stockpile is collected within a concrete vessel and flows via the surface water channel to the neutralisation plant. Water from the channel is tested at 2 intermediate points and following treatment discharged into the Martyn Klutch brook.

7.6.2.4 Mine Wastes

Mine wastes are classified and stockpiled on assigned areas for disposal, or used for backfilling as required.

7.6.2.5 Water Management & Effluents

There are two discharge points to the Martyn Klutch brook from within the mining licence area. Discharges to surface waters are permitted in accordance with national legislative requirements.

Water is pumped from a point source close to the Martyn Klutch brook, permit No 03-YK-136/08 issued 19 December 2009 valid until 31 December 2012, and issued by the Water Resource committee.

7.6.2.6 Emissions to Air

Air quality is monitored at ventiliation shafts and the batching plant which are the main sources of emissions.

7.6.2.7 Waste Management – General

A subsidence zone has developed in the northern part of the site, and has been utilised as a landfill for general site waste.

Sewage and domestic waste water is collected on site and stored in two septic tanks, the tanks are serviced/emptied by an outside contractor twice yearly.

Waste oil and greases are taken to Ridder testing laboratory for analysis. Each batch is tested twice and on these results the oils are either sent for recycling or disposal at a State landfill. Oily and greasy rags are collected on designated reciprocals and burnt on site in the landfill area and contaminated overalls are also either burnt or sent to a commercial laundry for washing. Waste tyres are sold to a third party for recycling and scrap metal from the vehicle workshops and mine is collected and stored close to the vehicle maintenance area and taken off site by a third party contractor.

> ***WAI Comment:*** *There is a reasonably efficient waste management system in place, whereby waste is sorted for either recycling or disposal in the subsidence zone.*

7.6.2.8 Hazardous Materials Storage & Handling

Explosives are delivered by road to an explosives magazine at a remote part of the site. From here the explosives required for underground use are transported in sealed containers and lowered in by shaft to the underground storage area prior to use.

Main contracts specify the lines of responsibility, and an additional contract 'Safety Ageement' is awarded to specify environmental liabilities during the contractual work.

It was noted that spillage/cleanup protocols and spill kits were available in the vehicle maintenance plant areas for oil spills. In addition, it was demonstrated and checked that the spill kits were adequately stocked and that staff were fully conversant with the procedures for spills.

> ***WAI Comment:*** *WAI considers that in general, storage areas are appropriate and well maintained, although some of the buildings do require some upgrading maintenance.*

7.6.2.9 General Housekeeping

Generally the housekeeping of Shubinskiy Mine is very good, with most open areas and operational buildings being maintained in a tidy and orderly manner.

7.6.2.10 Fire Safety

A fire protection plan has been developed for Shubinskiy Mine; this would appear to be very comprehensive for both above and below ground facilities.

All staff and contractors are given training in fire safety procedures, advised of emergency evacuation routes and fire fighting protocols, as part of the mine site induction programme. A designated person in each area is responsible for inspection and maintenance of fire extinguishers. Fire drills are scheduled to be undertaken twice a year. The State mine rescue team, located in Ridder is responsible for underground fire management.

> ***WAI Comment:*** *The fire management systems appear appropriate to the size of the operations, and WAI considers that these issues are being well managed by the health and safety team. WAI understands that Kazzinc intends to introduce more drill training systems, and recommends that this should be implemented as a priority.*

7.6.2.11 Security

Security of the Shubinskiy mine is provided by Group 4 Security (G4S), a leading international firm, with much experience with mine sites.

7.6.3 Permitting

7.6.3.1 OVOS

An OVOS has been produced for Shubinskiy Mine, endorsed by an "expert" opinion of the State approved with the State Environmental Inspection. In 2007 Shubinskiy Ltd. OVOS (ESIA) was adjusted and approved with the State Environmental Inspection.

7.6.3.2 Environmental Permits

The environmental emissions permit, prepared by Kazzinc and agreed with the Ministry of Environmental Conservation (RK), has been granted to the Shubinskiy Mine. This permit regulates water and air emissions at Shubinskiy and regulates the volume, location and toxic elements of such emissions. The permit is renewable every year (permit No 0054698) and the requirements for the whole complex in 2010 are as follows:

- Emmissions to air 6,381317t
- Discharge to water 177,869t

7.6.4 Environmental Management

Details of Kazzinc's environmental policy and company approach and environmental management staff & resources are specified elsewhere. There are no specific additional initiatives implemented for Shubinskiy Mine separate to the company plan.

7.6.4.1 Systems and Work Procedures

The Environmental Monitoring Programme has been agreed and endorsed by the Ministry of Environmental Protection Republic of Kazakhstan (MoE RK) and is renewable each year.

The monitoring plan includes information on location of sampling, frequency and determinants' for groundwater, surface water, discharge points, snow, air, dust and soils.

7.6.4.2 Environmental Monitoring, Compliance & Reporting

Surface water is sampled at two locations upstream/downstream from the Martyn Klutch brook and 5km downstream the mine. Samples are collected monthly. Maximum Allowable Discharge from the mine site is set out in accordance with the State Law and specified in the environmental permit.

Two monitored discharge points are sampled monthly for pH, copper, iron, lead, zinc, cadmium, manganese, ammonium salts, nitrate, and nitrite. At the neutralisation treatment facility, the pH of the neutralised minewater and metals is monitored on a daily basis.

Air is monitored at 3 locations at the batching plant, for flow, temperature on exit, speed, volume, and particles using the fabric from the filter (this is reportedly 97% efficient). Air is also monitored in the shafts.

A working level of 80dB is permitted, and where this is exceeded, Personal Protective Equipment (PPE) is provided, or mechanical improvements are made.

No soils testing are currently undertaken at site, since according to the monitoring program approved with the State environmental authorities such tests are not required.

Radioactivity is not monitored at site since no excess background radiation levels or potential radiation sources (equipment, materials, etc.) were found at the time of OVOS development.

Water monitoring reports are prepared each month and submitted on a quarterly basis to the Ministry of Environmental Protection with a full report annually. Furthermore, annual State environmental inspections are conducted to assess compliance and environmental management practices. Compliance at Shubinskiy is reported to be acceptable.

> ***WAI Comment:*** *The scope of the environmental monitoring programme has been developed in line with national legislative requirements. The Kazzinc Environmental Managers, Laboratories and third party specialists are aware of compliance requirements and analytical and sampling methodologies required to assess these. WAI therefore considers that the monitoring programmes are fully compliant with national standards. However WAI considers there is an insufficient number of groundwater sampling points.*
>
> *WAI appreciates that the history of mining in the area has already resulted in previously disturbed and depleted soil formations, but consideration should be given to developing a monitoring plan. The mine closure plan will contain an assessment of soil contamination levels, which will inform technical decisions for closure.*

7.6.4.3 Emergency Preparedness & Response

A total of 15 Emergency Preparedness and Response Plans covering various possible scenarios has been developed. These outline responsibilities, actions and reporting requirements should an incident occur. However, these plans have not been tested via any actual drills.

> ***WAI Comment:*** *The plans are comprehensive, and appear adequate, but testing via drills should be implemented.*

7.6.4.4 Training

There are no specific additional initiatives implemented by Shubinskiy Mine separate to Kazzinc company plan.

7.6.5 Social and Community Management

There are no specific additional initiatives implemented by Shubinskiy Mine separate to Kazzinc's company plan.

7.6.6 Health & Safety

7.6.6.1 Health & Safety Management Arrangements

Shubinskiy Mine has a H&S Manager who is responsible for the H&S of both above and below ground operations. A total of 34 ''Safe Work Procedures' are in existence covering each aspect of work undertaken at the mine. All safe work procedures are on a 3 year rolling programme for review and updating by the H&S Manager or designated H&S team member. The H&S Manager has a team of 3 inspectors per shift which cover both above and underground works. The Company is also accredited for H&S management under the OHSAS 18001 scheme.

Visitors to the site are given an H&S induction, and provided with appropriate Personal Protective Equipment (PPE). Prior to anyone commencing work at the mine (employee or subcontractor) H&S training and an H&S induction pack is given. After induction, further training relating to specific working areas is also given, and refresher training in undertaken on a regular basis.

The environmental training covers issues such as dealing with spills, waste management, water and power consumption and how to manage and minimise environmental risks. Managers also receive training every 3 years from State authorities.

The Company also has a special clinic in Ridder where all employees receive annual medical checks. The Company has a contract with the State ambulance provider and the nearest hospital with an Accident and Emergency (A&E) department is in Ridder. Spot tests for drugs and alcohol are also performed.

> ***WAI Comment:*** *WAI considers that health and safety is very well managed at the site, with appropriate training being provided, and good response systems in place for personnel. PPE was worn in all areas to some degree, however it was noted that some staff did not have hi-vis PPE on or hard hats.*
>
> *In general the H&S managers and their teams are proactive in their management of H&S issues, both with regard to national and international requirements. WAI would recommend implementing drills to test Emergency Response Plans, to assess their adequacy.*

7.6.6.2 Performance and Accident Records

Shubinskiy Mine generally has a good safety health and safety record with no significant incidents occurring in the last few years. Accident logs are maintained, and in the event of an accident a full inspection and investigation, with remedial action, takes place. Daily informal inspections are performed, and there are also annual inspections from various State departments including: Sanitary Epidemiological, Police, Safety and Emergency Response units. Formal internal inspections are performed three times per year.

> ***WAI Comment:*** *WAI considers that appropriate record keeping and improvement procedures are in place to deal with incidents. Overall the safety records are considered good, and all personnel seem committed to improving safety management.*

7.6.7 Mine Closure & Rehabilitation

Shubinskiy Mine initial estimate of potential closures costs was made as 131,000,000 Tenge, and this sum of money was deposited in a protected Company fund. Typical closure measures include demolition of surface structures, stripping of utilities, and stabilisation of mining related features, earthworks and re-vegetation.

It is reported that standard post-closure monitoring includes: rivers and groundwater, minewater, dust, soils, flora and fauna. It is also reported that post-closure aftercare is likely to include the continuation of minewater treatment and pumping. At Shubinskiy Mine this is envisaged for 10 years before the workings are allowed to flood and groundwater rebound occurs. It is reported that the closure fund will periodically be updated during the mine life, with a final detailed closure plan to be prepared 2 years before actual closure, in line with State requirements.

> ***WAI Comment:*** *Pursuant to subsoil use & mining Contract a closure plan will be undertaken in 2012. It is important to ensure that the Mine Closure and Rehabilitation Plan (MCRP) includes all environmental and social aspects, including post closure monitoring, and that the supporting closure fund is adequate to provision for this, including in the event of unplanned closure.*
>
> *The closure fund should be based on site specific closure requirements and should include provision for post closure monitoring and treatment. The proposed figure should be reassessed once all site liabilities and ongoing treatment requirements have been fully investigated.*

7.6.8 Conclusions

7.6.8.1 Environmental and Social Liabilities & Risks

Potential soil contamination could arise via runoff from temporary stockpiles and dumps, deposition of contaminated dusts, oil/grease spills, and emergency situations, such as a chemical spill or containment breach.

Kazzinc has acquired a number of historic mining features at Shubinskiy mine for which the liability remains with the State. When acquiring the deposit, the representatives of the State authorities and Kazzinc drew up the Mine Environmental Condition Certificate. It is recommended that Kazzinc undertake a detailed baseline assessment of current environmental liabilities and prepare measurable benchmarks during the mine closure plan development.

Historical voids have been recorded by underground surveys and measures have been taken to prevent collapse due to lack of backfilling. In the event of claims with regard to subsidence, attributable to historic workings, the liabilities are stipulated by national legislation.

7.6.8.2 Compliance with Local and International Standards and Expectations

WAI considers that Company environmental management and monitoring is performed in line with national requirements, and Kazzinc has achieved accreditation under international environmental, quality and health and safety management systems. Development of the systems is annually audited by international certification authorities and a sustainable improvement is observed.
The environmental expenditure budget is linked to corporate priorities and should include appropriate levels of contingency.

Health and safety is generally well managed, and site security is good, but does require a more proactive approach for visitors to site.

The existing closure plan and budget estimate is not considered sufficient to cover current international requirements or post closure monitoring and treatment obligations. The need for and duration of post closure minewater treatment and rehabilitation of State owned facilities should be clarified.

7.6.8.3 Recommendations for ESAP

In general all the areas visited at Shubinskiy Mine were well managed and appeared to operate efficiently. Environmental management is good with monitoring being undertaken regularly which appears to be appropriate to control migration of contaminates. However, specific recommendations include:

- Batching Plant: Develop a programme of dust monitoring;
- Waste Disposal and Management: Investigate run-off from mine roads constructed with waste rock; and
- Subsidence Zone: Develop a structured approach to landfilling.

8 DOLINNOE AND OBRUCHEVSKOE DEPOSITS

8.1 Introduction

Development of the Dolinnoe and Obruchevskoe deposits, situated immediately east of Ridder, is planned. With depleting reserves at Ridder-Sokolniy (15 years production remaining), Tishinskiy (9 years) and the re-processing of low grade tailing dumps; it is evident that increasing the reserve base with primary ore is essential. The former deposit consists of two lodes referred to as 'North-Eastern' and 'South-Western' and the latter comprises 'Northern' and 'Southern' lodes and a 'Central Prospect' interpreted from relatively widely spaced drilling.

8.1.1 Location & Access

The Dolinnoe and Obruchevskoe deposits, included administratively within the city of Ridder, are located 7.5 and 11km respectively from Ridder (nearest railhead), 2.5km from the outer limit of the town, and are readily accessible from a graded gravel road that links Ridder with Biysk in the Russian Federation. The Dolinnoe deposit occurs 4550m below the Bystrushinskoe reservoir, see Photo 8.1.

Photo 8.1: Dolinnoe and Obruchevskoe – Location Adjacent to Ridder

8.1.2 Topography and Climate

The Dolinnoe deposit is situated at depths between 450-650m below the Bystrukha River valley, which flows into the Bystrushinskoe reservoir, and a significant part of the area is marshy. Elevations vary between 840-860m, increasing to 910m in the northwest.

The Obruchevskoe deposit is situated at a depth of 800-1,000m on a gentle northern slope at the foot of the Ivanovskiy Ridge, on the southern side of the Bystrukha river.

Ground elevations above the Baltic Sea datum range from about 930m on the northern side to 1,100-1,150m on the southern side. The terrain is dissected by shallow boggy valleys of streams flowing into the Bystruha river. Surface formations comprise moraine, alluvial and deluvial deposits ranging in thickness from 120m to 475m. The area is overgrown with deciduous forest predominated by birch trees

Seismicity of the region reaches 7 on the Richter scale

8.1.3 Infrastructure

The deposits are located in the immediate hinterland of Ridder and have access to all the necessary services and utilities.

8.1.4 Mineral Rights and Permitting

Kazzinc acquired the right to explore and mine the Dolinnoe and Obruchevskoe deposits by tender in April 2004. A geological exploration licence comprising two separate areas covering the Dolinnoe and Obruchevskoe deposits was issued to Kazzinc by the Ministry of Energy and Mineral Reserves in July 2004.

The contract was issued on 20 August 2007 (Contract No 2450), and registered on 24 August 2007, and allows Kazzinc to explore for, and mine, zinc, lead, copper, gold and silver from the Dolinnoe and Obruchevskoe deposits. It is valid for 19 years, including six years of exploration. The exploration period may be extended for a further two years if required.

An integral part of the Contract is the Work Programme, which was submitted by Kazzinc and accepted by the Ministry of Energy and Mineral Reserves after an expert review. The Work Programme includes a detailed exploration budget of US$12,392M which forms the contractual commitment for exploration of the Dolinnoe and Obruchevskoe deposits.

A geological exploration licence over the Dolinnoe deposit covers an area of 3.3km^2 with the boundary defined by eight corner points as detailed in Table 8.1.

Table 8.1: Dolinnoe Geological Licence Boundaries

Boundary Points	Geographical Coordinates		Local Coordinates *[1] VKTGU 3 degree zone	
	Latitude N	Longitude E	X	Y
1	50°40'49"	83°34'00"	67,516	77,928
2	50°20'22"	83°34'56"	66,856	78,930
3	50°20'22"	83°36'02"	66,856	80,192
4	50°19'34"	83°36'09"	65,412	80,328
5	50°19'05"	83°35'57"	64,529	80,092
6	50°19'26"	83°34'50"	65,114	78,790
7	50°20'20"	83°34'50"	66,824	78,786
8	50°20'46"	83°33'52"	67,486	77,832

A geological lease over the Obruchevskoe deposit covers an area of 1.61km^2 with the boundary defined by four corner points as detailed in Table 8.2.

Table 8.2: Obruchevskoe - Geological Lease Boundaries

Boundary Points	Co-ordinates	
	Latitude N	Longitude E
1	50°18'36"	83°36'33"
2	50°19'20"	83°36'33"
3	50°19'20"	83°37'33"
4	50°18'36"	83°37'33"

8.2 Geology and Mineralisation

8.2.1 Regional Geology

The Dolinnoe and Obruchevskoe deposits are situated in the centre and deepest south-eastern portions respectively of the Ridder mining district, in the Rudnyi Altay geotectonic block (Figure 8.1), and the unusual grabber structure (see section 5.2.1).

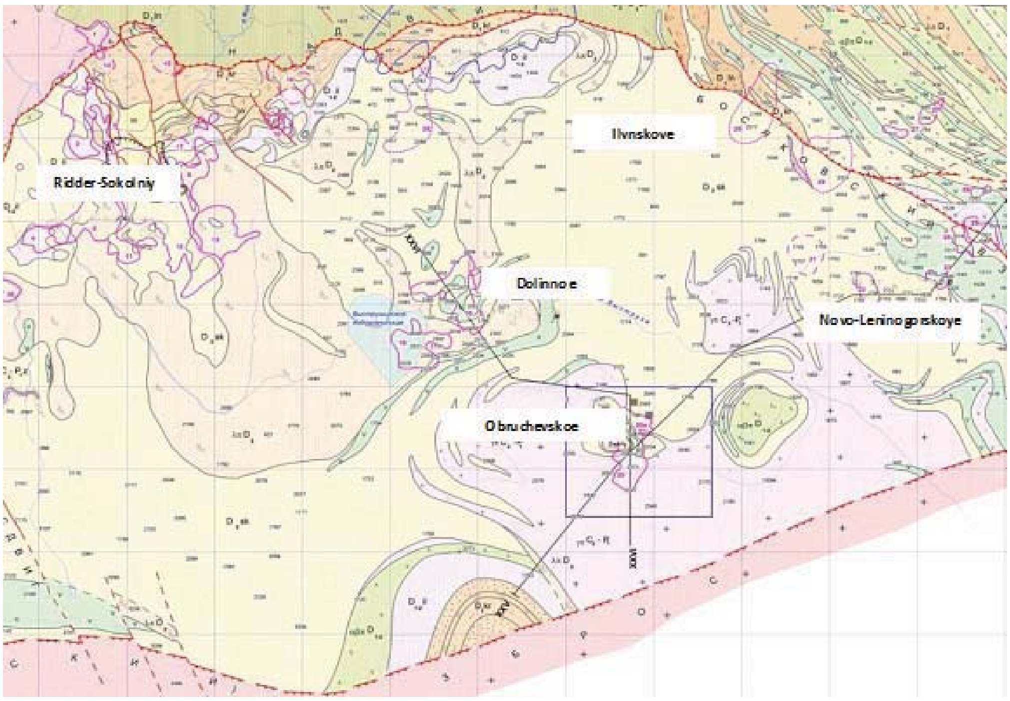


Figure 8.1: Dolinnoe and Obruchevskoe-Geological Setting and Section Lines XXVI and XXV
(Legend Refer to Figure 8.7)
(Scale Each Grid Division = 1000m; True North - Vertical Grid Lines)

The graben is filled with four volcano-sedimentary formations comprising, in ascending order, Zavodskaya (S2-D1zv), Leninogorskaya (D1ln), Kryukovskaya (D1kr), Ilibnskaya (D1-2il) and Sokolnaya (D2sk) formations (refer to 5.2.1).

The Kryukovskaya Formation hosts most of the polymetallic mineralisation known within the confines of the graben structure. It consists of two horizons of predominantly fine¬grained sedimentary rocks (siltstones of highly variable composition) separated by felsic volcanogenic rocks (agglomerates, tuffs and their reworked derivatives), which attain 350m in thickness at the Ridder-Sokolniy deposit and thin out rapidly in the south¬western, southern and south-eastern flanks of the Ridder-Sokolniy deposit. Calcareous siltstones at the top of the Kryukovskaya Formation, with their distinctive green-grey or ash-grey colour, serve as the marker horizon throughout the Ridder graben. Isolated reef limestones are also locally known at the same level.

The gold-silver-copper-led-zinc mineralisation occurs at four stratigraphic horizons, the most important horizons are:

- Stratiform, predominantly polymetallic mineralisation with a high content of gold and silver, hosted by a microquartzite-sericite quartzite horizon (called the Critical Horizon) in the upper part of the Kryukovskaya Formation (Middle Devonian), and underlain by subvertical stockwork-vein feeder channels; and
- Stratigraphically deeper, predominantly copper and copper-zinc mineralisation, at the interface of the Kryukovskaya Formation and the Leninogorskaya Formation, also underlain by subvertical stockworks and veins.

8.2.2 Deposit Geology

8.2.2.1 Stratigraphy and Lithology

The stratigraphy of the Leninogorskoye ore field includes metamorphic slates of early Palaeozoic, igneous-sedimentary formations of Lower and Middle Devonian and Quaternary deposits.

*Zavodskaya Suite (**S_2D1zv**)*: comprises strongly metamorphosed rocks of greenstone stage (greenschist facies), derived from siltstones, silty sandstones and sandstones, not intersected at Obruchevskoe, as a consequence of the considerable depth of the ore deposit structures. The suite is unconformably overlain by the following.

*Leninogorskaya Suite (**D_1ln**)*: the Lower Devonian igneous-sedimentary formations of the suite consists of molasse with irregularly alternating igneous-sedimentary formations of typically tuff, tuffites, tuffaceous volcanomictous sandstones, siltstones, silty sandstones, coarse sandstones and gravelstones; the basal horizon has rare lenses of basal conglomerates and gravelstones including fragments of metamorphic slates and angular rose quartz. Contact with the overlying Kryukovskaya suite is diffuse, the number of fragments gradually decreases and greenish grey tuffaceous siliceous cement is gradational into dark grey aleuropelites or grey/light grey sericitised micro-quartzites.

*Kryukovskaya suite (**D_1kr**)*: represents a sub-marine succession, characterised by a wide variety of facies with complex transition zones and considerable variation in thickness, intersected over their entire thickness at both Dolinnoe and Obruchevskoe. At the former the suite is 500-540m thick of which effusive pyroclastic formations (breccias of andesitic porphyrites and porphyry lavas) account for 170-230m and at the latter the thickness is 300-400m, represented by sedimentary formations.

Lithologies include calciferous, siliceous, carbonaceous/siliceous, carbonaceous/argillaceous siltstones with intercalations of sandstones and gravelites, with horizons of sub-rounded porphyrite breccias (Dolinnoe).

The mineralised zone is 70-100m thick and is characterised by a significant development of metasomatically altered lithologies *viz*. as micro-quartzites, siliceous and carbonate/sericitic rocks, sericitic and quartziferous rocks, sericitolites and chloritolites.

An upper light grey siltstone (0-103m thick) in the upper part of the suite has been identified in the hanging wall of the ore horizons (hanging wall schists) and acts as a consistent marker horizon. The Kryukovskaya suite is locally unconformably overlain by the *Ilyinskaya suite* (D1-2il); represented by mottled greenish, reddish, greyish green, sub-aerial and shallow marine sedimentry-volcanogenic rocks, the basal horizon consisting of tuffaceous sedimentary fragmental rocks viz. tuffaceous gritstones, grit stones with interlayers of tuffaceous sandstones and silty sandstones.

The suite is formed by irregularly interstratified tuffs of felsic to intermediate and mafic composition, tuffites, tuff gritstones and tuff sandstones, siliceous hydro-micaceous rocks, sandstones, grit stones, and calciferous silty sandstones with rare fossil horizons. Andesitic porphyrite bodies of sub-volcanic facies occur in abundance; the thickness of the suite is not regular and varies from 80-170to 330-530m. At Obruchevskoe the suite is fragmentary occurring as outliers in a thick and complex deposit of andesitic porphyrites and quartz porphyry, 450-500m thick including the magmatic rocks.

The *Sokolnaya suite (**D_2sk**)* completes the Devonian succession and is composed of dark grey, almost black siltstones and aleuropelites, argillites with minor interlayers and quartzo-feldspathic sandstones; intermittently, in the lower part of the suite there is an increase in the calcareous component of the sedimentary formations; thicknesses at Dolinnoe are 250-300m and at Obruchevskoe 200-250m.

Quarternary sediments almost everywhere overlie the Paleozoic rocks the thickness changing at Dolinnoe from a few metres near the north-east wall of the Bystrukhinskoye storage lake to 70-135m in the Bystrukha River valley. Quaternary sediments at the Obruchevskoe deposit vary from 90-150m in the north to 350-450m in the south.

8.2.2.2 Igneous Rocks

Igneous rocks are ubiquitous at both deposits and account for 25-30% of the explored deposit sections; at the Obruchevskoe deposit they amount to >50% where Lower Devonian extrusive, subvolcanic rhyolitic and dacite rhyolitic porphyries occur, mainly in the Kryukovskaya suite, forming a sub-concordant sheetlike body with thickness of up to 150-250m and almost over the whole of the Dolinnoe deposit, excluding its southern edge.

Lower Devonian extrusive subvolcanic andesite, andesitic and basalt porphyrites and their breccias are predominantly confined to sediments of the Ilyinskaya suite and are much rarer in sediments of the Kryukovskaya suite. Porphyrites form sheet bodies with dimensions from tens to hundreds of metres and with both sub-concordant and discordant contacts. The feeder zones (roots) of the intrusive bodies are of a discordant character.

Middle Devonian extrusive subvolcanic rhyolitic and dacite rhyolitic (quartziferous albito-phyres) porphyries are restricted to the base of terrigenous sediments of the Sokolskaya suite. They form sub-concordant sheet bodies, which are continuous on strike, and have thicknesses from 200-280m to >400m. The epicentre of extrusive rocks of the complex is on the Obruchevskoe deposit, where the feeder zone occurs in the upper horizons of the Kryukovskaya suite and causes shearing in the ore zone.

The late Devonian – Lower Carboniferous complex of gabbroic intrusions is represented by diabase dykes striking north-east (35-70°) with steep to vertical dips, from 0.5 to 50m thick. Dykes, the longest of which can be traced for more than 2km, intrude almost all Devonian rocks and ores zones and are frequently displaced en echelon.

Upper Carboniferous – Lower Permian plagiogranite porphyries, granite-porphyries and granites are confined to the formations of the Sokolnaya suite and form sheets of varying thickness up to 250-400m.

8.2.2.3 Structure

The Dolinnoe and Obruchevskoe deposits are located in the Leninogorskaya graben/syncline, which is constrained to the north by an overthrust, to the south by the Obruchevskoe upthrust fault and to the east and west by the marginal Uspensko-Karelinskaya and Kedrovsko-Butachikhinskaya shear zones.

In the graben/syncline mineralised deposits with a shallow dip, plunge at 7-10° to the south, south-east and south-west from the Ridder-Sokolniy ore deposit in conjunction with the trace of the upper contacts of the Kryukovskaya and Zavodskaya suites. Gently dipping fold structures (striking north-east) can be identified viz. an anticline to the north with the associated deposits of Ridder-Sokolniy and Bakhrushinsk, and a southern plunge associated with the Novo-Leninogorskoye and Obruchevskoe deposits separated by the Central syncline.

Complex fold structures are evident to the south on Dolinnoe superimposed on the shallow dipping limb of the Ridder-Sokolnaya dome. The fold axes which also plunge southwards, strike to the north-northwest.

In the central area of Dolinnoe synclinal folds with an amplitude of 400-600m are evident and to the north-east, and south-west anticlinal structures are developed complicated by discontinuous curved and longitudinal faults with downthrows to the south west of 95-160m and up to 130m respectively. The longitudinal faults also exhibit lateral movement of up to 95m.

The Obruchevskoe deposit is located on the plunging southern limb of the southern anticline where low angle thrusts with a north western strike occur. Faulting is not evident in the ore zones of the Obruchevskoe deposit.

Frequent interformational erosional and crush zones and flexure-like structures are evident in the higher grade gold-polymetallic and polymetallic ore zones which are sub-concordant with the enclosing rocks.

8.2.3 Morphology and Mode of Occurrence of Lodes

8.2.3.1 Dolinnoe

The two lodes comprising the Dolinnoe deposit are centred on anticlinal structures with gentle to moderate dips in all directions (domes) separated by a shallow trough. The distance between the lodes is approximately 900m (Figure 8.2).

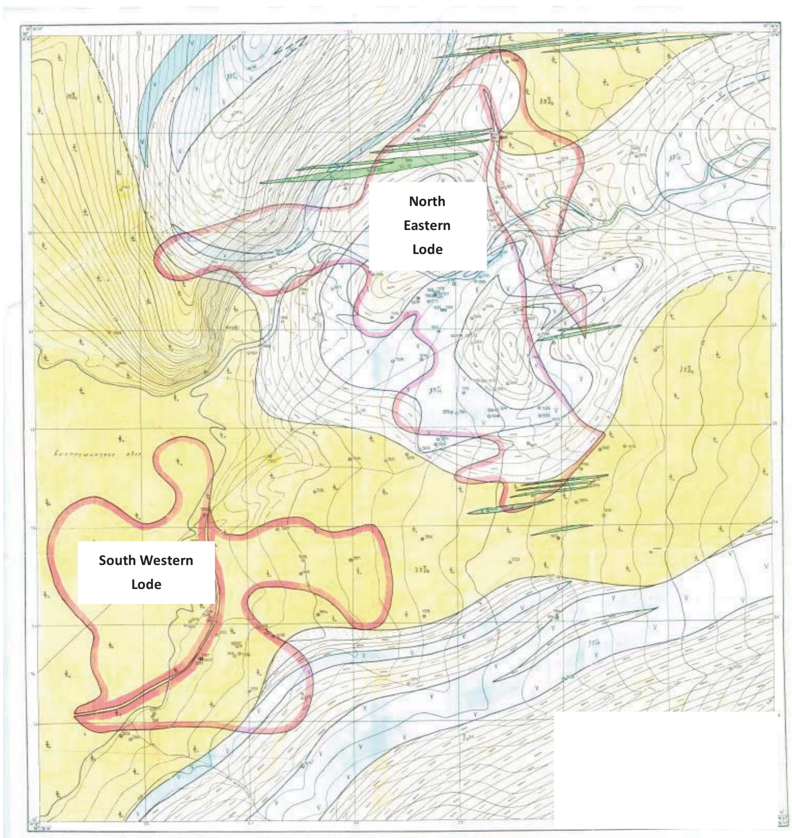


Figure 8.2: Dolinnoe – Outline of the North Eastern and South Western Lodes
(Scale Each Grid Division = 200m; True North-Vertical Grid Lines)

Each lode contains discrete mineralised bodies of two contrasting forms:

- Conformable, stratabound, mineralised mound-like bodies in altered calcareous siltstones; and
- Moderately to steeply dipping cross-cutting lenticular vein-like bodies in the gravelites which underlie the calcareous siltstone horizon.

The stratabound mineralisation is confined to zones of pervasive silica alteration which are locally referred to as microquartzite and which have developed within the calcareous siltstone horizon of the Kryukovskaya Formation. This is the key stratigraphic marker horizon identical to that observed at the Ridder-Sokolniy mine. It is widely believed that the calcareous siltstone acted as a screen preventing the ascent of mineralising solutions after it had been silicified and forcing precipitation of contained metals. However, venting of hydrothermal fluids and the direct precipitation of sulphides to form sub-marine sulphide rich mounds appears to be a more plausible possibility.

The steeply dipping lenticular bodies occur in intensely silicified tuffaceous gravelites and also in rhyolites and rhyolite breccias wherever these are present. These vein-like bodies are considered to be feeder channels to the stratabound sulphide accumulations.

Hydrothermal alteration assemblages associated with the mineralisation are predominantly quartz-sericite, chlorite-sericite-quartz and quartz. With increasing depth alteration changes from pervasive replacement of calcareous siltstone by silica (microquartzite) to quartz-sericite and quartz alteration of the underlying tuffogenic gravelites, porphyritic rhyolites and rhyolite breccias. Laterally, microquartzites pass into quartz-sericite, and locally, into quartz-barite-carbonate-sericite alteration assemblages found on the fringes of the hydrothermal system. Elevated silver content has been noted in the syncline separating the two domes. Sericite alteration occurs in the footwall of massive gold-polymetaliic mineralisation.

North-Eastern Lode

The stratabound North-Eastern Lode is represented by a single mineralised body, (Orebody 3) and a few small and isolated lenses. The underlying mineralisation is represented by a swarm of twelve moderately to steely dipping north-northwest trending veins.

Orebody 3 is centred on a north-northwest elongated half dome abutting against the Prodolnyi I Fault (see typical cross-section in Figure 8.3). Its thickness exceeds 25m in the saddle-shaped hinge zone. The south-western limb has a variable geometry due to variations in dip and the presence of subsidiary folds. Dips are generally shallow to moderate south-west but due to folding locally change through subhorizontal to shallow to the north-east. Widths are also variable, typically in the range of 5m or less. The hinge zone is cut by the Prodolnyi II Fault, a reverse fault parallel to Prodolnyi I Fault with throws of up to 35m. In plan view, Orebody 3 covers an elongated north-northwest trending area: 800m in length, with four west-pointing embayments and one 500m long offshoot. The lateral extent in plan varies from 200-400m.

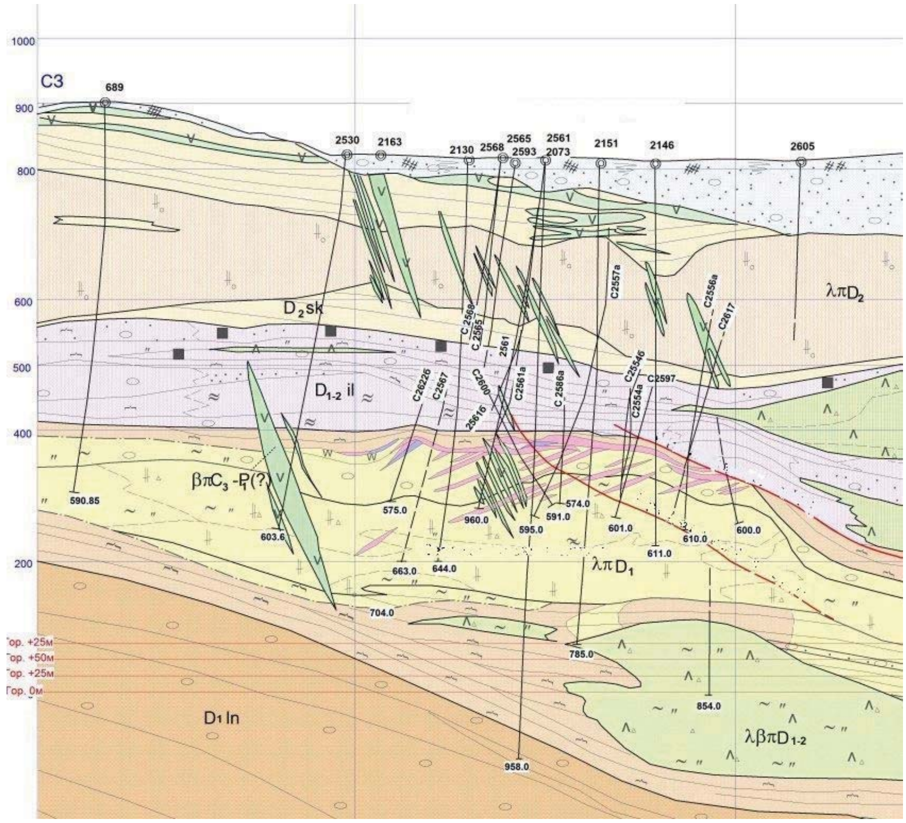


Figure 8.3: Dolinnoe-North Eastern Ore Body
(Scale Each Grid Division = 100m)

Orebody 3 displays pronounced vertical zoning with low sulphide barite mineralisation at the top passing down into massive barite-polymetallic and pyritic-polymetallic mineralisation, which in turn passes into massive polymetallic mineralisation. Barite is not pervasive and when it is absent, massive polymetallic mineralisation extends through the whole thickness of the mineralised zone. The hanging wall contact with the overlying siltstones is generally very sharp. Host siltstones are brecciated and contain hydrothermal barite, chlorite and sericite. The footwall rocks display very strong sericitic alteration with coarse crystalline barite and nests of sulphides.

Rich gold-silver-polymetallic massive sulphide mineralisation (>20% ZnEq) forms four lenses along the hinge zone of the dome. The largest lens extends for 300m with widths of 100-175m and up to 10.5m thick.

Steeply dipping mineralised zones strike 355° and dip 65°SW. The overall strike length of the whole swarm is 600m, vertical extent is at least 200m and the overall width is 130m. True widths of intercepts of individual zones range from 0.35m to over 7m (Table 8.3). Geolen interpreted these zones as being sub-parallel to each other and continuous within tuffogenic gravelites by analogy with similar zones at the neighbouring Bystrushinskaya Lode of the Ridder-Sokolniy deposit (which is in production) and at the Novo-Leninogorskiy deposit, which is located 5km to the east.

***WAI Comment:** This interpretation is acceptable with regard to the overall structural pattern but the drilling data so far accumulated are insufficient to allow the grade continuity to be confidently interpreted between all drillhole intercepts on the steeply dipping vein mineralisation.*

Table 8.3: Dolinnoe-Parameters of Steeply Dipping Mineralised Bodies from the North-Eastern Lode

Mineralised Body	Dimensions (m)				№ of Drillhole Intercepts	Average Grades				
	Strike Length	Downdip Extent	True Width From	To		Cu %	Pb %	Zn %	Au g/t	Ag g/t
6	100	35	1.0	1.1	2	0.05	1.05	2.02	3.16	9.95
7	75	25	1.0	1.3	2	0.53	1.64	2.36	3.72	15.21
8	450	120	0.4	5.3	13	0.18	1.04	2.10	3.49	11.86
9	260	50	0.7	7.1	10	0.26	1.11	2.05	3.25	19.66
10	130	80	0.4	4.2	13	0.18	1.22	2.45	3.10	7.78
11	200	70	0.6	2.9	5	0.20	2.14	4.36	3.13	13.05
12	354	130	0.5	3.6	15	0.26	1.00	1.86	6.29	14.27
13	260	80	0.7	2.4	10	0.15	0.67	1.11	6.50	7.38
14	400	115	0.4	3.4	16	0.22	1.08	2.46	6.17	12.84
15	340	85	0.4	3.0	13	0.23	1.36	2.35	4.68	47.76
16	150	80	0.4	2.4	6	0.25	1.03	2.58	5.13	9.14

8.2.3.2 South-Western Lode

This lode occurs at a depth of 495m to 580m below surface. Steeply dipping feeder veins extend to deeper levels. Grades tend to be lower than those recorded in the North-Eastern Lode.

The stratabound mineralisation forms a gently rising dome and is cut on the south¬eastern side by the Dugovoy Fault. The lode extends over a strike length of 145m to 420m and over a distance of 120m to 460m down dip.

Stratabound mineralised lenses, which are hosted by calcareous siltstones altered to microquartzites, cover an area of 600x600m-800m. The thickness of the altered microquartzites varies from a few metres to 55m. It appears that the favourable calcareous siltstone horizon had been locally eroded prior to deposition of the overlying llinskaya Formation.

Mineralised bodies are numbered in ascending order from the footwall to hanging wall of the calcareous siltstone horizon. Based on the approved reserve conditions, there are five subconformable lenticular mineralised bodies: 3, 3.1, 3.2, 3.3 and 3.4. In addition, there are 27 drillhole intercepts outside these lenses within the same lithological horizon.

There is no information to assess if these intercepts indicate flat lying stratabound lenses of mineable dimensions or just local mineralised partings within barren siltstones.

Drilling completed during the preliminary exploration stage also encountered steeply dipping mineralised structures within tuffaceous gravelites and volcanic rocks below the calcareous siltstone horizon. Geolen allocated these intersections to seven steep veins not exceeding 30-50m in strike and downdip directions with widths from tens of centimetres to a few metres.

The Dugovoy Fault has an eastern downthrow of less than 300m. Extensions of all mineralised lenses have been confirmed by drilling in the downthrown block but not followed along strike and dip.

Other Mineralisation

More recent drilling has identified mineralisation between Ridder-Sokolniy Mine and the South Western lode at Dolinnoe (Figure 8.4). Information on reserves and grade has not been obtained.

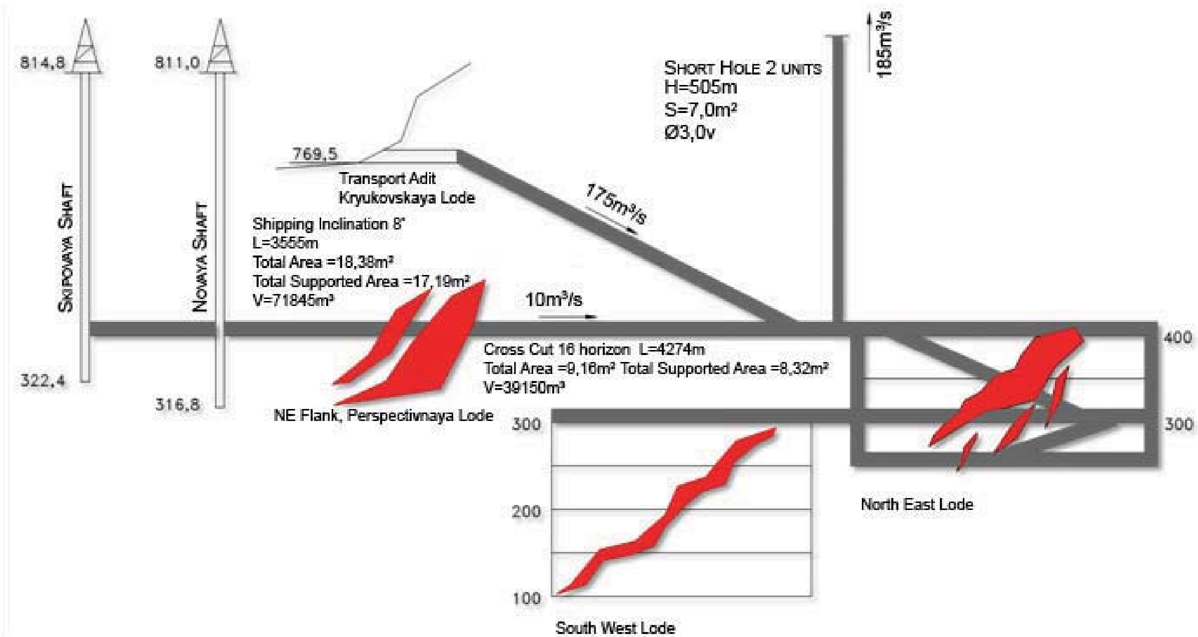


Figure 8.4: Dolinnoe-Diagrammatic Representation of Proposed Development with Perspectivnaya Extension

8.2.4 Obruchevskoe

Stratabound polymetallic sulphide mineralisation has been outlined by drilling in two cupola-shaped lodes, Southern and Northern, and traced over the intervening ground, called Central Prospect, within the same lithological unit of the Kryukovskaya Formation. At the Southern and Northern lodes (Figure 8.5 and Figure 8.6) the mineralisation occurs as tabular and lenticular veinlet-disseminated and massive sulphide bodies on three horizons that are identified, from top to bottom, as Nos.1, 2 and 3. Relatively thin and generally low grade polymetallic mineralisation encountered at the Central Prospect is correlated with the No.1 horizon.

The three horizons are embedded in altered rocks with disseminated pyrite and low grade polymetallic mineralisation. In a broad sense, the stratabound mineralisation so far delineated at the Southern and Northern lodes spans a vertical interval of 110-160m and 60-80m respectively.

The Northern Lode is situated at a depth of 850-920m below surface and the Southern Lode is at a depth of 930-1,155m. This is due to a combined effect of the south sloping topography and the generally south-west shallow dipping bedding within the Kryukovskaya Formation.

The continuity between drillhole intercepts is supported by the following observations:

- Mineralisation occurs below a distinctive horizon of calcareous siltstones within the upper part of the Kryukovskaya Formation;
- Drill intercepts display similar mineralogical composition, degree of concentration of copper, lead, zinc, gold and silver and similar vertical zoning patterns;
- Similar textures observed in neighbouring drillhole intercepts;
- Confirmation of earlier interpreted continuity by infill drilling to 25-50m spacing between drillholes; and
- Similar geophysical signatures on downhole geophysical logs.

Composition of the mineralisation is polymetallic but changes from predominantly lead-zinc in the Northern-Lode to copper-zinc in the Southern Lode, the latter with predominant copper or zinc varieties. Vertical zoning is represented by an increase of copper to lead ratio with depth. The lateral zoning seems to be related to hydrothermal alteration. Lead-zinc mineralisation with low copper occurs in association with sericite-quartz alteration in the Northern Lode, whilst mineralisation with copper and zinc predominating over lead occurs in association with chlorite alteration in the Southern Lode, particularly in its south-eastern part. On a local scale, however, pockets of polymetallic mineralisation also occur in the Southern Lode, particularly to the north west.

Gold and silver grades tend to be higher at the apical and 'apron' parts of mineralised lenses and decrease with depth. However, it is possible that gold and silver may occur in small shoots similar to those at the Ridder-Sokolniy deposit, at intersections of north-west and north-east trending faults. Oleynik (2003) mentioned three shoots in the Southern Lode, one in the Northern Lode and one in the Central Prospect. Gold also occurs in low sulphide quartz veins.

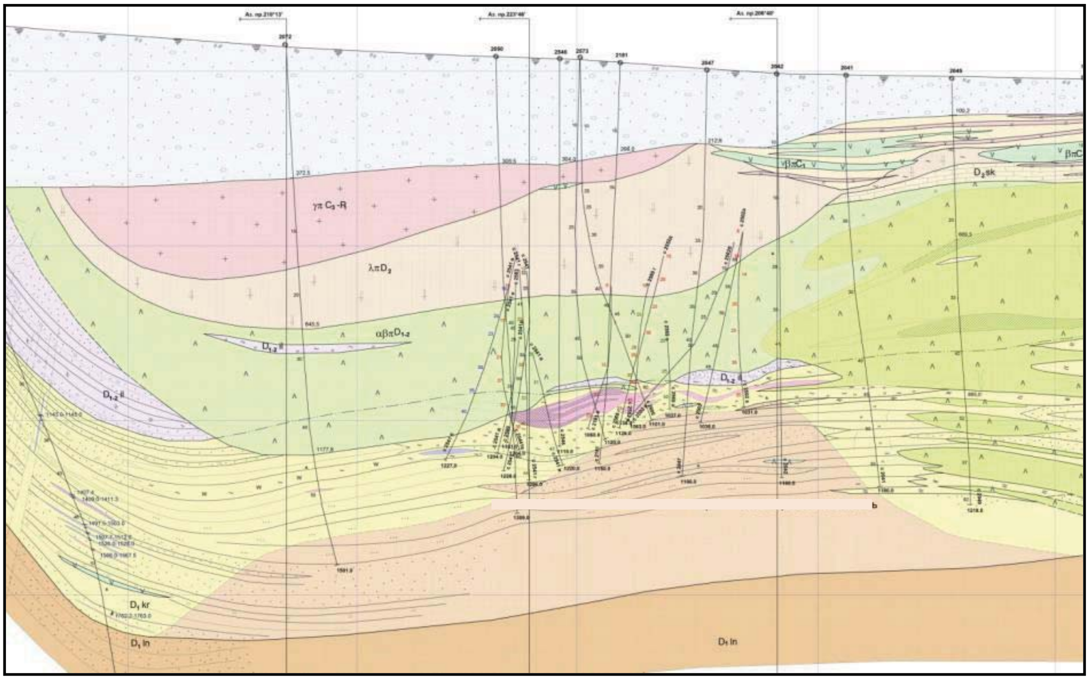


Figure 8.5: Obruchevskoe-Southern (Left) and Northern (Right) Lodes from Section XXV
(Scale: Grid Division= ±100m)

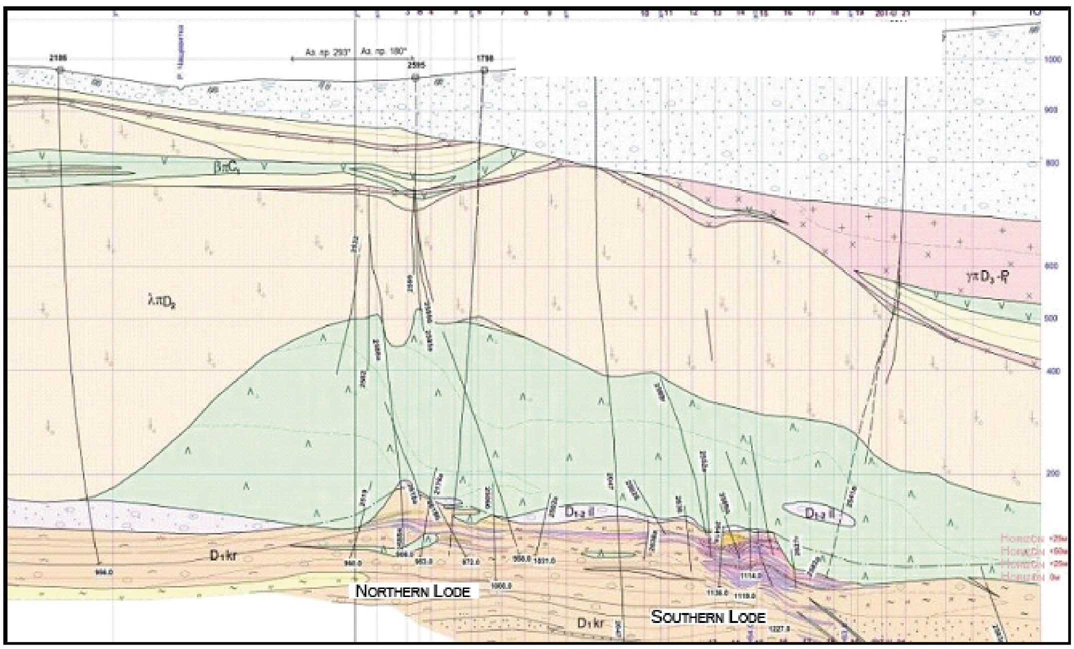


Figure 8.6: Obruchevskoe Northern (Left) and Southern Lodes (Right) from Section XXVI
(Scale: Grid Division= ±100m)

8.2.4.1 Southern Lode

This is by far the larger of the two lodes. It consists of three stratabound mineralised bodies denoted as 1, 2 and 3 from top to bottom and enclosed in an envelope of low grade disseminated pyrite-copper-zinc mineralisation. Mineralised Body No 1 has been traced over a distance of 970m throughout the deposit from the Southern Lode through the Central Prospect to the Northern Lode. The lode peters out rapidly on the western flank and is truncated by a porphyritic andesite intrusion of complex geometry on the eastern and southern flanks.

The proportions of base metal minerals varies considerably, with polymetallic and copper-zinc (>0.5% Cu) mineralisation types being the most common. Veinlet-type and disseminated mineralisation predominate over massive sulphide mineralisation.

The average Cu:Pb:Zn ratio is 0.3:1:2.4. There is a good correlation of lead with zinc (0.839) and gold with silver (0.759), some correlation of silver with lead (0.660), zinc with silver (0.568) and gold with lead (0.541). Copper alone has no correlation with the other elements.

The host rocks are gravelites, sandstones and siltstones of the Kryukovskaya Formation, all subjected to predominantly chloritic alteration. Chlorite is variously associated with quartz, pyrite and carbonate. Sericite alteration is local and insignificant in volume.

8.2.4.2 Northern Lode

The Northern Lode consists of three stratabound mineralised bodies denoted as 1, 2 and 3 from top to bottom and enclosed in an envelope of low grade disseminated pyrite-polymetallic mineralisation. The three mineralised bodies correlate with those in the Southern Lode. On the north-eastern side, the Northern Lode is truncated by small intrusive bodies of porphyritic andesite and quartz-albite rhyolite.

The average Cu:Pb:Zn ratio is 0.2:1:1.9. There is a very good correlation of zinc and lead (0.943) and good correlation of copper with zinc (0.802) and lead (773). Gold and silver have no correlation with the three base metals and only very weak correlation with each other (0.406), except in high-grade lead-zinc mineralisation where the correlation coefficient exceeds 0.9.

The host rocks are mottled gravelites and siltstones of the Kryukovskaya Formation, both subjected to intense carbonate-quartz-sericite alteration. The area displays two intersecting cleavages and brecciation of the more competent rock types.

8.2.4.3 Central Prospect

As interpreted from relatively widely spaced drilling, this prospect contains a single horizon of stratabound mineralisation correlated with the No 1 horizon of the neighbouring lodes. Base metals grades are relatively low but some high-grade gold and silver-bearing polymetallic mineralisation was reported.

8.2.5 Mineralogy

The mineral composition of the Dolinnoe and Obruchevskoe, Ridder-Sokolniy and other deposits in the Leninogorskiy ore field, is relatively uniform and are all characterised by the same complex of ore and non-metallic minerals.

The former comprise sphalerite, galena, chalcopyrite and pyrite which, depending on their content in different ores, can constitute major, abundant or accessory minerals, although in solid sulphide polymetallic ores they may all be major components. Accessory minerals include tetrahedrite and tennantite with a high zinc content (fahlores which can be present in varying amounts); rare minerals consist of free gold, gold-silver alloy (electrum), pyrrhotite and specifically, at Obruchevskoe, bornite and some chalcopyrite and arsenical pyrite.

Non-metallic minerals in the ore zones are quartz, calcite, sericite and chlorite and accessory minerals are dolomite, hematite and feldspar. Apart from the above mentioned minerals ore may contain apatite, marcasite, molybdenite, rutile, barite and sphene.

8.2.5.1 Dolinnoe

Veinlet-disseminated sulphide mineralisation predominates (95% of the total volume), with massive sulphides (sulphides >50% vol) making up the balance. The average ratio of Cu:Pb:Zn is 0.3:1:2 in the North-Eastern Lode and 0.3:1:1.8 in the South-Western Lode. The average Cu:Pb:Zn ratio in barite-polymetallic and massive pyritic barite-polymetallic (>15% pyrite plus >6% barite) mineralisation in the North-Eastern Lode is 0.4:1:1.3.

Table 8.4 lists minerals described and mentioned in the Dolinnoe exploration reports of 1994 and 1997.

Table 8.4: Dolinnoe Mineralogy

Abundance	Metalliferous Minerals	Gangue Minerals
Main	Sphalerite, galena, chalcopyrite, pyrite	Quartz, calcite, sericite, barite
Abundant	Silver-zinc tetrahedrite, silver-bearing zinc tetrahedrite, silver-bearing zinc tennantite-terahedrite	Dolomite, chlorite, hematite
Accessory	Native gold, electrum, pyrrhotite, kerargyrite	Rutile, apatite, monazite

Gold is the main mineral of economic interest. Results of phase analyses conducted by VNIItsvetmet and Kazmekhanobr (see Table 8.5) demonstrate that about 30% of gold occurs as free milling (native) gold particles with clean surfaces, about 5% as native gold covered with oxidised films and about 40% as native gold in intergrowths with other minerals. About 25% of gold occurs in crystal lattices of sulphides, mainly in sphalerite and, to a lesser degree, in galena, chalcopyrite and fahlores (tetrahedrite-tennantite group). Pyrite also carries gold though the available reports do not elaborate on the type of association. In addition, small amounts of gold occur in quartz.

The highest gold concentrations are found in massive polymetallic and pyritic-polymetallic mineralisation and, also, in massive barite-pyritic-polymetallic mineralisation. Steeply dipping veins in gravelites are generally lower in gold content.

The highest silver grades occur in massive pyritic-barite-polymetallic mineralisation. Phase analyses of silver showed that 10-15% of silver is free, 45-50% is intergrown with other minerals and 25-30% occurs in association with sulphides. Up to 8.5% of silver is held in grey ores (fahlore-tetrahedrite-tennantite group).

The reviewed reports mention molybdenum grades in the range of 0.004-0.015% at which levels it is a potentially economic by-product.

The main deleterious element is antimony, which ranges from 0.01-0.035%. Another possibly deleterious element is cadmium, which occasionally exceeds 0.025%. Geolen estimated the average content of cadmium as 0.01%.

Table 8.5: Dolinnoe- Forms of Gold and Silver Occurrence Based on Phase Analyses		
Forms of Gold and Silver Occurrence	**Percentage**	
	Gold	**Silver**
Laboratory Sample No 1 (VNIltsvetmet, 1991)		
3.0g/t Au, 30g/t Ag		
Free		
with clean surface	26.7	3.9
covered with oxidised films	4.5	11.4
Intergrown		
with clean surface	12.7	31.0
covered with oxidised film	30.6	17.8
Associated with sulphides	24.0	30.2
namely:		
with anglesite and cerussite	3.0	4.1
with galena	4.3	6.4
with sphalerite	6.8	7.1
with pyrite	8.4	8.7
with chalcopyrite	1.5	3.9
Associated with gangue	1.5	5.7
Large Laboratory Sample No 1 (Kazmekhanobr, 1992)		
3.43g/t Au, 91.4g/t Ag		
Free	34.4	10.9
Intergrown	34.4	33.8
Covered with films	6.6	10.4
Associated with sulphides	24.6	24.2
Associated with gangue		10.3
In kerargyrite		10.4

8.2.5.2 Obruchevskoe

Veinlet-disseminated polymetallic mineralisation predominates, and consists of sphalerite (1-6%, average 4%), galena (0.5-3%, average 2%) and chalcopyrite (average <1%). Sphalerite occurs as porphyroblasts (up to 4mm across) among aggregates of sphalerite, galena and chalcopyrite with black ore inclusions. The predominant grain size of these minerals is 0.5-1mm. Massive sulphides (total sulphides >50% by volume) form about 12% of the mineralisation volume and hold approximately 40% of the contained metal.

Copper-zinc mineralisation (>0.5% Pb) is also abundant particularly in the Southern Lode of combined sulphides).

Table 8.6: Obruchevskoe Mineralogy		
Abundance	**Minerals**	
	Metalliferous	**Gangue**
Main	Pyrite, sphalerite, galena,chalcopyriteBlack ores (silver-bearing zinc tetrahedrite, zinc tennantite, tetrahedrite)	Quartz, sericite, chlorite, calcite
Abundant	Proustite-pyrargyrite, lead sulphosalts, electrum, arsenopyrite, pyrrhotite,	Dolomite, feldspar, talcum
Accessory	bornite, wurcite, magnetite, marcasite	Apatite, rutile, sphene, barite,monazite

8.3 Exploration

8.3.1 Historical Work

8.3.1.1 Dolinnoe

The Dolinnoe deposit was discovered by DDH 1799 which drilled in 1987 to follow up results of integrated deep drilling, geophysical prospecting and geochemical investigations. DDH 1799 intercepted low grade polymetallic mineralisation at depths between 600-800m.

The result was considered to be encouraging enough to warrant further drilling. A prospecting-assessment programme, focused on the area where the discovery hole was sited, was conducted in 1989 to 1990. Several drillholes intercepted polymetallic mineralisation with significant gold grades, paving the way for further investigations.

The ensuing exploration was conducted in two stages:

- Qtr 1 1991 - Qtr 2 1993: Preliminary Exploration focused on the North-Eastern Lode and including the South-Western Lode, with both targets drilled on a grid of 50x50 -50x25-50m to a depth of 800m; and
- Qtr 3 1994 - Qtr 2 1996: Additional Exploration of the North-Eastern Lode, essentially an infill drilling programme to reduce the spacing between drillhole intercepts to about 25m and to upgrade the delineated reserve from the C_2 to the C_1 category.

Both exploration stages were conducted by Ridder-based TOO Geolen (Geolen).

8.3.1.2 Obruchevskoe

Mineralisation was discovered by DDH 1798 which drilled in March 1987 to follow up a geochemical anomaly. DDH 1798 intersected pyritic-polymetallic mineralisation between depths of 869.20-917.20m, including 4.35m of high-grade massive sulphide mineralisation that promoted follow-up exploration

A prospecting-assessment programme focused on the area where the discovery hole was sited was conducted from March 1989 to January 1991. A total of 35,563m was drilled in 40 drillholes, including 19 wedged deflections. DDH 2181, sited 600m south of the DDH 1798, intersected 66.5m of pyritic-polymetallic mineralisation which was confirmed in the wedged deflection No 2181a. Subsequent drilling delineated a body of mineralisation which was identified as the Northern Lode. The programme was carried out by the state-owned Leninogorsk Geological-Exploration Expedition; a scoping study conducted at the end of this programme recommended further drilling.

From July 1991 to November 1995 a total of 55,519m was drilled in nine parent holes and 80 wedged deflections, including 15 metallurgical holes. The drilling focussed on the Northern Lode and a newly discovered mineralised area, the Southern Lode. The drilling programme was 85% complete when it was terminated due to the withdrawal of State funding. The programme was carried out by TOO Geolen (Geolen), the privatised former state-owned Leninogorsk Geological-Exploration Expedition.

8.3.2 Drilling

8.3.2.1 Dolinnoe

A total of 70 diamond coring exploration holes with 94 wedge deflections was drilled in the two exploration stages. In addition, 30 technical holes were completed to obtain samples for process testwork. Table 8.7 provides essential summary figures.

Table 8.7: Dolinnoe Drilling Summary								
Description	Number of Drillholes	Metres Drilled and Core Recovery						
		Total Drilled	Including intervals in Kryukovskaya Formation			Including Intervals with Gold Polymetallic Mineralisation		
			Interval	Core Recovery		Interval	Core Recovery	
		m	m	m	%	m	m	%
Preliminary Exploration (1991-1993)								
Exploration Holes	146	70,309	15,419	12,758	827	1,315	1,102	84.0
including								
Parent Holes	66	41,426	7,645	6,383	83.5	472	399	84.7
Wedged Deflections	80	28,883	7,774	6,376	82.0	840	703	83.6
Technological Holes	21	1,471	980	785	80.1	320	258	80.5
Additional Exploration (1994-1996)								
Exploration Holes	18	6,737	746	617	82.7	-	-	-
including								
Parent Holes	4	2,149	161.4	138.67	85.9	-	-	-
Wedged Deflections	14	4,588	584	478	81.8	-	-	-
Total								
Technological Holes	9	466	173	138	79.5	-	-	-

Drillhole trajectories were surveyed with MIR-36 and MI-40 inclinometers at 20m intervals over 40-100m depth increments. Control measurements were taken every 200-300m, at the beginning of each mineralised intercept and at the end of each hole.

Core recoveries were recorded by geologists after each run. In addition, weight and actual core diameters were recorded for all core recovered from mineralised, allowing. monitoring of the core recovery by weight.

The recovery was in the range of 80-86% which is not considered to be good by current industry standards.

A statistical analysis of grades against core recovery was conducted by Almaty-based consulting firm Geoincentr, who concluded that there was no differential loss.

8.3.2.2 Obruchevskoe

Borehole intersections form an irregular grid pattern:

- Northern Lode – 40x40m in the centre to 70x130m on the flanks;
- Southern Lode - an irregular grid with drillhole intersections at intervals of 25-60m; 60-100m on the north-eastern and south-eastern flanks; and
- Central Prospect - intersections 90-250m apart.

Start-up holes were drilled with roller bits at near vertical angles (average inclination 84-90° from horizontal) and a series of wedged deflection was made from those parent holes and from deflected holes to reach target depths at specific points.

Hole trajectories were surveyed with MIR-36 inclinometers at 20m intervals over 40-100m depth increments. Drilled depths were checked each time drilling advanced by 200-300m, at the top or end of each mineralised intersection and at the end of hole.

Core recovery was checked by geologists after each run and, from mineralised intercepts, additionally by hydrostatic weighting and measuring the diameter of the recovered core.

Core recovery from the mineralised section of the Kryukovskaya Suite averaged 79% at the prospecting-assessment stage and 81% at the preliminary exploration stage. Low core recovery (less than 70%) was reported in DDH 1783, 2501 and 2569. Grades over these intervals were compared against RRK logging data with satisfactory results.

8.3.3 Sampling

8.3.3.1 Introduction

Core was sampled over all intercepts of mineralised zones and hydrothermally altered hangingwall siltstones of the Kryukovskaya Suite.

At the preliminary exploration stage, the whole core was taken for sample preparation and analysis except for intervals selected for technological tests, which comprised half core. Bulk density determinations by hydrostatic weighing were performed on all samples.

At the additional exploration stage, core was sawn in half, one half sent for sample preparation and analysis and the other half used for mineralogical and petrographic studies and for compositing into technological samples. Prior to splitting, all core was weighted in water and air and those results were combined with measurements of the core diameter to estimate core recoveries by weight and bulk density.

Minor accompanying elements and potentially deleterious elements were determined on composite (grouped) samples comprising pulverised duplicates of routine samples by combining small portions of three to four successive samples, with the weight of each portion being proportional to the length it represented. The composite sample intervals were selected to ensure that each sample represented the same type of mineralisation and/or more or less uniform grade range.

Sample Preparation

Sample preparation was carried out at the Geolen sample preparation facility. The method was based on the Richard-Czeczott formula Q=kda, where Q is the minimum sample quantity at a given stage of volume reduction, d is the diameter of the largest fragments defined as the screen size that retains the largest 5% of the mass, к is a coefficient dependent on the distribution irregularity of the mineral of interest and a is a coefficient related to the roundness of mineral grains (generally approximately 2).

The coefficient к is the key parameter. Initially the sample preparation scheme was based on the к coefficient of 0.16. From March 1995, on GKZ's recommendation and after tests on 0.5kg subsamples of core crushed to -1 mm, the coefficient к was changed to 0.6. The scheme used for Obruchevskoe samples was ased on K-0.16. In general terms, the lower coefficient к is, the better it accounts for the erratic distribution of minerals. The tests conducted in 1995 showed that relative differences in copper, lead and zinc grades remained relatively constant at different values of k. Relative differences in gold and silver grades showed some variations but those variations reportedly remained within a narrow range.

8.3.3.2 Dolinnoe & Obruchevskoe

A total of 9,789 samples were collected during the preliminary exploration stage. The samples ranged from 0.3-2m in length, the average length being 1.3m. The average length of the core samples from the additional exploration stage was 1.15m, with a range from 0.20m to 1.80m.

Other samples collected during the preliminary exploration stage included:

- 1958 geochemical samples representing core intervals of up to 10m in length (6m on average) made up of small chips broken off along groves parallel to core axis;
- Samples from the lithological horizon hosting the stratabound mineralisation (also referred to as "upper mineralised zone") for silicate analysis to determine the silica content and 21 samples for self ignition tests;

- 39 waxed samples from different types of mineralisation and host rocks for the determination of bulk density, moisture content and porosity;
- 234 specimens for thin sections and 36 specimens for polished sections; and
- Samples for process testwork comprising: one 2,247.15kg sample; two laboratory scale samples, 558.8kg and 315kg in weight respectively; and five samples totalling 306.3kg in weight to be used for metallurgical mapping.

Analyses

Selection of core samples for accurate analysis was done on the basis of results of spectral analyses and spectral gold analyses. Samples that gave spectral grades greater than 0.3% Cu, Pb or Zn were submitted for chemical analysis of the three metals. Samples showing > 0.3g/t Au and Ag were submitted for fire assay. Samples submitted for gold and silver assays averaged 200-300g in weight, but the available reports do not mention the sample charge. Gold and silver beads were measured using the gravimetric method.

Composite samples collected at the preliminary exploration stage were analysed for Cu, Pb, Zn, Ba: Cd, Bi, Se, Te, total sulphur, sulphide sulphur, Sb, As, Ge, Ga, In, Tl, Ag, Fe, Mo, and Co.

Due to financial difficulties during the additional exploration stage, analytical work was limited to routine analyses for copper, lead, zinc, gold, silver and barium.

All involved laboratories (see Table 8.8) were certified in Kazakhstan in the 1990s and have maintained their certification status.

Table 8.8: Laboratories Used for Dolinnoe Exploration Analyses

Elements	Principal Laboratory	Control Laboratory	Method
Preliminary Exploration			
Multi-element Scan	AO Leninogorskiy Geolog	CChl VK PGO	Spectral
Gold	CL PGO Vostkazgeolgia Leninogorsk GRE CChl LPC	CL PGO Samrkandgeologia	Fire Assay
Silver	CL PGO Vostkazgeolgia Leninogorsk GRE CChl LPC	CL PGO Samrkandgeologia	Fire Assay
Copper, lead, zinc	AO Leninogorskiy Geolog	CL PGO Yuzhkazgeologia	Chemical
Barium	CL PGO Vostkazgeolgia Leninogorsk GRE CChl LPC	CL PGO Yuzhkazgeologia	Chemical
Other elements	VNIItsvetmet CChL YuK PGO	CL PGO Yuzhkazgeologia	Various
Process testwork	VNIItsvetmet		
Additional Exploration			
Multi-element scan	AO Leninogorskiy Geolog		Spectral
Gold	AO LPK		Fire Assay
Silver	AO LPK		Fire Assay
Copper, lead, zinc	AO LPK		Chemical
Barium	AO LPK		Chemical
Process testwork	VNIItsvetmet		

Abbreviations:
CChL LPC Central Chemical Laboratory Leninogorsk Polymetallic Combination
CChL YuK PGO Central Chemical Laboratory South Kazakhstan Geological Administration
CChL VK PGO Central Chemical Laboratory East Kazakhstan Geological Administration

Quality assurance and control (QA/QC) relied on internal and external control analyses of analytical duplicates. Internal control analyses did not indicate any problems, except for large relative differences for zinc analyses in Class 5% to 10% in the first six months of 1991. As most samples contain less than 5% Zn, this is not considered to have a material impact on zinc grade estimates.

With regards to external analytical control, 14.5% of samples were submitted for external chemical analyses and 6% for external fire assays for gold and silver during the preliminary exploration stage. Due to the lack of adequate funding, no samples were submitted to external laboratories during the additional exploration stage.

Relative differences between results of external and routine analyses for samples submitted at the preliminary exploration stage at times exceeded the 95% confidence limit (see Table 8.9). To assess the impact on grade estimates, Geolen carried out two sets of estimates of zinc equivalent based on all samples from the North-Eastern Lode that were analysed in the first six months of 1993. The first set of estimates was based on the results as reported by Geolen and the second set on values for the grades of the same samples calculated from regression formulas, derived from the comparison of routine and control results. The overall impact turned out to be insignificant: 0.02% reduction in tonnage and 0.46% reduction in contained copper and zinc.

Table 8.9: Grade Classes with Relative Differences in Excess of 95% Confidence Limit

Element	Grade Class	Period	Lode
Cu	0.01 – 0.2%	Otr 1-2 1991	South-Western
	0.2 – 0.5%	Otr 1-2 1993	North-Eastern
	0.5 – 1%	1991	South-Western
Zn	0.2 – 0.5%	Ote 1-2 1993	North-Eastern
Au	1 – 4g/t	1991	South-Western
	0.2 – 1g/t	1992	South-Western

8.3.3.3 Dolinnoe & Obruchevskoe

According to tables in the Geolen report, 10,356 core samples were taken Including 2,520 samples taken at the prospecting-assessment stage, 6,403 samples taken at the preliminary exploration stage and 1,433 samples taken from earlier prospecting boreholes, including the discovery hole. The average sample length was 1.6m. Other samples included:

- 7,781 geochemical samples representing core intervals of up to 10m in length (9.2m on average) made up of small chips broken off along groves parallel to the core axis;
- Samples from the lithological horizon hosting the stratabound mineralisation for silica analysis;
- 35 waxed samples from mineralised intercepts for the determination of bulk density, moisture content and porosity; and
- 126 specimens for thin sections and 63 specimens for polished sections.
- Samples for process testwork and metallurgical mapping.

Analyses

As for Dolinnoe, spectral analyses were conducted at the laboratory of the Leninogorsk GRE (Geolen), as well as chemical analyses for copper, lead and zinc. Fire assaying for gold and silver was conducted at the central laboratory of Vostkazgeologia in Ust-Kamenogorsk, the laboratory of Leninogorsk GRE and the central laboratory of the Ridder-Sokolny mine.

Vostkazgeologia and VNIItsvetmet analysed composite samples for potential by-products and deleterious elements.

External control analyses were performed at the laboratories of Altay Geophysical Expedition (spectral analyses), Yuzhkazgeologia (chemical analyses) and Samarkandgeologia (fire assays). All participating laboratories were locally accredited.

Internal and external control analyses did not reveal any significant random or systematic errors.

8.3.4 Bulk Density

8.3.4.1 Dolinnoe

Bulk density was determined by hydrostatic weighing of about 7,000 core specimens. However, the development of unambiguous regression formulas for different types of mineralisation proved to be very difficult. Various formulas were tested but they either overestimated tonnage at the expense of contained base metals or vice versa.

It appears that the main reason for wide density fluctuations and low correlation between density and combined base metals is the presence of barite. However, the distribution of barite is erratic as it tends to be localised at discrete horizons, particularly in the upper portions of the stratabound mineralisation. High silver grades, often recorded in barite-rich mineralisation, further complicate the issue. Incorporation of barite in the regression formulae reportedly did not help in improving them and the issue requires further investigation.

Table 8.10 lists regression formulae used in the current reserve estimates. Relative differences between densities derived from the formulae and mean densities calculated from actual measurements for various grade classes average approximately from 1% to 7% depending on the combined content of base metals.

Table 8.10: Regression Formulae Used in Current Reserve Estimates

Lode	Mineralisation Style	Regression Formula
North-Eastern	Stratabound balance and off balance	D =2.6874 + 0.0469Cu + 0.054575PB +0.019296Zn
	Rich Stratabound	D = 2.8807975 + 0.075101Cu + 0.048542Pb + 0.013674Zn
	Steep dipping veins	D = 2.663691 + 0.015273Cu + 0.017277Pb + 0.021154Zn
South-Western	All Stratbound	D = 2.395283 + 0.90559Cu + 0.01636Pb 0.001154Zn
	Steeply dipping veins	D = 2.663591 + 0.015273Cu + 0.017277Pb + 0.021154Zn

8.3.4.2 Obruchevskoe

The same density determintation applies of Obruchevskoe and the following regression formula was developed based on 1,094 measurements:

D = 2.743429+0.022552 Cu + 0.069044 Pb + 0.017698 Zn

> ***WAI Comment:*** *Drilling methods, sampling and, sample preparation appear to be satisfactory based on historical reports and at both projects QA/QC (internal and external control analyses) did not reveal any significant random or systematic errors.*

8.3.5 Future Exploration

The Work Programme, submitted by Kazzinc and accepted by the Ministry of Energy and Mineral Reserves, includes detailed exploration (which is also Kazzinc's contractual commitment) phased over a six-year period and comprises:

- Pilot drilling at the sites selected for three shafts - the main shaft to access the Dolinnoe and Obruchevskoe deposits (Ventilation Shaft), an exploration shaft at Dolinnoe and an exploration shaft at Obruchevskoe;
- Capital development, comprising sinking of Ventilation Shaft with the development of a haulage crosscut on 18 level and sinking of the exploration shafts with the development of drives on 16 and 17 levels;
- Underground exploration, including 20,200m of diamond core drilling to infill the current drilling grid to 12x12-30m;
- Technical-Economic Assessment (TEO) to define reserve conditions; and

- Reserve estimation and submission of results for approval to the Government Commission for Natural Reserves (GKZ).

Table 8.11: Schedule and Budget for Detailed Exploration of the Dolinnoe and Obruchevskoe Deposits

Scope of Work	Year/Exploration Expenditure (US$)						Total US$
	1	2	3	4	5	6	
Pilot Drilling	160	-	-	-	-	-	**160**
Ventilation Shaft sinking	-	3,462	1,224	-	-	-	**4,686**
Obruchevskoe Exploration Shaft	-	209	1,220	400	-	-	**2,229**
Horizontal development on Obruchevskiy Level 22	-	-	-	-	435	880	**1,315**
Dolinnyi exploration shaft	-	1,040	183	-	-	-	**1,538**
Haulage crosscut on Dolinnoe and Obruchevskoe	-	-	-	418	560	560	**1,126**
TEO conditions and reserve estimation	-	-	-	-	-	115	**115**
Total	**160**	**5,111**	**2,627**	**818**	**1,415**	**2,261**	**12,392**

The Work Programme also includes conceptual mining plans for Dolinnoe and Obruchevskoe based on mining output capacities of 250,000tpy and 350,000tpy respectively. These capacities and all other parameters related to mining are subject to revision and renegotiation at the appropriate time.

To date, Kazzinc has drilled two pilot holes: 850m deep hole at the proposed site of the Dolinnoe exploration shaft and a 1,020m deep hole at the site of the Obruchevskoe exploration shaft.

8.4 Historical Reserve Estimates

8.4.1 Dolinnoe

Previous geological reserve estimates were made on completion of both stages of exploration, in 1994 (GKZ) and 1997 (Geolen).

8.4.1.1 Conditions

The reserve conditions on which the estimates were based were approved in December 1991 (Protocol No 2-t dated 30 December 1991) after a review of TEO by the State Committee produced to justify detailed exploration of the Dolinnoe deposit, and are summarised below:

- Sample cut-off grade= 3% Zn_{Eq};
- Minimum economic (commercial) grade in the block = 4.45% Zn_{Eq};
- The Zn_{Eq} formula used: Zn% + (Cu% x 0.7) + (Pb% x 0.7) + (Au g/t x 0.73) + (Ag g/t x 0.02);
- Grades below 0.12% Zn, 0.01% Cu, 0.03% Pb, 0.28g/t Au and 3.5g/t Ag are not convertible into Zn_{Eq};
- Minimum grade in intersections along the margins of the estimation blocks = 3.3% Zn_{Eq};
- Minimum thickness of ore = 1m, or appropriate grade x thickness values at thicknesses less than 1m;
- Maximum thickness of waste and low grade partings = 4m; and
- Minimum tonnage in isolated mineralised bodies (as below):

Content of Conditional Zinc %	Unit of Measurement	Reserves in Ore Zones and Blocks Isolated from the Main Ore Bodies		
		100m	150m	200m
5	kt	3.1	4.6	6.1
6	kt	1.5	2.2	2.9
8	kt	0.7	1.0	1.4

- Cut-off grade for off balance reserves: 2% Zn_{Eq};
- Zinc, lead, copper, gold, silver, cadmium and sulphur grades determined for the balanced and off-balance reserve blocks.

8.4.1.2 GKZ Reserves (1994/1995)

The Dolinnoe deposit was explored by core drilling from surface to a depth of 800m, the programme comprising 146 holes with a total depth of 70,309m.

The reserves were approved by the RK State Mineral Reserves Protocol No 26 dated 23.03.1995, confirming the provisional estimates of 01.08.1994 and are summarised in Table 8.12.

Table 8.12: Dolinnoe - Reserves Approved by GKZ KR (1995)

	Units	Balanced			Off Balance
		C_1	C_2	C_1+C_2	
Ore Reserves:	**kt**	**1029.4**	**1497.9**	**2527.3**	**949.3**
Copper	kt	6.0	4.8	10.8	1.4
Lead	kt	18.8	18.7	37.5	5.0
Zinc	kt	37.7	34.5	72.2	10.0
Gold	kg	10,887.4	9,329.9	20,217.3	2,175.8
Silver	t	147.9	118.9	266.8	30.5
Cadmium	t	-	252.7	252.7	-
Sulphur	'000t	-	88.7	88.7	-
Average Grade					
Copper	%	0.59	0.32	0.43	0.15
Lead	%	1.83	1.25	1.48	0.53
Zinc	%	3.67	2.30	2.86	1.05
Gold	g/t	10.58	6.23	8.00	2.29
Silver	g/t	143.72	79.38	105.57	32.13
Cadmium	%	-	0.01	0.01	-
Sulphur	%	-	3.51	3.51	-

8.4.1.3 TOO "Geolen" Reserves (1997)

Additional exploration of the high grade gold-polymetallic ores in the North-Eastern Lode was carried out during the period from 1994 to 1997 in accordance with Geological Assignment No. 99, when the drilling grid was reduced further to 20-25x25m.

The reserve classification was based on the drilling grid density; 20-30x25-30m - 40-50x50m were classified as C_1 with the remaining reserves (50-130x100-140m), allocated to the C_2 category, as were small isolated lenses.

The Geolen estimates are summarised in Table 8.13.

Table 8.13: Dolinnoe – Reserves (TOO «Geolen » 1997)					
	Units	**Balanced**			**Off Balance**
		C_1	**C_2**	**C_1+C_2**	
Ore Reserves	kt	945.82	1686.2	2632.02	949.3
Copper	kt	5.95	4.86	10.81	1.4
Lead	kt	19.81	19.63	39.44	5.0
Zinc	kt	38.92	38.01	76.93	10.0
Gold	kg	11358.8	9334.2	20693	2175.8
Silver	t	188.83	96.84	285.67	30.5
Average Grade					
Copper	%	0.63	0.29	0.41	0.15
Lead	%	2.09	1.16	1.50	0.53
Zinc	%	4.11	2.25	2.92	1.05
Gold	g/t	12.01	5.54	7.86	2.29
Silver	g/t	199.65	57.43	108.54	32.13

8.4.2 Obruchevskoe

The Obruchevskoe deposit was explored by drilling from surface on an initial exploratory grid of 50x70 m and 100x150 m during the same exploration programmes described for Dolinnoe.

8.4.2.1 Conditions

The RK State Mineral Resources Commission of the Ministry of Geology and Subsoil Resources protection approved temporary conditions for Obruchevskoe deposit ores by Protocol No. 8 dated 25.02.1994.

For balance reserves:

- The cut-off grade of the conditional zinc in samples – 4.0%;
- Minimal commercial content of conditional zinc in the evaluation block – 6.0%;
- Conversion factors to conditional zinc-zinc = 1.0; copper = 0.87; lead = 0.33. In the conversion to conditional zinc the following grades, zinc <0.2%, lead <0.1% and copper <0.03% are not considered;
- Minimum thickness of ore bodies included in the reserve evaluation = 3m; in the case of a lesser thickness and higher grade the appropriate grade x width equivalent is used. For each block, the average grade of the contained intersections are weighted by respective intersection lengths and the average grade of each intersection having been pre-estimated by the weighted average method (grade x sample length/intersection length) are calculated;
- Maximum acceptable thickness of waste partings and low grade material included in the ore body = 3.0m;
- A cut-off grade of 1% conditional zinc used for delineating off balance ores; and
- Zinc, lead, copper, gold, silver, cadmium and sulphur grades are to be determined for the balanced and off-balance reserve blocks.

8.4.2.2 GKZ Reserves (1995)

The reserves were not submitted for approval to the State Mineral Reserve Commission but recorded in the State Balance.

Table 8.14: Obruchevskoe – Reserves not Submitted for Approval to the SMR Commission (1995)					
		C_1	C_2	C_1+C_2	*Off-balance*
Ore Reserves	**kt**	**964.3**	**2,792.3**	**3,756.7**	-
Copper	kt	12.5	30.2	42.7	-
Lead	kt	51.7	62.1	113.8	-
Zinc	kt	123.8	194.3	318.1	-
Gold	kg	1,121	,1979	3,100	-
Silver	t	31.6	58.0	89.6	-
Average Grade					
Copper	%	1.30	1.08	1.14	-
Lead	%	5.36	2.22	3.03	-
Zinc	%	12.84	6.96	8.47	-
Gold	g/t	1.16	0.71	0.82	-
Silver	g/t	32.77	0.71	23.85	-

8.4.2.3 TOO "Geolen" Reserves (2002)

Introduction

The most recent reserve estimation was conducted by Geolen in 2001 using all previously generated drilling data, reported as at 01.01.2002.

Conditions

- Sample cut-off grade to define mineralised intercept intervals - 2.0% Zn_{Eq}, where Zn_{Eq} = 1.8xCu + Zn + 0.2xPb, with grades less than 0.06% Cu, less than 0.3% Zn and less than 0.1% Pb not being subject to conversion;
- Minimum grade in estimation block = 5% Zn_{Eq};
- Minimum thickness = 1m; in the case of a lesser thickness and higher grade the appropriate grade x width equivalent is used;
- Maximum acceptable thickness of waste partings and low grade material included in the ore body = 3.0m; and
- Cut-off grade for delineating off balance reserve = 0.9% Zn_{Eq}.

The conditions are of an interim character and have not been submitted for GKZ RK approval.

> ***WAI Comment:*** *The formula for the ZnEq is based on the metal recoveries reported for the Ridder-Sokolny ores and on outdated metal prices. The formula does not account for gold and silver, both of which significantly contribute to the Ridder-Sokolny mine revenue. The use of cut-off grades based solely on the base metals may have led to a reserve understatement.*

Database

The database was compiled in 1993-1995, and consists of over 9,000 core samples with copper, lead, zinc, gold and silver grades from 136 boreholes, including some holes completed in the area before the prospecting-assessment programme was initiated. It also includes complete records of downhole surveys.

Geological Interpretation

Geological interpretation was originally performed manually on cross sections, longitudinal sections and plans of the levels. They were based on primary drilling and geophysical data and present the key geological features such as borehole intersections and traces, stratigraphy, lithology, mineralised bodies, folds, intrusives, faults, orientation of contacts, etc.

The main weakness is that the perimeters of mineralised bodies are constrained by the reserve condition parameters which make the interpretation appear final. Widths and grades of borehole intersections are shown only for composited lengths within the perimeters of mineralised bodies, which makes alternative interpretation impossible without the substantial effort involved in searching for, and plotting sample results.

The interpretation is based on 688 intersections in 111 drillholes, comprising 279 intersections with weighted average grades of 2-10% Zn_{Eq} and 92 intersection with >10% Zn_{Eq} (balance reserves) and 317 intersections with 0.9-2% Zn_{Eq} (off balance reserves).

Estimation Method

Geolen used the method of geological blocks on horizontal projection. In this method mineralised bodies are subdivided into large blocks with boundaries defined by faults or other geological boundaries, grade drift or arbitrarily on the drilling density. Each block is assigned the average grade of the contained intersections weighted by respective intersection lengths, the average grade of each intersection having been pre-estimated by the weighted average method (grade x sample length/intersection length). The area of each block is estimated in plan view, multiplied by the average vertical length of the contained intersections to estimate the volume, which is multiplied by the bulk density to obtain the block tonnage.

Geolen performed the estimates using the MORE program developed by Almaty-based consulting firm Geoincentr.

Geolen divided the reserve into two groups:

- Reserves in geological blocks (as defined above); and
- Reserves in small blocks centred on single isolated borehole intercepts.

Altogether, there are 82 blocks, including 33 blocks with the average Zn_{Eq} grade equal to, or greater than 5% Zn_{Eq}, which according to the reserve conditions were classified as a balance resource.

The blocks were sorted into three groups corresponding to the grades defined in the reserve conditions; namely:

- Blocks 20-52 - balance reserve (>2% Zn_{Eq});
- Blocks 1-13 – high grade reserves (>10% Zn_{Eq}) estimated within the balance resource; and
- Blocks 101-138 - off balance reserve (0.9-2% Zn_{Eq}).

In addition, Geolen estimated tonnage and grades of gold-bearing mineralisation contained within the balance reserve. These estimates were subdivided into:

- Low grade gold-bearing reserve(0.5-2g/t Au);
- Average gold-bearing reserve(2g/t-16g/t Au); and
- High grade gold-bearing reserve (>16g/t Au).

Low sulphide, gold-bearing quartz veins were estimated separately outside Kazzinc's reserve conditions criteria, and block volumes were converted to tonnages using the bulk density regression formula.

Reserve Classification

Geolen based its reserve classification on drilling density. Reserves in blocks with a drilling grid of 30-50x50-60m were classified as C_1 category, and with a drilling grid of 50-70x60-100m classified as C_2. Reserves in small blocks assigned to single isolated mineralised intercepts were also classified as C_2 and reported separately to the reserve contained in large geological blocks.

The results reported by Geolen are presented in Table 8.15.

Table 8.15: Obruchevskoe – Reserves (TOO «Geolen » 01.01.2002)					
	Units	**C_1**	**C_2**	**C_1+C_2**	***Off-balance***
Ore Reserves:	**'000t**	**4,047.201**	**581.531**	**4,628.732**	**6,657.947**
Copper	'000t	45.114	6.474	51.587	25.456
Lead	'000t	162.013	11.563	173.576	30.605
Zinc	'000t	388.958	26.562	415.520	94.874
Gold	kg	7,298.8	441.1	7,739.9	1,691.9
Silver		221.409	104.012	325.421	83.107
Cadmium	t	-	-	1455.3	-
Antimony	t	-	-	560	-
Arsenic	t	-	-	1816	-
Selenium	t	-	-	41.6	-
Molybdenum	t	-	-	225	-
Pyrite	'000t	-	-	490.342	-
Average Grade					
Copper	%	1.11	1.11	1.11	0.38
Lead	%	4.0	1.99	3.75	0.46
Zinc	%	9.61	4.57	8.98	1.42
Gold	g/t	1.8	0.76	1.67	0.25
Silver	g/t	54.71	178.86	70.3	12.48
Cadmium	%	-	-	0.03144	-
Antimony	%	-	-	0.0121	-
Arsenic	%	-	-	0.03924	-
Selenium	%	-	-	0.0009	-
Molybdenum	%	-	-	0.00529	-
Pyrite	%	-	-	4.5116	-

8.4.3 Mining Reserves

The mining reserves (commodity ore) used in the June 2008 TEO life of mine and other projections are derived from the geological reserves and the application of loss and dilution factors specific to each deposit (Table 8.16). By back calculation the net increases in the geological reserve comprise 11.7% and 12.7% for Dolinnoe and Obruchevskoe respectively.

Table 8.16: Dolinnoe & Obruchevskoe-Commodity Reserves of Ore and Metals Accepted for the Final Feasibility Study [TEO]						
	Tonnage	**Grade/Metal Content**				
		Zn ***%***	**Pb** ***%***	**Cu** ***%***	**Au** **g/t**	**Ag** **g/t**
Obruchevskoe						
Ore (t)	**5,214,300**	7.45	3.11	0.92	1.39	58.35
Metal (t)		388,511	162,293	48,234	7,237kg	304,268kg
Dolinnoe						
Ore (t)	**2,883,700**	2.54	1.30	0.36	6.82	94.20
Metal (t)		73,160	37,507	10,280	19,679kg	271,672kg
Total						
Ore (t)	**8,098,000**	5.70	2.47	0.72	3.32	71.12
Metal (t)		461,671	199,800	58,514	26,916kg	575,940kg

8.5 Current Mineral Resource Estimate (2011)

8.5.1 Dolinnoe & Obruchevskoe

8.5.1.1 Introduction

The mineral resource estimate presented here for the Dolinnoe ore zone in the Ridder-Sokolnoe deposit, is based on the model prepared by Kazakhstan Mineral Company (KMC) in January and February 2011. The mineral resource model considers drilling available to this date and the effective date of this resource estimate is 02/02/2011.

8.5.1.2 Database Compilation

The sample database was supplied by Kazzinc in Datamine Format. Checks were made for overlapping or duplicate samples and the database was found to be in good order. The database consists of 224 surface drillholes. Table 8.17 below lists the number of samples and Table 8.18 the basic statistics of the global database. The samples were analysed for Au, Ag, Cu, Pb and Zn.

Table 8.17: Summary of Assay Data

Sampling Method	Number of Holes	Total Metres	Average Hole Length	Number of Assays
Drillholes	224	152,445.7	680.6	10,858

Table 8.18: Basic Global Statistics

Element	Number Of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	CV
AU (g/t)	10858	0.01	213.80	1.16	35.50	5.96	5.14
AG (g/t)	10858	0.01	7187.00	15.43	20289.77	142.44	9.23
CU (%)	10858	0.00	6.50	0.08	0.09	0.30	3.99
PB (%)	10858	0.01	20.10	0.27	0.93	0.96	3.56
ZN (%)	10858	0.01	32.90	0.51	3.53	1.88	3.69

8.5.1.3 Domaining

Mineralised zone wireframes were created to a cut-off grade of 1.7% Zinc Equivalent (ZnEq) where:

ZnEq= 1.02*Pb + 3.15*Cu + 1.66*Au +0.03*Ag + Zn

Where grades of less than 0.03% Pb, 0.01% Cu, 0.28g/t Au, 3.5g/t Ag and 0.12% Zn were not converted. The minimum thickness of mineralised intercept is 2m and the maximum included thickness of barren or below cut-off grade samples is 2m. Examples are shown in Figure 8.7 of a plan view and an isometric view respectively. There are 2 ore zones within the deposit; Upper and Lower. Ore bodies from Upper Ore Zone are bedded, Lower Ore Zone bodies are steeply dipping to south-west.

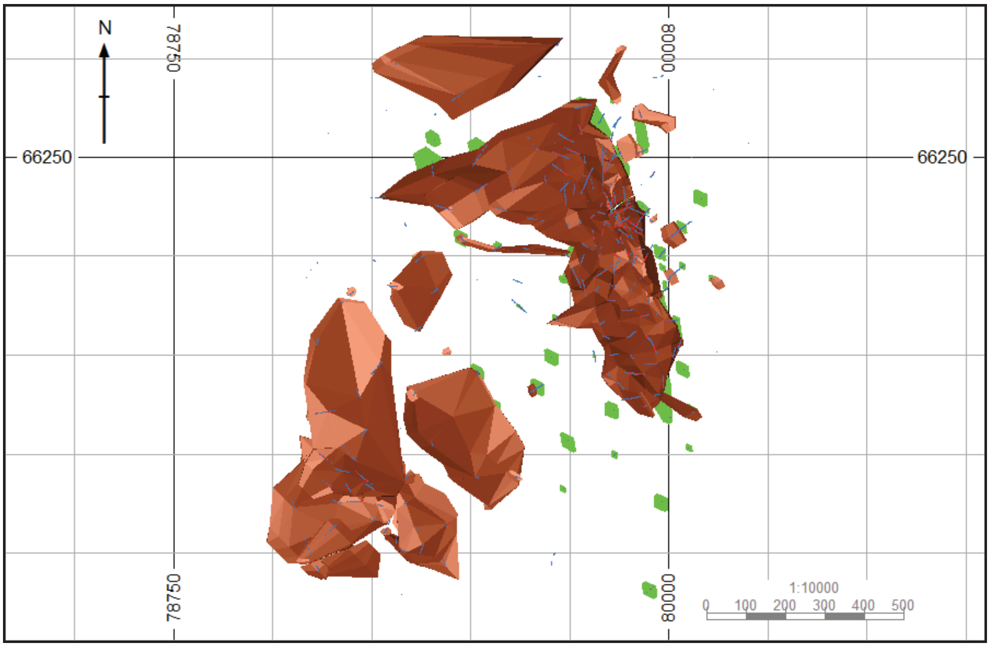


Figure 8.7: Plan View With Upper (Brown) and Lower (Green) Mineralised Zones

8.5.1.4 Global Geostatistical Analysis

Samples were selected within the mineralised zone wireframes for further processing. The basic statistics of those samples from within the mineralised zones are listed in Table 8.19.

Table 8.19: Basic Statistics of Selected Samples

Element	Number Of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	CV
AU (g/t)	3383	0.01	213.80	3.29	107.25	10.36	3.15
AG (g/t)	3383	0.01	7187.00	43.94	63912.31	252.81	5.75
CU (%)	3383	0.01	6.50	0.20	0.26	0.51	2.59
PB (%)	3383	0.01	20.10	0.73	2.57	1.60	2.20
ZN (%)	3383	0.01	32.90	1.40	10.03	3.17	2.26

The populations of each element are roughly log-normal with a positive skew. Each of the elements has a small high grade outlier population. To identify the need for top cuts, the log probability plots and quantile distribution for each domain were studied to identify the level of any outlier values. Top cuts were applied to these outliers in order to reduce any undue influence during grade estimation. Values above the top cut value are reduced to that value. These values were initially chosen as points on log probability plots where the sample population distribution changed. Assays above this level were checked to make sure that, spatially, the samples to be top cut were not clustered forming a distinct high grade zone. For Au a separate higher grade domain was given a higher top cut level. A summary of this top cutting is given in Table 8.20.

Table 8.20: Top Cutting Summary				
Element	Cutting Level	No Samples Cut	Average Grades	
			Before Top Cutting	After Top Cutting
Au for ore body 1-1	98	4	6.29	5.81
Au for other ore bodies	55	9	2.59	2.53
Ag	2254	12	43.94	39.45
Cu	3.97	19	0.20	0.19
Pb	11.25	25	0.73	0.71
Zn	26.3	13	1.40	1.39

8.5.1.5 Variography

Experimental variograms were calculated and models fitted for the Dolinnoe ore zone in Micromine using the composited samples for Au, Ag, Cu, Pb and Zn in both the Upper and Lower ore zones. Figure 8.8 shows the along strike and down dip variogram models for Zn in the Lower ore zone.

8.5.1.6 Block Modelling

Block models were created within each of the mineralised zone wireframes. A parent cell size of 5m x 10m x 5m was selected. Sub cell splitting to a minimum block size of 1m x 2m x 1m in the X, Y and Z directions.

8.5.1.7 Density

A Regression formula was used post grade estimation to calculate density in to the block models based on metal content. The formula for bulk density used at Dolinnoe is:

Bulk Density = 2.8 + 0.047*Pb + 0.014*Cu + 0.013*Zn.

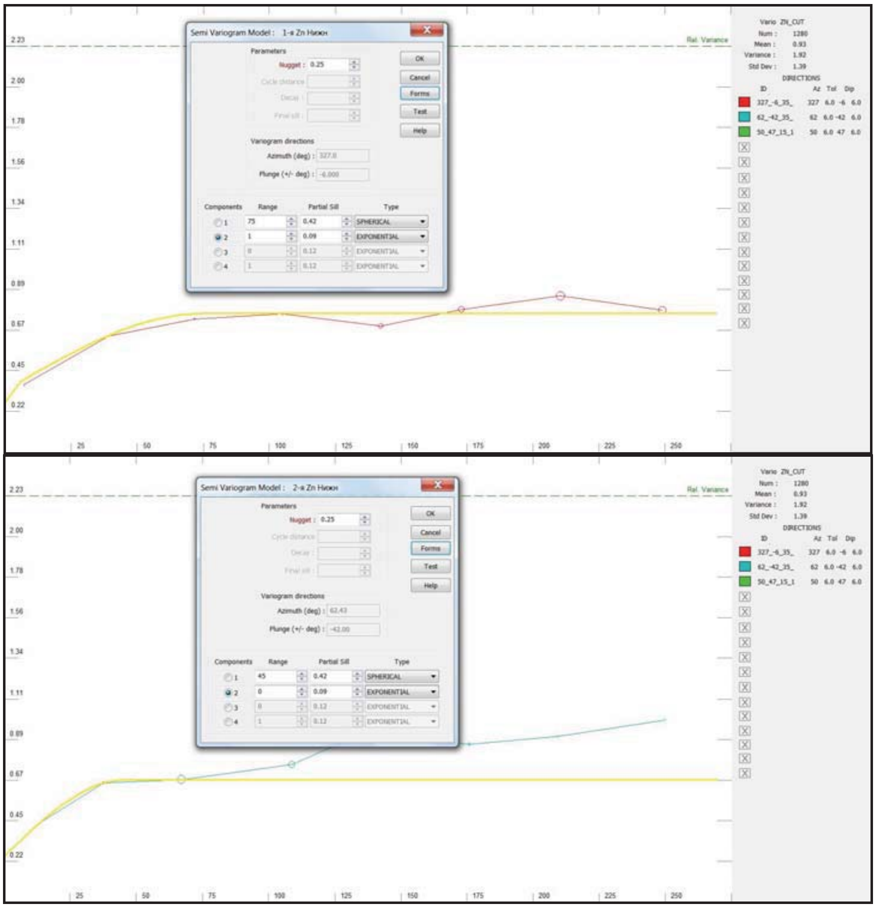


Figure 8.8: Along strike (Top) and down dip (Bottom) variogram models for Zn in the Lower Ore Zone.

8.5.1.8 Grade Estimation

Grade estimation was carried out using Ordinary Kriging (OK) as the principal interpolation method with Inverse Power of Distance (IPD) used for comparative purposes. Grades were estimated in to the block model using a zonal control based on domain. Grades were estimated into each individual subcell. Sample weighting during grade estimation was determined by the variogram model parameters for the OK method. Cell discretisation was set at 2 x 2 x 2. A multiple pass approach was used with up to three expansions of the search ellipse if minimum conditions were not met in the first or second ellipse. The first search ellipse is equal in size to 2/3 of the variogram ranges as outlined above. The second ellipse was equal in size to the range of the variograms and the third equal to three times the variogram range. The key estimation parameters are listed below in Table 8.21 for the Upper ore zone and in Table 8.22 for the Lower ore zone.

Post estimation a zinc equivalent grade (EqvZn) is calculated in to the block models from the estimated grades using the formula;

$$EqvZn = 1.02*Pb + 3.15*Cu + 1.66*Au + 0.03*Ag + Zn.$$

Table 8.21: Estimation Parameters for Upper Ore Zone of Dolinnoe deposit (KMC 2011)						
Ellipse	Parameter	Au	Ag	Cu	Pb	Zn
1st	Search Radii 1 (m) – across strike	41	52	23	23	24
	Search Radii 2 (m) – along strike	48	27	20	20	20
	Search Radii 3 (m) – down dip	20	3	5	6	5
	Minimum Composites	4	4	4	4	4
	Minimum Drill Holes	2	2	2	2	2
2nd	Search Radii 1 (m) – across strike	61	78	35	38	36
	Search Radii 2 (m) – along strike	72	40	30	30	30
	Search Radii 3 (m) – down dip	30	5	7	9	7
	Minimum Composites	2	2	2	2	2
	Minimum Drill Holes	2	2	2	2	2
3rd	Search Radii 1 (m) – across strike	183	234	105	114	108
	Search Radii 2 (m) – along strike	216	120	90	90	90
	Search Radii 3 (m) – down dip	90	15	21	27	21
	Minimum Composites	1	1	1	1	1
	Minimum Drill Holes	1	1	1	1	1

Table 8.22: Estimation Parameters for Lower Ore Zone of Dolinnoe deposit (KMC 2011)						
Ellipse	Parameter	Au	Ag	Cu	Pb	Zn
1st	Search Radii 1 (m) – across strike	39	47	32	47	51
	Search Radii 2 (m) – along strike	17	44	53	23	30
	Search Radii 3 (m) – down dip	15	19	32	13	10
	Minimum Composites	4	4	4	4	4
	Minimum Drill Holes	2	2	2	2	2
2nd	Search Radii 1 (m) – across strike	58	70	48	70	76
	Search Radii 2 (m) – along strike	26	66	80	35	45
	Search Radii 3 (m) – down dip	22	29	48	19	15
	Minimum Composites	2	2	2	2	2
	Minimum Drill Holes	2	2	2	2	2
3rd	Search Radii 1 (m) – across strike	174	210	144	210	228
	Search Radii 2 (m) – along strike	78	198	240	105	135
	Search Radii 3 (m) – down dip	66	87	144	57	45
	Minimum Composites	1	1	1	1	1
	Minimum Drill Holes	1	1	1	1	1

8.5.1.9 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 interpolated model grades.

8.5.1.10 Resource Classification

The Mineral Resource estimates for Dolinnoe set out below are classified in accordance with the guidelines of the JORC Code (2004)

The basis of the definition of the resource categories was the pass used in estimating blocks. Essentially this gives approximate drillhole spacings for the allocation of resources which can be summarised as 20m x 4m for *Measured* resources (along strike and down dip), 36m x 7m for Indicated resources and 108m x 21m for *Inferred* resources. Figure 8.9 is a plan view of Dolinnoe showing JORC classification.

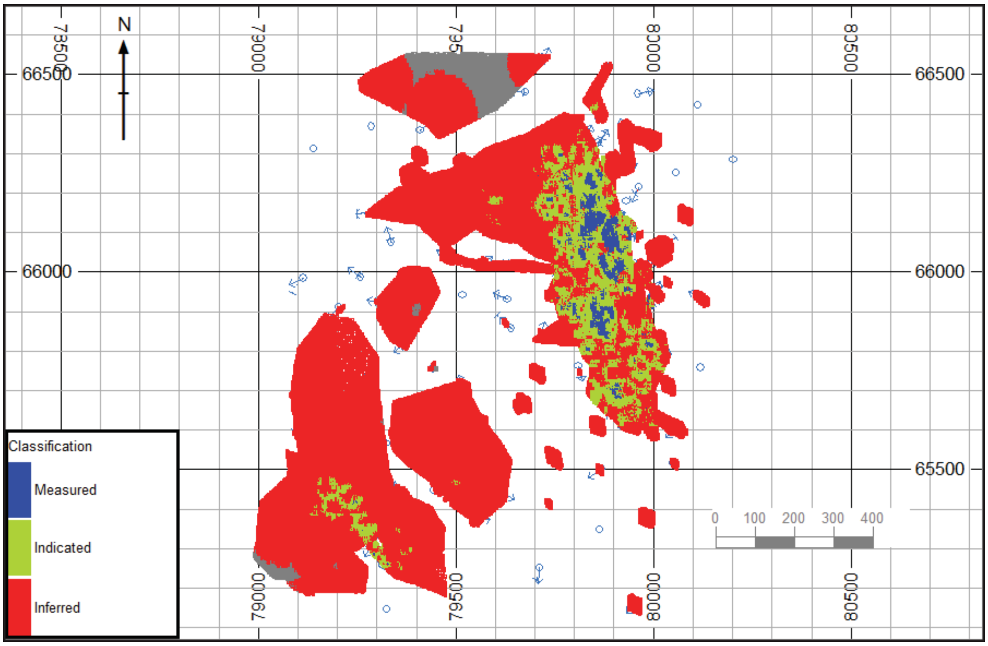


Figure 8.9: Plan view of JORC Classification at Dolinnoe.

8.5.1.11 Resource Evaluation

Grade-tonnage information for Dolinnoe is summarised in Table 8.23 for total in-situ resources at a cut-off grade of 1.7% EqvZn. The grades and classifications are reported as estimated by OK method with an effective date of 01 January 2011.

Table 8.23: Dolinnoe Polymetallic Deposit Total In-situ Resources - For All Orebodies at a cut off grade of 1.7% EqvZn (In Accordance with the Guidelines of the JORC Code (2004)) (WAI 01.01.2011)

Classification	Tonnage (kt)	Density	Pb		Zn		Cu		Au		Ag		EQV
			%	kt	%	kt	%	kt	g/t	oz	g/t	oz	%
Measured	5,043	2.86	0.74	37.32	1.39	70.10	0.20	10.09	3.85	624,252	50.47	8,183,378	10.65
Indicated	2,707	2.84	0.48	12.99	1.00	27.07	0.14	3.79	2.32	201,879	28.05	2,440,821	6.60
Measured +Indicated	7,750	2.85	0.65	50.31	1.25	97.17	0.18	13.88	3.32	826,131	42.64	10,624,199	9.24
Inferred	6,907	2.84	0.48	33.15	0.86	59.40	0.12	8.29	1.59	353,085	15.88	3,526,405	4.84

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.

8.5.2 Obruchevskoe

8.5.2.1 Introduction

The mineral resource estimate presented here, is based on the model prepared by KMC and is in accordance with the guidelines of the JORC Code (2004) and considers drilling carried out as at 01.01.2011.

8.5.2.2 Database Compilation

The sample database was supplied by Kazzinc in Datamine format, and consists of 134 surface drillholes. Checks were made for overlapping or duplicate samples and the database was found to be in good order. Table 8.24 lists the number of samples and Table 8.25 the basic statistics of the database.

Table 8.24: Summary of Assay Data

Sampling Method	Number of Holes	Total Metres	Average Hole Length	Number of Assays
Drillholes	134	153,380.5	1,144.6	5,122

Table 8.25: Basic Global Statistics

Element	Number Of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation	CV
AU (g/t)	5,122	0.00	71.20	0.59	8.86	2.98	5.04
AG (g/t)	5,122	0.00	5,019.20	18.94	12,055.61	109.80	5.80
CU (%)	5,122	0.00	11.30	0.33	0.60	0.78	2.39
PB (%)	5,122	0.00	26.80	0.99	10.24	3.20	3.24
ZN (%)	5,122	0.00	43.70	2.33	37.73	6.14	2.64

8.5.2.3 Domaining

Mineralised zone wireframes were created to a cut-off grade of 1.7% Zinc Equivalent (Zn_{Eq}) where:

Zn_{Eq}= 0.94xPb + 2.78xCu + 1.13xAu +0.02xAg + Zn

The minimum thickness of mineralised intercept is 1m and the maximum included thickness of barren or below cut-off grade samples is 3m. Eight major zones were domained in the process with two more consisting of smaller lenticular bodies (see Figure 8.10). Due cognisance was taken of the geological interpretation to limit the mineralised domains by the interpretation of the position of the host unit.

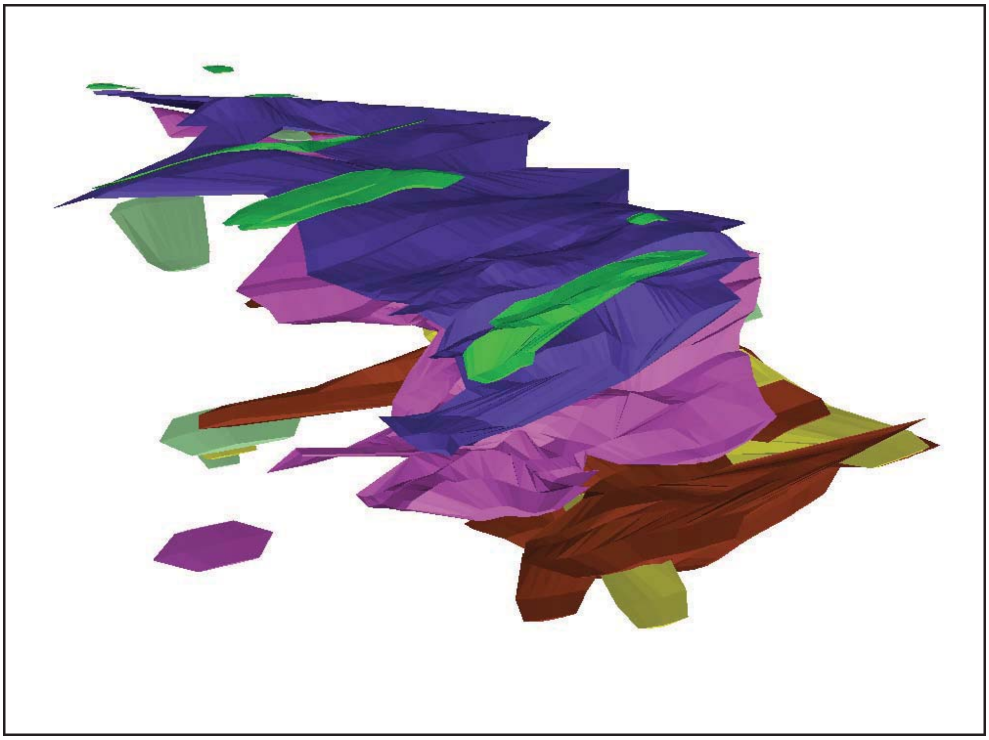


Figure 8.10: Isometric View Looking North East of Mineralised Zones

8.5.2.4 Global Geostatistical Analysis

Samples were selected within the mineralised zone wireframes for further processing. The basic statistics of those samples from within the mineralised zones are listed in Table 8.26.

Table 8.26: Basic Statistics of Selected Samples							
Element	**Number Of Samples**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**CV**
AU (g/t)	1,936	0.00	71.20	1.25	21.13	4.60	3.69
AG (g/t)	1,936	0.00	5,019.20	44.25	30,609.95	174.96	3.95
CU (%)	1,936	0.00	11.30	0.76	1.21	1.10	1.45
PB (%)	1,936	0.00	26.80	2.46	23.37	4.83	1.96
ZN (%)	1,936	0.00	43.70	5.74	80.02	8.95	1.56

The populations of each element are approximately log-normal with a positive skew. Each of the elements has a small high grade outlier population.

A composite length of 1.5m was chosen for further processing in order to give a consistent level of support for geostatistical analysis.

The same approach for top cuts was applied as for Dolinnoe.

A summary of this top cutting is given in Table 8.27.

Table 8.27: Top Cutting Summary				
Element	Cutting Level	No Samples Cut	Average Grades	
			Before Top Cutting	After Top Cutting
Au	15	34	1.27	0.97
Ag	346.5	36	39.74	32.16
Cu	5.2	10	0.75	0.74
Pb	20.2	10	2.26	2.25
Zn	36.8	9	5.43	5.41

8.5.2.5 Variography

A single set of experimental variograms were calculated and models fitted for the Obruchevskoe ore zone in Micromine® using the composited samples for Au, Ag, Cu, Pb and Zn. Figure 8.11 shows the along strike and down dip variogram models for Zn.

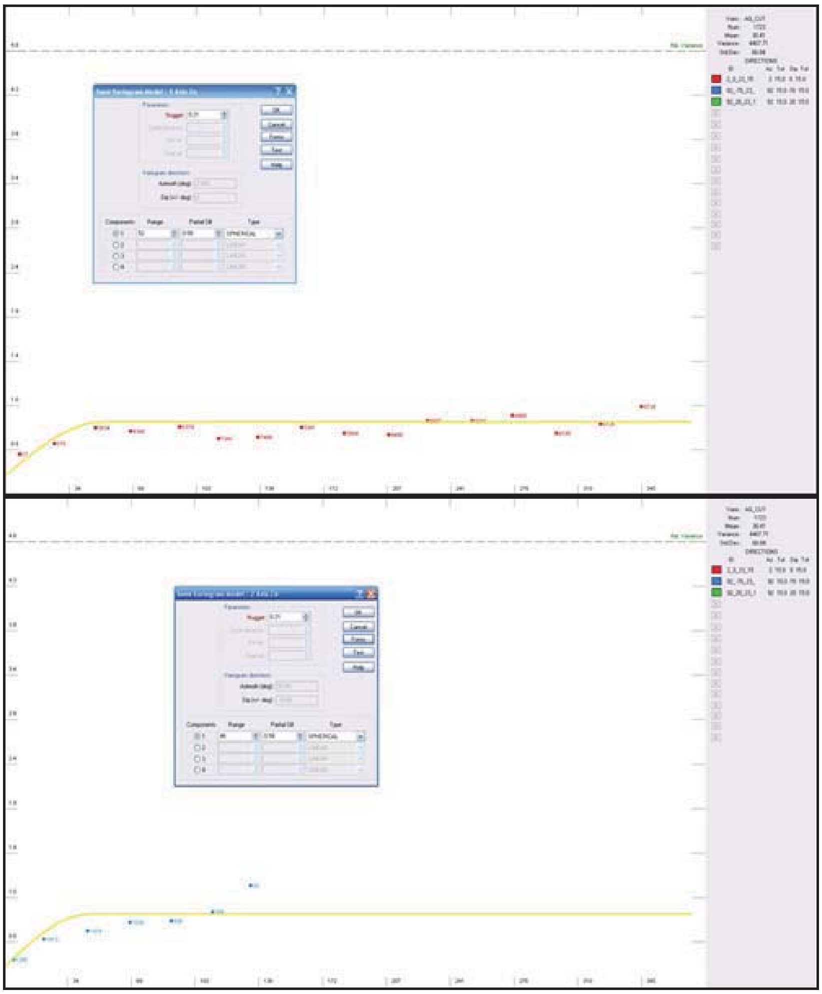


Figure 8.11: Along strike (Top) and down dip (Bottom) variogram models for Zn

8.5.2.6 Block Modelling

Block models were created within each of the mineralised zone wireframes. A parent cell size of 15m x 15m x 10m was selected. Sub cell splitting to a minimum block size of 3m x 3m x 2m in the X, Y and Z directions.

8.5.2.7 Density

A Regression formula was used post grade estimation to calculate density into the block models based on metal content. The formula for bulk density used at Obruchevskoe is:

Bulk Density = 2.743429 + 0.069044xPb + 0.022552xCu + 0.017698xZn.

8.5.2.8 Grade Estimation

A similar approach to grade estimation as for Dolinnoe was applied. The key estimation parameters are listed below in Table 8.28.

Post estimation a zinc equivalent grade (EqvZn) is calculated in to the block models from the estimated grades using the formula;

EqvZn = 0.94xPb + 2.78xCu + 1.13xAu +0.02xAg + Zn.

Table 8.28: Estimation Parameters for Obruchevskoe

Ellipse	Parameter	Au	Ag	Cu	Pb	Zn
1st	Search Radii 1 (m) – across strike	51	47	43	23	31
	Search Radii 2 (m) – along strike	37	62	37	41	35
	Search Radii 3 (m) – down dip	29	20	29	36	26
	Minimum Composites	9	9	9	9	9
	Maximum Composites	16	16	16	16	16
	Minimum Drill Holes	3	3	3	3	3
2nd	Search Radii 1 (m) – across strike	77	70	64	35	46
	Search Radii 2 (m) – along strike	55	93	56	61	52
	Search Radii 3 (m) – down dip	43	30	44	54	39
	Minimum Composites	6	6	6	6	6
	Maximum Composites	12	12	12	12	12
	Minimum Drill Holes	2	2	2	2	2
3rd	Search Radii 1 (m) – across strike	154	140	128	105	115
	Search Radii 2 (m) – along strike	110	186	112	183	130
	Search Radii 3 (m) – down dip	86	60	88	162	98
	Minimum Composites	4	4	4	4	4
	Maximum Composites	8	8	8	8	8
	Minimum Drill Holes	1	1	1	1	1

8.5.2.9 Validation

The same approach to model valuation was followed as for Dolinnoe.

8.5.2.10 Resource Classification

The Mineral Resource estimates for Obruchevskoe set out below are classified in accordance with the guidelines of the JORC Code (2004).

The basis of the definition of the resource categories was the pass used in estimating blocks. Essentially this gives approximate drillhole spacings for the allocation of resources which can be summarised as 35m x 31m for measured resources (along strike and down dip), 52m x 46m for Indicated resources and 130m x 115m for inferred resources. Figure 8.12 is an isometric view of Obruchevskoe showing JORC classification.

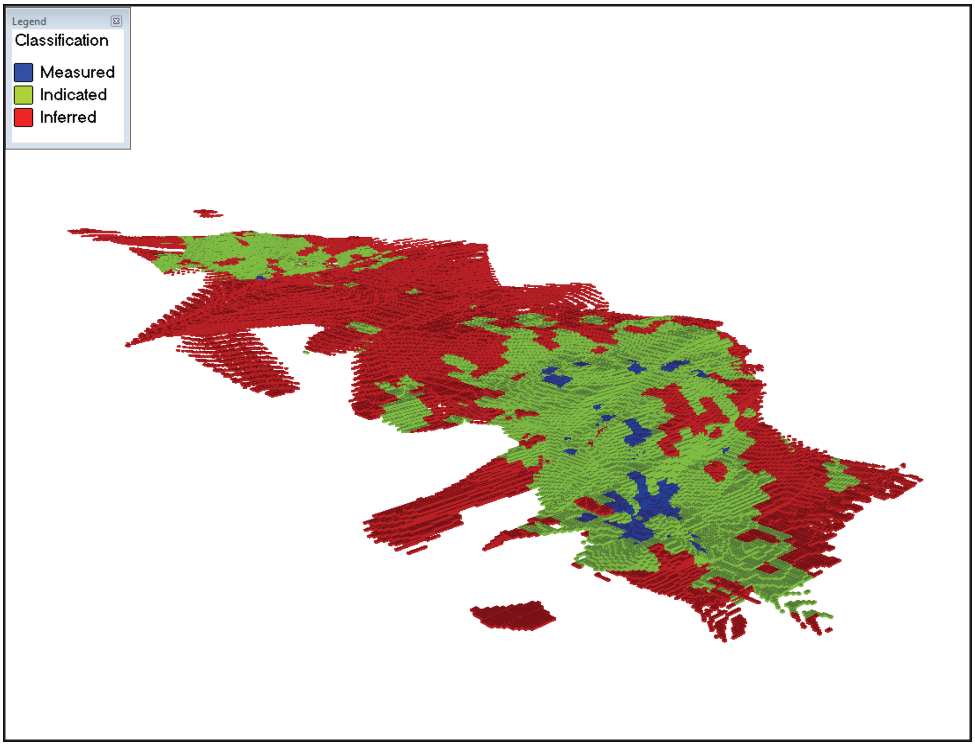


Figure 8.12: Isometric View to the North East view of JORC Classification at Obruchevskoe

8.5.2.11 Resource Evaluation

Grade-tonnage information for Obruchevskoe is summarised in Table 8.29 for total in-situ resources. The grades and classifications are reported as estimated by OK method to a cut off grade of 1.7% EqvZn with an effective date of 9 February 2011 and in accordance with the guidelines of the JORC Code (2004).

**Table 8.29: Obruchevskoe Mineral Resource Estimate
For All Orebodies at a Cut off Grade of 1.7% EqvZn
(WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))**

Classification	Tonnage (kt)	Density	Zn		Pb		Cu		Au		Ag		EQV
			%	t	%	t	%	t	g/t	oz	g/t	oz	%
Measured	1,154	3.20	8.87	102,360	4.02	46,391	0.88	10,155	1.62	60,105	40.68	1,509,308	17.75
Indicated	7,783	2.96	4.64	361,131	1.78	138,537	0.73	56,816	0.67	167,654	25.36	6,345,814	9.61
Measured +Indicated	8,937	2.99	5.18	462,937	2.07	184,996	0.75	67,028	0.79	226,992	27.34	7,855,636	10.66
Inferred	5,500	2.83	1.75	96,250	0.64	35,200	0.41	22,550	0.50	88,415	24.97	4,415,423	4.56

8.6 Mining

8.6.1 Production Rate

8.6.1.1 Dolinnoe

Productivity of the mine has been based on the number of stopes in production at the any one time on each level and in each lode. The production rate is calculated on the assumption that one stope can be mined from the South-West Lode at a rate of 80ktpa and three stopes in operation from the North-East Lode at a rate of 50ktpa. Therefore the productivity of the mine will be 250ktpa.

8.6.1.2 Obruchevskoe

The production rate is calculated on the assumption that two stopes can be mined from the upper levels at a rate of 120ktpa and one stope in operation from the lower levels at a rate of 110ktpa. Therefore the calculated production rate will be 350ktpa. The combined Dolinnoe-Obruchevskoe production rate will be 600ktpa.

8.6.2 Life of Mine

For the Dolinnoe mine, the mine life is expected to be 11.5 years based on the reserve of 2.88Mt being mined at 250ktpa and for Obruchevskoe, the mine life will be 14.9 years based on 5.21Mt being mined at 350ktpa.

Taking into account the time required for mine development, the production overlap and the winding down of the mines, the mine life of the Dolinnoe-Obruchevskoe mine is expected to be 16 to 18 years.

8.6.3 Mine Design

The development of the Dolinnoe and Obruchevskoe deposits has been determined taking into account the following mining, geological, geographical and engineering conditions:

- The two deposits are only 2km apart;
- The depth of the ore bodies occur at 600 to 700m;
- Surface topography, depth of cover over the deposit, of thick mass of loose sediments and bed of the flowing Bystrukha river, flowing into Bystrushinskoye water reservoir;
- Spatial arrangement of main (in terms of reserves) gently dipping and steeply pitching ore chutes and bodies; and
- Annual productivity of the mine by extraction of ore and its value.

In local and international mines, where the working levels are anticipated to be at a depth of 800m and beyond, the mine is developed using two vertical shafts and two hoisting systems.

The Feasibility Study proposed to develop Dolinnoe and Obruchevskoe deposits using vertical shafts. The efficiency of combining the two deposits has been confirmed by a technical and economic assessment into the feasibility of developing, supplementary exploration and drilling of Dolinnoe and Obruchevskoe deposits performed by "Republic of Kazakhstan mining-and-processing integrated works" in 2004. Taking into account the combined development of both deposits, Kazzinc concluded with the Competent body of the Republic of Kazakhstan for conducting exploration and production of zinc, lead, copper, gold and silver at Dolinnoe and Obruchevskoe deposits, Registration number 2450 dated 20 August 2007.

8.6.3.1 Mine Design Options

Several project designs have been undertaken to date.

The 2008 Feasibility Study considered three scenarios for accessing the Dolinnoe and Obruchevskoe deposits:
Option 1: Three shafts from surface independent of Ridder-Sokolniy mine;

Option 2, 2A and 2B: Opening jointly with Ridder-Sokolniy mine; and
Option 3: Using two shafts from surface but independent of Ridder-Sokolniy mine.

Following an economic evaluation, Kazzinc selected Option 3 as this provided the best NPV and Internal Rate of Return (IRR) as well as lowest discounted payback period. As Option 3 was developed from Option 1, the two options are described below.

Option 1

This option involves opening of both deposits with three shafts, Dolinnoe and Obruchevskoe vertical shafts and a ventilation shaft which will service both mines. The depth of vertical shafts has been determined by depths of the ore bodies within deposits; Dolinnoe deposit from +420 to +250 level for the North-East orebody and from +350 to +250m level for the South-West orebody and at Obruchevskoe deposit from –20 to +130 level.

The ventilation shaft is to be driven from surface to –40 level, the Dolinnoe shaft to +250 level horizon and the Obruchevskoe shaft to -20 level. The height between the levels has been designed at 50m to allow for the development and mining activities on the different levels from both mines simultaneously. The stopes are expected to be either vertical or steeply dipping with varying widths ranging from 1.5 – 10m.

The ventilation shaft is to be 7.3m in diameter and equipped with one cage winding and a double-skip hoist facility. It will be designated for the movement of people, materials and equipment, and for the delivery of mined rock and supply of fresh air. The total depth of the shaft is 1,000m. A drift will developed at the +890 level so that when the rail cars are hoisted to surface, they are hauled to the unloading facility at the end of the drift and returned underground.

The Dolinnoe and Obruchevskoe shaft will be 5m in diameter and a depth of 640m from surface to +250 level, and 960m from surface to the -20m level, respectively. The shafts will be equipped to hoist materials and small mobile equipment and for the emergency winding of personnel.

The inclined Dolinnoe deposit will be opened by three main horizons; +350, +300 and +250 levels to access reserves in the North-East and South-West orebodies. There is also the +440m and +150m and +100m levels dedicated to the separate orebodies. The +440 level will serve as an air return level. All levels passing through both the North-East and South-West orebodies will be connected via internal decline, hoisting and ventilation shafts.

The +250m and +100m level will be a main loading and transportation drifts where the electric haulage equipment, ore and waste-passes at 150-200m intervals will be located.

There will be four levels developed in the Obruchevskoe mine, the +130m, +80m, +30m and the -20m levels. The levels will be connected via an internal decline, hoisting and ventilation shafts. The -20m horizon will be the main loading and transportation drift and will be equipped with electric haulage. Ore and waste passes on this level will be located at 150-200m intervals to optimise the movement of material along this level to the ventilation shaft for hauling to the surface.

The broken rock from the levels will be delivered to the sectional ore and waste passes and then to the main loading facility and transported to the ventilation shaft using electric haulage. Waste is hoisted to the +890m level where it is taken and discharged on surface. Ore is moved to the bottom of the mine via ore passes to the -40m level where it is fed into skips and hoisted to surface. From there it is tipped into an ore storage bin and then moved by conveyor to an ore stockpile on the surface and delivered to the Ridder plant by road transport.

Option 3

Option 3 is the preferred Option and is similar to Option 1 in that the ore is hoisted to surface and moved by road transport to the Ridder plant. The exception is that there are only two shafts. The ore is moved underground from both mines to the bottom of the Obruchevskoe mine and hoisted up the Obruchevskoe shaft.

The differences between Option 3 and Option 1 are:

- Removal of one cage winding shaft. The supply of fresh air and the hoisting of ore to the surface will be performed through Obruchevskoe shaft with ore handling facilities being constructed near this shaft;
- Fresh air for the Dolinnoe mine will travel along a dedicated drive from the Obruchevskoe shaft on the +100m level. It will be 1,370m in length with a cross sectional area of 10m^2;
- Return air from the Obruchevskoe mine, will pass along a air crosscut which will be built at +130 level for 3,530m with a 12m^2 section to connect to a exhaust raise up to the +470 level and further on through bleeding pit onto the surface; and
- For delivery of contaminated air from North-West orebody of Dolinnoe deposit at the rate of 30m^3/s, an exhaust raise will be constructed at the main workings at +470 level and further on through dedecated raises to the surface. Contaminated air from South-West orebody will be delivered along the Dolinnoe shaft at the rate of 40m^3/s.

In this option, ore from the Dolinnoe mine is transported to Obruchevskoe shaft at +100m level and travels through the ore pass system to the -20m level. Ore from the Obruchevskoe mine also moves through the internal ore pass system to the –20m level for loading into skips, hoisting to surface and is delivered to the concentrating plant by road transport.

8.6.4 Mine Scheduling

The mine scheduling for Option 3 is based on the geological cut-off grade of the Dolinnoe and Obruchevskoe deposits and envisages an ore production rate of 600ktpa as shown in Table 8.30. The mine reserve for Dolinnoe is 2,883,677t and for Obruchevskoe is 5,214,258t.

Table 8.30: Mine Schedule for Option 3 (ktpa)

	Total	2014	2015	2016	2017	2018	2019	2026	2027	2028	2029	2030	2031	2032	2033
Option 3															
Obruchevskoe deposit	**5,214,258**			70	350	350	350	350	350	350	350	350	150	94.3	
Dolinnoe deposit	**2,883,677**	100	250	250	250	250	250	33.7							
Total for Dolinnoe-Obruchevskoe mine	**8,097,936**	100	250	320	600	600	600	383.7	350	350	350	350	150	94.3	

8.6.5 Mining Method

When selecting the mining methods, the following mining and technical factors were taken into consideration:

- The depth of the ore bodies - 450-670m for Dolinnoe and 800-1,000m for Obruchevskoe;
- Mine safety;
- Inclination of ores and rocks of deposits in relation to seismic events;
- High ore value; and
- The need to preserve surface integrity at Dolinnoe with the Bystrushinskoye water reservoir above the mine.

Taking all of these mining and technical conditions into account, the extraction of ore bodies at Dolinnoe will be using the chamber mining method with backfill, and for Obruchevskoe chamber mining with caving. During the development of deposits, it is planned to use both conventional mobile equipment, if thickness of ore bodies is from 1 to 3m, and self-propelled equipment if the thickness of the ore bodies is greater than 3m. The technical and economic parameters are shown in Table 8.31.

Table 8.31: Summary of Technical & Economic Parameters

Indicator	Unit	Deposit, method	
		Dolinnoe, chamber with backfill	Obruchevskoe, chamber-solid block with caving
Thickness of ore bodies	m	1-18	15-25
Angle of inclination	degrees	0-20	0-50
Density of ore	tons/m^3	2.8	2.8
Density of rock	tons/m^3	2.7	2.7
Rock, ore hardness ratio	-	4-12	4-12
Ratio of fragmentation	-	1.5	1.5
Standard quality block	mm	400x400	400x400
Specific gravity of the system	%	100%	100%
Annual output - total	kt	250.0	350.0
Annual amount of preparatory works	m^3	15,000	12,000
Annual amount of first workings	m^3	9,000	28,000
Specific volume of mining preparatory works per 1,000 tons	m^3	60.0	37.0
Specific volume of stoping works per 1,000 tons	m^3	36.0	20.0
Losses	%	4.9	6.5
Dilution	%	13.2	17.0

8.6.6 Services

8.6.6.1 Supply of Compressed Air

The air supply to both Dolinnoe and Obruchevskoe mines will be from portable compressors with 3 compressors envisaged for Dolinnoe and 5 compressors for Obruchevskoe.

8.6.6.2 Water Supply

The water supply for drilling, washing of mine faces, fire fighting purposes and additional needs for both mines will be sourced from the Bystrushinskoye water reservoir.

8.6.6.3 Heat Supply

The demands for heating energy in relation to Option 3 are expected to be about 1.0MW, excluding administrative and household facilities.

The heating energy demand from the backfill plant is the same regardless of which option is used. It is expected that 0.372MW will be required and the plant will have its own heating system.

8.6.6.4 Electricity Supply

The combined supply of power to Dolinnoe and Obruchevskoe mines will be from the existing 110/35/6kV substation at the Ridder complex and then onto the relevant shafts by 6kV double-circuit high-voltage power lines each 1km length.

8.6.6.5 Surface Transport

It is anticipated that 40t capacity Volvo trucks will be used as the main vehicles for deliverying ore to the plant. On the return journey, the trucks will be used to deliver tailings to the backfill plant for the filling of worked out stopes underground. This will require construction of a dedicated 10km haulage road and along the road it is planned to construct two 360mm water pipelines to supply the backfill plant. The road will also be used for the movement of material and supplies to the mine.

8.6.7 Manpower

The manpower requirements for Option 3 have been determined by the "Kazgiprotsvetmet" OJSC for 2002 and have been summarised in Table 8.32 below.

Table 8.32: Number of Mining Personnel

Department	Option 3		
	Total	Workers	Engineers and technicians
Mining Operations			
Dolinnoe-Obruchevskoe mine	294	259	35
including:			
Backfill section	12	10	2
Mine Management	6		6

8.7 Financial

8.7.1 Introduction

Kazzinc have chosen Option 3 using two shafts from surface but independent of Ridder-Sokolniy mine, as this option provides the greatest Net Present Value (NPV) and Internal Rate of Return (IRR) and the lowest Discounted Payback Period. Table 8.33 summarises the saleable products from Dolinnoe and Obruchevskoe deposits

Table 8.33: Salable Products from Dolinnoe-Obruchevskoe Deposits

Item	Unit	Option3		
		Dolinnoe	**Obruchevskoe**	**Total**
Period of development of deposits		2014 - 2026	2016 - 2032	2014 - 2032
Production of concentrates including:				
Zinc concentrate	t	101,901	635,746	737,647
Lead concentrate	t	51,486	243,440	294,927
Copper concentrate	t	88,556	7,237	95,793
Gravitational concentrate	t	28,310	145,563	173,873
Sales of salable products	US$	666,745,155	1,207,575,033	1,874,320,187

8.7.2 Capital Expenditure

Estimated Capital Costs from the company's five year plan are summarised in Table 8.34 below.

Table 8.34: Summary of Capital Costs for Option 3	
Item	**Cost (US$M)**
Mine Construction	82.6
Mining Equipment	53.5
Environmental & Infrastructure Facilities Construction	35.7
Design Works	6.9
Total of Capital expenditures	**178.7**

8.7.3 Operating Costs

The operating costs have been calculated based on real costs from Tishinskiy mine which has a similar orebody, mining method, backfill techniques and pay structure. The operating cost for Option 3 are presented in Table 8.35 below.

Table 8.35: Summary of Operating Costs	
Item	**Cost (US$/t)**
Dolinnoe	
Mining	36.96
Royalty	10.02
Transportation	2.31
Processing	8.57
General and Administrative	5.71
Total costs	63.58
Obruchevskoe	
Mining	26.14
Royalty	9.76
Transportation	2.31
Processing	9.77
General and Administrative	5.71
Total costs	53.69

8.7.4 Economic Assessment

The key indicators of the Feasibility Study are presented in Table 8.36 which shows the payback period of 15.5 years, from the beginning of construction, and 10.5 years from the beginning of production. The IRR is 15.13% with a discount rate of 12.5% and a NPV of US$27.6M.

Table 8.36: Summary of the Financial Performance for Option 3		
Production Period	**years**	**2014 - 2032**
Zn Price	US$/t	1,900
Pb Price	US$/t	1,300
Cu Price	US$/t	4,000
Au Price	US$/oz	750
Ag Price	US$/oz	14.00
Internal rate of return (IRR)	%	15.13
Discounted payback period from construction	**years**	**15.5**
Rate of discount	%	12.50
Net present value (NPV)	US$	**27,621,904**
Production/processing of ore		10,215,800
Volume of Zinc concentrate	t	737,647
Volume of Lead concentrate	t	294,927
Gravitational concentrate	t	95,793
Copper concentrate	t	173,873
Sales	US$	1,874,320,187
Operating Costs	US$	(840,569,296)
EBIT with Amortisation	US$	1,033,750,891
Depreciation	US$	(228,498,733)
EBIT	US$	805,252,158
Income tax	US$	(247,032,286)
Net income	US$	558,219,872
Capital Costs	US$	(247,922,667)
Depreciation	US$	228,498,733
Accumulated net cash flow	US$	538,795,938
Discounted payback period from production	**years**	**10.5**

8.8 WAI Ore Reserve Estimation

8.8.1 Introduction

WAI has carried out stope design, and produced Ore Reserves for the Dolinnoe and Obruchevskoe deposits in accordance with the guidelines of the JORC Code (2004), based upon the most recent Mineral Resource Block Model. WAI has used GijimaAST Mine2-4D® (Mine2-4D) software to prepare the stope design.

8.8.2 Mine 2-4D Software

Mine2-4D is an automated mining software package that permits accurate design mine of excavations (development and stoping), and to apply time-dependent mining activities, such as backfilling and cable bolting, in a three-dimensional graphical environment. Following the design of excavations and associated activities, mining activities can be sequenced with time delays built into the sequence where appropriate.

8.8.3 Mining Parameters

The stope blocks for Dolinnoe and Obruchevskoe have been designed to a minimal average block grade of 4% Zn, or 4% Zinc Equivalent (ZnEQV).

8.8.4 Mine Layout

The Dolinnoe and Obruchevskoe mines have been laid out following the proposed mining methods laid out in section 8.6.

8.8.5 Dolinnoe

Dolinnoe is a greenfield site, and as such, principal mining levels have been designed in order to optimise mineral extraction. Stoping levels have been created at 15m vertical intervals from the 245m to 425m Levels.

All proposed new stopes for the Dolinnoe deposit have been designed as sub-level chamber stopes with backfill as outlined in section 8.6. Figure 8.13 shows the proposed stoping blocks for Dolinnoe.

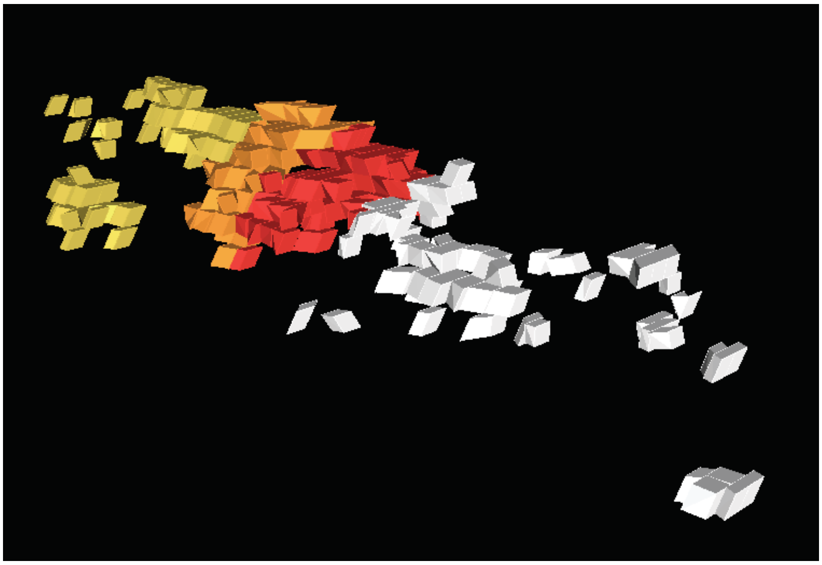


Figure 8.13: Dolinnoe Stope Blocks

8.8.6 Obruchevskoe

Stoping levels have been created at 15m vertical intervals from the -15m to +135m Levels. All proposed new stopes for the Obruchevskoe deposit have been designed as sub-level chamber stopes with backfill as outlined in section 8.6. Figure 8.14 shows the proposed stope layout for Obruchevskoe.

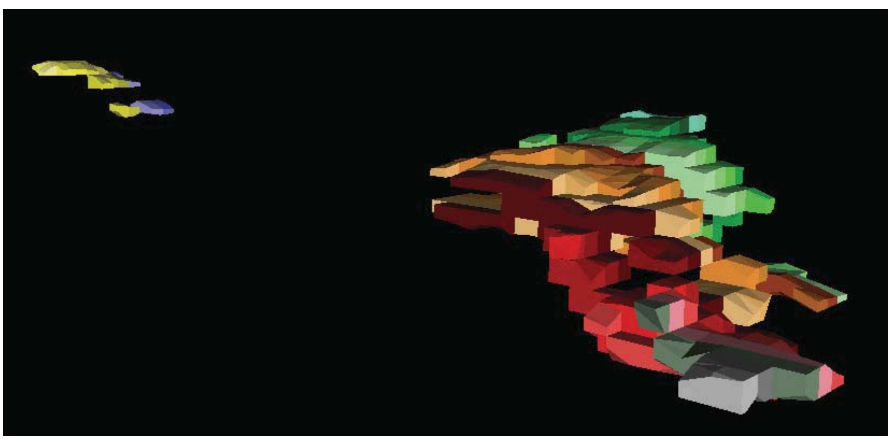


Figure 8.14: Obruchevskoe Stope Blocks

8.8.7 Ore Reserves

Ore Reserves for the Dolinnoe and Obruchevskoe deposits have been estimated in accordance with the guidelines of the JORC Code (2004). A summary of the Ore Reserves is presented in Table 8.37.

> ***WAI Comment:*** *It should be noted that in addition to the Ore Reserves outlined, 43,000t of Inferred material (at 3.52g/t Au, 18.42g/t Ag, 0.13% Cu, 0.68% Pb & 1.24% Zn) at Dolinnoe and 36,764t of Inferred material (at 3.79% Zn, 0.92% Cu, 0.79% Pb, 0.24g/t Au and 12.39g/t Ag) at Obruchevskoe have been modelled, but not reported as Ore Reserves. This material has been included in the production schedule, as it is not realistic to leave this material in-situ.*

Table 8.37: Dolinnoe & Obruchevskoe Ore Reserve Estimate (WAI 01.01. 2011)

(In Accordance with the Guidelines of the JORC Code (2004))

Deposit	Reserves	Ore (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
			Grade (g/t)	Metal Content (oz)	Grade (g/t)	Metal Content (oz)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)	Grade (%)	Metal Content (t)
Dolinnoe*	**Proven**	3.66	3.93	462,258	53.76	6,325,710	0.20	7,385	0.75	27,351	1.41	51,730
	Probable	0.96	2.38	73,822	29.82	923,296	0.14	1,338	0.50	4,849	1.02	9,792
	Total	***4.62***	***3.61***	***536,080***	***48.77***	***7,249,006***	***0.19***	***8,723***	***0.70***	***32,380***	***1.33***	***61,523***
Obruchevskoe	**Proven**	0.89	1.73	49,363	42.80	1,219,753	0.81	7,161	4.27	37,829	8.98	79,581
	Probable	3.25	0.90	94,019	33.21	3,466,977	0.83	26,845	2.66	86,520	6.50	211,092
	Total	***4.14***	***1.08***	***143,382***	***35.26***	***4,686,731***	***0.82***	***34,006***	***3.01***	***124,349***	***7.03***	***290,673***

*Dilution of 20% and losses of 5% applied

8.9 Processing

8.9.1 Dolinnoe Deposit

Technological studies of the ores from the Dolinnoe deposit were conducted on one technological sample (#1) weighing 339.5kg, and 9 low-volume samples at the "Kazmekhanobr" institute in 1992. Sample #1 is polymetallic and was selected from ore body 3 from the central part of the deposit, which is considered to represent the main value and characteristics.

Ores are of sulphide type, and oxidation is insignificant. The main ore minerals are; chalcopyrite, galenite, sphalerite, pyrites and fahl ore, and predominantly fine to medium grained.

Rational analysis of gold of sample #1 is presented in Table 8.38 below.

Table 8.38: Analysis of Sample #1 of Dolinnoe Ore

Occurrence form of gold and silver	Content, %	
	Gold	Silver
I. Free:		
a) with clean surface	26.7	3.9
b) covered with oxidized films	4.5	11.4
2. In intergrown pieces:		
a) with clean surface	12.7	31.0
b) covered with oxidized films	30.6	17.8
3. Associated with ore minerals, including:	24	30.2
a) with anglesite and cerussite	3.0	4.1
b) with galenite	4.3	6.4
c) with sphalerite	6.8	7.1
d) with pyrites	8.4	8.7
e) with chalcopyrite	1.5	3.9
4. Associated with barren rock	1.5	5.7
Head grades, (g/t)	3.0	30.0
Yield by class –0.074mm	70	
Colour	Reddish-yellow	
Form	Twig-like arborescent crystals, octahedrons	
Size of gold grains, (mm)	0.05-0.025	

Tests were performed using bulk-selective flotation flowsheet similar to the one used currently for processing of Pb-Zn ore at Ridder, recommended as the model for conducting tests of small-volume technological samples.

Test results on samples from Dolinnoe obtained the following concentrates; gold-containing gravitational concentrate with content of gold from 94.71-141.05g/t, copper – with content of copper 27% and recovery 71-72.38%, lead – with content of lead 52.79 – 55% and recovery 71.13-75.5%, zinc – with content of zinc 54.81 – 56% and recovery 75 – 76%, extraction of gold in commercial concentrates – upto 77.15%.

Due to similar processing properties of ore from Dolinnoe deposit, and Pb-Zn ore of Ridder-Sokolniy deposit, ore from Dolinnoe will be concentrated using the existing types of process equipment of concentrating plant of Ridder complex in accordance with processing flowsheet of polymetallic ore of Ridder-Sokolniy deposit.

Processing of Dolinnoe deposit envisages the following operations:

- Three-stage crushing;

- Two-stage grinding;
- Bulk flotation of sulphide minerals;
- Selection of bulk concentrate with acquisition of gold-containing flotation and zinc concentrates;
- Separation of gold-containing flotation concentrate into lead and copper concentrates, and
- Drying of concentrates.

Separation of the gold as a gravity concentrate from ore during the grinding cycle is performed on shaking tables. Five products are obtained as a result of beneficiation of polymetallic ore from Dolinnoe deposit: gravity concentrate, copper concentrate, lead concentrate, zinc concentrate and tailings.

8.9.2 Obruchevskoe Deposit

Technological studies of the Obruchevskoe deposit were conducted on technological sample (#1) weighing 2.87t ("Kazmekhanobr", 1993) and laboratory sample (#2) weighting 243.7kg ("VNIItsvetmet", 1991-1993), together with 11 low-volume samples weighting 34.6-88.7kg at "VNIItsvetmet" institute in 1991-1993.

Polymetallic ore prevail and copper-zinc and zinc ores differ from polymetallic ores only by low content of galenite and chalcopyrite. The content of oxidized and secondary minerals of copper fluctuates from 6% to 32.5%; of lead – from 2% to 27%; and of zinc – from 1-3%.

Tests were performed on the samples using a bulk-selective flotation flowsheet similar to the one used for processing of Pb-Zn ore at Ridder-Sokolniy deposit.

Test results on samples of Obruchevskoe obtained the following concentrates; copper – with content of copper 28.5% and recovery 88.7%, lead – with content of lead 59% and recovery 86.3%, zinc – with content of zinc 56% and recovery 90.9%. Tests on gravity separation of gold from ore of this deposit were not conducted, possibly due to low content of gold in the samples. Hence, only gross recovery of gold from ore is quoted as 74%.

Similar results were obtained by "Kazmekhanobr" on a bulk laboratory sample weighing 2.5t in semi-industrial conditions. With similar processing properties with Pb-Zn using ore from Ridder-Sokolniy, ore from Obruchevskoe will be concentrated using existing types of process equipment as at the Ridder complex. The same operations and products will be achieved at Obruchevskoe as at Dolinnoe.

8.10 Environmental

8.10.1 Introduction

The Dolinnoe and Obruchevskoe deposits are located, approximately 7.5km and 11km respectively, to the east of Ridder, and Dolinnoe is 2.5km from Bystrukha village. The two sites are in close proximity to each other and are both still at the exploration stage with no site development. Both sites are accessed by the Ridder-Biysk road Russian Federation.

8.10.2 Environmental & Social Setting and Context

8.10.2.1 Landscape, Topography

The Dolinnoe deposit is situated in Bystrukha valley with the Bystrukha River crossing the southern part of the deposit. The area is predominantly gently undulating topography with hills varying from 650m to 1,000m in height. The valley is surrounded by the Prohodny Ridge to the south, up to 1,800m in height. To the north, the Ivanovsky Ridges rise to 2,300m, with other lower mountain ranges also present.

Ground elevations range from 950m to 1,060m above Baltic sea datum and the site is dissected by shallow wetlands and streams flowing into the Bystrukha River. Surface formations comprise moraine, alluvial and deluvial deposits ranging in thickness from 120m to 475m. The area is overgrown with deciduous forest predominated by birch trees.

The region is seismically active with earthquakes up to 4 on the Richter scale being reported in the Ridder area in the last 10 years.

Soil conditions, flora and fauna are similar to those found at Ridder.

8.10.2.2 Climate

Climatic conditions are similar to those at Ridder.

8.10.2.3 Land Use and Land Cover

No development of the Dolinnoe and Obruchevskoe deposits has yet been undertaken. Dolinnoe is made up of two polymetallic deposits referred to as the north-eastern and south-western. Above the deposit is the Byslrushinshoye Reservoir which covers an area of 6,700m^2. The Bystrukha River flows into the Byslrushinshoye Reservoir on the western part of the deposit and is the primary the source of recharge water.

To the south of the Byslrushinshoye Reservoir is an abstraction point, and Kazzinc hold a permit to abstract up to 18Mm^3/year for the annual operational requirement of the mine. The water is pumped via a pipeline for this purpose.

8.10.2.4 Water Resources

No design for water supply has been developed for either project, but it is believed that if any additional water is required it will come from the reservoir.

> ***WAI Comment:*** *A full water balance assessment to establish the sustainable water needs of the Company to use the reservoir as its primary water source is assessed annually, and agreed with the State Water Resource body. The mine requirement will not be more than 5% of the discharge from the reservoir, and is not expected to have a negative affect on the water balance.*

8.10.2.5 Communities and livelihoods

A village is located c2.5km from the Dolinnoe deposit and although a mining plan is not yet available, it is understood from the outline plans that the village could be undermined.

> ***WAI Comment:*** *The mine design, as agreed with State authorities, ensures that measures to avoid subsidence in the areas of dwellings are implemented, such as mining occurring below 400m, and backfilling of voids. The OVOS for the project will address potential social impacts that may affect the local community.*

8.10.2.6 Infrastructure & Communications

There is no infrastructure related to mining on either site, both sites are connected by the same road.

8.10.3 Project Status, Activities, Effects, Releases & Controls

8.10.3.1 Project Description & Activities

Both Dolinnoe and Obruchevskoe deposits are polymetallic and were discovered in about 1979 following a programme of drilling. Further exploration work was undertaken from 1989 to 1990. As yet a project design has not been developed for either site.

> ***WAI Comment:*** *Details of the proposed mining of the deposit would appear to be at an early stage and no designs have been submitted to the state for endorsement.*

8.10.3.2 Land Ownership and Tenure

Kazzinc acquired the right to explore and mine the Dolinnoe deposit by tender in April 2004. The same tender included the nearby Obruchevskoe deposit. A geological allotment comprising two separate areas covering the Dolinnoe and Obruchevskoe deposits was issued to Kazzinc by the Ministry of Energy and Mineral Resources in July 2004. The contract was issued in August 2007 (contract No 2450). The contract defined the working boundary for Dolinnoe as 3.3km^2 and Obruchevskoe as 1.61km^2.

8.10.3.3 Energy Consumption & Source

There is currently no demand for energy at the projects, however, the energy requirements for mine construction and development will be substantial.

8.10.3.4 Mine Wastes – Rock

There are currently no waste rock dumps at the sites, however, when mining commences a considerable amount of waste rock will be generated and will require managed storage. This material may have the potential to generate Acid Rock Drainage (ARD) and leach heavy metals. It is intended to use this waste as backfill for the old (Andreevsky) open pit at Ridder. Nonetheless, testwork to assess ARD potential will be required.

8.10.3.5 Water Management & Effluents

For the planned project, mine water will be pumped to the water treatment facilities at Ridder. In 2011, the water treatment facilities at Ridder will be expanded to deal with the increase in volume.

8.10.3.6 Emissions to Air

There is a potential that wind blown dust/particles from exposed surfaces and exploration activities will be generated, and when the projects are developed, dust will be generated from a variety of sources such as vehicle movements, excavation, loading and deposition, which will need appropriate control measures.

8.10.3.7 Waste Management – General

Given the status of the projects, there are no man-made wastes currently present.

8.10.3.8 Security

Access to the sites is not currently restricted. It is understood that Kazzinc will secure their exploration activities.

8.10.4 Permitting

8.10.4.1 ESIA/OVOS

An OVOS has not been undertaken for proposed activities at either site. A single OVOS will be produced for both sites, given their proximity. Operation-specific 'Maximum Allowable Emissions' (MAEs) and 'Maximum Allowable Discharges' (MADs) will be determined and approved as part of the OVOS process for both sites.

> ***WAI Comment:*** *WAI would recommend that an ESIA conforming to international best practice is undertaken prior to implementation of further works. When the OVOS process is initiated, there is the opportunity to expand the EMP to meet the requirements of an ESIA. This has cost benefits and would*

> *be strongly recommended, however, it should be noted that additional monitoring, in terms of scope and frequency (seasonality), will be required for the ESIA.*

8.10.4.2 Environmental Permits and Licenses

There are no current environmental permits held by Kazzinc for Dolinnoe and Obruchevskoe.

8.10.5 Environmental Management

8.10.5.1 Environmental Policy, Management, Training and Resources

This is not applicable at this time, given the current status of the projects. However, when operations commence, it is assumed that the Company-wide corporate management system and training will be adopted at both sites.

8.10.6 Social and Community Management

8.10.6.1 Stakeholder Dialogue and Grievance Mechanisms

Although not applicable at this time, it is assumed that the existing Company-wide approach will be adopted at both sites. Public hearings, regarding Dolinnoe and Obruchevskoe mine development, were held in Ridder in December 2010.

8.10.6.2 Social Initiatives and Community Development

Given the status of the projects, this has not been considered in detail at this stage. However, the Company demonstrates a commitment to community development at other controlled assets and performs a wide range of social initiatives at Ridder-Sokolniy Mine.

8.10.7 Health & Safety

8.10.7.1 Health & Safety Management Arrangements

Apart from Exploration-related health and safety practices, there is no current requirement for health and safety management at the sites. When development commences, it is assumed that Standard Operating Procedures (SOPs) will be developed for individual activities and that the Corporate Health and Safety management system and ethos will be applied.

Kazzinc is accredited under the internationally recognised OHSAS 18001 system.

8.10.8 Mine Closure & Rehabilitation

8.10.8.1 Mine Closure Plans

In the absence of mining rights, a conceptual Closure Plan has not been considered.

> ***WAI Comment:*** *WAI appreciates that it will not be necessary to consider closure issues until the mining licence has been granted and understands that although a preliminary closure cost estimate will be undertaken, a formal Closure and Rehabilitation Plan (MCRP) will not be prepared until the final phases of mine life. WAI would recommend that, in line with international best practice a fully costed framework MCRP is drafted as a priority.*

8.10.9 Conclusions

8.10.9.1 Environmental and Social liabilities & Risks

WAI understands that at present project designs for either site have not been formulated and subsoil mining licences have not been applied for. Anticipated dates for these applications are not confirmed. The current exploration licence for both sites was granted in 2007 and is valid for 19 years.

At this stage Kazzinc has not started or commissioned an OVOS for either site. However, WAI considers that Kazzinc demonstrates good environmental monitoring and management practices at other operational sites and demonstrates a commitment to meeting national legislative requirements.

Recommendations for ESAP

WAI would recommend that when the project is approved, an ESAP is prepared. This would also ensure that the following documents/systems/assessments are undertaken:

- Preparation of Environmental Management System to fulfil national and international standards
- Commissioning of an OVOS;
- Site Assessment and structural survey of all existing buildings;
- Collection of baseline data;
- Commence air quality monitoring; and
- Undertake a sustainable water resource assessment on the Byslrushinshoye Reservoir if/when the project designs are developed.

9 CURRENT EXPLORATION PROGRAMS IN THE RIDDER AREA

9.1 Exploration Drilling on Ridder-Sokolniy Flanks

Both surface and underground exploration is conducted annually at Ridder-Sokolniy. A summary of this drilling work for the period 2000-2008 is given in Table 9.1 below. Notably drilling was significantly curtailed during 2009 due the financial crisis.

Table 9.1: Exploration Drilling Summary 2000-2008

Type of Drilling	Unit	Period								
		2000	**2001**	**2002**	**2003**	**2004**	**2005**	**2006**	**2007**	**2008**
Surface	m	15,672	14,872	8,712	10,953	11,256	13,250	18,542	56,502	62,539
U/Ground	m	45,329	49,110	47,410	51,645	55,142	55,794	57,613	71,401	77,200

9.1.1 South-Eastern Flank of Perspectivnaya Lode

Historically a Kazzinc drilling programme was conducted in 2005 to 2007 over a 2.2km^2 area situated between the Belkina, Perspectivnaya and Kriukovskaya lodes of the Ridder-Sokolny deposit on one side and the Bakhrushinskoye and Dolinnoye deposits on the other. It consisted of 66 diamond core drill holes, with wedge deflections, for a total of 43,990m.

The drill holes were sited on lines set up on an azimuth 55˚ at 150-200m intervals and generally angled north-east. Actual drilling grids, after deflections, varied from 100m by 50m to 100m in the central part of the area to 200m by 200m in the outlying areas. Spacing between drill holes on the south-eastern margin of the investigated area varied from 220m to 520m.

The drill hole diameter was 76mm and occasionally 59mm. The average core recovery was 83%.

Routine analyses, including fire assays for gold and silver, were performed at the chemical laboratory of the Ridder-Sokolny mine. External control analyses were performed by DGP VNIItsvetmet in Ust-Kamenogorsk. It is understood that internal and external control analyses did not reveal any analytical problems.

The drilled area contains shallow dipping stratabound gold-silver-bearing polymetallic and low sulphide mineralisation underlain by steeply dipping vein swarms with gold-polymetallic, polymetallic and low sulphide gold mineralisation.

9.1.2 Western, South-Western, South, South-Eastern and North-Western Flanks of the Ridder-Sokolny Deposit

Targets in these areas were first drilled from December 2006 to December 2008. The drilling programme was terminated as a result of an exploration budget cutback following the worldwide economic crisis of late 2008. A total of 169 diamond core drillholes were completed for 109,741m, which represented approximately 50% of the proposed drilling programme.

The following targets were drilled:

- South-western flank of the Bystrushinskaya Lode (named Sokolok Lode);
- Western flank of the Bystrushinskaya Lode;
- Southern flank of the Bystrushinskaya Lode;
- Western flank of the Ridder-Sokolny deposit;
- An area covering the south-eastern flanks of the Pobeda and Perspectivnaya lodes and the north-western and northern flanks of the Dolinnoye deposit;
- Northern and north-western flanks of the Bystrushinskaya Lode; and
- Western flank of the 2nd Ridder Lode

The programme involved deep diamond core drilling from surface, using directional wedges where required, targeting previously postulated P_1 prognosticated resource areas. Drillholes were sited on a 100m to 150m by 200m to 50m to 60m by 50m to 100m grid patterns and occasionally reached Level 22 (175 above Baltic Sea datum or 610m below surface). Drill core was taken over a total interval of 67,691m, with an average core recovery of 84%.

The **Sokolok Lode** is a south-western extension of the mineralisation that was earlier delineated in the South Bystrushinskaya area between drive lines 29 and 34.

The upper part of this lode consists of several thin stratabound lenses of gold-silver bearing polymetallic mineralisation situated at different levels within the upper member of the Kriukovskaya Formation. The largest of these lenses, and also hypsometrically highest, is called Orebody No 5. It is more or less an isometric body measuring 250m in diameter in plan view and 1 to 8m in thickness. All stratabound lenses follow a shallow NW-trending dome structure.

The lower part of the Sokolok Lode consists of a swarm of predominantly N-S trending veins that have been traced for 500m along strike.

The delineated resource is contained at levels 13 to 22 between drive lines 35-45 and crosscut lines 22-34.

Drilling on the **western flank of the Bystrushinskaya Lode** encountered a new lens of stratabound barite-polymetallic mineralisation in altered siltstones within Horizon I. The lens is centered on a shallow NW-trending dome and extends over a strike length of 200m with lateral extent of 40m to 80m and thickness of several meters.

The lens is underlain by a series of steep dipping veins of a lower grade polymetallic and gold-bearing polymetallic composition. The veins generally strike almost north-south, range from 1m to 3m in width and do not exceed 100m in strike and downdip extents. Some isolated drill intercepts returned high gold grades (>18 g/t) suggesting the presence of small high grade gold veins. A delineated resource is located at levels 10 to 15 between drive lines 36 to 41 and crosscut lines 8-15.

Several steep dipping veins were also delineated on the southern flank of the Bystrushinskaya Lode.

9.1.3 Exploration 2010

An aggressive exploration programme has been conducted by Kazzinc in 2010. This work has been conducted on the flanks (margins) of Ridder-Sokolniy, particularly on the south eastern flank of Perspectivnaya lode towards the Dolinnoe deposit (x2 drill rigs); the south-western flank of Pobeda lode (x4 rigs rigs) and on north-west flank of Perspectivnaya lode (x2 drill rigs) The location of the exploration holes are shown in Figure 9.1 below.

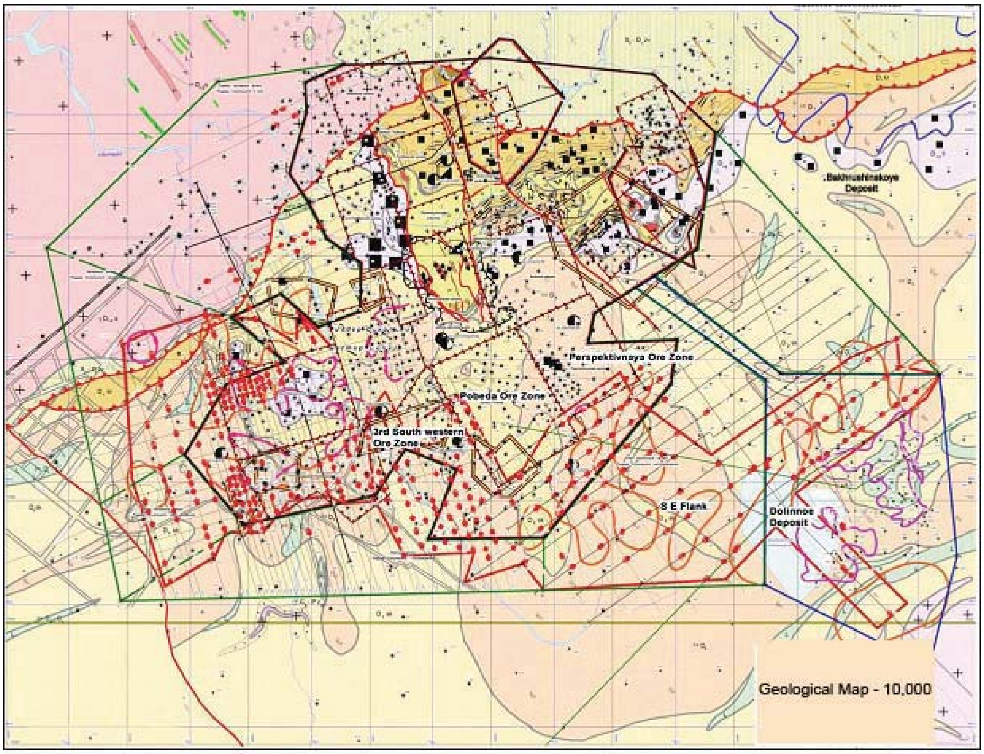


Figure 9.1: Plan showing Location of Exploration Drill holes on the Flanks of Ridder-Sokolniy Deposit (Collar locations shown in red) (Grid:200m)

In total x8 drill rigs are being utilised in the programme, some 65km of drilling for 2010, which will continue through 2011. Two contractors have been employed to conduct the works, namely LLP Geolen and LLP Corrund.

LLP Geolen is currently drilling on the south-western flank of the Pobeda ore zone. LLP Corrund is currently drilling the south eastern flank of the Perspectivnaya ore zone towards the Dolinnoe deposit, with both a Boart Longyear LF 90 rig and a Russian manufactured ZIF 1200 rig (see Figure 9.2 below).

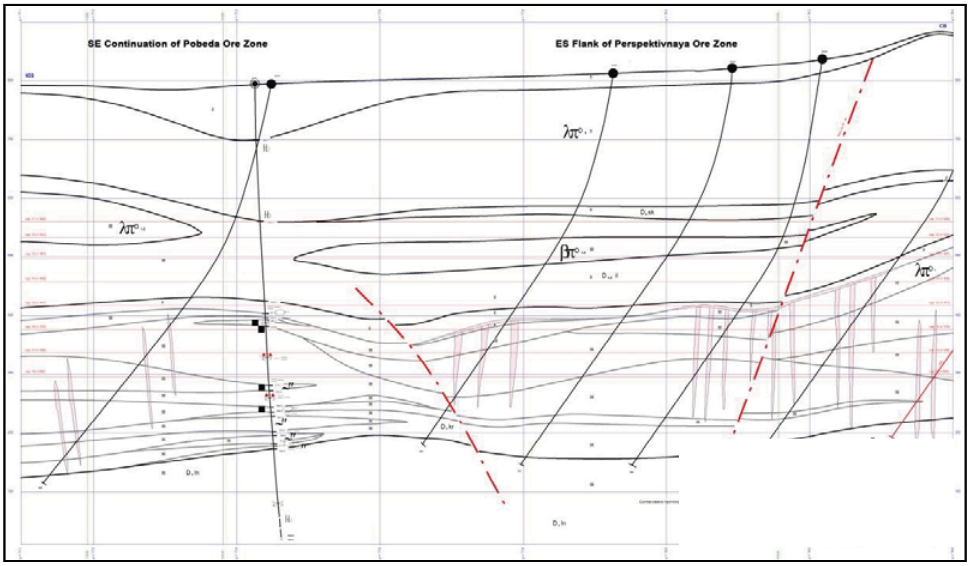


Figure 9.2: Cross Section of Proposed Exploration drilling on the South-eastern Flank of Perspectivnaya Ore Zone (proposed holes shown in red) (Grid:200m)

Additional drilling is also planned for the northern and southern flanks at the Dolinnoe deposit (15,000m of drilling) in an attempt to identify potential gold targets around the main known ore bodies. A cross section illustrating the proposed programme is shown in Figure 9.3 below.

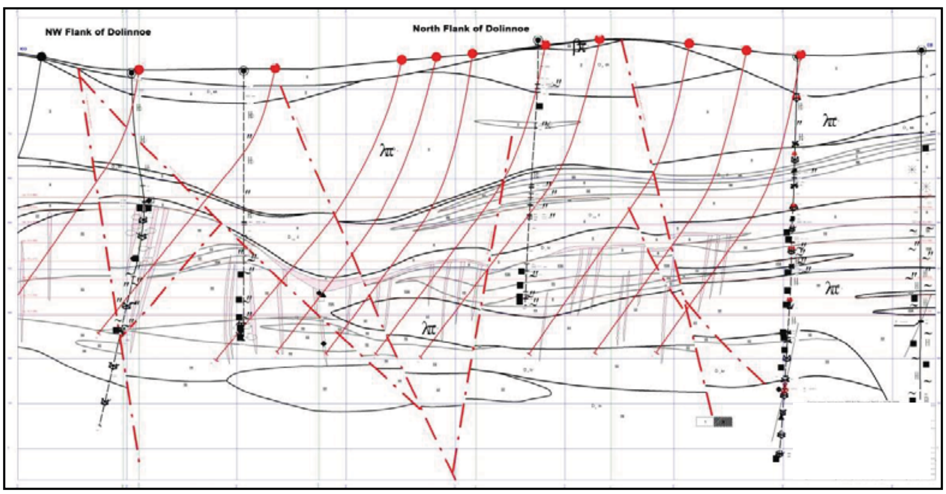


Figure 9.3: Proposed Exploration Drilling on the Northern Flank of Dolinnoe Deposit (proposed holes shown in red)

Exploration drilling is to be continued through 2011. The exploration budget and programme is given in Table 9.2 below.

WAI Comment: *WAI has reviewed the current drilling practices on site and has found them to be well implemented. The recording of drill hole data (logging and surveying), sampling procedures and drilling practices were of very good standard.*

WAI has reviewed the proposed exploration programme and budget for 2011 and has no reason to believe it is not fit for purpose.

Table 9.2: Planned Exploration Works For 2011 Proposed by Kazzinc

Object	**Units**	**Total 2011**	**Same by Quarters**			
			1	**2**	**3**	**4**
1. NW Flank of Bystrushinskaya Lode	m	5,600	1,400	1,400	1,400	1,400
	US$	685,714	171,429	171,429	171,429	171,429
	k tenge	100,800	25,200	25,200	25,200	25,200
2.Western Flank of Zavodskaya Lode	m	4,000	1,000	1,000	1,000	1,000
	US$	489,796	122,449	122,449	122,449	122,449
	k tenge	72,000	18,000	18,000	18,000	18,000
3.Western Flank of 2nd Ridderskaya Lode	m	8,000	2,000	2,000	2,000	2,000
	US$	979,592	244,898	244,898	244,898	244,898
	k tenge	144,000	36,000	36,000	36,000	36,000
4. Deep Horizons of Belkina Lode	m	20,000	5,000	5,000	5,000	5,000
	US$	2,448,980	612,245	612,245	612,245	612,245
	k tenge	360,000	90,000	90,000	90,000	90,000
5. South-East Flank of Pobeda Lode	m	11,000	3,000	3,000	2,500	2,500
	US$	1,346,939	367,347	367,347	306,122	306,122
	k tenge	198,000	54,000	54,000	45,000	45,000
6. Ore Occurence of Borehole No. 2426	m	10,400	2,600	2,600	2,600	2,600
	US$	1,273,469	318,367	318,367	318,367	318,367
	k tenge	187,200	46,800	46,800	46,800	46,800
7. Ore Occurence of Borehole No. 2437	m	10,400	2,600	2,600	2,600	2,600
	US$	1,273,469	318,367	318,367	318,367	318,367
	k tenge	187,200	46,800	46,800	46,800	46,800
8. Ilinskoye Ore Occurence	m	4,800	1,200	1,200	1,200	1,200
	US$	587,755	146,939	146,939	146,939	146,939
	k tenge	86,400	21,600	21,600	21,600	21,600
9. Bakhrushinskoye Deposit	m	16,000	4,000	4,000	4,000	4,000
	US$	1,959,184	489,796	489,796	489,796	489,796
	k tenge	288,000	72,000	72,000	72,000	72,000
10. Deep horizons of Tishinskiy Mine	US$	897,959	224,490	224,490	224,490	224,490
	k tenge	132,000	33,000	33,000	33,000	33,000
	m	**96,200**	**24,300**	**24,300**	**23,800**	**23,800**
Total Exploration Works of Ridder Mining and Processing Complex	**US$**	**11,942,857**	**3,016,327**	**3,016,327**	**2,955,102**	**2,955,102**
	k tenge	**1,755,600**	**443,400**	**443,400**	**434,400**	**434,400**
	US$	897,959	224,490	224,490	224,490	224,490

9.2 Exploration Drilling for Deep Horizons and Flanks at Tishinskiy

Kazzinc has proposed an extensive exploration programme of deep drilling at Tishinskiy in 2011. These holes envisage intersecting the main ore zone at depth for extension of the mine and on the flanks to

The programme consists of:

- Drilling of x17-18 drill holes in the center and on the western flanks (and to some extent on the eastern margin) of the Main ore zone between the -410 and -470m elevations at 18 to 21 levels; and
- Drilling x4 or 5 very deep directional drill holes to intersect the Main ore zone at the -600m elevation which lies just below -22 level. These holes will be from 1,500-1,600m long.

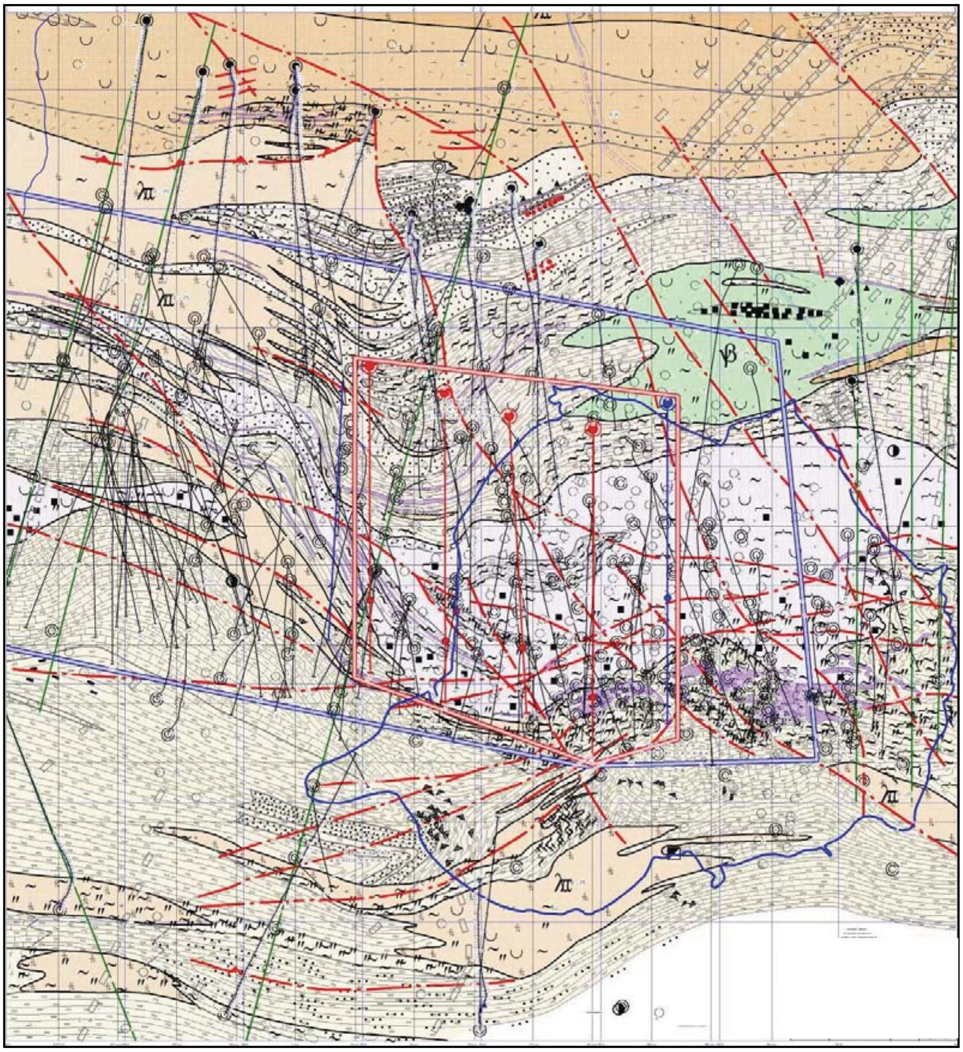


Figure 9.4: Plan showing Location of Deep Horizon (collars and traces shown in red) and Flank (collars shown in black with traces in green) Exploration Drill holes (Grid:200m)

Figure 9.5: Long Section showing Location of Deep Horizon (Collars and targets shown in red) and Flank (targets shown in green) Exploration Drill holes (Grid:200m)

10 CHEKMAR

10.1 Introduction

The Chekmar deposit comprises three polymetallic ore zones, Chekmar, South-Eastern and Gusliakov. They are located in close proximity to each other in the centre of an 8km long mineralised trend which extends along the south-western slopes of the Gusliakov and Bolshoy Chekmar Mountains on the east side of the Uba River.

The deposit is situated 46km north of Ridder and linked with Ridder by a graded road, which is passable over the whole year except for short spells of wet weather and heavy snow fall.

Ground elevations at the sites range from approximately 860-1,196m above sea level at the peak of the Malyi Chekmar. The neighbouring peaks rise to 1,561m (Bolshoy Chekmar). The Uba valley is approximately 650m deep and its tributaries invariably have deep and steep sided V-shaped valleys.

The site and surrounding mountains are covered by thick forest with fir, birch, some spruce and pine and a variety of shrub species. The River Ulba is renowned as a prime trout fishing venue.

The climate is extreme continental with the average annual temperature of 1.6°C and seasonal variations from +35°C in July to -45°C in December. Mean annual precipitation is approximately 1,000mm, of which maximum falls as snow in winter months. Snow cover lasts from early October to April, and longer on north facing slopes and in forested river valleys.

> ***WAI Comment:*** *Notably the licence lies close to significant watercourse, which is a renown tourist site and lies with a designated nature reserve.*

10.1.1 Mineral Rights and Permitting

Kazzinc won a tender for the rights to carry out exploration and extraction of the polymetallic ore zones comprising the Chekmar deposit on 26 December 2008.

A geological lease was granted to Kazzinc on 21 May 2009, covering an area of 6.9km^2. The boundaries of the lease are defined as detailed in Table 10.1 below.

The Chekmar subsoil use contract is expected to be finalised and registered with the relevant Kazakhstan authority by the end of 2011. The Chekmar mining plan, which was submitted by Kazzinc to the relevant Kazakhstan authority, is currently going through the approval process.

Table 10.1: Chekmar Lease Boundary Points

Boundary Points	Geographical Co-ordinates	
	Latitude N	Latitude N
1	50°40'10"	83°35'35"
2	50°40'09"	83°36'32"
3	50°37'52"	83°37'56"
4	50°37'55"	83°36'02"

10.1.2 Project History

The three ore zones comprising the Chekmar Group (Gusliakov, Chekmar and South-Eastern) were explored during the period 1971-1981. Detailed exploration was completed at Gusliakov (1975) and Chekmar (1981), whilst exploration of the South-Eastern ore zone was stopped on completion of a preliminary exploration programme in 1977.

Subsequent activities funded by the government of the Republic of Kazakhstan included:

- The partial completion of a railroad from Ridder to the site;
- The development of a small trial open pit in the central part of the Chekmar ore zone;
- Process testwork on a 2,000t bulk sample conducted by VNIItsvetmet in Ust; and
- Limited exploration to define a silver resource at Chekmar, which was abandoned due to a shortage of funds in November 1993.

All works on the property were terminated in 1996. Kazzinc is currently undertaking a Feasibility study for re-starting the open pit on the Chekmar ore zone.

10.2 Geology and Mineralisation

10.2.1 Local Geology

The three ore zones are genetically associated with submarine rhyolite volcanism and comprise predominantly stratabound accumulations of sphalerite, galena, chalcopyrite and pyrite, which precipitated from hydrothermal fluids on or below the seafloor. The Chekmar ore zone is underlain by a stockwork feeder pipe with copper-zinc and copper mineralisation. No such features have so far been recognised at the Gusliakov and South-Eastern ore zones, which are developed in a stratigraphically higher horizon (see Table 9.2), and have not been explored to the same depths as the Chekmar ore zone.

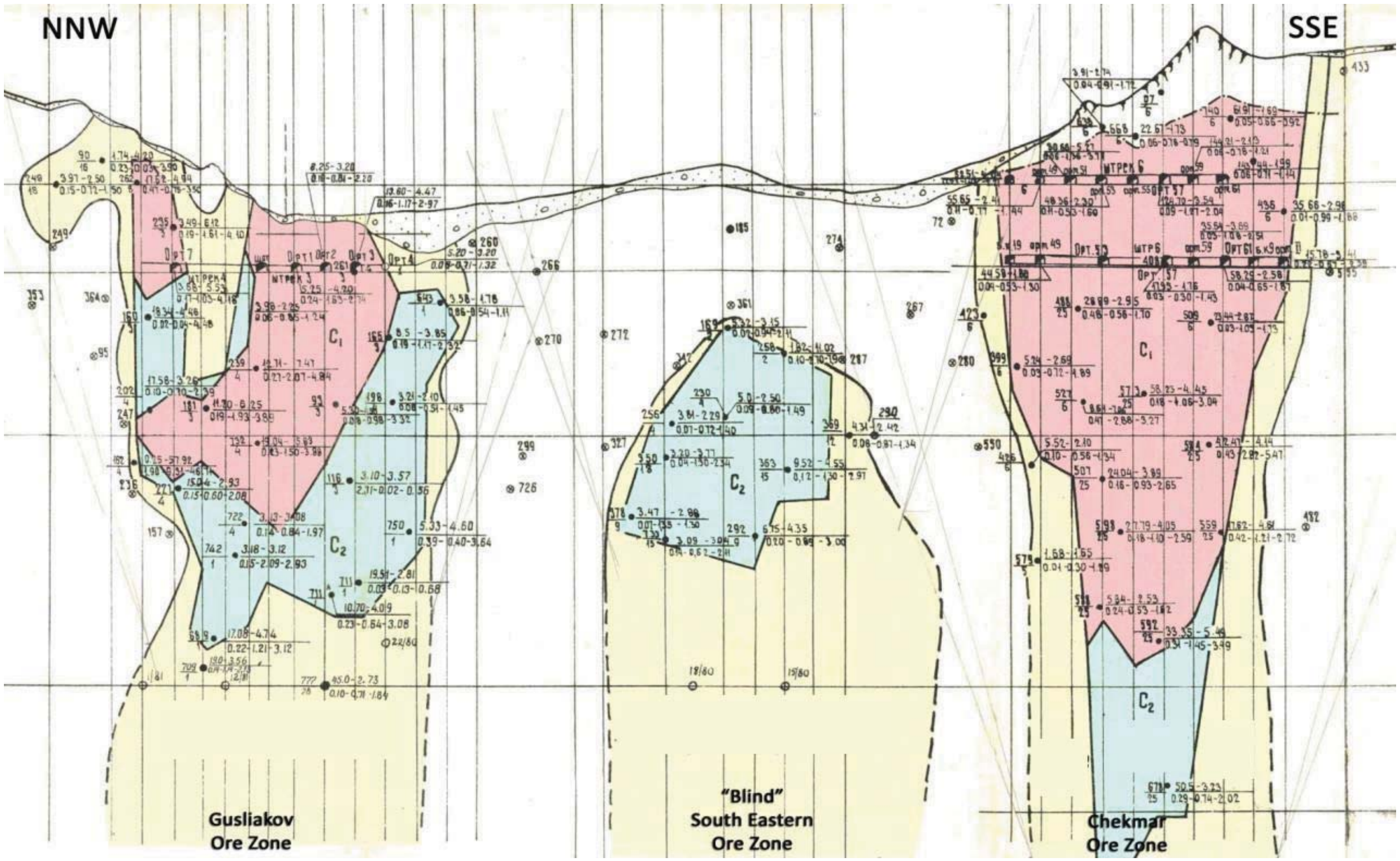


Figure 10.1: Long Section showing Three Ore Zones

Table 10.2: Stratigraphic Position of Polymetallic Mineralisation of the Chekmar Ore Zones

Stratigraphic Unit			Occurrence	Lithology	Mineralisation
Uspenskaya Formation	Upper Subformation	D_2e_2-gv_1us_2	From 20-30m at Chekmar to 70-100m at Guslakov	Carbonate Siltstones with carbonaceous matter and lenses of crinoidal limestones Tuffs, lavas and lavobreccias of intermediate composition with intercalations of green epicalastic rocks	
	Lower Subformation	Upper effusive pyroclastic member (UEMP) D_2e_2-$gv_1us_1^4$	Laterally most extensive member, sharply variable composition, fairly constant thickness: 250-300m at Chekmar, 300-400m at South-Eastern and 170-250m (not bottomed) at Guslakov	Lithic-vitric acid tuffs and flows of quartz rhyolite and quartz-feldspar rhyolite, lenses and intercalations of tuffaceous siltstones, siliceous siltstones, tuffaceous sandstones and tuffs of acid composition	Tuffaceous units host significant mineralisation at South-Eastern and Guslakov deposits
		Tuffogenic-sedimentary member (TSM) D_2e_2-$gv_1us_1^3$	Mainly at Chekmar (60-170m in thickness), reduced to 0-25m at South-Eastern Deposit	Siliceous tuffaceous siltstones sandstones, tuffites and tuffs derived from mottles quartz rhyolite, lenses of albitophyres and quartz rhyolites and crinoidal limestone at the top of the unit Boundinages and brecciated microquartzite horizon of variable thickness developed at Chekmar at the uppermost levels immediately below crinoidal limestone horizon, associated relics of hematite-quartz and hematite-carbonate-quartz cherts	The microquartzite horizon is the main host for stockwork mineralisation at the South-Western Zone of Chekmar
		Lower effusive-pyroclastic member (LEPM) D_2e_2-$gv_1us_1^2$	Thickness varies from 170-200m at Chekmar to 90-120m at South-Eastern presumably continuing into Guslakov	Alternating mottled quartz rhyolite and rhyolite tuffs, locally subordinate reworked tuffs Strong chlorite-sericite-quartz alterations, particularly at South-Western mineralised zone of Chekmar	Hosts mineralisation at Central and South-Western zones of Chekmar, mineralised at South-Eastern deposit
		Sedimentary-volcanic member (SVM) D_2e_2-$gv_1us_1^2$	A 90-110m in thickness in the core of Chekmar dome	Rhyolite lavas with intercalations and lenses of tuffaceous sandstones and crinoidal limestones at the top of the unit	Hosts mineralisation at Central and North-Eastern zones of Chekmar

Syn and post-depositional faulting appears to have played a key role in focusing the flow of the hydrothermal fluids. Subsequent folding and regional metamorphism strongly modified forms of mineralised bodies and metal distribution. Despite these modifications all three ore zones are stratabound and display many relic features normally associated with Kuroko-type VMS deposits.

10.2.1.1 Chekmar Ore Zone

The Chekmar ore zone contains about 40 mineralised bodies, predominantly of stockwork and stringer type, contained in a tight to isoclinal north-west striking double plunging anticline with two small subsidiary anticlines on the south-western and north-eastern limbs. Mineralised bodies that form the central part of the deposit, starting from the gossan cap and descending to about the 500m level (at 350-400m depth), are grouped together as Central Zone. Mineralised bodies on the south-western and north-eastern limbs of the anticline are grouped as South-Western and South-Eastern zones respectively.

The largest mineralised bodies occur in a boudinaged and brecciated microquartzite horizon, which appears to represent submarine siliceous precipitates within the tuffogenic-sedimentary member (TSM) of the Lower Uspenskaya Formation. In plan view, this horizon attains 600m in width and drapes over the hinge of the anticline and onto its south-western and north-eastern limbs. The mineralisation does not breach the overlying upper effusive-pyroclastic member (UEPM) The host microquartzite horizon contains cherty manganiferous hematite quartz and hematite-carbonate-quartz which are most probably relics of an exhalite horizon that formed an apron around the ore zone.

Pyrite-polymetallic stockwork with an iron content of more than 10% is found in the apical part of the Central Zone. Elsewhere, stockwork, stringer and disseminated lead-zinc mineralisation predominates and extends down to levels 600m and 500m, below which lead and zinc grades decrease and mineralisation becomes polymetallic (copper-lead-zinc). The same tendency has been noted within larger mineralised bodies as the distance from the hanging wall increases. Small massive sulphide bodies are locally developed but represent an insignificant proportion of the total volume. As indicated by drilling, the South-Western Zone does not extend below the -100m level (i.e. below a depth of 900-1,000m below the surface).

Copper-zinc and copper mineralisation assemblages occur at deeper levels of the Central Zone in the:

- Lower effusive-pyroclastic member (LEPM);
- Sedimentary-volcanic member (SVM);
- North-Eastern Zone, where they form a single NNW-elongated pipe, some 200-300m in width; and
- Footwall of lead-zinc and polymetallic mineralisation.

The associated hydrothermal alteration changes from sericite-quartz to chlorite-sericite-quartz. The pipe of the North-Western Zone is open at depth of 1,100-1,200m below the surface (level -200m).

Mineralised bodies included in the resource range from 100-400m in strike length, from 230-900m in down dip extent and from 6-230m in width.

The Central Zone crops out on top of the Chekmar Mountain, where it has given rise to massive gossan capping. The trial open pit is located in this capping and also exposes the adjacent microquartzite-hosted pyrite-polymetallic stockwork. Effects of superficial oxidation extend down to 200-300m below the surface and mixed partially oxidised mineralisation forms over 45% of the resource so far delineated at Chekmar.

10.2.1.2 South-Eastern Ore Zone

This ore zone was discovered whilst exploration was conducted on the Gusliakov and named to reflect its location in relation to Gusliakov. This is a blind ore zone of which only the top part has been explored. The structure is a NW-trending tight to isoclinal anticline with a moderate to steep (40-60°) NW plunge. It is thought to be an extension of the north-eastern subsidiary anticline of Chekmar rather than an extension of the main Chekmar anticline. The fundamental difference in relation to Chekmar is that the mineralisation occurs in tuffaceous units within the UEPM from where it has been traced down into the LEPM. The TSM fades out at this locality. The mineralisation is polymetallic with barite present at the uppermost and intermediate depth levels.

Eleven lenses of stockwork and stringer mineralisation have been delineated with strike lengths in the range of 175-290m. Down-dip dimensions vary from 90-185m and widths from 3-11m. Lead grades are about 40% higher than at Chekmar and zinc grades are around 25% higher.

10.2.1.3 Gusliakov Ore Zone

This ore zone occurs in a NW-trending double plunging tight to isoclinal anticline, which is correlated with the anticline of the South-Eastern ore zone. Due to faulting, this area is lifted up at least 200m in relation to the South-Eastern ore zone. As at the South-Eastern ore zone mineralisation at Gusliakov is polymetallic, with barite-rich lenses at the highest stratigraphic levels. Stockwork, stringer and disseminated mineralisation predominates but massive sulphide lenses are more abundant than at the South-Eastern deposit and account for over 5% of the mineralised volume. The mineralisation occurs in tuffaceous units within the UEPM. Twenty four mineralised lenses have been delineated with strike lengths of 60-150m. Down dip dimensions vary from 100-375m and widths from 7-10.5m. Average lead grades are 75-80% higher than at Chekmar and zinc grades are generally twice as high.

10.2.2 Mineralisation

Three natural primary sulphide mineral associations have been recognised within the Chekmar ore zones; they are: lead-zinc, polymetallic and copper-zinc. The polymetallic mineralisation differs from the lead-zinc mineralisation in that it contains more than 0.1% Cu and from the copper-zinc mineralisation in that it contains more than 0.1% Pb. The Chekmar ore zone displays lateral zoning with copper-zinc and polymetallic mineralisation in the centre passing outwards into polymetallic mineralisation, which in turn gives way to lead-zinc mineralisation on the flanks.

The main primary metalliferous minerals are sphalerite, pyrite and galena. Chalcopyrite is much less common except in the copper-zinc mineralisation. Accompanying minerals include arsenopyrite and tetrahedrite-tennantite. Barite occurs with polymetallic mineralisation, locally in significant quantities. The main gangue mineral is quartz. Calcite and fluorite occur in subsidiary quantities.

The polymetallic and copper-zinc mineralisation contains gold-bearing zones. A series of such zones have been traced for a strike length of 300-350m and for 300-350m down dip between Profiles 47-61 in the central part of the Chekmar ore zone. Gold-bearing zones have also been identified within lead-zinc mineralisation in the footwall of the South¬Western Zone. With exploration being focused on base metals, the importance and extent of the gold-bearing zones still remain to be assessed.

Gold in primary sulphide mineralisation occurs predominantly in its native form and as electrum intergrowths with quartz and sulphide minerals. Isomorphic admixtures in sulphide minerals, particularly in chalcopyrite, are less common. In the central part of the Chekmar deposit, for instance, only 7% of the gold occurs within sulphides. The average gold contents in sulphide minerals were stated as follows: 3.5g/t Au in chalcopyrite, 1.83g/t Au in pyrite, 0.6g/t Au in galena and 0.1g/t Au in sphalerite.

The size of gold is generally within a range of 0.003 - 0.07mm but grains as large as 0.45mm have been noted. Fineness varies from 740 to 890.

At the Chekmar deposit, superficial oxidation extends from the gossan cap down to depths ranging from 20-70m. Fracture zones show signs of oxidation to a depth of 120-150m. The main secondary minerals are: hydrogoethite, hydrohematite, chalcocite, malachite, limonite-cuprite, cerussite, smithsonite, plumbojarosite, covelite, cuprite, lepidochrosite, azurite, goslarite, chalcanthite, melanterite and relatively rare chrysocolla.

The weathered zone is enriched in gold, which is frequently found as high purity native grains occurring as nests in kaolin among partially oxidised sulphide veinlets.

The oxidation zone at Chekmar is underlain by a leached transition zone to primary sulphide mineralisation. Secondary supergene minerals (which include secondary galena, chalcocite and covellite) and effects of partial leaching extend approximately to a depth corresponding to the Ulba river (at absolute elevations of 650-670m or approximately at a 200m depth) and locally to a depth of 250m along fracture zones. This zone consists of partially oxidised mineralisation with secondary supergene sulphides alternating with primary unaltered sulphide mineralisation and hence is referred to as mixed mineralisation. The average Cu:Pb:Zn ratio is 0.2:1:3.

The classification into oxidised, mixed and sulphide mineralisation types is determined on the basis of phase analyses of lead and zinc in combination with the cyanide soluble copper analysis (see Table 10.3 below).

Table 10.3: Criteria for Classification into Mineralisation Types

Mineralisation Type	Sulphide Forms		CyanideSoluble Copper Content
	Lead	Zinc	
	%		%
Oxides	<50		>0.2
Mixed/Transitional	50-85	75-90	01-02
Sulphides	>85	>90	<0.1

10.3 Exploration Works

10.3.1 Sample Collection

The ore zones were explored by core drilling from surface augmented by underground exploration from two adits at Chekmar (Adit No.1 at +800m level and Adit No.2 at +670m level) and from another adit at Gusliakov (at +600m level).

The drilling grid density was selected on completion of early geological mapping and prospecting, when the targeted ore zones were assessed to represent Group III in the 4-tier classification of mineral deposits with respect to geological complexity. According to GKZ instructions, polymetallic lead-zinc deposits of Group III require a sampling grid of 50m by 50m for the delineation of a C1 category resource.

10.3.1.1 Surface Core Drilling

The drilling profile lines were cut at 50m spacing perpendicular to the overall strike. At the Chekmar and South-Eastern ore zones, profiles were cut on a bearing of 235°. At the Gusliakov ore zone, profiles were cut on a bearing of 230°. Profiles on the north-western and south-eastern flanks of the Chekmar ore zone were oriented at azimuths of 300° and 0° respectively to account for strike changes. In addition, four drill holes at Gusliakov were oriented either NW or SE to test a postulated NE strike.

The initial drill hole inclination angle depended on target depths. Most drill holes with target depths less than 300m were angled, at the start, at 75-78˚ from horizontal, whilst deeper drill holes were angled at 78-89°. Drilling trajectories were then controlled by utilising natural deviation tendencies resulting from the orientation of folded beds (holes at Chekmar had a tendency to deviate 15-25° clockwise and holes at Gusliakov 20-25° anticlockwise) and directional wedging. Overburden and barren formations at the Gusliakov and South-Eastern ore zones (to depths of 50-400m) were drilled by non-coring bits, but drill holes at Chekmar, where overburden depths are negligible, were cored throughout.

Some drill holes targeting shallow near-surface parts of the Chekmar and Gusliakov ore zones were drilled at angles ranging from a few degrees to 60° from horizontal.

External diameters at the start of each hole were variably 112mm, 93mm or 76mm, with subsequent reductions to 76mm, 59mm and 46mm respectively. Mineralised intervals were generally drilled with 59mm and 46mm drill bits, which recover core with diameters of 42mm and 31mm respectively. All unstable intervals were cased and cemented. Specialised drilling muds were used at Gusliakov.

Historical average core recoveries were estimated for each mineralised intercept included in the Soviet compliant resource estimates of 1981, (as length of core recovered)/(intersection length) but reported as percentages of the total number of intercepts included in the estimated C_1 and C_2 resource categories. An abridged summary for all mineralised intercepts included in that resource estimate (C_1 and C_2 categories combined) is given in the Table 10.4 below. It was also reported that recoveries of less than 70% were essentially limited to older drill holes and to shallow intercepts through weathered, leached and locally cavernous ground to depths of 200-250m. It was also mentioned that some caverns reached 1-3m in size. At that time, normal practice required that average recoveries from intercepts included in resource estimates had to be better than 70%.

No recent verification drilling has been undertaken at Chekmar to confirm the reliability of the data.

Table 10.4: Reported Drill Core Recoveries

Ore Zone	Average Core Recovery	Number of Intercepts	Proportion
	%		%
Chekmar	>70	678	83.4
	60 - 70	100	12.3
	50 - 60	15	1.8
	<50	20	2.5
Gusliakov	>70	278	62.5
	60 - 70	97	21.8
	50 - 60	41	9.2
	<50	29	6.5
South-Eastern	>70	33	73.3
	60 - 70	9	20.0
	50 - 60	2	4.4
	<50	1	2.2

10.3.1.2 Underground Exploration

Underground exploratory workings consisted of crosscuts developed at 100m intervals, drives and raises, some of which followed surface drill holes. Additional crosscuts were developed at 50m spacing through the largest mineralised bodies at Chekmar and occasionally at 25m spacing though gold-bearing mineralised bodies. Mineralised intervals totalled 5,920m, of which 4,990m were exposed at the Chekmar workings.

The geological mapping geology was documented on 1:50 scale drawings of walls, backs and faces (at approximately 5m intervals between successive rounds).

Direct underground exploration was augmented with underground diamond core drilling, aimed essentially at detailed delineation of morphology and compositional variations of the mineralised bodies, verification of structural and compositional continuity and, where necessary, infilling to a 50m sampling grid.

Inclination of underground drill holes ranged from +48° to -90° (vertical). Deviations were in the range of +/-5° for azimuth and up to 3° per 100m for inclination, which equates to approximately 2-3m per 100m.

The average core recoveries from mineralised intercepts were reported as 71% for Chekmar and 72% for Gusliakov.

10.3.1.3 Exploration Summary

A summary of all exploration undertaken at Chekmar is given in Table 10.5 below.

Table 10.5: Summary of Work Undertaken at Chekmar from 1956 To 1981

Stage and Scope of Work	Units	1956-1960	1961-1971	1972-1973	1973-1975	1975-1977	1977-1980	1980-1981	Total
Surface and subsurface prospecting:		100	50						150
Prospecting traverses	km	728	2,564						3,292
Prospecting pits	m	2,171	17,329		37,112				56,612
Diamond drilling	m	800	2,911		4,824				8,535
Sampling (drill core, channel)	samples								0
Prospecting-assessment:			13,422			7,564 654			
Diamond drilling	m		249						
Percussion drilling	m		3,178						
Sampling (drill core, channel)	samples								
Preliminary exploration:				1,252		16,549	2,258		
Trenching	m			3,063		300	64,933		
Underground exploratory workings	m			17,824		1,188	310		
Diamond drilling	m			135					
Percussion drilling	m			5,106					
Sampling (drill core and channel)	samples								
Detailed exploration:					103		12,044	5,964	
Underground exploratory workings	m				29,060		3,500	108,531	
Diamond drilling	m				443			595	
Percussion drilling	m				5,340			76,780	
Sampling (drill core and channel)	samples								

Gusliakov Ore Zone:
Commencement of work in 1971
Prospecting-assessment in 1971-1972
Preliminary and detailed exploration in 1972-1975
Technical-economic assessment (TEO) and resource estimation completed in 1975
South-Eastern Ore Zone: Commencement of work in 1973 Prospecting-assessment in 1974-1975 Preliminary exploration in 1976-1977 Interim resource estimation completed in 1977
Chekmar Ore Zone:
Preliminary and detailed exploration in 1976-1981
TEO completed in 1978
Detailed exploration completed in 1981

10.3.1.4 Drill Core Sampling

Drill core from mineralised intercepts was divided into samples not exceeding 1m in length or approximately 1m to coincide with visible contacts. Drill core intervals with low grade sulphide mineralisation were sampled over lengths not exceeding 2m.

Up to 1973, drill core samples from the Gusliakov and South-Eastern deposits were split by hand. From 1974, the whole core was taken except for short 10 to 15cm specimens from sample intervals where mineralisation appeared to be homogenous.

Drill core samples from Chekmar had been hand-split until 1978, when it was realised that control analyses on duplicate halves of core correlated poorly with the original results. The whole core was sampled thereafter.

Several different approaches were tried to test the reliability of core sampling. Results were generally ambiguous and simply indicating a high variability of the polymetallic mineralisation, copper in particular. Gold and silver analyses were not included in these tests. For instance, channel sampling in six raises, which were developed along traces of drill holes drilled from surface, and in five crosscuts to verify results of underground drilling, revealed only high short range variations in copper grades.

The most comprehensive test involved drilling of 15 underground holes and taking both the drill core and sludge samples over the same 2m intervals. The holes were angled from 30-60˚ from horizontal and ranged from 15-20m in length. It was established that copper, lead and zinc grades were on average lower in core samples than in the sludge. The results were then analysed in relation to core recoveries. Copper grades in solid drill core were on average 7% to >20% lower at the 0.14-0.23% Cu grade range. Lead and zinc grades correlated reasonably well for core at >70% core recovery, but differential loss of galena and sphalerite was clearly demonstrated for lower core recoveries.

Further work at Chekmar, involving control channel sampling and direct RRK measurements on drill core with different recoveries, confirmed the differential loss of galena and sphalerite. Average lead and zinc grades on drill core with 60-70% recovery were 16-24% lower than grades of the corresponding channel samples and RRK readings. Average lead and zinc grades in drill core with recoveries less than 60% were 35-44% lower.

Control channel sampling at Gusliakov returned totally different results suggesting a differential gain in respect of copper, lead and zinc at drill core recoveries below 60%. Core with recoveries in excess of 60% correlated reasonably well with channel sampling.

10.3.1.5 Underground Channel Sampling

Crosscuts at Gusliakov were channel sampled along horizontal lines over mineralised intervals exposed in crosscuts and from wall rocks up stepping out 3-5m from visible limits of the sulphide mineralisation. Channels were 10cm wide, 3cm deep and generally 1m in length. Only one side wall, generally north-western, was sampled, except in zones with sharp short distance variations, where samples were cut from both walls. Horizontal face samples were collected from advancing faces in drives.

Crosscuts at Chekmar were generally sampled along both walls. Drives on the Adit No.1 level were generally sampled along one side wall and from faces and results averaged for each round interval. Drives in Adit No.2 were sampled from faces only and those results were pooled and averaged with RRK readings taken along drive walls. As rocks at Chekmar were too hard to sample manually, mechanically driven chisels was developed by Leninogorsk GRE.

A variety of control sampling ranging from repeat sampling to bulk sampling did not indicate any systematic bias.

10.3.1.6 Composite Sampling for Gold and Silver Determination

The exploration in the Chekmar district was focused on lead-zinc and polymetallic (copper-lead-zinc) mineralisation. Precious metals were not the subject of detailed investigations and therefore their grades were determined on the basis of analyses of composited samples and statistical correlation with copper, lead and zinc.

Compositing was done over core intervals which passed the approved resource criteria with respect to copper, lead and zinc grades. Composited samples were then screened by multi-element spectral analysis, which included silver, and by gold spectral analysis. Samples with high spectral silver and gold were sent for fire assay. This method however, would have missed gold and silver mineralisation outside the mineralised bodies included in the sulphide resource model.

10.3.2 Sample Preparation

The sample preparation scheme used at all exploration stages was based on the Richard-Czeczott formula Q=kda. The coefficient k is the key parameter. In general terms, the lower coefficient k is, the better it accounts for the erratic distribution of minerals. In this instance, the k parameter of 0.16 was selected after a series of experiments using schemes with varying k coefficients.

Samples were jaw crushed in two stages to pass a 2mm sieve and then divided to obtain an approximately 1kg sub-sample. That sub-sample was ground in a roll grinder, sieved through a 0.8mm screen and pulverised to pass a 0.07mm screen. Sub-samples were then collected for Cu, Pb and Zn analyses (50g) and for gold and silver analyses (450g). A 500g duplicate of each was retained for reference purposes.

Control analyses on sample splits taken after each volume reduction did not reveal any systematic errors. Random errors were small. In total, 121 control samples were analysed for lead and zinc and 33 samples were analysed for copper.

10.3.3 Sample Analysis

Selection of core and channel samples for accurate analysis was made on the basis of results of 16-element spectral analysis. Samples that showed spectral values greater than 0.3% of Cu, Pb or Zn were submitted for chemical analysis for copper, lead and zinc and fire assay for gold and silver.

Chemical analyses for Cu, Pb and Zn were conducted by LPC and Vostkazgeologia laboratories.

Vostkazgeologia performed most of other analytical work, including:

- Fire assays for gold and silver and XRF analyses for barium on composited samples;
- Chemical analyses for copper, lead, zinc, cadmium, selenium, tellurium, indium and sulphur, atomic absorption analyses for iron, XRF analyses for barium and fire assays for gold and silver on grouped samples; and
- Phase analyses for copper, lead and zinc on grouped and routine samples.

Routine samples from the gold-bearing zone at Chekmar were analysed by spectral gold method and fire assay at LPC and partly at Vostkazgeologia.

Chemical spectral analyses for antimony and bismuth on grouped samples were performed by Altay Geological-Geophysical Expedition.

The methods used for accurate analyses for key elements are listed in Table 10.6.

Table 10.6: Analytical Methods Used During Chekmar Exploration					
Element	Method	Detection Limits			HCAM Norm
Copper	Polarographic	0.05	3	%	155-XC
	Iodometric	0.05	~10	%	
	Atomic absorption	0.01	30	%	
Lead	Polarographic to 1977		5	%	155-XC
	Complex analysis	2	~10	%	
	Atomic absorption	0.05	30	%	
Zinc	Polarographic	0.05	5	%	155-XC
	Complex-metric	3	~10	%	
	Atomic absorption	0	50	%	
Cadmium	Polarographic	0	~10	%	
	Atomic absorption	0	10	g/t	
Gold	Fire assay	0.1		g/t	
Silver	Fire assay	2		%	
Barium	Yadro-physical	0.01		%	97-ЯФ

Internal and external control analyses were performed on duplicates of 5-10% of routine samples. Neither revealed any significant random or systematic errors.

10.3.4 Bulk Density Determinations

Bulk densities have been determined on the basis of measurements on laboratory scale waxed specimens (Chekmar -109, South-Eastern - 48, Gusliakov - 260) and systematic hydrostatic weighing of drill core specimens (Chekmar >6,000, Gusliakov c. 3,000).

Bulk densities are independent of moisture content, which varies from 0.02% to 0.44%.

Barite-bearing specimens were divided into three groups:

1. Barite-bearing (>1.3% BaO);
2. Barite-polymetallic (0.5-<1.3% BaO); and
3. Polymetallic (<0.5% BaO).

Based on statistical analysis, lead-zinc, polymetallic and copper-zinc mineralisation varieties follow the same trend but pyritic polymetallic mineralisation follows a different trend. The boundary between is approximately $3t/m^3$. It was further established that bulk densities of mixed mineralisation are only slightly lower than bulk densities of sulphide mineralisation (by 0.05 to $0.07t/m^3$).

The following regression formulas were developed (Jc = Cu + Pb + Zn, or Cu + Pb + Zn + Ba for barite-polymetallic mineralisation, C = correlation coefficient):

Chekmar Ore Zone:

1. Sulphide (based on 369 samples) d=0.01247 (Jc) + 2.72585 C = 0.57
2. Mixed (based on 375 samples) d=0.01841 (Jc) + 2.66849 C = 0.52
3. Average for sulphide and mixed d=0.0165 (Jc) + 2.69717
4. Massive polymetallic (128 samples)d=0.03251 (Jc) + 3.209 C = 0.44

Gusliakov and South-Eastern Ore Zones:

1. Polymetallic (93 samples) d=0.02375 (Jc) + 2.72215 C = 0.94
2. Barite-polymetallic (147 samples) d=0.02675 (Jc) + 2.89318 C = 0.65
3. Barite-bearing (85 samples) d=2.897 + 0.03715 BaO C = 0.81

The low correlation for massive sulphides is due to variations in the pyrite content. The relationship is non-linear because the pyrite content decreases with the increase in the content of base metals.

Bulk densities estimated from these regression formulas were compared with arithmetic means of the specimens used in the derivation of the formulas. Differences between bulk densities calculated from the regression formulas and arithmetic means were within a range of -3% to +4% (-3% for mixed, +4% for massive, -1% for sulphide mineralisation).

As a control measure, bulk density was also determined on 37 bulk samples (26 from Chekmar and 11 from Gusliakov). The average bulk density for the three ore zones is 2.93t/m^3 (for average 0.14% Cu, 0.89% Pb and 2.45% Zn). The average bulk density of host rocks was determined as 2.58t/m^3.

10.4 Mineral Resources

At present no Mineral Resources or Ore Reserves have been calculated in accordance with the guidelines of the JORC Code (2004) for the Chekmar deposit.

11 STAROYE TAILINGS DEPOSIT

11.1 Introduction

11.1.1 Location & Access

The Staroye tailings dam contains waste material discharged from the Leninogorsk[1] Polymetallic Combinat (LPC) during the period 1926 to 1953 inclusive, the western area until 1946 and the eastern area from 1946 to 1953. The feed for the concentrator during that period was derived exclusively from processing of polymetallic ores from the Ridder-Sokolniy deposit. Sulphide ores formed the bulk of the concentrator feed whereas oxidised ore fluctuated from 3% to 18% of the total feed.

From 1973 tailings material (from several sources) was transported to the Irtysh Polymetallic Combinate to be processed as fluxes. Thus from the western Staroye tailings dam some 24,334t were extracted (in 1986), and from the eastern Staroye tailing dam a further 47,010t (in 1988-1992) was removed.

Photo 11.1: North-West view over the Staroye Tailings Dam
(Ridder Concentrator to the left of the image)

The Staroye tailings dam is located some 300m to the northeast of the Concentrator of formerly named, Leninogorsk Polymetallic Complex (MCLPC) at Ridder (Figure 11.1). The dam extends from Ridder Hill eastwards towards the Filipovka River and the Malaya Talovka tributary. It is easily accessible by sealed road from Ridder and is some 5.5km from the Kazzinc main office.

The tailings cover a north-south elongated rhomboid outline approximately 1km in length and 700m wide. The surface elevation is at approximately 740m AOD and the average thickness is estimated to be 12.3m, with a maximum of around 21.0m.

In the 1970's some attempts to re-process tailings at the MCC were made. Based on exploration work carried out in 1969 by "Kazgiprotsvetmet" Research Institute, within the western Staroye tailings dam, the project for tailings re-processing was developed in 1976 indicating the main technical and economical indices. This evaluation demonstrated that the project could be viable. Thus having an 'approved' reserve of 1,490kt, within a designed pit outline, and excluding a protective pillar for the railway, containing 1.74g/t Au (2,606.3kg Au) and 16.44g/t Ag (24.5t Ag), a tailings re-processing capacity of 200ktpa over 7 years was proposed. At 59.3% gold recovery to concentrates and capital investments of 2.2M rubles, the total profit was estimated to be 4.8M Rubles for a payback period of 3.4 years. Furthermore the project did not consider additional recovery of gold from bulk flotation tailings by cyanidation due to the unsuitable performance of this process, as demonstrated in previous testwork by the "Kazgiprotsvetmet" Research Institute; "Feasibility study on processing of current quartz tailings of Leninogorsk Concentrator" in 1973.

[1] The name Leninogorsk was abandoned in June 2002 and replaced with the pre-1941 Revolution name Ridder.

Figure 11.1: Outline of Staroye Tailings Dam

11.1.2 Mineral Rights and Permitting

Kazzinc holds the right to mine the Staroye tailings under the terms of Contract No 559 dated 7 November 2000. The same contract covers the nearby Chashinskoye tailings. The Staroye mining lease covers an area of 61.5ha, the boundaries of which are defined by 8 corner points as detailed in Table 11.1 and shown in Figure 11.1.

Table 11.1: Staroye Tailings Mining Lease Boundaries

Boundary Points	Geographical Co-ordinates		Local Rectangular Co-ordinates		Local Interim Co-ordinates	
	Latitude N	Longitude E	X	Y	X	Y
1	50°21'39"	83°32'35"	98,191	-14,507	84,179.39	80,856.64
2	50°21'50"	83°32'55"	98,550	-14,180	84,550.12	81,337.03
3	50°21'50"	83°33'00"	98,548	-14,004	84,552.03	81,446.06
4	50°21'36"	83°33'20"	98,101	-13,607	84,128.33	81,859.58
5	50°21'23"	83°33'10"	97,693	-13,823	83,714.91	81,661.09
6	50°21'19"	83°33'06"	97,567	-13,903	83,586.76	81,586.20
7	50°21'14"	83°32'57"	97,436	-14,079	83,449.77	81,416.20
8	50°21'21"	83°32'41"	97,655	-14,381	83,652.40	81,105.58

11.1.3 Production History

Table 11.2 presents a summary of tonnage and grades of material deposited into the Staroye tailings dam from 1926 to its closure in 1953. The data were compiled by Durnev and Golubtsov (2001) on the basis of historical records of the LPC. It was recorded that a total of 14.3Mt of tailings were deposited at the site and a small amount (151kt) was subsequently removed prior to the start of the reprocessing operation in 1991, mainly from the eastern part of the impoundment, and sent to the Irtysh Polymetallic Combinat to be used as flux.

The average gold grade in the deposited tailings dropped from 11.1g/t Au in 1926/1927 to 2.15g/t Au in 1936, fluctuated at around 2.0-2.6g/t Au from 1937 to 1945 and then, as recoveries at the LPC gradually improved, fell to 0.5g/t Au in 1953. Silver grades are more erratic while copper, lead and zinc grades show a similar pattern. The grade variations reflect both concentrator metal recoveries, which in turn are directly linked to the history of the LPC, as well as mined grade. The first concentrator was destroyed by fire in 1927 and a new concentrator (Concentrator No 2) was constructed in 1928, but required two years of fine tuning to achieve target metal recoveries. Another concentrator (Concentrator No.3) was commissioned in 1936.

Table 11.2: Tailings Deposited in the Staroye Tailings Dam (1926-1953)

Year	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
1926/27	2,850	11.10	25.60	0.40	2.30	14.10
1927/28	5,855	8.24	34.85	0.29	1.90	9.90
1928/29	23,068	4.00	31.00	0.24	0.68	6.26
1929/30	45,496	6.00	29.00	0.11	1.05	5.85
1930	14,805	7.40	37.00	0.17	0.94	5.59
1931	106,487	6.60	33.00	0.16	0.84	4.33
1932	113,383	4.70	16.50	0.14	0.42	2.77
1933	118,170	3.67	13.26	0.18	0.48	2.05
1934	144,601	2.73	13.12	0.11	0.39	1.31
1935	207,003	2.56	15.23	0.09	0.65	1.68
1936	299,806	2.15	17.36	0.08	0.50	1.31
1937	382,709	2.48	18.00	0.07	0.47	1.39
1938	397,714	2.57	18.02	0.08	0.49	1.32
1939	473,622	2.26	20.67	0.08	0.46	1.26
1940	590,234	2.05	22.45	0.08	0.44	1.22
1941	878,000	2.01	22.35	0.10	0.44	0.88
1942	655,889	2.09	21.45	0.08	0.46	1.36
1943	435,684	2.40	27.20	0.09	0.55	2.26
1944	467,534	2.14	19.97	0.07	0.55	1.93
1945	471,513	2.36	21.00	0.08	0.53	1.75
1946	548,847	1.89	21.30	0.08	0.47	1.52
1947	549,223	1.39	13.40	0.07	0.42	1.43
1948	679,183	0.94	11.39	0.07	0.36	0.96
1949	812,022	0.87	10.40	0.05	0.30	0.56
1950	1,048,463	0.74	9.08	0.05	0.24	0.48
1951	1,212,948	0.66	8.40	0.05	0.21	0.37
1952	1,592,227	0.60	7.50	0.03	0.23	0.33
1953	1,997,111	0.50	8.70	0.03	0.24	0.26
Subtotal	14,274,447	1.47	14.45	0.06	0.37	0.98
Extracted 1970-1990	151,156	1.08	17.04	0.05	0.25	0.82
Total (1.01.1991)	14,123,291	1.48	14.42	0.06	0.37	0.98

11.2 Geology and Mineralisation

The Staroye tailings dam is a man-made feature and does not possess any recognised geological structure. The tailings are the waste products of the processing of gold and polymetallic ores (primarily silver, copper, lead and zinc) from the Ridder Mining and Concentrating Complex (MCC) and their composition thus reflects the major constituents of the ore.

11.2.1 Impoundment Development Stages

Tailings with the highest gold content were dispersed over a 100-400m wide area immediately to the north-east of the concentrator. Remains of old collection tanks and pipes indicate that the outlet for the old tailings was at 761.5-763.5mAOD level. The area filled from that outlet is known as the "upper tailings impoundment". The area located immediately to the east and south-east of the "upper tailings impoundment" was filled with tailings by approximately 1946. This area is known as the "lower tailings impoundment" and its upper level is at 740-741mAOD.

Thereafter the impoundment expanded to the same level eastwards and north-eastwards, which necessitated the diversion of the Filipovka River. The last dam was built along the eastern perimeter of the present impoundment area in early 1950's. The crest of the dam is at 746-750mAOD and the top of the tailings disposed during that period reached 745mAOD.

Subsequent to decommissioning, the western and south-western parts of the impoundment served as a dumping ground. Dumped material ranges from metallurgical slags to industrial and domestic rubbish.

11.2.2 Mineralisation

Microquartzite (silicified calcareous siltstone), quartz and other gangue vein material make up over 95% of the tailings material. Sulphides and much less common oxidised ore minerals occur in accessory amounts. Pyrite is the most common sulphide mineral. Oxidised ore minerals are common and include smithsonite, cerussite, anglesite and limonite. Marcasite and azurite occur in minor amounts.

Sphalerite occurs as inclusions in gangue minerals and less commonly as intergrowths with pyrite. Its size varies from 8-700µm, with the 50-60µm range being predominant.

Galena forms minute inclusions in sphalerite and gangue minerals and also forms rare intergrowths with pyrite and chalcopyrite. The grain size varies from 3-100µm with 20-35µm size predominating.

Chalcopyrite occurs intergrown with gangue minerals of sphalerite, pyrite and occasionally with galena. The grain size ranges from emulsion to 40µm with the 10-20µm range being most common.

From an economic standpoint the most important mineral is gold. Phase and rational analyses indicate that free gold contributes about 15% of the total gold (see Table 11.3). Its composition borders electrum and the principal grain size is 20-40µm.

Over 50% of silver occurs as intergrowths with other minerals and approximately 45-50% in sulphides or gangue minerals. Kerargirite was noted as one of the silver-bearing minerals.

Table 11.3: Forms of Gold and Silver Occurrence

Forms of Occurrence	Gold %	Silver %
Free with clean surface	4.6	
Free with film-covered surface	11.2	
Intergrowths with clean surface	53.4	45.1
Intergrowths with film-covered surface	3.0	6.9
In sulphides	27.8	42.2
Including pyrite	23.3	28.6
Associated with gangue	-	5.8

11.2.3 Granulometry

The granulometric composition of the tailings was determined by a sieve analysis on 110 samples at the process testwork facility of the LPC and at the Kazmekhanobr laboratory in Ust-Kamenogorsk. The samples came from holes drilled in 1974, mainly in the eastern and south-eastern parts of the tailings impoundment. Drillhole RU 94-3 was drilled in the northern part of the impoundment and drillhole RU 94-13 in the centre ("lower tailings impoundment"). None of the samples represent the "upper tailings impoundment".

Course sand faction is insignificant (about 2.0%) whereas medium and fine sands are generally widespread (about 48.0%) with the prevailing being silt, clay and fine sandy (about 50.0%). According to the data received from VNIITSVETMET Institute the tailings are represented by fine ground material with 48.0 to 60.0% grain size <0.074mm.

As seen in the percentage grain size distribution chart in Figure 11.2 below, silt size particles (shown in violet and indigo) dominate, making up 57.4% of the total. Sand grain size particles, which form the balance, range from very fine sand to very coarse sand. The overall proportion of coarse and very coarse sand is 3.9%. It is not unreasonable to expect that the tailings in the "upper tailings impoundment" would contain a higher proportion of sand size particles.

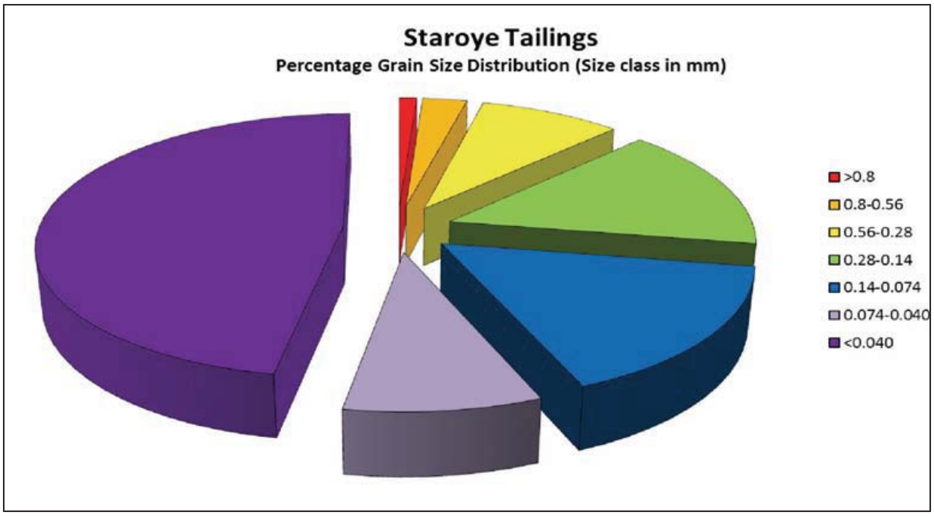


Figure 11.2: Percentage Grain Size Distribution in the Satroye Tailings

It must be borne in mind that the chart in Figure 11.2 (above) presents the average granulometric composition based on seven drillholes only. The average depth of the drillholes was 16m and samples were generally collected over 1m intervals. When considered individually, the drillholes display very diverse granulometric patterns, with the silt content ranging from 21.4% to 83.4%.

As a result of process efficiency the area of the dam (western) deposited from 1926 to 1946 displays a coarser, medium grain sand, fraction compared to the rest of the dam area which is characterised by a finer grain size. It is also noted that the percentage of silt and clay sized particles increase with depth.

11.2.4 Sample Analysis

The assay result of run-of-mine and composite samples (1974, 1991-1994) indicate that the gold content in Staroye tailings dam is approximately between 0.3-0.4g/t and 1.0-1.2g/t Au. From total number of these samples (437) only 8 samples contain less than 0.3g/t Au and 11 samples more than 2.5g/t Au. Samples from the first programme of drillholes (1969) have only 15 samples with a maximum value ≥3.0g/t Au. Boreholes 14 (10.8g/t Au) and 16 (11.5g/t Au) possess the highest gold content value but have only one sample for their entire length and are thus unreliable.

A collection of 347 run-of-mine samples from the last analysis period (1991-1994) demonstrate a relatively clear grain size distribution limit between 0.3-0.4g/t Au and less clear for 0.8-0.9g/t Au and 1.0-1.1g/t Au and with some degree of convention >2.0g/t Au.
There is evidence of a higher grade concentration in the western part of the tailings dam, which was filled during the early stages when richer ore was being mined and the process efficiency was lower. From 1947 onwards the depositional grade and grain size is more homogenous.

Grain size distribution has been studied using a number of drillholes from the 1994 period (RU94-3, RU94-5, RU94-6, RU94-7, RU94-13, RU94-15, RU94-16, and RU94-20) in the central and eastern parts of the dam (profiles II and VIII). This was also previously studied in 1974 by the Kazmekhanobr Institute in the western part of the tailings dam (1969 drilling period) at 100x100m grid using engineering-geological boreholes and excavated holes.

As a whole the average results were as follows:

- grain size >0.56mm (rough and coarse sands) = 1.2g/t Au (sample selection is not significant);
- grain size 0.56 – 0.10mm (medium and fine sands) = 0.72g/t Au; and
- grain size <0.10mm (fine sands, silt and clay particles) = 0.54g/t Au.

As seen in Table 11.4 below, and Figure 11.3 (below), there appears to be no correlation between particle size and the gold distribution. The same applies to silver and base metals.

Table 11.4: Grade Distribution versus Grain Size

Class	Yield	Grade						Distribution (%)					
		Au	Ag	Cu	Pb	Zn	Fe	Au	Ag	Cu	Pb	Zn	Fe
mm	%	g/t	g/t	%	%	%	%						
Feed Material		2.34	18.4	0.06	0.43	1.22	1.48						
0.4	4.7	2.9	20.1	0.07	0.5	1.3	1.7	5.8	5.1	5.0	5.6	5.0	4.3
0.2	15.6	2.1	18.2	0.05	0.2	0.5	1.0	13.9	15.4	11.7	7.5	6.7	8.9
0.1	26.8	2.9	13.4	0.04	0.3	0.9	1.2	32.8	19.5	17.6	15.2	20.6	17.3
0.074	5.8	2.6	18.6	0.05	0.3	1.4	1.5	6.4	5.6	4.7	3.8	6.7	4.7
0.044	7.8	2.7	21.3	0.07	0.5	1.5	2.6	9.0	9.0	8.4	8.1	9.6	11.0
0.022	10.2	2.3	19.6	0.07	0.5	1.4	2.4	10.0	11.8	11.2	10.7	11.7	13.3
0.011	9.3	1.7	17.8	0.07	0.5	1.8	2.5	6.8	10.8	10.1	9.7	13.7	12.4
0.005	8.7	2.2	21.2	0.19	0.8	2.0	2.5	8.2	10.1	12.2	16.2	14.2	11.7
<0.005	11.1	1.5	17.2	0.14	0.9	1.3	2.8	7.1	12.7	19.1	23.2	11.8	16.4

NB: The Class +0.074mm on scrambler. Class -0.074mm in sludge analyser

Source: Durnev and Golubtsov (2001) quoting results reported by VNIltsvetmet (1970-1972)

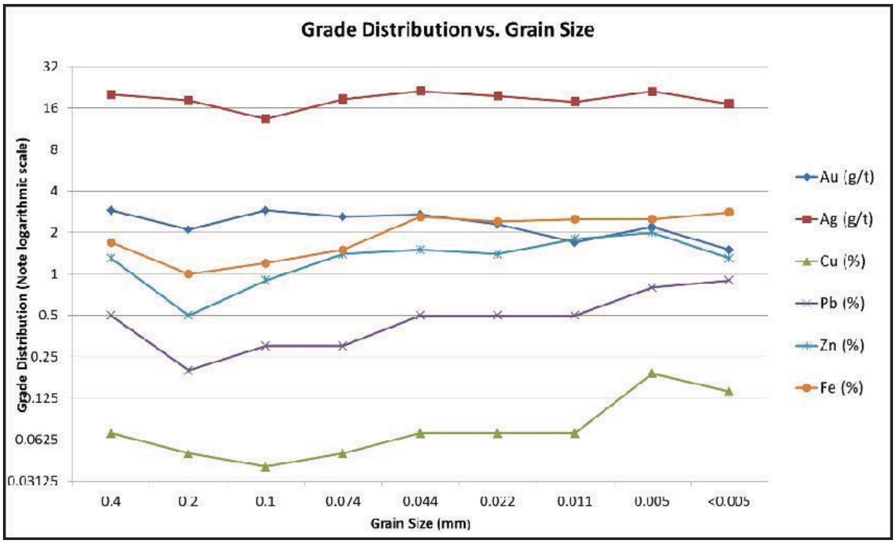


Figure 11.3: Grade Distribution versus Grain Size (Au, Ag, Cu, Pb, Zn, and Fe)

11.3 Topographic Base and Surveying Data

The local survey grid, which was used during topographic surveys and for locating drillhole sites, is linked to 2nd and 3rd class triangulation points. The relative point location error is reported to be 0.12m.

Locations of drillhole collars were surveyed by theodolite traversing from survey grid points or by resection from a minimum of three co-ordinated points. Collar elevations were measured by trigonometric levelling from not less than two co-ordinated points. The mean square positional error did not exceed 0.9m in plan and 0.2m in elevation.

11.4 Exploration Works

11.4.1 Drilling

Drilling has been conducted in several campaigns since 1969 and a total of 98 holes (1,374m) have been completed on a nominal spacing of 50m x 50m, and up to 150m x 200m. The 1969 drill campaign produced a single sample for the entire length or intersection of tailings material but later campaigns sampled at 5.0m down to 1.0m intervals. The samples were analysed for gold, silver, copper, lead and zinc at the Leninogorsk GOK laboratory.

11.4.1.1 Historical Drilling

The tailings were evaluated from data obtained during three stages of drilling in 1969, 1974 and 1993-1994 respectively (Table 11.5). In addition, four verification holes were drilled in 1991-1993. A total of 98 evaluation holes were drilled on 22 section lines for a total of 1,374m. Most holes were drilled until they reached the base of the tailings impoundment.

Table 11.5: Staroye Tailings Evaluation Drilling

Year	Drilling method/drill rig	No of Drillholes	Total Depth	Grid	Main Purpose
1969	Cable tool/UTB-5A	40	490	50m x 50m	Evaluation of the western part of the tailings impoundment
1969	Cable tool/UTB-5A	7	103	120m x 250m	Preliminary assessment of other parts of the impoundment Infill in the 150-200m wide
1974	Cable tool/UKC-22	24	387	50m x 50m	Fringe around the previously drilled western fringe
1991	RC hydrolift/UGB-ZUK	3	42		Verification drilling
1993	Vibrodrill	1	19		Verification drilling
1993-1994	Vibrodrill	23	333	150m x 150m	Northern and Eastern parts of the impoundment
Total		**98**	**1,374**		

The first drilling programme (1969) focused on the western part of the impoundment, which contained tailings deposited during the period 1926-1946 and considered to be the richest. A total of 47 cable tool boreholes were completed, including 40 holes on a 50x50m grid to depths of 11-15m over the western part of the impoundment and 7 holes to 12-20m depth elsewhere. All holes were 110mm in diameter.

The 1974 drilling, again by cable tool method, covered the central area of the impoundment immediately to the east of the area drilled in 1969. A total of 24 holes were drilled on a grid of 50x50m to a depth of between 9m and 21m. All holes were 10 inches (254mm) in diameter.

In 1991, JV Kazgold drilled three 13-15m deep verification holes within the area drilled in 1969. Hole diameters varied from 377mm to 219mm. A verification hole was drilled to a depth of 19m using a vibrocore rig in 1993 with a diameter of 59mm.

In 1993-1994, Goldbelt Resources Limited and Pegasus completed, as part of the feasibility study for the tailings re-treatment, 23 vibroholes on a grid of 150x150m in the northern and eastern parts of the tailings area which had not been investigated earlier. The holes varied in depth from 1.7-23m and the hole diameter was 59mm. In addition, 15 holes were drilled for specific purposes, such as hydrogeological, geotechnical, etc. but these holes are not included in the database used in the resource estimates.

> ***WAI Comment:*** *WAI has not observed any of the drilling and sampling process followed but has reviewed the protocols applied, and in keeping with typical Soviet standards, considers the methods to be industry standard and appropriate for use in a mineral resource and ore reserve estimate in accordance with the guidelines of the JORC Code (2004).*

11.4.2 Sample Collection

Different sampling methods were employed at different stages of drilling. In the 1969 drilling programme only one sample per hole was collected. Each sample was dried, mixed and quartered. Approximately 8kg of material was then taken and reduced to 1kg.

In 1974 samples were collected over intervals of 5m, plus the remainder at the end of each hole ranging from 1m to 7m. Samples were stored, dried, mixed and quartered to between 10 and 12kg.

> ***WAI Comment:*** *Collecting one sample per drillhole is very poor practice. Sampling over 5m intervals is better but still inappropriate as it does not characterise the layered composition of the tailings. However, as grade distribution with depth is considered to be relatively uniform, and no selective mining is likely, this should not be too detrimental to the resource evaluation.*

During the 1991 verification drilling programme, samples were taken from each metre. This material was mixed and quartered to 50kg. After drying each sample was reduced in a Jones riffle splitter to about 19-20kg.

> ***WAI Comment:*** *This is a more appropriate sampling technique which allows a reasonable definition of metal grades within the vertical profile and reduces the risk of random errors being extrapolated over large volumes. It also provides representative material for other tests.*

In 1993-1994 samples were taken generally over 1m intervals with each sample reduced in a Jones riffle to 3kg.

Samples from the 1991 and 1993-1994 programmes were subdivided in a Jones riffle into:

- Samples for fire assay and chemical analysis (approximately 600 g each);
- Duplicate samples (approximately 600g each);
- Samples for sieve analysis (2.5-3kg each) taken from the western part of the tailings impoundment;
- Samples (12kg each) for compositing into technological samples;
- Control samples (150g each);
- Samples for process testwork (0.5kg); and
- A general 5kg sample was taken from the primary drillhole for drain size analysis for Au, Ag, Cu, Pb and Zn. The general sample comprised 100-300g duplicate samples (depending on the hole depth).

Samples for granulometric analysis and determination of gold, silver, copper, lead and zinc in various grain size classes obtained for every other hole by compositing 100-300g duplicates into 5kg composites.

> ***WAI Comment:*** *Sampling and sub-sampling methods implemented in the 1991 and 1993-1994 drilling campaigns are understood to conform to typical industry standards except for the apparent lack of measures to monitor the quality.*

11.4.3 Sample Analysis

All analyses were performed at the central chemical laboratory of Leninogorsk GOK. These include fire assay for gold and silver (475 samples) and chemical analyses for copper, lead and zinc (472 samples). A total of 218 samples were sent for fire and chemical analyses for Au, Ag, Cu, Pg and Zn by grain sizes at the central laboratory of Leninogorsky Metallurgical Complex. Granulometric analyses were performed on 110 samples at the process testwork laboratory of the LPC and from 1974 at the laboratory of Kazmekhanobr.

Results of internal and external control analyses are not available. Durnev and Golubtsov (2001) only mention that relative differences between results of control and routine analyses were within two standard deviations (+/-5%) from the population mean.

The following analysis has been undertaken:

Fire assay for Au and Ag was completed in the assay section of Leninogorsky Metallurgical Complex central laboratory and a total of 475 samples were assayed.

Chemical assay for Cu, Pb and Zn grade was also made in the central laboratory of Leninogorsk GOK (472 samples assayed).

- 218 fire and chemical analyses for Au, Ag, Cu, Pg and Zn by grain sizes were made in the central laboratory of Leninogorsky Metallurgical Complex;
- Quarterly internal and external geological control was taken to check sampling accuracy; and
- The findings of internal and external geological control proved acceptable tolerance of grade test results.

Grain size (granulometry) sieve analysis of residues was made in the experimental and research section of Leninogorsky Metallurgical Complex (110 samples). The samples taken along the western extension of the tailing dam were analysed in 1974 at the Kazmekhanobr laboratory.

11.4.4 Bulk Density, Moisture Content and Porosity

Density and moisture content were determined on 17 samples collected in December 1991. Wet bulk density above the water saturation level was found to be 1.42-1.64t/m^3 (average 1.52t/m^3) rising to 1.71-1.91g/cm^3 (average 1.80g/cm^3) below the water saturation level.

Porosity was determined on drill samples from depths of 2-12m by KazGIIZ in 1964 and 1970-1971. The measurements varied from 45.6-51.05%, with the average porosity being 48.1%. Based on the 48% porosity, dry bulk density was estimated to be in the range of 1.40-1.44t/m^3.

> ***WAI Comments:*** *It would appear that the results for bulk density, moisture content and porosity are based on 17 samples taken in December 1991 as well as density determination values from the 1964 testwork programme, which Durnev and Golubtsov believe to be most representative.*

11.5 Extraction and Reprocessing

Extraction and processing data for the period from 1991 to April 2009 inclusive are provided in Table 11.6 and Table 11.7.

Table 11.6: Staroye Resource Depletion Data (1991-April 2009 Inclusive)						
	Tonnes (Mt)	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
Deposited Tailings	14.274	1.5	14.4	0.06	0.37	0.98
Extracted prior to 1991	0.151	1.1	17.0	0.05	0.25	0.82
Mined 1991-2008	4.679	1.4	12.4	0.07	0.31	0.79
Mined Jan-April 2009	0.284	1.4	14.0	0.07	0.35	0.90
Mining Losses (4.4%)	0.000002	1.4	12.5	0.07	0.31	0.79
Balance Remaining	9.161	1.5	15.4	0.06	0.40	1.09

Currently the Staroye tailings are being mined at a rate of approximately 0.5Mtpa and re-processed at the Ridder plant. The process involves a two stage grinding followed by a gravity-flotation scheme with the recovery of gravity concentrate, auriferous concentrate and zinc concentrate. Production data for the period from 1991 to 2008 inclusive are given in Table 11.6. Note that various process routes have been tried since 1991 before the scheme summarised was established.

During the period from 1991 to 2008 inclusive, Kazzinc recovered approximately 4.0t of gold, 31.1t of silver, 18,000t of zinc, 6,000t of lead and almost 2,000t of copper from 4.679Mt of processed tailings.

Based on the metallurgical balance the average head grade of the re-processed tailings was 1.37g/t Au, 12.43g/t Ag, 0.07% Cu, 0.31% Pb and 0.79% Zn.

Resource depletion calculations based on the extraction data are given in Table 11.7 below. The depletion includes resource losses during mining and transportation, which Kazzinc has estimated at 4.4% of the total depleted resource. Using the tonnage and grades as at 01 January 1991 (see Table 11.7) as the initial resource, the resource remaining at the Staroye impoundment, as at 1 May 2009, should amount to 9.1Mt at 1.5g/t Au, 15.4g/t Ag, 0.06% Cu, 0.4% Pb and 1.1% Zn.

Table 11.7: Production Figures for Staroye Tailings Reprocessing (1996 - September 2010)						
Year	**Tonnes (t)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
1996	64,563	2.25	16.42	0.08	0.36	1.25
1997	42,971	1.30	10.47	0.04	0.26	0.52
1998	2,047	0.98	7.82	0.05	0.20	0.29
1999	0	-	-	-	-	-
2000	178,606	1.73	13.08	0.10	0.37	0.80
2001	484,766	1.28	10.30	0.05	0.27	0.58
2002	490,660	1.49	11.64	0.07	0.29	0.67
2003	572,932	1.62	11.69	0.08	0.30	0.76
2004	755,630	1.03	9.82	0.05	0.22	0.57
2005	942,336	1.14	12.66	0.06	0.32	0.75
2006	286,142	1.01	11.70	0.08	0.35	0.89
2007	346,230	1.73	16.79	0.07	0.44	1.18
2008	421,880	1.65	16.56	0.07	0.37	0.92
2009	937,143	1.76	14.99	0.15	0.35	0.97
2010	414,636	1.12	11.16	0.06	0.26	0.72
Total	**5,940,542**	**1.41**	**12.70**	**0.08**	**0.31**	**0.79**

Mining and re-processing of the tailings material continues, as witnessed during the WAI site visit, with production expected to increase to 1.0Mtpa with the proposed construction of a new dedicated process plant in the near future. The plant is also intended to re-process tailings from Chashinskoye tailings dam as well.

11.6 Kazzinc Reserve Estimation

The previous resource estimation by Kazzinc, as of 01.01.2000, updating that produced at 01.09.1994, was completed taking into account all historical exploration, chemical-analytical works and technological research performed since the tailings dam ceased being operational in 1953, and is summarised in Table 11.8 and Table 11.9.

Table 11.8: Staroye Tailings Reserve Estimate (01.09.1994)

Tonnes (kt)	Grade and Contained Metal				
	Au g/t (kg)	Ag g/t (kg)	Cu % (t)	Pb % (t)	Zn % (t)
12,545.5	1.25 (15,538.3)	13.07 (162,752)	0.04 (5,287)	0.31 (38,207)	0.74 (92,228)

Table 11.9: Staroye Tailings Reserve Estimate (01.01.2000)

Tonnes (kt)	Grade and Contained Metal				
	Au g/t (kg)	Ag g/t (kg)	Cu % (t)	Pb % (t)	Zn % (t)
12,345	1.24 (15,335.7)	13.06 (161,226)	0.04 (5,216)	0.31 (37,863)	0.74 (91,195)

Subsequent to the previous reserve estimations, and in light of production from Staroye tailings, Kazzinc have completed an updated reserve estimate (as at 01.01.2010) as summarised in Table 11.10 below.

Table 11.10: Staroye Tailings Reserve Estimate (Kazzinc 01.01.2010)

Classification	Tonnes (Mt)	Grade				
		Au g/t	Ag g/t	Cu %	Pb %	Zn %
C_1	0.5788	2.55	26.43	-	-	1.17
C_2	6.311	1.04	13.16	-	-	0.70
Total	**6.9**	**1.17**	**14.23**	**-**	**-**	**0.74**

11.6.1 Method Validation and Estimation Limits

The Staroye tailings dam has largely been studied during two phases of investigation by drillholes from 1969 to 1974, when 71 cable percussion holes were completed in the western area on a 50m by 50m grid, and again in 1993-1994 when 23 'vibrocore' holes on irregular 150m spacing were completed over the remainder of the dam. There were an additional 4 verification holes completed on the dam in 1991 and 1993.

Based on this drilling, and to assess the contained resource, 22 vertical sections were constructed at 50m intervals in an east-west orientation. In total 98 boreholes were included in this resource estimation; including 69 boreholes of 'Block 1' drilled at 50x50m grid, and 29 boreholes of 'Block 2' drilled at 150x150m grid.

The surface area of the tailings dam considered for the resource estimation is 870x1,160m.

11.6.2 Categorisation and Reserve Distribution per Block

Based on the drilling density it is possible to distinguish two resource 'blocks'. Block 1 occupies the western area of the tailings dam on a 50m grid and is presented on profiles 8 – 19 and is considered to represent C_1 category 'reserves'. Block 2 occupies the remainder of the tailings dam and is studied largely on 150m spacing and represents C_2 category 'reserves'. It should be noted that when estimating the 'reserves' in Block 2 the bordering drillholes from Block 1 were included for grade determination.

11.6.3 Grade and Density Determination

Previous studies have considered the grade distribution within the tailings deposit and the use of a gold cut-off-grade (COG) in its evaluation. The VNIITSVETMET Institute considered that a 1.48g/t Au COG be applicable and several other studies considered separating the low grade (<0.5g/t Au) and high grade (>3.0g/t Au) zones into definable areas. However, mechanical extraction of tailings material and the gold distribution throughout the deposit means that selective mining is not an option and the deposit as a whole is considered economically viable.

Consequently the entire drillhole intersection, along each vertical profile, is considered in determining the average grade.

No cut-off has been applied by Kazzinc to the mineral resource. Equally no grade factors or top-cuts to grades have been applied in estimating the mineral resource grades.

Average grade is determined by its weighted average over the entire sample length, exceptions are boreholes from 1969 where the entire thickness of tailings equates to 1 sample.

In addition two boreholes from this early period (№ 14 and 36) have high gold content of 10.8 and 11.5g/t Au respectively (14 also containing 47.7g/t Ag). However while estimating the 'reserves' of Block 1 these were omitted. These gold values are more than 5 times the average value of the block (3 times more for silver).

The bulk weight was accepted by comparison with previous 'reserve' estimation and on the basis of earlier research (KazGIIZ, 1964, 1970-1971) which is accepted as being 1.6t/m3.

Estimation of block section areas, using 1:1,000 scale, was carried out with the aid of computer generated drawings (AutoCAD-14).

11.6.4 Historical 'Reserve' Estimation

A summary of the 'reserve' data for gold and all metals per separate blocks are shown in Table 10.11 below. Analysis of these data shows that one part of the tailings dam drilled at 50x50m grid (Block 1: C1 'reserve' category) contains 3,087kt with an average grade of 2.04g/t Au, 17.27g/t Ag, 0.05% Cu, 0.42% Pb and 1.06% Zn. The main percentage of these reserves concentrates from profiles 8 to 14. Interestingly, profiles 15-19 are characterised by an average gold content higher than from profiles 8-14.

Block 2 contains 'reserves' of C_2 category equal to 9,498kt with an average grade of 1.03g/t Au, 12.40g/t Ag, 0.04% Cu, 0.28% Pb and 0.66% Zn.

A comparison of the historic estimation figures are presented in Table 11.11 below.

Table 11.11: Historical 'Reserve' Estimates

	Tonnes (kt)	Grade (g/t)		Metal (kg)		Notes
		Au	Ag	Au	Ag	
1969 (LPC)	2,191.84	2.80	19.70	6,337.2	43,179.2	Western part of tailings dam only
1974 (LPC)	472.56	1.54	16.50	729.5	7,777.0	
Total 1969 and 1974	2,664.40	2.65	19.12	7,066.7	50,956.2	
1994 (LPC)	**12,454.54**	**1.25**	**13.07**	**15,538.3**	**16,2751.6**	
Block 1	3,053.29	2.01	16.94	6,137.1	51,722.8	
Block 2	9,401.25	1.00	11.81	9,401.2	111,028.8	
2000 (LLP «Geos»)	**12,585.0**	**1.28**	**13.59**	**16,103.0**	**171,065.0**	
Block 1	3,087.0	2.04	17.27	6,310.0	53,327.0	
Block 2	9,498.0	1.03	12.40	9,793.0	117,738.0	

Table 11.11 shows that the western part of the tailings dam (Block 1) contains 24.5% of the total gold 'reserve' bearing sands but almost 40% of the contained gold and 31% of the contained silver. The average gold grade in Block 1 is twice that in Block 2.

According to the estimate made in 1969, the far western part of the tailings dam is characterised by an elevated gold content with a grade of 2.8g/t Au and it is estimated that the protective pillar of the railroad contains 450kt with an average grade of 2.89g/t Au.

Estimation of the C_1 category 'reserves' shows the following results:

- average thickness 13.62m;
- total block area 156,350m^2;
- weighted average gold grade is 2.02g/t;
- total 'reserve' at 1.6g/cm^3 bulk weight is 3,407kt;
- contained gold is 6,882.5kg (214koz).

Due to the low values of copper and lead (oxidised form) in the tailings, and the inability to recover these elements at this time, they are not included within the 'reserve' estimation.

Differences in the historic 'reserve' estimation are considered a result of several factors as follows. The grain size distribution is represented by fine sand and 'slime' fractions of <0.1mm. The tailings dam has a surface area of approximately 0.6km^2 and is open to the elements, apart from in the far southwest which is occupied with a shallow pond. Some parts of the dam area have also succumb to incursion of poplar trees.

Climatic conditions, high and prolonged periods of windy conditions, are therefore considered to have effectively removed surface layers of the dam and further losses have occurred as a result of storm water (rain and flooding) and washout from the Phillipovka River.

In addition an unreliable metallurgical balance from the Leninogorskaya Concentrator in the 1920's and 30's may also have an impact on these historic discrepancies.

It should also be noted that some areas of the Staroye tailings dam were used, and therefore removed, as inert filler for producing mortar and finishing mixes as well as concrete for local urban development.

11.7 WAI Mineral Resource Estimate

11.7.1 Topography

Two digital terrain models (DTMs) were created for the Staroye TMF. The first DTM was of the current surface level of the TMF (dated 09/09/2010) and was created from information supplied by Kazzinc.

The survey file was drawn as a set of planar strings which were imported and manipulated in Datamine Studio 3 to the correct elevations. The DTM was then created from these strings.

The second DTM is of the base of the TMF. This was created primarily using information from 22 cross sections (dated 01/01/2000) drawn across the deposit at an azimuth of 080˚ showing the base of the TMF based on drilling results. These sections were supplied by Kazzinc in five separate dxf format files. Stings were created within Datamine Studio using this information and from these an initial DTM was created. This DTM was then adjusted to take in to account the drilling information between the section lines.

11.7.2 Database Compilation

All of the sample data collated for the current project are summarised below in Table 11.12. A general plan of these drillholes are shown in Figure 11.5. The holes were drilled in 5 campaigns concentrating on different areas of the TMF with the 1991 campaign primarily being for the purpose of verification of older holes. The

1969 campaign, mainly concentrated in the west of the TMF, was drilled on a close spaced pattern with holes drilled on a grid 50x50m with profile lines aligned towards 080˚ azimuth. The 1974 campaign was essentially an extension of this 50m x 50m grid in to the centre of the TMF. The holes drilled between 1993 and 1994 were mainly in the eastern part of the TMF on an irregular basis with holes usually between 80m and 110m apart. All holes were drilled vertically.

Table 11.12: Sample Data Summary

Campaign	Type	No. Of Holes	No. Of Samples	Holes Lengths (m)	
				Total	Average/Hole
1969	Cable Percussion	**47**	47	593	13
1974	Cable Percussion	**24**	104	387	16
1991	RC	**3**	42	42	14
1993	Vibrocore	**1**	19	19	19
1993-1994	Vibrocore	**23**	291	333	14

Different sampling methods were used for different stages of the exploration of the Staroye TMF. In the 1969 drilling program only one sample for each hole was collected, effectively a composite sample over the entire length of each hole but only 1kg. During the 1974 campaign samples were generally collected over a length of 5m with an irregular sample length at the end of the hole though these end of hole samples range from 1-7m in length indicating an inconsistent strategy. These samples consisted of 10-12kg of material. For the 1991 and later holes samples were taken from each metre of drilling with each sample consisting of 19-20kg of material.

Statistical analysis was carried out on the samples by campaign to identify any potential bias that may be present within the data. The campaigns for which the most samples were taken show similar variances and roughly log-normal populations. The campaigns show differences in mean grade but no significant bias is considered to be present as a result of the different sampling methods.

The campaigns show differences in mean grade of Au. The basic statistics for Au by year sampled are listed in Table 11.13 and this shows a general decrease in Au grade for the later campaigns apart from the 1991 holes drilled as twinned holes amongst the 1969 holes. This is expected because of the locations of the different campaigns and the recorded decrease in discharge grade over time. The initial campaign was located close to the discharge point and subsequent campaigns more distal. The initial deposition of tailings was very high grade at 11.1g/t Au in 1926/27 dropping to 2.15g/t Au by 1936 and then dropping to 0.5g/t Au by 1953 as plant recovery improved (see Figure 11.4). The higher grade material is concentrated around the discharge point towards the south west of the site where the earlier campaigns were concentrated. Because of this and the similar variances shown by the campaigns no significant bias is considered to be present from the different campaigns. However, RC drilling is not considered to be a suitable method of sampling for the material and so the three holes from 1991 were removed from the database.

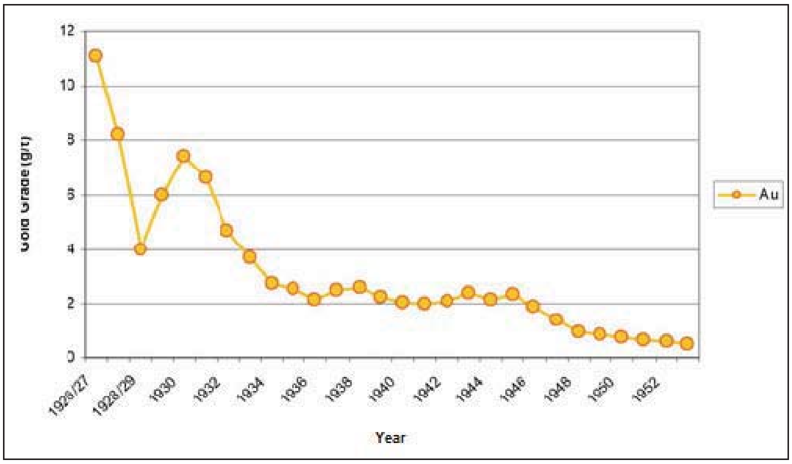


Figure 11.4: Average Au Grade of Deposited Tailings over Time (Based on LPC Data)

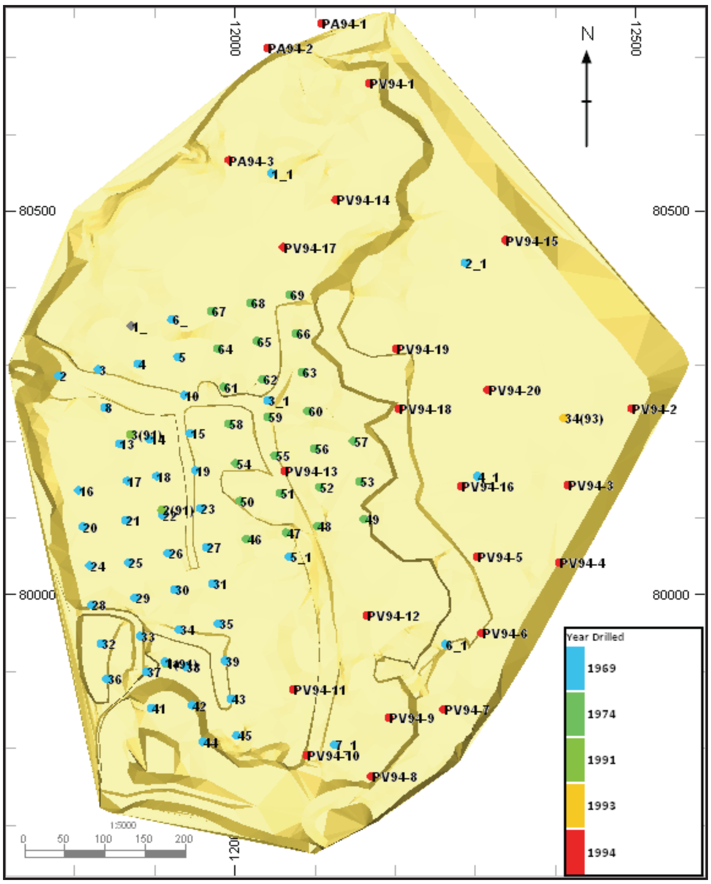


Figure 11.5: Plan View of Current Survey of Staroye TMF with Borehole Locations by Year Drilled

Table 11.13: Statistical Summary of Au by Year Drilled								
Year	Number	Minimum	Maximum	Mean	Variance	Standard Deviation	Log Estimate of Mean	Coefficient of Variation
1969	47	0.80	11.50	2.49	3.90	1.97	2.42	0.79
1974	104	0.00	3.20	0.88	0.53	0.73	1.16	0.83
1991	42	0.30	4.80	1.76	0.85	0.92	1.83	0.52
1993	19	0.50	1.10	0.81	0.03	0.16	0.81	0.20
1994	292	0.00	4.00	0.82	0.25	0.50	0.87	0.61

11.7.3 Sample Data Processing

The DTMs of the base and the current surface of the TMF were used to select and allocate the drillhole data. The data selected is summarised in Table 11.14. Most of the rejected samples are from parts of the TMF that have already been excavated. For those holes where only a single sample was taken the samples were reduced in length to fit to the current surface topography.

Table 11.14: Selected Sample Summary				
	No. Of Holes	No. Of Samples	Holes Lengths (m)	
			Total	Average/Hole
1969	44	44	208	5
1974	23	33	132	6
1993	1	17	17	17
1993-1994	24	181	210	9

All of the drillhole samples have been assayed for Au, Ag, Cu, Pb and Zn. Most of these indicate a single roughly log-normal distribution. A statistical summary of the selected samples is shown in Table 11.15 below.

Table 11.15: Statistical Summary of Selected Samples								
FIELD	Number	Minimum	Maximum	Mean	Variance	Standard Deviation	Log Estimate of Mean	Coefficient of Variation
AU	271	0.00	11.50	1.11	1.34	1.16	1.15	1.04
AG	271	0.00	47.70	11.92	40.30	6.35	12.11	0.53
CU	271	0.00	0.69	0.04	0.00	0.05	0.04	1.04
PB	271	0.00	2.33	0.33	0.05	0.23	0.33	0.70
ZN	271	0.00	3.20	0.69	0.20	0.45	0.70	0.64

A decile analysis was completed on the selected gold grades, as shown in Table 11.16 below. From this and the log-probability plot a top cut of 4g/t Au was applied to the selected

Table 11.16: Decile Analysis of Au Grades							
Q%_FROM	**Q%_TO**	**NSAMPLES**	**MEAN**	**MINIMUM**	**MAXIMUM**	**METAL**	**METAL%**
0	10	27	0.16	0.00	0.30	4.39	1.43
10	20	28	0.41	0.30	0.50	11.50	3.75
20	30	27	0.54	0.50	0.60	14.60	4.77
30	40	28	0.61	0.60	0.70	16.95	5.53
40	50	27	0.74	0.70	0.80	20.00	6.53
50	60	28	0.83	0.80	0.90	23.34	7.62
60	70	27	1.04	0.90	1.20	27.95	9.13
70	80	28	1.40	1.20	1.70	39.18	12.79
80	90	27	1.94	1.70	2.20	52.25	17.06
90	100	28	3.43	2.25	11.50	96.10	31.38
90	91	2	2.28	2.25	2.30	4.55	1.49
91	92	3	2.38	2.34	2.40	7.14	2.33
92	93	3	2.45	2.40	2.50	7.35	2.40
93	94	3	2.51	2.50	2.54	7.54	2.46
94	95	3	2.63	2.60	2.65	7.90	2.58
95	96	2	2.80	2.80	2.80	5.60	1.83
96	97	3	2.95	2.86	3.00	8.86	2.89
97	98	3	3.30	3.20	3.40	9.90	3.23
98	99	3	3.65	3.46	3.82	10.96	3.58
99	100	3	8.77	4.00	11.50	26.30	8.59
0	**100**	275	1.11	0.00	11.50	306.26	100.00

The most common sample interval in the selected sample set was 1m, but a significant amount (30%) of the samples, from the earlier drilling campaigns, are longer than this. A composite length of 5m was chosen for the data. This length was chosen to ensure that there was no significant decompositing of data which would lead to an unrealistic view of the data which would suggest less sample variability than actually exists. Approximately 5% of samples were logged as being longer than 5m; all these were from the 1969 campaign from which a single sample was assigned to the entire drillhole length.

A statistical summary of the composites is shown in Table 11.17. The statistical summary shows an increase in grade from the sample mean because the longer samples from the west of the TMF, which are effectively decomposited, are generally of higher grade than the 1m samples which are drilled in the east of the deposit. The decompositing leads to these holes having a greater influence on the mean grade. The effect of this during grade estimation was mitigated by the estimation parameters used as described below.

Table 11.17: Statistical Summary of Composites								
FIELD	**Number**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**Log Estimate of Mean**	**Coefficient of Variation**
AU	131	0.00	4.00	1.43	0.84	0.91	1.46	0.64
AG	131	0.00	47.70	14.37	43.05	6.56	14.45	0.46
CU	131	0.00	0.29	0.05	0.00	0.03	0.05	0.61
PB	131	0.00	2.33	0.39	0.11	0.33	0.38	0.83
ZN	131	0.00	3.20	0.86	0.23	0.48	0.88	0.56

11.7.3.1 Variography

Variography was undertaken test for any spatial continuity of the TMF metal grades, and to assist with the selection of suitable search parameters upon which to base the resource estimation.

For the vertical direction variography was carried out using the 1m sample data from the holes drilled in the 1993-1994 campaigns as these were considered the most suitable. The holes from the earlier campaigns give an unrealistic view of grade continuity downhole as they were sampled over a much greater length.

Variography was carried out horizontally on the 5m composite data contained within the modelled TMF zone. Absolute, as well as relative variograms were generated for AU grades, with a spherical model being used for modelling purposes. Variograms were generated and modelled for the principal directions at azimuths 080˚ and 350˚ in which significant anisotropy was apparent.

These indicated ranges of approximately 233m towards 080˚, 52m towards 350˚ and 2m vertically.

11.7.4 Block Modelling

A model prototype was set up with a 20x20x5m parent block size. The details of the prototype are shown in Table 11.18 below. This prototype was then used as the basis to set up a volumetric model, as controlled by the TMF wireframes. Sub-blocks were generated within the structure near the edges, in order to better fit the wireframe geometry, down to minimum dimensions of 5x5x0.5m.

Table 11.18: Block Model Prototype

	Origin	Maximum	Distance	Block Size	Number
X	11,700	12,600	900	20	45
Y	79,600	80,800	1,200	20	60
Z	700	780	80	5	16

11.7.5 Density

An overall average density value of 1.60t/m^3 was used.

11.7.6 Grade Estimation

Grade estimation was carried out using Inverse Power of Distance Squared (IDW^2) for all metal grades, although Ordinary Kriging (OK) and Nearest Neighbour (NN) were also used for comparative purposes on the AU grade.

The estimation was run in a three pass plan, the second and third passes using progressively larger search radii to enable the estimation of blocks unestimated on the previous pass. Grades were estimated using the 5m composite drillhole data. The search parameters were derived from the variographic analysis, with the first search distances corresponding to the distance at 2/3^{rds} of the variogram sill value and the second search distance approximating up to the variogram range. Block discretisation was set to 5x5x2 to estimate block grades. Sub-cells received the same estimate as the parent cell. A summary of the estimation parameters is shown in Table 11.19.

Table 11.19: Estimation Parameters

Search Distances (m)						No. Of Composites	
1st Search			2nd Search				
350˚	80˚	Vertical	350˚	80˚	Vertical	Min	Max
34	150	1	50	230	2	3	5

- 3rd search up to 500m with a minimum of 1 composite required
- IPD^2 used for Au, Ag, Cu, Pb, Zn grades

11.7.6.1 Resource Classification

The resource classification for the Staroye TMF is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves JORC Code (2004).

In the case of the Staroye TMF the resource classification is driven by sample quality and sample spacing. Approximately 47% of holes were drilled in the 1969 campaign and 24% of holes were drilled in the 1974 campaign. The sampling methodology for these campaigns is unlikely to give a true representation of the layered nature of the TMF and therefore variations in grade vertically. The 1969 drilling campaign assumes constant grade over the hole length from a relatively small sample and the 1974 campaign assumes sample continuity over 5-7m lengths. This is at odds with the information from the holes drilled in the 1993-1994 campaign which were sampled at 1m intervals.

The reliability of grades recorded in the 1969 and 1974 campaigns is questionable and these holes are the main source of information for half of the TMF. Over the rest of the TMF, though samples are of better quality, holes are irregularly spaced up to 200m apart in the direction of 080˚ azimuth and generally over 100m apart in the direction of 350˚ azimuth. The variographic studies carried out during this resource estimation suggest a drillhole spacing of 200x50m in these directions would be required for classification as an Indicated resource. This drillhole spacing is only present in the Staroye TMF where the holes are of uncertain quality.

The following criteria were established for the classification of resources:

Measured: No material was in this category, owing chiefly to lack of closely spaced sample data;
Indicated: Blocks within this category had to be covered by at least 50x50m spaced drillhole data; and
Inferred: All other blocks within the TMF, but outside of 50x50m drilled areas, were assigned as inferred.

A plan view showing this resource classification methodology is shown in Figure 11.6.

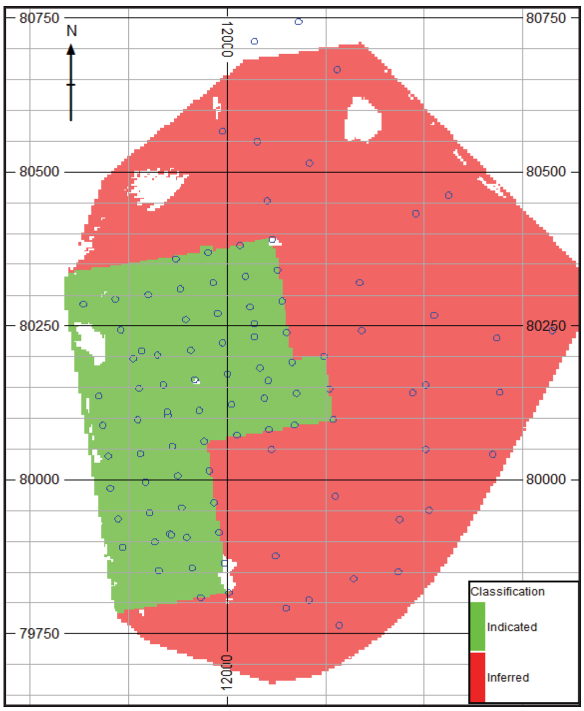


Figure 11.6: Plan View of the Staroye TMF Showing JORC Classification

11.7.7 Validation

Following grade estimation a statistical and visual assessment of the block model was undertaken to assess successful application of the estimation passes and to ensure that as far as the data allowed, all blocks within the defined TMF domain were estimated. The model validation methods carried out included:

- A visual assessment of grade;
- Global statistical grade validation; and
- Model grade profile (swath plot) analysis.

A visual comparison of composite sample grade and block grade was conducted in cross section and in plan. An example section through the block model, with the estimated AU grades, is shown in Figure 11.7. Visually the model was generally considered to spatially reflect the composite grades.

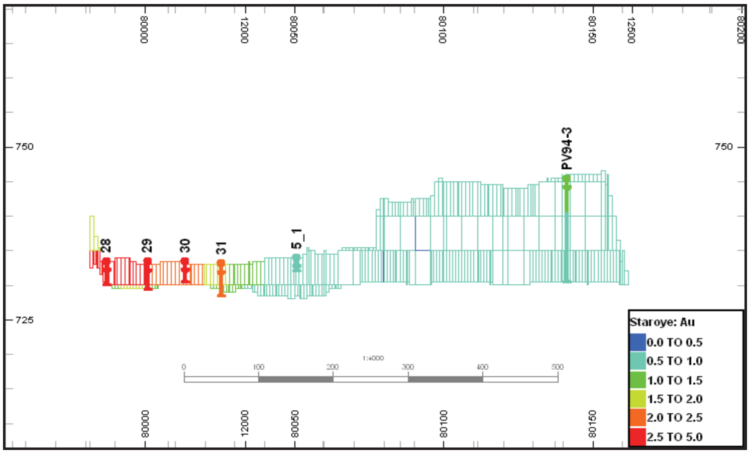


Figure 11.7: Vertical Section at 10 x Vertical Exaggeration
(Approximately E-W Section Showing Composite Vs Block Au Grades)

A comparison of global average grades is summarised below in Table 11.20. As well as the principal inverse-distance grades in the block model, this table includes alternative estimates made using nearest neighbour and ordinary kriging. Overall these global average grades compare very well to the original sample grades and not the mean composite grade which is slightly skewed as a result of decompositing of several of the higher grade samples from the earlier campaigns which had a single grade assigned to their entire lengths.

Table 11.20: Comparison of Global Average Grades

FIELD	Unit	Average Grades		Block Model Grades		
		Samples	Composites	ID	NN	OK
AU	g/t	1.14	1.44	1.05	1.01	1.06
AG	g/t	12.02	14.34	12.09	-	-
CU	%	0.04	0.05	0.04	-	-
PB	%	0.33	0.39	0.32	-	-
ZN	%	0.72	0.86	0.69	-	-

Notes:

1. No cut-off grades applied
2. OK (ordinary kriging), NN (nearest neighbour), ID (inverse distance)

Model grade profiles (swath plots) were generated from the model by averaging both the drillhole composites and blocks along 20m-spaced E-W slices. Overall the drillhole composite average grades and the block model average grades compare extremely well.

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.

11.7.8 WAI Resource and Reserve Evaluation

The resource classification for the Staroye TMF 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 of the TMF was used as the basis for resource evaluation. Summary results of the evaluation of the resources are shown in Table 11.21.

No cut off has been applied as it is assumed that there will be no selection of material mined and the whole deposit will be processed.

Table 11.21: Staroye Tailings Mineral Resource Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AuEq (g/t)
Indicated	0.82	2.01	18.78	0.05	0.48	1.11	3.19
Inferred	5.90	0.91	11.16	0.04	0.30	0.63	1.72

Notes:

- AuEq calculation based on prices of:
  - Au 1287 US$/oz
  - Ag 23 US$/oz
  - Cu 7341 US$/t
  - Pb 2422 US$/t
  - Zn 2420 US$/t

WAI has calculated Ore Reserves based upon the above Mineral Resource Estimate. WAI has applied losses of 4.4% based upon historical data, and dilution of 0.5% to take into account minimal dilution from the bunds and floor.

Table 11.22: Staroye Tailings Ore Reserve Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))

Classification	Tonnes (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (koz)	Grade (g/t)	Metal Content (koz)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)
Proven	-	-	-	-	-	-	-	-	-	-	-
Probable	0.79	2.00	50.97	18.69	475.89	0.05	0.38	0.48	3.80	1.10	8.72
Total	***0.79***	***2.00***	***50.97***	***18.69***	***475.89***	***0.05***	***0.38***	***0.48***	***3.80***	***1.10***	***8.72***

12 CHASHINSKOYE TAILINGS DEPOSIT

12.1 Introduction

12.1.1 Location & Access

The Chashinskoye tailings dam contains waste material discharged from the concentrator of the Leninogorsk Polymetallic Combinat (LPC) during the period 1953 to 1978 inclusive, after the closure of the Staroye tailings dam.

Photo 12.1: General View of the Chashinskoye Tailings Dam (looking northeast)

The Chashinskoye tailings dam is located on the southern side of the Filipovka River valley, opposite to the confluence with the Chashino stream, and some 3.5km east of the, formerly named, Leninogorsk Mining and Concentrating Complex (MCC) at Ridder (Photo 12.1). The impoundment is readily accessible from the Ridder-Chekmar deposit graded road, which runs along the northern margin of the site and is a road distance from the Kazzinc office at Ridder of approximately 11km.

The tailings were deposited by the LPC during the period 1953-1978, after the closure of the Staroye tailings site, and cover an area of some 2.2km (ESE) by 1.7km (NNE). The tailings dam was filled by the products of processing Ridder-Sokolny and Tishinsky polymetallic and pyrite-polymetallic ores at Leninogorsk Concentrator. Up to 1964 only Ridder-Sokolniy tailings were deposited, and since 1965 both Ridder-Sokolny and Tishinsky tailings were deposited. In addition, and over different time periods, the concentration plant processed insignificant amount of other ores from the deposits throughout the former Soviet Union (oxidized ores of the Far East, Zhairem deposit ores; Dzhidinskoye and Gorevskoye deposits' ores in Trans-Baikal).

Tailing deposits generated in previous years of Ridder-Sokolny processing were dumped in the Staroye tailings dam. Since late 1978 tailings from the LPC have been deposited into the new Talovskoye tailings dam.

From 1973, and for many years, some of the deposited tailings were sent to the Irtysh Polymetallic Combinate to be processed as fluxes. From Chashinskoye tailings dam some 103,185t (in 1973-1983, 1987) was removed for this purpose.

The initial dam, which was constructed at the northern end of the impoundment, is now entirely covered by tailings. The current earth dam runs along the western and northern margins of the impoundment. The crest of the dam is at an elevation of 825-827m above Baltic Sea datum whist the elevations of the top surface of the tailings are generally in the range of 815-825m. The average thickness is 17.25m with a maximum thickness of 60m at a trough which extends over an area of 900m (ESE) by 100-350m, adjacent to the northern dam.

The south-eastern part of the site is utilised as a settlement pond for water pumped from the Talovkoye tailings site. The depth of the pond varies from less than a metre to over 5m; and as a consequence no drilling has occurred in this area of the tailings dam.

The base of the tailings impoundment is composed of fine grained calcareous loam, locally with abundant rock debris. The thickness of the loam layer reportedly varies from several metres to 50m and due to its high clay content this layer is considered impervious.

12.1.2 Mineral Rights and Permitting

Kazzinc holds the right to mine the Chashinskoye tailings under the terms of Contract No 559 dated 7 November 2000. The same contract covers the nearby Staroye tailings and is valid for 30 years until 6 November 2030. The Chashinskoye mining lease covers an area of 265.1ha, the boundaries of which are defined by 56 corner points as detailed in Table 12.1 and shown in Figure 12.1.

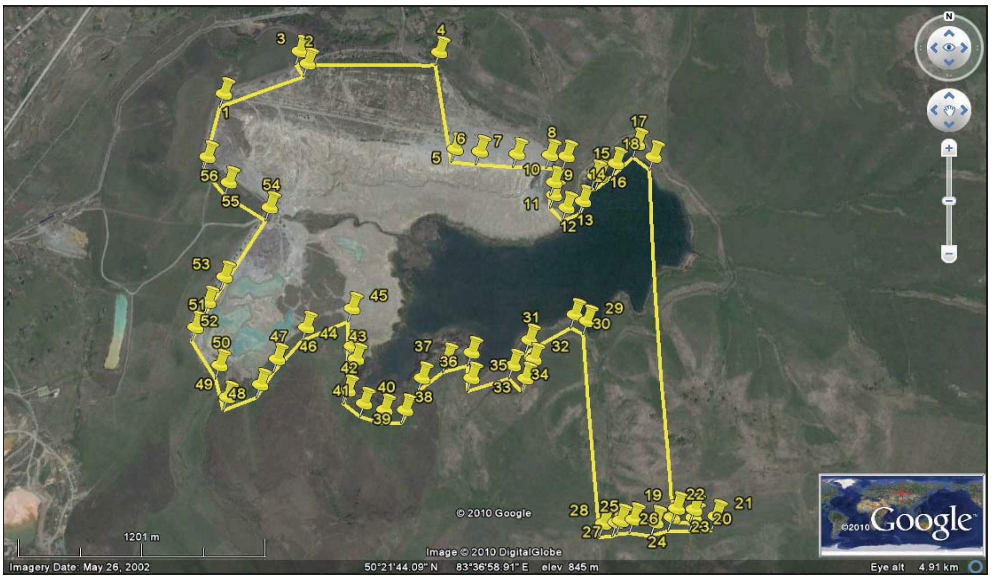


Figure 12.1: Chashinskoye Licence Outline

Table 12.1: Chashinskoye Tailings Mining Lease Boundaries

Boundary Points	Geographical Co-ordinates		Local Rectangular Co-ordinates		Boundary Points	Geographical Co-ordinates		Local Rectangular Co-ordinates	
	Latitude N	Longitude E	X	Y		Latitude N	Longitude E	X	Y
1	50°22'10"	83°34'48"	29,125.0	-11,875.0	29	50°21'34"	83°36'16"	28,030.0	-10,130.0
2	50°22'15"	83°35'08"	29,280.0	-11,482.5	30	50°21'35"	83°36'13"	28,040.0	-10,202.5
3	50°22'17"	83°35'06"	29,372.5	-11,127.5	31	50°21'31"	83°36'02"	27,911.0	-10,420.0
4	50°22'17"	83°35'40"	29,355.0	-10,837.5	32	50°21'28"	83°36'03"	27,827.0	-10,400.0
5	50°22'1.3"	83°35'44"	29,225.0	-10,760.0	33	50°21'25"	83°36'01"	27,795.0	-10,442.5
6	50°22'0.9"	83°35'50"	29,102.5	-10,650.0	34	50°21'27"	83°35'58"	27,816.0	-10,495.0
7	50°22'0.5"	83°35'59"	28,985.0	-10,465.0	35	50°21'25"	83°35'48"	27,731.3	-10,685.2
8	50°22'0.3"	83°36'07"	28,917.5	-10,309.0	36	50°21'29"	83°35'48"	27,877.5	-10,700.3
9	50°22'0.1"	83°36'11"	28,844.0	-10,237.5	37	50°21'28"	83°35'42"	27,836.2	-10,818.3
10	50°21'56"	83°36'08"	28,704.0	-10,295.0	38	50°21'25"	83°35'36"	27,740.5	-10,923.1
11	50°21'54"	83°36'08"	28,595.0	-10,295.0	39	50°21'20"	83°35'32"	27,604.9	-11,001.3
12	50°21'52"	83°36'11"	28,560.0	-10,238.0	40	50°21'20"	83°35'27"	27,594.6	-11,101.1
13	50°21'53"	83°36'15"	26,600.0	-10,155.0	41	50°21'21"	83°35'22"	27,637.8	-11,205.3
14	50°21'57"	83°36'18"	28,735.0	-10,102.5	42	50°21'23"	83°35'18"	27,698.9	-11,277.4
15	50°21'57"	83°36'19"	28,780.0	-10,067.5	43	50°21'28"	83°35'20"	27,823.5	-11,236.1
16	50°21'59"	83°36'23"	28,785.0	-9,995.0	44	50°21'30"	83°35'19"	27,910.6	-11,262.3
17	50°22'02"	83°36'28"	28,880.0	-9,890.0	45	50°21'36"	83°35'19"	28,081.0	-11,272.5
18	50°22'00"	83°36'32"	28,815.0	-9,810.0	46	50°21'33"	83°35'08"	27,985.7	-11,480.2
19	50°21'5.6"	83°36'36"	28,680.0	-9,740.0	47	50°21'28"	83°35'01"	27,851.1	-11,621.7
20	50°21'5.5"	83°36'40"	26,657.5	-9,662.5	48	50°21'24"	83°34'57"	27,719.7	-11,700.6
21	50°21'4.5"	83°36'45"	28,351.0	-9,557.5	49	50°21'22"	83°34'49"	27,658.1	-11,850.9
22	50°21'4.2"	83°36'40"	28,265.0	-9,667.5	50	50°21'27"	83°34'47"	27,798.4	-11,905.1
23	50°21'4.2"	83°36'35"	28,248.0	-9,765.0	51	50°21'33"	83°34'41"	27,984.3	-12,021.8
24	50°21'3.5"	83°36'31"	28,040.0	-9,640.0	52	50°21'37"	83°34'44"	28,132.0	-11,955.8
25	50°21'3.9"	83°36'26"	28,175.0	-9,938.0	53	50°21'41"	83°34'48"	28,257.7	-11,879.3
26	50°21'3.7"	83°36'23"	28,100.0	-10,000.0	54	50°21'52"	83°34'59"	28,581.1	-11,660.5
27	50°21'3.4"	83°36'22"	27,995.0	-10,028.0	55	50°21'56"	83°34'49"	28,700.8	-11,845.2
28	50°21'3.2"	83°36'19"	27,955.0	-10,088.0	56	50°22'00"	83°34'44"	28,832.2	-11,941.0

12.1.3 Production History

Based on the LPC records some 95.4Mt of tailings were discharged during the period 1953-1978. Of these 3.1Mt were taken directly from the LPC for backfill and other purposes and the balance (92.3Mt) deposited at the Chashinskoye impoundment. During the period 1963-1978 a further 4.4Mt of tailings were removed from three shallow pits in the western part of the impoundment for use as backfill and construction material. This leaves a balance of 87.9Mt remaining in the current impoundment site (Table 12.2 below).

Table 12.2: Material Disposal at Chashinskoye Tailings Dam

Year	Tailings Deposited (t)	Grade		Tailings Removed		Sub-total (t)
		Au (g/t)	Ag (g.t)	From LPC (t)	From Dam (t)	
1953	2,002,529	0.50	10.00			2,002,529
1954	2,390,667	0.67	9.18			4,393,189
1955	2,741,804	0.62	8.49			7,134,993
1956	2,875,690	0.55	9.30			10,010,683
1957	2,961,119	0.58	10.70			12,971,802
1958	3,046,960	0.76	9.40			16,018,762
1959	3,156,682	0.87	8.90			19,175,444
1960	3,333,322	0.78	8.80			22,508,766
1961	3,423,848	0.87	8.24			25,932,614
1962	3,506,688	1.08	7.11			29,439,302
1963	3,602,690	0.89	5.64		23,800	33,018,192
1964	3,614,513	0.80	6.26		4,200	36,628,505
1965	3,717,838	0.81	5.21		2,700	40,343,643
1966	4,071,120	0.87	5.29		1,400	44,413,363
1967	4,244,248	0.72	5.26		54,800	48,602,811
1968	4,263,521	0.70	4.01		30,000	52,836,332
1969	4,352,837	0.63	3.62	55,543	62,350	57,071,276
1970	4,370,394	0.63	4.19	228,792	47,570	61,165,308
1971	4,434,219	0.59	4.16	167,751	36,410	65,395,366
1972	4,521,107	0.71	4.95	318,560	134,758	69,463,155
1973	4,600,214	0.63	4.41	315,500	81,025	73,666,844
1974	4,533,428	0.62	3.82	334,893	79,311	77,786,068
1975	4,683,286	0.52	3.16	438,842	139,257	81,891,255
1976	4,380,272	0.47	2.70	510,933	110,060	85,650,534
1977	4,514,692	0.39	4.51	467,814	99,244	89,598,168
1978	2,079,617	0.45	4.43	277,092	172,149	91,228,544
1979-1992					3,311,417	87,917,127
Total	**95,423,298**	**0.68**	**5.81**	**3,115,720**	**4,390,451**	**87,917,127**

With regards to the gold grade, tailings deposited during the period from 1958 to 1970 averaged more than 0.6g/t Au, with a high of 1.08g/t Au in 1962, though on the whole remained fairly constant (≈0.7g/t Au). Gold grade dropped below 0.5g/t Au in 1976 while silver grade shows a gradual drop from 10.7g/t Ag in 1957 to less than 5.0g/t Ag from 1968 onwards, as shown in Figure 12.2.

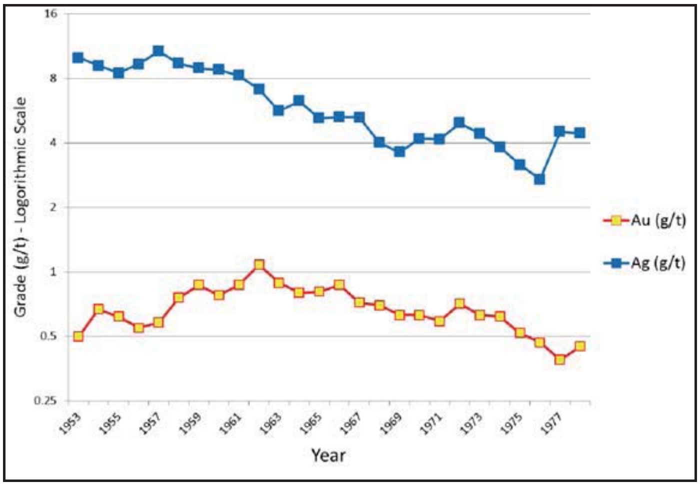


Figure 12.2: Production Grades from 1953 to 1977

The composition of the tailings should correspond to the types of ore processed at LPC during the period of 1953-1978, with inevitable secondary changes due to oxidation of sulphides and leaching of metals. Over the first 12 years from 1953 the feed for the concentrator consisted almost exclusively of auriferous polymetallic sulphide ores from the Ridder-Sokolniy mine. From 1965, the LPC also treated pyritic-polymetallic ores from the Tishinsky mine. The proportion of Tishinsky ores varied from 1.4-16% of the total ore feed, for the initial 3 years, but increased to 20-30% from 1969 onwards. The overall contribution of the Tishinsky ore to the end of 1978 was 14%.

Mining and treatment of copper-zinc and copper ores from the Ridder-Sokolniy mine began in 1970 and gradually increased to 23% of the total Ridder-Sokolniy mine output by 1978. Overall, the proportion of these ores to the total ore processed at LPC during the period to 1978 inclusive was estimated at 5%. By inference, therefore, the proportion of tailings derived from processing of auriferous polymetallic sulphide ores should be approximately 20% of the total tailings deposited at the Chashinskoye site.

It has also been estimated that sulphide ores made up 89% of ore processed at the LPC during the period from 1953 to 1978. The balance came from processing of oxidised ores mined at shallow levels of the Ridder-Sokolniy deposit and from the open pit at the early years of the Tishinskiy mine operation.

The LPC also periodically processed ores from various polymetallic mines in the former Soviet Union, including oxidised and sulphide ores from the Zhayrem deposit in central Kazakhstan and from the Gorevskoe deposit in the Transbaikal region. However, the proportion of tailings derived from those sources is not considered to be significant.

12.2 Geology and Mineralisation

The Chashinskoye tailings dam is a man-made feature and does not possess any recognised geological structure. However, there is some evidence that in vertical section the tailings display a layered structure in terms of fluctuating gold grade though there is no discernable pattern.

Deposition of tailings into a tails dam is largely driven by the beaching characteristics and the water accumulation on the surface of the dam. Because of this very little continuity can be expected to exist. As the tailings are deposited in layers some minor vertical continuity may exist as the coarser/heavier particles tend to drop out first and the lighter finer particles tend to migrate more into the centre of the dam. However, this is further complicated by channelling which can cause coarser grained material to deposit closer to the centre.

The tailings are the waste products of the processing of gold and polymetallic ores (primarily silver, copper, lead and zinc) from the Ridder Mining and Concentrating Complex and their composition thus reflects the major constituents of the ore.

12.2.1 Mineralisation

According to a mineralogical examination conducted at the laboratory of Kazzinc concentrator at Zyryanovsk (Kostorev 2007), 95% of material deposited (i.e. waste from the concentrator and not material sampled directly from the tailings) in the Chashinskoye tailings dam consists of quartz-sericite rock, with minor admixture of sandstone, quartz and carbonate. The observed particle size was in the range of 0.006-2.0mm and the content of primary sulphide minerals was less than 4% by volume, of which the pyrite content was estimated at 1.5-2% by volume. Ore minerals identified included sphalerite, chalcopyrite, galena and minor amounts of grey ore (tetrahedrite-tennantite group). Magnetite was an accessory mineral. The results of this study are summarised in Table 12.3. Secondary minerals mentioned in the above report (Kostorev 2007) include:

- Secondary sulphides (chalcocite and covelline), which occurred as films on surfaces of sphalerite and rims around chalcopyrite;
- Smithsonite, ochres of limonite and hematite; and
- Jarosite, which formed rims around pyrite grains and veinlets infilling microfractures within pyrite.

The examination was conducted on a sub-sample obtained from a 150kg sample of tailings, which contained 0.8g/t Au, 0.3% Zn, 0.1% Pb and 0.06% Cu. The report did not specify the location from which the sample was collected.

Table 12.3: Sulphide Mineralogy of Chashinskoye Tailings (*Kostorev* 2007)

Component	Sulphide Content (% by volume)		Size of Liberated grains (mm)	Composition and Size of Intergrowths (mm)
	Free	Intergrowths		
Sphalerite	51	49	0.006 to 0.1 Sporadically 0.12-0.045 to 0.1-0.36	ZnS (0.0015-0.075) intergrown with gangue (0.06-0.1)
Chalcopyrite	44	56	0.015 to 0.72-0.36	CuFeS2 (0.0015-0.054) intergrown with gangue (0.06-0.1)
Galena	35	65	0.008 to 0.03	ZnS (0.03-0.075)
Pyrite	≈100	0.006-0.06		
Gangue	90	10	0.006 to 2.0	CuFeS2 (0.22) intergrown with ZnS (0.075)
Magnetite			0.012-0.03	

The main ore minerals within the tailings material are pyrite, sphalerite, galena and chalcopyrite; as summarised below:

- **Sphalerite** is mainly found in aggregations with rocks, galena and chalcopyrite; sometimes with hydrous ferric oxides and smithsonite. Grain size – from 7-30 µm (more often) up to 100-300µm.
- **Galena** occurs in the form of inclusions in enclosing rock and sphalerite, in aggregations with pyrite, chalcopyrite, sometimes with anglesite. Grain size varies from 5 to 30µm, prevailing size being 7-15µm.
- **Pyrite** occurs in aggregations with rocks and sphalerite. Grain size – from 7 to 15-20µm, mainly 7µm.
- **Chalcopyrite** is found in aggregations with rocks, sphalerite and pyrite; grain size varies from emulsive impregnation to 7-40µm (prevailing grain size 10-20µm).

12.2.2 Gold and Silver Occurrence

The data on phase analysis of the Concentrator material deposited, including western Staroye tailings dam, show that gold is found in different forms: nuggets, aggregations, association with sulphide, and sometimes with rocks. About half of the gold occurs in aggregations; up to 1/3 of the gold is found in association with sulphides (pyrite, chalcopyrite and galena). Free gold amounts to around 10% (sometimes up to 20%). Gold grain size is 20-75µm and the grains are generally of plate-like or tabular form though sometimes found in isometric form.

Silver is mainly found in aggregations or association with sulphides, occasionally occurring as nuggets or sulphides (fahl[2] ores) or chlorides (cerargyrite[3]). Gold and silver mineralisation occurs in a variety of forms and associations (see Table 12.4) with the proportion of free gold roughly estimated at 10% of the total gold and with a dominant size range of 25-75µm.

Moderately rich and rich classes are found mainly the deepest part of the impoundment, along the western and northern margins. This is most likely due to gravity settling of heavier particles, including gold, close to the discharge outlet for the disposed tailings. Durnev and Golubtsov (2001) mentioned that tailings were generally deposited along the northern dam in warmer seasons and along the western dam during the winter period.

The distribution of gold in various grain size classes was investigated in 2-5kg samples collected from 12 drillholes (2, 7, 7/7, 10, 13, 17, 19, 21, 23, 25, 29 and 33) completed in 1992. Results showed that gold grades are generally in the range of 0.3-1.0g/t Au, with the higher grades correlating with the coarse grain fractions.

Table 12.4: Forms of Gold and Silver Occurrence in Chashinskoye Tailings Dam

Form of gold & silver occurrence	Collective flotation tailings sample № 7/72 (Qtr 4 1974)				Collective flotation tailings sample № 11/76 (Qtr 1 1965)				Disposed tailings (1974)		Quartz Tailings (1977)	
	Gold		Silver		Gold		Silver		Gold		Silver	
	g/t	%	g/t	%	g/t	%	g/t	%	Abs %	Rel %	Abs %	Rel %
Free	-	-	-	-	-	-	-	-	0.00004	20.0	0.008	38.1
With clean surface	0.03	7.0	ND	-	0.04	10.5	ND	-	-	-	-	-
Covered with oxidation films	0.02	4.6	ND	-	ND	-	ND	-	-	-	-	-
In intergrowths	-	-	-	-	-	-	-	-	-	-	-	-
With clean surface	0.23	53.5	0.85	44.7	0.19	50.0	0.91	50.3	-	-	-	-
Covered with oxidation films	0.01	2.3	0.28	14.7	0.02	5.3	ND	-	-	-	-	-
Associated with ore minerals	-	-	0.39	20.6	0.11	28.9	0.87	48.1	0.00016	80.0	-	-
1. Oxidised lead minerals	0.03	-	-	-	-	-	-	-	-	-	-	-
2. sphalerite	0.02	-	-	-	-	-	-	-	-	-	-	-
3. other	0.07	-	-	-	-	-	-	-	-	-	-	-
Sulphide forms	-	-	-	-	-	-	-	-	-	-	0.002	9.5
With gangue	0.02	4.6	0.38	20.0	0.02	5.3	0.03	1.6	-	-	-	-
Kerargyrite	-	-	-	-	-	-	-	-	-	-	-	-
Insoluble residue	-	-	-	-	-	-	-	-	-	-	0.011	52.4
Total	**0.43**	**72.0**	**1.90**	**100**	**0.38**	**100**	**1.81**	**100**	**0.0002**	**100**	**0.021**	**100**
Size of free gold (mm)	**0.025 – 0.075**											

[2] Grey copper ore (fahlerz)

[3] Former name for chlorargyrite; mineral form of silver chloride (AgCl)

12.2.3 Granulometry

Initial grain size composition of Chashinskoye tailing dam was defined by a number of boreholes located in 2 profiles: in profile 5 – boreholes 17, 19, 21, 23, 25 (1992-1993), in profile 1 – boreholes 103, 110, 116, 126, 129, 133 (1993).

According to the fragment classification (V.A. Priklonsky and V.V. Okhotin) several material size classes were combined which allowed a more general characterisation of tailings material. Subsequently three size classes were specified: > 0.56 mm – coarse, gravel sands; 0.56-0.1 – medium and fine sands; <0.1mm – fine sands, coarse dust, silty and clay particles.

Table 12.5: Grain Size Classification according to Priklonsky & Okhotin

Fractions by grain size (mm)							
>1.6 - 1.25	**1.25 - 0.8**	**0.8 - 0.56**	**0.56 - 0.28**	**0.28 - 0.10**	**0.10 - 0.071**	**0.071 - 0.04**	**<0.04**
"coarse, gravel sands"			**"medium and fine sands"**		**"fine sands, coarse dust, silty and clay particles"**		
Very coarse sand	Very coarse & coarse sand	Coarse sand	Medium sand	Fine sand	Ultra-fine sand	Ultra-fine sand & coarse dust	Silt & clay particles

The coarse fraction forms an insignificant part of the tailings volume with an average percentage from 0.2-8.0% (averaging to 3.3 %). The most common fractions are sand fraction (medium and fine sand) and fine fragmented fraction (ultra-fine sand, silt and clay particles).

The tailings dam was filled by pulp discharge from the main discharge pipeline to the dam's inner part, starter dam filled first. As the dam was filled the floodwalls (along which the pulp was discharged) were subsequently built up by tailings. Coarser sand material accumulated at the point of pulp discharge and finer sand dispersing along the slope to the deeper parts of the pond. As a result the sand fractions prevail (over 50% of total volume) in the upper part of the tailings while the lower part contains mainly ultra-fine particles. Such occurrence is also likely to be a result of the function and efficiency of the plant; initially the average grinding coarseness was 0.11mm while later it was increased to 0.18-0.21mm due to the increased recovery of valuable ore components. Farther from the pulp discharge point there is a lower sand component content.

The percentage of coarse sand is quite insignificant (about 2.0%) while fine and medium sands are more common (48.0%). Fine fragmented fractions – silt, clay and ultra-fine sand fractions prevail (over 50.0%). According to "VNIItsvetmet" Research Institute data, the tailings are formed by finely ground material (48.0-60.0% of -0.074µm) with great amount of slimes.

The granulometric composition of the tailings was determined on samples from 6 drillholes on Profile 1 and 5 drillholes on Profile 5. As seen in the chart of percentage grain size distribution in Figure 12.3, silt size particles (shown in two shades of blue 45.1% and 18.2% respectively) predominate. The balance (36.7%) are sand grain size particles (shades of brown and yellow) ranging from very fine sand to very coarse sand. The overall proportion of coarse to very coarse sand is 3.1%.

Durnev and Golubtsov (2001) also noted that the overall grain size decreases southwards from the initial northern dam and eastwards from the western dam. This is consistent with the sequence of tails disposal, which began at the initial northern dam, and also with the quality of grinding. Earlier deposited tailings were ground to an average size of 0.11-0.21mm. The grinding quality improved in the 1970's and reached 48-60% of 200 mesh (0.074mm). The overall conclusion is that the fine fraction (silt, clay and very fine sand) predominate; particularly in the central, eastern and southern parts of the impoundment. The coarse sand fraction forms only about 2% of the total volume and the medium and fine sand fractions form 48% of the volume.

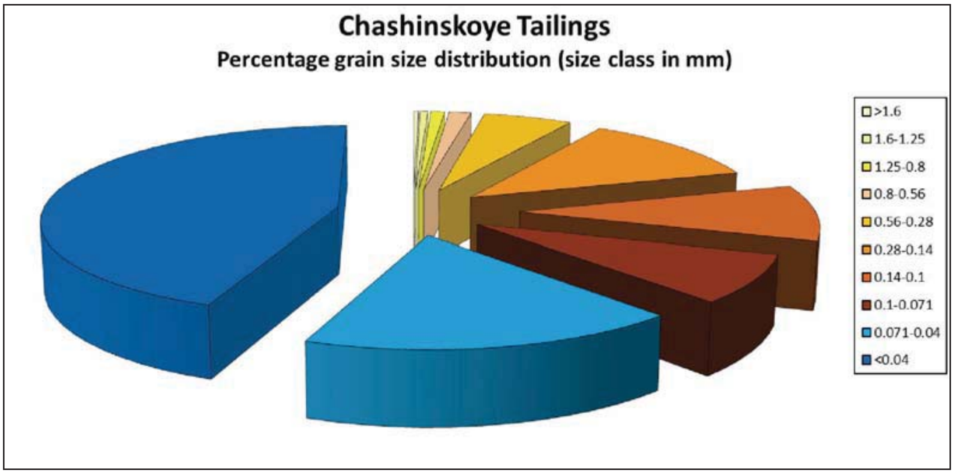


Figure 12.3: Percentage Grain Size Distribution Chashinskoye Tailings

12.3 Exploration Works

12.3.1 Topographic Base and Surveying Data

The topographic base map, which was used in the conventional resource estimates of 2000 and in a Datamine model that was prepared by Kazzinc, was compiled from data generated in two topographic surveys, both on a 1:1,000 scale. The earlier survey of 1960 covered the northern and eastern part of the area while the more recent survey, completed in 1988, covered the western part of the area. Archive material, including a 1:2,000 topographic map, thought to date from 1939, and engineering plans from 1974 were also utilised. The map compiled in this way has a scale of 1:2,000 and shows 5.0m contour lines for the base of the tailings impoundment.

The current configuration of the tailings top surface is derived from a 1:5,000 scale map prepared by the survey department of Kazzinc in April 2009. The map shows spot heights at various points, and the outline of three shallow pits in the western and central parts of the impoundment left after the extraction of material for backfill and construction, and locally contour lines at 1m intervals.

The local survey grid, which was used during topographic surveys and for locating drillhole sites, is linked to 2nd and 3rd class triangulation points. The relative point location error is reported to be 0.12m.

Locations of drillhole collars were surveyed by theodolite traversing from survey grid points or by resection from a minimum of three points with known co-ordinates. Collar elevations were measured by trigonometric levelling from at least two reference points. The mean square positional error was reported as less than 0.9m in plan and 0.2m in elevation.

12.3.2 Drilling

The Chashinskoye tailings dam has been evaluated during two stages of drilling. Stage 1 involved 25 holes being completed for 846m by Leninogorsk Mining and Processing Combinat (Leninogorsk GOK) from November 1991 to January 1992. From May to June 1993 a further 36 holes for a total of 873.9m completed by Goldbelt Resources Limited (Canadian company).

12.3.2.1 Historical Drilling

Drillholes were sited on nine profiles set up on an azimuth of 015° with spacing between profiles of 300m in Stage 1, reduced to 150m for Stage 2. The drillholes completed during Stage 1 were sited at 75-280m intervals along profile using a cable tool rig (UGB-ZUK) with an initial diameter of 377mm and reduced diameters of 273mm and 219mm as depth increased. Some of those holes were stopped above the base of the tailings.

For Stage 2, drillholes were sited at intervals of 75-125m along profiles using a Canadian vibrating rig ('vibrocore') with a hole diameter of 59mm. All drillholes completed during this stage reached the base of the tailings.

The resultant overall drilling grid varied from 150x100m in the western and northern parts of the impoundment to 300m by 100-200m in the area adjoining the settlement pond. All drillholes were vertical and there is no drilling over the area covered by the settlement pond in the southeast.

Table 12.6: Chashinskoye Tailings Evaluation Drilling

Year	Drilling method/Drill rig	No of Drillholes	Total Depth	Spacing	Main Purpose
1997-92	Cable tool/UGB-ZUK	25	846.0	300	Qualitative and quantitative characterisation
1993	Vibrocore	36	873.9	≈150	Confirmatory drilling
Total		**61**	**1,719.9**		

> ***WAI Comment:*** *WAI has not observed any of the drilling and sampling process followed but has reviewed the protocols applied, and in keeping with typical Soviet standards, considers the methods to be industry standard and appropriate for use in a mineral resource and ore reserve estimate in accordance with the guidelines of the JORC Code (2004). In addition this drill data was used for a GKZ approved resource estimation in 2000 that offers additional assurance to the standards applied and protocol followed.*

12.3.3 Sample Collection

Most drillholes were sampled over intervals of 1.0m though sample recoveries are not mentioned in the available documentation.

Samples from Stage 1 were collected in metal trays, mixed, quartered and reduced to 50kg weight. The next step involved reduction to about 20kg in a Jones riffle. The following samples were obtained by further splitting in the Jones riffle:

- Samples for fire assay (approximately 600g each);
- Duplicates for external control analyses (approximately 600g each);
- Samples for compositing into a 12kg technological sample;
- Control sample (150g);
- Samples for sieve analysis (2.5-3.0kg each) from Profile 5;
- Samples for process testwork (0.5kg); and
- Samples for determination of gold, silver, copper, lead and zinc in various grain size classes (taken from every other hole).

Samples collected during Stage 2, each weighing about 3kg per metre, were subdivided to obtain the following reduced samples:

- Samples for fire assay and chemical analysis (approximately 600g each);
- Duplicates (approximately 600g each); and
- Samples for sieve analysis from Profile 1 (1.8kg each).

12.3.4 Sample Analysis

The results of standard fire assay showed that the gold content in Chashinskoye tailing dam varies from 0.3-0.4 and up to 0.8-1.0g/t Au. Of 1,754 samples only 6 samples contained <0.3g/t Au and 107 samples >1.2g/t Au.

The maximum values of 2.0-3.3g/t Au were found in 11 samples. This analysis suggests a homogenous distribution of grade classes.

All analyses were performed at the central chemical laboratory of Leninogorsk GOK. They included fire assays for gold and silver, chemical analyses for copper, lead and zinc and granulometric analyses.

The quality assurance/quality control (QA/QC) programme relied on internal and external control analyses of analytical duplicates. External control analyses were conducted on analytical duplicates of 24% of routine samples by "Voskazgeologia" in Ust-Kamenogorsk.

Results of control analyses are available only in summary tables arranged by grade classes. Average relative differences for paired internal control analyses for gold and silver are within a range of 0-2%, which suggests a high precision of fire assays at Leninogorsk GOK. A summary of external control analyses shows similar results with the exception of gold analyses performed in 1992. There, results of external analyses were on average higher than the corresponding results of routine fire assays (28.86% for the 0.25-0.5g/t class, 11.31% for the 0.5-1.0g/t class and 6.21% for the 1.0-4.0g/t class). According to the ex-Soviet NSAM instruction, which remains in use in Kazakhstan, these results suggest that Leninogorsk GOK have understated gold in 1992. However, without results on certified reference material or arbitrary analyses by an independent laboratory, it is impossible to judge which laboratory was correct. There are no sample preparation rejects or analytical duplicates left in storage that could be submitted for check analysis.

The QA/QC summary data for copper, lead and zinc, which were presented in the 2001 report of Durnev and Golubtsov, are not specific to samples from the Chashinskoye drilling programmes. According to Durnev, they refer to the overall internal control results of the Leninogorsk GOKs laboratory in 1991-1993. The grade classes shown in the summary are too high for the grades encountered at the Chashinskoye tailings. It is therefore impossible to assess the reliability of analytical results for copper, lead and zinc. However, as these elements occur at very low concentrations, resulting uncertainties will have no material impact on resource estimations.

12.3.5 Bulk Density, Moisture Content and Porosity

Due to a highly variable composition of the Chashinskoye tailings, determinations of dry bulk density proved to be a challenging task. Durnev and Golubtsov (2001) quote results of particle density and porosity determination conducted in 1964, 1970-1971 and 1991. The results of those determinations were erratic and inconclusive. Consequently Durnev and Golubtsov resolved to select a dry bulk density of 1.6t/m^3 based on a porosity of 48% indicated by measurements reported in 1964. The density of 1.6t/m^3 was used in the resource estimate of 2000.

Specific weight (γ - g/cm^3) and gravimetric (natural) moisture content (WB %) were determined from 17 samples taken from the deposits in early December 1991. In addition to specific weight, WB determination of porosity (n %) and dry bulk weight, it was required to calculate bulk weight (δ - g/cm^3).

Analysis of granulometric parameters shows that the deposit, in section, is generally heterogeneous and the deposit cannot be classified by granulometric composition. Therefore using of "δ" value (given in reference literature specifying "n" values) for calculations may lead to substantial errors.

The "KazGIIZ" studies (1964 and 1970-1971), using samples taken from 2-12m depth, estimated that the "n" value varied from 45.6%-51.05%, averaging to 48.3%.

Assuming that tailing deposits are mainly formed by silt-sand of medium density (density coefficient e = 0.7), then:

$$n = e / 1 + e * 100 = 41\%$$

According to the "KazGIIZ" research data, the dry density (δск) is slightly variable from 1.34-1.47g/cm^3, averaging to 1.40g/cm^3. Reference literature values vary from 1.30 to 1.85g/cm^3.

At different "n" values, calculated values according to the formula: δск = γ (1 – n) are as follows:

n = 48.3 % – δск = 1.42 ÷ 1.46 g/cm^3, average – 1.44g/cm^3;
n = 41.0 % – δск = 1.62 ÷ 1.67 g/cm^3, average – 1.64g/cm^3.

According to the "KazGIIZ" data in 1964 the bulk weight of the deposits (δск) averaged 1.77g/cm^3; and in 1970-1971 – 1.51g/cm^3.

Calculated values determined by the formula δ = δск (1+ WB), and based on the analysis of available data, surmises that the following conclusions may be made:

1. Probability value: n ≈ 48%, δск = 1.40-1.44 g/cm^3.
2. Above the water level, bulk weight varies from 1.42-1.64g/cm^3 average value being 1.52g/cm^3 (samples No.1, 9, 13, 14, 15, 16, 17), below the water level bulk weight varies from 1.71-1.91g/cm^3 averaging at 1.80g/cm^3 (samples № 3-8, 10, 12).

As the valuable component content was determined for dry raw material (tailings) to calculate tailings and metal 'reserves', the (dry) bulk weight must be used. At a porosity value of n = 41.0% the average calculated (dry) bulk weight was 1.64g/cm^3 which generally complies with the data of "KazGIIZ" Research Institute.

> ***WAI Comment:*** *WAI has reviewed the methods employed and protocols followed in acquiring the sample data (drilling and sampling) and testwork procedures; including assay, granulometry, bulk density, moisture content and porosity, and believes that the results obtained are appropriate for use in a mineral resource estimate in accordance with the guidelines of the JORC Code (2004). Furthermore this data has been accepted by GKZ in their approval of the resource/reserve estimate of 2000. It should also be noted that a dry bulk weight (specific gravity) of 1.6g/cm^3 has been applied to all resource/reserve estimates.*

12.4 Process Testwork

The results of the most recent (2003 to 2007) process testwork on the Chashinskoye tailings conducted at the mineral processing laboratory of the Zyryanovsk concentrator were reviewed and reported by Kostorev (2007). The same report mentions earlier testwork results and these results are also summarised in Table 12.8.

The sample (150kg) used for the 2007 testwork at the mineral processing laboratory of the Zyryanovsk concentrator contained 0.82g/t Au. In particle size terms, it contained approximately 75% of very fine to medium sand and 17.85% of minus 200 mesh size particles. The bulk of the gold (73.5%) was contained in the sand size particle.

By comparison, the average grade of all drillhole intercepts at the Chashinskoye tailings is 0.7g/t Au and the average content of sand size particles is approximately 35-40%. Therefore the sample tested represents only the sand fraction of the tailings material which cannot be selectively extracted (as it is interlayered with silt size material) but could be separated by screening or gravity concentration prior to processing.

Table 12.7: Specific Weight, Bulk Weight, (Dry) Bulk Weight, Porosity and Gravimetric Moisture of Chashinskoye Tailings (based on samples taken on 4-9.12.1991 at Chashinskoye tailing dam)

№	Sampling Point	Sampling depth (m)	Position Relative to Waters Level	Parameter Values		Calculated Parameter Values				
						(Dry) Bulk Weight δ ск., g/cm^3		Deposits Bulk Weight, δ, g/cm^3		
						at porosity value (n), %		"KazGIIZ" data	Calculated at porosity value (n), %	
				Specific Weight (g/cm^3)	Gravimetric Moisture Content (%)	48.3	41.0	$\delta_{ск}$ = 1.40 г/см^3	48.3	41.0
1	Borehole 1/17	0.0 – 1.0	above, but increased water saturation during drilling	2.76	14.8	2.43	1.63	1.61	1.64	1.87
2		10.0 – 11.0		2.76	27.1	1.43	1.63	1.87	1.81	2.07
3		30.0 – 31.0		2.80	23.7	1.45	1.65	1.73	1.79	2.04
4		40,0 – 41.0	below	2.80	27.9	1,45	1.65	1.79	1.85	2.11
5	Borehole 2/19	10.0 – 11.0	below	2.78	21.5	1.44	1.64	1.70	1.75	1.99
6		20.0 - 21,0	below	2.81	29.3	1.45	1.66	1.81	1.87	2.15
7		30.0 – 31.0	below	2.77	22.0	1.43	1.63	1.71	1.74	1.99
8		38.0 – 39.0	below	2.77	30.4	1.43	1.63	1.83	1.86	2.13
9	Borehole 3/21	0.0 – 1.0	above	2.81	9.1	1.45	1.66	1.53	1.58	1.81
10		10.0 - 11.0	below	2.82	30.5	1.46	1.66	1.83	1.91	2.17
11	Borehole 11/	0.0 - 1.0	above, though water saturated	2.77	20.5	1.43	1.63	1.69	1.72	1.96
12		5.0 - 6.0	below	2.77	26.5	1.43	1.63	1.77	1.81	2.06
13	Outcroppings	0.2	above	2.82	2.0	1.46	1.66	1.43	1.49	1.69
14	Northern wall of pit No.3	0.3	above	2.75	3.6	1.42	1.62	1.45	1.47	1.68
15	Deposits intake for filling works	1.0	above	2.77	1.4	1.43	1.63	1.42	1.45	1,65
16		2.0	above	2.83	1.9	1.46	1.67	1.43	1.49	1,70
17		2.0	above	2.75	15.0	1.42	1.62	1.61	1.63	1,86
Average		-	-	**2.78**	**--**	**1.44**	**1.64**	-	-	-

The testwork carried out on this sample focused on gravity and flotation. Overall concentrate yields, grades and recoveries are summarised in Table 12.8.

Flotation tests were carried on the tailings feed (without any pre-concentration) and on gravity tails. These tests were performed on feed material comprising various proportions of -0.074mm and -0.020mm size fractions. The tests demonstrated that flotation recoveries tend to deteriorate as the proportion of the -0.020mm increases due to fine silica forming screens around metallic minerals.

The best results on the tailings feed were obtained by close circuit flotation on the feed grind 85% to -0.074mm using a 60:40 mixture of Butyl Xanthate and Butyl Aeroflot, or alternatively of Max Gold S101104 collector and X-133 frothing agent. The concentrate grade was 15.3g/t Au equating to the gold recovery of 65.5% and concentration degree 3.6 (see Table 12.8).

The best results on the gravity tails were obtained by close circuit flotation after additional grinding to 97% -0.074mm. The concentrate yield was 1.74%, concentrate grade was 19.0g/t Au equating to 58.4% gold recovery and concentration degree 33. The use of Max Gold S101104 combined with Aeroflot improved the gold recovery to 62.5% but the concentrate yield increased to 2.24% and concentrate grade was 16g/t Au.

Results of previous testwork are also shown in Table 12.8. It should be noted that direct cyanidation of tailings recovered 62.5% of gold, however no report on those test are available and were therefore not reviewed by Kostorev (2007).

Table 12.8: Summary of Results of Process Testwork on Chashinskoye Tailings

Testing Laboratory	Year	Processing Method	Product	Feed Grade	Yield	Product Grade	Gold Recovery
				g/t Au	%	g/t Au	%
Kazmekhanobr	2003	Direct cyanidation	Doré	0.62	0.23	16.90	62.50
		Gravity concentration followed by cyanidation of gravity tails	Doré	0.62	0.07	50.43	57.77
Ridder Concentrator	2005	Gravity-flotation	Gravity concentrate	0.62	1.08	10.50	17.45
			Flotation concentrate	0.62	1.05	21.00	33.92
TOMS	2007	Gravity-flotation	Gravity concentrate	0.56	0.18	43.60	14.14
			Flotation concentrate	0.56	2.27	8.00	32.42
Zyryanovsk Concentrator	2007	Close circuit flotation	Flotation concentrate	0.82	3.67	15.30	70.00
		Gravity concentration in Mozley	Gravity concentrate	0.82	72.46	0.96	84.08
		Gravity concentration in Knelson separator	Gravity concentrate	0.82	8.40	3.40	34.96
		Flotation of Knelson gravity tails	Flotation concentrate	0.82	1.74	19.00	58.35

12.5 Mineral Resource and Ore Reserve Estimation

12.5.1 Introduction

The mineral resource of Chashinskoye tailings dam has been investigated following each stage of investigation. In 1994 Leninogorsk MCC estimated the deposit based on previous studies and drilling works per-formed by

"Kilborn" company. The most recent, and considered current, estimation was completed using all available data as of 01.01.2000.

12.5.2 Kazzinc Reserve Estimation

The current (2000) estimate was based on the results of exploration works carried out by Leninogorsk Exploration Company (drilling performed in November 1991-January 1992) and Goldbelt Resources Limited (check drilling in May-July 1993) with co-operation of Leninogorsk Polymetallic Combinate (LPC) and Comptoir International du Commerce C.A. (Comptair) from Luxembourg.

The works were commenced based on the Letter of Intent dated December 15, 1990 and concluded between Australian Mineral Resources (AMR), acting as the agent of Comptoir and the State Committee on Foreign Economic Relations of Kazakh SSR. The agreement stating the main terms and conditions of the Joint Venture Agreement was signed on June 27, 1991 and on December 15, 1991 the Parties signed the Letter – protocol on agreement to final contract award. The Agreement between LPC and Comptoir on formation of a Joint Venture ("Kazgold") for processing of gold bearing concentration tailings was concluded on February 19, 1992 and registered by the Ministry of Finance of the Republic of Kazakhstan (registration No. 290).

According to the Protocol of intent, on formation of the Joint Venture for processing the Concentrator's tailings dated October 15, 1991, and signed by LPC and AMR, drilling works at Chashinskoye tailing dam were performed in November 1991 – January 1992. Drilling works were aimed to qualitative and quantitative characterise the tailings dam material and tenor of gold bearing mineralisation. Leninogorsk Exploration Company drilled 25 exploration boreholes along five profiles located 300m away from each other for a total of 846m.

It should be noted that initial program of drilling and sampling works, suggested in the Protocol of intent by AMR, was completely changed and developed by the Geological department of Leninogorsk MCC and then completed under the supervision of Mr G.S. Durnev, Chief Geologist, in the presence and under control of the AMR represented by Mr. J. Searle. The drillhole pattern was completely changed, the scope of works was performed in full, and the sampling interval was reduced to 1m (instead of 4m).

As a result of these works a 'reserve' was calculated as being 82,765.99kt containing 48,081kg of gold at an average grade of 0.58g/t. In addition grain-size classification of the tailings was studied (5 boreholes in one profile) with classification of valuable components by classes; samples were taken for metallurgical analysis; chemical composition of the waters of Chashinskoye, Talovskoye tailing ponds and Ridder-Sokolniy mine water treatment facilities was studied; specific weight (density) and gravimetric (natural) moisture content of the Chashinskoye tailing dam deposit were investigated.

In February 1992 LPC prepared and approved the report based on the results of performed works.

In January 1993, the National Kazakhstan Agency for foreign investments addressed the European Bank for Reconstruction and Development (EBRD) with a proposal to carry out an independent examination of the previous exploration works. In May-June 1993 confirmatory drilling was performed according to a drillhole pattern at half the distance of previous investigation, and as required by independent examination.

Results obtained confirmed the information obtained during the work completed in 1991-1992. In addition to sampling and chemical-analytical study, grain size composition of tailings material in one profile, parallel to the earth-fill dam, was studied. All the works were carried out by LPC and the drilling was contracted to a Canadian company (Goldbelt Resources Limited).

Variation coefficient for gold and silver content determined from 1,687 standard samples analyses is 56.2% and 47.0% respectively. These values are compared with coefficient variations for the deposits of Rudny Altay referred to complexity Class III. According to the classification of the State Reserves Committee of the Republic of Kazakhstan, the Chashinskoye tailing dam can be referred to Class III of geological structure complexity.

12.5.3 Historical Mineral Resource Estimates

The first resource estimate was prepared in 1993 and is summarised in Table 12.9 below. However, the Leninogorsk Mining and Concentrating Complex (MCC) removed 401,391t from the Chashinskoye tailings dam in 1996-1998.

Table 12.9: Chashinskoye Tailings Dam Resource Estimate (01.09.1993)					
Tonnes (kt)	Metal (grade/metal reserves)				
	Au g/t (kg)	Ag g/t (kg)	Cu % (t)	Pb % (t)	Zn % (t)
87,373.8	0.62 (53,834)	5.01 (437,901)	0.04 (37,216)	0.13 (114,315)	0.36 (317,238)

As a result of this removal of material, a revised resource estimate was completed which is summarised in Table 12.10 below.

Table 12.10: Chashinskoye Tailings Dam Resource Estimate (01.01.2000)					
Tonnes (kt)	Metal (grade/metal reserves)				
	Au g/t (kg)	Ag g/t (kg)	Cu % (t)	Pb % (t)	Zn % (t)
86,972.4	0.62 (53,575)	5.01 (435,819)	0.04 (36,528)	0.13 (113,064)	0.36 (315,710)

12.5.4 Approved Resource ("Balance Reserve")

Resource estimation has been undertaken in 1994 and in 1999, the former are now only of historical significance, the latter being performed by Durnev and Golubtsov and reported in their report dated 2001. Their estimate was reviewed and approved by the Government Commission for Mineral Resources of the Republic of Kazakhstan ("GKZ RK").

For purposes of tonnage and grade estimation, Durnev and Golubtsov divided the tailings area into four blocks as follows:

1. Block I drilled on the 100x150m grid.
2. Block II drilled on the 100x300m grid.
3. Block III covering the strip of ground along the margin of the settlement pond and tested by six drillholes along its north-western perimeter.
4. Block IV covering the settlement pond and devoid of any drilling.

Estimates for Blocks I and II were performed using the conventional cross sectional method. Blocks III and IV were also estimated by the same method with average grades for both blocks estimated from the six holes around the north-western perimeter of Block III.

The tailings were considered to be bulk mineable and therefore no cut-off grade or top cut was applied. Average grades for each drillhole intercept were estimated as arithmetic means of sample grades weighted by respective sample lengths. Each cross section was assigned the arithmetic mean of the drill intercepts located on and close to the section. These grades were then weighted by volumes assigned to each cross section. The areas were calculated in AutoCAD-14 and the dry bulk density of $1.6t/m^3$ was used to convert volume into tonnage.

In accordance with the ex-Soviet Classification of Reserves and Prognosticated Resources (which remains in use in Kazakhstan), the resource contained within Block I was classified as C_1 category and within Block II as C_2 category, and both as 'Balance Reserve'. Tonnages and grades estimated for Block III and Block IV were classified as Prognosticated Resources of P_1 and P_2 categories. The combined tonnage was estimated to be

88.1Mt and the average grades are 0.6g/t Au and 5.0g/t Ag. The tonnage is 2.8% higher than the tonnage of the deposited tailings but the gold and silver grades are lower by 11.5% and 15.7% respectively (Table 12.11).

Durvev and Golubtsov also estimated copper, lead and zinc grades and the average grades are 0.04%, 0.13% and 0.36% respectively. GKZ took a view that only gold and silver resources should be placed on the state balance and that copper, lead and zinc should be considered as by-products.

GKZ RK reviewed and approved those estimates in June 2001 (Protocol No 103-01-U dated 27 June 2001). In the same document, GKZ RK also confirmed its acceptance of the estimated Prognostic resource.

Table 12.11: Balance Reserves and Prognostic Resources for the Chashinskoye Tailings (01 January 2000)

Category	Tonnes (Mt)	Grade	
		g/t Au	g/t Ag
GKZ Approved 'Balance Reserve'			
C_1	33.278	0.76	5.97
C_2	20.001	0.58	4.49
GKZ RK Accepted Prognostic Resource			
P_1	22.147	0.49	4.40
P_2	12.739	0.49	4.40

The reserves quoted in the table above have not been prepared in accordance with the guidelines of the JORC Code (2004). These reserves have been classified under the standards of the State Commission on Mineral Reserves (Gosudarstvennaya Komissia po Zapasam) of the Republic of Kazakhstan (GKZ (RK)) which were adopted from GKZ of the Russian Federation.

The JORC Code (2004) differs in certain respects from GKZ (RK), but WAI considers that the standards are broadly comparable. The level of detail of information required to support a submission of mineral "resources and reserves" to GKZ (RK) is both systematic and comprehensive and classification under GKZ (RK) is subject to rigorous review comparable with the guidelines of the JORC Code.

WAI have completed a Mineral Resource and Reserve estimate of the Chashinskoye tailings deposit which is in accordance with the guidelines of the JORC Code (2004), and is presented in Section 12.6 below.

12.6 WAI Mineral Resource and Ore Reserve Estimation

12.6.1 Introduction

WAI have completed a Mineral Resource and Ore Reserve estimate of the Chashinskoye tailings deposit in accordance with the guidelines of the JORC Code (2004) and based on data provided by the Client including a drillhole database and topography plans.

12.6.2 Topography

The wireframe envelope of the TMF deposit was supplied by KMC. The current surface topography is derived from a 1:5,000 scale map prepared by the survey department of Kazzinc in April 2009. Some parts of the TMF are up to 60m deep.

12.6.3 Database Compilation

All of the sample data collated for the current project are summarised below in Table 12.12.

Table 12.12: Sample Data Summary				
	No. of Holes	No. of Samples	Hole Lengths (m)	
			Total	Average/Hole
Drillholes	89	2,243	2,393	27

A general plan of these drillholes are shown in Figure 12.4. As can be seen from the plan, the drillholes have generally been laid out approximately on a 150m spaced sections, with spacings of between 80–120m along each section line. All drillholes are vertical and no down hole survey conducted. The area of the tailings dam towards the southeast is furthest away from the original discharge points and thus has the lowest elevation. Consequently this part of the dam is covered in water to a depth of around 5m and as a result has not been subject to drilling.

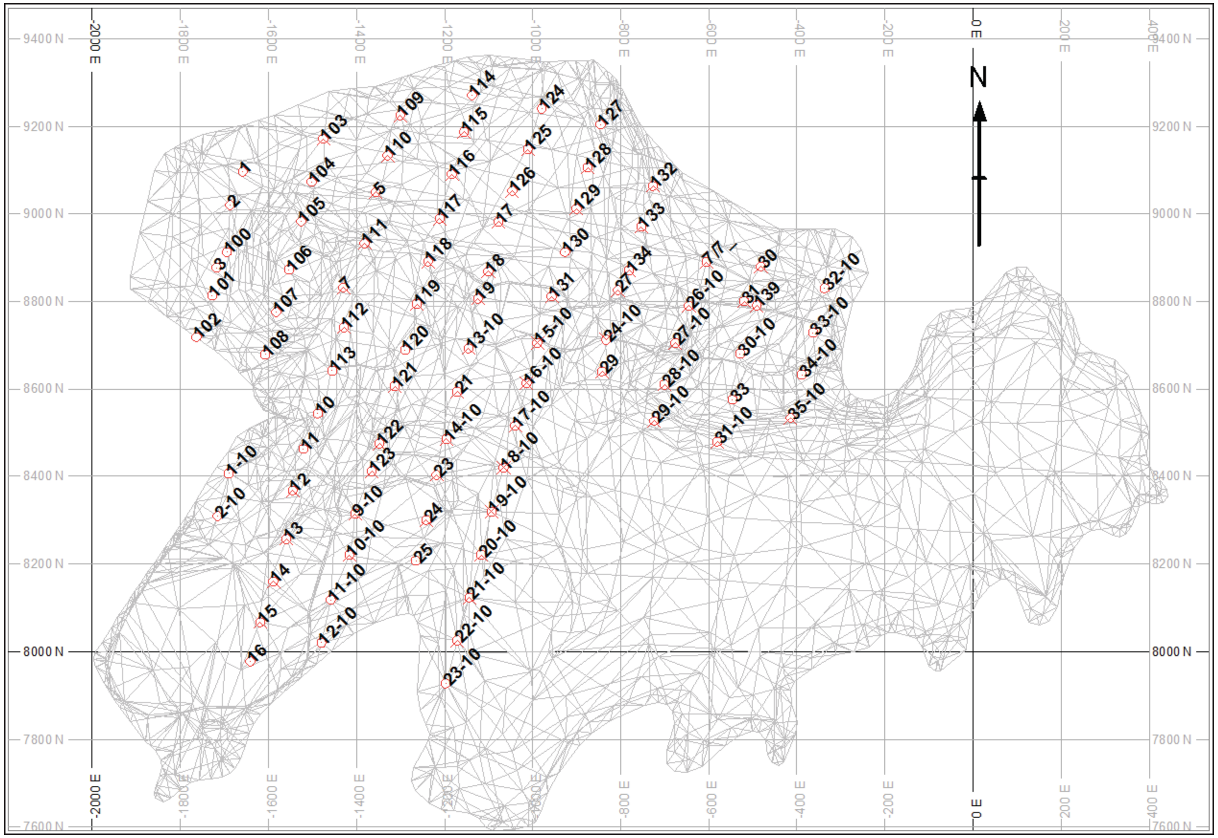


Figure 12.4: Plan View of Drillholes and WAI Resource Envelope

12.6.4 Sample Data Processing

The supplied wireframe model was used to select and allocate all of the drillhole data.

Table 12.13: Selected Sample Summary				
	No. of Holes	No. of Samples	Hole Lengths (m)	
			Total	Average/Hole
Drillholes	89	2,126	2,278	26

Most of the 117 rejected samples are from parts of the tailings dam that have already been removed for re-processing.

All of the drillhole samples have been assayed for Au, Ag, Cu, Pb and Zn. Most of these indicate principally a single log-normal distribution. A statistical summary of the selected samples is shown in Table 12.14. Most samples were 1m in length.

Table 12.14: Statistical Summary of Selected Samples								
Field	Number	Minimum	Maximum	Mean	Variance	Standard Deviation	Log Estimate of Mean	Coefficient of Variation
AU	2,126	0.2	4.1	0.65	0.09	0.30	0.65	0.46
AG	2,126	0.3	45.0	5.08	5.55	2.36	5.07	0.46
CU	2,126	0.0	0.42	0.05	0.00	0.02	0.05	0.54
PB	2,126	0.01	17.0	0.15	0.06	0.25	0.14	1.66
ZN	2,126	0.02	2.08	0.38	0.02	0.16	0.38	0.41

A decile analysis was also completed on the selected gold grades, as shown in Table 12.15. From this and the log-probability plot, there does not appear to be a significant problem with outlier grades, so no top-cuts were applied.

Table 12.15: Decile Analysis of Au Grades							
Q%_FROM	Q%_TO	NSAMPLES	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	212	0.30	0.2	0.4	63.0	4.4
10	20	213	0.40	0.4	0.4	85.2	6.0
20	30	212	0.45	0.4	0.5	94.8	6.7
30	40	213	0.51	0.5	0.6	108.1	7.6
40	50	213	0.60	0.6	0.6	127.8	9.0
50	60	212	0.63	0.6	0.7	134.0	9.4
60	70	213	0.73	0.7	0.8	154.7	10.9
70	80	212	0.82	0.8	0.9	173.3	12.2
80	90	213	0.98	0.9	1.0	208.2	14.7
90	100	213	1.26	1.0	4.1	269.3	19.0
90	91	21	1.00	1.0	1.0	21.0	1.5
91	92	21	1.00	1.0	1.0	21.0	1.5
92	93	21	1.08	1.0	1.10	22.6	1.6
93	94	22	1.10	1.1	1.1	24.2	1.7
94	95	21	1.18	1.1	1.2	24.7	1.7
95	96	21	1.20	1.2	1.2	25.2	1.8
96	97	22	1.23	1.2	1.3	27.1	1.9
97	98	21	1.31	1.3	1.4	27.5	1.9
98	99	21	1.42	1.4	1.5	29.9	2.1
99	100	22	2.10	1.5	4.1	46.1	3.3
0	**100**	**2,126**	**0.67**	**0.2**	**4.1**	**1,418.33**	**100**

The data was then converted into 5m composites. A statistical summary of the composites is shown in Table 12.16.

Table 12.16: Statistical Summary of Composites								
FIELD	Number	Minimum	Maximum	Mean	Variance	Standard Deviation	Log Estimate of Mean	Coefficient of Variation
AU	462	0.23	1.42	0.65	0.04	0.20	0.65	0.31
AG	462	2.37	13.10	5.08	3.55	1.88	5.07	0.37
CU	462	0.02	0.13	0.05	0.00	0.02	0.05	0.37
PB	462	0.07	1.70	0.15	0.01	0.10	0.15	0.70
ZN	462	0.17	0.93	0.38	0.01	0.12	0.38	0.32

12.6.4.1 Variography

Variography was undertaken to test for any spatial continuity of the metal grades, and to assist with the selection of suitable search parameters upon which to base the resource estimation.

Variography was carried out on the 5m composite data contained within the modelled zones. Absolute, as well as relative variograms were generated for AU grades, with the spherical scheme model being used for modelling purposes. Experimental and fitted model variograms, horizontally and vertically, are shown in Appendix 1. These indicated ranges of approximately 300m vertically and 30m vertically. In the horizontal direction no particular anisotropies were apparent.

12.6.4.2 Block Modelling

A model prototype was set up with a 50x50x10m parent block size. The details of this prototype are shown in Table 12.17.

Table 12.17: Block Model Prototype					
	Origin	Maximum	Distance	Block Size	Number
X	-2,200	500	2,700	50	54
Y	7,500	9,500	2,000	50	40
Z	750	850	100	10	10

This prototype was then used as the basis to set up a volumetric model, as controlled by the supplied TMF envelop. Sub-blocks were generated within the structure near the edges, down to minimum dimensions of 10x10x1m.

12.6.4.3 Density

An overall average density value of 1.60t/m^3 was used, as had been determined from previous Kazzinc measurements.

12.6.4.4 Grade Estimation

Grade estimation was carried out using Inverse Power of Distance Squared (IDW^2) for all metal grades, although Ordinary Kriging (OK) and Nearest Neighbour (NN) were also used for comparative purposes on the AU grade.

The estimation was run in a three pass plan, the second and third passes using progressively larger search radii to enable the estimation of blocks un-estimated on the previous pass. Grades were estimated using the 5m composite drillhole data. The search parameters were derived from the variographic analysis, with the first search distances corresponding to the distance at 2/3^{rds} of the variogram sill value and the second search distance approximating up to the variogram range. Block discretisation was set to 5x5x2 to estimate block grades. Sub-cells received the same estimate as the parent cell. A summary of the estimation parameters is shown in Table 12.18.

Table 12.18: Estimation Parameters					
Search Distances (m)				No. Of Composites	
1st Search		2nd Search			
Horizontal	Vertical	Horizontal	Vertical	Min	Max
150	15	300	30	5	15

Note:

- 3rd search >1,000m, with a minimum of only 1 composite
- ID^2 used for all Au, Ag, Cu, Pb, Zn grades

12.6.4.5 Resource Classification

The resource classification for the Chashinskoye tailings dam is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves JORC Code (2004).

The following criteria were established for the classification of resources:

- ***Measured:*** No material was in this category, owing chiefly to lack of closely spaced drillholes and sample data;
- ***Indicated:*** Blocks within this category had to be covered by at least 150x150m spaced drillhole data. These blocks were assigned using perimeters defined on each 10m bench of the TMF; and
- ***Inferred:*** All other blocks within the TMF, but outside of 150x150m drilled areas, were assigned as inferred.

An example horizontal model section through the block model, showing this resource classification, is shown in Figure 12.5.

12.6.4.6 Validation

Following grade estimation a statistical and visual assessment of the block model was undertaken to assess successful application of the estimation passes and to ensure that as far as the data allowed, all blocks within the defined domain were estimated. The model validation methods carried out included:

- A visual assessment of grade;
- Global statistical grade validation;
- Model grade profile (swath plot) analysis; and
- Comparison with historical estimates.

A visual comparison of composite sample grade and block grade was conducted in cross section and in plan. An example section through the block model, with the estimated AU grades, is shown in Figure 12.6. Visually the model was generally considered to spatially reflect the composite grades.

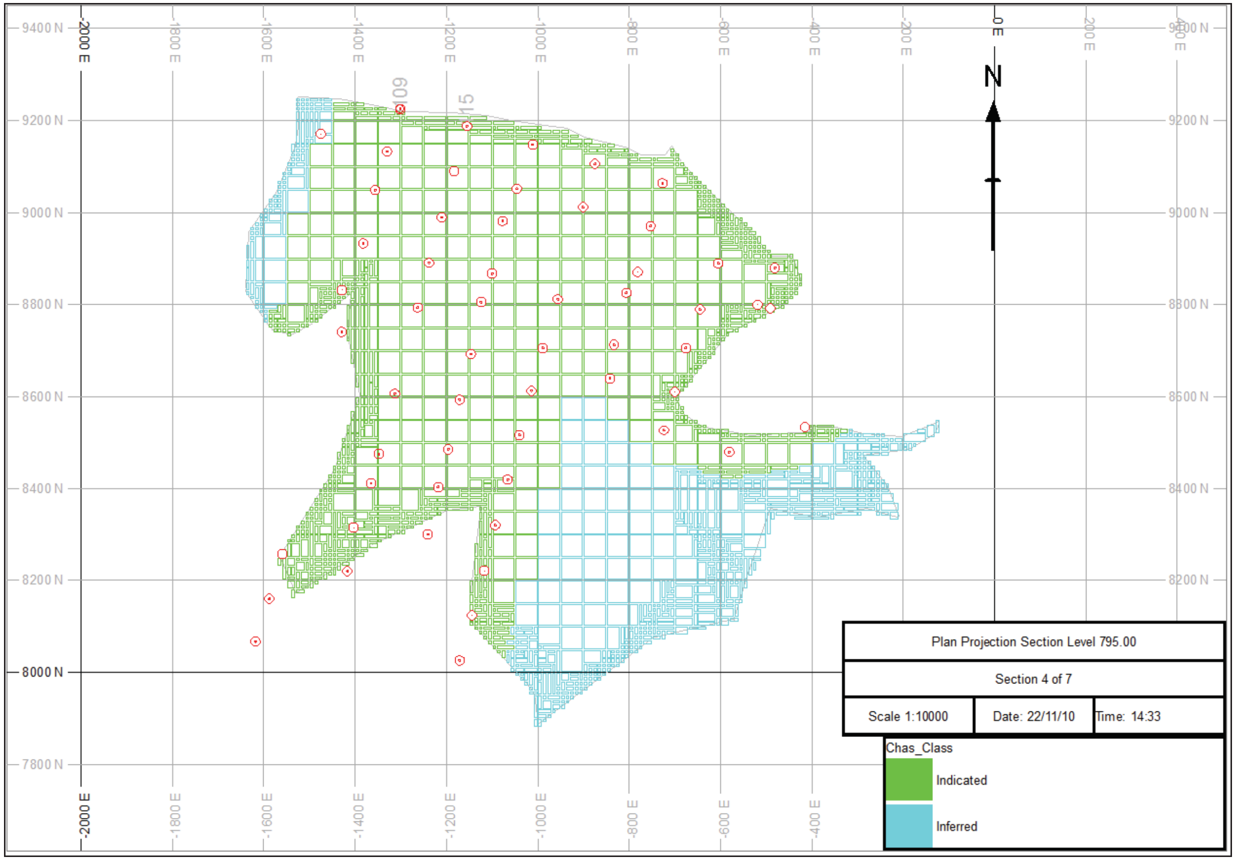


Figure 12.5: Bench Plan at 795mRL showing Resource Classification (Green = Indicated, Blue = Inferred)

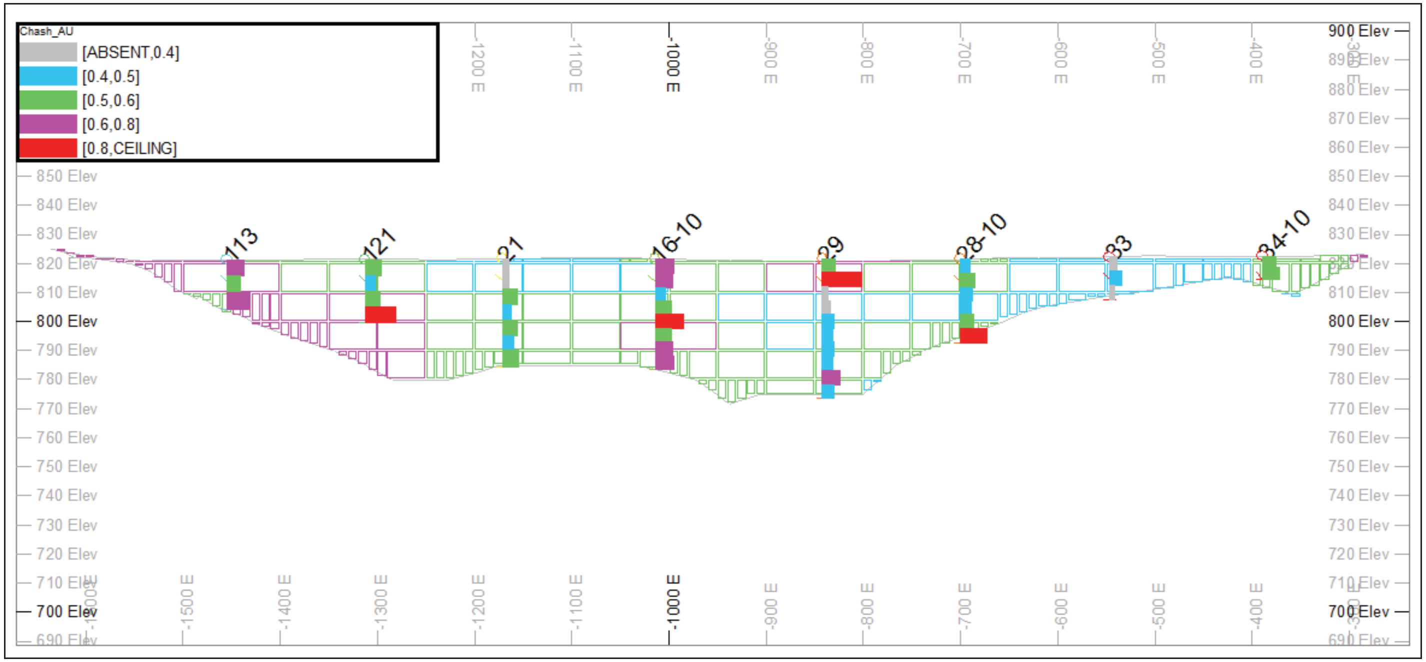


Figure 12.6: Vertical Section at 8,600mN (Vertical exaggeration), Showing Composites and Block Model Au Grades

A comparison of global average grades is summarised below in Table 12.19. As well as the principal inverse-distance grades in the block model, this table includes alternative estimates made using nearest neighbour and ordinary kriging. Overall these global average grades compare very well.

Table 12.19: Comparison of Global Average Grades

FIELD		Average Grades		Block Model Grades		
		Samples	Composites	ID	NN	OK
AU	g/t	0.65	0.65	0.67	0.68	0.67
AG	g/t	5.08	5.08	5.25		
CU	%	0.05	0.05	0.04		
PB	%	0.15	0.15	0.15		
ZN	%	0.38	0.38	0.38		

Note:

- Block model grades are for Indicated resources only
- No cut-off grades applied
- OK (Ordinary Kriging), NN (Nearest Neighbour), ID (Inverse Distance)

Model grade profiles (swath plots) were generated from the model by averaging both the drillhole composites and blocks along 100m spaced W-E slices. The model average grades were derived from indicated resources only. Overall the drillhole composite average grades and the block model average grades compare extremely well.

A comparison of this resource estimate is shown with reference to an earlier Kazzinc estimate from 2001 in Table 12.20. This shows a small increase in the estimated gold content (indicated versus C_1+C_2) of approximately 5%.

Table 12.20: Historical Comparison of Resource Estimates

	Kazzinc (2000)			WAI (01.01.2011)		
Classification	**C_1 + C_2**	**P_1**	**Total**	**Indicated**	**Inferred**	**Total**
Tonnes	53.3	22.1	**75.4**	57.8	30.0	**87.8**
Au g/t	0.69	0.49	**0.63**	0.67	0.50	**0.61**
Ag g/t	5.42	4.40	**5.12**	5.16	4.57	**4.96**
Cu %	0.04	0.05	**0.04**	0.05	0.06	**0.05**
Pb %	0.13	0.13	**0.13**	0.15	0.19	**0.16**
Zn %	0.35	0.38	**0.36**	0.38	0.45	**0.41**

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.

12.6.5 Resource and Reserve Evaluation

The resource classification for the Chashinskoye tailings deposit is classified in accordance with the guidelines of the JORC Code (2004).

The final block model of the tailings deposit was used as the basis for resource evaluation. Summary results of the evaluation of the resources are shown in Table 12.21.

Table 12.21: Chashinskoye Tailings Mineral Resource Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))							
Classification	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AuEq* (g/t)
Indicated	57.8	0.67	5.16	0.05	0.15	0.38	1.15
Inferred	30.0	0.50	4.57	0.06	0.19	0.45	1.06

* AuEq (Gold Equivalent) is based on the following commodity prices:

- Au = US$1,287/oz
- Ag = US$23/oz
- Cu = US$7,341/t
- Pb = US$2,422/t
- Zn = US$2,420/t

WAI has calculated Ore Reserves based upon the above Mineral Resource Estimate and in accordance with the guidelines of the JORC Code (2004). WAI has applied losses of 4.4% based upon historical data, and dilution of 0.5% to take into account minimal dilution from the bunds and floor.

Table 12.22: Chashinskoye Tailings Ore Reserve Estimate (WAI 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))											
Classification	Tonnes (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (koz)	Grade (g/t)	Metal Content (koz)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)
Proven	-	-	-	-	-	-	-	-	-	-	-
Probable	55.53	0.70	1,245	5.37	9,589	0.05	28.9	0.16	86.7	0.40	219.6
Total	**55.53**	**0.70**	**1,245**	**5.37**	**9,589**	**0.05**	**28.9**	**0.16**	**86.7**	**0.40**	**219.6**

13 TISHINSKIY TAILINGS DEPOSIT

13.1 Introduction

The Tishinskiy slimes ponds (tailings) are the accumulation of 'fines' deposited via a pipeline from the Ridder concentrator where, prior to August 2002, sulphide ore from the Tishinskiy deposit was processed. In August 2002 a filter press was commissioned at the RMCC which allowed dewatering of the slimes to occur and to be re-processed.

In the summer of 2003 an initial exploration programme was undertaken on slimes pond No.2 whereby mapping and 7 drillholes were completed in an effort to evaluate their re-processing potential as well as reduce their environmental impact. The ponds contain significant quantities of non-ferrous metals that have been exposed to oxidation through natural processes, including filtration into the underlying ground water and Ulba River.

13.1.1 Location & Access

The ponds are located some 500m south and southwest of the Tishinskiy complex, and are formed within the body of the waste rock dump derived from overburden stripped between 1965 and 1976 from the Tishinskiy deposit.

Pond No.1 is currently used as an emergency discharge facility for the Ridder complex and is flooded, however once this purpose ceases to be required it is intended to dewater and assess (complete a small drilling programme) as for Pond No.2.

13.1.2 Mineral Rights and Permitting

Kazzinc holds the right to mine under the terms of Contract No 2693 dated 19 June 2008 which expires on 19 May 2013. The Tishinskiy slimes collector's No. 1 and 2 covers an area of 30ha, the boundaries of which are defined by 8 corner points as detailed in Table 13.1 and shown in Figure 13.1.

Table 13.1: Tishninskiy Slimes Mining Lease Boundaries

Boundary Points	Geographical Co-ordinates	
	Latitude N	Longitude E
1	50°15'23"	83°21'28"
2	50°15'28"	83°21'29"
3	50°15'34"	83°21'28"
4	50°15'42"	83°21'16"
5	50°15'38"	83°21'46"
6	50°15'33"	83°21'53"
7	50°15'28"	83°21'52"
8	50°15'21"	83°21'38"

Figure 13.1: Outline of Tishinskiy Tailings Licence

13.2 Geology and Mineralisation

The Tishinskiy slimes ponds are a man-made feature and does not possess any recognised geological structure. The tailings are the waste products of the processing of gold and polymetallic ores (primarily silver, copper, lead and zinc) from the Ridder Mining and Concentrating Complex (MCC) and their composition thus reflects the major constituents of the ore.

13.3 Exploration Works

13.3.1 Drilling

13.3.1.1 Introduction

The Tishinskiy slimes ponds are relatively small structures, less than 500m in length and 200m wide, and so have few drillholes. Slimes Pond No.1 has had no exploratory investigation at it is flooded and still in use as an emergency discharge facility and therefore retains a high moisture content making it impossible to safely support a drill rig. It is intended to dewater this pond once its use as an emergency discharge facility has come to an end and engage in a drilling programme. Conversely Slimes Pond No.2 has been investigated by 2 drill campaigns in 2003 and 2008 using cable tool technique.

13.3.1.2 Historical Drilling

The initial drilling campaign in 2003 involved the completion of only 7 holes for a total of 123m using a cable percussion rig and 220mm diameter tools. The drill casing is driven in advance of the sampling tool (shell and bailer) to minimise cross-contamination and improve drilling and sampling results. In 2008 a further 12 holes were completed for an additional for 181m (Table 13.2).

All holes have been completed by LLP "Trias LTD" using cable percussion technique at a diameter of 220mm and are vertical.

Table 13.2: Drilling Campaigns at Tishinskiy Slimes Pond No.2			
Profile	Drillhole No	Depth (m)	
		Planned	Actual
7-7	AQS. 1	13	13
7-7	2 NCR.	23	23
7-7	AQS. 3	19	19
6-6	AQS. 4	21	21
6-6	AQS. 5	26	26
5-5	AQS. 6	12	12
2-2	AQS. 7	9	9
	Total 2003	**123**	**123**
6-6	7-p	14	14
5-5	8-p	15	15
5-5	9-p	15	15
3-3	10-p	16	16
4-4	11-p	16	16
3-3	12-p	16	16
2-2	13-p	16	16
2-2	14-P.	16	11
2-2	15-p	12	12
1-1	16-p	17	16
1-1	17-p	17	17
1-1	18-p	17	17
	Total 2008	**187**	**181**
	TOTAL	**310**	**304**

The interval between drillhole profiles is 80-110m, with hole spacing along profiles of approximately 40-50m, which complies with the requirements of Recommended practices of exploration and evaluation according to ETS GKZ at MG & ON RK dd. 2 June 1995 for evaluation of C_1 category reserves.

13.3.1.3 Sampling

To ensure qualitative and quantitative evaluation of the material found in the slime ponds samples were taken from each meter providing a total number of 295 samples (Table 13.3). There were no samples taken from the last 9m of hole 19-P since the hole passed through into the underlying material of the waste rock dump. All slime samples were processed in accordance with the scheme developed based on the existing grinding equipment, NSAM (analytical methods research council) requirements (Methods of geological control of analytical sample, VIMS, MG USSR, 1982). Samples are treated in accordance with the scheme developed at K=0.1, final grinding up to 0.07mm. All slime samples were analysed for copper, lead, zinc, gold and silver.

Combined samples (totalling 5) were taken at various levels of holes to analyse for trace elements and harmful impurities.

21 samples of 5kg were taken at various depth to analyse grain size composition and physical and mechanical properties of the slimes. Four bulk samples (1m^3 each) were taken to determine bulk weight and moisture content.

There were no samples taken for analysis of underground waters chemical composition since the holes were dry.

Table 13.3: Summary of Sampling from Slimes Pond No.2				
Type	**Unit**	**No.**	**Analysis**	
			Reduced	**Complete**
Slimes sample for chemical analysis	Sample	295	295	
Combined samples	Sample	5		5
Grain size, water-physical and physical and mechanical properties	Sample	21		21
Bulk samples for determination of bulk weight and moisture content	Sample	4		4

13.3.2 Laboratory Analysis

13.3.2.1 Introduction

Analytical studies were conducted in Kazzinc JSC RMCC chemical-analytical laboratory in compliance with the existing (Kazhak) requirements of quality assurance and accuracy of measurements.

Samples were processed (prepared) using a jaw crusher and disk pulveriser based on standard practices as per Chechett formula:

$$Q = kd^2$$

Where:

- Q = representative weight of sub-sample (kg);
- k = mineralisation discontinuity coefficient for Tishinsky ore equal to 0.1; and
- d = maximum diameter of sample chips (mm).

The initial ≈61.8kg was mixed and quartered until a sub-sample of 3.8kg which was subject to crushing (passing 2mm), splitting to 1.9kg and pulverised, and final splitting for a 1.3kg duplicate sample, 0.1kg sample for Cu, Pb and Zn analysis and a 0.5kg sample for Au and Ag analysis.

As well as standard internals controls the sample set also underwent external control at the "DGP Vniitsvetmet" laboratory for Au, Ag, Cu, Pb, and Zn, and in accordance with recommended State guidelines, with the results deemed acceptable.

13.3.2.2 Chemical and Analytical Study

Determination of grain size composition, water-physical and physicomechanical properties were carried out at the pilot production area of the concentrator. Samples were analysed for grain size composition; bulk weight, specific density, porosity, moisture content, moisture holding capacity as well as content of metal ions of copper, lead, zinc, iron in water extract were also determined; and chemical analysis for 15 elements was undertaken.

The granulometric composition of the slimes is summarised in Table 13.4 below, and as can be seen the slimes possess a very fine grain size with over 70% of the slimes -0.011mm but with grade being more evenly distributed throughout.

Table 13.4: Granulometric Composition of the Slimes									
Class Size (mm)	**Removed**	**Grade (%)**				**Extraction (%)**			
	%	**Cu**	**Pb**	**Zn**	**Fe**	**Cu**	**Pb**	**Zn**	**Fe**
-0.56 +0.32	1.05	0.29	0.41	1.86	4.46	0.98	0.96	0.76	0.87
-0.32 +0.12	4.8	0.14	0.19	1.05	3.50	2.18	2.03	1.97	3.11
-0.12 +0.074	7.32	0.21	0.22	1.61	4.36	5.02	3.60	4.60	5.90
-0.074 +0.044	1.7	0.30	0.28	2.20	4.84	1.66	1.07	1.46	1.52
-0.044 +0.022	1.02	0.62	0.79	4.69	8.96	2.05	1.80	1.87	1.69
-0.022 +0.011	9.76	0.76	0.72	6.60	9.70	24.2	15.7	25.13	17.5
-0.011 +0.005	30.71	0.34	0.48	2.93	5.72	34.04	32.92	35.10	32.47
-0.005	43.64	0.21	0.43	1.71	4.58	29.87	41.92	29.11	36.94
Source	**100**	**0.31**	**0.45**	**2.56**	**5.41**	**100**	**100**	**100**	**100**

13.4 Topographic Base and Surveying Data

The local survey grid, which was used during topographic surveys and for locating drillhole sites, is linked to 2nd and 3rd class triangulation points. The relative point location error is reported to be 0.12m.

Locations of drillhole collars were surveyed by theodolite traversing from survey grid points or by resection from a minimum of three co-ordinated points. Collar elevations were measured by trigonometric levelling from not less than two co-ordinated points. The mean square positional error did not exceed 0.9m in plan and 0.2m in elevation.

13.5 Kazzinc Reserve Estimation

Kazzinc have produced an internal reserve estimate (01.11.2008) as summarised in Table 13.5 below.

Table 13.5: Kazzinc Reserve Estimate for Tishinskiy (01.11.2008)						
Class	**Tonnes (kt)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
C_1	491.4	0.37	9.52	0.23	0.71	2.49

However it should be noted that as the initial deposition (pre 01.06.1992) into the slimes dam was during State ownership there is an amount within the dam that is the property of the State. Resultantly the slimes belonging to Kazzinc JSC are 279,457t (01.11.2008).

According to the working programme between MÈiMR and Šubinskoe LLP, for the processing of slimes at a rate of 200ktpa, it is anticipated that this reserve will be depleted in 3 – 5 years (before 2013). Slimes from No.2 pond are loaded onto road trucks, or rail cars, and transported to the beneficiation plant at Ridder a distance of 20km for processing together with ore from Tishinskiy mine.

13.6 WAI Mineral Resource Estimate

13.6.1 Topography

Two digital terrain models (DTMs) were created for the Tishinskiy tails. The first DTM was of the base of the TMF and was supplied by Kazzinc. Figure 13.2 is a plan view of the survey supplied by Kazzinc showing the base of the Tishinskiy TMF The second DTM is of the current surface level of the TMF (dated 27/09/2010). For each of the survey files the relevant strings were imported in to Datamine Studio® edited to the correct elevation and used to create wireframe surfaces. The two resultant wireframes delineated a volume, representing the remaining material, for sample selection and block model creation as described below.

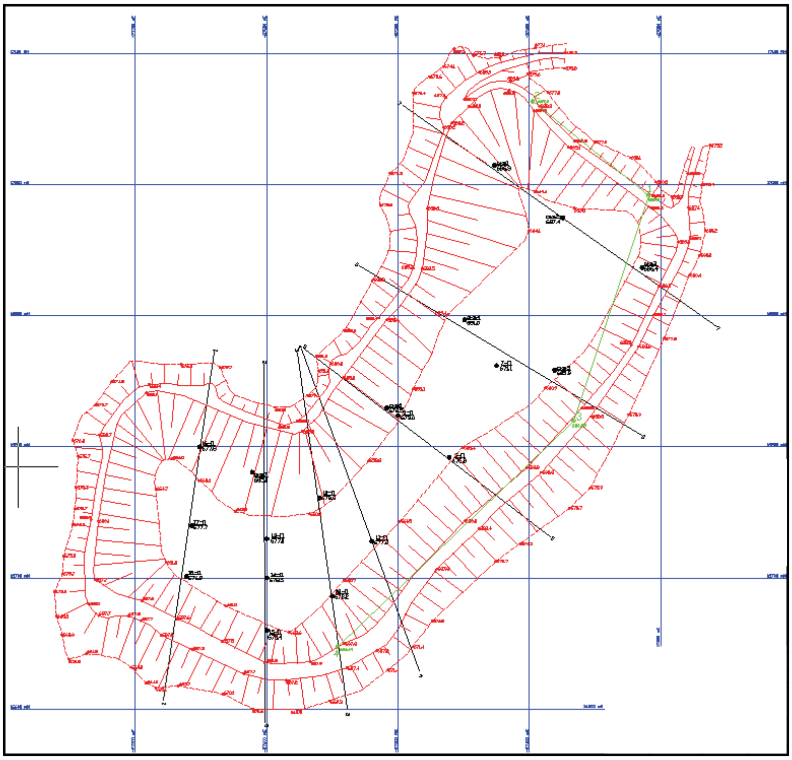


Figure 13.2: Plan View of Survey of Base of Tishinskiy TMF (100m grid)

13.6.2 Database Compilation

All of the sample data collated for the current project are summarised below in Table 13.6. A general plan and cross-section of these drillholes are shown in Figure 13.3 and Figure 13.4 respectively. The holes were drilled in two campaigns and as can be seen from the plan, the drillholes have generally been laid out on irregularly spaced sections between approximately 50m and 110m apart, with spacings of between 10m and 65m along each section line. The profiles were laid out across the strike of the direction of flow of the slimes. All drillholes are vertical.

Table 13.6: Sample Data Summary

	No. Of Holes	No. Of Samples	Holes Lengths (m)	
			Total	Average/Hole
Drillholes 2003	7	123	123	18
Drillholes 2008	12	181	181	15

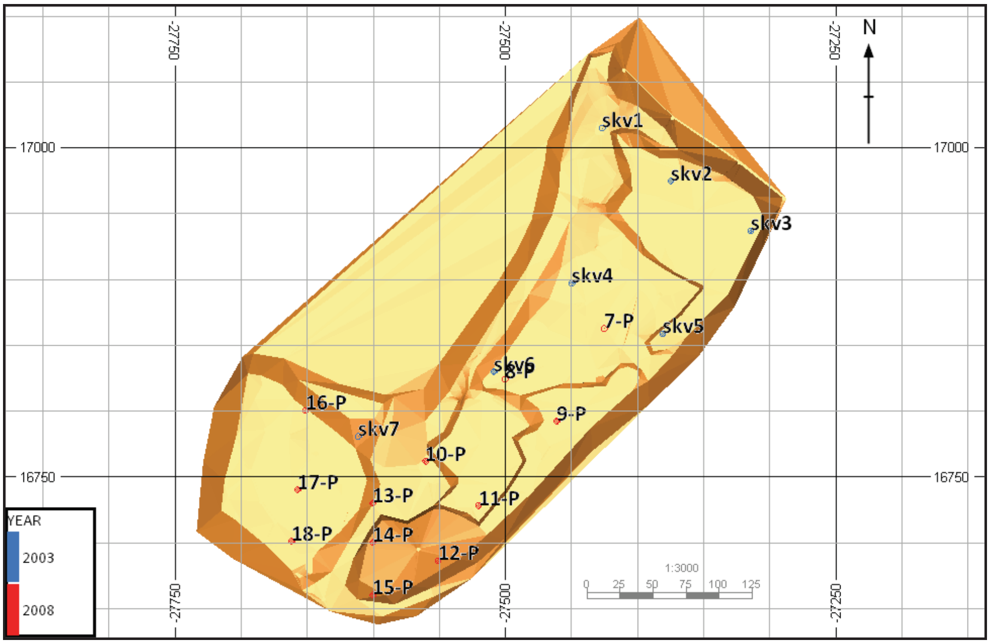


Figure 13.3: Plan View of Drillholes and TMF Resource Envelope

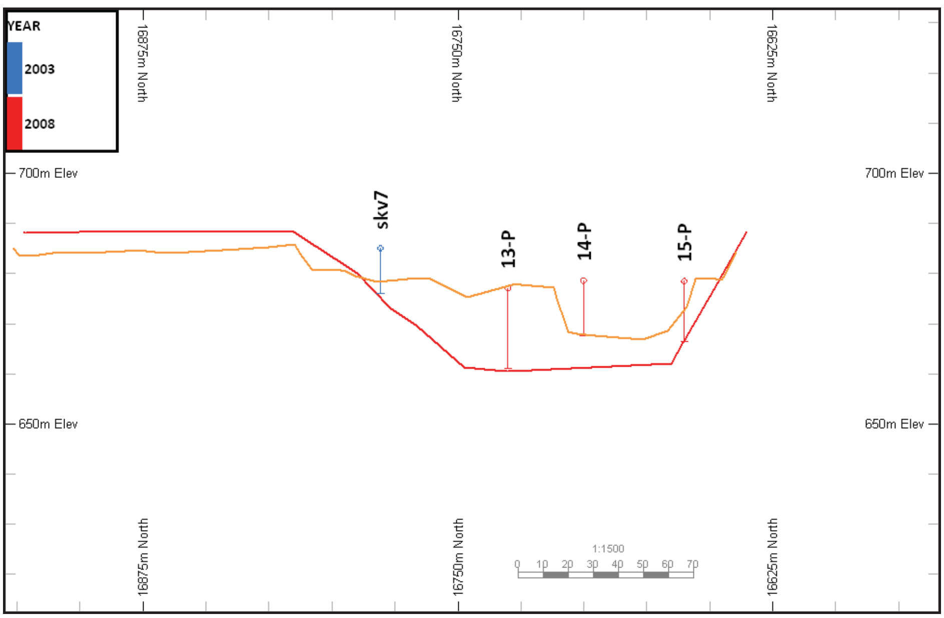


Figure 13.4: Vertical Section at 2 x Vertical Exaggeration -27600mE

13.6.3 Sample Data Processing

The supplied wireframe envelope was used to select and allocate all of the drillhole data. The data selected within the envelope is summarised in Table 13.7. Most of the 164 rejected samples are from parts of the pond that have already been excavated for processing.

Table 13.7: Selected Sample Summary

	No. Of Holes	No. Of Samples	Holes Lengths (m)	
			Total	Average/Hole
Drillholes	16	140	140	9

All of the drillhole samples have been assayed for Au, Ag, Cu, Pb and Zn. Most of the probability plots principally indicate a single log-normal distribution. A statistical summary of the selected samples is shown in Table 13.8. All samples were 1m in length. The two drilling campaigns show differing mean values but a similar variance and so are treated as a single population.

Table 13.8: Statistical Summary of Selected Samples

FIELD	Number	Minimum	Maximum	Mean	Variance	Standard Deviation	Log Estimate of Mean	Coefficient of Variation
AU	140	0.10	1.90	0.34	0.04	0.19	0.34	0.57
AG	140	1.00	17.40	9.61	11.71	3.42	10.32	0.36
CU	140	0.03	0.44	0.22	0.00	0.07	0.23	0.32
PB	140	0.04	1.29	0.72	0.04	0.20	0.75	0.28
ZN	140	0.18	3.95	2.39	0.50	0.70	2.47	0.29

A decile analysis was also completed on the selected gold grades, as shown in Table 13.9. From this and the log-probability plot, there does not appear to be a significant outlier population and so no top-cuts were applied to the data.

Table 13.9: Decile Analysis of Au Grades

Q%_FROM	Q%_TO	NSAMPLES	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	14	0.10	0.10	0.10	1.40	2.94
10	20	14	0.18	0.10	0.20	2.50	5.25
20	30	14	0.20	0.20	0.20	2.80	5.88
30	40	14	0.30	0.30	0.30	4.20	8.82
40	50	14	0.30	0.30	0.30	4.20	8.82
50	60	14	0.36	0.30	0.40	5.10	10.71
60	70	14	0.40	0.40	0.40	5.60	11.76
70	80	14	0.40	0.40	0.40	5.60	11.76
80	90	14	0.49	0.40	0.50	6.90	14.50
90	100	14	0.66	0.50	1.90	9.30	19.54
90	91	1	0.50	0.50	0.50	0.50	1.05
91	92	1	0.50	0.50	0.50	0.50	1.05
92	93	2	0.50	0.50	0.50	1.00	2.10
93	94	1	0.50	0.50	0.50	0.50	1.05
94	95	2	0.50	0.50	0.50	1.00	2.10
95	96	1	0.60	0.60	0.60	0.60	1.26
96	97	1	0.60	0.60	0.60	0.60	1.26
97	98	2	0.60	0.60	0.60	1.20	2.52
98	99	1	0.70	0.70	0.70	0.70	1.47
99	100	2	1.35	0.80	1.90	2.70	5.67
0	**100**	140	0.34	0.10	1.90	47.60	100.00

The data were then converted into 2m composites. A statistical summary of the composites is shown in Table 13.10.

Table 13.10: Statistical Summary of Composites								
FIELD	**Number**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**Log Estimate of Mean**	**Coefficient of Variation**
AU	74	0.10	1.03	0.34	0.02	0.15	0.34	0.44
AG	74	1.00	14.20	9.56	7.42	2.72	9.78	0.28
CU	74	0.04	0.38	0.22	0.00	0.06	0.22	0.27
PB	74	0.08	1.02	0.71	0.03	0.17	0.73	0.23
ZN	74	0.29	3.72	2.39	0.34	0.59	2.43	0.25

13.6.4 Variography

Variography was undertaken test for any spatial continuity of the metal grades, and to assist with the selection of suitable search parameters upon which to base the resource estimation.

Variography was carried out on the 2m composite data contained within the modelled zone. Absolute, as well as relative variograms were generated for AU grades, with a spherical model being used for modelling purposes. These indicated ranges of approximately 47m horizontally and 5m vertically. In the horizontal direction no particular anisotropies were apparent.

13.6.5 Block Modelling

A model prototype was set up with a 10x10x4m parent block size. The details of this prototype are shown in Table 13.11.

Table 13.11: Block Model Prototype					
	Origin	**Maximum**	**Distance**	**Block Size**	**Number**
X	-27,700	-27,200	500	10	50
Y	16,600	17,100	500	10	50
Z	640	700	60	4	15

This prototype was then used as the basis to set up a volumetric model, as controlled by the supplied TMF envelop. Sub-blocks were generated within the structure near the edges, in order to better fit the wireframe geometry, down to minimum dimensions of 1x1x0.5m.

13.6.6 Density

According to data received from water-physical testwork on the properties of the slimes the following parameters apply:

- specific density ranges 1.36 - 3.36t/m^3 (average 2.44t/m^3); and
- volumetric weight is 0.675 - 1.205t/m^3.

The mean volumetric weight is considered to be 0.88t/m^3 in dry weight taken from 17 samples from percussive drilling and 4 bulk samples (1m^3) to determine the volumetric weight.

WAI have consequently applied an average (dry) density value of 0.88t/m^3 to mineral resource estimation.

13.6.7 Grade Estimation

Grade estimation was carried out using Inverse Power of Distance Squared (IDW^2) for all metal grades, although Ordinary Kriging (OK) and Nearest Neighbour (NN) were also used for comparative purposes on the AU grade.

The estimation was run in a three pass plan, the second and third passes using progressively larger search radii to enable the estimation of blocks unestimated on the previous pass. Grades were estimated using the 2m composite drillhole data. The search parameters were derived from the variographic analysis, with the first search distances corresponding to the distance at 2/3rds of the variogram sill value and the second search distance approximating up to the variogram range. Block discretisation was set to 5x5x2 to estimate block grades. Sub-cells received the same estimate as the parent cell. A summary of the estimation parameters is shown in Table 13.12.

Table 13.12: Estimation Parameters					
Search Distances (m)					
1st Search		2nd Search		No. Of Composites	
Horizontal	Vertical	Horizontal	Vertical	Min	Max
31	3	47	5	5	15

- 3rd search up to 250m with a minimum of 1 composite required
- IPD^2 used for Au, Ag, Cu, Pb, Zn grades

13.6.8 Resource Classification

The resource classification for the Tishinskiy slimes pond is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves JORC Code (2004).

The following criteria were established for the classification of resources:

Measured No material was in this category, owing chiefly to lack of closely spaced and sample data;
Indicated Blocks within this category had to be covered by at least 40x40m spaced drillhole data; and
Inferred All other blocks within the TMF, but outside of 40x40m drilled areas, were assigned as *Inferred*.

An example horizontal model section through the block model, showing this resource classification, is shown in Figure 13.5.

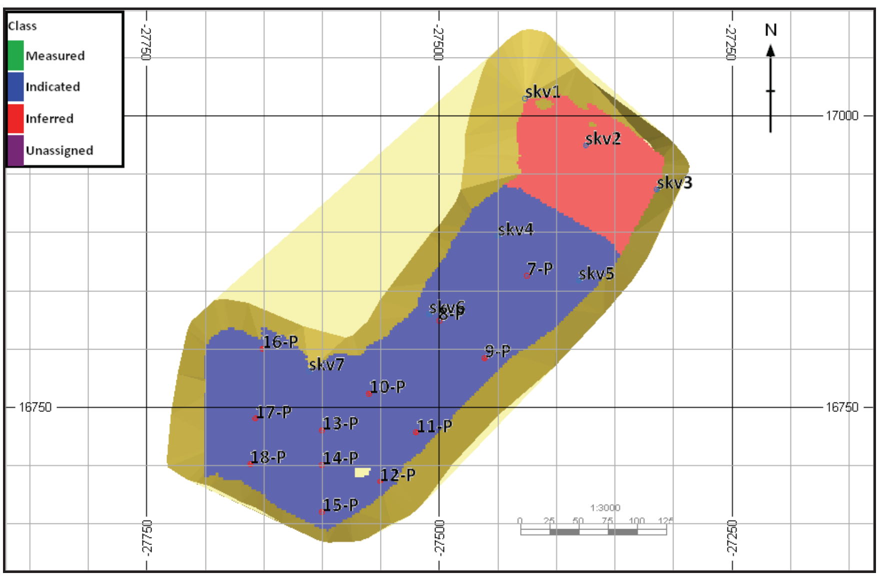


Figure 13.5: Plan View at Surface Showing Resource Classification

13.6.9 Validation

Following grade estimation a statistical and visual assessment of the block model was undertaken to assess successful application of the estimation passes and to ensure that as far as the data allowed, all blocks within the defined TMF domain were estimated. The model validation methods carried out included:

- A visual assessment of grade;
- Global statistical grade validation; and
- Model grade profile (swath plot) analysis.

A visual comparison of composite sample grade and block grade was conducted in cross section and in plan. An example section through the block model, with the estimated AU grades, is shown in Figure 13.6. Visually the model was generally considered to spatially reflect the composite grades.

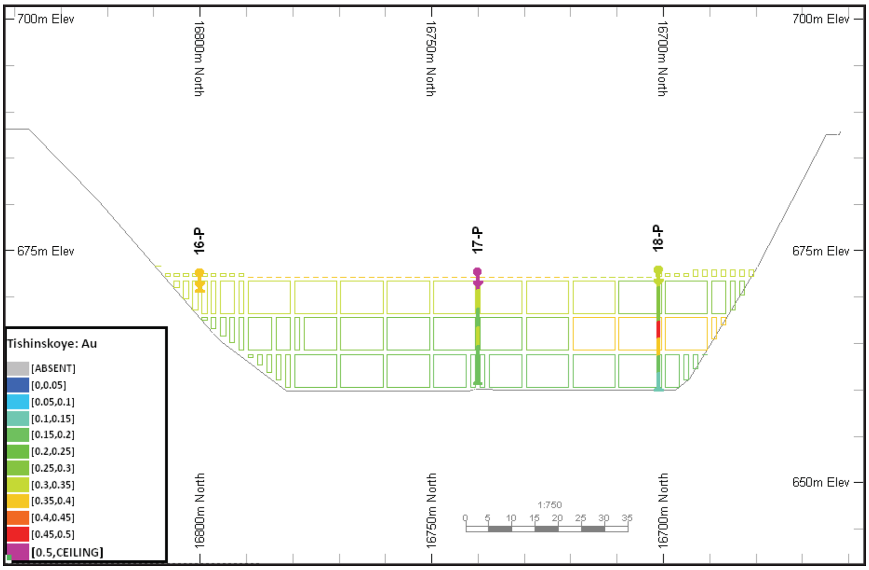


Figure 13.6: Vertical Section at -27660mE Showing Composite and Block Model Au Grades (2 x Vertical Exaggeration)

A comparison of global average grades is summarised below in Table 13.13. As well as the principal inverse-distance grades in the block model, this table includes alternative estimates made using nearest neighbour and ordinary kriging. Overall these global average grades by compare very well.

Table 13.13: Comparison of Global Average Grades

FIELD	Unit	Average Grades		Block Model Grades		
		Samples	Composites	ID	NN	OK
AU	**g/t**	0.34	0.34	0.34	0.35	0.33
AG	**g/t**	9.61	9.56	9.85		
CU	**%**	0.22	0.22	0.23		
PB	**%**	0.72	0.71	0.74		
ZN	**%**	2.39	2.39	2.44		

Notes:

1. Block model grades are for indicated resources only
2. No cut-off grades applied
3. OK (ordinary kriging), NN (nearest neighbour), ID (inverse distance)

Model grade profiles (swath plots) were generated from the model by averaging both the drillhole composites and blocks along 40m spaced N-S and E-W slices. The model average grades were derived from indicated resources only. Overall the drillhole composite average grades and the block model average grades compare extremely well.

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

13.6.10 Resource and Reserve Evaluation

The resource classification for the Tishinskiy simes pond 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 resources are shown inTable 13.4, and are valid as at 01.01.2011.

Table 13.14: Tishinskiy Slimes Mineral Resource Estimate (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Volume (m^3)	Tonnes * (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AuEq (g/t)
Indicated	378,744	333	0.33	9.93	0.22	0.76	2.46	2.77
Inferred	50,909	45	0.58	8.73	0.23	0.56	2.64	3.01

* Based on volume and an average density value (mean volumetric weight) of 0.88t/m^3.

Notes:

- AuEq calculation based on prices of
  - Au 1287 US$/oz
  - Ag 23 US$/oz
  - Cu 7341 US$/t
  - Pb 2422 US$/t
  - Zn 2420 US$/t

WAI has calculated Ore Reserves based upon the above Mineral Resource Estimate and in accordance with the guidelines of the JORC Code (2004). WAI has applied losses of 4.4%, based upon historical data, and dilution of 0.5% to take into account minimal dilution from the bunds and floor.

Table 13.15: Tishinskiy Slimes Ore Reserve Estimate (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004)

Classification	Tonnes (Mt)	Gold (Au)		Silver (Ag)		Copper (Cu)		Lead (pb)		Zinc (Zn)	
		Grade (g/t)	Metal Content (koz)	Grade (g/t)	Metal Content (koz)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)	Grade (%)	Metal Content (kt)
Proven	-	-	-	-	-	-	-	-	-	-	-
Probable	0.32	0.33	3.41	9.89	101.79	0.22	0.71	0.76	2.42	2.44	7.82
Total	**0.32**	**0.33**	**3.41**	**9.89**	**101.79**	**0.22**	**0.71**	**0.76**	**2.42**	**2.44**	**7.82**

14 SHAIMERDEN DEPOSIT

14.1 Introduction

The Shaimerden structure was discovered in 1956 during prospecting for bauxite, exploration for which continued in the area until 1992. In 1989 a programme to investigate scandium levels in the bauxite deposits was commenced and was later extended to detect additional elements. The expanded investigations revealed anomalous lead and zinc concentrations in the Shaimerden area. Exploration drilling to define the resource was undertaken in 1993 by the Stepnaya exploration team of the Zhetigera Expedition, as part of a State run regional investigation programme, which was terminated in 1995 consequent upon a lack of finance.

In 1996 the deposit was taken over by the Zinc Corporation of Kazakhstan (ZCK) comprising joint venture between Ennex International plc and Acier, who implemented a cored borehole drilling programme with the intention of undertaking a bankable feasibility study into the potential for developing the deposit through an open pit operation.

In 2001 ZincOx, formed as a Dutch company in 1997 by Andrew Woollett and Noel Masson formerly with Reunion Mining plc acquired a 95% interest in the Shaimerden zinc oxide deposit.

Due to the deteriorating market conditions, ZincOx sold its 95% interest in the deposit to Kazzinc, Kazakhstan's largest producer of zinc, in December 2003. The consideration for the sale was US$7.5M in cash and, subject to the reconciliation of expected and actual zinc contained additional deferred payments.

The deferred payments, which were made in the January following the year end, amounted to US$0.2375 in every dollar that the zinc price was above US$800/t on the first 200kt Zn mined from the deposit. In order to ensure that the payments were spread over a number of years, there were deemed minimum 40ktpa and maximum 60ktpa zinc mining rates assumed in the agreement.

By utilising heavy duty machinery for the project Kazzinc revised and reduced the term of the pit development by a factor of more than 4 times. ZincOx envisaged mining over 17 years averaging 250ktpa whilst Kazzinc planned to complete development of the same volume within 5.5 years averaging up to 1Mtpa. Latest projections indicate that stockpiled ore will provide raw material to the Ridder Zinc Plant until 2019.

The Shaimerden mine commenced production on 17 September 2006. In spite of considerable problems caused by water ingress to the open pit, which culminated in it flooding in March 2008, the water problem was stabilised and the operation is currently extracting zinc at the rate of about 30kt per month.

14.1.1 Location & Access

Shaimerden is located some 200km SSW of the city of Kostanay and about 110km south west of Lisakovsk, in the Kamysty region of the Kostanay District of northern Kazakhstan (Figure 14.1).

The nearest settlements comprise the village of Krasnogorsk 7km south-east and KrasnoOktyabrsky 14km north-east from the deposit, whilst the regional centre, Kamysty lies 50km to the west.

Figure 14.1: Shaimerden - Location Plan

14.1.2 Access and Infrastructure

The access to the mine from Kostanay is via metalled roads, initially of a reasonable standard comprising a four lane highway to Rudnay but deteriorating thereafter. The open pit lies about 7kms west of Krasnogorsk a small village where the mine hostel is located. Travelling time from Kostenay is from 2.5 hours upwards depending on immediate weather conditions.

Rail links are excellent, connecting the Kostanay district with Russia and the rest of Kazakhstan at Tobol and Arka stations and there exists a short spur to the mine loading facility from the main line.

Mining and ancillary industries developed in the region (within a radius of 200km) include KrasnoOktyabrsky Bauxite Mine (KBRU - Aluminum of Kazakhstan OJSC branch) which is contiguous with the Shaimerden licence, Zhitikara Asbestos Complex, Lisakovsky Iron Ore Mining & Dressing Complex (LGOK) and Sokolovsko-Sarbaisky Iron Ore Mining & Dressing Complex (SSGPO).

There are no permanent rivers in the area, but there are many fresh and saline water lakes, the largest being Lakes Sorkol, Koyandykopa, Sunaly and Tunkuyukty. The nearest permanent water source is the Tobol river which is in 60km north of the deposit.

The region lacks power resources; coal from Karaganda and Kushmurun, and oil products are brought in and electric power is supplied from the Ural power system lines (Russia) via the Troitsk –Rudny–Lisakovsk–Krasnogorsk 110kv power line. Potable and domestic water supply is provided from underground sources.

In addition to the widespread bauxite deposits there are numerous clay deposits suitable for brick production and sand and stone suitable for construction.

There is an available supply of skilled labour in the region.

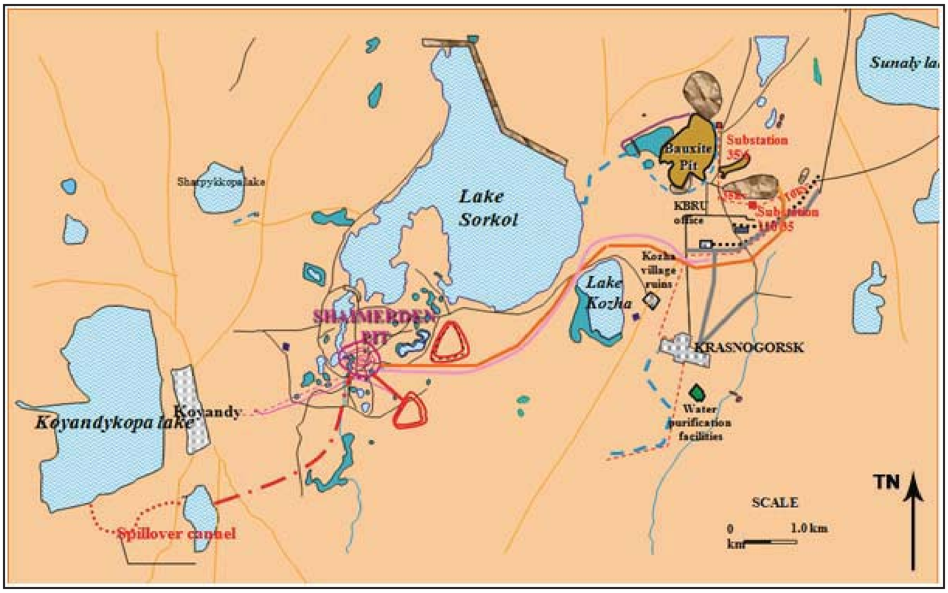


Figure 14.2: Shaimerden – Detailed Location Plan

14.1.3 Topography and Climate

The local relief is flat with absolute levels in the area of the deposit in the range of 239–247m, although in some places undulating with lakes, salt pans and occasional 'ravines' and gullies.

Vegetation is minimal comprising sparse grassland with very occasional low stunted bushes; the whole area is dotted with salt pans and the high water table has produced a level of salinity in the soil inimicable to plant life, in addition to the problems caused by water inflow to the pit.

The climate of the region is typical continental, with short, hot dry summers and extended extremely cold winters. The average annual air temperature is +3°C (Dzhetikara meteorological station +2.66°C). The hottest month is July, with an average monthly temperature of +20.5°C, and a highest temperature of 39.3°C. The lowest temperatures are normally in January, with an absolute minimum of -42.3°C and average temperature varying in the range 12.5 to – 22.2°C. Winds are a permanent feature of the regional climate.

The amount of precipitations per month from the Zhetigara weather station for period from November 2007-March 2010 is summarised in Table 14.1.

Table 14.1: Precipitation Recorded at the Zhetigara Weather Station

Month	1	2	3	4	5	6	7	8	9	10	11	12	Total
2007	-	-	-	-	-	-	-	-	-	-	21.7	5.0	-
2008	8.5	27.0	40.0	26.5	46.6	22.7	23.3	1.0	11.3	21.2	6.1	5.8	240.0
2009	17.1	10.4	22.2	27.3	36.7	28.1	10.8	29.6	1.9	42.3	1.1	50.4	277.9
2010	21.7	13.7	21.6	-	-	-	-	-	-	-	-	-	-

The deepest snow cover in the open areas is not more than 25cm which produces soil freezing (down to 2.0-2.5m) during the winter months.

> ***WAI Comment:*** *The Shaimerden mine is located in an area of unremarkable steppe, and although winter weather conditions can be harsh, the location presents no major obstacles to the operation of the mine.*

14.1.4 Mineral Rights and Permitting

The licence for the right of subsoil use for prospecting and development by the "Shaimerden Joint Venture" for 25 years was approved on 04.05.1997. The licence, comprising 5 years exploration and 20 years mining, comfortably accommodates the revised mining and ore processing schedule noted in section 13.4.5. The mining licence covers an area of 3.23km^2, and the total area under licence is 12.1km^2. The mining licence co-ordinates are given in Table 14.2 below.

Table 14.2: Co-ordinates of Mining Licence Area

Point	Northing	Easting
1	51°59'21"	62°14'35"
2	51°59'21"	62°16'14"
3	51°58'32"	62°16'14"
4	51°58'32"	62°14'35"

> ***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.*

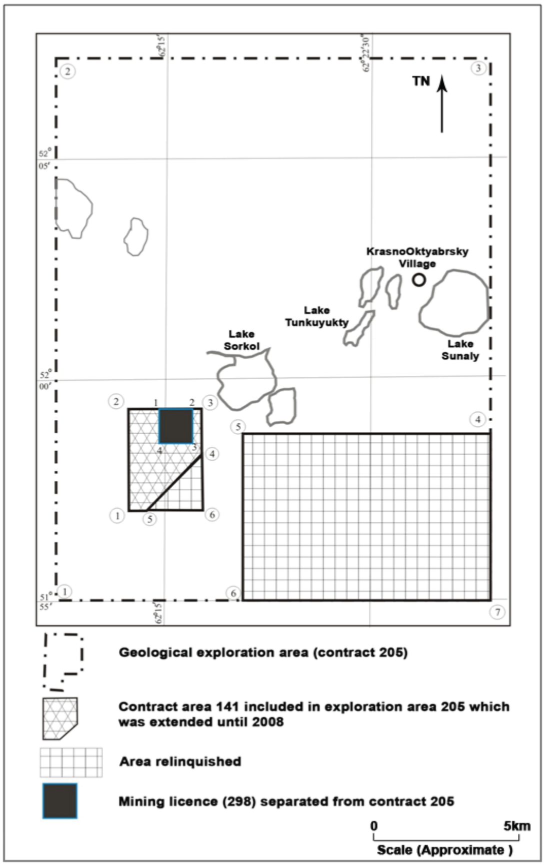


Figure 14.3: Shaimerden – Plan of Exploration and Mining Licence

14.2 Geology and Mineralisation

14.2.1 Regional Geology

The Urals Orogenic belt is divided by the major north-south Main Urals fault. To the west are Precambrian and early Palaeozoic shield and intraplate rift basins. The east is characterised by a mainly oceanic crustal regime with ophiolitic complexes and island arc related volcano-sedimentary basins (andesite-basalt, spilite and trachyte), as well as volcano-plutonic belts and intervening sedimentary basins characteristic of an active continental margin.

Arc complex rocks of Silurian to Carboniferous age are preserved adjacent to the Main Urals fault, while late Devonian to Permian synorogenic granitoid complexes are found in the southern section of the eastern Urals. Shaimerden is located on the eastern side of the north-south trending transcontinental Urals Orogenic belt within the 700km long Valerianovsky belt characterised by Carboniferous sediments and volcanics, and at the southern end of the Kostenay megasyncline and on the eastern limb of the Krasnoktyabr syncline.

The smaller Krasnoktyabr syncline strikes north-south and comprises sedimentary-volcanogenic rocks of Lower Carboniferous age. These rocks are generally characterised as andesite porphyries and occasionally andesite-basalt porphyries, along with their corresponding tuffs, breccias, sandstones and limestones. These volcanic units are cut by dioritic intrusive rocks of Lower Carboniferous age which form massifs and dykes.

Considerable weathering has occurred and the entire sequence is overlain by early Mesozoic alluvial weathering crust formations and Cainozoic sedimentary overburden.

Geological units which occur in the Shaimerden area are summarised below in order of decreasing age.

14.2.1.1 Basement Rocks

The Palaeozoic basement rocks in the Shaimerden area were formed by intensive volcanism, alternating with periods of carbonate accumulation and the formation of terriginous rocks. The thickness of the sedimentary-volcanogenic sequence exceeds 2,000m.

The Shaimerden deposit is hosted by the middle Sokolov formation which was formed in a shallow sea during a period when volcanic activity was dormant. The strata are characterised predominantly by limestone, which is described as organic, dark grey to pink, fine grained, massive and infrequently bedded. Remains of algae, foraminifera, crinoids, corals and brachiopods have been logged within the limestone and dolomitisation is present in places.

The limestone band has a northeasterly strike with a strike length of approximately 25km. The width of the limestone varies from a few hundred metres to more than 3km. In restricted areas, the central part of the limestone band is interbedded with tuffs. At the margins of the limestone unit, transitional zones including volcanic ash and vegetation detritus are noted and the limestone tends to be dark grey and clayey.

The volcanic rocks to the northwest and southeast of the limestone band comprise predominantly fine grained andesite porphyrite which is intensely altered. Tuffs consisting of unsorted volcanic fragments in a chloritised matrix are also present.

Intrusive formations in the Shaimerden area include small massifs and dykes and are composed of diorite or dioritic porphyrite. The composition of these bodies is generally consistent with the composition of the volcanogenic sediments.

14.2.1.2 Weathering Crust

A long break in sedimentation occurred through the Triassic and early Cretaceous periods, accompanied by the formation of a weathering crust on the Palaeozoic rocks. Weathering crusts were preferentially developed over tuffs, porphyrites and strata comprising interbedded limestone and volcanogenic rocks. In areas of comparatively pure limestone, weathering was restricted to a thickness of a few metres and was concentrated in fractured zones.

Although the composition of the weathering crust varies according to the source rocks and the position, the weathering profile is generally represented by a reduction of silica, alkaline and alkaline earth elements. Composition of the weathering crust is generally clay, except over granite porphyry where mica and quartz may be present and in areas where ferric oxides have formed carbonate or chalcedony formations. The thickness of the weathering crust is highly variable, and can extend to between 50 and 200 m in basement depressions. The degree of weathering is strongly controlled by structure. Tectonic faults in the Palaeozoic basement have resulted in the generation of deeply developed, linear weathering crusts, limestone karst formation and development of karst erosion forms in the pre-Cretaceous surface. Linear weathering crusts are also formed over contacts between limestone and volcanogenic rocks.

14.2.1.3 Cretaceous Sediments

Overlying the weathering crust is a sequence of Cretaceous continental clayey sediments which include all the bauxite deposits in the area. The clays have limited regional extent but are present over most of the Sorkofov limestone which hosts the Shaimerden deposit. The Cretaceous clays generally infill karstic and erosional basins in the underlying rocks and their thickness is therefore highly variable; the clays are fairly thin on a regional basis, but locally the thickness can reach 80-100m.

The base of the Cretaceous clay unit is represented by multicoloured fragmental clays which result from minimal transportation and reworking of the weathering crust clays. Sedimentary material tends to be unsorted and brecciated, and inclusions of fragments of Palaeozoic rocks have been noted. In the upper section of Cretaceous clays, argillic bauxite and kaolin are more prevalent.

14.2.1.4 Alluvium

The overburden in the Shaimerden area is of Oligocene age and everywhere comprises alluvial sediments. The sediments are between 30 and 40 m in thickness and include a mixture of laminated clays, sands and sandy clays, with occasional pebble or gravel inclusions. The clays are yellow, grey, brown and generally firm or stiff. The sands comprise micaceous quartz, while pebbles have been described as rounded siliceous rocks. At the base of the alluvial sequence a stiff, plastic olive green clay layer is commonly intersected, with a thickness between a few centimetres and several metres and is referred to as the Chigansk Clay. In some areas the Chigansk Clay is replaced by a quartz glauconitic sand but over the Shaimerden deposit itself it is represented by a dense plastic clay. The Chigansk Clay tends to be present in lenses up to a few kilometres in width and one such lens is present over the Shaimerden deposit.

14.2.1.5 Structure

The limestone sequence of basement rocks identified at the Shaimerden deposit extends approximately 25km in a northeast-southwest orientation and has a width which ranges from a few hundred metres to more than 3km. The northwestern and southeastern margins of the limestone unit are interpreted to be conformable contacts with the surrounding volcanic rocks and represent a change in depositional environment. The abrupt truncation of the limestone unit at the southwestern and northeastern boundaries is attributed to the presence of northwesterly orientated faults. A series of faults with this orientation are interpreted to cut the limestone unit in the Shaimerden region.

Tectonic breaks in the area follow three main structural trends:

1) Northeast-southwest trending breaks which parallel the main strike of the sedimentary-volcanogenic sediments and are associated with a regional structure called the Livanian Fault.
2) Crosscutting northwest-southeast trending faults which in some cases are associated with intrusive bodies and may have controlled volcanism.
3) Minor faults of various orientations with limited offset and throw.

The depth to unweathered basement rocks is strongly controlled by these tectonic features and varies considerably throughout the area. A number of basement troughs have been identified which are associated with tectonic breaks or with lithological contacts. Gravity surveying was used to help locate positions of these troughs.

14.2.2 Local Geology

14.2.2.1 General

In the Shaimerden area the lithologies comprise a sequence of intertidal to open-marine carbonates and evaporites of Viséan (Early Carboniferous) age and andesitic volcanics that are intruded by small diorite and granite bodies (Figure 14.4). Upper Cretaceous sediments fill karst depressions within the Carboniferous limestone and host bauxite deposits. Overlying the whole area are 40m of Tertiary-Quaternary sands and clays (Figure 14.4, Figure 14.5 and Figure 14.6).

The Shaimerden lead zinc orebody is located in a deep depression in karstic limestone of the Sorkolov formation. The depression is associated with a west northwest tectonic break which is accompanied by elongated linear intrusive bodies composed of andesite and diorite porphyry/rite. Cross-cutting structures are also present with the orebody being located at the intersection of these structures.

The alluvium in the deposit area is relatively uniform in thickness while the Cretaceous clays and weathering crust are variable in thickness. The greatest thickness of clay material occurs at the deposit location. The deep troughs of weathered limestone, which appear to follow faults, are narrow in extent and steeply dipping.

Table 14.3: Mine Stratigraphic Succession

Geochronological Period	Lithology	Thickness (m) and Occurence
Quaternary (Q)	Soil, sandy loam, loam	1 – 2m to 3-5m (north west area)
Paleogene(Oligocene) (P_3)	Sandy clay and fine grained sand with minor Clay component	20-25m to 35-40m (western area)
Paleogene(Oligocene-Eocene) (P_{2-3})	Predominantly glauconitic clay	5 – 15m from east to west (Level 220-210)
Upper Cretaceous (K_2)	Hard and soft bauxitic clay' Variegated dense clay and lignite	5-10 to 30-40m (pit perimeter, NE to SSW (Level 210 – 200)
Upper Trias – Lower Cretaceous ($T_3 – K_2$)	Weathered alluvial shales and limestones	Levels 210 – 190m
Lower Carboniferous (C_1)	Grey limestones and dark grey shales	Ore hosting horizons below Levels 200 = 190m

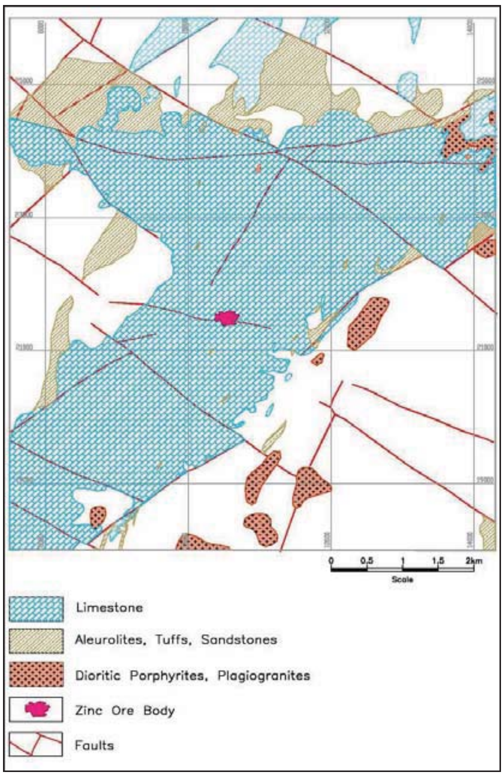


Figure 14.4: Shaimerden – Area Geology

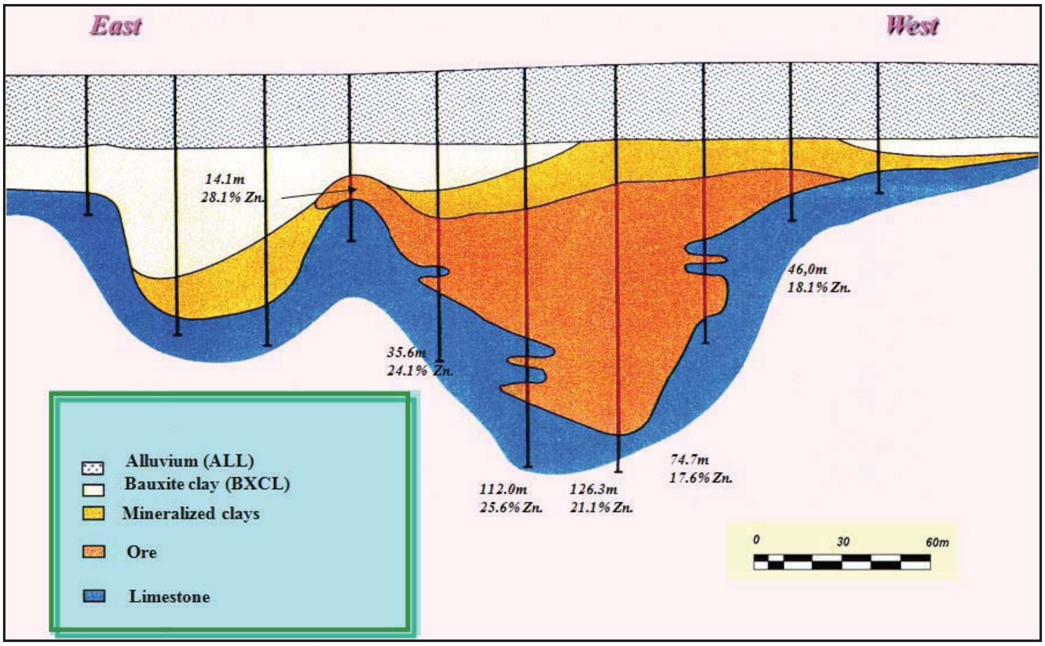


Figure 14.5: Geology of the Shaimerden Pit Area – East-West Section

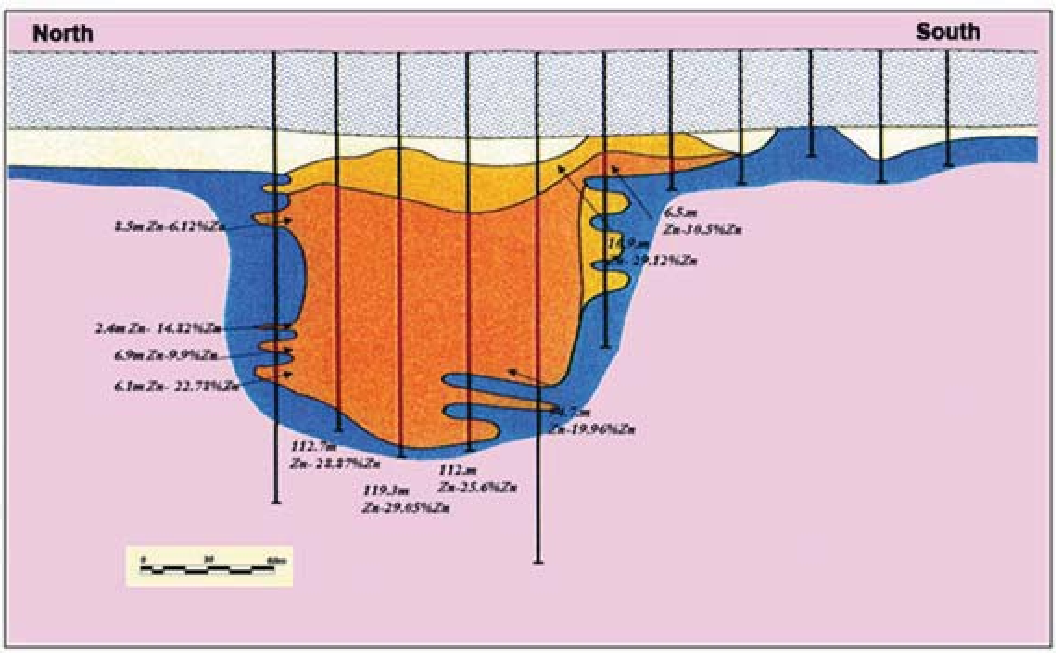


Figure 14.6: Geology of the Shaimerden Pit Area –North-South Section. (Legend as for Figure 14.5)

14.2.2.2 Ore Body

The deposit is hosted in a massive, clean Carboniferous limestone and has resulted from the in-situ oxidation during the Triassic-Cretaceous period of a body of massive sphalerite mineralisation. The deposit occurs within a weathered depression measuring 450m east-west, 150m north-south with mineralisation occurring to a depth of 240m below the surface topography.

Emplacement of the orebody is thought to have resulted from the circulation of ore-bearing geothermal solutions in the period after deposition of the rocks and prior to the Triassic weathering. The tectonic structures present in the area provided a channelling device and an extensional zone during emplacement. The limestone rocks are likely to have acted as a geochemical barrier resulting in the deposition of the ores.

Although drilling did not intersect faulting, significant faulting in the area is suggested by the presence of polymict debris flows comprising a wide range of carbonate facies and by large variations in micropaleontologic dates. Sulphide deposits replaced hydrothermally dolomitized carbonates and were subsequently reworked into polymict conglomerates of probable Carboniferous age that were deposited in a marine environment.

14.2.3 Mineralisation

14.2.3.1 Ore Body Composition

The main Shaimerden orebody comprises supergene oxidised zinc ores with remnant sulphides towards the centre, indicating oxidation from the outside margins to the centre of the deposit. There is an outward zoning comprising five categories, based on macroscopic and microscopic studies of the ore:

- Massive sulphides – (2.8% of the reserves) preserved in the centre of the orebody and consisting mainly of remnant massive sulphides consist of 90% sphalerite with minor galena and pyrite and an average grade of 46% Zn, 1.2% Pb, 6.1g/t Ag;
- Massive hemimorphite-smithsonite – (11.7% of the reserve) surrounding and intimately related to the massive sulphides and varying from a massive fine-grained ore to a brecciated vuggy rock; the zone comprises fine grained hemimorphite-smithsonite clasts in a fine grained carbonate-hemimorphite-smithsonite matrix which has a sharp 2-5mm transition to the massive sulphides, with a similar texture and always intimately related to the massive sulphides. There are additional gangue mineral clays, carbonates and oxides. This ore type assays 30 to 45% Zn, with an average resource grade of 35.2% Zn;
- Stony ore – (8.2% of the reserve) a dark green to dark red weathered, competent and sometimes friable material with a relict breccia texture. It is an intermediate weathered material between the massive hemimorphite-smithsonite and the mineralised clays;
- Mineralised clays – characterised by their grey/green colour and high percentage of gritty and fragmental material. The zinc occurs mostly as small fragments of hemimorphite and smithsonite and partly as clay minerals. The detailed composition is:
    - CL3 - (38.1% of the reserve) grey-green chlorite-smectite, kaolinite and/or montmorillonite clays with more than 40% gritty material (hemimorphite and smithsonite with minor sauconite and zincite) and a gangue of lesser calcite, dolomite, siderite and quartz and grades averaging 24.9% Zn,
    - CL2 - (28.8% of the reserve) similar grey-green clays to CL3 but with 10-40% gritty material (hemimorphite and smithsonite again) with grades averaging 13.1% Zn.
    - CL1 - (4.5% of the reserve) mottled, multicoloured, white to orange brown, massive and plastic clays with a grit content of less than 10% and averaging 1.8% Zn which is found above and sometimes laterally to the main massive sulphides and massive hemimorphite-smithsonite.
- Mineralised limestone – (5.9% of the reserve) occurs along the margin of the weathered trough and consists of intervals of mineralised clays within the limestone.

Weathering of the sulphide mineral deposits took place during the Triassic Period, following uplift during the late Paleozoic. The weathering occurred in situ, and small intervals of relict sulphides were preserved in the centre of the deposit.

14.2.3.2 Mineralogy

Historically, smithsonite, or zinc spar (ZnCO3), was identified with hemimorphite before it was realised that they were two distinct minerals. The two minerals are very similar in appearance and the term calamine has been used for both, leading to some confusion.

Smithsonite is a variably colored trigonal mineral which only rarely is found in well formed crystals. The typical habit is as earthy botryoidal masses. It has a Mohs hardness of 4.5 and a specific gravity of 4.4-4.5. It occurs as a secondary mineral in the weathering or oxidation zone of zinc-bearing ore deposits. It sometimes occurs as replacement bodies in carbonate rocks and as such may constitute zinc ore. It commonly occurs in association with hemimorphite, willemite, hydrozincite, cerussite, malachite, azurite, aurichalcite and anglesite. It forms two limited solid solution series, with substitution of manganese leading to rhodochrosite, and with iron, leading to siderite.

Hemimorphite ($Zn4Si_2O_7(OH)_2$ H_2O), is a sorosilicate mineral which has been mined from days of old from the upper parts of zinc and lead ores, chiefly associated with smithsonite. The silicate was the rarer of the two, and was named hemimorphite because of the hemimorph development of its crystals. This unusual form, which is typical of only a few minerals, means that the crystals are terminated by dissimilar faces. Hemimorphite most commonly forms crystalline crusts and layers, also massive, granular, rounded and reniform aggregates, concentrically striated, or finely needle-shaped, fibrous or stalactitic, and rarely fan-shaped clusters of crystals. Some specimens show strong green fluorescence in shortwave ultraviolet light (253.7mm) and weak light pink fluorescence in longwave UV.

14.3 Exploration Works

14.3.1 Historical Works

The Shaimerden structure was discovered in 1956 during prospecting for bauxite, exploration for which continued in the area until 1992. In 1989 a programme to investigate scandium levels in the bauxite deposits was commenced and was later extended to detect additional elements.

The expanded investigations revealed anomalous lead and zinc concentrations in the Shaimerden area. Exploration drilling to define the resource was undertaken in 1993 by the Stepnaya exploration team of the Zhetigera Expedition, as part of a State run regional investigation programme, which was terminated in 1995 consequent upon a lack of finance.

The subsequent exploration of the deposit comprising vertical coring holes from surface was completed by "Shaimerden" a Kazakh-British joint enterprise and consisted of a single phase. The scope and types of main exploration works are summarised in Table 14.4.

Table 14.4: Summary – Exploration Programme Works		
Scope and Types of Geological Exploration	**Meas. units**	**Work Volume**
Exploration-evaluation works	R.m./Boreholes	6501.1/58
Exploration	R.m./Boreholes	5853.6/51
Drilling for sampling	R.m./Boreholes	551.3/3
Ore body outline (contour) drilling	R.m./Boreholes	239.9/3
Geological and engineering research	R.m./Boreholes	716.2/9
Hydrogeological and engineering works	R.m./Boreholes	714.8/14
Sampling:		
Samples;	Samples	3,935
Geochemical samples;	Samples	598
Hand specimen	Specimen	342
Lab research:		
Spectral testing;	Tests	4,329
Chemical testing for main components	Tests	3,935
Technological investigation of laboratory samples	Samples	267
Geophysical research: - directional logging	B/hole	15

The density of the exploration borehole grid was 25-50x12.5-25m for the C_1 category and 50x100m for the C_2 category; borehole profiles were orientated east–west and the average recovery of core through the mineralised intersections used in the reserve estimate was 87%.

Reserves of oxide-bearing ore were estimated to a depth of 250m. The majority of boreholes intersected the ore zone fully, with the exception of three (Bhs 8831, SH9, and 9004) where drilling was stopped in the ore body.

The deepest bore-hole (Bh 9021 in profile 21500) was drilled and sampled to 305m, and in consequence no conclusions could initially be drawn about the probability of finding sulphide ore at depth below the main ore body and the genesis of the deposit.

Ore intervals were sampled by sectional method according to lithological variation. Hand specimens were used to study the material constitution of the ore, its texture and structural features, as well as physical and mechanical properties.

In the course of borehole exploration geophysical downhole logging was used to a limited extent. From this it was evident that future exploration in the area for zinc-lead deposits similar to Shaimerden would involve x-ray and radiometric well logging and x-ray and radiometric core sampling which would facilitate the evaluation of zinc grade in field conditions (with 10-15% accuracy), reduce the volume of expensive core sampling, and enable continual adjustments to be made to the drilling programme grid in accordance with the prospectivity or complexity of the mineralised intersections.

In 1998 Ennex announced the outcome of analysis and checking of its confirmatory drilling programme completed in 1997. The results confirmed both the thickness and grades obtained within the main mineralised body which was estimated to contain a resource of approximately 4Mt at a grade of 25% Zn.

This resource was calculated originally by indigenous operators and verified during 1997 by Ennex and its consultants. The company also announced that samples from seven earlier drillholes not previously assayed had been analysed and that a new detailed drilling programme was being prepared for feasibility study purposes, including mine design and production scheduling.

Table 14.5: Results from Ennex Drilling (1997)					
Hole No	**From (m)**	**To (m)**	**Interval (m)**	**Zn %**	**Notes**
8830T[1]	51.00	221.00	170.00	27.80	Twinned
8605T[1]	112.25	149.45	37.20	15.56	Twinned
SH1[2]	62.98	90.38	27.40	24.90	New
SH2[2]	72.55	121.85	19.30	18.03	New
9011[3]	68.4	171.3	102.9	30.89	Previous
9020[3]	84.0	196.0	112.0	25.60	Previous
9021[3]	73.6	164.2	90.6	21.97	Previous
9022[3]	53.5	79.3	25.8	24.19	Previous

[1] Twinned with earlier drilling. Both were drilled to greater depths and each encountered the original mineralisation, as well as additional mineralisation at lower grades

[2] Sited within the mineralised body but away from previous drillsites. The results from these holes also confirmed the presence of economic grade mineralisation

[3] Four boreholes of the seven not assayed previously, sited within the mineralised body

The other three drillholes outside the main mineralised body contained thin intercepts (3–6m) with low grade zinc (5.0-7.5% Zn).

All the analytical results were calculated using a 5% grade cut-off.

> ***WAI Comment:*** *The Western Torgai exploration asset, comprising the Karabaitalsky and Sakharovsko-Adaevskiy sites, lies immediately to the south west of the original Shaimerden exploration licence and contracts are in the process of registration at the Ministry of Industry and New Technologies. At the Karabaitalsky site, a 3 year programme of drilling (19,000m) and geophysics is planned to investigate previously identified anomalies representing porphyry gold and polymetallic deposits; at the Sakharovsko-Adaevskiy site a similar 3 year programme has been drawn up to investigate the Klochkovskoye quartz diorite porphyry Cu/Mo deposit (intersected in old soviet boreholes) with 13,000m drilling and geophysics.*

14.3.2 Sample Collection

14.3.2.1 ROM Ore Sampling

Sampling at Shaimerden is carried out in pit during mining and before and after crushing. Blast hole sludge sampling is routinely undertaken in all blast areas in the pit, and the number of blast holes checked depends on the components of the block being mined i.e. ore, contact zone or waste.

A shift geologist is always present during mining operations to make an initial assessment of the grades of the samples from the blast holes and the ore being mined, using a portable XRF Spectrometer, and the information is supplied to the pit foreman and a controller from the technical and analytical control services. The latter makes an informed decision on the appropriate ore storage 'zone', based on the initial grade estimate, lithology and moisture content of the material and gives directions to the pit foreman, geologist and dump truck operator for its dispatch.

The ROM ore (±500mm size) is transported to the crushing plant storage 'zone' in a 'Caterpillar' dump truck of 30m^3 capacity, after passing over a 'two way' weighbridge; here a member of the sampling department directs the driver and records the truck number, weight of ore, unloading zone, lithology, sample number and time of load arrival. A composite grab sample from three trucks weighing >2kg is transferred to the laboratory.

14.3.2.2 Crushed Ore Sampling

Crushed ore is sampled by an automatic belt sampler and a composite sample, of 25 or 50 'cuts' is collected, representing 50 or 100t batches respectively. The sampling process is controlled automatically and the crushed ore is transferred to the various compartments (8 @ 400t each) of the storage bin by a movable conveyor belt.

14.3.3 Sample Preparation

All samples sent to the laboratory are split, one part sent for moisture determination and the other for analysis; the sample for analysis is dried at 105-110°C and crushed to <10mm through a jaw crusher. A reduction in sample size and further drying is followed by grinding in a rotary mill.

A laboratory sample of >0.25kg is ground in a ring mill, and sieved through an 0.080mm mesh, the oversize being re-circulated through the mill and sieve. The <0.080mm fraction is quartered and two samples of >100g are separated, one for analysis and the other sent to storage as a duplicate for re-analysis if required.

The sample material is compressed into a disc prior to analysis in the Spectro Expos Spectrometer.

> ***WAI Comment:*** *The laboratory is clean, well organised and run efficiently and is fit for purpose to provide analysis for the stockpiles of mined material and the crushed ore from the crushing plant, transferred to the storage bins, where blending occurs.*

14.3.4 Quality Control

10% of all the mining, geological and crushing plant samples are subject to internal checks using duplicate and standard samples. External sample checks include the dispatch of 3% of all samples to the certified SEVKAZGRA laboratory in Kostanai, every three months.

The average deviations for the four randomly chosen 'orders' range from -0.5%-0.21% Zn whilst the lowest and highest individual sample deviations are zero to 1.0%Zn; the two data sets are plotted on the regression curve (Figure 14.7).

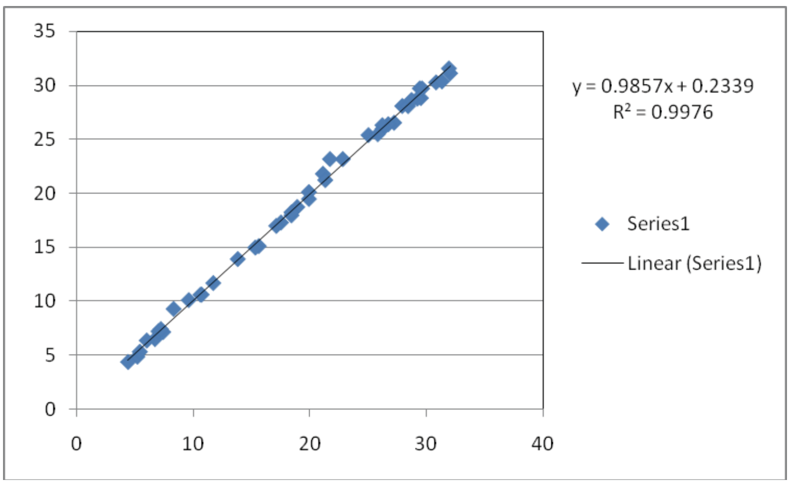


Figure 14.7: Regression Curve - Original Shaimerden Analyses (Horizontal Axis) and the Duplicate SEVKAZGRA Control Analyses (Vertical Axis)

> ***WAI Comment:*** *The two sets of analyses exhibit a high level of correlation and indicate that the Shaimerden laboratory is run competently and has the appropriate QA/QC protocols in place.*

The zinc plant at the Ridder Metallurgical Complex (RMC) samples 10% of the Shaimerden product; the error tolerance between the two sets of analyses are summarised in Table 14.6.

Table 14.6: Tolerated Error – Shaimerden/Ridder Metallurgical Complex

Component	Ore grade, %	Relative Tolerated Error (%)
Zn	>5	3.5
	3 - 5	6.0
	1 - 3	8.0
	0.3 – 1.0	11.0

WAI Comment: *It was stated that Shaimerden has never incurred any penalties and that the RMC has always confirmed the validity of the original analyses within the tolerances given above. Although both operations are within the Kazzinc organisation, the reconciliation data from the external laboratory supports this statement.*

14.4 Mining

14.4.1 Introduction

The Shaimerden mine has been in production since 2005 but is expected to cease mining in May 2011. The near surface zinc orebody has meant it is suitable for conventional truck and shovel open pit mining. Zinc ore is mined from a single pit, hauled to stockpiles, crushed and transported by rail to Ridder in east Kazakhstan.

Figure 14.8 below is the general Shaimerden site layout showing the location of the pit, stockpiles, fuel storage and the railway loading facilities.

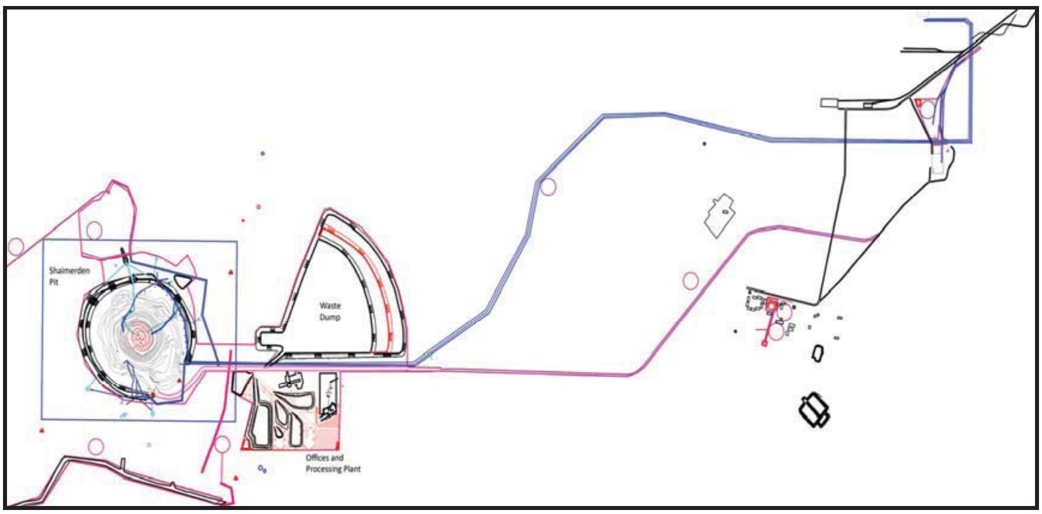


Figure 14.8: Shaimerden Site Layout

14.4.2 Geotechnical Assessment

The initial pit slope designs are based on criteria developed by Golder Associates Ltd. ("Golder") in their 1999 report. Golder recommended slope angles of 22° for the weathered alluvium material, ranging from 11-15° for the cretaceous/triassic clays and 56◦ for the limestones.

Golder have been regular trips to site over the duration of the mine life and the current recommended slope angles are ranging from 12-17° for the weathered alluvium material , 50◦ for the cretaceous/triassic clays and 70° for the limestones.

Photo 14.1: Shaimerden Pit Looking East

> ***WAI Comment:*** *Having reviewed the geotechnical reports, WAI feel that all aspects of the final wall stability have been covered and the ongoing stability of the pit are in hand.*

14.4.3 Pit Outlines

The Shaimerden deposit has 12 orebodies. The main bodies are No.1 and No.3, which contain 96% of the ore reserve. The pit outlines are determined for each ore body according to lateral and longitudinal geological cross sections, based on the condition of mining the maximum amount of ore reserve.

The deposit has been developed as one pit. The balance reserve is represented by ore bodies, having confluent bedding and being mined in one section. For the purposes of mine design, a balance reserve for the Shaimerden zinc ore of C_1+C_2 category was accepted on 01 January 2003 and approved by the State Reserve Committee of the Republic of Kazakhstan.

Table 14.7: Reserves Approved for Pit Design

Indices	Units	C1 +C2 Balance reserve
Balance Reserve of the deposit as on January, 1st of 2003		
Raw ore reserve	kt	3808.5
Dry ore reserve*	kt	3475.7
Zinc reserve	kt	914.0
Average zinc grade	%	26.29
Including industrial reserve within the pit outline		
Raw ore reserve*	kt	3805.3
Dry ore reserve	kt	3472.8
Zinc reserve	kt	913.2
Average zinc grade	%	26.30
Commodity reserve (3.7% losses, and 4.5% dilution)		
Raw ore reserve*	kt	3853.3
Zinc reserve	kt	879.4
Average zinc grade	%	22.82
Dry ore reserve	kt	3501.9
Zinc reserve	kt	879.4
Average zinc grade	%	25.11
*Average moisture of 9.12%		

14.4.4 Pit Design

The initial pit designs were performed by Golder Associates in 1999 using Gemcom software with the overall open pit wall slopes varying from 11-56° as part of the 1999 Feasibility Study. This design has since been modified by Golder Associates to incorporate several final wall slips that occurred in 2006 in the soft rock at the top of the open pit mine.

The current pit design parameters for the Shaimerden pit are:

- Height between safety berm – 10-15m;
- Bench height – 5m, mining on 2.5m flitches;
- Safety berm width – 6m, with a 15m berm at the 205 level;
- Haul road width – 24m above the 110 level, two way traffic;
  – 16m below the 110 level, one way traffic;
- Haul road gradient – 1:10;
- Final Wall Angles: 12-70°, and
- Cut-off Grade: 2% Zn.

The mine dimensions are approximately 745x845m at the top of the pit and approximately 300m in diameter at the bottom. At the time of the visit, the production activity was on the 100 level and mining will finish on the 40 level. Table 14.8 summarises the pit dimension, and volume.

Table 14.8: Pit Dimensions and Volumes

Design Dimensions		Total volume, (000m^3)	Overburden volume (000m^3)	Av. Strip Ratio	
Dimensions (m)	Depth (m)			m^3/t	m^3/ m^3
710 x 845	207	22,350	20,847	5.95	13.87

14.4.5 Production Schedule

The development of the mine commenced in 2005 and ore was first mined in 2006. Since the mine commenced operations, a total of 21.6Mbcm has been excavated up until 1 October 2010 and the breakdown of materials excavated are shown in Table 14.9 below.

Table 14.9: Total Mining Volumes from the Inception of Mining until 01.10.2010

Component	m^3
Total mined material :	**21,587,500**
Overburden Including :	
Waste dump	11,660,804
Road construction, (dumps)	6,862,853
Bauxite stockpile	53,229
Off- balance stockpile	63,028
Limestone stockpile	1,715,846
Total Overburden	**20,355,760**
Ore	**1,231,740**

The production rate has gradually increased over the life of the mine to peak at 950ktpa in 2009. The reason for the slow start up was the constant delays to production caused by the inflow of ground water into the pit. This situation is now under control and production is expected to reach 895kt for 2010 and 410kt for 2011 with the mine being exhausted in May 2011. Table 14.10 below shows the historical and scheduled production for the life of the Shaimerden pit.

Table 14.10: Historical and Scheduled Production for Shaimerden Pit									
Work Title	Unit	Quantity	Mining period						
			2005	2006	2007	2008	2009	2010	2011
Dry ore mining	kt	3,502	-	53	439	755	950	895	410
Overburden operations	(000)m^3	20,847	4,253	5,277	3,263	3,135	3,343	1,375	201
Total mined material	(000)m^3	22,350	4,253	5,300	3,451	3,459	3,750	1,760	377

All of the ore from the pit is hauled to various stockpiles corresponding to zinc grade. The stockpiled ore is blended through a crusher to maintain a constant head grade. As the crushing rate is lower than the mining rate, it is intended that the crusher will continue after the open pit has been exhausted. Table 14.11 shows the historical and scheduled production from the stockpiles.

Table 14.11: Historical and Scheduled Production from Shaimerden Stockpiles			
Year	Ore from Stockpile (kt)	Zinc grade (%)	Zn (t)
2006	53	19.27	10,212
2007	439	20.64	90,610
2008	755	20.39	153,982
2009	-	-	-
2010	-	-	-
2011	80	23	18 400
2012	100	23	23 000
2013	280	22,5	63 000
2014	296,272	21,73	64 381
2015	300	21,66	64 975
2016	300	21,66	64 975
2017	300	21,66	64 975
2018	300	21,66	64 975
2019	300	21,66	64 975
2020	315,157	21,85	68 275

14.4.6 Mining Operations

Mining is based on 5m benches for ore and 10m for waste; excavated on 2.5m flitches in ore and 5.0m flitches in waste. Safety berms are left every 10 or 15m depending on the ground conditions. The main ground engaging tools are three CAT 5110B backhoe excavators with 85 tonnes CAT 777D haul trucks.

Shaimerden is also mining small quantities of limestone which may be used as a raw material in the processing plant at the Ridder Works.

14.4.6.1 Drilling and Blasting

Drilling and blasting at Shaimerden is very straightforward as the site uses diesel drilling rigs with hydroperforators and a modern emulsion explosive. All blasting is performed by contractors who deliver the explosives on the day of the blast so there is no need for magazine facilities.

An Atlas Copco DM45 rig is used for drilling 165mm diameter blast holes. The drilling grids are 3.0m x 3.0 and 4.0m x 4.0m depending on the ground conditions. Two smaller rigs, an Atlas Copco RocL8 and Ingersoll Rand CM780D rigs are used to drill 102mm and 114mm diameter holes.

The main explosive is an emulsion matrix which has a chemical gassing agent added to it in the explosive truck. It is pumped from the truck through a hose into the blast holes and becomes sensitized on discharge from the hose. The main charge is initiated with cast boosters and non electric detonators. The explosive supplier also provides the shotfiring services.

14.4.6.2 Selective Digging

There are 4 ore types at Shaimerden based upon zinc grade. The selective digging of the different ore types is supervised by mine geologists who take zinc grade readings using a handheld x-ray fluoresces (XRF) spectrometer which can quickly indicate the zinc grade. The geologist then tells the truck operator which stockpile the ore has to be taken.

14.4.6.3 Load and Haul

Ore is hauled from the Shaimerden pit on a well maintained haul road with a gradient of 10% by Shaimerden Ltd personnel in the CAT 777D haul trucks to the various stockpiles depending on the zinc grade. The stockpiles are:

- SP25: 2-5% Zn;
- SP23: 5-15% Zn;
- SP22: 15-25% Zn, and
- SP24: over 25% Zn.

Ore is then taken from the stockpiles and blended through a crusher to maintain a constant head grade prior to being shipped by rail to the Metallurgical Complex in Ridder. All material with a Zn grade less than 2% is hauled to the waste dump.

14.4.6.4 Pit Dewatering

The Shaimerden pit has had a history of water management as the site was located in a lake when it was discovered. Bunding was constructed to allow the exploration and the mine development to take place. A dewatering programme was implemented to minimise disruptions to production.

The dewatering system consists of 3 in-pit bores located in the bottom of the pit and several bores located on the 205 level. New bores are being drilled on the 205 level by contractors. The diameter of these bores will 490mm to a depth 210m and will give a 30m draw down from the final working level.

Submersible pumps are being used to pump water from the bores and out of the pit. Currently the mine is moving approximately 3,600m^3/hr. The water from the pit is discharged into holding pond to the north west of the pit and then pumped to a lake 12km west of the site.

14.4.7 Mining Fleet

The current mining fleet includes the CAT 5110B Backhoe Excavators, CAT 777D Haul Trucks, CAT 16G and 16M Motor Graders, the three drill rigs and four CAT D9 tracked dozers.

The ancillary equipment includes smaller excavators and front end loaders for the stockpiles, water trucks, service vehicles and light vehicles. A complete maintenance shop is located adjacent to the mine site offices. Table 14.12 below shows the primary mining fleet and their function.

Shaimerden management have made plans for the sale or the redeployment of the equipment to other Kazzinc sites as the mining activity draws to an end in 2011. It is intended for the CAT777D haul trucks to be redeployed at Vasilkovskoye mine whereas the decision to sell the large excavators and drill rigs will be made when production finishes.

Table 14.12: Shaimerden Mining Fleet		
Type	**No.**	**Function**
CAT 5110 Backhoe Excavators	3	Mine Production
CAT 77D Haul Trucks	10	Mine Production
CAT D9 Tracked Dozers	4	Mine Production
Atlas Copco DM45	1	Mine Production
Atlas Copco Roc L8	1	Mine Production
Ingersoll Rand CM780	1	Mine Production
CAT 16 Motor Graders	2	Road Maintenance
CAT GS683E Road Roller	1	Road Maintenance
Hitachi 450H Backhoe Excavators	2	Stockpile Management
Hitachi 470H Backhoe Excavator	1	Stockpile Management
CAT 988 Wheel Loader	1	Stockpile Management
CAT 966 Wheel Loaders	3	Stockpile Management

14.4.8 Mine Personnel

There are 209 personnel associated with the mining activities including mine management, geology, mining, surveying and operations. Management and Technical Services work day shift 5 days a week, whereas the mining operations are based on two 12hr shifts (9-9) with 2 hours per shift dedicated to maintenance, shift change over and blasting. The miners work an 8 on/8 off roster with day shift changing to night shift after 4 days.

14.4.9 Operational Costs

Table 14.13 below summarises the mining operating costs for the Shaimerden mine.

Table 14.13: Shaimerden Mining Operating Costs			
		Planned	**Actual**
Volume Excavated	m^3	3,750,000	3,750,200
Ore Mined	t	1,000,000	951,057
Loading	US$	2,434,414	2,714,071
	US$/$m^3$	0.65	0.72
	US$/t ore	2.43	2.85
Hauling	US$	5,732,791	4,586,086
	US$/$m^3$	1.53	1.22
	US$/t ore	5.73	4.82
Drill & Blasting	US$	3,223,093	3,041,274
	US$/$m^3$	0.86	0.81
	US$/t ore	3.22	3.20
Dewatering	US$	1,866,933	2,559,527
	US$/$m^3$	0.50	0.68
	US$/t ore	1.87	2.69
Roads & Dumps	US$	2,248,349	1,503,977
	US$/$m^3$	0.60	0.40
	US$/t ore	2.25	1.58
TOTAL Operating Costs	**US$**	**15,505,579**	**14,404,935**
	US$/$m^3$	**4.13**	**3.84**
	US$/t ore	**15.51**	**15.15**
Indirect costs	US$	4,890,933	3,458,551
	US$/$m^3$	1.30	0.92
General & Administrative Costs	US$	5,845,545	1,563,851
	US$/$m^3$	1.56	0.42
Total Costs	**US$**	**26,242,057**	**19,427,337**
	US$/$m^3$	**7.00**	**5.18**
	US$/t ore	**26.24**	**20.43**

14.5 Ore Reserves

14.5.1 Introduction

The initial in situ reserve base and the ongoing depletion of reserves have been estimated 'in-house' using Datamine®, the former comprising 4,230kt at a grade of 21.92%.Zn containing 927,034t of zinc metal. The original model is shown in Figure 14.9.

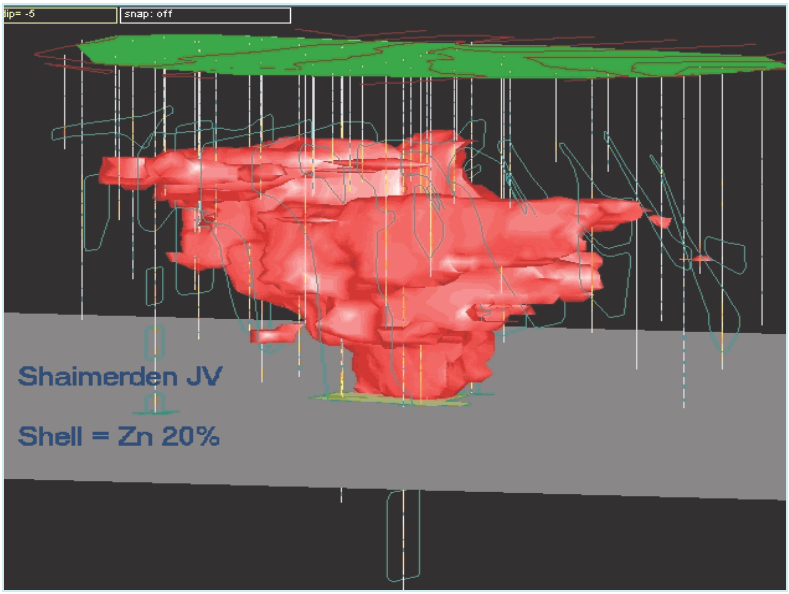


Figure 14.9: Original Datamine Model of the Shaimerden Ore Body

At the time of the WAI visit (12.10.2010) mining was being completed on the 105 Level.

Although the ore body extends to 20 Level the intention is to deepen the pit only as far as 40 level and leave the relatively inaccessible reserves between the 40 and 20 levels.

The blending of different grades to achieve a saleable product of ±21% Zn occurs after crushing and belt sampling in the laboratory.

14.5.1.1 Datamine® Summary of the Creation of the Block Model

Topographic Survey

The initial topography was surveyed in 1958-1959 on a scale of 1:2000 using a plane table in initial reconnaissance stages of the exploration of the KrasnoOktyabrsky bauxite deposit.

Top cutting

Top cutting of high values was not necessary, in the absence of any anomalous values for the components being modelled.

Parameters Used for Reserve Estimation (Conditions)

The 'conditions' constraining the modelling of the ore body were:

- Cut-off grade of zinc - 5%;
- Minimum mining thickness - 2m; and
- Maximum thickness of the waste rock and low grade mineralisation included in the ore body outline - 10 m.

Geostatistics

Reserve calculation was carried out using the inverse square method and therefore no variography was done.

Lithological Coding and Relative Density

Density was determined according to the lithologic code by estimating the relative densities of all lithological varieties of ores, limestones and other surrounding rocks. Relative densities were estimated separately for high and low grade ores and waste rock and for the individual lithotypes within each of these groups. These are summarised in Table 14.14.

Table 14.14: Lithotypes and Relative Densities

Code	Description of Lithotypes	Density
ALL	Chegansky Clays	1.92
BXCL	Bauxitic Clays (lignite/kaolin)	1.89
MCL	Multicoloured Clays	1.89
KCL		
BX	Stony Bauxites	2.30
	Weathering Crust (General)	
FF	Limonite	1.89
CL1	Clay Ores	1.75
	Low Grade	1.65
CL2	Rubble-Clay (10-40%)	2.06
	Rubble Clay (Low Grade)	1.81
CL3	Clay-Rubble Ores (>40%)	2.27
	Clay-Rubble Ores (Low Grade)	1.55
CSM	Stony, Dense, Breccia like Ores	2.35
	Stony Ore (Low Grade)	2.35
STO	Calamine (Smithsonite) Ore	2.56
	Calamine (Smithsonite) Ore (Low Grade)	2.56
SUL	Sulphide Ores	3.63
BXCL	Bauxitic Clays (Rich Ore)	2.30
LST	Limestone-High Grade Ore	2.68
LST	Limestone-Low Grade Ore	2.56
LST	Limestone	2.68
VOL	Volcanics	2.68

Cell Dimensions

The parent cell and sub-cell dimensions used in model are summarised in Table 14.15.

Table 14.15: Block Model Cell Dimensions (m)

Axes	Parent Cell Dimensions	Sub-cell Minimum Dimensions
X	5	1.5
Y	5	1.5
Z	2.5	0.1

Parameters of the search ellipsoid:

- the angles of the volume of the search: SANGLE 1 = 90, SANGLE 2 = 0, SANGLE 3 OF =-30;
- axes of rotation (1=X, 2=Y, 3=Z) SAXIS 1 = 3, SAXIS 2 = 1, SAXIS 3 = 2;
- the axes of the volume of the search: SDIST 1 = 37.5 m, SDIST 2 = 37.5m, SDIST 3 = of 3m;
- minimum samples required - 4, maximum – 20;
- form of the search volume – ellipsoid;
- axis multiplication factor – 2;
- no search on the octants; and
- a maximum of 4 samples per borehole.

Interpolation Parameters

Blocks were not discretised. Interpolation within separate ore bodies was limited to the use of those samples which lie within their outline (contour).

Checks

No checks on the adequacy of the procedures were carried out on the model.

Reserve Classification

The entire reserves of the ore body were classified in the C_1 category.

14.5.1.2 Mined and Processed Ore Stockpiles

A range of zinc grades is specific to each ore stockpile and the stockpile numbers and grade categories are summarised in Table 14.16.

Table 14.16: Stockpile and Dump Grade Summary

Stockpile No	Zn%	Notes
(No 20)	(18.06)	
No 21	>25	
No 22	15 - 25	
No 23	5 - 15	
	2 - 5	Out of balance
Rock	<2	Waste

14.5.2 Combined Pit and Stockpile Reserves (01 01 2011)

The Shaimerden 'reserves' as at 01 01 2011 comprised the following components:

a) Remaining 'in situ' reserves in the pit, and
b) Mined and processed ore including:
   - Stockpiles (ore dumps) classified on the basis of a range of zinc values;
   - 'Zones' consisting of ore stacked to allow draining of the high moisture content;
   - Railway stockpile, and
   - Overburden stock.

The reserves remaining 'in pit' as of 01.01.2011 are 474,420t of ore at a grade of 21.6% Zn containing 102,487t of metal. The remaining resources and reserves of the various stockpiles as at 01.01.2011 are summarised in Table 14.17.

Table 14.17: Stockpile Mineral Resources and Ore Reserves (WAI, 01.01.2011) (In Accordance with the Guidelines of the JORC Code (2004))			
Description	**Ore (kt)**	**Zn %**	**Total metal (t)**
Stockpile Ore *Indicated* Resources	**2,483.86**	**21.71**	**539,143**
Stockpile Ore *Probable* Reserves	**2,483.86**	**21.71**	**539,143**

14.6 Process

Kazzinc LLP purchased the Shaimerden oxide zinc deposit from Irish company ZincOx, in the spring of 2004. The deposit is situated in the territory of Kamistin district about 200km to the SW of the regional centre, Kostanay. In 1998 the approved reserves were 3.5Mt of ore containing 879,400t of zinc at an average zinc grade of 25.11% Zn. Subsequently, a mining and crushing operation was established at the site, with a rail connection to transport the crushed ore to the Ridder Metallurgical complex for processing operations began in 2005.

Ore from the open pit mine is transported to an intermediate ore stockpile using 777D Caterpiller trucks, each carrying approximately 96t each, where it is stockpiled ready for crushing. At the intermediate ore stockyard ore is stacked into heaps with the help of 2x HITACHI ZX 450 loaders in a particular position as to take advantage of the prevailing wind and ambient temperature to reduce the moisture content to a level of 9% as well as to perform ore blending to produce an ore grade in the range 18-21% Zn.

Run of Mine ore of -500+0mm size is crushed in three stages. This crushing cycle of Shaimerden ore operates from May to November 20 hours/day. Ore from intermediate ore stockyard (specially positioned heaps as to take advantage of prevailing wind) is moved to MMD 500 S154-0871 primary crusher (manufactured in England) with the help of CAT 988G loader. Final product size after primary crushing is -100+0mm.

A MMD 500 S154-0870 crusher (manufactured in England) is used for the secondary crushing to produce the final -40 +0mm crushed product. A Chinese 2DSKP 75200 crusher is installed for tertiary crushing to produce the final -10+0mm crushed product. In June 2010, a DG 1000 roll crusher manufactured by KazzincMash Ltd was installed in parallel with Chinese 2DSKP 75200 rolls crusher for tertiary crushing. The crushed product size is currently P80 =10mm.

For quality control purposes, before and during crushing, samples for analysis are taken after every 50-100t of ore has passed through the crushers. Sampling is performed with the help of an automatic sampling system and the Zn content of the sample is determined using a SPECTRO XEPOC X-ray spectrometer. Size analysis of the ore is also checked every shift and there are weightometers positioned within the crushing circuit to monitor and record the tonnes that have been crushed.

After crushing, the ore is delivered by conveyor to a stockyard with 8 sections holding 400t each from which the ore is blended in certain proportions using a CAT 988 G loader to achieve the required grade. This finished product is then transported in 25t MAN trucks to the railway siding. Ore is loaded from the intermediate stockyard to a receiving bin of the railway wagon loading complex with the help of a CAT 966 H Wheel Loader. Loaded rail wagons are shipped to Ridder Metallurgical Complex.

Note that the mining rate, currently about 1Mtpa, is not synchronous with the crushing rate, nominally about 330ktpa. The remainder of uncrushed ore is taken up by the stockpiles which have been increasing in size since the start of production to the present 2Mt. The mining operation is scheduled to cease in May 2011, whereas the crushing and delivery of ore will continue for a further 5 years after this.

WAI Comment: *The Shaimerden operation is modern, well managed and well maintained.*

14.7 Environmental

14.7.1 Environmental Setting

Shaimerden is located approximately 200km SSW of the city of Kostanay and approximately 110km southwest of Lisakovsk, in the Kamysty region of the Kostanay District of northern Kazakhstan. The nearest settlements comprise the village of Krasnogorsk, 7km southeast and KrasnoOktyabrsky 14km northeast from the project, whilst the regional centre, Kamysty lies 50km to the west.

There are no permanent rivers in the area, but there are many fresh and saline water lakes, the largest being Lakes Sorkol, Koyandykopa (a land allotment of the project), Sunaly, and Tunkuyukty. The nearest permanent river is the Tobol River which is in 60km north of the project.

The local relief of the project is flat with absolute levels in the area of the deposit in the range of 239-247m, although in some places undulating with lakes, salt pans and occasional gullies.

The vegetation within the vicinity and surrounding area of the project comprises saline steppe habitat and species. This land has been categorized as having no land use value by the state. Such a status means that it is not deemed suitable for any contemporary farming practices; however Red Book (rare) species of fauna are present within this area.

The climate of the region is highly continental, with short, hot dry summers and extended extremely cold winters. Annual precipitation is low with an average of 295mm.

14.7.2 Current Environmental Status

The licence for the right of subsoil use for prospecting and development by the "Shaimerden Joint Venture" is for 25 years and was approved on 04 May 1997. The licence comprises 5 years exploration and 20 years mining. The mining licence covers an area of 3.23km^2, and the total area under licence is 12.1km^2.

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

The land allotments are licensed to the project for 20 years, in consistency with the mining licence period.

The lease area of the neighbouring bauxite operation (KBRU) is adjacent to Shaimerden's mining lease area.

> ***WAI Comment:*** *Although the mining lease and several of the land allotments of the project are contained within the same geographical area as the bauxite operation, the projects operate on an independent basis which works well. The only obligation to which Shaimerden is subject, as per government requirements, is to stockpile any bauxite recovered from the open pit in their lease in a separate stockpile.*

The open pit is mined by the use of blasting and then removal of rock by truck. Waste rock material is taken to rock dumps and ore bearing rock is stockpiled according to grade and then crushed to form a blend with an average grade of 20% zinc. The ore is then taken by trucks to the rail head (spurring to the project from the main line through Krasnogorsk) and transported by rail to the Kazzinc Ridder Metallurgical Complex for further processing. No other processing is undertaken on site; however an assay laboratory to determine rock grades is located adjacent to the crusher.

14.7.3 Review of Environmental/Social Studies

A revised OVOS was issued in 2006 in relation to the Shaimerden project and is valid until the project closes, or there is a significant change in the nature of the operations. The OVOS includes all extraction, processing, ancillary and auxiliary facilities.

> ***WAI Comment:*** *The project is forecast to remain in an extraction phase until 2012, with crushing and transportation activities continuing until 2016, or later by State agreement.*

An environmental monitoring plan to enable the project's compliance with the OVOS is reviewed and updated on an annual bases, and is based on operational changes and project emissions and discharges. It was reported that the 2011 plan has been reviewed and passed by the state.

> ***WAI Comment:*** *Shaimerden is well aware of its obligations with regard to meeting national requirements and is performing the appropriate environmental studies in this regard. WAI was not able to review the complete OVOS document in English; however it is understood to be fully compliant with State requirements. It is suggested that a gap analysis should be performed to assess compliance with international requirements.*

14.7.4 Security

All security for the project is undertaken by an international subcontracted company called Group 4 Security (G4S). G4S provide round the clock surveillance for all of Shaimerden's facilities.

> ***WAI Comment:*** *The project is fragmented into 13 land allotments, therefore it would be impractical to prevent the public accessing the entire area. The security at the project is generally good.*

14.7.5 Environmental Management and Training

Environmental management is undertaken in accordance with the environmental improvement programme, which is stipulated for compliance with the OVOS. The programme is approved annually by the Kostanay Environmental Department. An annual environmental budget is drawn up in accordance with the programme.

All environmental training is undertaken by the personnel department and is tailored to the role of the individual member of personnel.

> ***WAI Comment:*** *Environmental management at the project is compliant with Kazakh requirements. The mine itself has not been accredited under ISO 14001. However, it is subject to the request of the corporate certification. All mining in the OP which affects the environment will be completed in 2011 and mine closure will then occur.*

14.7.6 Social Management and Training

Shaimerden employs 465 personnel, approximately 60 of which are on shift rotation (2 weeks on, 1 week off) from Kostanay (60km east). A number of the technical staff are not from Kazakhstan and have relocated for the project.

Shaimerden has a personnel manager and department. The role of the department is to:

- Source suitable personnel for the project by
  - Advertising employment opportunities
  - Identifying suitable candidates for employment
  - Identifying employed personnel who could be trained into higher positions
  - Providing regular training/refresher programmes to personnel in health safety and environment awareness etc
- Maintain suitable working conditions for personnel
  - Providing and maintaining suitable accommodation, services and facilities for rotational personnel i.e. laundry, television, canteen etc
  - Providing Personal Protective Equipment (PPE)
  - Provision and maintenance of the Medical Station (based in Krasnogorsk), whose roles include

- Testing for alcohol and drug use before shifts
- Providing regular health check to personnel
- Provision of a negotiated social package
- Provision of daily transportation from and to Lisakovsk and Krasnogorsk

- Maintain a balanced working relationship between personnel and managers
  - Providing an interface between managers and personnel
  - Organising meetings between managers and personnel
  - Provision of working contracts
  - Insuring all facets of the project are working within the law
- Maintain a strong, positive relationship with the local population of Krasnogorsk.

WAI Comment: *The personnel department provides a comprehensive, well planned and successful service to employees. This is in compliance with Kazakh and international requirements.*

14.7.7 Environmental Monitoring and Compliance

All environmental monitoring is undertaken in accordance with the requirements of the annual emissions and discharges report in line with the OVOS. The purpose of all environmental monitoring at the project is to ascertain whether the project is complying with specified standards.

Water quality monitoring is undertaken quarterly by Shaimerden internally and includes the following locations:

- 7 boreholes in and the 3 around the open pit;
- Settlement pond;
- Discharge pond (Karakol Lake);
- Monitoring boreholes around the projects sanitary protection zone (allotted as part of the OVOS); and
- Potable water borehole based in Krasnogorsk.

The water samples are sent for analysis at the Kazzinc laboratory based in Ust-Kamenogorsk. The analyses consists of heavy metals, pH, hydrocarbon compounds and explosives residues. The concentrations of contaminants identified in the discharge pond are the basis of the MPC permit issued in the revised OVOS. The environmental manager also undertakes rudimentary calculations to estimate the concentrations of contaminants outside of the sanitary protection zone.

Air quality monitoring is undertaken by the Kostanay Oblast Epidemiological Department on a quarterly basis. The monitoring consists of a number of points around the project and on the boundary of the sanitary protection zone. The monitoring is undertaken by hand held equipment and includes noise, vibration, dust, nitrogen oxide compounds (NOx) and carbon monoxide (CO). These determinands are undertaken to identify risks to human health, and risks to environmental receptors.

Soils and vegetation (plant uptake of contaminants) monitoring is undertaken within the project and along the boundary of the sanitary protection zone. The samples are collected by Shaimerden and sent to an independent laboratory in Kostanay for analysis.

WAI Comment: *An inventory of pollution sources for the operation was produced, and the monitoring programme has been developed in response to this. The monitoring programme has been designed to ensure compliance with Kazakh requirements, and would need to be compared against international standards to assess compliance with these.*

14.7.8 Stakeholder Dialogue and Grievance Mechanisms

The internal dialogue and grievance mechanism is managed and operated through the personnel department.

An annual review and forthcoming operations is provided to the Akim by Shaimerden. The Akim in turn presents this information to the populations of Krasnogorsk and Lisakovsk.

In case of complaints, the local populations are encouraged to communicate with the Akim, who then reports problems to the Personnel Manager or Director of the project. People are able to complain in person; however this has been deemed a more effective way of understanding the population's concerns.

> ***WAI Comment:*** *The internal stakeholder dialogue and grievance mechanism is appropriate to the size and nature of the operations. With regard to external dialogue, whilst national obligations are being met, it is recommended that external stakeholders have further opportunity to inform the company's decision making, to ensure that both company and community aspirations and concerns are satisfied.*

14.7.9 Water Management, Sourcing and Consumption

The abstracted water from the open pit is pumped via a pipeline into an unlined settlement pond to enable the reduction of suspended solids. The settlement pond was once an area of low lying wetland, which has been enclosed at its lower end by a waste rock berm. Once settled, the water is then pumped via a pumping station, adjacent to the settlement pond, to Lake Karakol where it is released. The pumping station has a capacity of 3,330m^3/hr via 2 pumps; a third is on standby. The evaporation potential of Lake Karakol is 25Mm^3/year. Lake Karakol was once a seasonal water body; however it is now a permanent feature as a result of the project.

Chemical analyses is undertaken at each stage of the abstraction process on a quarterly basis (Section 14.7.7), in accordance with the OVOS requirements. Non-soluble explosives are used in the open pit in order to reduce the contamination of groundwater.

Potable water is abstracted from 2 boreholes located in Krasnogorsk. The boreholes supply the village and the potable needs of the project. The water is transported to the project via pipeline to tanks to provide water for washing, and in containers for drinking and cooking purposes. The borehole abstracts from the limestone and is approximately 40m deep.

> ***WAI Comment:*** *The potential impacts on receiving waters from open pit discharge were assessed as part of the OVOS.*

14.7.10 Energy Consumption and Source

The region lacks power resources; coal from Karaganda and Kushmurun, and oil products are brought in and electric power is supplied from the Ural power system lines (Russia) via the Troitsk –Rudny–Lisakovsk–Krasnogorsk 110kv power line to a single substation owned by Shaimerden JSC. Another power supply source comprises three 6kV power lines from the KBRU-owned substation. Shaimerden has a reserve diesel 1MW/hr power generator should the electricity supply fail (the whole project requires approximately 7MW/hr at any one time).

> ***WAI Comment:*** *The substation is well managed and maintained, and energy usage is reviewed as part of the Company-wide resource minimisation scheme. The 5% target is thought to be met in the energy department, apart from with regard to heating.*

14.7.11 Waste Management

Scrap (including empty used oil barrels) is initially stored at the workshops and the settlement pond before being collected and stockpiled at the rail head to be transported to a scrap recycling company (Kozneasen) based in Kostanay. The amount of scrap produced is variable; however 100t was produced in 2009.

Used oil is emptied back into oil barrels and sent to the Ridder project by rail, where it is disposed of by burning.

Waste tyres are stockpiled outside the workshops. The tyres will remain in situ until there is a suitable facility identified for their disposal within a reasonable distance of the project, or until the project is closed. They will then be transported to the closest available facility.

Domestic waste and sewage are disposed of in a location known as the 'filtration field' a land allotment of the project located less than 0.5km and east of the open pit. The filtration field contains an open manmade pond into which sewage, pumped by tanker from sewage collector tanks, is deposited. Whilst still in the sewage tanks, the sewage is treated with chlorine to retard microbiological activity. Once disposed of into the sewage pond, no further treatment is undertaken. The sewage is intentionally allowed to passively drain into groundwater; hence the pond is not lined.

Domestic waste is disposed of in temporary trenches. The trenches are dug to a depth of 3.5m, 5m width and approximately 20m length. They are backfilled approximately every 5 days by the excavated soil arisings. No domestic waste separation is undertaken.

It was reported that no waste is produced by the assay laboratory.

The waste rock (>1% zinc) is stockpiled awaiting reprofiling as part of the project's closure plan.

From mid 2011 operations will cease and full mine closure will be initiated. The current dumping area will be rehabilitated.

14.7.12 Handling and Storage

The whole fuel farm area is concrete bunded below ground level and has soil bunds at surface. Below the rail refuelling point, is a subsurface sump and tank, within a further concrete bund and clay liner. The ground surface has been contoured to enable surface runoff to enter the sump. The total capacity of the subsurface tank is 140,000l. The sump is periodically pumped out to remove precipitation and this is taken to a specialist facility based in the neighbouring KBRU operation. It was reported that Shaimerden has never had a significant incident at the fuel farm.

Petrol is not stored on site and any required (for vehicles) is obtained from a public station in Krasnogorsk.

Oil for use in plant and machinery is brought in via train by barrel and is stored in the maintenance sheds in small quantities (<50 barrels).

The storage of spare machinery is next to fuel farm.

Ore stockpiles are classified into low (5-15%), medium (15-20%) and high (20-25%) zinc grade ore. Each stockpile is numbered. 24 stockpiles exist. The material from the stockpiles is taken by truck to the crusher. The stockpiles are blended as part of the crushing process. Crushing takes place between May and October. The crushed rock is then taken by trucks to the rail head for transportation.

In addition to zinc ore, the open pit also contains some bauxite (aluminium bearing rock). The bauxite is stockpiled separately. 21Mm^3 of bauxite has been mined at Shaimerden to date. The bauxite is given over to the neighbouring bauxite operation KBRU as a result of the Shaimerden operating inside the KBRU's mining licence.

> ***WAI Comment:*** *The project's storage areas are appropriate and well maintained, and there is an observation monitoring well to assess potential contamination in the vicinity of the fuel farm.*

14.7.13 General Housekeeping

General housekeeping at the project is good, with most areas being maintained in a tidy manner.

14.7.14 Fire Safety

Fire fighting equipment is located at various points around site facilities and plant. All staff and contractors are given training in fire fighting protocols. The project maintains a fire fighting team on a 24 hour basis.

> ***WAI Comment:*** *Fire fighting provisions appear appropriate to the size of the operations.*

14.7.15 Health and Safety Management, Training and Emergency Response

The project has a health and safety officer and deputy officer. Health and safety procedures at the project are undertaken in accordance with the state handbook on health and safety. In addition, there are site specific risk assessments for different tasks.

Employees are given training by the personnel department in health and safety procedures in accordance with their role within the project and with regard to emergencies such as fires or spills. All personnel are provided with adequate Personal Protective Equipment (PPE) and its use is enforced.

The management of health and safety includes the production of emergency plans, the inclusion of contractors and exterior personnel into site visit briefings and investigations after accidents have occurred.

Health and safety inspections are undertaken by the Kostanay Oblast Epidemiological Department on a quarterly basis. The inspections include noise, vibration, dust and potable water quality monitoring and reporting.

The stability of the rock dumps is measured via visual inspection by the project's surveyor on a monthly basis and also by instruments on an annual basis.

> ***WAI Comment:*** *Health and safety is fully compliant with state requirements. The site is not separately OHSAS 18001 accredited, but is covered by the corporate OHSAS 18001 accreditation.*

14.7.16 Environmental Liability

The environmental liabilities of the project are relatively minor given that all plant and machinery can be dismantled and removed and that other than crushing, no other mineral processing activities are undertaken on site. The most significant liabilities are the area of waste rock dumps and the open pit.

> ***WAI Comment:*** *The Company considers that the Mine Closure and Rehabilitation Plan (MCRP) addresses the main environmental liabilities posed by the project, and given the imminent closure of the project in 2011, feels these will be adequately addressed. The MCRP design has been submitted to the State authorities for expertise assessment.*

14.7.17 Mine Closure and Rehabilitation

Under State legislation, the local community provisions for mine closure should be made by the State. However, Kazzinc is also considering isues regarding community impacts of closure.

The closure plan for Shaimerden is being designed by a state registered institute concurrent with the writing of this report. Once completed, the report will be submitted to the Ministry of Environmental Protection for review. The closure fund has yet to be determined.

Although the closure plan has yet to be officially completed Shaimerden unofficially has made the following closure plans.

Prior to the construction of the project's facilities, the topsoil from these areas was stockpiled. The topsoil will be replaced once the facilities are dismantled and removed. Remediation will be progressive as facilities become redundant.

The remediation of the waste rock dumps will consist of the reduction in slope angle from 60° to 10°, producing a plateau to the edge of the rock dump land allotment. The area of the rock dumps was stripped of topsoil (36,000m^3) previously and as such the topsoil will be placed over this area and left for passive remediation. The re-profiling of the rock dumps will proceed in 2011 and will continue throughout the remaining life of the project (forecast 2016), subject to the removal of the ore stockpiles within this area (providing the required space).

It is proposed that the haul road to the open pit will be removed. The soil safety bund, currently present around the open pit, will be reinforced with crushed waste rock. The water pipelines will be removed. It is assumed that on closure of the open pit that this will be allowed to flood. It is assumed that the dam in Sorkol Lake will remain in place.

The waste rock berm at the settlement pond will be removed to allow this to once again become an area of wetland.

The local administration is in the town of Lisakovsk.

> ***WAI Comment:*** *The MCRP has been prepared by a local environmental contractor, and is currently undergoing environmental expertise review by the government. Approval is expected in April 2011. It is recommended that Kazzinc's community closure initiatives should be incorporated in a formal plan, with a supporting fund. The fund should be adequate to cover both environmental and community aspects of closure, including post closure monitoring.*

14.7.18 Social Initiatives and Community Development

Prior to the operation of Shaimerden by Kazzinc, the village of Krasnogorsk was failing. The village had no sewage system and an unreliable water supply, as well as high and increasing rates of unemployment.

The project has provided Krasnogorsk with street lighting, a sewage system, potable water, funding and equipment to local schools, employment to a large proportion to the viable population, funded the redevelopment of many of the town's buildings, replaced heating systems in houses and public places, provided machinery, roads, wage supplements to those employees with children and low incomes, vaccinations for children, an ambulance service and a medical station (staffed by nurses), amongst other initiatives.

The potential health impacts of the project on the local community have not been assessed by Shaimerden. However there have been no known illnesses reported as a result of the project.

> ***WAI Comment:*** *Shaimerden is working well with the local community, and has provided much needed and required assistance to it. Kazzinc always renders assistance to Krasnogorsk village. The following works are performed: cleaning of roads, school maintenance, purchasing of equipment and clothes for orphan homes and boarding schools of Lisakovsk town which is in compliance with social business responsibility. It would also be recommended to focus on skills provision to enable local business development in other sectors (outside of mining).*
>
> *WAI recommends carrying out a local needs assessment which can be used to inform a social development plan and initiatives.*

15 NOVOSHIROKINSKOYE DEPOSIT

15.1 Introduction

15.1.1 Location & Access

The Novoshirokinskoye gold-polymetallic mine is located in East Zabaykal in the Gazimuro region of Chita province, Russian Federation. The geographical coordinates of the centre of the licence area are 51°34´45"N and 118°42´15"E.

The deposit is located some 23km to the northeast of the rayon centre of Gazimurskiy village, 191km southeast of the railway station at Sretensk, and some 288km to the southwest of the railway station at Borzya (Figure 15.1).

Transport to the mine is via a 10km long un-metalled road which joins the Sretensk - Nerchinsk provincial highway. The total distance to the railway siding at Priiskovaya where concentrates are loaded into railcars for their transport to Ust-Kamenogorsk is some 230km.

Figure 15.1: Location of Novoshirokinskoye, Russian Federation

Travel time by car from Chita to the northwest, which is served by daily flights to Moscow, is between 7 – 8 hours, a distance of over 500km. The road conditions are generally good, particularly in the first half of the journey from Chita.

The deposit is bordered by China to the east (some 70km in a straight line) and Mongolia to the south.

15.1.2 Topography & Climate

The Gazimuro region is home to basic agriculture concentrated in the main valleys, whilst the majority of the land is typical boreal birch and pine forest.

The Novoshirokinskoye mine lies at an altitude of around 935m between the Gazimura and Uryumkana rivers. Relief is generally moderate, although there are some steeper slopes. Within the vicinity of the mine, absolute elevations vary from 700-1,200m, with relative changes of between 100-300m.

The climate of the region is sharply continental, and is characterised by a prolonged (5½ to 6 months) of winter with frosts as low as -60°C, and by a comparatively short summer with a maximum temperature of +39°C. The average annual temperature is -4°C, which has caused the development of localised permafrost on the northern slopes and in the valleys.

The annual average precipitation is 392mm, with approximately 85% (333mm) occurring during the short summer months (July and August), whilst the remainder falls as snow in winter (heaviest in January and February). Groundwater levels around the mine are quite low, and the mine only makes around 50m^3/hr of water at the current 750m depth.

Snow cover varies from 2-19cm, but can reach 33cm in extreme cases. First snows are evident in October and remain until March, sometimes April. In winter, all rivers, except the Gazimura and Uryumkana, are frozen throughout.

Winds are at their strongest in spring and in autumn, predominantly from a southeast and southwesterly direction.

The dryness of the air is a special feature of the climate of the Transbaykal. The average monthly relative humidity of air is fairly consistent, although the lowest relative humidity is seen in May (52%), whilst during the rest of the year it varies from 58 to 80% with an annual average value of 70%.

15.1.3 Infrastructure

With the recent opening of the mine (December 2009), infrastructure in the region has greatly improved, as without mining, there is little else in the vicinity of the mine with the exception of basic agriculture. The electric power supply for the mine comes from the Chita power system via a 110KVa line.

Although limited, the immediate region does produce building materials in the form of clay bricks, limestones, rubble stone, gravel, sand, and also timber for construction and firewood.

15.1.4 Mineral Rights & Permitting

The Novoshirokinskoye mine is surrounded by a 1.4km^2 Mining Licence which is granted for extraction of polymetallic ores to the 600m elevation, but with a larger Land Allotment to accommodate mining infrastructure and tailings surrounding the licence (Table 15.1).

Table 15.1: Mining Licence Coordinates

Coordinate	Easting	Northing
1	118° 41' 25"	51° 36' 20"
2	118° 42' 10"	51° 36' 00"
3	118° 43' 10"	51° 35' 50"
4	118° 43' 05"	51° 35' 35"
5	118° 42' 10"	51° 35' 40"
6	118° 41' 15"	51° 36' 15"

The licence was granted on 30th September 2004 for a period of 20 years, expiring 1st October 2024. The Novoshirokinskoye licence is held by a Russian legal entity, OAO Novoshirokinskiy Rudnik and Kazzinc owns 48.3% of the shares in OAO Novoshirokinskiy Rudnik.

> ***WAI Comment:*** *The licence documentation has been inspected and is in order. Moreover, the conditions of the licence are sufficient for the life of mine.*

15.2 Geology & Mineralisation

15.2.1 Regional Geology

15.2.1.1 Introduction

The area of the Novoshirokinskoye ore field is composed of sedimentary and volcanogenic-sedimentary formations of early-Middle Cambrian, Early Middle Jurassic, and Late Jurassic ages, with Upper Jurassic intrusive rocks.

15.2.1.2 Stratigraphy

The oldest rocks present in the area are those of a Cambrian age (Cm1-2al) sedimentary package (Altachinskaya series), comprising predominatly micaceous siltstones with interlayers of sandstones and quartzites. In the upper part, siltstones and schists, with interlayers of limestone and quartzites predominate.

These rocks are seen in the northwest part of the area, and are bounded to the southeast by the Uryumkanskim Fault.

Above the Cambrian lie rocks of Jurassic age. The Lower-Middle Jurassic (J1-2) sediments occupy a large portion of the southeastern part of the Novoshirokinskoye area, with less exposure in the west, where they are faulted against the Cambrian rocks of the Altachinskaya series. In the east, they overlap with Upper Jurassic sediments.

Compositionally, the Lower-Middle Jurassic is characterised by polymict sandstones with lens-like siltstones occurring in the upper parts, thin clay shales (1-2m thick), and localised conglomerates between 5-10m thick.

The Upper Jurassic (Shadoronskaya series) is composed of a volcano-sedimentary sequence (J3šd) well developed in the central part of the Novoshirokinskoye area where they form a large synclinal structure, elongated in a northeasterly direction. In detail, the series comprises a lower sequence (J3šd1) of tuffaceous sandstones, siltstones, conglomerates, tuff breccias, and thin andesite porphyries. This sequence is approximately 700m thick and attains 1,400m width in the central part of the area.

Above this, the middle sequence (J3šd2) is divided into a lower unit (J3šd21) which comprises andesite and andesite-basalt porphyries, interbedded lava-breccias, tuffs, tuffites, and tuff breccias of those. The thickness of this lower layer is 800m.

Quaternary sediments (Q) of the region include sands, sandy clays, clays, silts, and pebbly gravels in the valleys and terraces, which have thickness from 5 to 30m. Recent, thin (0.1-5m), eluvial-deluvial sediments, which occur on mountain slopes, mostly comprise rudaceous debris, with sand-clay infill. Figure 15.2 shows the regional geology of the Novoshirokinskoye area and the main stratigraphic units.

15.2.2 Local Geology

As outlined above, the Novoshirokinskoye area is characterised by a sequence of Jurassic sediments folded into a synclinal structure with variable dip, but most commonly 45-55° and 30-40°. Sediments are cut by many faults with hydrothermal alteration and brecciation associated with their development. Four main fault directions have been identified: Northeasterly, Northwesterly, North-South, and East-West. The Northwesterly and East-West trending faults can be traced for up to 6km.

The principal magmatic activity within the area is represented by a series of Late Jurassic intrusions comprising granodiorite stocks and dykes which are sometimes, but not always, related to the steeply dipping zones of hydrothermal alteration which host the mineralisation. It is likely that these intrusives, and particularly a small stock adjacent to the main area of mineralisation may well be the driving engine for the hydrothermal activity and subsequent metal emplacement.

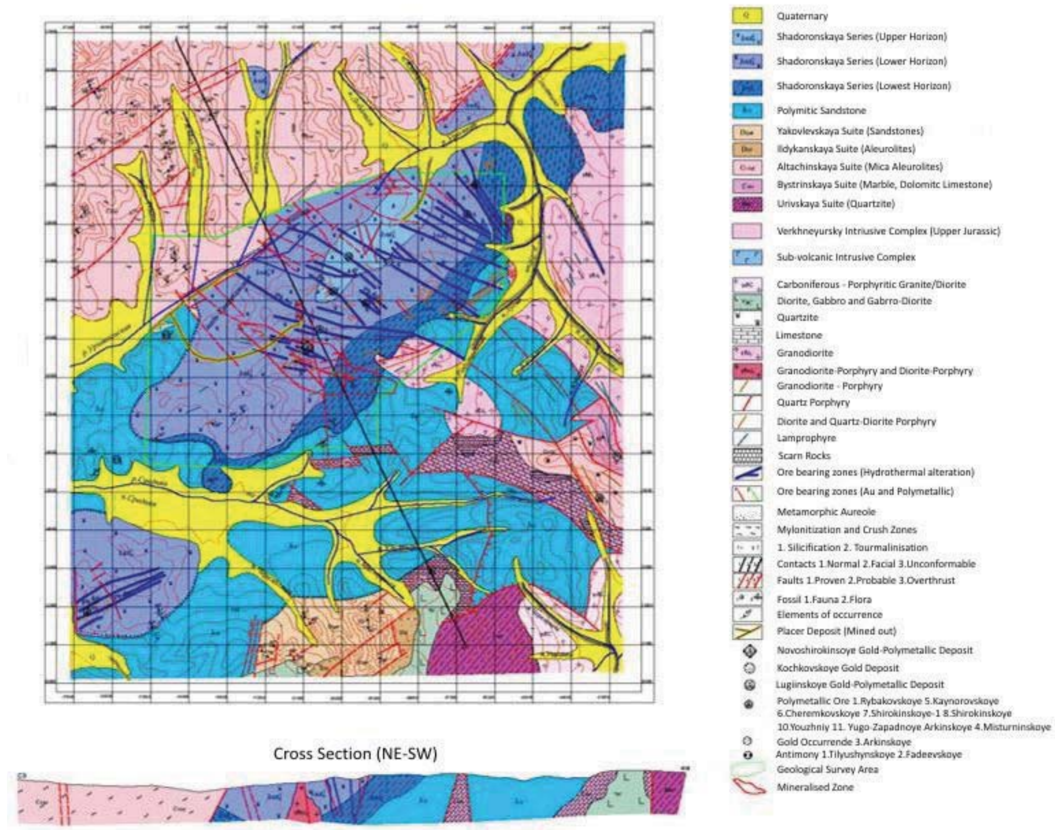


Figure 15.2: Regional Geology of the Novoshirokinskoye Area

15.2.3 Mineralisation

15.2.3.1 Introduction

Economic mineralisation at Novoshirokinskoye is constrained within broader zones of hydrothermal alteration, which in turn are related to several fracturing systems. The ore-bearing phases may be later than the principal hydrothermal phase – the fluids using these zones as pathways for migration and precipitation.

Mineralisation within the alteration zones is seen as lensoid and can be continuous over several hundred metres along strike and has been traced to over 900m in depth where the structure remains open in some parts of the system.

Orebodies are generally oxidised in their upper parts (where close to surface), followed by a thin transition zone grading into sulphide where the predominant economic mineralisation comprises galena and sphalerite with associated gold and silver.

The Main ore zone is the most persistent, and in the central part, 16 orebodies have been identified. In the footwall of the Main orebody are found orebodies 5, 11, 7, 4, 3, 2, 1, 13, 14 and 16, whilst in the hangingwall 8, 6 and 12 are located.

15.2.3.2 Orebody Morphology

The Novoshirokinskoye tectonic zone is characterised by strong metasomatism of quartz-micaceous-dolomite composition. The width of the metasomatic zone varies from 20-300m, and has been traced down-dip for more than 750m. Ore mineralisation is concentrated in the centre of the zone.

The footprint of the mineralisation at surface and on the 853m level is practically identical and comprises 2,130m and 2,150m of strike length respectively. With depth, on the 750m and 650m levels, due to some thinning on the southwest flank, strike extent decreases 1,900-1,960m, whist at 550m this decreases further to 1,780m, at 450m to 1,150m, and at 350m some 1,060m.

Figure 15.3 shows a typical section through the mineralisation.

The internal structure of the mineralised zone is very complex, and is seen as a series of en echelon-like fractures, with mineralisation concentrated within them and disseminated within the surrounding metasomatised envelope.

Disseminated mineralisation predominates which is developed along the systems of small fractures, brecciated mineralisation is also common, as is massive ore which is found within the breccias and less commonly symmetrically-banded and crustiform textures which are characteristic for mineralisation filling cavities.

The disseminated ore types can only really be defined by assay, whilst the breccias and massive ores show more definitive, visual mineralisation.

Overall, the Novoshirokinskoye zone can be considered a linear stockwork with a strike varying from 255-335°, and dipping generally to the southwest at angles from 30-90°. The zone extends for 50-1,450m along strike, and from 40-760m (Main orebody) down-dip, and economic mineralisation has been proven down to the 450m level. A likely finite depth to mineralisation exists at around the 200m level, where the change of volcanic rock favourable to mineralisation by unfavourable pyroclastic and fragmentals occur.

Deep drilling has also revealed a "blind" granodiorite stock of Late Jurassic age in which although orebodies are outlined, they appear to be thinner and branching.

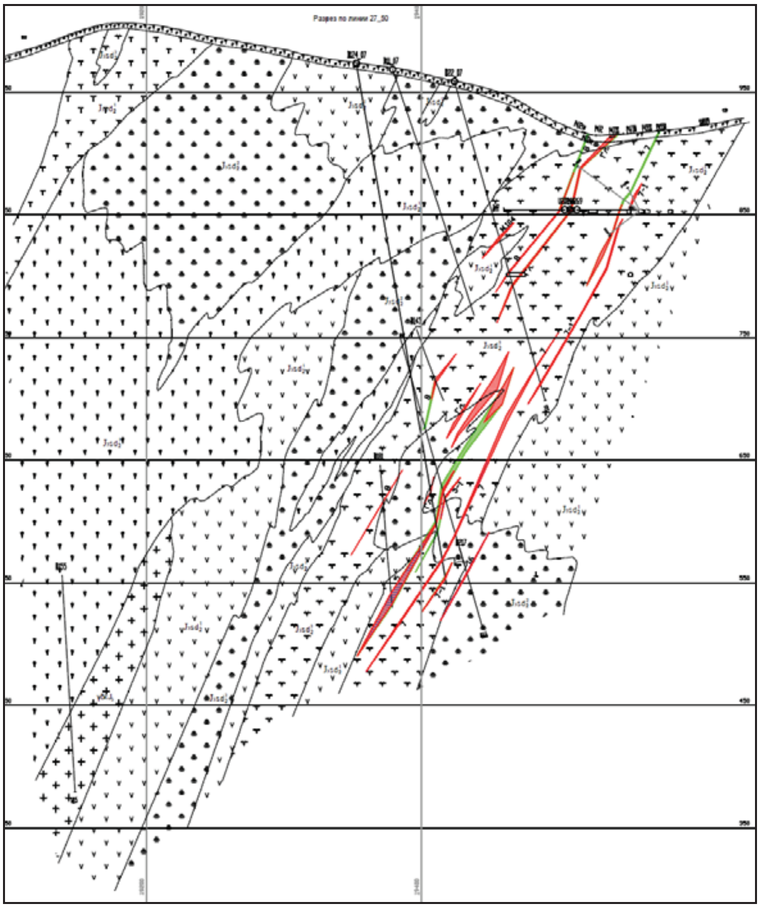


Figure 15.3: Section 27+50 through Main Zone Mineralisation

15.2.3.3 Orebodies

Introduction

The principal orebodies within the Novoshirokinskoye mineralised area are Main, 5 and 7. The details of these are given below.

Main Orebody

The Main orebody typically pinches and swells (and nearly merges with Orebody 5 on section line 95), but in the centre is persistent over considerable distance, with an overall length of 1,330m, maximum down-dip depth of 760m, strike of 290-300°, and dip of between 33-82°. Thickness varies from 0.20 to 22.1m. The thicker areas are often related to breccias (tectonic) zones.

At a 3g/t Au equivalent cut-off grade (calculated in 2007 – see below), the average thickness is 4.52m. On the surface, the length of the orebody is 915m; at 853m it reaches a maximum of 1,330m, whilst at horizons 750m and 550m, the strike length decreases to 1,020 and 1,040m respectively. At these lower horizons, barren areas create a column-like appearance to the ore zone. The most extensive of them occurs at the southeastern end where mineralisation has been traced to the 350m level.

The orebody is folded within the metasomatic envelope of predominantly quartz-dolomite composition with the dissemination, veins, and groups of veins of pyrite, galena and sphalerite.

Orebody 7

This orebody is the second largest and occurs in the central and southeastern part of the deposit on the footwall of the Main and Orebody 5. The orebody thins to the southeast and northwest both along strike and down-dip. At the surface, the length of the orebody is some 480m, whereas at the 700m level, strike length approaches 1,510m, with a maximum down-dip extension of 460.

Morphologically, Orebody 7 is column-like with a "blind" offshoot in the region of profiles 16-27 with a length of up to 600m. In general, Orebody 7 has relatively consistant strike and dip exemplified on the 650m level where the strike only varies between 280-290°, although there are occasional abrupt changes.

However, the general dip is to the southwest, varying from 54-80°, and averaging 65°. Thickness varies from 0.26-15m, with the average thickness at a 3g/t Au equivalent cut-off grade is 2.73m. Mineralisation comprises an uneven dissemination, and by the veins of pyrite, galena and sphalerite.

Orebody 5

Orebody 5, which is the third most important orebody at Novoshirokinskoye, lies in the central part of the area, in the footwall of the Main orebody, some 5-50m away from it, although sometimes merging, although down dip, the orebodies become more seperate.

The maximum strike length (270-335° azimuth) of Orebody 5 is some 1,060m, and has been traced down-dip for some 775m. Thickness is highly variable from 0.19 – 18.15m, with an average of 3.04m using the 3g/t Au equivalent cut-off grade.

Orebody 5 is morphologically quite complex, varying in dip (steeper on the flanks than in the centre), but with an average of 61°, and structure.

Mineralogically, the mineralisation is characterised by a pyritised metasomatic zone of chlorite-dolomite and quartz-chlorite-dolomite composition, predominantly breccia-like in texture. In the majority of cases, mineralisation is related to the quartz veins and accumulations of dolomite and quartz. Principal ore-bearing minerals are galena and shalerite.

Orebody 8

Orebody 8 lies in the hangingwall of the Main orebody at a distance from 10 to 30m from it. Broadly, the orebody structurally mimics the Main orebody, generally dips to the southeast at between 79-80°, but overall at 65°. Localised sharp bends in strike are noted.

Overall strike length is around 945m, with a delineated down-dip length of 460m. Thickness varies from 0.22-12.66m, average 3.21m at a 3g/t Au equivalent cut-off grade.

Other

A number of other ore zones occur within the Novoshirokinskoye area. Zones vary in width from 0.2-17.24m, with averages between 6.25-11.8m. Strike lengths can be >500m, but down-dip depths are around 100m.

In addition, several smaller ore zones, localised in the hangingwall of the Main orebody, also occur, dipping to the southeast at 40-74°. Zones tend to be approximately 100m along strike and 100m down-dip.

15.2.3.4 Mineralogy

More than 60 ore and vein minerals have been identified at Novoshirokinskoye. Principal mineralogy comprises galena, pyrite and sphalerite, with hydroxides of iron and manganese, lead ochres, quartz, dolomite and sericite. The typical average grades are 3.72% lead, 1.79% zinc, 3.51g/t Au and 96.91g/t Ag.

Secondary minerals include chalcopyrite, hematite, covellite, malachite, smithsonite, cerussite and calcite.

Three types of mineralisation have been identified:

- copper-pyrite;
- quartz-polymetallic; and
- carbonate-polymetallic.

There does appear to be some zonation of the ore types, with at the northwestern end, carbonate-polymetallic ores predominating, in the centre, quartz-polymetallic, whilst at the southeastern end, the copper-pyrite ores are most common.

The most significant gold contents are found in the copper-pyrite stage of mineralisation, where values up to 186.1g/t have been found. In the ores of the quartz-polymetallic stage, significant quantities of gold are also encountered where the gold is found in nearly all sulphides, vein quartz and the hematite. The ores of the carbonate-polymetallic stage are the least gold-bearing, with gold associated almost exclusively with galena.

Economic quantities of silver in the form of isomorphous admixtures within galena and fahl ore is seen, the latter is capable of accumulating silver in significant quantities. In addition, up to x10 more silver reports to the copper concentrate than the lead concentrate, whilst comparatively small quantities of silver are connected with virgin gold.

The bulk of the zinc is isolated in the quartz-polymetallic stage of mineralisation, and its content somewhat decreases with depth. The majority of the copper is found in the cupper-pyrite stage of mineralisation, whilst a smaller quantity is found in the quartz-polymetallic stage. In the ores of the latest, carbonate-polymetallic stage, the content of copper is very insignificant.

Alteration zones are quite common, but unless ore-bearing, are mostly pyrite with very low galena and sphalerite.

The ores of the deposit are divided into two types: sulphide and mixed (oxidised-sulphide). The oxidation zone is weakly developed with an average depth of 18m, whilst the mixed ores do not exceed 2% of the total.

15.2.3.5 Exploration Potential

There are two properties nearby, not owned by the Company, one within 1km (Kochkovka), another within 5km (Arkya). These may be available for acquisition, but they are believed to be relatively small.

15.3 Exploration Works

15.3.1 Background

The Novoshirokinskoye mine has had a long history of exploration before production finally commenced late in 2009. Work commenced in 1960 and followed a series of specified protocols which has continued to the present day.

15.3.2 Historical Works

Table 15.2 summarises data on exploration undertaken at Novoshirokinskoye from 1960 – 2010.

Table 15.2: Historical Exploration Works

Description	Units	Total Works	Until 1962	1962 – 1/7/2010
Geological Survey Work at 1:10,000 Scale	km^2		50	
Geological Survey Work at 1:2,000 Scale	km^2	?		
Geophysical Studies (Magnetic, Electrical)	km^2	55	55	
Hydrogeological Survey	km^2	150	150	
Metallometric Survey	km^2	20	20	
Trenches	m^3	73,725	73,725	
Pitting	m	9,282	9,282	
Pit Samples of 1.8m^2 Area	m	3,029	3,029	
Surface Channel Samples	m	4,128	4,128	
Surface Drillholes	m	95,660	73,630	22,030
Core Samples	m	36,998	20,291	16,706
Borehole Logging	%		50	
Hydrogeological Drillholes	No.	14	10	4
Pump Tests	No.	31	31	
Technological Sample Tests	No.	15	6	9
Underground Exploration Development	m	2,960.8	1,284	1,676.8
Underground Mine Development	m	8,037.5		8,037.5
Production Development	m	7,090.5		7,090.5
Stopes	m	2,249		2,249
Underground Channel Samples	No.	3,323	1,299	2,024
Underground Drilling	m	24,309	1,385	22,924
Core Samples	No.	13,786	781	13,005

15.3.3 Sample Collection

From the Russian protocols, the original surface core drilling, the majority of which took place before 1962, sent half cores for assay, whilst underground drilling, due to the small size of the core, sent whole cores for assay.

Due to the very historic nature of much of the exploration at Novoshirokinskoye, unfortunately none of the original drill cores were available for inspection.

Samples were often 1m in length, but varied from 0.25-2.2m. For the underground channel samples with dimensions of 10x5cm, which were collected along all development drives and cross-cuts (often on both sides of a cross-cut), sample lengths varied from 0.2-2.0m, with an average of 1m.

In addition, as part of the mine's development, a number of Technological Samples were collected. These included 11 lab samples with weights from 200-2,000kg, and 5 semi-industrial samples from 5-1,100t.

15.3.4 Sample Preparation

The sample preparation flowsheets for the core and channel samples differed slightly, however, as with all Russian sample preparation for assay, the ultimate grain size for the final sample split for both methods was 0.074mm.

> ***WAI Comment:*** *Although it was not possible to review the historic methods used, from experience of auditing many former Soviet operations, sample preparation is done in a very prescriptive manner and to a consistent good standard. In that respect, WAI is satisfied that these works will have been done properly.*

15.3.5 Sample Analysis

The early exploration works saw some 3,500 channel and core samples sent to the Chita Central Laboratory (1952-57) for lead, zinc and some copper analysis, as well as 17,800 fire assays for gold and silver, and some 23,000 tests done under an umbrella group of laboratories. In addition, analyses of copper, sulphur, cadmium were executed on 110 group tests.

The determination of the content of lead and zinc was done using a classical volumetric chromate method in the tests on samples with a lead content of >2%.

ICP analysis was undertaken at "Alex Stewart" (Moscow), "SGS" (Chita) and the Transbaikal scientific research institute (Chita) from 2007 until 2009. In 2010, analyses were undertaken at the mine laboratory.

15.3.6 Quality Control

As with all deposits explored under the regime of the Former Soviet Union, the level of sample quality control which was undertaken followed strict GKZ protocols, with detailed internal and external control on assaying. Novoshirokinskoye is no exception to this.

The internal and external control of analyses was conducted for the purpose of the detection of random and systematic errors within chemical and fire assays during the entire period of the exploration works.

For internal checks, some 6.6% of samples were re-run for lead and 6.1% for zinc at the Chita laboratory, and some 3.8% and 3.4% respectively at the group laboratories. The test laboratory also controlled some 1.7% of analyses for gold and silver.

Rather than present the voluminous data that relates to this topic, WAI has undertaken a thorough inspection of these data and can conclude that with the exception of occasional discrepancies (which were always rectified at the time), the quality of the data can be deemed to be good and fit for use in any resource estimation exercise.

However, what is clearly evident from these data, and particularly from an analysis of channel sample "twins" is that there is a high degree of variability within the gold values seen.

In isolation this may have proved a problem, although the issue is somewhat ameliorated by the sheer volume of available data giving the reader comfort as to the approximate overall grade of the deposit. Moreover, recent production data has borne out the grades estimated from the historic exploration programme.

15.3.7 Current Exploration & Grade Control

Exploration is now undertaken using channel samples and underground Diamec 251 and 252 rigs, producing a small diameter 32mm core.

Fans of holes are drilled to achieve a 12x12m orebody pierce-points, with whole core sent for assay. Sludge drilling takes this spacing down further to 6 x 6m.

Channel samples are taken approximately every 3m along strike drives and faces, and occasionally as duplicates on either side of x-cuts.

These data are used to update the model and modify the underground development programme.

All exploration assays are currently done in the laboratory on site, although during the time of the visit, there were major sample preparation bottlenecks.

As an example, the mine produces some 200 channel samples (weighing approximately 15kg each), 600 drill chip samples (20kg each), and 1,000 cores (1-4kg each) per month. These relatively large samples make the sample preparation process very slow.

As a consequence, there is an estimated 8,000 sample back-log in the laboratory caused by insufficient and poorly utilised equipment.

Of these, some 3,500 are ground, whilst the remainder are un-prepared. However, the sample preparation facility currently can handle only 44 samples per day.

> ***WAI Comment:*** *The back-log of unassayed samples within the sample preparation facility basically renders the current grade control measures undertaken at the mine obsolete as the mine will have moved on before the assays are known for a particular stope or development drive (WAI was informed that the laboratory runs the grade control holes as a priority, although this was not confirmed). Nevertheless, the laboratory is meant to handle some 400 samples per day, but is currently only doing some 240. The issue has been raised to shareholders, but no action has yet been taken.*
>
> *If the mine has been working efficiently without the benefit of good grade control data, the volumes of samples needed for this exercise may not be necessary, given the lack of blending and generally relatively wide stopes Suffering slightly more dilution might be preferable to the collection and analysis of so many control samples.*

15.4 Mineral Resources

15.4.1 Introduction

Resources were first approved in 1967 by GKZ at B + C_1 for both mixed oxide (near surface) and the larger sulphide resource at depth. These were updated in 2010 (shown on Form 5GR) to indicate the minor production that had taken place.

In November 2010, WAI produced a new resource and reserve estimate, prepared in accordance with the guidleines of the JORC Code (2004).

15.4.2 Density Determination

The volumetric mass of the sulphide ores was determined at a laboratory of construction materials in Irkutsk (Oblpromstroma) and gave a value of 3.10t/m^3.

The volumetric mass of the mixed ores was determined by the laboratory "MITsMIZa", and gave a value of 2.89t/m^3.

Additionally, the calculation of the volumetric mass is carried out statistically using the correlation of the contents of elements and volumetric mass of the existing models, in which were determined the contents of Pb, Zn, Au and Ag.

In the TEO, for the calculation of reserves, an equation for the regression between the volumetric mass (Y) and the content of lead:

$Y = 0.0249Pb + 2.9016$

15.4.3 WAI 2010 Mineral Resource Estimate

15.4.3.1 Introduction

Resources using three-dimensional geostatistical block models were initially estimated in 2008 which included all ore bodies within the deposit. In 2010 Resources were updated for five of the ore bodies (Main, 5, 7, 8 and 16).

15.4.3.2 Database Compilation

Drilling

Copies of the drillhole databases were provided to WAI in Excel format. The database included 563 holes drilled since 2005 and 261 holes drilled prior to 2005.

The density of drilling is greatest in the central part of the deposit where drilling includes underground fan drilling from three horizons (+930m, +850m and +800m Levels). A total of 117,083m of diamond drilling is now included in the drillhole database and includes 35,868 samples with Au assays, 34,566 samples with Ag assays, 25,414 samples with assays for Pb, and 42,885 samples with Zn assays.

The location of all drillholes at Novoshirokinskoye are shown in Figure 15.4.

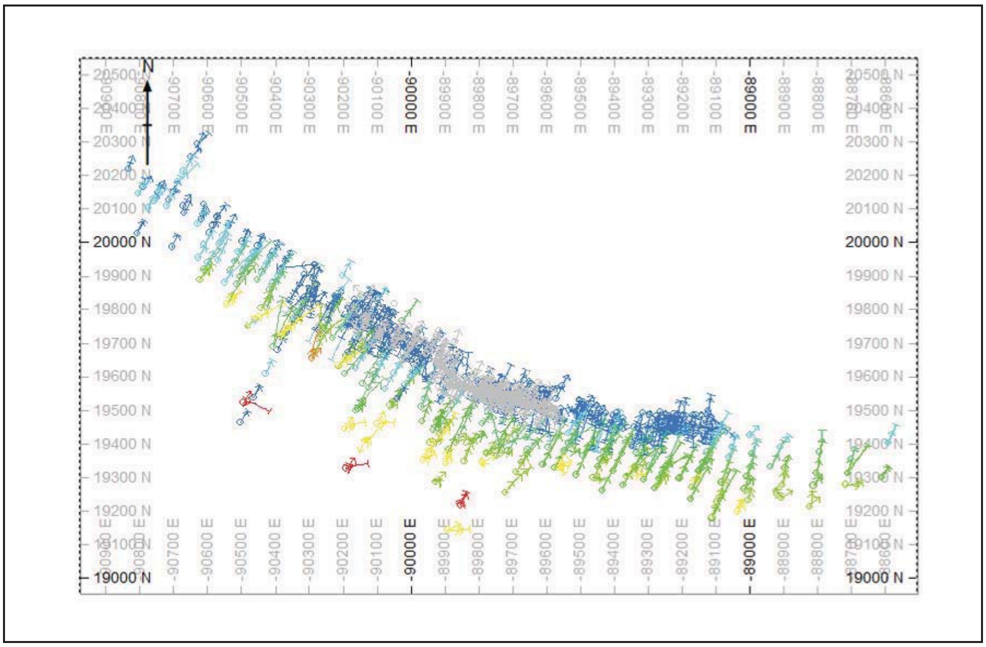


Figure 15.4: Novoshirinskoye Drillhole Locations

Channel Sampling

Channel sampling has been carried out at the Novoshirokinskoye deposit from the underground drives. A total of 7,476 channel samples with Au, Ag, Pb and Zn assays are included in the database.

15.4.3.3 Geological Interpretation

Introduction

Wireframes are constructed based on a 3g/t gold metal equivalent (Aueq) the prices and recoveries used in constructing the wireframes is detailed in Table 15.3. The prices used were those of 2007, the markets have seen significant changes since 2007 and as such metal prices have gone up significantly thus potentially impacting the geological interpretation. As such WAI undertook a sensitivity analysis of changing metal prices and cut-off grades (COG).

Table 15.3: GKZ Novoshirokinskoye Au(eq) 2007

Metal	cost per unit (LME) US$	price units	Convert Unit	Price per Unit, US$	Recovery	Co-Ef
Au	603.46	oz (troy)	1g	19.4	0.92	1
Ag	11.54	oz (troy)	1g	0.37	0.887	0.0184
Pb	1288	t	1%	12.88	0.875	0.632
Zn	1637	t	1%	16.37	0.787	0.722

Sensitivity Analysis

A sensitivity analysis was undertaken using metal prices taken from averages for the next 5 years, the details of which are shown in Table 15.4. The price of gold has increased disproportionately to the other metals and as such the new metal equivalent grades tend to be reduced slightly. Due to the increase in metal prices, a COG for the wireframes of 3g/t Aueq is too high and as such it is deemed a lower COG of 2g/t Aueq would be more feasible.

Table 15.4: Novoshirokinskoye Au(eq) 2010

Cost per Unit (LME) US$	Price Units	Convert Unit	Price per Unit, US$	Recovery	Co-Ef
1287	oz (troy)	1g	41.37865	0.92	1
23	oz (troy)	1g	0.739479	0.887	0.01723
2422	t	1%	22.21	0.875	0.510496
2420	t	1%	20.8	0.787	0.430005

Visual checks were done on sections looking at the wireframes based on the original 2007 3g/t Aueq alongside the drillholes coded with the new metal equivalent.

An analysis was also undertaken in Excel whereby the drillhole orebody intercepts were coded and cross-checked with the new metal equivalent grades.

As a result of both of the checks, it was identified that changing metal prices would have no significant effect on the geological interpretation and wireframing. Due to the dip of the orebodies and the dip of the drillholes the changes in metal equivalent would potentially add less than half a metre in actual width to an orebody and this is only in limited sections.

Therefore, for reporting continuity, WAI has continued with the 3g/t Aueq for modelling purposes. Figure 15.5 shows wireframes in plan for Novoshirokinskoye, whilst Figure 15.6 shows a typical cross-sectional view through the deposit.

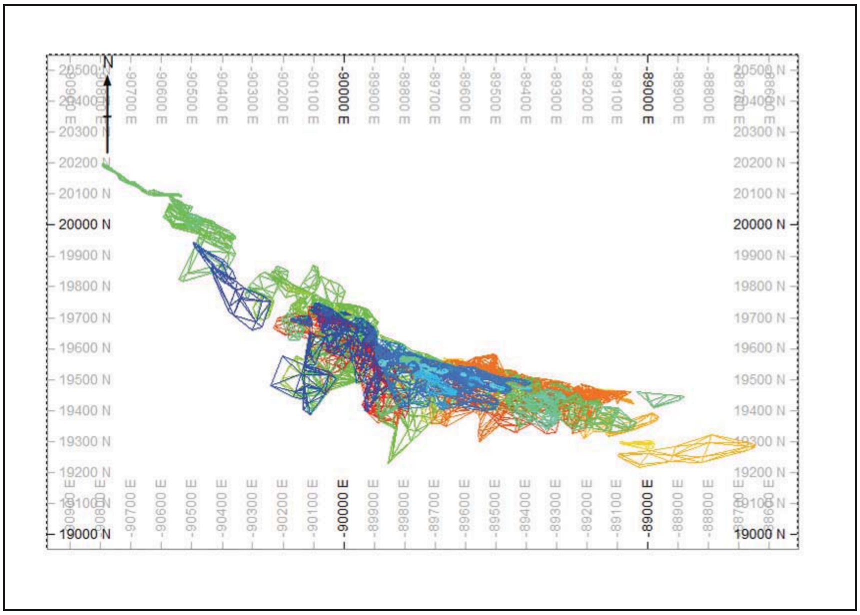


Figure 15.5: Plan View of Novoshirokinskoye Orebody Wireframes

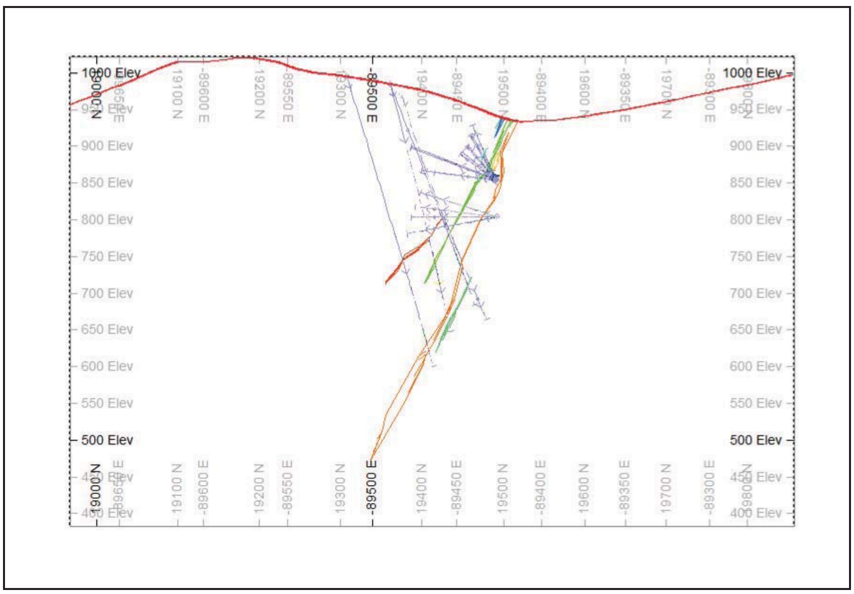


Figure 15.6: Cross Sectional View of Deposit

15.4.3.4 Sample Data Processing

The drillhole samples contained within the orebody wireframes were selected for further data processing. The samples were coded according to each orebody. Gold data show strong log normality.

Top cutting was applied to Au, Pb, Zn and Ag, with the value validated by using decile analysis. The top-cut values used were:

Pb: 16.6%;
Zn: 6.82%;
Au: 22.62g/t; and
Ag: 360.43g/t.

It has been assumed that drillhole sample intervals where assay values are absent were not sampled as they were not considered to contain mineralisation. Assay values which are absent have therefore been replaced with zero values.

A 1m composite length was used to provide a consistent level of support. WAI checked this and agrees that a 1m composite interval is suitable.

The statistical analysis of the Novoshirokinskoye sample (incl Channel samples) database is summarised below:

- Chip samples have been included in the resource estimate;
- Top-cutting was applied; and
- A 1m composite interval has been applied to standardise sample length.

15.4.3.5 Variography

Introduction

WAI undertook variography to review the estimation and search parameters that had been used as well as:

- To estimate the presence of anisotropy in the deposit;
- To derive the spatial continuity of mineralisation along the principal main anisotropic orientations;
- To produce suitable variogram model parameters for use in geostatistical grade interpolation; and
- To check the validity of search parameters upon which the resource estimates were based.

The variography was carried out based on the 1m composite intervals, and analysis was performed using Datamine Studio v3 software.

Variogram Parameters

Directional semi-variograms for the along strike, down dip and downhole directions were generated for Au, Ag, Pb and Zn using 1m composite sample data. The nugget variances were modelled from average downhole variograms based on a 1m lag reflecting the downhole drillhole composite spacing. Separate semi-variograms were generated for each element and each orebody.

Variography Interpretation

The principal direction of continuity was selected from the generated experimental semi-variograms and modelled with two-structure spherical models. The three orthogonal orientations represented the predominant along-strike, down-dip and cross-strike directions. Overall the semi-variograms generated are considered to be well structured and interpretable.

15.4.3.6 Block Modelling

Separate block models were constructed for each orebody in the deposit, each one having its own prototype. Each model comprises parent cells of $2m^3$ with sub cell splitting to a minimum block size of $0.5m^3$ used where additional cell resolution is required. The models are all un-rotated. WAI recommends that future modelling should aim to include all the ore bodies within one model prototype with an integrated coding for each ore zone.

15.4.3.7 Density

Density has been calculated based on lead content of each block, the formula outlined below was calculated based on regression analysis.

$$SG=2.7971+Pb*0.0316$$

15.4.3.8 Grade Estimation

Grade estimation was carried out using Inverse Power of Distance Cubed (IDW^3). Each of the ore bodies was estimated separately.

The estimation process comprised six different search radii, each one progressively larger than the last. These six search radii and the sample constraints are shown in Table 15.5.

Table 15.5: Novoshirokinskoye Search Estimation Parameters

Interpolation Run No	Search Axis (m)			Min No Samples	Max No Samples	Min No Holes/Channels
	Strike	Down Dip	Across strike			
1	2.5	2.5	2.5	1	12	1
2	15	15	5	3	12	2
3	30	30	10	3	12	2
4	60	60	20	3	12	2
5	120	120	40	1	12	1
6	240	240	80	1	12	1

For the estimation, a block discretisation of 2x2x2 was applied, each parent cell in the model was estimated individually with all sub-cells of a given parent cell receiving the same grade. Octants were used in the estimation with a maximum number of samples per octant of 3.

15.4.3.9 Validation

Introduction

A statistical and visual assessment of the block models was undertaken to assess the robustness of the grade estimations within each model and to ensure that the grade estimates and search radius passes were acceptable. The model validation methods carried out included a visual assessment of grade, global statistical grade validation and SWATH plot (model grade profile) analysis.

Visual Assessment of Grade Estimation

A visual comparison of composite sample grade and block grade was conducted in plan view and cross section. Visually the model was generally considered to spatially reflect of the composite grades.

SWATH Analysis

SWATH plots have been generated from the model by averaging both the composites and blocks along northings, eastings and vertically. The dimensions of each panel are controlled by the dimensions of the block size. SWATH plots were generated for all block model estimation methods. Each estimated grade should exhibit a close relationship to the composite data upon which the estimation is based.

WAI Comment: *Globally, no indications of significant over or under estimation is apparent in the models 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.*

15.4.3.10 Resource Classification

The resource classification for Novoshirokinskoye is classified in accordance with the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code (2004)). Criteria for defining resource categories were also derived from the geostatistical studies.

Classifications are:

- ***Measured*** resources - 30 x 30m search radii (along-strike x down-dip);
- ***Indicated*** resources - 60 x 60m search radii (along-strike x down-dip); and
- ***Inferred*** resources - within defined mineralised zones.

Adjustments to the classifications were also made on a visual basis. Where ore bodies had very limited number of intercepts, the classification was downgraded, typically in small, separate outlying areas.

15.4.3.11 Resource Estimate

The grades in the final resource block model were derived from diamond drillhole and channel sample composites based on Inverse Power Distance Cubed method for Au, Ag, Pb and Zn. Complete block models were built reflecting both the remaining in-situ parts, as well the parts which has already been mined out.

The resource estimate for Novoshirokinskoye is given in Table 15.6 below.

Table 15.6: Novoshirokinskoye Mineral Resource Estimate - COG of 3g/t Au_{eq} (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

Classification	Volume (m^3)	Tonnes (Mt)	Density (t/m^3)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
Measured	835,326	2.43	2.90	4.43	87.74	3.43	1.47
Indicated	1,603,523	4.64	2.89	4.30	94.82	3.07	1.15
Measured + Indicated	2,438,849	7.06	2.89	4.34	92.39	3.19	1.26
Inferred	525,673	1.51	2.87	2.08	57.02	2.44	1.81

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. Grade represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

WAI Comment: *Given the nature of the mineralisation at Novoshirokinskoye, the relatively simple structure of the ore zones and their good continuity, WAI is satisfied that the resource model presented above is a good reflection of the magnitude and tenor of mineralisation within the deposit.*

15.5 Mining

15.5.1 Introduction

Operations at Novoshirokinskoye mine started relatively recently with industrial scale production first achieved early in 2010. Currently the mine is producing around 35-38kt of ore per month, targeting 450kt annual production.

15.5.2 GKZ Resources and Reserves

The latest GKZ-approved resource/reserve estimation was performed in 2007. Since that time the overall tonnages have been depleted to correspond with the actual mined material. During the 2007 estimation economical and technical parameters were applied and formed the base for an equivalent gold ore grade estimate.

It should be noted that since 2007 the original economical factors used for the evaluation have not been updated. As the prices for metals contained within this deposit have changed, there is a strong potential for a resource/reserve base improvement.

15.5.3 Mine Design and Current Mining Activities

The original mine design dates to the late 1960's when the deposit was intensively studied and pilot underground mine design took place. The design had been revisited in the late 1980's and a new mine was mostly built. Due to a number of reasons (mainly due to financial and political instability in Russia during that time) the project remained inactive until 2004.

In 2004 an updated design was performed, considering the requirement for rehabilitation works (as the underground workings remained abandoned for some time) and modernised infrastructure facilities. According to this design, the mine is accessed via two shafts (which have been constructed but required some repairs – as shown in Table 15.7). One of the shafts "Skiovaya" is used to hoist the ore (In skips), whilst the other "Kletievaya" is used for man, materials and waste hoisting (cage), providing an annual production rate of 450kt.

> ***WAI Comment:*** *In summary, the design has inherited old, labour-intensive mining methods, involving overhead tracked loaders, scrapers and tracked electrical haulage.*

Currently, mining operations are performed to the design described above, with the majority of production coming from sub-level stoping (79% of production) and ore shrinkage stoping (conventional short hole shrinkage stoping). The shafts extend to the 750m level (with skip loading and weighing facilities below that level).

First production from the mine took place in January 2010, with full year production details available in Table 15.7 below.

Table 15.7: Summary of Novoshirokinskoye Production (2010)						
January	**Ore**			**kt**	**10,890**	
	Metal	%	Pb	t	2.06	224
		%	Zn	t	0.84	91
		g/t	Au	kg	3.02	33
		g/t	Ag	kg	62.17	677
February	**Ore**			**kt**	**18,882**	
	Metal	%	Pb	t	2.73	516
		%	Zn	t	0.80	151
		g/t	Au	kg	4.16	79
		g/t	Ag	kg	74.54	1,407
March	**Ore**			**kt**	**21,346**	
	Metal	%	Pb	t	2.04	436
		%	Zn	t	0.80	171
		g/t	Au	kg	3.34	71
		g/t	Ag	kg	65.66	1,402
April	**Ore**			**kt**	**22,357**	
	Metal	%	Pb	t	2.35	525
		%	Zn	t	0.87	195
		g/t	Au	kg	3.71	83
		g/t	Ag	kg	76.69	1,714
May	**Ore**			**kt**	**24,700**	
	Metal	%	Pb	t	2.12	523
		%	Zn	t	0.88	218
		g/t	Au	kg	2.80	69
		g/t	Ag	kg	63.72	1,574
June	**Ore**			**kt**	**27,407**	
	Metal	%	Pb	t	2.19	600
		%	Zn	t	0.97	266
		g/t	Au	kg	3.18	87
		g/t	Ag	kg	61.47	1,685
July	**Ore**			**kt**	**34,902**	
	Metal	%	Pb	t	2.21	770
		%	Zn	t	0.91	316
		g/t	Au	kg	4.10	143
		g/t	Ag	kg	55.40	1,934
August	**Ore**			**kt**	**38,505**	
	Metal	%	Pb	t	2.21	851
		%	Zn	t	0.74	285
		g/t	Au	kg	3.57	137
		g/t	Ag	kg	59.90	2,306
September	**Ore**			**kt**	**37,550**	
	Metal	%	Pb	t	2.23	837
		%	Zn	t	0.85	319
		g/t	Au	kg	4.1799	157
		g/t	Ag	kg	64.3	2,414
October	**Ore**			**kt**	**37,183**	
	Metal	%	Pb	t	2.24	83
		%	Zn	t	1.02	380
		g/t	Au	kg	3.52	131
		g/t	Ag	kg	63.84	2,374
November	**Ore**			**kt**	**38,025**	
	Metal	%	Pb	t	0.86	327
		%	Zn	t	1.88	715
		g/t	Au	kg	3.56	135
		g/t	Ag	kg	58.91	2,240
December	**Ore**			kt	33,990	
	Metal	%	Pb	t	0.77	262
		%	Zn	t	2.14	727
		g/t	Au	kg	2.87	976
		g/t	Ag	kg	66.40	2,325

The main orebodies which are currently being mined are Glavnoye, Orebody No.5, and Orebody No.7. Stope access is from the 933m, 850m, 800m and 750m levels which are the only developed levels. Production in 2011 will mainly be focused on Orebody No.7 (30% of overall quantity of the ore). A long section of Orebody No.7 is given in Figure 15.7 below.

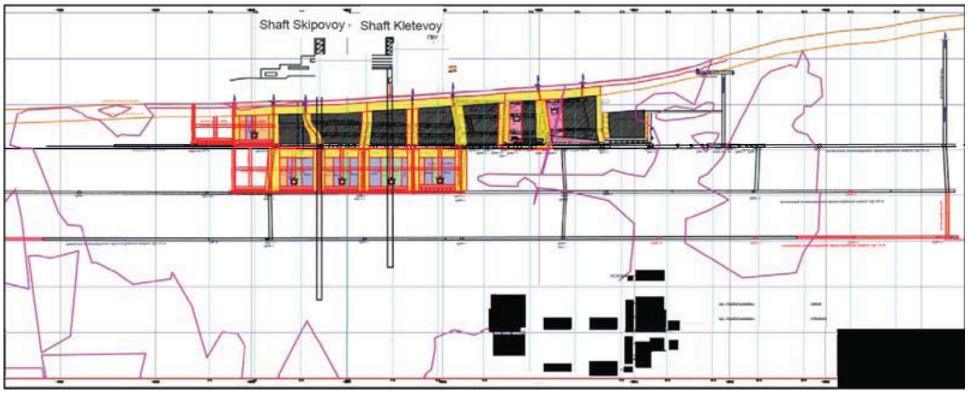


Figure 15.7: Long Section through Orebody No.7. Extracted areas – Hatched; Areas to Mine in 2011 – Shown in Colour; Safety Pillars –Yellow; Orebody No.7 Contours – Pink

Mining works are progressing from the centre of the deposit to the flanks, mining reserves on the top levels first. Currently, one of the most developed levels is the 850m level, a plan of which is presented in Figure 15.8 below. The plan shows the Glavnoye orebody, Orebody No.7 and Orebody No.5 accessed by drifts and a number of access drives used for an overhead loader.

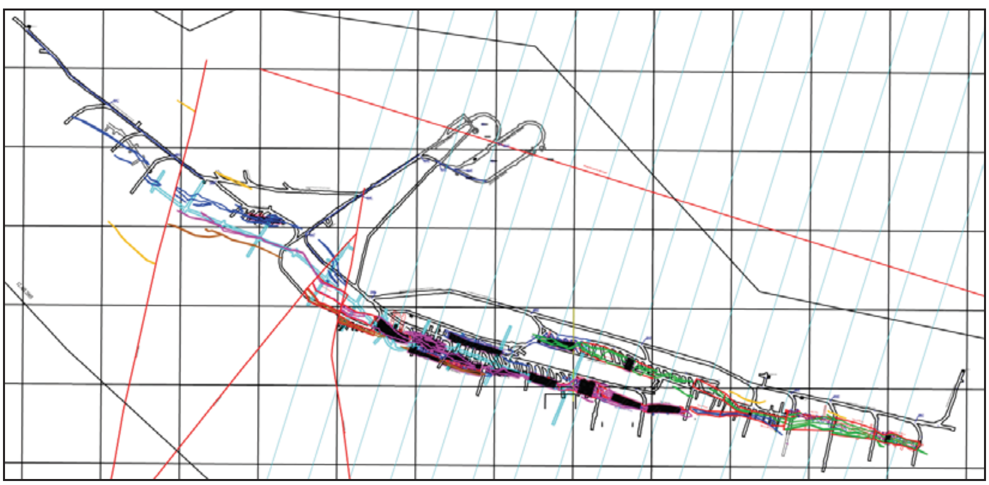


Figure 15.8: Plan of 850m Level

15.5.4 Mining Schedule

It is intended that the current production rate of 450ktpa be maintained in 2011 and increased to 550ktpa by 2013. The summary of mining schedule for 2011 is given in Table 15.8 below.

Table 15.8: Summary of Mining Schedule for 2011									
	Ore Mined (Total)	Pb		Zn		Au		Ag	
Unit	**t**	**%**	**kt**	**%**	**t**	**g/t**	**kg**	**g/t**	**t**
January	32,500	2.55	828	1.18	384	2.62	85	70.22	2,282
February	34,000	2.48	842	1.31	446	2.44	83	73.03	2,483
March	34,000	2.37	805	1.39	472	2.53	86	72.53	2,466
I Quarter	**100,500**	**2.46**	**2,475**	**1.30**	**1,302**	**2.53**	**254**	**71.95**	**7,231**
April	35,500	2.60	924	1.36	484	2.76	98	72.14	2,561
May	39,000	2.71	1,055	1.26	492	3.51	137	68.54	2,673
June	39,500	2.69	1,064	1.18	468	3.85	152	66.10	2,611
II Quarter	**114,000**	**2.67**	**3,043**	**1.27**	**1,444**	**3.39**	**387**	**68.82**	**7,845**
July	39,000	2.74	1,070	1.14	444	3.54	138	64.36	2,510
August		3.30	1,303	1.34	530	4.25	168	63.72	2,517
September	39,000	2.98	1,162	1.29	502	4.18	163	67.13	2,618
III Quarter	**117,500**	**3.01**	**3,535**	**1.26**	**1,476**	**3.99**	**469**	**65.06**	**7,645**
October	39,000	3.12	1,216	1.22	475	3.92	153	53.51	2,087
November	40,000	2.95	1,180	1.21	483	3.88	155	54.48	2,179
December	39,000	2.96	1,156	1.39	544	3.82	149	56.87	2,217
IV Quarter	**118,000**	**2.91**	**3,552**	**1.23**	**1,502**	**3.76**	**457**	**52.01**	**6,483**
Total	**450,000**	**2.80**	**12,605**	**1.27**	**5,724**	**3.48**	**1,567**	**64.90**	**29,205**

In addition, it is planned that 2,050m (17,600m^3) capital development works; 2,438m (9,750m^3), primary development works; 2,549m (15,294m^3), preproduction mining and 240m (1,152m^3) of exploration works are to be carried out during 2011.

Expected dilution and mining losses are 12.5% and 8.8% for sublevel stoping respectively; and 21.7% and 6.1% for shrinkage stoping respectively.

The majority of the ore in 2011 will be mined via a sub-level open stoping method. The remainder is planned to be mined via conventional shrinkage stoping. A description of the mining methods is given below.

15.5.5 Mining Methods

15.5.5.1 Introduction

The majority of production at Novoshirokinskoye is now performed using hand-held jack-leg mining equipment with scraper winches. The principal haulage is by tracked equipment.

The main mining method to date is sub-level open stoping (75% of total ore tonnage produced) accompanied by short hole stoping and shrinkage stoping (21% of total ore tonnage produced). In accordance with the design, four ore types are present at the deposit, and are designed to be mined employing different methods.

Type I: steeply dipping continuous orebodies with medium thickness of 5-15m, to be mined using sub-level stoping;
Type II: steeply dipping orebodies with medium thickness of 3-5m, to be mined using longhole shrinkage;
Type III: steeply dipping thin orebodies (up to 3m) are supposed to be mined using shorthole shrinkage;
Type IV: shallow dipping (40-55O) orebodies with thickness from 0.8m to 7.0m, to be mined using stull mining. Such orebodies are not common in the deposit and therefore this system is used very rarely;

15.5.5.2 Sub-level Open Stoping

The majority of ore is extracted using the Sub-Level Open Stoping method. The average block parameters for this system area as follows:

- Length – 50m;
- Rib pillar – 10-12m;
- Width – defined by orebody thickness; 7.5m on average; and
- Total height of full level – 50m.

The ore is accessed via waste haulage cross-cut drives and block raises. Subsequently access drives are cut and drilling drives are made. Extraction starts by developing a slot between the stope and rib pillar, which represents an expanded slot raise. The ore is blasted in vertical layers, having two or three such layers in a blast. Ore is withdrawn via a discharge hole at the bottom of the stope, where it is mucked by overhead loader into a wagon.

Blasting of ore is in vertical blast hole rings. The ore is broken in several sublevels at a time (over the whole chamber height without taking the crown pillar into account), with the upper sublevel being ahead of the lower one by 2-4m in order to maintain the access to the next hole rings. The ore is drawn from stoping zone by two methods:

1. Through the ore passes to scraper drift worked out in the bottom of a mining unit. Then it is delivered to wagons via scraper winch; and
2. Through the ore passes to access cross-drifts in the bottom of a mining unit as well, then it is loaded by pneumatic rocker shovel (PPN type) into wagons.

After a stope has been mined, temporary pillars are removed and the stope is left unfilled. This may potentially lead to caving of the rock above. Such caving, may hole to surface causing subsidence crater emergence, exemplified by holes, fractures and crack zones. The surface area potentially affected by caving is thoroughly assessed and fenced at surface. All major buildings are located beyond the boundaries of this area, taking into account appropriate distances for safety pillars.

15.5.5.3 Shrinkage Stoping

Short Hole Shrinkage

This method is applied when mining thin (up to 3m), steeply dipping ore bodies in medium to stable rock. The method is based on the temporary storage of crushed ore within the stope (two thirds of overall volume), providing a floor for further hole drilling.

After the ore is blasted, a potion of the crushed ore from the bottom of the stope is fed to wagons from draw points using LHD. The faces are accessed via a timber ladder-way (raise) which would normally be surrounded by crushed rock.

The method is used for extraction of around 21% of the overall tonnage. Typical shrinkage stope parameters are as follows:

- Length – 50m;
- Width – defined by orebody thickness, average 2.0m;
- Crown pillar – 4m; and
- Height – 50m.

Long Hole Shrinkage

The method is similar to conventional shorthole shrinkage. It is used for thicker orebodies (up to 5m) than shorthole shrinkage, providing a higher production rate by blasting more material at a time.

Blastholes are drilled from chambers driven in block raises in every 8.4m (chamber height is 2.4m). Holes are drilled from the horizontal to vertical, and hole length varies from 4 to 30m. As for shorthole shrinkage, a portion of the ore is stored within the stope to provide support for the walls.

> ***WAI Comment:*** *The majority of ore is extracted employing sub-level stoping using low-productivity equipment and techniques, such as hand-held drill rigs, scrapers and tracked overhead loaders. A more effective mining method should be considered in order to increase production.*

15.5.6 Rock Properties and Geotechnical Conditions

The ore of the deposit is generally stable and hard; the hardness factor for the ore is in the region of 8-14, and for waste 11-16 on the Protodyakonov scale. The density of the sulphide ore is 3.10t/m^3, transitional ore 2.89t/m^3, and ore bearing rock and waste 1.5 to 1.6t/m^3. Average moisture content is 4-5%.

15.5.7 Drilling and Blasting

Hand-held drill rigs are widely used for drilling and blasting works. The majority of short holes are made by PP-63V and PT-48A Russian-manufactured rigs. 42mm diameter holes are normally used, and the depth of holes varies, depending on direction: horizontal holes 1.8-2.0m; and vertical holes 1.4-1.6m. Ammonit No.6 ZhV (explosive mix of Ammonium Nitrate, trotyl and fatty acids salts) is used for blasting these holes, initiated by both electric (EDNZ) and non-electric (SINV-Sh, DSh) detonators.

Long holes (mostly blasting) are normally drilled by a BP-100N and/or LPS-3U drill rig to a maximum depth of 30m. Standard hole diameter is 110mm. Granulated explosives, such as Granulit A-6, AS-8 and Igdanit are used for charging the holes using a pneumatic charging machine. Explosives are delivered to the site from local underground magazines.

15.5.8 Losses and Dilution

The inconsistent shape of the orebodies forming the deposit, and the bedding conditions, dictate that dilution and losses are variable. The two mining methods used for ore extraction also produce variable losses and dilution values. Monthly production reports containing detailed data on actual dilution and losses were provided to WAI at the site. Monthly results show that the expected dilution and mining losses are 12.5% and 8.8% for sub-level stoping; and 21.7% and 6.1% for shorthole shrinkage respectively.

15.5.9 Ore and Waste Transportation

15.5.9.1 Underground

Ore and waste transportation from stopes is performed by K-10 and 7KRM1 electric locomotives. Each locomotive is capable of towing up to 11 wagons, with volume of 2.2 or 0.8m3 depending on the type of material to be moved. Smaller wagons are used for waste transportation. Wagons are discharged by rotary wagon tippers (OVK-1-2.2-750. At the top of the Kletyevaya Shaft there is a wagon exchange and discharge complex installed for waste removal.

The average transportation distance within a haulage horizon (750, 800 or 850m) is normally around 0.6km (see plan of 850m level on Figure 14.22 above), and the standard locomotive travel speed is 7-12km/h.

Ore from stopes is normally loaded into 2.2m^3 wagons by PPN overhead loaders and then delivered to ore passes connected to the 750m level crusher. The ore pass also functions as an ore accumulating buffer before the crushing facilities. After the ore is crushed to the required size, it is loaded into one of two 3.0m^3 capacity skips and hoisted to the surface via Skipovaya Shaft. At the surface the skip is discharged to a bunker which is directly connected to the processing plant.

15.5.9.2 Hoisting Facilities

The Skipovaya shaft is equipped with an SHPU 2Ts-4x2.3 hoisting machine (drum diameter 4m, width 2.3m) which is used for ore hoisting (two 3.0m^3 skips). The winder is equipped with an AKN-14-16-10 electrical motor, with an installed capacity of 630kW providing a maximum skip travel speed along the shaft of 6.2m/s.

The Kletyevaya Shaft has the same hoisting machine as the Skipovaya Shaft, operating with cage and counterweight. This shaft is used for waste hoisting, men and materials delivery. The shaft is equipped with an AKN-13-66-10 electric motor, and the installed capacity of 500kW providing a maximum cage travel speed of 6.2m/s.

> ***WAI Comment****: Inspection of the hoisting facilities showed that these are operated to a high standard of maintenance and are in good working condition. Based on the Skipovaya Shaft technical parameters, it is expected that it can operate to a higher capacity than current.*

15.5.10 Dewatering System

There are two pumping stations at the mine. One is located in close proximity to the Kletyevaya Shaft, and another (smaller) at the Skipovaya Shaft. Water from Kletyevaya Shaft is collected at the bottom of the shaft and then drained to Skipovaya shaft. Then water is pumped to the horizon 750m settling sumps, using two TsNS-38-66 pumps (one operating, one on stand-by). The specification of the pumps installed at both stations is given in Table 15.9 below.

After the water has settled, it flows into collecting sumps and then is pumped to the surface.

Expected water inflow into the shafts is 92m^3/h. The inflow rate into horizontal development at levels 850m, 800m and 750m is 200, 325 and 454m^3/h respectively.

Table 15.9: Major Dewatering Facilities

Installation area	Pump Model	Capacity (m^3/h)	Head (m)	Quantity Installed
Skipovaya Shaft	TsNS-38-66	38	66	2
Kletyevaya Shaft	TsNS-300-240	300	240	5

15.5.11 Ventilation

The mine has forced ventilation. Fresh air is supplied through Kletyevaya Shaft by a VO-24K main ventilation fan (a second fan is on stand-by of the same make and model). The main ventilation fan is reversible, performed by changing the fan blade direction. The supplied air is heated when required.

The air is delivered to the operating levels via the Kletyevaya shaft, and then it splits into two directions – east and west. Both ventilation paths provide air to the operating faces, and connect to ventilation raise leading to surface (eastern and western ventilation raise respectively). Airflow for each path is adjusted by isolating doors. Skipovaya shaft is also used for exhaust air discharge.

The forced ventilation scheme benefits from the ability to provide fresh, heated air from a centralised location, minimising shaft freezing; and reducing unintentional air intake from collapsed stopes and cracks connected to the surface.

Remote areas, such as active production faces, are supplied with air by means of local fans (VME-6, VME-5) using vent ducting.

The approximate distribution of airflow is as follows:

- Eastern Sector – 38.5m³/s;
- Central Sector – 66.5m³/s;
- Western Sector – 12m³/s.

WAI Comment: *It was noticed by WAI during the underground visit that ventilation is operating to a good standard, providing fresh air to remote parts of the mine as required.*

15.5.12 Mine Personnel

A total of approximately 1,150 people are employed by Kazzinc at Novoshirokinskoye. The mine has a developed structure of departments, each of which has specific designated areas and scope of works. A list of departments together with number of administrative and non-administrative staff employed is given in Table 15.10 below.

Table 15.10: Mine Administrative and Non-Administrative Personnel

Mine Administrative		Mine Non-Administrative	
Management	8	Underground Mining Department	172
Engineering Service	16	Underground Transportation Department	126
Geology Service	6	Drilling and Blasting Department	53
Surveying Service	5	Ventilation Service	4
Health and Safety Service	9	Maintenance Service (UG)	38
Financial Service	15	Equipment Maintenance Service	8
HR Department	15	Compressor Service	5
Commercial Service	12	Explosives storage	1
Project Manager Team	4	***Total for the mine personnel***	***407***
Quality Control Service	1	Processing Plant	159
Total Administration	***91***	Quality Control Department	15
Mine	32	Laboratory	26
Processing Plant	14	Transport Department	79
Plant Quality Control Department	4	Maintenance Service (Surface Equipment)	18
Transport Department	2	Heating and Water Supply Service	20
Maintenance	2	Power Supply Service	9
Power Supply Service	1	Electrotechnical Laboratory	2
Electrotechnical Laboratory	3	Measuring Facilities Service	4
Measuring Facilities Service	4	Automation Service	5
Automation Service	3	Engineering Service	45
IT Department	5	Building and Construction Service	3
Building and Construction Department	1	Storage Facilities Department	20
Storage Facilities Department	5	Domestic Services Department	105
Domestic Services Department	2	Safety Department	6
Safety Department	6	**Total**	**884**
Plumbing and Sewerage Department	3		
Total	**269**		

WAI Comment: *The mine is remote from any inhabited area, and therefore all labour resources are brought to the site, mainly from Chita. Considering the fact that this part of Russia is known for its historical and present mining activities, no major employment issues are anticipated.*

15.5.13 Prospective Mine Development

It is intended that the production rate of 450ktpa be maintained in 2011 and increased to 550ktpa by 2013. Most of the mining works will still be focused on the three main orebodies Glavnoye, Orebody No.5, and Orebody No.7. The increase of the production is planned to be achieved by employing mobile trackless equipment.

A preliminary assessment of potential mining method options, and estimated operational and capital expenditures has been made. On the basis of these estimations, a LOM production schedule has been designed.

15.5.14 WAI Reserve Estimation

WAI has performed an evaluation of the Novoshirokinskoye mine Ore Reserves in accordance with the requirements of the JORC Code (2004). The evaluation has been performed using geological block model produced by WAI dated 01.01.2011 and considers modifying factors, such as dilution and losses, with respect to the current mining operations and methods.

The stope boundaries were generated to correspond with the standardised dimensions of the existing stopes and development in the mine and also be above the minimum cut-off grade for exploitation. A summary of typical stope parameters is given in Table 15.11 below.

Table 15.11: Typical Stope Parameters

Length (m)	50
Width	Decided upon orebody thickness and method: SLOS: 5-15m LongHole Shrinkage: 3-5m Conventional Shrinkage: 0.8-3m
Height (m)	50
Minimal Au_{Eq} Average Grade (%) For a MU	3.0
Dip Angle (Deg)	50-90

Datamine® Mineable Stope Optimiser® (MSO) software was used to assess whether mineable stopes can be generated for certain parts of the deposit. MSO computes the optimal size, shape and location of stopes for an underground mine using an input block model containing grades or values. $Au_{Equivalent}$ grades were used for Novoshirokinskoye evaluation.

MSO searches for the optimal mineable shapes taking into account the orebody geometry. The programme generates stope shape outlines on adjacent sections, and links these to create a wireframe shape for evaluation against the block model. The sectional outlines are defined by four points on the roof and floor. Constraints are to be applied on the dip and strike of the final stope shape.

The stopes produced by MSO are then evaluated against the geological block model in order to quantify average grades of the economic minerals and the metal content, together with ore and waste tonnages.

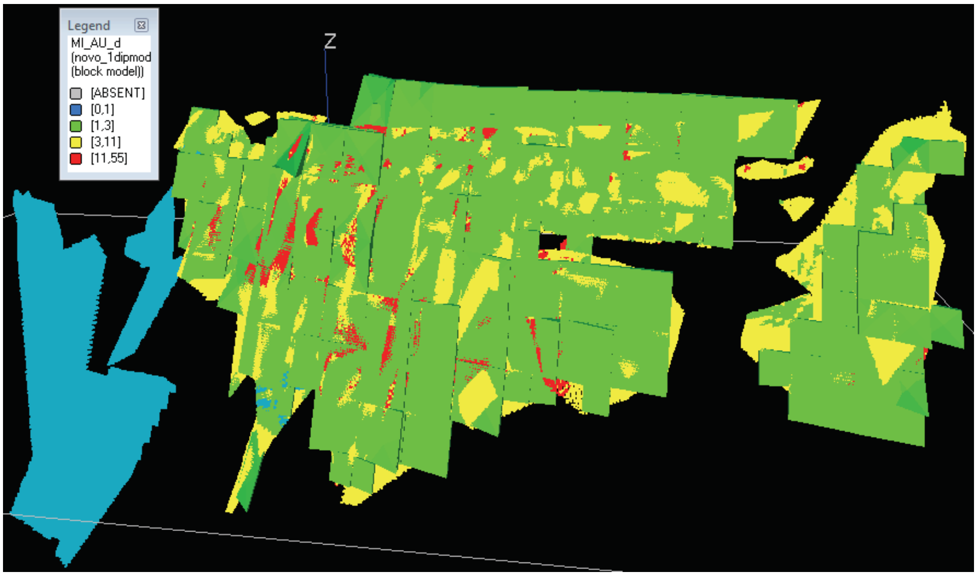


Figure 15.9: Stope Layout for Novoshirokinskoye (Orebody No.1)

(Blue – low grade or Inferred material; green – MSO stopes, yellow, green, and red cells – model blocks with 0-1, 1–3, 3–11, and >11 g/t of AuEq in Measured and Indicated categories)

A total of approximately 3,500 stopes were produced to justify the Novoshirokinskoye ore reserves. The greatest part of the reserve is contained in the central part of the deposit, but there is a significant portion of reserves located in the western part of the deposit. A summary of ore reserves is given in Table 15.12 below.

Table 15.12: Novoshirokinskoye Ore Reserves (WAI 01.01.2011)
(In Accordance with the Guidelines of the JORC Code (2004))

	Tonnes (Mt)	Au_{Eq} (g/t)	Au_{Eq} (kg)	Ag (g/t)	Ag (kg)	Au (g/t)	Au (kg)	Pb (%)	Pb (t)	Zn (%)	Zn (t)
Proven	2.44	7.89	19,233	77.0	187,676	3.89	9,473	2.98	72,601	1.28	31,089
Probable	4.43	7.78	34,463	84.3	373,276	3.89	17,200	2.69	118,896	0.99	43,912
Total	***6.87***	***7.82***	***53,695***	***81.7***	***560,953***	***3.89***	***26,673***	***2.79***	***191,498***	***1.09***	***75,000***
Inferred material within stopes	*0.048*	*5.16*	*249*	*53.3*	*2,577*	*1.5*	*74*	*1.94*	*937*	*1.14*	*549*

15.5.15 Conclusions

The Novoshirokinskoye mine is a relatively new operation at which full capacity has only been approached in 2010. Currently developed reserves and exploration campaigns performed in the area ensure availability of a reserve until 2025 and beyond.

The mine is operated to a high standard. Employment of qualified and skilled personnel provides stable production rates, although existing underground infrastructure has a spare capacity in case an increase in production rate is planned.

Despite the fact that the mine is remotely located and mining methods involved are labour-intensive, reasonable mining costs for such a type of operation have been achieved.

15.6 Process

15.6.1 Introduction

The plant at Novoshirokinskoye began treating ore in December 2009. Construction of the plant buildings commenced in the early 1990's but work was stopped due to the break up of the Soviet Union. The plant uses the standard minerals processing techniques of SAG-ball milling, gravity and flotation to produce a gold-rich lead concentrate and a zinc concentrate.

15.6.2 Flowsheet

Novoshirokinskoye ore processing flow sheet comprises the following stages:

- Coarse ore crushing (≤300 mm) in the underground jaw crusher to 150 mm;
- Semiautogenous grinding and sizing to 80% - 1mm passing size;
- Jigging of 1st stage grinding discharge;
- Ball milling and sizing to 82% -0,074mm passing size;
- Jigging of 2 stage grinding discharge;
- Sizing and after-grinding of jig concentrate in the ball mill;
- Gravity concentration of crushed concentrate on the shaking table and in the centrifugal concentrator;
- Cyanide-free flotation of gravity tailings (82% -0,07mm passing size) to produce Pb-Au-Pyrite concentrate;
- Selective flotation of bulk concentrate to produce gold-rich lead and pyrite concentrates;
- Flotation of bulk flotation tailings to produce Zn concentrate;
- Fine grinding of pyrite concentrate and sizing to 80% -0,020mm passing size;
- Gravity flotation of ground pyrite concentrate in the centrifugal separator to produce gold-rich concentrate; and
- Thickening and filtration of gold-rich lead concentrate and zinc concentrate.

15.6.2.1 SAG and Ball Milling

Ore is crushed underground and hoisted directly into coarse ore bins (four times 320t and two times 550t.) Ore is hoisted 6½ days per week.

Ore is recovered from the bins using 16 vibrating feeders and passes via a conveyer belt, fitted with a weightometer, magnet ainstalled above the conveyoer belt nd a metal detector, to the milling section. There are two grinding lines each consisting of one MPS 50x23 SAG mill and one MShTs 3.2x3.1 ball mill. The circuit is flexible and during the WAI site visit one SAG mill and two ball mills were being used.

1st stage grinding discharge undergoes two sizing cycles – primary sizing is performed in КСН-24 classifier with underflow recycled in the bank of Cavex 400CVX10 hydrocyclones. The hydrocyclone underflow is fed to a 2nd sizing cycle with discharge being the primary feed for rougher bulk flotation.

The through put capacity of one grinding line is 350ktpa. During the WAI site visit this through put was being exceeded which necessitated the use of the second Ball mill. The grinding circuit is shown in Photo 15.1.

The total grind in capacity is therefore 700,000tpa.

Photo 15.1: Novoshirokinskoye Grinding Circuit

Gold-rich lead concentrate in the grinding cycle is produced in MOD-2 jigs with the underflow being rough gravity concentrate and overflow pumped for sizing.

The rough gravity concentrate is fed to the after-treatment section for sizing in hydrocyclones and after-grinding in a MShTs 2.1x3.0 ball mill is then treated on SKO-15 shaking tables with three tables used for main after treatment and one for cleaner flotation of middlings after the main after treatment.

Rougher and cleaner flotation tailings are the fed to a KC-CVD20-1 centrifugal separator with continuous discharge. After the separator, the concentrate undergoes scavenger concentration on the shaking table. Rougher, after-treatment and scavenger concentrates are combined and fed to the thickener. Scavenger concentration tailings are pumped to the leaf thickener with its discharge fed to flotation.

15.6.2.2 Flotation

Bulk and selective flotation is applied to produce saleable lead and zinc concentrates.

The hydrocyclone discharge (82% -74µm) is pumped to the bulk lead-pyrite flotation cycle comprising rougher, scavenger and cleaner flotation stages. A conventional reagent suite of xanthate, pineoil frother and sodium carbonate for media control is applied. Bulk flotation tailings are pumped to zinc flotation.

The bulk flotation concentrate is pumped to lead flotation comprising rougher lead flotation, one cleaner and one scavenger flotation cycle. Flotation lead concentrate is fed to thickening stage, where it is mixed with gravity concentrate. Lead flotation tailings are pumped to an ultrafine grinding section for gravity separation in the centrifugal separator.

The zinc flotation cycle comprises cleaner zinc flotation with three cleaner and one scavenger stages. Zinc concentrate is pumped for thickening. Zinc scavenger tailings are waste tailings, they are pumped to the TMF.

Lead concentration tailings (the pyrite concentrate) are reground in a Metso Vertimill METSO VERTIMILL VTM-150WB to 80% passing 20µm. They are passed through KC- CV-20 Knelson concentrators with continuous discharge to recover free gold from the ground lead concentration tailings. The produced concentrate is mixed with both the flotation lead concentrate and the gravity concentrate from the milling circuit to produce a single final gold rich lead concentrate product.

Tailings from gold recovery cycle (from the pyrite product) are combined with zinc flotation tailings and pumped to the TMF.

All cells in the plant are new and supplied by CETCO (South Africa). The flotation plant has generally limited process control and instrumentation. The cells have built-in pH meters although lime addition is manual. The cells have manual level controls. There is no On-Stream-Analyser (OSA). An improved SCADA system is planned for next year. The Novoshirokinskoye flotation plant is shown in Photo 15.2.

Photo 15.2: The Novoshirokinskoye Flotation Plant

15.6.2.3 Concentrate Dewatering

Combined lead flotation concentrate, together with the gravity concentrates are dewatered in a conventional STs-9 thickener. The zinc concentrate is also dewatered in a second STs-9 thickener. There is a third SP-6 leaf thickener to dewater the gravity tailings streams.

The lead and zinc thickener underflow products are filtered using VDFK-30 - 30m^2 Ceramec vacuum filters. These are considerably oversized (concentrate filtering area), with the lead filter operating only 8 hours per day, and the zinc filter operating 4 hours per day.

The filtered concentrates are transferred at 7-8% moisture to Big Bags using mechanical grabs. The Big Bags are currently trucked a distance of some 250km, to a rail head at Priiskovai and then transported by rail to Ust-Kamenogorsk. A new rail link is currently being constructed which will result in a rail head some 17km, from the mine.

15.6.3 Plant Metallurgical Balance

The plant metallurgical balance for the month of October 2010 is given in Table 15.13.

Table 15.13: Plant Metallurgical Balance for October 2010							
					Grades		
Product	Weight %	Tonnes	Au g/t	Ag g/t	Cu%	Pb%	Zn%
Ore		37183.0	3.51	65.20	0.14	2.17	1.06
Pb Conc	5.8	2166.8	46.98	955.14	2.05	33.84	4.01
Zn Conc	1.5	574.9	4.78	126.68	0.52	1.53	43.92
Tails	92.6	34441.3	0.75	8.19	0.01	0.19	0.16
					Distribution		
		Product	Au	Ag	Cu	Pb	Zn
		Ore	100	100	100	100	100
		Pb Conc	77.99	85.37	86.94	90.79	21.97
		Zn Conc	2.11	3.00	5.87	1.09	63.79
		Tails	19.91	11.63	7.19	8.12	14.24

The Metallurgical Balance shows a satisfactory degree of upgrading. The overall gold recovery is lower than the target figure of 88.4%.

15.6.4 Analytical Laboratory and OTK Department

15.6.4.1 Introduction

The management of the analytical facility and the OTK (quality control department) are the responsibility of one individual. There are a total of 37 people employed in the analytical laboratory and 21 are employed within the OTK department

15.6.4.2 Analytical Laboratory

A Laboratory manager is responsible for this facility and is supported by two chemical engineers who are heads of the Fire Assay and Wet Chemistry Departments. The laboratory works 24 hours per day and 7 days a week. The Assay Laboratory processes mine exploration, plant and a limited number of environmental samples.

The sample preparation laboratory is equipped with two soil drying ovens, Rocklabs jaw and rolls crushers as well as ring mills for fine grinding. The sample preparation facility has proved to be a serious bottleneck and its capacity is limited to 42 samples per shift. There is therefore a serious backlog of several thousand mine exploration samples. The company is planning to expand this facility and will be using an adjacent building as additional work space and storage in the near future.

Gold analysis is undertaken in duplicate using standard fire assay techniques. Analysis is undertaken on 10–20g samples using a flux manufactured from borax, lead oxide, silica and soda ash. The lead products are subjected to the standard cupellation process and the beads from cupellation are parted using nitric acid, the final gold bead is weighed on a microbalance to an accuracy of 0.001mg.

Lead, zinc and iron analyses are undertaken using an Ametek XRF analyser. The analysis of lead and zinc concentrates is undertaken using titrimetric techniques. The laboratory is also equipped with two Varian atomic adsorption spectrometers (AAS). One is used for Cu, Pb, Zn, Fe, and Ag. The other is used for gold analysis using an organic solvent method. The laboratory also undertakes a limited number of environmental analyses for dust, noise, light and radioactivity.

The laboratory is attested by the federal agency "Technical Regulation and Measurements" (Chita Region). This is valid until 2013. The plant utilises a regular program of external cross check analysis with an accredited laboratory. The internal control consists of resubmitting 10% of samples as recoded duplicates. The laboratory initially used internationally recognised standard reference materials as part of its QA/QC procedures, but has recently began manufacturing in-house standards with cross check analyses at accredited laboratories.

15.6.4.3 OTK Department

The OTK department is responsible for monitoring plant performance and the preparation of metallurgical balances. There are automatic samplers installed on the flotation feed, flotation tailings and thickener feed streams. Separate samples are taken of the lead flotation concentrate, the gravity concentrates, and the combined concentrates in order that an overall balance on each process stream can be made. The automatic samples are taken every hour and then combined on a shift or daily basis and the results of the analyses are used to prepare metallurgical balances. The plant's belt weightometers are checked by the automation department using control weights and the truck weigh-scales are checked on an annual basis using a certified contractor.

15.6.5 Conclusions

The plant at Novoshirokinskoye achieves a satisfactory metallurgical performance using a combination of Russian and western equipment. The plant could be improved by the addition of further process control and instrumentation. The grinding section is capable of processing up to 700ktpa of ore, although further flotation cell capacity may be required to achieve this throughput. The plant appears to be well managed and there is a high standard of housekeeping.

15.7 Environmental

15.7.1 Introduction

Novoshirokinskoye mine is located approximately 450km east of the city of Chita in Eastern Siberia, Russia. The mine is located within the Zabaikalskig Krai Oblast with Chita as the Regional Oblast Centre. It is connected by a metalled road, and freight and passenger railway lines (part of the Trans Siberian Railway) are present at the rail head at Priskovaya approximately 250km from the mine. The Mine operates its own siding at this railhead. A new railway line for Norilsk Nickel is currently under construction which when complete will come within 17km of the Site.

15.7.2 Environmental & Social Setting and Context

15.7.2.1 Landscape, Topography

Land around Novoshirokinskoye Mine is predominantly undulating topography with hills varying from 650-1,000m in height. The region is not particularly seismically active but earthquakes up to 4 on the Richter scale have been reported in the last 10 years.

Soil horizons are expected to be thin and lacking in fertile topsoil.

There are numerous tree and grass species, in a mosaic habitat with deciduous species including birch found predominately at higher altitudes. There are wooded hillsides (birch, larch and scrub) around the mine site and village.

15.7.2.2 Climate

The climate is sharply continental with hot summers reaching temperatures in excess of 30°C and cold winters dropping to below -40°C. Snow cover is present from November to April. The prevailing wind is from the northwest.

15.7.2.3 Land Use and Land Cover

The site has unoccupied buildings mostly dating from the later part of the 20th century but most of the current mine facilities are modern in design and construction dating from the last few years. A series of hardcore roads link the infrastructure at site and public highways link the mine to the mine camp and dangerous goods storage areas.

15.7.2.4 Water Resources

The mine currently has water ingress of 40-60m^3/h and is pumped at a rate of 450m^3/d to surface for use in the mineral processing circuit and dust suppression on the roadways around the Mine Site. It is understood that 5m^3 water per tonne of ore treated in the mineral processing plant is required, with makeup water supplied by return water from the Tailings Management Facility (TMF) and from local streams (when flowing).

Potable water is supplied by boreholes from the ground water at an abstraction rate of up to 900m^3/d.

15.7.2.5 Communities and Livelihoods

A mine camp is located approximately 1km from the main site entrance and the main village of Novoshirokinskoye a few kilometres further along the same asphalt road. The village developed originally with the early mining activities and contains administration buildings and schools serving a wider region. Mining and agricultural activities form the principal livelihoods.

15.7.2.6 Infrastructure & Communications

The mine is served by asphalt roads. Currently the closest rail head is over 200km from the site but it is understood that a new rail line is under development and will bring the rail link to within 17km of the mine.

The total (permanent) population of the village is around 600.

A shift camp for Mine personnel has been established comprising dormitories, sauna, gym and camping facilities. The field camp is served by a 400kW boiler fuelled by heavy oil and is equipped with a 7.0m^3 oil tank. Solid waste is collected from the camp and taken to the village landfill. It is understood that the possibility of changing to coal firing from the present heavy oil system is under consideration. Such a change would be implemented after the opening of the rail project that would enable a rail head to be created 17km from the Mine Site.

> ***WAI Comment:*** *WAI notes the mine plans to utilise a new rail head once the line has been constructed. WAI considers that this will offer environmental advantages and reduce road haulage mileage considerably. Once operational, further consideration will be given to replacing diesel oil boilers with a coal fired boiler. These actions will be subject to the OVOS procedure and will need approval by the environmental regulatory authorities.*

15.7.3 Project Status, Activities, Effects, Releases & Controls

15.7.3.1 Project Description & Activities

The project comprises an active underground mine, all surface facilities associated with the operation of such a mine, a mineral processing plant producing separate lead and zinc concentrates, a TMF for disposal of the tailings from the mineral processing plant, water storage tanks and pumping stations, waste rock stockpiles, heating boilers at the mine and, shift camp and village, a shift camp for the accommodation of staff and off-site storage of hazardous materials (explosives and sodium cyanide).

Vehicle maintenance facilities are located in (old) buildings within the mine complex with refuelling facilities provided by dedicated diesel road tanker (parked semi-permanently minus the tractor unit).

Two shafts and an adit access the underground workings. An electric railway line (narrow) gauge is installed in the adit and is used to bring ore to the surface.

The mineral processing plant consists of a conventional circuit comprising crushing and grinding using two SAG mills followed by flotation to produce separate lead and zinc (sulphide) concentrates. Pyrite is present in the ore and flotation is inhibited by the use of cyanide. Currently cyanide is used at a rate of 50g per tonne of ore processed.

Principal (solid) wastes produced by the project include waste rock, from mine development, tailings from the mineral processing operation, general wastes from industrial operations, sludges from the water treatment plant and ash if coal fired boilers are used.

Aqueous effluents are produced at the mineral processing plant and are transported with solid tailings (20% solids) to the TMF. A water treatment facility treats all sewage from the Mine Site and following solids removal treated water is discharged to the TMF. Waste rock and tailings are known to contain pyrite and hence acid rock drainage (ARD) could be expected.

> ***WAI Comment:*** *The mine sub-station and transformers are modern and should not contain potential contaminants such as PCB's.*

15.7.3.2 Land Ownership and Tenure

OAO Novoshirokinskiy Rudnik Mine JSC holds a licence to exploit (mine) polymetallic ores for 20 years from 2004. Kazzinc owns 48.3% of the shares in OAO Novoshirokinskiy Rudnik Mine JSC. As is normal for such licences, details are contained concerning expected production rates and when activities are expected to cease. After cessation of mining the Mining Company has 2 years to carry out a closure plan. Currently this has to have occurred by 2026. It is believed that the mining licence can be extended in duration by mutual agreement with the regulatory authority.

> ***WAI Comment:*** *WAI has not examined the mine licence and permit in any detail but believes that these have been obtained in line with State requirements. It is understood that the current mining and any future expansion of mining activity would not necessitate relocation of communities within the current licence boundary. Furthermore, mining induced subsidence is not anticipated to represent a significant risk to present habitations.*

15.7.3.3 Energy Consumption & Source

Power to the Mine is supplied from the National Grid via two transformers. The Mine operates a sub-station owned by South East Electricity Network Company. Kazzinc operate a Hydroelectric Power Plant at Bukhtarma in Kazakhstan, but this will not supply to Novoshirokinskoye.

WAI understands that one of the boilers in the village has retained coal as the energy source using coal from the Urtuisky Mine which is understood to be the closest thermal coal. This coal has a heating value of 17.8MJ/t, sulphur content of 0.2% and ash content of 7.3%. Ash from this boiler is currently sent to the municipal landfill. It is understood that this coal would be the preferred supply should the Mine change from oil.

At the mine site there is a (modern) boiler house with 4x1750kW boilers. These are supplied from 5 single-skin unbunded tanks, each of 90m^3 capacity. Off-gas quality (CO, NOx) is measured regularly to determine the operational efficiency of the plant. A derelict old boiler plant (estimated to date from 1960's) including brick stack remains at the site but it is believed that this has never been used by the current Mine owners.

Mine energy usage is quoted at 3MkWh/month. This figure covers power demand at all facilities including administration and accommodation at the shift camp. Electricity is supplied direct from the National Grid and the Mine operates a substation owned by South East Electricity Network Company and 2x10MVA transformers. These are of a modern design and installed within the last few years. It is understood that (most) nearby power stations are coal fired.

15.7.3.4 Mine Wastes – Waste Rock and Tailings

Novoshirokinskoye Mine produces a small quantity of waste rock which is dumped adjacent to the road connecting the Mine with the TMF. Tailings produced by the mineral processing plant are disposed of in the TMF (see above section for more detail).

A dedicated TMF was constructed and completed in 2009 to receive up to 450,000tpa tailings by damming an existing valley with an earth dam. The dam walls are protected by rip-rap rocks. The facility has a total storage capacity of 8.7Mt and is considered adequate for the current 12 year mine life. A first, upstream raise is planned for Spring 2012.

Topsoil and vegetation was not stripped from the site of the TMF prior to operation and no artificial liner system is included in the design. It is understood that naturally clays underlying the TMF offer some protection to the underlying groundwater. It is understood that the design allows for a minimum of 1m free-board and was designed to accommodate a 1:20 year storm event. It is noted that no spillway is included in the design. Downstream of the TMF drainage ditches are present to intercept any contaminated waters. These are pumped back to the TMF. There are three boreholes 200m downstream of the TMF for groundwater pollution control purposes. A control station is located 500m downstream in the Pad Shirokaya Brook.

> ***WAI Comment:*** *WAI notes that the current TMF is not lined. Seepage rates were calculated in 2008 by Mekhanobr Engineering JSC (St Petersburg, Russia) for each year of the TMF operation and concluded that seepage was not expected and did not predict any significant impact on water resources in the area. The current TMF operation includes dam stability control points, monitoring boreholes, visual dam slope inspection and other regular control procedures all performed in accordance with current environmental legislation.*

15.7.3.5 Water Management & Effluents

The mine operates a water treatment facility (biological treatment process using bacteria) and treated water is delivered by pipeline a further 1km to the TMF. The Water Treatment Plant became operational in October 2009.

Water balances have been made for the site. Ground water is monitored from 3 boreholes around the TMF, from within the TMF and 500m downstream in the River below the TMF. Water is analysed for a comprehensive suite of parameters and although tested for, no cyanide has ever been detected in any of the analyses. WAI understands from the mine management that results of monitoring are not needed to be sent to the regulatory authorities.

A series of interception ditches exist downstream of the main TMF dam. These are designed to intercept any seepage through the dam and prevent ingress into the local river network. There is no spillway included in the TMF design.

Pollution of surface waters could result from:

- Acid rock drainage and leaching from ore/waste rock/tails stockpiles;
- Run off from contaminated soils at the Mine site;
- Unplanned discharge of untreated minewaters;
- Uncontrolled seepage (e.g. catastrophic failure) from the TMF avoiding the intercept ditches; and
- Road run off.

Additional processes which may result in groundwater contamination include:

- Unsegregated deposition of (hazardous)waste to landfill;
- Absorption/dissolution of explosives residues and gases;

- Hydrocarbons, especially diesel spillages from the mobile fleet;
- Leakage from oil storage tanks;
- Leaching from contaminated soils; and
- Rebound of contaminated minewater at cessation of/or interruption to pumping.

> ***WAI Comment:*** *WAI considers that ARD test work on tailings and waste rock should be undertaken. However, it is noted that at the time of the site visit there were no visible signs of ochre staining which might be indicative of ARD. WAI notes that as much water as possible is reclaimed from the TMF for reuse in the processing plant. Given the use of cyanide in operations, it is recommended that the Company seeks compliance of its activities with the International Cyanide Management Code.*

15.7.3.6 Emissions to Air

The background air quality in the Novoshirokinskoye area is considered to be good and there are no other significant industrial sources in the immediate locality.

An emission stack associated with the main boiler plant at the Mine is included in the monitoring regime and has not revealed any unacceptable emissions that would pose significant risks to either the environment or site users. Road sprinkling is carried out to prevent windblown dust. At the time of the site visit, the entire site was covered with a few centimetres of snow, which did not display evidence of any dust deposition. Air quality monitoring is carried out weekly in the sanitary protection zone.

Emissions of combustion gases resulting from underground explosions and particulates from the underground mining activities will be potentially emitted from any ventilation shafts.

Air pollution at Novoshirokinskoye could result from the following processes:

- Exhaust gases from mobile fleet and combustion processes;
- Gases from explosives usage; and
- Localised chemical and solvent volatilisation.

> ***WAI Comment:*** *WAI do not consider that the current activities cause a significant negative impact on air quality. If the boiler plants are converted to run on coal; further studies will be carried out to comply with environmental requirements of the legislation. Emission control at each emission source and at the boundary of the residential area is carried out according to approved procedures*

15.7.3.7 Waste Management – General

General waste management is reasonably well managed. The mine assists with the operation of the local landfill that is utilised by the external local community. Tailings and waste rock arising from the mining operations are well managed.
Scrap metal is stored around the site and sent for recycling.

> ***WAI Comment:*** *In general WAI considers that there is a reasonably efficient waste management system in place at the mine. Hazardous waste management is under the control of government authorities and is performed in accordance with legislative requirements.*

15.7.3.8 Hazardous Materials Storage & Handling

Fuel delivery is subcontracted and the mine maintains a number of oil storage tanks.

Explosives are delivered by road to an explosives magazine remote from the mine site and village. The storage facility comprises a number of semi-underground bunkers designed to withstand blasts and the facility is protected by armed guards. From here, the explosives required for underground use are transported to the mine in sealed containers in dedicated road vehicles and lowered down the shaft to an underground storage area prior to use.

Adjacent to the explosive storage area is a containerised storage area dedicated to the storage of solid sodium cyanide in sealed metal canisters. The containment area is equipped with air conditioning to regulate (summer) temperatures. An outside shower is provided for emergency use.

The dangerous goods storage area is located due east of the village and was sited taking into consideration the prevailing wind direction. The facility has to be inspected by external Environmental Specialists from the State on a quarterly basis.

> ***WAI Comment:*** *Contractors are made aware of the company policy and their responsibilities with respect to materials storage handling and recycling They are fully conversant with the Company protocols clear lines. Clear lines of responsibility, in the event of an incident, such as a cyanide or fuel spill, exist and emergency contingency planning included for spillage clean up. It is noted that the location of the hazardous and dangerous materials store means that both explosives and cyanide is transported on public road adjacent to areas where people live (e.g. the shift camp). Transportation of hazardous goods is effected by a permit issued by the authorised government agency with special measures indicated which need to be taken in the event of, for example, a cyanide spillage.*

15.7.3.9 General Housekeeping

General housekeeping at Novoshirokinskoye mine is considered to be good, with most open areas and operational buildings maintained in a tidy and orderly manner. The recent age of many of the mine buildings facilitates this aspect.

15.7.3.10 Fire Safety

A fire protection plan has been developed for Novoshirokinskoye mine, this would appear to be comprehensive and cover both above and below ground facilities. The plan covers fire drills and training and the company ensures that contractors are included.

> ***WAI Comment:*** *The fire management systems appear appropriate to the size and nature of the operations.*

15.7.3.11 Security

Security at Novoshirokinskoye mine is provided by Group 4 Security (G4S), a leading expert in mine security. Security is in place on all internal roads/site accesses used during the visit. The security service is provided round the clock.

> ***WAI Comment:*** *It was noted that security was in attendance at site. There is a special register at a secure checkpoint for visitor registration and issue of temporary passes.*

15.7.4 Permitting

15.7.4.1 ESIA/OVOS

An OVOS have been produced for Novoshirokinskoye mine that is in accordance with the requirements of Russia legislation.

15.7.4.2 Environmental Permits and Licenses

OAO Novoshirokinskoye Rudnik Mine JSC holds the right to utilise polymetallic ores under the terms of a licence which is valid for 20 years from 2004. A water abstraction licence and discharge for potable water is in place.

> ***WAI Comment:*** *The WAI team has viewed a number of mine licences and permits and considers the mine licences to have been obtained in line with National requirements.*

15.7.5 Environmental Management

15.7.5.1 Environmental Management Staff & Resources

Environmental management is evolving rapidly at Novoshirokinskoye Mine. Currently there is no separate Environmental Department with Stepanvna Guseva, the Senior Environmental Engineer reporting directly to the H&S Director. The central analytical laboratory employs two assistants responsible for environmental monitoring.

15.7.5.2 Systems and Work Procedures

Environmental effects need to be examined and integrated into all existing work procedures at the Mine. Presently, this has not yet commenced and should be seen as a priority for the Mine.

15.7.5.3 Environmental Monitoring, Compliance & Reporting

A programme of monitoring has been agreed with the regulatory authorities although it would appear that in many cases formal transmission (reporting) to the authorities is not required.

> ***WAI Comment:*** *The scope of the environmental monitoring programme has been developed in line with national legislative requirements. Maximum permissible discharge limits cover all releases from the project and are enforced in current environmental legislation. Groundwater monitoring occurs around the TMF. This includes quarterly chemical and bacteriological analysis.*
>
> *WAI notes that the mine has committed to a weekly monitoring programme needed to establish baseline data for the sanitary protection zone at two locations in the sanitary zone for noise and air quality (NOx, SOx, CO and Dust) during 2010-11. Russian legislation is developed with mandatory health risk assessment for air emissions. WAI considers that the aerial emissions associated with the Mine, processing plant boiler house are relatively insignificant and minimal impact on health risk is expected.*
>
> *WAI recommends extension of monitoring to include ARD and leaching tests.*

15.7.5.4 Emergency Preparedness & Response

Emergency Response plans include for all potential abnormal situations, including catastrophic failure at the TMF and cyanide spillage. WAI considers that the current response planning should be regarded as a Work in Progress.

15.7.5.5 Training

The mine is committed to the implementation of Kazzinc Corporate policy of Environment and training matters.

16 KAZZINC SMELTER

16.1 Introduction

The principle metallurgical processing facility is located at Ust-Kamenogorsk and comprises a number of integrated individual plants including;

- Conventional lead smelter;
- Conventional electrolytic zinc refinery;
- Precious metal refinery;
- Existing acid plants;
- ISAMELT™ Copper furnace, converters and electrolytic tank house (under construction); and
- Dual contract absorption technology acid plant (under construction).

At Ridder the metallurgical processing facilities comprise a conventional zinc refinery, using essentially similar technology to that in the zinc refinery at Ust-Kamenogorsk, and an (existing) acid production facility. Until recently a conventional lead (Pyrometallurgical) processing plant with an annual production capacity of 15ktpy was located at Ridder alongside the zinc refinery. However, it is understood that this is currently mothballed and not in use and was not visited as part of the October 2010 site visit.

16.2 Provenance of Ores Treated

The Metallurgical Complexes operated by Kazzinc treat primarily ores produced at Kazzinc mining operations. Historically, installed production capacity is greater than that required just to treat internally produced concentrates and ore has been accepted from 3rd parties. The current situation in this respect is as follows:

16.2.1 Zinc

Production of refined zinc at the two zinc smelters (i.e. At Ust and Ridder combined) has been at 300kt since 2008. Of this total approximately 240,000t is derived from Kazzinc produced ore. Kazzinc continues to expect to purchase up to 130kt of zinc concentrates with grades of up to 45% zinc from external sources. Hence around 60kt of refined metal is expected to be produced from concentrates purchased from 3rd parties.

16.2.2 Lead

Kazzinc has the capacity to smelt 380kt of the lead concentrates. Between its own lead concentrates, and precious concentrates which will be processed through the lead smelter, Kazzinc will utilise approximately 280kt out of 380kt capacity. This means that Kazzinc is able to buy 100kt of lead concentrates and secondary lead feed together. The average content of this feed is approximately 50-60% Pb, so Kazzinc is going to produce on average 55kt of the refined lead from purchased concentrates.

16.2.3 Copper

After fully commissioning the Cu-Isa-smelt furnace and tank house, Kazzinc expect to have the capacity to produce 87.5kt of refined metal. Approximately 50kt of metal will derive from Kazzinc Cu-concentrates, an additional 6kt will continue to result from Cu-cakes produced as a by-product of lead smelting and 4kt from zinc smelting making a total of 60kt of refined copper from Kazzinc produced ore. From 2012, Kazzinc expect to purchase 50kt of concentrate with an average copper content of 20% Cu and by 2015 Kazzinc intend to purchase up to 137.5kt concentrates giving the ability to produce an additional 27.5kt of refined metal from concentrates produced by 3rd parties.

16.3 Ust-Kamenogorsk Metallurgical Complex Technology

16.3.1 Existing Lead Smelter

16.3.1.1 Background

Currently, lead is produced at the Ust-Kamenogorsk metallurgical complex (Ust) using a traditional flow sheet involving feed preparation, sintering, smelting in blast furnace refining, and lead bullion sales. Refined lead output is projected to be over 109kt in 2010, which is up significantly from the production level of 79,494t in 2009. The tonnage of lead produced in 2009 was dramatically lower compared to the previous few years (production between 2002 and 2008 was never lower than 87kt with a high of 104,507t in 2002). A lead recovery of >94.6% (lead in feed reporting to lead bullion) is anticipated in 2010. Recoveries have remained consistent between 2001 and 2010 varying between 94.01% and 95.49%. However, it should be noted that if lead recovery from by-products downstream is included overall total recovery of lead is in the region of 98-98.5%. Approximately 1,225 people are employed in the lead production section of the Complex.

16.3.1.2 Ore Preparation

Ore arrives at Ust by rail in large bags. It is offloaded from rail trucks to a feed storage area from where ore is blended to provide a consistent feed grade to the sinter plant. Three ore stockpiles of approximately 3kt each are kept on site, each containing lead sulphide concentrate (27-70% PbS) together with smaller stockpiles of materials including; fluxes, limestone, silica and iron oxide (to control slag viscosity). Coke is added from a separate stockpile prior to feeding to a traditional Dwight-Lloyd updraft sintering machine.

16.3.1.3 Sintering

A traditional updraft sinter process is used in the sinter plant with oxygen enriched air. An oxygen plant capable of producing up to 15,000m^3/h, of oxygen serves the lead and zinc plant at the complex.

The sinter plant has a maximum rate of 150tph (usually operating at between 130-150tph) of feed comprising ore plus all flux and coke. Heavy fuel oil (HFO) is used to supply heat to the pre-combustion zone of the sinter plant. The sinter machine operates semi-continuously with a planned 2hr maintenance shut down every Thursday. SO_2 containing off-gas with sulphur dioxide (SO_2) contents of > 4-5% goes to the 'Haldor Topsoe' acid plant for conversion to sulphuric acid (H_2SO_4).

Sinter lumps with a particle size of larger than 100mm are regarded as "coarse sinter" and sinter with a particle size of 0-30mm as "fine sinter". Sinter fines are cooled and returned via milling to the feed preparation area. Sinter forms the feed to the lead blast furnace and is transferred at a nominal temperature of 250-350°C. Exhaust from the end stage of the sinter plant goes via bag filtration to atmosphere.

16.3.1.4 Blast Furnace Section

There are 3 traditional blast furnaces within the lead plant, 2 of which are operational at any one time and one of which is held as a reserve. One of the blast furnaces receives lead sinter feed and zinc bearing (lead) slags. The zinc content is fumed with the collection of zinc oxide dust and molten lead bullion (95% lead purity) tapped to ladles which are transferred to the refining section. The second operating blast furnace is used for short duration re-melting of recycled lead products. Any copper content is removed as a matte (molten sulphide phase) which is sent to traditional (small scale) copper convertors to process to blister copper. It is assumed that once operational this will be carried out in future at the Pierce-Smith Convertors installed adjacent to the Cu-ISASMELT™ furnace. Currently however, there are 4x12t capacity convertors (2 operational, 1 in reserve and 1 undergoing maintenance). The average copper content of the matte is reported to be 40% and the copper grade of the blister copper that is the product of the convertors is 97% Cu (blisters weighing approximately 1 tonne are cast).

Zinc (from zinc bearing lead slag) is removed as a fume of zinc oxide (ZnO) by blowing air and coal dust into the blast furnace. The off-gases (at around 1,000°C) go via a waste heat boiler to recover energy with the zinc rich dusts collected at around 250°C.

Wearing of respirators is mandatory in the lead plant and the current occupational hygiene standards are maintained at a maximum of 4% dust in the atmosphere with a limit of 0.01% Pb content.

Photo 16.1: Tapping Lead Bullion from Lead Blast Furnace

16.3.1.5 Refining

Molten lead bullion (95% Pb) is transferred by ladle to the refining kettles. Principal impurities (not in strict weight percentage order) comprise Cu, Sb, As, Ag, Au, Bi, Se and Tl. All are recovered separately and with the exception of Arsenic sold as products.

Lead is charged to the first refining kettle at approximately 800°C and dross removed. The molten bullion is then charged to a second kettle and allowed to cool to 400°C over 24hrs. Sulphur is added during a decopperisation stage (ca 6hr) with copper removed as a sulphide. In the same kettle tellurium is removed during a vitrolisation process. The tellurium rich material is sent for alkaline leaching to produce ultimately elemental tellurium. Arsenic, tin and antimony are removed in a Harris Furnace at temperatures between 550 and 700°C.

Desilverisation is performed in the next stage of refining where silver and gold is removed as a silvery crust. This material is sent to the Precious Metals Refinery for recovery of pure silver and gold. During the last refining stage debismuthisation occurs by an oxidation process with the high purity lead bullion sent to one of two casting machines (induction furnaces), each of which is capable of producing up to 600tpd lead ingots. The ingots are placed onto pallets and loaded by forklift truck directly onto rail wagons for export.

> ***WAI Comment:*** *Currently a traditional lead circuit is adopted, sintering – blast furnace smelting – refining. The actual process metallurgy is rather complex and the circuit has been developed over the years to ensure maximum recovery of by products. It is noted that during 2011 an alternative front-end lead extraction process will become operational and ultimately the existing sinter plant and blast furnaces will become redundant. WAI believes that the current operation is well understood and well managed. Dust extraction could be improved in the ore handling section. WAI considers that the sinter plant and blast furnaces are well managed and that there is little scope for further process optimisation. Likewise the (current) operation of the refining section is well understood and further optimisation is probably not viable. Although it is envisaged that the refining section will remain "as is" following commissioning of the ISASMELT™ furnace there will inevitably be subtle metallurgical differences which will require investigation (and time) to understand complete and allow optimisation to the same extent as present.*

16.3.2 Zinc Refinery

16.3.2.1 Introduction

The zinc refinery at Ust uses conventional zinc roasting leaching purification process flow-sheet. This flow-sheet is essentially replicated at the zinc refinery at the Ridder Metallurgical Complex. The existing refinery at Ust-Kamenogorsk was constructed in the early 1960's as an upgrade and extension to an earlier (now completely decommissioned and demolished) zinc refinery. The plant has been gradually expanded and improved (installation of computer control, installation of modern fluidised bed roasters using oxygen enriched air etc) and has produced >190kt salable zinc per year (pure metal and zinc alloy) since 2008. Presently almost 1,100 people are employed in the zinc production section of the Complex. High levels of overall zinc recovery to final product (>98%) are achieved at the plant.

16.3.2.2 Feed Preparation

Concentrate is delivered to Site by rail and off-loaded into charge bins from where it is sent by conveyor to crushers and the feed hoppers to the fluidised bed roasters.

16.3.2.3 Roasting

Roasting is performed in 4 fluidised bed roasters. Three have a diameter of 5.5m and are 14m high and one is slightly larger with an effective volume of 42m³. The three standard sized roasters can produce between 400-450tpd calcine whereas the larger reactor can produce >500tpd calcine. Only two roasters are operational at any one time, one is kept in reserve and one is usually undergoing planned maintenance. The roasting step takes place at a nominal temperature of 980°C and utilises oxygen enriched air (25-35% O_2).

Presently all charging is computer controlled from a central control room and feed materials are maintained so that lead in feed is at or below 2% and a maximum concentration of 0.06% arsenic is tolerated.

Off-gases from the roasters are cleaned of particulate using electrostatic precipitators (ESP's) to remove particulate levels to below 200mg/m^3 and the SO_2 rich gas delivered to the existing single contact acid plant.

Zinc slurry, zinc rich filtercake and some slag material is processed in one of two Waelz kilns. The smaller of the two kilns is 40m long and has an internal diameter of 3.6m and the larger is 60m long and has a diameter of 4m. These kilns can produce 185 and 240tpd of calcined material. The kilns use coke from Zarinsk, Russia, as primary fuel and the heat requirement is quoted at 692MJ/tonne feed. There are essentially three sections in each kiln. The first (10-15m) is where drying takes place, followed by the reaction zone (ca. 20m) and finally clinker formation. Temperatures in these three sections are 750-850°C, 1,100-1,250°C and 1,100°C respectively. Off-gas from the kilns goes to a mechanical bag filter where solid zinc oxide (ZnO) is removed and sent back to the leaching process, prior to venting to atmosphere (note: these gases are largely sulphur free).

The smaller of the two kilns was installed in 1971 and the larger is more modern, installed in 2006. Dust losses are reported as 10%. However, during the site visit fugitive emissions were noted from the (small) kiln and it is not known whether this figure includes fugitive emissions.

> ***WAI Comment:*** *The ore preparation and roasting sections of the plant can be considered to be entirely conventional in design and operation and similar to other zinc smelters around the World. Manning levels would appear to be above average for the zinc plant as a whole. Recent innovations have included computer control and optimisation of the use of oxygen enriched air and WAI believe there is little to optimise metallurgical performance further in this section.*
>
> *WAI believes that investigation of fugitive dust losses from the kilns, especially from the older, smaller kiln is warranted to establish whether the metallurgical and environmental performances can be improved.*

16.3.2.4 Leaching Section

Roasted ore is transferred by elevated conveyer via ball mills to the leaching plant where oxidised zinc ore is leached using spent electrolyte solution (essentially sulphuric acid) at a pH of around 1.5. Leaching is carried out in agitated tanks (two banks of 4 agitators) and during the leaching process pH gradually increases to around 4.5 when the solution is pumped to 18m diameter thickeners. Zinc solution containing 130g/l zinc in solution is pumped at a rate of 6,500m³/day to the refinery with a copper/cadmium residue sent to other parts of the Complex for further processing. The zinc solution is filtered using a Difenbach press filter.

> ***WAI Comment:*** *The leaching section comprises conventional zinc processing technology and is essentially similar to most other electrolytic zinc plant around the World (including the plant at Ridder) and can be regarded as an essentially optimised circuit.*

16.3.2.5 Refinery

Zinc solution at a temperature of around 45°C is mixed with spent electrolyte solution at a ratio of 1:15 to optimise the zinc concentration for the electrolysis process, pumped to vacuum distillation equipment and ultimately delivered to the tank house (commissioned in 1967). The tank house comprises a total of 282 cells arranged in 4 banks of cells connected in series. Anodes in the cells are made of zinc metal (containing 1% silver) and cathodes made from aluminium. Busbars supply direct current at 22 amps and 99.99% purity zinc is plated onto the cathodes. An average of 1,200kW/t zinc plated is consumed in the process. The current electrolytic circuit comprises relatively modern polymer-concrete cells that replaced the previous lead lined cells (over the last 5 years). Other improvements have been made to the electrolytic cooling system, with cooling now effected by Hamon air cooling towers.

Up to 525tpd cathode zinc is removed manually and transferred by fork lift truck, 50kW induction furnaces of which there are 9, for re-melting and casting into high purity zinc ingots. At this stage aluminium can be added and a zinc-aluminium alloy produced if required. Production of these speciality alloys can be up to 4kt/month. Ingots are cooled batched on pallets and loaded onto railway trucks for export.

> ***WAI Comment:*** *The electrolytic process employed at Ust-Kamenogorsk can be considered as conventional technology. The fundamental process is little changed since its introduction in the 1950s' although recent improvements have been made to the electrolytic cell design and materials of constructions and cooling efficiencies. Overall there would appear to be little scope for significant improvements to process efficiency and the plant is considered to be operating at, or close to, optimised levels.*

16.3.3 Precious Metals Refinery

Ust-Kamenogorsk Metallurgical Complex operates a precious metals refinery producing 99.99% silver and 99.99% gold ingots. Estimated 2010 outputs of refined silver and gold are 171,514kg and 18,480kg respectively.

A schematic process flow sheet for the Precious Metals Refinery is shown in Figure 16.1.

High levels of recovery for silver and gold are reported, with the principal being to final discard slag. Minor variations are apparent but over the last few years have been relatively stable at around 12% or less.

All refined silver and gold is exported from site, up to three times per month, by rail using dedicated (Kazzinc) bullion wagons.

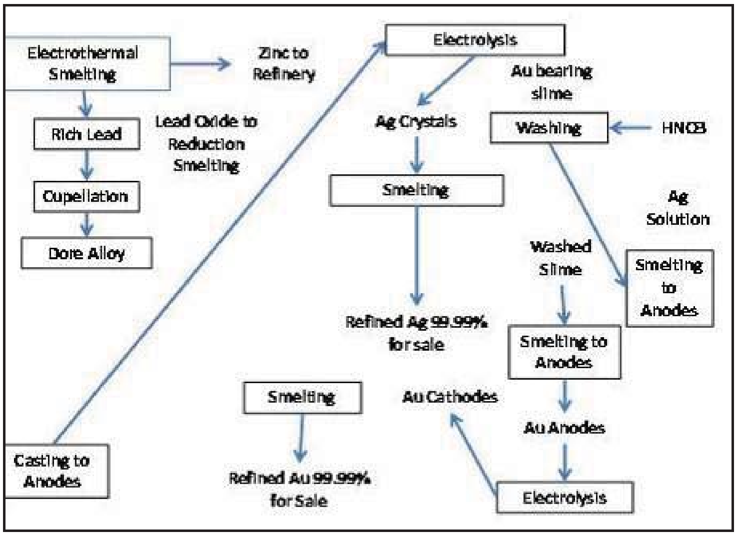


Figure 16.1: Precious Metals Refinery Ust-Kamengorsk Schematic Process Flow Sheet

The electrolytic cells used to produce silver crystals are protected by Kazzinc Patent No. KZ(A) 5555 (filed 15.12.1997) and have the ability to produce 4 nines grade (i.e. 99.99%) silver in a single stage. The silver is removed from the electrodes and smelted to form ingots (28-32kg in weight). Facilities exist to enable production of powered metals should this be required by the customer, although the vast majority of production is as ingots.

Security at the metallurgical complex is provided by G4S however, State regulations dictate that security within the previous metals refinery is provided by the State owned security company Kuzet and movement into and out of the buildings is controlled. As would be expected, housekeeping in the plant is excellent.

The refinery comprises 12 electrolytic cells and was commissioned in 1991 with a capacity to produce between 45-60t of silver per month. The gold plant at the refinery first produced refined gold in 1992.

Photo 16.2: 99.99% Silver Ingots, Ust-Kamenogorsk

Photo 16.3: 99.99% Gold Ingot for Export

> ***WAI Comment**: The precious metals refinery is well managed with good process efficiencies. The metallurgy has been optimised over a number of years and the ability to precipitate silver at the required grade in a single stage is impressive. WAI considers there is little potential for enhanced gold or silver recoveries*

16.3.4 Existing Acid Plants

Two different acid plant technologies were operational in October 2010:

- Single contact process using zinc roaster gases; and
- Wet Catalytic (also known as Wet gas Sulphuric acid – WSA) using Danish Haldor Topsoe technology with blend of off-gases from lead and zinc processing.

The single contact plant was commissioned originally in 1955 (although it should be noted that the oldest parts of the current plant are approximately 10 years old) and has a limit of 0.3%SO_2 in the inlet gas. Any gases below this concentration of SO_2 are vented directly to atmosphere. In 2004, a 895tpd wet catalytic (Haldor Topsoe) Danish technology acid plant was installed capable of treating 125,000Nm3/h gases with SO_2 contents as low as 0.06%SO_2 and with a maximum concentration at the inlet to the reactor of 6.5%. Installation of the Haldon Topsoe plant reportedly decreased SO_2 emissions into the atmosphere from the complex by an estimated 40%.

These traditional technologies for sulphur capture and acid production will be supplemented by the use of a dual contact "state of the art" acid plant designed by SNC Lavalin which will become operational in 2011. This plant is designed to treat (all) high sulphur content off-gases produced by the ISASMELT™ copper furnace (Off-gas from the ISASMELT™ lead furnace will be treated in the current Haldon Topsoe plant).

An environmentally significant impurity in the off-gas (once particulate matter has been removed) is mercury and the relatively small amounts of elemental mercury are removed in filter cakes and sold as a by-product. Off-gas for both plants should be relatively particle free and it would appear that the gas cleaning performance (i.e. gas filtration) standards for SO2 containing gases are established by the acid plant minimum requirements.

All acid, produced by whichever route, is either re-used within the metallurgical processes on site or piped to rail tanker for export.

> ***WAI Comment****: Kazzinc deploys conventional technologies to scrub sulphuric gases to form sulphuric acid. The single contact plant cannot reach the scribbling efficiencies of the existing (Haldon Topsoe), stage process or the dual contact plant currently under construction.. Adding a second stage to the single contact plant would result in higher scrubbing efficiencies although the overall effectiveness and cost implications would need to be understood further before such a strategy could be recommended.*

16.3.5 ISASMELT™ Lead Furnace

16.3.5.1 ISASMELT™ Process

ISASMELT™ is a bath smelting process utilising lance technology (ISASMELT™ lance). The lance is inserted into a molten slag bath contained within a stationary, refractory lined furnace (ISASMELT™ Furnace). The injection of air, or oxygen enriched air, through the lance results in the formation of a highly turbulent molten bath. Feed materials (ore, sinter etc) falling into the molten bath from above can react rapidly resulting in extremely high productivity for a relatively small bath volume (high intensity smelting rates).

The ISASMELT™ process was first developed by Mt Isa Mines (MIM) and developed into a commercial scale during the 1980s' and 1990s' at MIM's smelting operation at Mt. Isa in Australia. Commercial operations now exist in at least 8 different countries, including applications for secondary lead, secondary copper, copper concentrates and lead concentrates. Further applications of the technology for tin smelting have been investigated. Key features of the ISASMELT™ process report in the literate include:

- Relatively simple design;
- Flexibility of fuel (coal, coke, natural gas);
- Long lance life;

- Ease of operation;
- Minimal feed preparation requirements;
- High specific smelting rates;
- Low dust production;
- Small size (footprint);
- Efficient gas capture; and
- Ability to utilise oxygen enriched air.

16.3.5.2 Other ISASMELT™ and similar Lead Plants

Although the ISASMELT™ technology was originally developed to smelt Mt Isa lead ores the original pilot plant was decommissioned in 1995 and currently Mt Isa does not employ an ISASMELT™ furnace.

Secondary lead smelting has been carried out successfully in an ISASMELT™ furnace by Britannia Refined Metals at Northfleet, UK, since 1991 and at Metal Reclamation Industries in Palau Indah, Malaysia. A lead ISASMELT™ plant was designed and commissioned in 2005 by Yunan Metallurgical Group, Qujing, Yunnan Province China, to produce 80ktpa of lead from concentrates. A lead blast furnace is included in the process flow sheet and the lead ISAMELT™ furnace effectively replaces the sinter plant of a traditional lead smelter. AUSMELT™ is essentially a similar lance technology to ISASMELT™. Recyclex operates an AUSMELT™ furnace for primary and secondary lead smelting at Nordenham and Hindustan Zinc operate a primary lead AUSMELT™ furnace at its smelter in Chanderiya.

16.3.5.3 ISASMELT™ Furnace – Ust- Kamenogorsk

A lead ISASMELT™ furnace is currently under construction at the Ust-Kamenogorsk Metallurgical Complex. The furnace has been designed by Xstrata (owners of the ISASMELT technology) and construction is being supervised by engineers from Xstrata. Commissioning is scheduled for 2011. Total investment in this furnace is estimated at over US$100M.

Blended ore will be fed to the lead ISASMELT™ furnace at a maximum feed rate of 150tph and the main plant has been designed to produce 140ktpa of lead metal. The ISASMELT™ facility will effectively replace (eventually) the sinterline and it is noted that the blasé furnaces will remain operational after the ISASMELT™ furnace has been commissioned. Product from the ISASMELT™ furnace will be fed to a hot briquetting plant and will form feed to the blast furnaces.

> ***WAI Comment:*** *WAI considers that ISASMELT™ technology is well proven and that there are commercial examples of the technology operating to produce lead. However, there have been difficulties in the past in attaining economic viability. WAI notes that the furnace design, the supervision of construction and training of key operatives is all under the management of Xstrata and considers this to be important. Operation of the ISASMELT™ furnace will enable more efficient collection of off-gas and conversion of SO_2 to H_2SO_4 resulting in reduced emissions of SO_2 to atmosphere. The feeding of hot-briquetted feed and operation of the lance bath should result in (significantly) improved particulate emissions compared to the levels produced currently during the sintering process.*
>
> *ISASMELT™ is considered an appropriate technology for implementation at Ust-Kamenogorsk and has the potential to reduce manning levels and produce lead more cost effectively than in the current process flow sheet as well as improve environmental performance It is understood that commissioning of the lead ISASMELT™ furnace is planned for the third quarter 2011.*

16.3.6 ISASMELT™ Copper Furnace and Tank House

16.3.6.1 ISASMELT™ Copper Process

A number of ISASMELT™ furnaces for the production of copper from both primary ore and secondary sources have been installed and are operational or are under construction or waiting commissioning around the World. The first furnace, a 700ktpa copper smelter treating primary copper concentrates was installed in 1992 by FMI Miami in Arizona, USA. This was followed rapidly by a 1Mtpa furnace operated by Mount Isa Mines Ltd in Australia. Subsequently furnaces have been commissioned by Sterlite Industries in Tuticorin, India (500ktpa); Yunnan Copper at Kunming, China (800ktpa); Mopani Copper Mines at Mufulira, Zambia (650ktpa); Southern Peru Copper at Ito, Peru (1.2Mtpa); Yunnan Copper at Chambishi, Zambia (350ktpa); and Yunnan Copper at Chuxiiong, China (500ktpa). In addition to the furnace under construction at Ust-Kamenogorsk, other Cu-ISASMELT furnaces (up to 1.36Mtpa throughputs) are under construction in China, India and Peru.

16.3.6.2 Cu-ISASMELT™ Furnace and Ancillary Equipment at Ust-Kamenogorsk

A US$400M investment has been made to construct an ISASMELT™ furnace (and ancillary equipment) and electrolytic tank house. The plant was largely complete at the time of the site visit (October 2010) and it is understood that commissioning is scheduled for the first quarter of 2011. Off-gases from the smelting section will be cleaned using Electro Static Precipitators (ESP) to reduce the particulate content to below 50mg/m^3 and delivered to a new acid plant which is also under construction and which has been designed to be commissioned alongside the ISASMELT™ furnace (cold commissioning planned for end February 2011). Once fully operational this facility is expected to produce up to 87.5kt of refined copper.

The ISASMELT™ furnace is fitted with waste heat boilers to recover heat. Molten product (matte) from the furnace is transferred to conventional Pierce-Smith Rotary Coverters where blister copper is produced. Blister copper is tapped to an anode furnace for casting into (impure) copper anodes for further refining in the electrolytic tank house.

> ***WAI Comment:*** *WAI considers that the choice of ISASMELT™` technology is most appropriate. There are many examples of successful operation of this process around the World and the technology choice is considered low risk. The technology offers the potential for Kazzinc to become a relatively low cost copper producer and offers the potential to toll treat third party ores to supplement production from internal ores. In this respect the ability to treat arsenic bearing concentrates may be significant as there are few sites that can treat high arsenic concentrates and therefore may attract a premium. WAI believes that the significant involvement of Xstrata staff in the training of Kazzinc operators is important and should assist commissioning and start up of commercial operation.*
>
> *All other unit operations in the new plant which is under construction can be regarded as conventional and represent a low level of technical risk. The design of the smelting operation allows for flexibility and ease of operation and incorporates mechanisation wherever possible. The involvement of outside staff in the training of Kazzinc operators is important and should facilitate successful commissioning and operation.*

16.3.6.3 Copper Tank House

As part of the new investment a traditional, modern copper tank house comprising 264 electrolytic cells in 6 sections is under construction. Cold commissioning is underway and the whole facility should be operational in the first quarter of 2011.

> ***WAI Comment:*** *The tank house is a conventional design and WAI do not consider that there are any significant risks associated with the technology. Kazzinc believe that commissioning should be relatively straight forward and WAI has no reason to doubt this.*

16.3.7 (New) Dual – Contact Absorption Process Acid Plant

Kazzinc has invested approximately US$150M in a new acid plant designed by SNC Lavalin using a dual-contact absorption process. This plant will meet all off-gas from the copper ISASMELT™ process.

The plant has been subjected to rigorous appraisal and (together with the copper smelter) has an OVOS which has been reviewed and approved by the Ecological Committee of the Republic of Kazakhstan.

Extensive off-site training has been provided by SNC Lavalin for Kazzinc employees. The new acid plant underwent cold commissioning at the end of 2010 and is designed to become operational once the copper ISASMELT furnace commences operation.

> ***WAI Comment:*** *WAI notes that the new acid plant represented state of the art technology and should convert all SO_2 to H_2SO_4 in most of the plant operations.*

16.4 Ridder Metallurgical Complex Technology Description and Comment

16.4.1 History

The Ridder Metallurgical Complex (RMC) was built to incorporate separate lead and zinc production areas. However, the lead plant is currently mothballed and it is understood that the plant is unlikely to work again, especially as a new ISASMELT™ lead furnace is under construction at UST-Kamenogorsk.

Construction of a zinc plant to process ore from Tishinskiy Mine started in 1959, with the roasting section completed in 1963 and the 1st stage acid plant commenced operation in October 1964. The first zinc (metal) was produced on 27 March 1966. In 1981 a hydrometallurgical plant, comprising high temperature leaching of zinc concentrates in acid with iron removed in the form of jarosite was commissioned, and in the 1990s' the process was changed to the current process flowsheet.

16.4.2 Current Process Flow-Sheet

Zinc concentrates from the Ridder Mining and Concentrating Complex (Tishinskiy Mine, Ridder-Sokolniy Mine and Shubinskiy Mine) are treated at RMC in a conventional roasting (fluidised-bed) – leach circuit and ore from the Shaimerden Deposit is treated at RMC in a Waelz Kiln – leach circuit. A simplified, schematic process flow-chart is shown in Figure 16.2. The design capacity for the current zinc plant is 105,00tpy of saleable zinc (since 2004 sales of zinc metal from RMC have exceeded 105ktpa with sales of 110,847 in 2009).

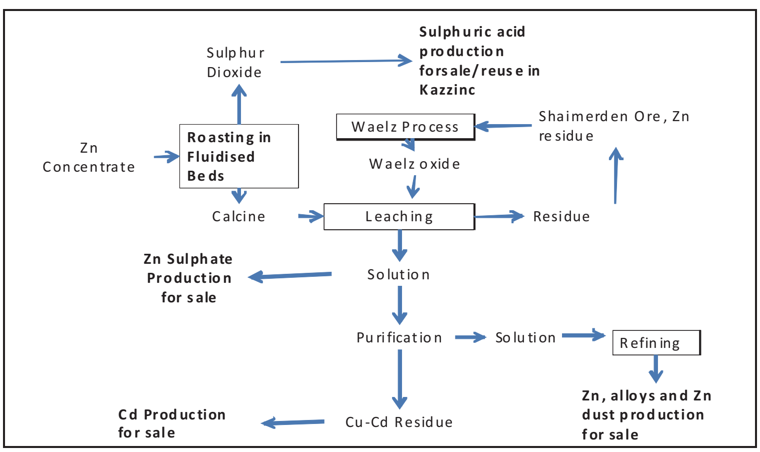


Figure 16.2: Schematic Process Flow Sheet – Showing Zinc Production Process at RMC

16.4.2.1 *Roasting*

Zinc sulphide ores are treated in one of 4 fluidised bed roasters. These have a hearth area of 31-32m² with a diameter of 6-7m. At any one time there are 2 working fluidised bed roasters one, held in reserve and one undergoing maintenance. The fluid-bed roasters are refractory lined and are relined every 4- 5 years after approximately 3,000 operating hours.

16.4.2.2 *Waelz Section*

Three 5m diameter Waelz kilns (70 and 75m long) are present at Ridder. The newest kiln was installed after a US$30M investment in 2008. The kilns can each produce up to 77t of Zinc per day and are equipped with waste heat boilers and a bag-house to collect dust with particulate emissions to atmosphere reported at 0.02g/m³. The kilns are inclined at 2.5% and rotate at up to 1rpm. The additional investment in 2008 was primarily so that Shaimerden ore (and zinc residues) can be processed. A schematic process flow sheet is shown in Figure 16.3.

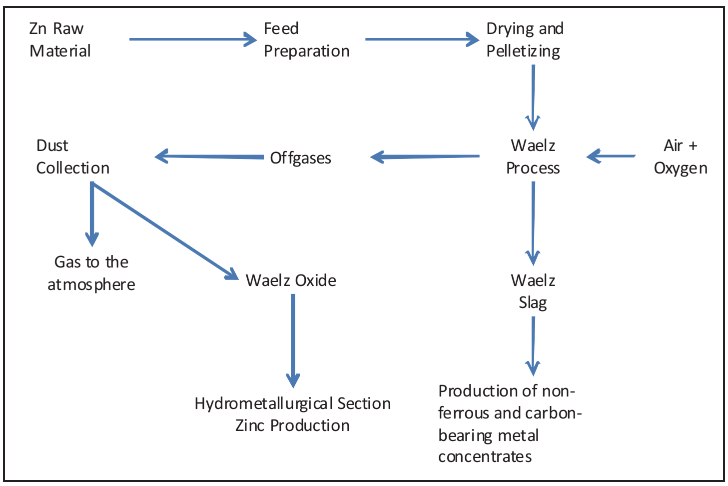


Figure 16.3: RMC Waelz Section: Schematic Process Flowsheet

16.4.2.3 *Hydrometallurgical (Leaching) and Electro winning Sections*

Waelz Oxide, or calcine, produced in the fluidised bed roasters is leached using sulphuric acid in standard agitated tanks (130-150g/l Zn in solution) and fed via thickeners and a Dieffenbach filter to electro winning cells. Zinc cathodes are produced by electrolysis of the solution. The hydrometallurgical/ electro winning sections of the RMC are essentially similar to those at the Ust-Kamenogorsk plant. The principal difference between the two metallurgical complexes is that at Ridder pneumatic agitators are employed whereas mechanical agitators are used at Ust. Three vacuum evaporators are used in the mixing/cooling section.

Solution containing up to 150g/l zinc is transferred at a rate of 125m³/h from the leaching plant to the electrolytic cells. Zinc is predicated as cathodes with spent electrolyte solution (essentially H_2S0_4) recycled to the leach section. Schematic process flow sheets are shown as Figure 16.4 and Figure 16.5. Zinc metal is produced by the electrochemical decomposition of an aqueous solution of acidified zinc sulphate by the application of a direct current passing through the cells. Aluminium cathodes and Pb-Ag cathodes act as the electrodes. Zinc metal is stripped manually from the cathodes once per day and sent to induction furnaces where high purity ingots are cast. These furnaces operate at 430-500°C at a specific power consumption of 94.5 to 100kW/h. Each furnace has a capacity of up to 40t molten metal. Ingots are produced on a batch basis.

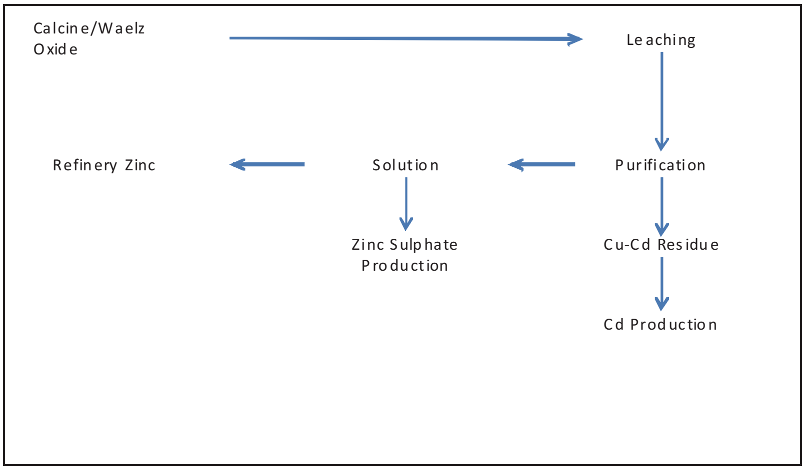


Figure 16.4: RMC Leaching Area: Schematic Process Flowsheet

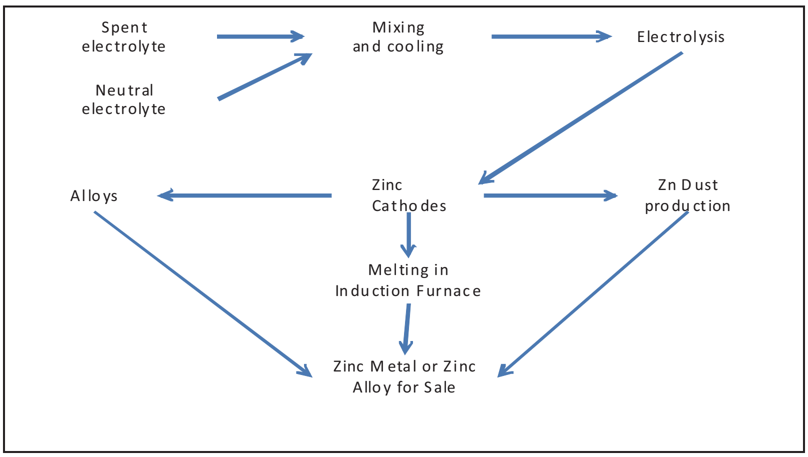


Figure 16.5: RMC Leaching Area: Schematic Process Flowsheet

In addition to high purity zinc metal, specific zinc alloys can be produced in the same induction furnace by addition of high purity alloying metals. Zinc ingots/alloys are exported from the RMC to consumers by rail.

16.4.2.4 Acid Plant

Sulphur dioxide gas (SO_2) produced by the roasting of zinc (sulphide) concentrates is ducted to the acid plant. The acid plant can be considered reasonably conventional and comprises 3 sections.

1) Weak acid washing;
2) Drying; and
3) Contacting (using a Vanadium Catalyst).

The process is based on the reaction $SO_2 + ½\ 0_2 \leftrightarrow SO_3$. The process flow-sheet is shown as Figure 16.6. Product sulphuric acid (H_2SO_4) is pumped to storage tanks from where it is loaded into railway tank wagons for delivery to customers. The plant produces 160-170,000tpa H_2SO_4 with a reported efficiency of SO_2 recovery of 98.7 – 98.9%

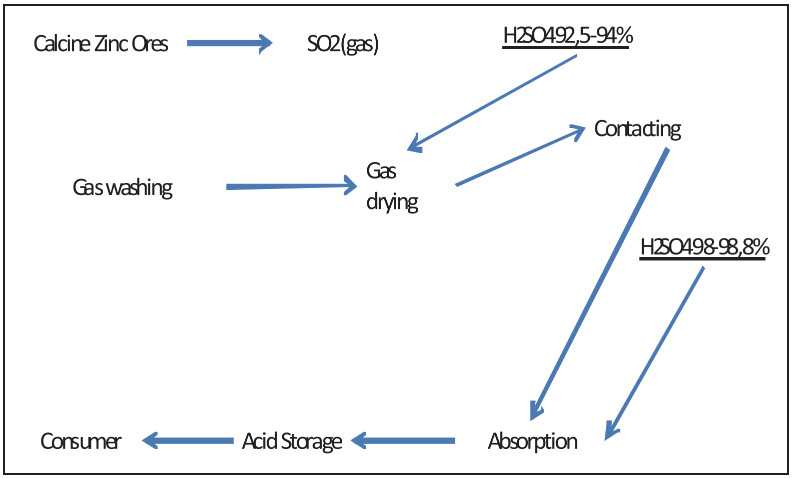


Figure 16.6: Schematic Process Flowsheet, Sulphuric Acid Production, Ridder Metallurgical Complex

16.5 Environmental

16.5.1 Background

Ust-Kamenogorsk is the capital of the East Kazakhstan Region. It is a highly industrialised city situated at the confluence of Irtysh and Ulba Rivers. The city has over 300,000 inhabitants (2010) with over 750,000 people living in the Region. It is served by an International airport and is located on a railway (branch) line with services to Ridder (eastwards) and westwards into Russia. Immediately north-east of Ust-Kamenogorsk the land rises towards the Altai Mountains and the border with Russia. The nearby West Altai National Park was created in 1991.

At Ust-Kamenogorsk metallurgical industries (including the Kazzinc Metallurgical Complex) and supporting engineering industries are important employment providers as well as uranium processing and thermal power generation. The city, befitting its Regional status, contains the administrative centre, university and college establishments and primary healthcare (hospital) facilities for the Region. Industry around Ridder is dominated by the activities of Kazzinc, namely mining, mineral processing and metals production.

East Kazakhstan has an extreme continental climate. Average January temperatures range between -19 and -4°C with average July temperatures between 19 and 26°C. Extremes in the East Kazakhstan Region can reach -40°C and +35°C respectively. Rainfall (precipitation) is relatively low with around 400mm in Ust-Kamenogorsk, although levels are higher in the mountains around Ridder.

Kazakhstan has a complex ethnic mix which explains the Country's religious diversity. Ethnic Kazakhs are historically Sunny Muslims and Islam is the largest faith. However, over one third of the inhabitants belong to the Russian Orthodox faith.

Kazzinc operates corporate integrated management systems, including:

- Quality Assurance (ISO 9000);
- Environmental Management Systems (ISO 14001); and
- Health & Safety Management (OHSAS 18001).

These management systems are somewhat generic at the moment but the desire is to roll out across all sites in the Company. Ultimately, the existing high level documents will need to be supported by individual site specific documents which will detail how environmental issues (and Health & Safety) are managed and performance standards maintained at each operating site.

16.5.2 Ust-Kamenogorsk Metallurgical Complex

16.5.2.1 Current Status – General Comments

The Ust-Kamenogorsk Metallurgical Complex is situated in a highly industrialised sector of the city. Adjacent to the site boundary is a major thermal power station (operated by AES) and a Uranium processing facility. Residential areas and a hospital complex are located just over 1km from the site boundary.

An OVOS covers all the activities at the Metallurgical Complex (including the new plant currently under construction) and establishes maximum permissible concentrations for all significant emissions from the complex. Discussions with environmental managers at the site indicate that no closure plans exist for the complex as these are not required (and would not be considered appropriate) under current Kazakh legislation/permitting.

WAI witnessed a number of individual monitoring results, all of which demonstrated compliance with requirements of environmental and occupational hygiene legislation. Occupational hygiene monitoring and general site air quality monitoring around the Complex is also carried out regularly for a variety of parameters including, NO_2, SO_2, CO_2, Pb, As, Zn and noise and vibration. General air quality testing is carried out around the Complex daily (Monday-Friday) using the Company's mobile testing laboratory.

Particular attention is paid to the occupational hygiene limits for lead and blood lead levels for workers exposed to lead (i.e. those working in the lead section) and are measured twice yearly.

General standards of housekeeping were good around the Complex at the time of the site visit (October 2010), especially considering the amount of construction (and demolition) activities taking place at the time. Major construction projects which were on-going at this time included the Cu-ISASMELT™ furnace and tank house, and the Pb-ISASMELT™ furnace and new acid plant. Regular dust suppression of the internal roadways occurred using dedicated road sweepers.

There is no electricity generation at the Complex and all power is obtained via the National Grid. It is noted however, that Kazzinc operate a hydroelectric power station, the Bukhtarma Hydroelectric Power Station, located on the Irtysh River and are responsible for the upkeep of roughly 650km of electricity transmission lines.

Security at the Complex is provided by G4S. All access to the Complex is controlled through one principal entry which is gated. All vehicles are inspected on entry and on leaving the site. G4S provide all internal site security including the provision of (armed) guards for the Precious Metals Refinery.

> ***WAI Comment**: WAI has not had the opportunity to review the OVOS in any detail but WAI considers that the document has satisfied National requirements.*
>
> *It is believed that Kazzinc has established some form of bond to facilitate closure of adjacent mining areas but it is unclear whether the Ust-Kamenogorsk (or Ridder) Metallurgical Complex is included within the provisions of this bond. WAI would recommend that the provisions of this bond are examined and if necessary extended to include the metallurgical complexes. Ground water samples are obtained from 20 monitoring boreholes across the site on a quarterly basis. WAI have been informed that asbestos materials are not present and that in 2010 an inventory of waste and active PCB materials was made. A National programme of work PCB disposal at a centralised site for the whole of Kazakhstan is under development currently.*
>
> *WAI notes that the new plants under construction (principally the copper and lead ISASMELT™ furnaces and the dual contact acid plant) should improve the overall environmental performance of*

> *the Complex, in particular by improving local air quality (reduced SO_2 and particulate emissions) and improved energy efficiency of copper and lead production.*
>
> *Electrical power is supplied to the Complex from the National Grid and hence, in reality, the electricity could have been generated by any route. However, as Kazzinc operate a hydro-electric plant that supplies an amount of power to the Grid similar to that consumed at Ridder and Ust, it could be argued that the Complex should be regarded as essentially carbon neutral.*

16.5.2.2 Air, Water and Solid Emissions

Air Emissions

Emissions to atmosphere represent arguably the most significant environmental impacts posed by the Complex. Local air quality has been poor historically (both visible emissions resulting from particulates as well as SO_2 emissions) but has improved in recent times partially as a result of environmental improvements implemented by Kazzinc. SO_2 concentrations in air around the city have decreased by almost two thirds between 1997 and present day, although are currently just above the maximum permitted concentration (MPC) limits. It is expected that the completion of the lead smelter upgrade and copper smelter in 2011 will ensure SO_2 concentrations in air decrease to below the MPC levels. The Complex has a large number (114 individual stacks, 6 of which are over 80m in height) of point source (stack) emissions, all of which requiring abatement to a greater or lesser degree. Currently most stacks are equipped with bag filters prior to venting to atmosphere but none are equipped with any form of continuous monitoring for particulates or SO_2. Deviations from normal operation of abatement equipment (vacuum, temperature, gas flow rate) are monitored from the control room by alarms. IS0-kintic air sampling to demonstrate compliance with the maximum permitted particulate emission rate of <5mg/m³ is carried out as necessary.

Commissioning of the new lead and copper plants will result in capture of a greater proportion of the SO_2 produced at the Complex and the commissioning of the new acid plant is designed to coincide with the commissioning of the copper plant.

> ***WAI Comment:*** *WAI considers that SO_2 represent the most significant environmental impact from the Complex. WAI considers that the introduction of new process plant in the form of ISASMELT™ furnaces and a new dual contact acid plant in 2011 will reduce the impact of the complex on local air quality.*
>
> *WAI recognises that the emissions from smelting operations are (under normal operation) relatively stable and predictable. The Environmental Department predicts likely emissions and determines what sampling frequency is appropriate. Emissions monitoring occurs once a month and this frequency can be increased if warranted. Furthermore, the complex utilises an on-site mobile laboratory which can obtain air samples from around the site, several times a day.*
>
> *WAI understands that the Ust-Kaemenogorsk atmosphere air monitoring programme requires a centralised monitoring system of main sources of emissions (stacks) within the city.*

Water Emissions

No water is abstracted currently from surface rivers. All make up water used in processes at the Complex is obtained from boreholes. The complex has a licence to abstract 1,000m3pa from groundwater.

Water at the Complex is used in an essentially closed system (about 6,000m³/h water in circulation at any one time, 4,000m³/h of which needs to be treated and an estimated 2,000m²/h which is not contaminated and not subject to treatment). Discharges to the river (<2% of the water usage) are made periodically after the water has been treated in a water treatment facility where pH is controlled by the addition of "lime milk" and suspended solids precipitated by addition of a flocculent. The water treatment plant was designed to treat 4,500m³/h process water.

Water chemistry is established at the Environmental Analytical Laboratory using principally standard atomic absorption spectrometry (AAS) instruments. Samples of pre and post treatment are analysed on a regular basis for quality control as well as routine analysis of river samples (upstream, downstream and at the discharge point

> ***WAI Comment****: WAI considers that water management is good at the Complex and has not seen any evidence (monitoring results) that indicate that the plant has been responsible for emitting (grossly) contaminated water to the river system.*

Solid Waste Disposal

A number of above ground, solid waste dumps are present within or immediately adjacent to the Complex. These include stockpiles of slags (4 in total) and calcium arsenate (2 in total). The slags include stockpiles of historic (+15year old) arising (estimated at a number of Million tonnes in total) from pre-Kazzinc operations which are believed to remain in the ownership of the State who retains any liability for the stockpile as well as 600kt slag for which Kazzinc is responsible. The slag stockpiles are all uncovered and have not been subject to any closure/remedial operations. It should be noted that the State regards all these stockpiles as "temporary" in nature and they are regarded as "assets" containing potentially recoverable metal values. As such, they are not subject to any National requirements for closure planning. The historic dumps are sources of particulate emissions.

The Ust-Kamenogorsk Metallurgical Complex recovers most useful components from the primary ores. Arsenic is associated with many sulphide ores due to the presence of the mineral arsenopyrite – (iron arsenide sulphide – FeAsS – the most ubiquitous arsenic bearing mineral species) and is present in a number of ores processed at Ust-Kamenogorsk. In theory arsenic could be recovered at the plant as either arsenic trioxide or as elemental arsenic. However, practically there are no economically viable markets for arsenic remaining as the vast majority of its former uses (e.g. in wood preservatives) have been banned in many countries. Hence, for a number of years arsenic has been removed (as calcium arsenate and calcium arsenite from the metallurgical circuits and considered as a solid waste.

There is currently a (filled) capped calcium arsenate and arsenite stockpile and a current (two cell) active calcium arsenate stockpile present on site. The filled stockpile/dump remains in the ownership of the State who retains all liabilities associated with the stockpile. (see Photo 16.4). The stockpile has been capped with a relatively thin layer of clay over a synthetic (impermeable) geosyntheic membrane. Once capped the stockpile was subject to revegetation trials in 2003 and plants cover much of the exposed surface.

Photo 16.4: Historic Capped Calcium Arsenate Stockpile
Showing Instability on Cap

Signs of instability are evident on the slopes of the stockpile (where the clay has slumped) where the membrane has been exposed.

Current arsenic disposal is to a single cell with multi-layered impervious walls with a clay lined base with concrete and asphalt surfaces. The cell is estimated to be half full with a further 18-plus months of disposal volume remaining. Cell construction methods and disposal operation has been with government approval.

Boreholes to the groundwater have been installed up gradient and down gradient of the calcium arsenate and arsenite disposal areas and are monitored regularly. It is reported that no groundwater pollution plume of arsenic has been identified by this monitoring.

Kazzinc internal R&D teams are evaluating and finalising the design of an alternative arsenic disposal method using Scorodite, a hydrated iron arsenate mineral with the chemical formula $FeAsO_4.2H_2O$, which is significantly less soluble in water compared to calcium arsenate. The ultimate plant will have the capacity to produce up to 9,000tpa scorodite but the detail of the disposal containment structures and ultimate location are uncertain at present. These works are at preliminary design status and completion is anticipated in the first half of 2011. It is understood that the primary impetus for this development work was the decision to construct a copper ISASMELT furnace as this will generate significantly more arsenical residues than are produced currently.

> ***WAI Comment***: *WAI notes that the historic slags present on site remain the responsibility of the State.*
>
> *WAI notes that the capping on the historic (State responsibility) calcium arsenate dump would benefit from remedial measures however, it is noted that legal ownership resides with the state.*
>
> *The current calcium arsenate disposal cells will become redundant once the scorodite process that is currently under development is implemented. WAI understands that this is contemplated to be by the middle of 2012. As per project approval the existing cells will require capping and revegetation and should be considered as permanent disposal locations.*
>
> *WAI concurs that scorodite is a more stable arsenic species compared to calcium arsenate or calcium arsenite. Waste tips in Saxony that are many hundreds of years old have been found to contain scorodite and suggest that this species is appropriate for long term disposal of arsenic. It is important to appreciate that scorodite is not completely stable environmentally and good operational control of the process is essential to ensure optimum performance. It is recommended that all process designs are examined in detail before the process is implemented but, in principle, WAI would support the concept and consider disposal of arsenic as scorodite to offer environmental advantages compared to the present system.*

16.5.3 Ridder Metallurgical Complex (RMC)

16.5.3.1 Current Status

The RMC is situated in the town of Ridder approximately 160km east of Ust-Kamenogorsk. Construction of a zinc plant started in 1959 with metal and acid production by the early 1960's. In April 2008 Kazzinc acquired the lead production facilities from Kaz-Tyumen JSC. This plant had an annual production capacity of 15,000t lead but from 2009 production was switched to the metallurgical complex at Ust-Kamenogorsk and the plant is currently mothballed.

Current activities at RMC are restricted to the production of zinc (design capacity 105,000tpa – 2009 production 110,800t) and sulphuric acid (design capacity 480tpd). Sulphuric acid is supplied to external customers as well as used on-site within the metallurgical process. Some materials are delivered to the metallurgical complex at Ust-Kamenogorsk for further reprocessing (Waelz fumes, lead cake).

Environmental challenges at the Ridder Complex are essentially the same as those faced at Ust. However, it should be appreciated that the RMC is significantly less complex than the Ust Complex.

Issues are tackled in a similar way. Aerial emissions are dominated by particulates and sulphur dioxide gases and mitigated in the same manner as in Ust. SO_2 rich gas is cleaned of particulates and sent to a catalytic acid plant to be converted into sulphuric acid. The Waelz kilns now treat Shaimerden low sulphur (up to 1%) ore which has decreased overall SO_2.

At the time of the site visit the general level of housekeeping across the site was regarded as reasonably good.

Untreated aqueous effluent is absent 18,000,000m³ water is treated at two waste water treatment plants annually and the site produces little solid wastes for disposal presently. Historic waelz slag and jarosite from previous processing activities are dumped in above ground stockpiles. It is understood that Kazzinc do not possess any liabilities associated with these historic dumps. Liabilities are retained by the State who remains the owner of the dumps. As these are regarded as temporary structures (awaiting development of new process technology) by the State they are not subject to closure planning or permanent disposal plans.

> ***WAI Comment:*** *WAI considers that many of the comments made regarding the metallurgical complex at Ust are equally applicable to Ridder.*
>
> *The impact of historical dumps on groundwater is assessed quarterly across odservation holes.*

16.5.4 Community Relations

Kazzinc is a major employer in East Kazakhstan. Senior management reported that the Company enjoyed good relationships with the local community in both Ust-Kamenogorsk and Ridder and the Company supported many local causes and organisations. This includes sponsorship of the regional ice hockey team in Ust-Kamenogorsk and the local football team in Ridder.

> ***WAI Comment****: WAI would concur that based on limited time spent in the region that good relationships exist with the local community. Corporate Social Responsibility (CSR) initiatives undertaken by Kazzinc are documented in memoranda. The community is informed in accordance with the Company's Community and Media Relations Plan.*

16.6 Summary and Conclusions

There is a long history of metallurgical processing at both Ust-Kamenogorsk and Ridder ensuring a well qualified, knowledgeable workforce is available. The metallurgical complex at Ust-Kamenogorsk is particularly complex with recovery of a number of metals, principally lead, zinc, copper, gold and silver.

The current process flow sheets are considered to be fully optimised and further process efficiencies will be difficult to realise.

WAI considers that the copper ISASMELT™ process (with associated dedicated tank house) offers significant potential for Kazzinc to become a low cost producer. In addition to Kazzinc ore there should be potential to toll treat third party concentrates. The process and site infrastructure should be capable of testing high arsenic containing copper ores which might attract a premium smelting rate. There are numerous examples of operational ISASMELT™ furnaces smelting primary copper concentrates and WAI considers that the technical risk associated with this technology to be low. The close involvement of Xstrata during the commissioning and operator training period is noted and considered vital to the successful introduction of this technology.

There are fewer operational lead ISASMELT™ furnaces smelting primary concentrates. It is noted that the current investment programmes should result in reduced total environmental emission at Ust-Kamenogorsk.

Current arsenic disposal at Ust-Kamenogorsk is in the form of calcium arsenate and arsenite production and disposal to pits with three impermeable layers. Internal research has identified a scorodite (iron arsenate) as the preferred arsenic species for long term disposal. WAI would agree with this evaluation. WAI understands that this is scheduled for mid-late 2012.

WAI notes the continuing commitment of Kazzinc to the implementation and maintaining of internationally recognised management systems, including ISO 14001 and OSHAS 8001. Implementation of these systems is regarded as best practice.

17 BUTACHIKHINSKO-KEDROVSKIY

17.1 Introduction

17.1.1 Butachikhinsko-Kedrovskiy Block

The Butachikhinsko-Kedrovskiy Block is located in the East Kazakhstan Oblast, some 40km northeast of Ust-Kamenogorsk and to the southwest of Ridder (Figure 17.1). The licence block covers an area of 700km^2 and a contract was sent to the Ministry for approval, and which is awaited to be received by Kazzinc. It should be noted that the Tishinskiy deposit/mine is enclosed within this licence block.

Kazzinc was awarded the tender by the Ministry of Energy and Mineral Resources of Kazakhstan for regional exploration for non-ferrous metals and primary gold in the Butachikhinsko-Kedrovsky district in November 2009.

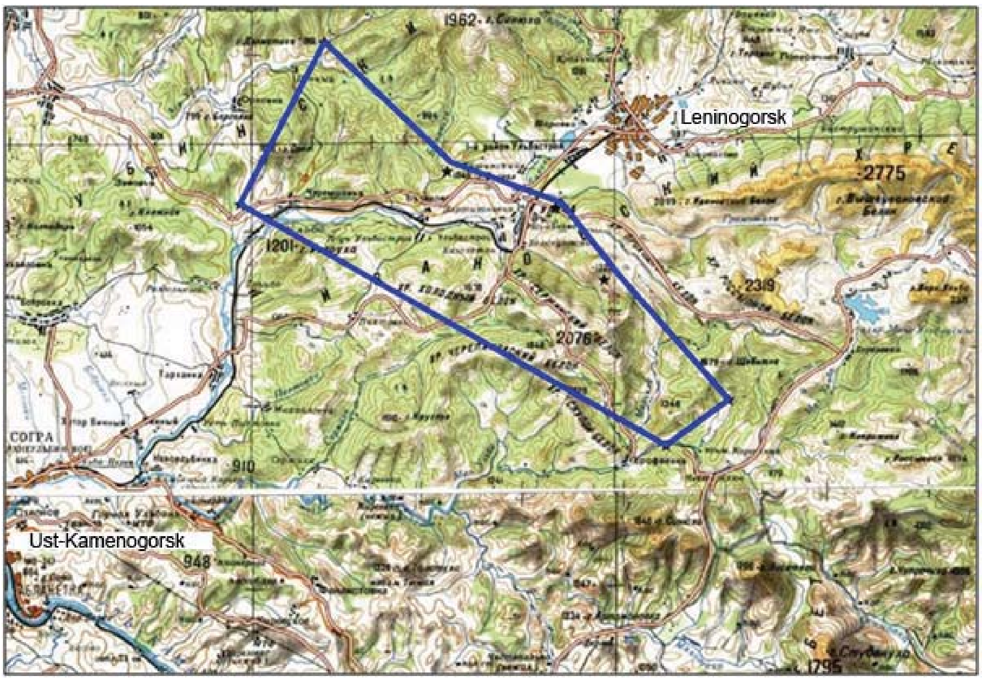


Figure 17.1: Location of Butachikhinsko-Kedrovskiy Block

The Butachikhinsko-Kedrovskiy Block lies within the Rudny Altay (VMS) province and predominantly comprises Lower to Upper Devonian geological units, with a small area of Carbonaceous in the extreme west of the block. The Rudny Altay province extends over 500km along the north eastern border of Kazakhstan. The southern half exposes Permian intrusive units while to the northeast massive granitoid units are present. The Mid to Lower Devonian units comprise intrusive host rocks though prophylitic alteration of Upper Carboniferous to Lower Permian age is evident within and around intrusive units and deposits.

Currently the most prospective target is the Grekhovskiy deposit, located to the west of the Tishinskiy deposit, which is proposed by Kazzinc to be explored to a depth of 1.5km.

Regional prospecting has been conducted within the Butachikhinsko-Kedrovskiy Block since the early 1900's, initially involving geological mapping and later through geochemical and geophysical surveys.

Although preliminary in nature this work has identified several prospective areas which will require considerable follow-up investigation to establish their true potentialExploration.

17.1.1.1 Introduction

Although Kazzinc have yet to receive approval for the licence a fairly significant amount of regional exploratory work has already been completed over the Butachikhinsko-Kedrovskiy Block including:

- High-resolution satellite images interpretation (5-6 m) with relief processing;
- Geochemical surveys using 'IONEX' – method and bed rock geochemistry;
- Complex airborne magnetic survey, scale 1:50 000 – 1:25 000, (magnetic survey + gamma-ray spectrometry);
- Geological and mineragenetic mapping at a scale 1:100 000 with 1:50 000 details, to define productive structures; and
- Petrophysical studies for analysis and interpretation of geophysical data.

Recent geophysical surveys have included Audio Magnetic Telluric (AMT) and Magneto Telluric (MT) 'sounding' in 2007 and 2010 along SW-NE profiles.

17.1.1.2 Historical Exploration

Early prospecting and information on the geology within the region is very fragmented and sparse. The first post-revolutionary prospecting by V.K. Kotul'skim Geolkoma started in 1917 (to 1920-1921) and drafting of the first geological maps of the area Ridder were produced in 1925. The first geological maps of the pre-war period are at a scale of 1:87,000 - 1:100,000 and from the northern area.

Systematic study of the area began in the 1950's culminating in a drafted geological map (geological map sheet m-44-XXVIII scale 1:200,000) in 1956. This was based on the whole of the Altay stratigraphic pattern, the principal provisions of which has survived to the present.

Between 1959 and 1973 most of the area was covered by survey work at a scale 1:50,000 (APSÈ). Survey work, including geophysical, formed the basis for the projection of metallogenic maps at a scale 1: 50 000 produced in 1973 by a team of geologists (VKTGU) and leading research institutes.

In 1988, Altai GGÈ completed the central part of Leninogorsk region covering Leninogorsk, Leninogorsk GOK, Tishynsky and Guslâkovskogo, at a scale 1:50,000, as well as producing a photographic library.

Thus, to date, almost the entire territory of the region (except the extreme South) is covered by the geological survey at a scale of 1:50,000.

Along with geological mapping, from 1953 to 1982 the "Leninskaya Geological Expedition" conducted detailed 'search and exploration', which included excavating pits and trenches, structural and exploratory drilling, and deep geochemical and geophysical research in the Butachikhinsko-Kedrovskiy Block. As a result of these works the Tishinskiy deposit was discovered with subsequent development and detailed underground exploration beginning in July 1963. Exploration was initially (to 1957) conducted by Altay Base Metals Exploration (ACMR), then jointly by Leninogorsk Geological Exploration Expedition (Leninogorsk GRE) and Leninogorsk Polymetallic Complex (LPC) and, from 1997, by OAO Kazzinc (successor to LPC). Actual mining commenced in 1964, from an open pit in the centre of the deposit, and operated until 1978, overlapping with underground mining which began in 1969.

Geological and mineral genetic mapping (scale 1:100,000) conducted (as shown in Figure 17.2) included 8,750m of trenching, 10,507m of 'structural' drilling, and 5,138m of core drilling.

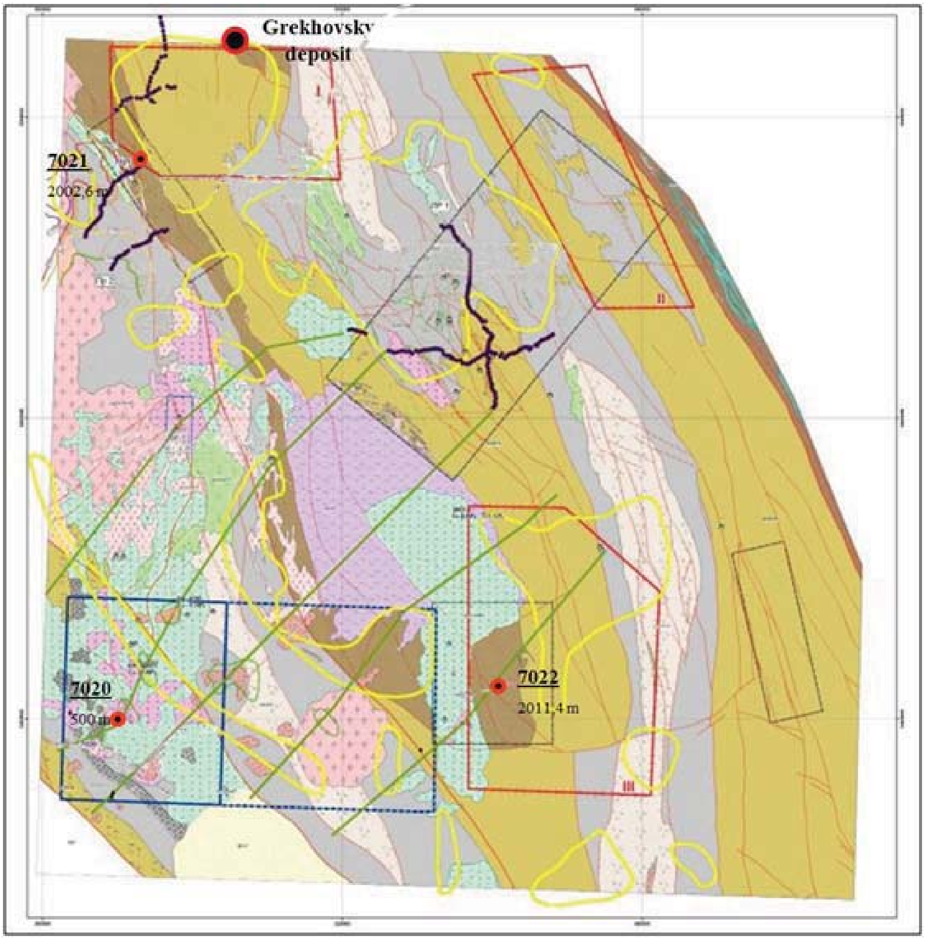


Figure 17.2: Geological and Mineragenetic Mapping (ГМК-100 (50)

In 1958 a drillhole was completed to a depth of 180m (No.779) in the southern area (Sub-block 6) to specifically target polymetallic mineralisation but instead intersected vein/stockwork gold mineralisation. Quartz veins, exposed on surface at 1.5 to 2.0m wide, reveal an east-west orientation with associated polymetallic mineralisation. Resultantly the area has become more of a gold target with high background gold geochemistry anomalies from sediment/soil sampling.

Historical drilling (limited) was only to shallow depths (300–400m) and was largely inclined due to the sub-vertical structures present in the region. Resultantly the coverage was irregular and fundamentally unreliable, particularly in previously considered areas of mineralisation, and their actual locations are also unclear.

17.1.1.3 Kazzinc Exploration

Since 2005 Kazzinc have undertaken a limited amount of preliminary (regional) exploration within the Butachikhinsko-Kedrovskiy Block, prior to the issuing of the licence but with State agreement. Initial work comprised the collation of available historic data, including drillholes and trench data. Subsequent to this a preliminary, confirmatory, programme was conducted with sampling that identified elevated levels of mineralisation, particularly copper.

Stratigraphic, structural, alteration (incl. hydrothermal) and geochemistry anomalies, as well as satellite images have all been combined and sequenced to form generalised images in an effort to identify the most promising areas. These will subsequently be investigated by Kazzinc in more detail through mapping, trenching and drilling, once the exploration licence is issued.

Geochemical 'Mode of Metal' occurrence surveys identified anomalous zones over the block, notably gold at Kedrovsky (Sub-Block 6) as well as several other locations for gold and polymetallic mineralisation. Prospective sites were considered on several parameters including historical and recent geochemistry, hydrothermal geology (potassium levels) and alteration etc. An initial 5km grid over the block was reduced to 1km and 250m grids over more prospective areas, including to the southwest of Tishinskoy which showed a strong anomaly and elevated levels of copper with zinc targets showing up further to the south and west.

Aero gamma/magnetic surveys, undertaken by a Moscow based contractor, have also been conducted to confirm the basic geological framework as well as reveal elevated levels of potassium. The use of AMT sounding surveys, initially tested on the Ridder area deposits to test the results which demonstrated positive correlation, has also demonstrated its potential value to exploration over the block.

17.1.2 Geological Setting

The Butachikhinsko-Kedrovskiy Block is located within the northwest striking, mid-Palaeozoic, Rudny Altai VMS (Volcanogenic Massive Sulphide) belt which is the host of numerous world-class VMS deposits, including Leninigorsk, Zyryanovsk and Maleevsky as well as smaller Grekhovskiy, Putintsevskie, Snegirevskie, and May. The Rudny Altai belt (VMS terrane) extends over 500km along the north eastern border of Kazakhstan and is ranked in the top four VMS belts of the world. As a note the Tishinskiy deposit is situated in the central portion of the Butachihinsko-Kedrova shear zone, which adjoins the south-western flank of the Ridder graben.

17.1.3 Resource Estimates

At present no Mineral Resources or Ore Reserves have been estimated in accordance with the guidelines of the JORC Code (2004) for the Butachikhinsko-Kedrovskiy Block.

***WAI Comment**: The amount of work (historical and recent), and level of detail, allied to a meticulous work ethic demonstrates a strong desire to fully investigate the Butachikhinsko-Kedrovskiy Block in its entirety. In addition the senior personnel involved with Kazzinc have a long and proven experience of working in these geological environments. The combination of this knowledge together with modern exploration techniques and a methodical programme provides a high possibility of success. Notwithstanding these comments there is a huge amount of exploratory work necessary before the true potential of the area, and an 'ore' deposit, can be revealed.*

18 SOLOVIEVSKIY

18.1 Introduction

18.1.1 Solovievskiy Block

The Solovievskiy Block is located in the eastern Kazakhstan region, Katon-Karagaiskiy District, some 100km south southeast of Ridder and 140km east southeast of Ust-Kamenogorsk. The nearest large town is Zyryanovsk where Kazzinc operate the Maleevskoye mine and processing plant. There is also a sample preparation facility as well as assay laboratory at the Zyryanovsk mine complex which was inspected by WAI as part of the site visit.

The main town and administrative centre for the region is Zyryanovsk, which is the centre of the Zyryan District, and is located outside the northern edge of the licence block with a population of around 45,000 (est). At the southern margin of the licence block is the village of Bolshenarymskoe, and Kazzinc operate a 'Geological Department' incorporating an exploration facility (office, core preparation and storage) at a location some 10km to the west of this town. This exploration office lies approximately 10km south of the Novokhairuzovskoie deposit.

Figure 18.1: Outline of Solovievskiy Block

The Solovievskiy Block lies within the Rudny Altai Palaeozoic province, or belt, and predominantly comprises Carboniferous and Devonian rocks. The Rudny Altai metallogenic belt extends over 500km along the eastern border of Kazakhstan from the Altaisky region of Russia southwards, through east Kazakhstan, into northwest China. The mineral hosting rocks are typically of late Devonian age.

The Solovievskiy Block is the most advanced exploration licence currently in operation by Kazzinc and the only one to possess a valid licence (at the time of the site visits in 2010). The Solovievskiy Block has an area of 4,300sq.km and is registered under Contract No.2114 dd. 25.07.2006.

Paved roads connect Ridder with Ust-Kamenagorsk and Zyryanovsk and access within the licence block is available via limited, but passable, dirt tracks. The distance from Ust-Kamenagorsk to Zyryanovsk is some 196km (by road) and takes approximately 3 hours, the licence area (south) is a further 1 hour travel time.

Kazzinc are undertaking a comprehensive and thorough exploration programme within the Solovievskiy block that commenced in 2005 culminating in a diamond drilling campaign as witnessed by WAI during the site visit in October 2010. This structured work has resulted in the identification of three targets; Novokhairuzovskoie (Au), Khairuzovskoie (Cu-Zn) and Greizenovoie target area, which are all located in the south-southwest of the licence block. The most advanced of which, Novokhairuzovskoie, is currently being drilled and a preliminary mineral resource estimation and model has been completed. To date (01.07.2010) some 30,510m of surface trenching and 16,544.8m of diamond drilling (in addition 41,180m of Reverse Circulation drilling for shallow geological/structural data acquisition) has been completed as since 2007.

Based on the work to date a preliminary resource estimate has been produced at different cut-off-grade (COG) for the Novokhairuzovsky deposit using Micromine® software to generate wireframes of Au and Re. However, the current exploration is widely spaced and as such can only be classified as P_1 in accordance with the Soviet classification system.

A preliminary mineral resource model has also been constructed by Kazzinc using Micromine® software to generate a 3D wireframe of the deposit and provide an estimate of Soviet classified P_1 resources of some 216.8Mt at 0.67g/t Au (0.3g/t Au cut-off grade).

18.1.2 Geological Setting

The region containing the Solovievskiy Block is considered by Kazzinc to be highly prospective, being within the Rudny Altai metallogenic, Volcanogenic Massive Sulphide (VMS), belt. Rocks of the Carboniferous and Devonian are developed within a strong NW - EW structural setting. Although the 'Central' part presents deeper (1 – 2km) targets, the northwest structure and southwest volcanics are considered to offer the better opportunities. In addition the southwest area shows evidence of ancient copper oxide (malachite) extraction and indication of copper oxide mineralisation is evident within granodiorites of the area. This area is also subject to potassic alteration, containing feldspars, biotite, phlogopite, and chlorite minerals, which is considered to represent an alteration stage or type in a porphyry copper deposit and is also typical of lode gold deposits.

As the Solovievskiy Block is some 120km in length (northwest-southeast) and some 60km in width the following description is focused on the Khairuzovsky area which encompasses the Novokhairuzovskoie, Khairuzovsky and Greizenovoie deposits.

18.1.3 Exploration

18.1.3.1 Introduction

Between 2005 and 2007 Kazzinc completed systematic geological exploration around the Leninogorsk and Ziryanovsk mining areas. These works included geophysical, geochemical, hydro-geochemical and remote (aero-geophysics) research aimed at rapidly gaining information to enable Kazzinc to evaluate the entire area for potential targets. To achieve this Kazzinc employed various organisations from Kazakhstan, Russia and Australia.

In 2005 some 25,000km^2, which includes the Leninogorsk and Ziryanovsky areas, were surveyed by geochemical studies on a 5km network, accompanied by sampling for assessment of physical properties. In 2006 this work culminated, together with geochemical data from previously completed work, in a thorough hydro-geochemical study at a scale of 1:200,000 - 1:500,000.

In summary, and following these initial studies, the Leninogorsk district presents the most prospective areas; however these have largely been explored and exhausted and the possibilities of discovering new deposits are low. Conversely, within the Solovievskiy Block the first stage work recognised areas of positive mineralisation, as well as identify similar geological settings with elevated mineralisation which form the primary targets in follow-up exploration.

18.1.3.2 Historical Exploration

Mineralisation, and ore minerals, has been recognised in the Solovievskiy Block since the 19th century (Murzincevskoe, Aleksandrovskoe II Hlopinskoe etc.). Although from this historical study the possibility of identifying prospective ore deposits within the block was considered poor. This may have been partially the result of mineral deposits being discovered and developed in the Ridder-Leninogorsk district and Maleevskoye. Conversely, their discovery, as well as a number of medium and small scale deposits, is also the rationale and counterargument for the areas potential.

The last complete and most consistent work on this area has been geological mapping at a scale of 1:50,000. More recently the area was relatively well studied following various geological surveys and simultaneous high quality geophysical studies, though these have mainly been shallow investigations.

Based on the set of criteria identified, three promising areas have been outlined:

1. South of Grekhovskiy deposit;
2. Murzincevskogo District (deposit); and
3. Nazarovskogo District.

The Solovievskiy Block has undergone an accelerated period of study by Kazzinc since 2005 with geological, geochemical, and geophysical studies. The main objective of this work has been to quickly assess the prospects within the area for pyrite-polymetallic mineralisation and to select prospective sites for follow-up investigation.

18.1.3.3 Kazzinc Exploration

Kazzinc have adopted a geostatistical approach to exploration, based on known parameters and characteristics to produce a 'mathematical probability' which has resulted in approximately 90% of the area being considered to be promising. However, in practice this process is seldom so simple because every mineral deposit has some uniqueness to its characteristics, such as alteration zoning. Nevertheless, the system has identified several targets which are currently subject to more detailed investigation and exploration, the most advanced of which is the Novokhairuzovskoie deposit which has already received a Prognostic resource estimate.

Having considered the entire Solovievskiy Block Kazzinc have focused on more detailed exploration of the Novokhairuzovskoie (Au) deposit, and the Khairuzovsky (Cu-Zn) and Greizenovoie mineralized zone, which are all located in the south-southwest of the licence block. Of the three the Novokhairuzovskoie deposit is the most advanced with 30,510m of surface trenching and 16,544.8m of diamond drilling (in addition 41,180m of Reverse Circulation drilling for shallow geological/structural data acquisition) completed as at 01.07.2010 (since 2007).

First stage mineralisation is related to volcanic and terrigenous andesite-basalt formations of the Upper Devonian, at the final stage of Bolshynarymsky brachysyncline formation. Stratiform copper pyrite mineralisation is formed at this period of time, and localised on several stratigraphic levels (Khairuzovskoye). Second stage is related to the volcanic and terrigenous formations of Larikhinsky series, and small porphyritic intrusions of Bukhtarminsky complex of rhyolitic and rhyodacitic porphyry. Predominant elements at this stage are copper and gold. Third stage is related to the diorite and granodiorite formations of the Zmeinogorsky complex. Gold-bearing copper and molybdenum mineralisation is referred to cataclasis and eruptive brecciation zones.

Sample Collection, Preparation, Analyses and QA/QC Procedures

This section refers specifically to the exploration conducted on the Novokhairuzovskoie deposit which is far more advanced than at the other prospects within the Solovievskiy block and comprises surface trenching and sampling and exploration drilling.

(a) Sample Preparation

Diamond drill core is transported off site to the Kazzinc facility near where the core is inspected by a geologist (partially logged) before being fully dressed and split with diamond saw. After this the core is logged in detail, and a basic magnetic survey conducted, before the core is marked with sample intervals.

Samples are typically 2–3m in length (half core) and small polished section samples are also removed for analysis. The remaining half core is retained in storage for further inspection, re-logging, and/or for technological samples.

The samples are bagged (in cloth bags) and clearly labelled with drillhole ID, depth and interval, before being transported to the sample preparation facility at Zyryanovsk. It should be noted that no control samples (blanks, duplicates or standards) are introduced at this facility.

As well as the basic drillers log from the field another 2 geological logs are produced with geological description (including sample interval) and for technological details, including sample details and magnetic survey data. Logs are entered into an Excel® spreadsheet daily and rock codes are assigned.

Hand drawn cross-sections are also updated daily, as logs are received, and include any other salient information as well as geophysics data (IP anomalies). Data is further managed and manipulated using MapInfo GIS software whereby additional information, including topographic surveys, surface geological maps, geophysics, aerial photographs and exploration data, is included for review.

> ***WAI Comment**: WAI visited the 'Geological Department' on 07.10.2010 and found the facility to be busy and well run. A clear protocol was in place and being followed. The facility was in a state of upgrading, particularly the office building, and so improvements are expected in the general layout and order of the office.*
>
> *However, the vast majority of core was stored outside and exposed to the elements. An effort should be made to improve this situation as contamination to the core can occur as well as oxidation and damage as a result of water ingress, and frost.*
>
> *The practice of control samples into the sample stream should be introduced as a matter of urgency at this stage, prior to the sample preparation facility, to both test and improve the QA/QC procedures. All companies need to obtain an accurate estimate of the quantity of an economic mineral in a given deposit. It is this estimate which ultimately drives the development decision and with the ever increasing sophistication in resource and process modelling, the precision and accuracy of the earliest stage of exploration sampling is becoming increasingly significant. Implementation of a rigorous well-conceived QA/QC programme at an early stage allows for the ready acceptance of the data and its conclusions by external organisations and saves both money and time by removing the necessity to back-track at the resource drilling or feasibility study stage in an attempt to obtain reliable and compatible data.*
>
> *Kazzinc should obtain Certified Reference Material (CRM) samples urgently to introduce into the sample stream, as well as inclusion of suitable blanks and duplicates. These QC samples should represent the deposit geology and mineralogy as well as form. Furthermore, QA/QC samples should be submitted 'blind', as part of the routine sample sequence to eliminate risk of spotting.*

(b) Sample Preparation Laboratory

The sample preparation laboratory, situated within the Zyryanovsk process facility, was visited by WAI on 07.10.2010 and escorted by the Head of the Sample Preparation Laboratory (Denikina Liliya Borisovna) during the visit.

This facility only handles samples from Kazzinc exploration programmes and does not prepare mine samples which are treated on their relative mine site laboratories.

Samples are received in batches, or 'orders', directly from the field and weighed before being dried as required or requested by the 'client'. A set of "Retsch" jaw crushers then operate a 2 Stage process to initially reduce to <5mm and secondly to <2mm (1 – 2mm). The crushed sample is then 'mixed', or rolled, and quartered to obtain two samples of 500g. The remaining material is disposed of and not stored or returned to the 'client'. The primary 500g sample is then pulverised (ring and puck) to passing a 200 mesh (0.074mm), mixed again and split into 2 sub-samples, 1 for assay determination and the other a duplicate.

After each sample the crusher is cleaned with compressed air and at the end of the shift (laboratory operates 2x12 hour shifts) is further washed with water and cloth. The same protocol is applied to the scales except that a fine brush is used between samples.

> ***WAI Comment****: The laboratory, although relatively small, is well laid out and only handles a limited amount of samples per shift (approximately 19 samples per person per 12 hour shift). However, samples are handled on square/angular sample trays whereas most international laboratories have introduced rounded trays to minimise cross-contamination and ease cleaning between samples. No control samples are introduced and no barren material or quartz is used to clean crushers/pulverisers between samples or at regular intervals. The floor of the laboratory is also uneven in places which is a potential safety issue that should be addressed.*
>
> *Preparation must ensure that the correct particle size and sample size reduction schemes are used in order to minimise errors. Entire samples should be crushed to 90% -2mm, then 1kg of crushed material should be riffle split and pulverised to 90% <75µm. In order to monitor this, laboratories should conduct wet screen analysis on a small but regular selection of samples.*
>
> *Overall the laboratory is acceptable in terms of facilities and sample preparation protocols. However, WAI would recommend that the use of rounded sample trays be introduced to replace the current angular ones and the use of barren quartz also be introduced between batches and/or at regular intervals to improve cleanliness and minimise cross-contamination between samples. The splitting of samples should also be conducted exclusively with the use of industry standard riffle splitters to improve the quality of the split. Coarse reject, as well as the pulp duplicate, should be returned to the 'client' for storage and QA purposes.*
>
> *Duplicate and blank samples should be randomly submitted into the sample stream as a standard procedure to monitor subsequent assay laboratory performance with regards accuracy and precision.*

Assay Laboratory

(a) Gold Assay Analysis

All the primary gold assay analysis for the Kazzinc exploration programmes is conducted at the TOPAZ laboratory in Ust-Kamenagorsk. In the period 2007 to 2010 a total of 22,227 samples have been analysed at this laboratory. This facility was visited by WAI on 06.10.2010 and it has operated as an independent laboratory for some 65 years. The office is currently undergoing an upgrade though this is not affecting the actual laboratory facilities. The laboratory is accredited to ISO 17025:2007 'Laboratory Management System' (plus ISO 9001:2000).

The laboratory receives a 'log' from Kazzinc that accompanies the samples and details the ID (trench or drillhole), depth and interval, and a brief geological description. The pulverised samples are received in 500g paper 'wraps' from the Kazzinc preparation laboratory in Zyryanovsk. The samples are logged in with all the relevant details into a hand written log book although the final results are provided in electronic format. An internal duplicate sample is taken every 5 or 6 samples and results are provided if requested or if there is a discrepancy. External laboratory control is conducted in Almaty or Karaganda depending on the analyses required or Client request.

Gold is analysed by standard fire assay techniques as follows:

- Samples mixed with flux (flour potassium);
- Samples placed into cup and into furnace in batches of 18;
- Glass material broken off and remaining 'plug' placed into cupel; and
- Remaining 'bead' is conveyed to the laboratory to be beaten, weighed and dissolved in nitric acid before being roasted and re-weighed to determine quantity of gold.

The final weighing process is undertaken in a purpose room where temperature and humidity is controlled (no external ventilation) and the electronic weighing scales are mounted onto a solid bench. The scale is calibrated annually by the manufacturer.

> ***WAI Comment**: WAI considers that the laboratory was clean, well organised and run efficiently and is fit for purpose to provide gold assay analysis. Internal and external QA/QC protocols are in place in order to meet the requirements of ISO 17025:2007 accreditation.*
>
> *It is recommended that Kazzinc obtain the results of all internal and external check analysis, regardless of results, as these can be used for QC monitoring and reporting. However, without the insertion of control samples (blanks, duplicates and standards) by Kazzinc there is a lack of QA protocol, which should be standard practice, to ensure that the laboratory being utilised is producing accurate and precise results continuously*.

(b) Polymetallic Assay Analysis

All polymetallic (multi-spectral) analysis is conducted at the "OC RPC Plazma-Analyt" (Open Company Research and Production Centre) in Ust-Kamenagorsk In the period 2007 to 2010 a total of 27,744 samples have been analysed at this laboratory for 34 analyte (not including Au). This facility was visited by WAI on 08.10.2010 and as it also operates as a uranium facility the level of security is very high and access restricted.

Samples are received in paper 'wraps' and registered electronically giving each sample a new laboratory ID, in addition to the client (Kazzinc) ID. The sample is mixed and 'cone and quartered' to reduce the original sample to a 50g weight, the final analysed sample is only 0.5g. The electronic scale used to weigh these sub-samples is audited annually (by the manufacturer) as well as being checked daily, and possibly through the day if deemed to be required, using registered weights.

Several methods for analysis are available depending on the anticipated grade or detection limit required. For lower grade samples ICP-MS (Inductively Coupled Plasma - Mass Spectroscopy) is used whereby the sample is dissolved (acid dissolution) to obtain a 50ml sample which goes forward to produce a 1:10 (sample:solution) ratio. The ICP-AES (Inductively Coupled Plasma - Atomic Emission Spectroscopy) is applied where there is a higher concentration of elements and a 1:1 ratio, i.e. a higher solution concentration is used.

Approved' laboratories in Kazakhstan enter into a 'round robin' series of testing in order to obtain accreditation. The "National Centre for Accreditation" disperses samples to different laboratories, and in effect acts as the client as results are monitored by the centre who then advises the laboratories on results.

> ***WAI Comment**: It was clearly evident from the visit by WAI that the laboratory operates under a strict protocol in terms of internal and external quality control, as well as under stringent security measures. However, to enable the quality control samples to be of any use it is important that Kazzinc obtain the results in order to monitor and report these checks as part of the whole QA/QC process. In addition it is also essential that Kazzinc introduce their own quality control samples into the original sample set at an early stage and monitor the results accordingly in respect of the sample preparation and subsequent analysis.*

Conclusion and Recommendations

Both (independent) analytical laboratories were visited by WAI and are considered to operate and provide a high calibre service. The TOPAZ facility has achieved ISO 17025:2007 (ISO 9001:2000) and so demonstrates appropriate and approved management systems. The OC RPC Plazma-Analyt laboratory in Ust-Kamenogorsk introduces National standard (sourced from Ingredmet) and blank samples, prepared in-house, into each batch of samples. In addition random samples are selected, re-coded, and re-introduced into the sample stream as part of an internal control. External control is also undertaken but only at the 'clients' request.

Whilst the laboratories both present a good standard of assay analysis the lack of control samples introduced by Kazzinc presents a serious issue with regards QA.

Quality Assurance relates to the planned or systematic actions necessary to provide adequate confidence in the data collection and estimation process, and Quality Control means the systems and mechanisms put in place to provide the Quality Assurance. The four steps of quality control include; setting standards; appraising conformance; acting when necessary and planning for improvements.

QA/QC must be addressed during the collection, recording and storage of any of the data ultimately used in mineral resource and ore reserve estimation. This programme should be concerned with, but not limited to: data verification, drill sample recovery, sample size, sample preparation, analytical methods, the use of duplicates/blanks/standards, effects of multiple periods of data acquisition and consistency of interpretation in three dimensions. The results of the QA/QC programme form part of the database and must be recorded.

18.1.4 Metallurgical Testwork

At this stage only preliminary metallurgical testwork has been conducted on 3 samples (Table 18.1). Subject to continued positive results it is proposed to undertake more decisive testwork as the project proceeds.

Table 18.1: Ore Process Characteristics of Tested Samples

	Grade (%)						Grade (g/t)	
	Pb	**Zn**	**Cu**	**Fe**	**S total**	**C**	**Au**	**Ag**
Sample T-1	0.03	0.05	0.04	3.20	0.83	0.25	**0.40**	0.60
Sample T-2	0.01	0.07	0.05	3.40	0.95	0.30	**0.90**	1.5
Sample T-3	0.03	0.08	0.18	3.53	1.31	0.25	**1.30**	3.8

Ore process characteristics have determined the following:

1. Gold is the most valuable metal in the tested ore. The gold in the ore has basically ultrafine and fine structure (<0.5–2.0mk^2) with occasional larger dust-like gold grains up to 7.4x16.9mk^2;
2. Gravity and flotation testing as well as successive coupling of these testing methods was applied with the following findings made:
   - This type of ore can be processed by gravity separation in centrifugal devices.
   - Flotation re-extraction of gold from gravity separation tailings seems to make sense for higher gold extraction; and
3. Combined gravity-flotation separation of ore is recommended with total gold recovery to gravity and flotation concentrates 85.69% with total gold content in combined concentrate 85.69% with combined concentrate output 10.73%.

18.1.5 Resources

There are currently no Mineral Resources estimated in accordance with the guidelines of the Jorc Code (2004) for the Solovievskiy Block.

***WAI Comment**: Considerable resource definition will be required in order to realise the full potential of the deposit.*

Furthermore, the lack of a full QA/QC programme by Kazzinc during their drilling programme will make verification of the database inconclusive. The inadequacies or apparent problems with the sample and analytical database will have to be resolved before an estimate in accordance with the JORC Code of these resources is possible. Fortunately the core is halved and so 50% remains which could be re-assayed at an internationally accredited and independent laboratory so as to provide credence to results obtained to date. However, this should be removed from an urgent evaluation of the current QA/QC protocol.

19 WESTERN TORGAI

19.1 Introduction

The Torgai Basin in northern Kazakhstan represents a complex petrographic province in which the distinctive features of the magmatism are a consequence of its location in a region where the influence of two different orogenic events (Caledonides of Kazakhstan and the Hercynides of the Urals) is evident. Volcanism and intrusive magmatism are ubiquitous over a significant area, exhibit a wide range of compositions from ultrabasic to alkaline and acid varieties and developed over a long period of time from the Precambrian to the Triassic, inclusive.

The Valeryanovskaya synclinorium, in the north west of the Torgai Basin and within which the Western Torgai exploration area lies, is limited to the west and east by successive upthrow faults and has an asymmetric structure, its axis being displaced to the west and plunging to the north. It is divided into a several large blocks by WNW-ESE, E-W and ENE-WSW striking faults, delineated from the results of magnetic and gravity surveys, and extends for a considerable distance to the north and south beyond the exploration area.

Almost the entire Western Torgai area has been covered by aerial photography on a scale of 1:50,000 and geochemical and geophysical surveys were completed at the same time. The whole of the Valeryanovskaya synclinorium has been subject to extensive investigation, and drilling through the iron ore and bauxite horizons intersected gold, copper, polymetallic and other minerals (titanium, molybdenum).

As a consequence five promising sectors were identified, of which two; Karabaytalsky and the Sakharovsko-Adaevskiy, have been prioritised for further investigation and specifically the Belozersk and Kamyshlykolskaya target areas in the former and the Klochovskaya target area in the latter.

The Klochkovskaya showing of copper-porphyry mineralisation has been delineated as the most promising target area in the Sakharovsko-Adaevskiy Sector in the southern part of the Adaevskogo 'ore' unit. However, there is insufficient exploration data to allow the estimation of resources.

19.2 Background, Location, Access, Topography and Climate & Infrastructure

19.2.1 Background

The earliest geological investigation of the Western Torgai region comprised exploration along the Trans-Siberian railway main line, and hydrogeological studies for water supply to migratory settlements in the 1880's.

Interest in the region developed at the beginning of the C20th with the discovery of the Dzhetygarinskogo gold deposit and Akkarginskoya chromites and the most active period of exploration resulted from the discovery and investigation of iron-ore, bauxite and lignite deposits in the northern part of the Torgai Basin which led to a systematic and detailed study of the region.

Almost the entire Western Torgai area has been photographed on a scale of 1:50,000 and geochemical and geophysical surveys were completed at the same time. The whole of the Valeryanovskogo structural basin (synclinorium) has been subject to extensive investigation, and drilling through the iron ore and bauxite horizons intersected gold, copper, polymetallic and other minerals (titanium, molybdenum).

The initial exploration effort was not directed towards the polymetallic potential of the region although the geological parameters for the discovery of polymetallic deposits had been identified.

In the Sakharovsko-Adaevskiy exploration site, copper-porphyry type mineralisation has been identified at Klochkovskoe, Kungurtausskoe and, to the south, Benkalinskoe, the first named being considered the most prospective.

19.2.2 Location and Access

The Karabaytalsky and the Sakharovsko-Adaevskiy exploration sectors extend for a further 70km to the south south-west of Shaimerden open pit zinc operation which is located some 200km SSW of the city of Kostanay and about 110km south west of Lisakovsk, in the Kamysty region of the Kostanay District of northern Kazakhstan (Figure 19.1).

Geographically the region comprises the Torgaiskaya-Kostanayskaya plain, which passes into the western Siberian steppe. The Karabaytalsky Sector in the southern part of the exploration area adjoins an elevated plateau (Aral-Irtysh watershed) which is the source of the Tobol and Torgai rivers.

The topography is gently undulating with elevations between 170-260m, characterised by numerous basins, small depressions and hollows and low ridges (5-15m).

The region is open to the north (towards the Arctic and Central Asian deserts) and has limited protection to the west (South Urals) and east (Kazakh uplands). Its location dictates the climate which is extreme continental, with little precipitation, prolonged sunshine, constant winds and a wide range of diurnal and annual air temperatures.

19.2.3 Infrastructure

The exploration areas are located in the southern part of the extensive north western mining region of Kazakhstan with well-developed asphalt road and railway networks; the access is good although the condition of the road surfaces is variable. Nearest settlements comprise Adaevka, Kamysty, Bestobe and Krasnooktyabrskiy, with the larger conurbations of Tobol and Lisakovsk to the north and the city of Zhitikara to the northwest.

Agricultural activity predominates in the region; there are no available local energy resources and coal and petroleum products are imported. Electric power is supplied from the branches of Urals power system (Russia) through lines via Lisakovsk and Krasnogorsk.

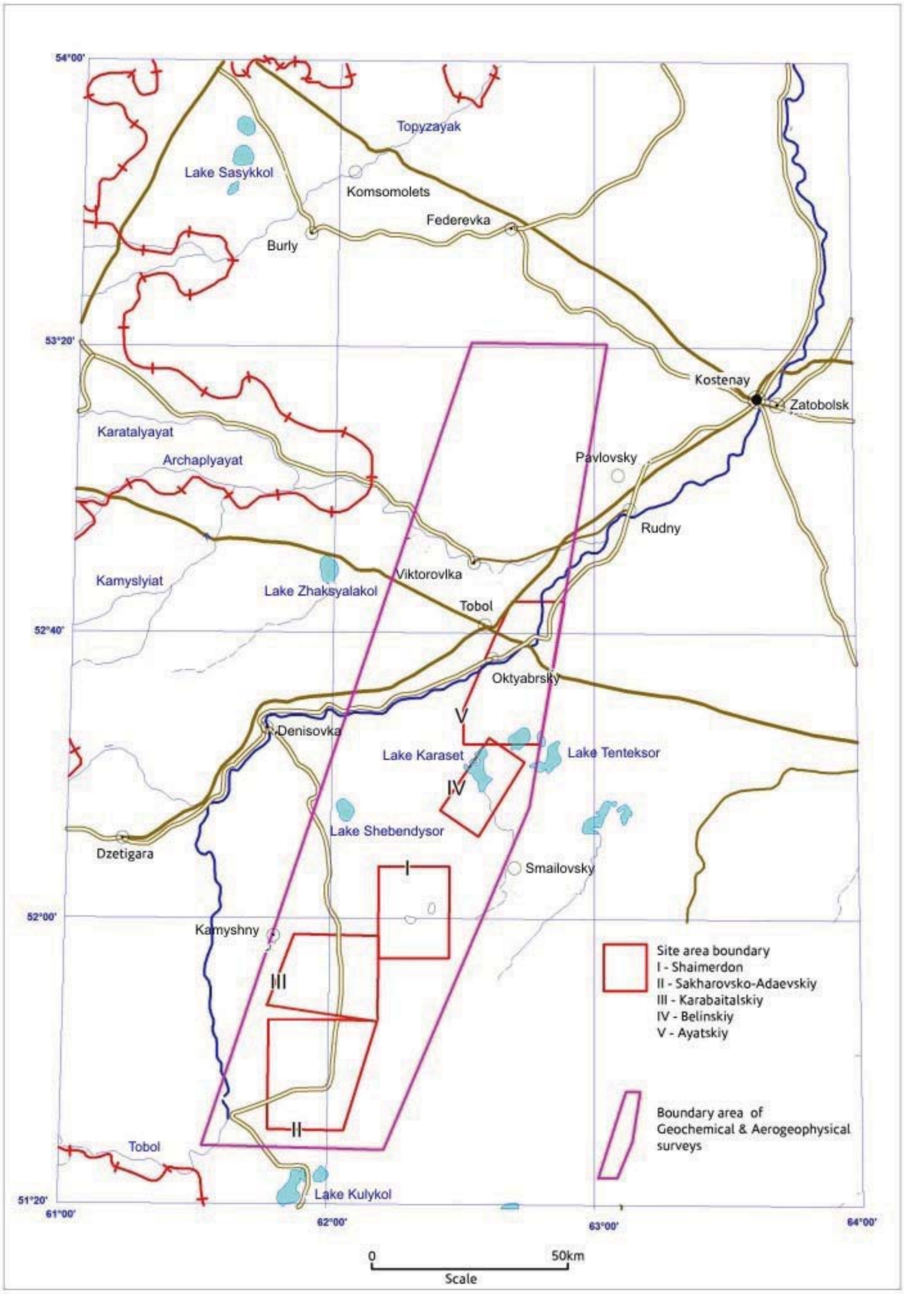


Figure 19.1: Location of the Karabaytalsky and the Sakharovsko-Adaevskiy Sectors of the Western Torgai Exploration AreaTopography and Climate

19.3 Mineral Rights and Permitting

Kazzinc has made applications for the various exploration licences and contracts are in the process of registration at the Ministry of Industry and New Technologies, approval is being awaited.

The boundary point co-ordinates of the limits of the original geochemical and aerogeophysical survey area and the Karabaytalsky and Sakharovsko-Adaevskiy sectors and their proposed target areas are summarised below (Table 19.1 to Table 19.5).

The Karabaytalsky and Sakharovsko-Adaevskiy sectors are 570km^2 and 734.4km^2 respectively.

19.3.1 Karabaytalsky Sector and Target Exploration Areas

Table 19.1: Karabaytalsky Sector-Boundary Co-ordinates

Boundary Points	Co-ordinates	
	Latitude N	Longitude E
1	51°58'13"	61°51'56"
2	51°58'07"	62°11'06"
3	51°46'13"	62°11'01"
4	51°48'24"	61°46'23"

Table 19.2: Belozerskoye Exploration Target Area-Boundary Co-ordinates

Boundary Points	Co-ordinates	
	Latitude N	Longitude E
1	51°58'00"	62°02'00"
2	51°58'00"	62°08'00"
3	51°54'00"	62°08'00"
4	51°54'00"	62°06'00"
5	51°53'00"	62°05'00"
6	51°53'00"	62°03'00"

Table 19.3: Kamyshlykolskoya Exploration Target Area-Boundary Co-ordinates

Boundary Points	Co-ordinates	
	Latitude N	Longitude E
4	51°53'38"	62°05'57"
7	51°53'38"	62°07'16"
8	51°52'17"	61°07'18"
9	51°52'16"	62°04'41"
10	51°53'04"	62°04'40"

19.3.2 Sakharovsko-Adaevskiy Sector and Target Exploration Area

Table 19.4: Sakharovsko-Adaevskiy Sector-Boundary Co-ordinates

Boundary Points	Co-ordinates	
	Latitude N	Longitude E
1	51°46'20"	61°46'29"
2	51°46'14"	62°11'02"
3	51°30'49"	62°03'39"
4	51°30'52"	61°46'29"

Table 19.5: Klochovskaya Exploration Target Area-Boundary Co-ordinates		
Boundary Points	Co-ordinates	
	Latitude N	Longitude E
1	51°42'00"	61°58'00"
2	51°42'00"	62°02'00"
3	51°39'00"	62°02'00"
4	51°39'00"	61°58'00"

19.4 Geology and Mineralisation

19.4.1 Regional Geology

19.4.1.1 Stratigraphy

Silurian deposits are the oldest formations underlying the western section of the area on the east of the Trans-Ural uplift, and comprise significant thicknesses of terrigenous sandstones- schistose lithologies in the lower part of the succession and predominantly volcanogenic rocks in the upper.

The stratigraphy of the Valeryanovskaya synclinorium consists of stratified deposits from the Lower-Middle Devonian to the Permian, divided into two complexes:

- Upper Devonian-Lower Carboniferous; and
- Middle-Upper Carboniferous-Permian.

The oldest deposits in the eastern part of the Valeryanovskaya synclinorium belong to the Frasnian Stage of the Upper Devonian; in the lower horizons a Givetian fauna has been recorded. The undifferentiated Frasnian Stage (D2gv-D3fr), comprising limestones and fine-fragmentary terrigenous lithologies occurs in the core of extensive anticlines in the southern sector of the synclinorium.

The overlying undifferentiated Famennian and lower Tournaisian Stages (D3fm-C1t1) are represented by intercalations of limestones, aleurolites and sandstones and subordinate andesite-basalts and their associated tuffs.

The Upper-Tournaisian-Lower-Visean substages (C1t2-v1) are undifferentiated and represented by a unique arenaceous-argillaceous sequence at the base of which argillites, aleurolites, shale and argillosilicon schists with a comminuted carbonaceous component and pyrite occur. The upper part of the sequence is characterised by the occurrence of frequent interlayers of tuffs and tuffstones and, at the top, interlayers and lenses of limestones are encountered.

The 3-4,000m sedimentary-volcanogenic lithologies of the Valeryanovskoy series, comprising the Sarbayskaya, Sokolovskaya and Kurzhunkulskaya formations, lie above the volcanogenic- carbonate- terrigenous sequence of the Upper Devon-Lower Visean (Carboniferous).

19.4.2 Geology of the Karabaytalsky and Sakharovsko-Adaevskiy Sectors

19.4.2.1 Karabaytalsky Sector

Geology and Structure

The Karabaytalsky sector is located on the interface of two structural-formation zones separated by the Livanovskaya fault, a major regional feature striking north-northeast. On the west lies the eastern flank of the Aleksandrovsk–Denisovskoy structural subzone consisting of folded, mainly Silurian and Devonian deposits, whilst the western edge of the Valeryanovskaya structural formation zone is located to the east and comprises predominantly Lower Carboniferous and, to a lesser extent, Permian deposits (Figure 19.2).

Intrusives of different ages and ranging from ultrabasic to acid in composition occur and the area is covered by flat lying Mesozoic-Cenozoic deposits up to 60m thick.

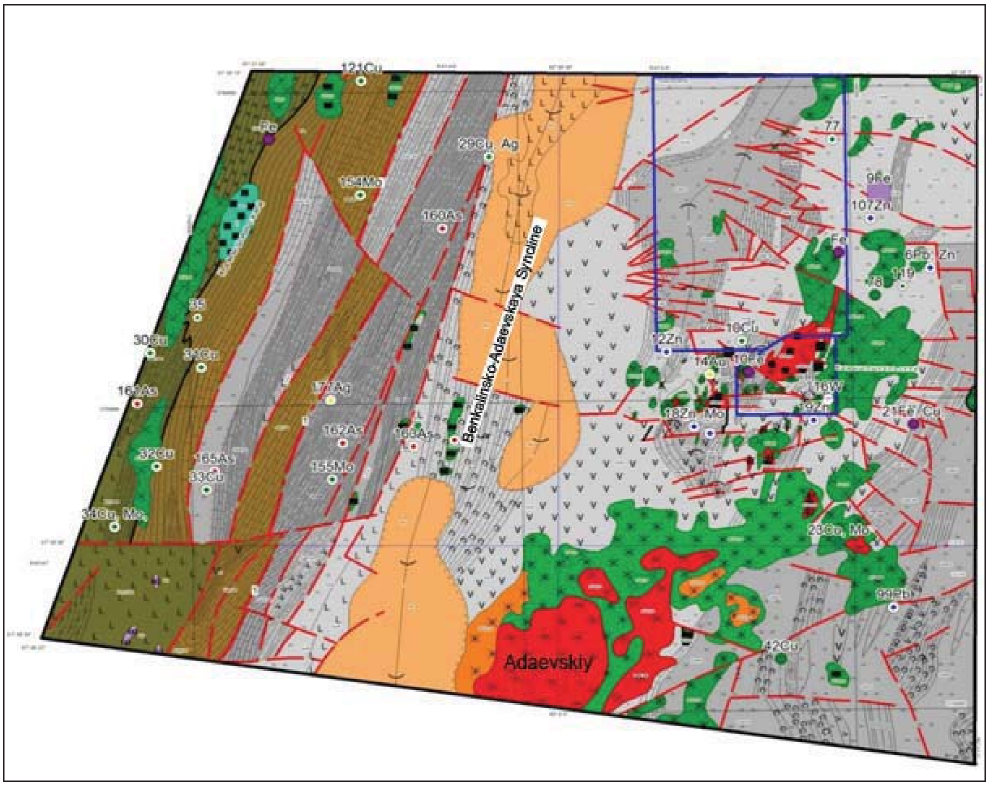


Figure 19.2: Karabaytalsky Sector-Geology showing the Western Edge of the Valeryanovskoy Structural-Formation Zone and the Belozersk (N) and Kamyshlykolskaya (S) Target Areas (Outlined in Blue)

Mineralisation

The Karabaytalsky Sector is located in the south eastern part of the Valeryanovskaya structural-formation zone (synclinorium), includes the southern flank of the main iron ore bearing horizon (skarn-magnetite) of the Torgai basin and is characterised by the presence of numerous deposits and showings of iron, bauxite, copper, polymetallic ores and gold.

Although the iron deposits are significantly inferior to those on the northern flank of the Valeryanovskaya zone, it appears that there is a corresponding improvement in copper deposits.

Amongst non-ferrous metals aluminium has the highest economic value followed in order of abundance by copper, polymetallic, nickel and cobalt deposits, although these are of limited extent, and molybdenum, tungsten, tin, arsenic, antimony and bismuth are even less abundant.

Increased concentrations of the natural associations of Cu-Ni-Co, Cu-Pb-Zn, Pb-Zn, accompanied by rare and noble metals have been identified both in the weathering profiles of Triassic-Lower Cretaceous age and in the volcanogenic-sedimentary thicknesses of the Devonian and Carboniferous, and are encountered in practically all stratified horizons.

These natural concentrations are diverse and relate to different genetic, mineralogical-geochemical and morphological types including magmatic, volcanogenic-sedimentary, skarn-magnetite, hydrothermal, sedimentary and weathering products (lateritic and eluvial). Leached polymetallic, eluvial clay rare-metals, gold-copper porphyry, gold-sulphide-quartz and pyritic polymetallic mineralisation have been identified.

Copper mineralisation is probably associated with acid intrusions; lead-zinc mineralisation, conversely, does not seem to be associated with the composition or presence/absence of volcanogenic lithologies.

19.4.2.2 Sakharovsko-Adaevskiy Sector

Geology and Structure

The major part of the Sakharovsko-Adaevskiy sector is occupied by the central section of the Adaevskaya syncline of the Valeryanovskaya structural formation zone whilst the formations of the eastern flank of the Aleksandrovsk-Denisovskoy structural sub-zone, occupy a narrow strip to the west. The two structural-formation zones are separated, as in the case of the Karabaytalsky Sector, by the Livanovskaya fault (Figure 18.3).

Intrusives of different ages and ranging from ultrabasic to acid in composition occur and the area is covered by flat lying Mesozoic- Cenozoic deposits with a thickness of 100m or more.

Mineralisation

Two areas of mineralisation identified in the Sakharovsko-Adaevskiy sector are Adaevskiy in the north east and Bestyubinskiy located south of Adaevskiy.

In the southern part of the Adaevskiy area, in addition to several copper and polymetallic showings, the Klochkovskoe copper-porphyry prospect has been identified as the most promising target.

The Bestyubinskiy mineralisation is characterised by a significant number of copper and polymetallic showings evidenced by the anomalous halos of copper, lead, zinc and the Kungurtauskim copper occurrence.

Mineralisation is associated with the granodiorite of the Milyutinsko-Mikhajlovsk intrusive complex with the maximum concentration of 0.36- 0.89%Cu, 0.2-3.15g/t Au and 1-3g/t Au in diorite intersected by several boreholes. The maximum content of gold is 6.4g/t whilst the copper content in volcanic rocks is usually from 0.1-0.31% and rarely up to 1.2%.

Gold in economic concentrations in the section has not been established. Nevertheless, borehole intersections at the Klochkovskom and Kungurtauskom demonstrate that gold grades of economic interest are present with copper-molybdenum.

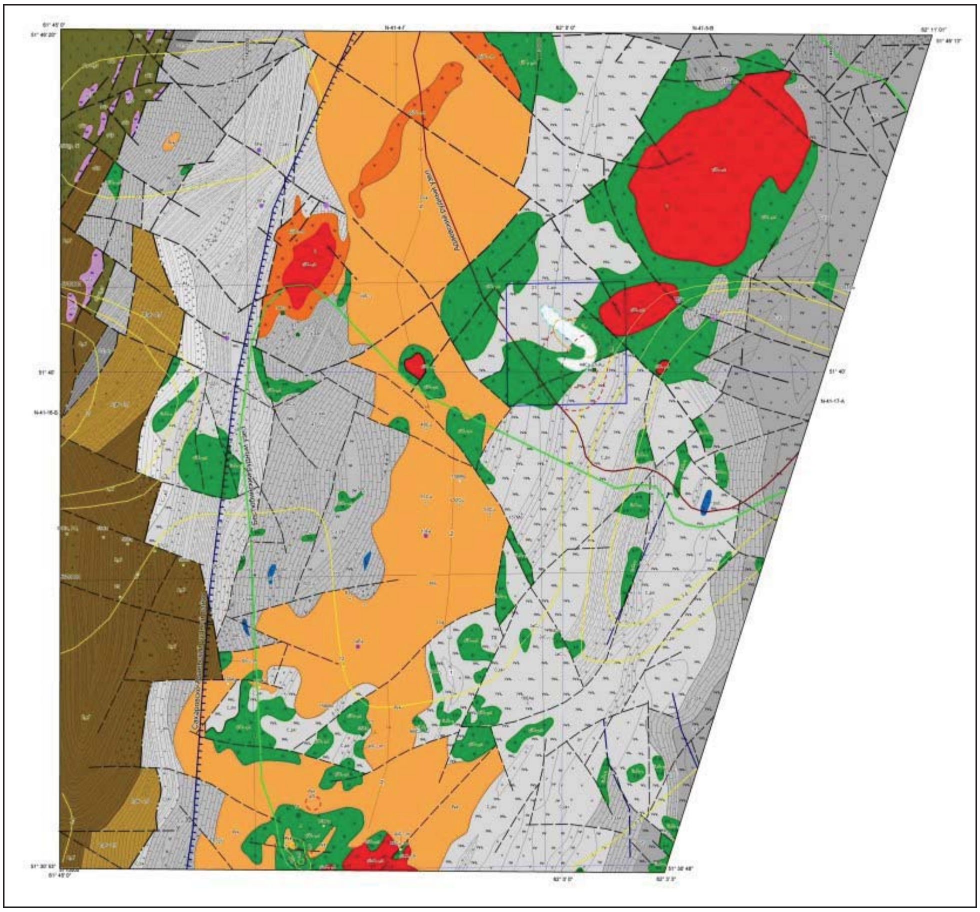


Figure 19.3: Sakharovsko-Adaevskiy Sector and the Klochovskaya Target Area (Blue square) Geology

19.5 Exploration

19.5.1 Introduction

Geological and mineral maps (1:200,000 scale), with brief explanatory notes covering the Western Torgai area, were produced as a result of geological surveys in the period 1954-60.

In the Karabaytalsky Sector the Karabidaiksky copper-molybdenum mineralisation and a number of lead and zinc anomalies, as well as the Karabaytalsky bauxite deposit, were identified. Further work in 1965-66 failed to produce any positive results at Karabidaikskoe.

During 1983-84 a significant number of boreholes (300m), and electrical geophysical surveys were completed and despite the intersection of small mineralised bodies with up to 5% Cu, economic deposits were not discovered, the prognostic resource (P_1) of copper being estimated as 95kt. However, the existence of copper-porphyry type mineralisation and economic concentrations of gold in weathering 'crusts' and karstic depressions was established and led to a recommendation for a continuation of exploration.

The majority of the Sakharovsko-Adaevskogo section has been mapped from aerial surveys on a scale of 1:50,000, with the exception of the eastern area which is covered by an earlier geological map (sheet M-17).

In the Adaevskiy granitoid massif several occurrences of bauxites, magnetite and sulphides of copper, lead and zinc, have been identified, as well as copper-gold porphyry (Kungurtauskoe) and magnetite (South-Sakharovskoe).

Geological and-geophysical exploration in 1971 identified the Klochkovskoe copper-gold mineralisation and in 1981 more detailed investigation established the mineralisation as being copper-porphyry type with up to 2% Cu.

On the basis of geological-geophysical exploration for copper-molybdenum in 1975-78, and the earlier 1:200,000 metallogenic regional map, it was concluded that in general the Valeryanovskaya structure zone copper mineralisation was associated with paleo-volcanic units and copper-porphyry type mineralisation with the contacts of granodioritic massifs.

In 2001-06 a complete geological reappraisal of the area led to the production of revised and updated geological and mineral maps on a scale of 1:50,000 and also maps of the geological-tectonic structure of the plicated Palaeozoic basement and its metallogeny. These served as a basis for the interpretation of the results of magnetic and gravitational surveys.

In 2007 Kazzinc and the Committee of Geology and Subsoil Use of the Ministry of Energy and Mineral Resources of the Republic of Kazakhstan, jointly commenced exploration on the Valeryanovsky structural formation zone of Western Torgai with specific reference to Shaimerden, Sakharovsky-Adaevskiy, Karabaytalsky, Belinsky and Ayatsky areas.

Exploration, focusing on areas prospective for polymetallic and gold mineralisation, was completed by 2009 and identified the optimum techniques for the location of copper-porphyry, polymetallic and precious metal deposits as well as the areas for further exploration work.

At present there are no Mineral Resource estimates in accordance with the guidelines of the JORC Code (2004) for the exploration target areas presented in this section.

19.6 Target Areas

19.6.1 Karabaytalsky

In the Karabaytalsky Sector two promising areas, Belozersk and Kamyshlykolskaya have been identified on the basis of structure and mineralised intersections established from earlier exploration programmes.

19.6.1.1 Belozersk

Numerous favourable factors for the occurrence of possible economic accumulations of both oxide (mainly zinc) and sulphide (pyritic-polymetallic) stratiform deposits are considered to exist over the Belozersk area.

Proposed Exploration Drilling

Inclined boreholes will be drilled on 14 identified promising mineralised zones in the Belozersk target area, with complementary down hole geophysical logging. A total of 26 boreholes, comprising 11,230m, are planned on the 14 zones at Belozersk to a depth of over 500m.

19.6.1.2 Kamyshlykolskaya

The area has been investigated by deep geochemical boreholes on seven east-west sections on an initial grid of 600x200m, with the subsequent infilling to 400x100m, and by three deep investigative boreholes. In geochemical bore holes, gold content varies from 0.1-1.0g/t and occasionally higher (borehole 80: 4.66g/t Au was recorded over 4.6m, including 1.1m at 18g/t Au).

Indications are favourable for the occurrence of copper-porphyry type mineralisation both in the Kamyshlykolskogo intrusive massif and its surrounding lithologies; gold appears to be directly associated with the copper and represents an additional positive feature.

Proposed Exploration Drilling

The presence of friable superficial deposits (≈40m), and the depth and steep dip of the predicted mineralised zones require the drilling of inclined (65°) coring boreholes with associated down hole geophysical surveys. A total of 15 boreholes, comprising 7,830m, on 4 profiles is planned to depths of up to 650m.

19.6.2 Sakharovsko-Adaevskiy

The Klochkovskoe showing of copper-porphyry mineralisation has been delineated as the most promising target area in the Sakharovsko-Adaevskiy Sector in the southern part of the Adaevskogo 'ore' unit.

19.6.2.1 Klochovskaya

In 1974 after completion of a comprehensive programme of geophysical surveys for the identification and evaluation of sulphide mineralisation, a north northwest striking IP anomaly extending for about 5km was delineated. In the centre of this (Klochkovskaya) anomaly, deep geochemistry revealed halos with 0.03-0.5% Cu and up to 0.3g/t Au.

Proposed Exploration Drilling

It is planned to drill 27 holes (total length 13,550m, and depth up to 730m), including a first phase of 9 holes (total length 4,820m) which will be drilled immediately adjacent to the three holes with mineralised intersections from the earlier programme. The completion of the holes will help in the verification of earlier predictions and facilitate any necessary modifications to the second phase.

20 BUKHTARMA HYDRO PLANT

20.1 Introduction

The Bukhtarma Hydroelectric Power Plant is a hydroelectric power plant on the Irtysh River, 5km upstream of the town of Serebryansk, in Eastern Kazakhstan. The plant has 9 individual turbines with a total generating capacity of 675MW. It was designed by a St Petersburg design bureau Lenhydroproject (now part of RusHydro) and constructed during 1953-1968. Since 1997 the plant has been operated by Kazzinc under a long-term concession which finishes in 2022. The owner of the Bukhtarma Hydro Plant is joint stock company Bukhtarmiskaya Hydro Power Plant which is part of "Samruk Kazyna" National Welfare Fund. The plant generates around 2.6 billion kilowatt-hours of electricity per year which covers 100% of the company's electricity needs. It is integrated into the Unified Power Grid of Kazakhstan and provides Kazakh national grid with a peak capacity. Financial and operational activities are the responsibility of Kazzinc and power generation is controlled by KEGOC JSC (Kazakhstan Electricity Grid Operating Company). The hydroelectric power generated by the plant is transmitted by high-voltage electric transmission lines at 220 and 110 kV to the Unified Power Grid of Kazakh

Outline technical details of the plant are given in Table 20.1 below.

Table 20.1: Technical Details of the Bukhtarma Hydro Plant

Unit Rated Capacity	75MW
Generator Rating	94,117kVA @ 0.8
Number of Units	9
Number of Functional Units (as on 18/11/2010)	7
Total Installed Capacity	675MW
Total Functional Capacity	525MW
Turbine weight (1 unit)	281t
Generator weight (1 unit)	655t
Generator rotary parts weight (1 unit)	363t
Generation Voltage	13.8kV
Transmission Voltage	110kV and 220kV

The main objectives of Bukhtarma Hydro Plant are:

- To generate reliable and clean energy for uninterrupted supply for the purposes of mining and processing energy needs of the Kazzinc operations; and
- To provide the Unified Power Grid of Kazakhstan with a peak capacity.

20.2 The Dam

The dam is a reinforced concrete gravity type dam. The intake structure comprises nine intakes permitting flow from the reservoir into the penstocks. Each intake comprises a series of trash racks and gates, each operated by hydraulics.

The dam's crest length is 380m, maximum height is 90m, dam's head is 68m, width on the top of the dam is 19m, width at the bottom is 70m and dam has one 18m long surface spillway span. The maximum height for the spillway gates to open is 5.6m. The dam spillway is equipped with a wave generator which produces waves on the water surface in the winter period to prevent ice formation. The river depth at the bottom of the dam is 12m. The dam has a lock located on the left bank.

20.2.1 Civil Engineering

The dam was robustly constructed to Soviet Russian engineering standards. The dam appears to be operating well and within its capability. There appeared to be no leakage at the abutments and to date the dam had been operating as required with no functional failures advised.

In 2002, 60 samples of dam's concrete were taken from different parts of the dam and tested by two independent labs. The results were the same for both labs: concrete strength increased compared with the samples taken after construction.

The banks of the river downstream of the dam are of rock origin and were reinforced with concrete to prevent erosion and to preserve integrity of the hydro plant.

20.2.2 Mechanical & Electrical

Gantry cranes were reported to be functional.

The design life of some of the electrical and mechanical equipment at the dam is likely to be in the region of 30 to 35 years. Modernisation and change of main items of equipment started in 1999 and is planned to be completed by 2011, the same year the plant is planning to run at full capacity.

20.3 Reservoir

Filling of the reservoir began in 1960; long-term regulation of the discharge has been in effect since 1966. The reservoir consists of two sections: the river section (along the valley of the Irtysh River) and Lake Zaisan, where a wide reach was formed. The reservoir has an area of 5,500km^2, volume of 53km^3, length of more than 500km, maximum width of 35km, and average depth of 9.6m. The Bukhtarma Reservoir is very important in ensuring an increase in the electric power capacity and production of the Bukhtarma and Ust'-Kamenogorsk hydroelectric power plants as well as of the hydroelectric power plants planned for construction lower on the river. Water is discharged from the reservoir every spring to irrigate hundreds of thousands of hectares of floodplains in of the East Kazakhstan city of Semipalatinsk, Pavlodar, and other oblasts. The reservoir creates a deep waterway and improves navigation conditions for ships on the Irtysh up to the city of Omsk in Russia.

Silt levels in the reservoir are low and are not problematic for the operation of a plant. The operational range for reservoir levels at the dam is reported as being around 7m.

20.4 Bukhtarma Lock

Bukhtarma Hydro Plant has a four section lock with a spillway from the second section. The lock is extremely important for shipping on Irtysh river and is not operated by Kazzinc.

20.5 Bukhtarma Plant

The power station is an integrated structure comprising an intake arrangement, turbine hall housing nine generating units, administration building and outlet arrangement. The station has an open switchyard (220kV) and separate building housing 110kV switchgear (see Figure 20.1).

The plant works in a fully automated mode. The plant has the following communication: telephone, trunked radio, power line communication and Kazzinc special channel connection.

20.5.1 Civil Engineering

There are no records of any settlement of the Hydro Station and leakage at abutments is not evident. Some leakage is evident at the upper gallery of the dam but it is believed that such leakage is insufficient to cause any concern.

Figure 20.1: Aerial view of the Bukhtarma Hydro Plant

20.5.2 Turbine, Generator & Auxiliaries

All generators are still in operation since the first installation and were supplied by Novosibirsk Turbogenerator Plant "Sibelectrotyazhmash" (The Science and Production Corporation "ELSIB"). The condition of the generators was reported to be satisfactory for machines of this age and design. Throughout the plant old relay protection equipment was being changed for modern microprocessor based protection. Electrical systems, turbine auxiliaries and control equipment appeared to be functional and gradually being changed for new. New voltage regulators were installed. The plant has a standby power supply and as an automatic monitoring system.

Two types of transformers are being installed: 5 transformers rated at 125MVA 13.8/110kV and 4 transformers rated at 120MVA 13.8/110/220/kV. The manufacturers are Zaporozhtransformator (Ukrane) and Tolyatinsky Transformator (Russia) factories. The transformers are located in a bunded enclosure to prevent oil spillages.

20.6 Access and Security

The station is closed for public access. Access to the plant is made via main security gate. A manned security lodge is located adjacent to the gate. The station has two security check points with guards checking for documents and looking for permission to access the plant. Armed patrols undertake regular walks around the plant. Security firms operating at Bukhtarma Hydro Plant are LLP Berkut Vostok and the specialised security service of East Kazakhstan region.

20.6.1 Roads, Paving and Railway

The station has a railway track access leading to the power house for transport of heavy loads. The road leading to the plant is in good condition and is made of concrete blocks. The roads and paving located within the confines of the plant are in good condition throughout.

20.6.2 Security

The perimeter of the station is protected by an adequate fence.

20.7 Generation Capacity

20.7.1 Reservoir Storage

- Reservoir surface area is – 5,500km^2;
- Total volume – 53km^3
- Useful volume – 30,810Mm^3
- Full Reservoir Level – 403m, and
- Minimum Drawdown Level – 395m.

The operational range for reservoir levels at the dam is reported as being around 7m.

20.7.2 Water Availability

Figure 20.2 shows Bukhtarma reservoir capacity between 2001-2009 and Figure 20.3 shows Bukhtarma Hydro Plant dam water balance for the same period.

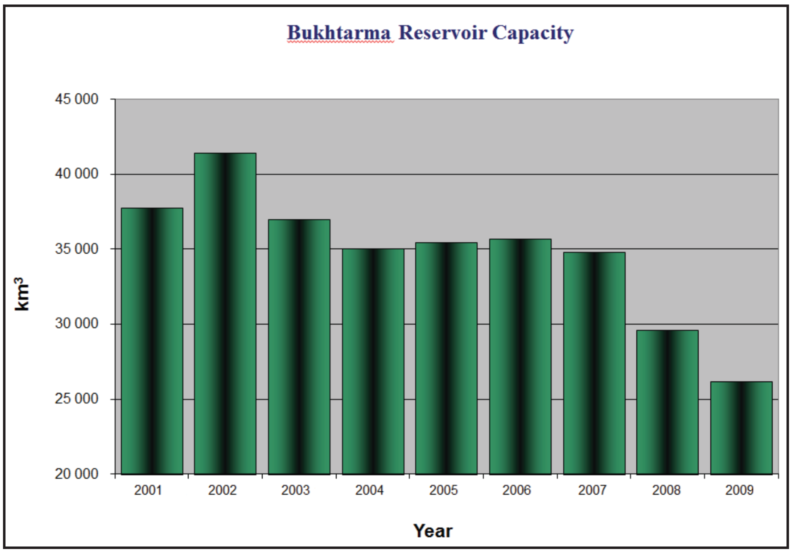


Figure 20.2: Bukhtarma Hydro Plant Reservoir Capacity

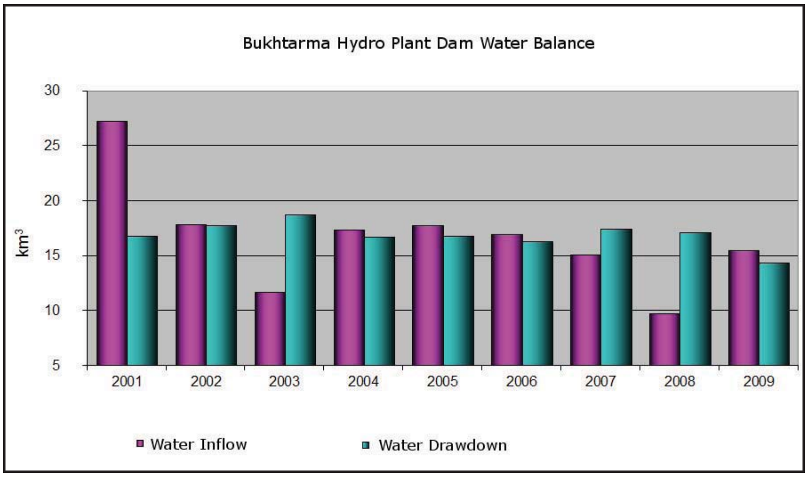


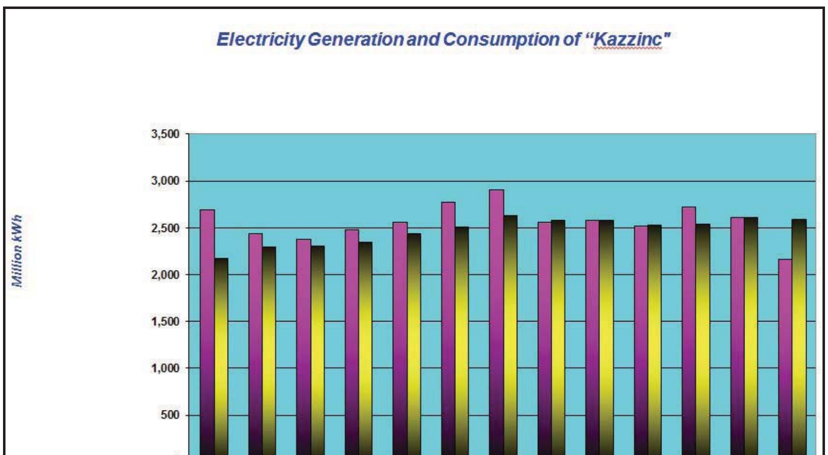
Figure 20.3: Bukhtarma Hydro Plant Dam Water Balance

Water flow through the turbines is approximately 126 – 142m^3/s and through the spillway (when open) is 440 – 975m^3/s. The maximum water flow through the hydro plant is 2,120m^3/s, including a total of 1,140m^3/s through the turbines.

The water usage licence was granted by the Committee for Water Resources of the Ministry of Agriculture of Kazakhstan for a period between 01/06/2010 – 31/12/2012.

20.8 Annual Generation

Since the beginning of its operation, Bukhtarma Hydro Plant has generated 116.5billion kWh of electricity.



	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009
Generation	2,704	2,441	2,381	2,484	2,571	2,777	2911	2,564	2,584	2,530	2,725	2,619	2,170
Consumption	2,184	2,304	2,315	2,350	2,441	2,512	2,635	2,588	2,592	2,537	2,548	2,614	2,602

Figure 20.4: Bukhtarma Hydro Plant Annual Power Generation and Kazzinc Consumption

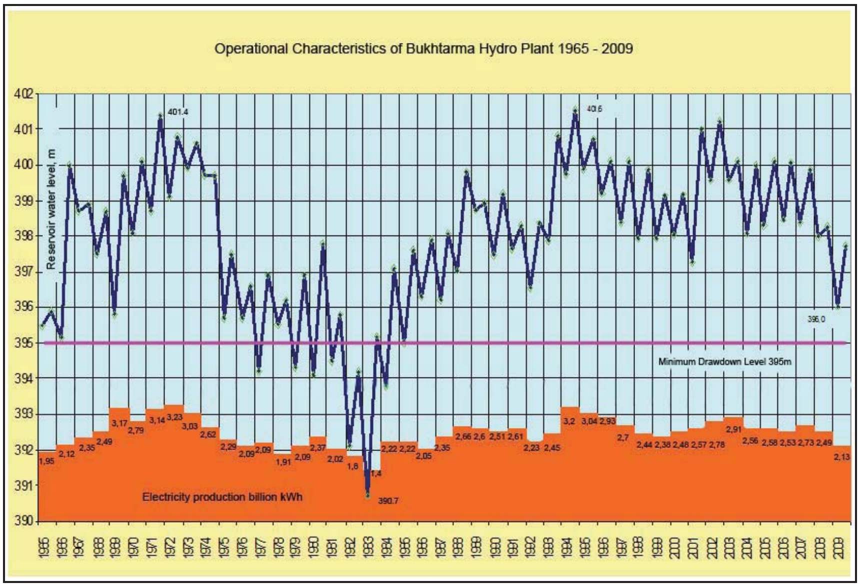


Figure 20.5: Operational Characteristics of Bukhtarma Hydro Plant

The plant's annual power generation and reservoir annual water level are shown in Figure 20.4. As can be seen from Figure 20.5, in 1983 the water level in reservoir was at its lowest because of low water inflow into the reservoir due to a weather anomaly. The possibility of this happening in the future is considered very low.

20.8.1 Capacity

All turbines have been changed for more efficient ones built with better fluid dynamics. As a result, the plant's generating capacity has increased from 675MW to 720MW with the efficiency increasing from 88% to 94.5%.

20.9 Environmental Consideration

In 2006 Kazzinc was certified by TUV under the ISO 14001:2005 management system international standard (valid till 18/05/2012). As part of Kazzinc the Bukhtarma Hydro Plant has implemented this standard also.

20.10 Health and Safety

Safety boards with explanations of what to do in emergency are available throughout the plant. First aid kits are available at the station. Also a health and safety processes and procedures are available in the powerhouse. It was reported that health and safety training sessions are take place frequently. Fire training for each department takes place every six months.

It is believed that no major accidents have happened during the history of the plant apart from a serious accident which took place in 1994 when transformer number 4 caught fire.

20.11 Financial Overview

Construction of the hydro plant cost 139.5M rubles in 1960's. The capital cost per kW installed was 119 rubles.

Total Kazzinc investments into equipment modernisation of the plant can be seen in Figure 20.6. The total capital investments into the plant for the period 1996 – October 2010 is US$93,451,897.11.

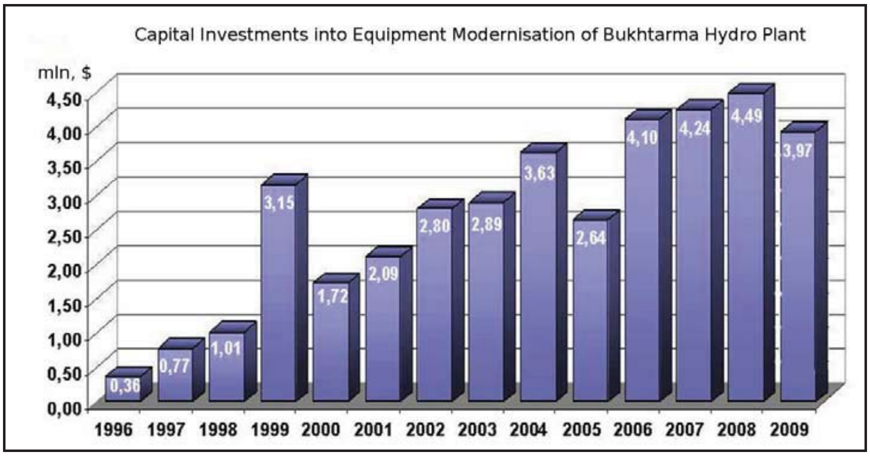


Figure 20.6: Total Investments into Equipment Modernisation

21 VALUATION

21.1 Financial Modelling Parameters

WAI has undertaken a valuation of the Kazzinc operations using discounted cash flow analysis to determine the Net Present Value ("NPV") of both the operations as a whole and each individual mine. A post-tax un-geared cash flow model was constructed for each operation based on the JORC Code (2004) compliant Ore Reserves estimated by WAI.

Ore production rates, operating costs and metallurgical recoveries for each mine were estimated on the performance over the last 3 years along with Kazzinc's forecast for 2011 production. Capital expenditure for 2011 to 2016 is based in Kazzinc's 5 year plan; thereafter, WAI has estimated likely capital expenditure based on the 2008 to 2016 expenditure.

In order to derive an individual valuation for each mine, a Net Smelter Return ("NSR") has been assumed for each concentrate product based on typical refining and treatment charges for that concentrate type. For the combined operations scenario, Kazzinc smelts its own concentrate and therefore benefits from the sale of by-products and other penalty elements, which are not included in the individual mine valuations. Third party concentrates smelted by Kazzinc are also included in the combined scenario but not in the individual mine valuations.

21.1.1 Operating Costs

The main operating costs for each operation have been divided into mining costs, processing costs and general & administrative costs. Group annual average operating costs are summarised in Table 21.1 below.

Table 21.1 Group Average Annual Operating Costs (2011 onwards)

Operation	Units	Mine Operating Costs	Process Operating Costs	General & Administrative Costs	Total Operating Cost
Vasilkovskoye	**US$M/Year**	40.54	71.34	18.17	130.5
Maleevskoye	**US$M/Year**	47.71	34.07	16.76	98.54
Ridder-Sokolniy	**US$M/Year**	69.67	22.63	12.04	104.33
Tishinskiy	**US$M/Year**	49.89	19.54	7.79	77.23
Novoshirokinskoye	**US$M/Year**	17.14	13.11	8.80	39.06
Shaimerden	**US$M/Year**	10.44	4.08	3.52	18.04
Dolinnoe	**US$M/Year**	9.08	2.67	1.40	13.15
Obruchevskoe	**US$M/Year**	8.84	4.09	1.93	14.86
Staroye Tailings	**US$M/Year**	1.06	6.44	3.83	11.33
Chashinskoye Tailings	**US$M/Year**	4.5	28.17	3.10	35.77

Smelter operating costs per tonne of finished metal product are shown in Table 21.2 below.

Table 21.2: Operating Costs at Kazzinc Smelter Complexes

Product	Production Costs (2010 actual)
Ust-Kamenogorsk Metallurgical	
1 tonne 99.99% Zinc	US$372.94
1 tonne 99.99% Lead	US$551.12
1 tonne Blister Copper	US$1016.88
1kg 99.99% Silver Bullion	US$12.26
1kg 99.99% Gold Bullion	US$743.77
Ridder Metallurgical Complex	
1 tonne 99.99% Zinc	US456.72

21.1.2 Capital Expenditure

The mining capital expenditure for 2011 to 2016 has been based on Kazzinc's 5 year plan. This data, as provided by the client, is summarised in Table 21.3, Table 21.4, Table 21.5 and Table 21.6.

Table 21.3 Kazzinc Capital Expenditure 2012-2016 (excluding VAT) (US$M)

Vasilkovskoye	Total	2012	2013	2014	2015	2016
Mining Capex	85.0	25.0	20.0	20.0	20.0	-
Tailings Dam	8.0	-	-	-	8.0	-
Contingency	3.3	2.5	-	-	0.8	-
Total	**96.3**	**27.5**	**20.0**	**20.0**	**28.8**	**-**
Maleevskoye Mine	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Capital Mining Works	79.3	3.7	4.3	4.6	48.9	17.8
Surface Facilities and Other Expenses	5.0	3.3	0.8	0.3	0.3	0.3
Other ZMC Facilities	1.0	0.2	0.2	0.2	0.2	0.2
Total	**85.3**	**7.2**	**5.3**	**5.1**	**49.4**	**18.3**
Ridder-Sokolniy Mine	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Drifting Operations	82.9	36.0	29.4	6.8	4.7	6.0
Stripping, Kryukovsky Section	11.8	-	3.2	4.4	3.2	1.0
Industrial Exploration	9.5	4.1	3.3	2.1	-	-
Bahrushinskaya Deposit	6.0	-	-	2.0	2.0	2.0
Other Surface Facilities	13.2	-	5.1	7.0	1.1	-
Total	**123.4**	**40.1**	**41.0**	**22.3**	**11.0**	**9.0**
Tishinskiy Mine	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Development of Levels 11-21	13.5	1.3	3.0	3.5	3.0	2.7
Capital Mining World Levels 11-16	1.9	1.9	0.6	-	-	-
Total	**16.0**	**3.2**	**3.6**	**3.5**	**3.0**	**2.7**

Table 21.4 Kazzinc Capital Expenditure 2012-2016 (excluding VAT) (US$M)

Novoshirokinskoye	Total	2012	2013	2014	2015	2016
Capital Mining Works	3.5	1.5	2.0	-	-	-
Concentrating Operations	0.5	0.1	0.1	0.1	0.1	0.1
Equipment Concentrating Operation	9.5	0.2	0.6	0.1	1.6	7.0
Infrastructure Facilities Construction	2.1	0.3	0.4	0.3	0.04	1.1
Design Prospecting Works	0.3	0.08	0.08	0.08	0.08	
Total	**15.9**	**2.18**	**3.18**	**0.58**	**1.82**	**8.2**
Dolinnoe & Obruchevskoe	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Mine Construction	82.6	4.3	11.5	18.0	28.9	19.9
Mine Equipment	53.5	-	-	0.4	0.9	52.2
Environmental Facilities Construction	35.7	3.6	9.1	12.4	5.1	5.5
Design Works	6.9	6.9	-	-	-	-
Total	**178.7**	**14.8**	**20.6**	**30.8**	**34.9**	**77.6**

Table 21.5 Kazzinc Capital Expenditure 2012-2016 (excluding VAT) (US$M)						
ZMCC	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Tailings Dump	1.8	-	-	-	1.8	0.04
Modernization of Equipment	27.6	14.1			10.0	3.5
Upgrade of Filtering Equipment	0.6	-	-	0.6		
Ore Treatment Flowsheet Improvement	2.5	0.5	0.5	0.5	0.5	0.5
Upgrade of secondary crushing equipment	1.7	1.7	-	-	-	-
Filtering Equipment Upgrade	1.1	1.1	-	-	-	-
Other	0.5	0.1	0.1	0.1	0.1	0.1
Total	**35.8**	**17.5**	**0.6**	**1.2**	**12.4**	**4.14**
RMCC	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Talovsky Tailings Dump Expansion	3.4	1.7	0.7	-	0.8	0.2
Reconstruction of Talovsky Tailnigs	4.4	1.5	1.6	-	-	1.3
Other Concentrator Facilities	42.5	14.5	0.1	1.4	25.9	0.6
Equipment Upgrade	8	-	-	-	8.0	-
Other	0.5	0.1	0.1	0.1	0.1	0.1
Total	**58.8**	**17.8**	**2.5**	**1.5**	**34.8**	**2.2**

Table 21.6 Kazzinc Capital Expenditure 2012-2016 (excluding VAT) (US$M)						
UKMC	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Zinc Concentrates Roasting Plant	35.7	10.6	6.1	11.3	7.5	0.2
Hydrometallurgical Plant	18.4	-	-	-	0.8	17.6
Electrolysis Plant	22	6.9	7.6	7.5	-	-
Total	**76.1**	**17.5**	**13.7**	**18.8**	**8.3**	**17.8**
Ridder Smelter	**Total**	**2012**	**2013**	**2014**	**2015**	**2016**
Electrolytic Plant	2.6	1.6	0.5	0.5	-	-
Integrated Plant No.1	13.3	3.7	2.3	2.3	2.5	2.5
Waelz Plant	6.6	3.2	3.4	-	-	
Hydrometallurgical Plant	1.0	1.0				
Lead Smelter	12.8	6.4	1.7	3.9	0.3	0.5
Total	**36.3**	**15.9**	**7.9**	**6.7**	**2.8**	**3.0**

21.1.3 Commodity Prices

WAI has modelled each cash flow based on Glencore commodity price assumptions, as summarised in the Table 21.7.

Table 21.7 Glencore Commodity Price Assumptions											
		2011	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**
Zinc	**US$/t**	2,500	2,475	2,431	2,388	2,326	2,228	2,195	2,162	2,130	2,100
Lead	**US$/t**	2,450	2,401	2,353	2,305	2,258	2,181	2,126	2,095	2,042	2,000
Copper	**US$/t**	9,600	9,208	8,822	8,347	7,880	7,240	6,859	6,397	6,036	6,000
Gold	**US$/tr. oz**	1,350	1,343	1,336	1,330	1,323	1,284	1,189	876	863	850
Silver	**US$/tr. oz**	29.00	28.89	28.79	28.68	28.58	27.78	20.12	14.42	14.21	14.00

21.1.4 Discount, Inflation and Taxation Rates

The purpose of the discount rate is to reflect not only the time value of money but also the investment risk of the project. Traditionally, mining type projects use higher than average discount rates, in the 10-15% range. This is due to the often longer term nature of the projects as well as the perceived higher level of risk to the investor. An annual discount factor of 10% has been applied to the Kazzinc cash flow.

Cost inflation has not been applied to the cash flow model on operating or capital costs.

The main taxation applied to the financial analysis is an excess profit tax applied to net income at the rate of 20% in 2011 and 2012, reducing to 17.5% in 2013 and 15% thereafter, in line with Kazakh tax policy. A recoverable Value Added Tax (VAT) at a rate of 12% is applied to all capital expenditure. Production royalties, land and property taxes and transport taxes are applied as operating costs.

21.2 Economic Statistics

A summary cash flow forecast for the operations as a whole and each individual mine - as modelled by WAI and using Glencore Commodity Price Assumptions - are presented in the following sections.

21.2.1 Combined Kazzinc Operations

A summary of the combined Kazzinc operations cash flow forecast, including third party concentrates, between 2011 and 2027 is provided in the table below. Total operating costs for the operations are estimated at US$9,941.5M with a cumulative operating free cash flow of US$8,395.2M and EBIT of US$10,186.1M. The operations generate a positive NPV of **US$4,513.8M.**

Table 21.8 Kazzinc Combined Operations - Summary Economic Statistics

Summary	Units	Value
Ore Mined	**Mt**	263.2
Zn Concentrate Produced	**kt**	6,490.1
Pb Concentrate Produced	**kt**	1,127.1
Cu Concentrate Produced	**kt**	1,209.6
Au Concentrate Produced	**kt**	1,085.4
Zinc Metal Recovered	**kt**	2,628.0
Lead Metal Recovered	**kt**	610.9
Zinc Metal Recovered	**kt**	314.1
Gold Metal Recovered	**t**	278.1
Silver Metal Recovered	**t**	1,523.9
Gross Revenue Generated	**US$M**	21,330.8
Operating Costs	**US$M**	9,941.5
Capital Expenditure	**US$M**	1,323.2
Depreciation	**US$M**	1,203.2
Cash Taxes	**US$M**	1,671.0
Total Operating Free Cash Flow	**US$M**	**8,395.2**
EBIT	**US$M**	**10,186.1**
NPV	**US$M**	**4,513.8**

21.2.2 Vasilkovskoye

A summary of the Vasilkovskoye forecast cash flow between 2011 and 2026 is provided in the table below. Total operating costs for the life of mine are estimated at US$2,072.1M , with a cumulative operating free cash flow of US$2,595.1M and EBIT of US$3,028.8M. The project generates a positive NPV of **US$1,466.6M**.

Table 21.9 Vasilkovskoye Summary Economic Statistics (as modelled by WAI)

Summary	Units	Value
Ore Mined	**Mt**	124.0
Gold Recovered	**koz**	5,617.8
Gross Revenue Generated	**US$M**	5,675.7
Operating Costs	**US$M**	2,072.1
Capital Expenditure	**US$M**	222.0
Depreciation	**US$M**	287.7
Cash Taxes (20%)	**US$M**	492.6
Cumulative Operating Free Cash Flow	**US$M**	2,595.1
EBIT	**US$M**	3,028.8
NPV	**US$M**	1,466.6

21.2.3 Maleevskoye

A summary of the Maleevskoye Mine forecast cash flow between 2011 and 2019 is provided in Table 21.10. Life of mine operating costs are estimated at US$633.0M, with cumulative operating free cash flow of US$1,420.7M and EBIT at US$1,825.4M. The Maleevskoye Mine generates an NPV of **US$1,075.3M**.

Table 21.10 Maleevskoye Summary Economic Statistics (as modelled by WAI)

Summary	Units	Value
Ore Mined	**Mt**	12.1
Zinc Recovered	**kt**	744.6
Lead Recovered	**kt**	149.8
Copper Recovered	**kt**	131.5
Gold Recovered	**koz**	158.0
Silver Recovered	**koz**	19,574.0
Gross Revenue Generated	**US$M**	2,393.3
Operating Costs	**US$M**	633.0
Capital Expenditure	**US$M**	146.3
Depreciation	**US$M**	61.6
Cash Taxes (20%)	**US$M**	325.7
Cumulative Operating Free Cash Flow	**US$M**	1,420.7
EBIT	**US$M**	1,825.4
NPV	**US$M**	1,075.3

21.2.4 Ridder-Sokolniy

A summary of the Ridder-Sokolniy forecast cash flow between 2011 and 2019 is provided in Table 21.11. The cash flow indicates life of mine operating costs of US$1,028.5M, cumulative operating free cash flow of US$221.5M and an EBIT of US$304.6M. Ridder-Sokolniy generates an NPV of **US$134.4M.**

Table 21.11 Ridder-Sokolniy Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	21.1
Zinc Recovered	**kt**	100.2
Lead Recovered	**kt**	57.9
Copper Recovered	**kt**	63.5
Gold Recovered	**koz**	837.2
Silver Recovered	**koz**	5,592.8
Gross Revenue Generated	**US$M**	1,469.5
Operating Costs	**US$M**	1,028.5
Capital Expenditure	**US$M**	43.5
Depreciation	**US$M**	10.4
Cash Taxes (20%)	**US$M**	50.1
Cumulative Operating Free Cash Flow	**US$M**	221.5
EBIT	**US$M**	304.6
NPV	**US$M**	134.4

21.2.5 Tishinskiy

A summary of the Tishinskiy Deposit forecast cash flow between 2011 and 2031 is presented in Table 21.12. The cash flow indicates life of mine operating costs of US$1,877.3, with a cumulative operating free cash flow of US$299.5 and EBIT at US$473.9. The project generates a positive NPV of **US$220.6M**.

Table 21.12 Tishinskiy Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	25.1
Zinc Recovered	**kt**	1,015.4
Lead Recovered	**kt**	125.6
Copper Recovered	**kt**	75.5
Gold Recovered	**koz**	338.0
Silver Recovered	**koz**	5,234.9
Gross Revenue Generated	**US$M**	2,601.8
Operating Costs	**US$M**	1,877.3
Capital Expenditure	**US$M**	161.8
Depreciation	**US$M**	84.3
Cash Taxes (20%)	**US$M**	96.9
Cumulative Operating Free Cash Flow	**US$M**	299.5
EBIT	**US$M**	473.9
NPV	**US$M**	220.6

21.2.6 Novoshirokinskoye

A summary of the Novoshirokinskoye Mine forecast cash flow between 2011 and 2027 is provided in the table below. Operating costs for the life of mine are estimated at US$569.3M, with a cumulate operating free cash flow of US$324.7M and EBIT of US$392.7M. The NPV of the project as calculated from the WAI model is **US$187.3M**.

Table 21.13: Novoshirokinskoye Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	7.3
Zinc Recovered	**kt**	84.3
Lead Recovered	**kt**	176.0
Copper Recovered	**kt**	17.2
Gold Recovered	**koz**	650.3
Silver Recovered	**koz**	13,029.9
Gross Revenue Generated	**US$M**	1,136.8
Operating Costs	**US$M**	569.3
Capital Expenditure	**US$M**	33.8
Depreciation	**US$M**	29.7
Cash Taxes (20%)	**US$M**	63.9
Cumulative Operating Free Cash Flow	**US$M**	324.7
EBIT	**US$M**	392.7
NPV	**US$M**	187.3

21.2.7 Shaimerden

A summary of the Shaimerden Mine forecast cash flow between 2011 and 2020, as modelled by WAI, is provided in the table below. The cash flow indicates life of mine operating costs estimated at US$216.6M, cumulative operating free cash flow of US$462.2M and an EBIT of US$535.5M. The Shaimerden Mine generates an NPV of **US$287.3M**.

Table 21.14: Shaimerden Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	2.5
Zinc Recovered	**kt**	539.2
Gross Revenue Generated	**US$M**	804.7
Operating Costs	**US$M**	216.6
Capital Expenditure	**US$M**	10.1
Depreciation	**US$M**	19.0
Cash Taxes (20%)	**US$M**	86.5
Cumulative Operating Free Cash Flow	**US$M**	462.2
EBIT	**US$M**	535.5
NPV	**US$M**	287.3

21.2.8 Dolinnoe

A summary of the Dolinnoe project forecast cash flow between 2011 and 2033, as modelled by WAI, is provided in the table below. The cash flow indicates life of mine operating costs estimated at US$255.5M, cumulative operating free cash flow of US$-67.8M and an EBIT of US$-78.8. The Dolinnoe project generates an NPV of **US$10.2M**.

Table 21.15: Dolinnoe Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	4.6
Zinc Recovered	**kt**	53.4
Lead Recovered	**kt**	28.0
Copper Recovered	**kt**	7.7
Gold Recovered	**koz**	169.3
Silver Recovered	**koz**	221.4
Gross Revenue Generated	**US$M**	285.7
Operating Costs	**US$M**	255.5
Capital Expenditure	**US$M**	82.6
Depreciation	**US$M**	82.6
Cash Taxes (20%)	**US$M**	-
Cumulative Operating Free Cash Flow	**US$M**	-67.8
EBIT	**US$M**	-78.8
NPV	**US$M**	10.2

21.2.9 Obruchevskoe

A summary of the Obruchevskoe project forecast cash flow between 2011 and 2028, as modelled by WAI, is provided in the table below. The cash flow indicates life of mine operating costs estimated at US$235.0M, cumulative operating free cash flow of US$355.1M and an EBIT of US$426.0M. The Obruchevskoe project generates an NPV of **US$65.8M.**

Table 21.16: Obruchevskoe Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	4.1
Zinc Recovered	**kt**	281.8
Lead Recovered	**kt**	115.3
Copper Recovered	**kt**	33.4
Gold Recovered	**koz**	107.5
Silver Recovered	**koz**	94.0
Gross Revenue Generated	**US$M**	849.9
Operating Costs	**US$M**	235.0
Capital Expenditure	**US$M**	165.3
Depreciation	**US$M**	165.3
Cash Taxes (20%)	**US$M**	70.9
Cumulative Operating Free Cash Flow	**US$M**	355.1
EBIT	**US$M**	426.0
NPV	**US$M**	65.8

21.2.10 Staroye Tailings

A summary of the Staroye Tailings forecast cash flow between 2011 and 2021, as modelled by WAI, is provided in the table below. The cash flow indicates estimated life of mine operating costs of US$99.8M, cumulative operating free cash flow of US$54.6M and EBIT of US$64.7M. The NPV of the project as calculated from the WAI model is **US$36.0M**.

Table 21.17: Staroye Tailings Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	6.9
Zinc Recovered	**kt**	23.5
Lead Recovered	**kt**	9.4
Copper Recovered	**kt**	2.0
Gold Recovered	**koz**	132.3
Silver Recovered	**koz**	1,454.8
Gross Revenue Generated	**US$M**	207.7
Operating Costs	**US$M**	99.8
Capital Expenditure	**US$M**	-
Depreciation	**US$M**	-
Cash Taxes (20%)	**US$M**	10.8
Cumulative Operating Free Cash Flow	**US$M**	54.6
EBIT	**US$M**	64.7
NPV	**US$M**	36.0

21.2.11 Chasihinskoye Tailings

A summary of the Chashinskoye Tailings forecast cash flow between 2011 and 2026, as modelled by WAI, is provided in the table below. Life of mine operating costs are estimated at US$515.1M, with a cumulative operating free cash flow of US$390.9M and EBIT of US$461.3M. The Net Present Value of the project as calculated from the WAI model is **US$188.4M.**

Table 21.18: Chashinskoye Summary Economic Statistics (as modelled by WAI)		
Summary	**Units**	**Value**
Ore Mined	**Mt**	55.5
Gold Recovered	**koz**	1,076.5
Silver Recovered	**koz**	3,659.2
Gross Revenue Generated	**US$M**	1,040.1
Operating Costs	**US$M**	515.1
Capital Expenditure	**US$M**	20.0
Depreciation	**US$M**	20.0
Cash Taxes (20%)	**US$M**	70.5
Cumulative Operating Free Cash Flow	**US$M**	390.9
EBIT	**US$M**	461.3
NPV	**US$M**	188.4

22 GLOSSARY OF TERMS

"°"	Degrees.
"°C"	Degrees Celsius.
"accretionary prism"	A mass of sediment and oceanic crust that is transferred from a subducting plate to the less dense, overriding plate with which it converges.
"acid"	An igneous or volcanic rock containing more than about 60% silica (SiO_2) by weight, most of the silica being in the form of silicate minerals, but with the excess of about 10% being free quartz.
"acid rock drainage" or "acid mine drainage"	Refers to the outflow of acidic water from metal mines or coal mines.
"actinolite"	A metamorphic ferromagnesian mineral of the amphibole group; forms a series with tremolite.
"adit"	A horizontal or sub-horizontal underground development providing access to underground workings from surface.
"adularia"	A colourless, moderate to low-temperature variety of orthoclase feldspar.
"aero-magnetic"	A geophysical prospecting (by air) method that maps variations in the magnetic field of the Earth that are attributable to changes of structure or magnetic susceptibility in certain near-surface rocks.
"Ag"	The chemical symbol for the element silver.
"agate"	A very fine grained form of silica.
"agglomerates"	Coarse accumulations of large blocks of volcanic material.
"Al"	Chemical symbol for aluminium.
"albite"	Sodic feldspar, Na(AlSi3O8); variety of plagioclase feldspars.
"albitization"	The development of the mineral albite in a rock as a result of metasomatism.
"allochthonous"	Formed or produced elsewhere than in its present place.
"alkaline"	A term applied to igneous rocks in which the feldspar is dominantly sodic and/or potassic.
"alluvial"	*Detrital* material which is transported by a river and deposited at points along the flood plain of a river.
"Alpine"	Alpine orogeny mountain-building event that affected a broad segment of southern Europe and the Mediterranean region during the middle Tertiary Period.
"alteration"	Changes in the chemical or mineralogical composition of a rock, generally produced by weathering or hydrothermal solutions.
"aluminosilicates"	Minerals consisting of aluminium and silica.
"alunite"	A trigonal mineral, $KAl_3(OH)_6(SO_4)_2$; massive or disseminated; in pale tints; formed from sulphuric acid acting on potassium feldspar in volcanic regions (alunisation), and around fumaroles. Formerly called alumstone, alum rock, alumite.
"amethyst"	A violet/purple variety of quartz.
"amphibole"	A mineral group characterised by double chains of silica tetrahedral, in the

	orthorhombic or monoclinic crystal systems.
"amphibolite"	A faintly foliated metamorphic rock developed during regional metamorphism.
"amphibolization"	The development of amphibole group minerals, usually at the expense of other minerals in the rock.
"amydaloidal"	Containing cavities in lava formed by the evolution of gas.
"andesite"	A fine-grained igneous rock with no quartz or orthoclase, composted of about 75% plagioclase feldspars, balance ferromagnesian silicates.
"anglesite"	A mineral in the supergene parts of lead-ore veins, a minor ore of lead.
"anhedral"	A term applied to mineral grains having no development of crystal form whatsoever.
"ankerite"	Is a calcium, iron, magnesium, manganese carbonate mineral with formula: $Ca(Fe,Mg,Mn)(CO_3)_2$.
"anticline"	A fold that is convex up and has its oldest beds at its core.
"anticlinorium"	A series of anticlines and synclines, so grouped that taken together they have the general outline of an arch.
"antiform"	An antiform is a fold that is convex up in which the stratigraphic sequence is not known.
"antimony"	An extremely brittle metal with a flaky, crystalline texture. Chemical symbol, Sb. Sometimes found native, but more frequently as the sulphide, stibnite.
"aphanitic"	Textural term used to describe igneous rocks such as fine grain size that the individual constituents are not visible to the naked eye.
"aphyric"	Said of the texture of a fine-grained or aphanitic igneous rock that lacks phenocrysts.
"apical"	Apex or tip.
"aplite"	A light-coloured igneous rock characterised by a fine-grained saccharoidal (i.e. aplitic) texture.
"apophyses"	A term applied to a body of igneous rock.
"Arc (Island Arc)"	A series of volcanoes that lie on the continental side of an oceanic trench, resulting from the subduction process.
"arenaceous"	A sedimentary rock consisting of wholly or in part sand size fragments of rock.
"argentite"	A silver sulphide mineral.
"argillic/argillite"	Pertaining to clay or clay minerals; e.g. argillic alteration in which certain minerals of a rock are converted to minerals of the clay group.
"arkosic"	A detrital sedimentary rock formed by cementation of individual grains of sand size and predominantly composed of quartz and feldspar. Derived from disintegration of granite.
"arsenic"	Metallic, steel-grey, brittle element. Chemical symbol, As.
"arsenopyrite"	Monoclinic mineral, 8[FeAsS]; metallic silver-white to steel grey; the most common arsenic mineral and principal ore of arsenic; occurs in many sulphide ore deposits.
"As"	Chemical symbol for arsenic.
"asl"	Above sea level.

"ASTER"	Advanced Spaceborne Thermal Emission and Reflection Radiometer) is an imaging instrument flying on Terra, a satellite launched in December 1999 as part of NASA's Earth Observing System.
"atomic absorption"	A wet chemical assay method.
"Au"	The chemical symbol for the element gold.
"aureole"	A circular or crescentic distribution pattern about the source or origin of a mineral, ore, mineral association or petrographic feature.
"aurichalcite"	A mineral found in oxidised zones of copper and zinc deposits. A guide to zinc ore.
"auriferous"	Pertaining to gold.
"axial trend"	A plane through a rock fold that includes the axis and divides the fold as symmetrically as possible.
"azimuth"	An angular measurement in a spherical coordinate system.
"azurite"	A supergene mineral in oxidised zones of copper deposits, an ore of copper.
"Ba"	Chemical symbol of Barium.
"ball mill"	A rotating drum containing steel balls used to grind ore.
"barite"	An orthorhombic mineral, 4[$BaSO_4$]; occurs as masses of crystals with sand and clay (desert roses); in veins or in residual masses on limestone; the principal source of barium.
"barren"	Of rock or vein material containing no minerals of value.
"basalt"	A fine-grained igneous rock dominated by dark-coloured minerals, consisting of plagioclase feldspars (over 50%) and ferromagnesian silicates.
"base metals"	Any of the more common and more chemically active metals, e.g., lead, copper.
basement"	Oldest rocks exposed in an area.
"batholith"	A discordant pluton that increases in size downward, has no determinable floor, and shows an area of surface exposure exceeding 100km^2.
"Bauxite"	A residual deposit of hydrated aluminium oxides formed under special climatic conditions in tropical regions. The most important aluminum ore.
"bedding"	A collective term used to signify existence of beds or layers, in sedimentary rocks.
"bedrock"	A mining term for the un-weathered rock below the soil.
"Bi"	Chemical symbol for bismuth.
"bioleach"	The use of bacteria to leach gold from sulphide-rich ores.
"biotite"	A monoclinic mineral of the ferromagnesian micas.
"bismuth"	A white crystalline, brittle metal with a pink tinge. Chemical symbol, Bi.
"block caving"	A general term that refers to a mass mining system where the extraction of the ore depends largely on the action of gravity.
"borehole intersection"	That part of a drilled hole that contains the economic mineralised section.
"bornite"	A copper ore mineral, Cu_5FeS_4; often found in hydrothermal veins.
"boudinage"	A structure formed by the stretching, thinning, and breaking at regular intervals into bodies resembling boudins or sausages. They form elongated parallel to the fold axes.

"breccia"	Clastic rock made up of angular fragments of such size that an appreciable percentage of rock volume consists of particles of granule size or larger.
"Bt"	Billion tonnes.
"bulk density"	The weight of a material (or object) divided by its volume.
"bullion"	Refined bars/ingots of gold or silver.
"bund(s)"	Any artificial embankment used to control the flow of water.
"cadmium"	Soft, bluish-white metal, chemical symbol, Cd.
"calcareous"	A substance that contains calcium carbonate. When applied to a rock name, it implies that as much as 50% of the rock is calcium carbonate.
"calcine"	Ore or concentrate after treatment by calcination or roasting and ready for smelting.
"calcite"	Mineral composed of calcium carbonate, CaCO3.
"caldera"	Roughly circular, steep-sided volcanic basin with diameter at least three or four times depth.
"Caledonian"	Major mountain building episode which took place during the lower Paleozoic Era.
"Cambrian"	Geologic period of time from 590 to 505Ma.
"carbonate"	Refers to a carbonate mineral such as calcite $CaCO_3$.
"carbonatite"	Intrusive or extrusive igneous rocks defined by mineralogy that comprises more than 50% volume carbonate minerals. Carbonatites may be confused with marble, and may require geochemical verification.
"Carboniferous"	A period of geologic time from about 345 to 280Ma.
"Category A"	Soviet "ore reserves" where the reserves in place are known in detail. The boundaries of the deposit have been outlined by trenching, drilling, or underground workings. The quality and properties of the ore are known in sufficient detail to ensure the reliability of the projected exploitation.
"Category B"	Soviet "ore reserves" where blocks are delineated by mine workings on three or more sides (approximately equivalent to *Measured* Mineral Resources under the JORC. CIM and IOM[3] Reporting Codes).
"Category C_1"	Soviet 'ore reserves" whose blocks are delineated by mine workings above and below (approximately equivalent to *Indicated* Mineral Resources under the JORC, CIM and IOM[3] Reporting Codes).
"Category C_2"	Soviet "ore reserves extrapolated from Category C_1 but with more complex geology or limited mine workings (approximately equivalent to *Inferred* Mineral Resources under the JORC, CIM and IOM[3] Reporting Codes).
"Category P_{1-3}"	Soviet "Prognostic" ore reserves extrapolated beyond more definable reserves and resources. The category is subdivided into three sub-categories P_1 to P_3, with the level of confidence decreasing progressively from sub category 1 to 3.
"Cathodes"	An electrode through which electric current flows out of a polarized electrical device.
"Cd"	Chemical symbol for cadmium.

"Cenozoic"	Era of geologic time, from the beginning of the Tertiary period to the present, considered to have begun about 65 million years ago.
"cerussite"	An aragonite group mineral found in the oxidised and carbonated parts of lead-ore veins. A source of lead.
"chacocite"	A copper ore, chemical symbol Cu_2S, found mainly in the enriched zones of sulphide deposits.
"chalcedony"	General name applied to fibrous cryptocrystalline silica with waxy lustre. Deposited from aqueous solutions and frequently found lining or filling cavities in rocks.
"chalcopyrite"	The mineral sulphide of iron and copper, CuFeS.
"chalcostibnite"	A subsidiary sulphide ore mineral of copper and antimony.
"channel samples"	Continuous rock-samples, where an even channel is cut into the rock to obtain the sample. If competently sampled, the quality of such sampling is comparable to drill-hole assays.
"chemical assay"	To analyse the proportions of metals in an ore via chemical analysis.
"chert"	Very fine grained silica.
"chlorite"	Tetrahedral sheet silicates of iron, magnesium, and aluminium, characteristic of low-grade metamorphism. Green colour, with cleavage like mica.
"chloritisation"	Alteration of rocks to chlorite as a result of low-grade metamorphism.
"chromite"	Mineral oxide of iron and chromium, $FeCr_2O_4$, only ore of commercial chromium. One of first minerals to crystallize from magma.
"chromium"	Rare mineral occurring in contact zones between ultramafic rocks and marble.
"chrysocolla"	A soft bluish green mineral forming in incrustations and thin seams in oxidised parts of copper mineral veins, a source of copper.
"CIL"	A recovery process in which a slurry of gold ore, carbon granules and cyanide are mixed together. The cyanide dissolves the gold content and the gold is adsorbed on the carbon. The carbon is subsequently separated from the slurry for further gold removal.
"CIM"	Canadian Institute of Mining, Metallurgy and Petroleum.
"CIP"	Similar to carbon-in-leach process, but initially the slurry is subjected to cyanide leaching in separate tanks followed by carbon-in-pulp. Carbon-in-pulp is a sequential process whereas carbon-in-leach is a simultaneous process.
"CIS"	Commonwealth of Independent States.
"clast"	A particle of broken-down rock. These fragments may vary in size from boulders to silt-sized grains.
"clasts"	Fragments of pre-existing rocks.
"cleavage – mineral"	Property possessed by many minerals of breaking in certain preferred direction along smooth plane surfaces.
"cleavage – rock"	Property possessed by certain rocks of breaking with relative ease along parallel planes or nearly parallel surfaces.
"clinochlore"	A chlorite group mineral occurring in greenschists.
"clinopyroxene"	A group name for monoclinic pyroxenes.

"CN"	Chemical symbol for Cyanide.
"CO_2"	Carbon dioxide.
"cobalt"	A tough, lustrous, nickel-white or silvery gray metallic element often associated with nickel, silver, lead, copper and iron ores.
"cockade"	Concentric bands surrounding earlier fragments of host rock or vein.
"coke"	Bituminous coal from which the volatile constituents have been driven off by heat.
"colbaltine"	Cobalt sulfarsenide (CoAsS).
"colloform"	Curved or contorted bands on various scales defined by variable composition, grain form and size.
"colloidal"	A dispersion of ultra-fine particles suspended in a dispersion medium.
"conglomerate"	Detrital sedimentary rock made up of more or less rounded fragments of such size that an appreciable percentage of volume of rock consists of particles of granule size or larger.
"Cordillera"	Mountain chain.
"costean"	Trench cut across the conjectured line of outcrop of a seam or orebody to expose the full width. In prospecting, to dig shallow pits or trenches designed to expose bedrock.
"covellite"	A copper ore mineral, chemical symbol CuS, found in the zones of secondary enrichment of copper veins.
"cratons"	Parts of the earth's crust which have attained stability, and have been deformed little for a long period.
"Cretaceous"	Geologic period of time from 144 to 65Ma.
"cross-cut"	A tunnel driven often perpendicularly to intersect underground mineralisation.
"crustiform"	Successive sub parallel bands defined by variable mineral compositions, grain form and size. The bands are usually sub parallel to the vein wall.
"cryptocrystalline"	Very fine grained rock in which the crystals are so small that they are indistinguishable by the naked eye.
"Cu"	The chemical symbol for copper.
"cuprite"	A red mineral found in oxidized parts of copper veins, and important source of copper.
"cut-off grade"	Lowest grade of mineralised material considered economic, used in the calculation of ore resources.
"cyanidation"	Is a metallurgical technique for extracting gold by leaching from low-grade ore, converting the gold to water soluble aurocyanide metallic complex ions.
"cyanide leach"	Chemical extraction method using a dilute cyanide solution to leach gold from the mineralisation.
"dacite"	Fine-grained igneous rock with composition between rhyolite and trachyte.
"Datamine®"	Complex mining software used primarily for orebody modelling, resource estimation and pit optimisation.
"deposit"	Coherent geological body such as a mineralised body.
"detrital"	Term applied to any particles of minerals or rocks which have been derived from

	pre-existing rock by the processes of erosion.
"Devonian"	Geological period of time from 408 to 362Ma.
"dextral"	Term applied to a fault to describe the apparent direction of relative movement of each side, in this case to the right.
"diabase"	Metamorphosed medium-grained igneous rock (see dolerite).
"diamond drill hole"	Hole made by a rotary drill using diamond-edged bits which produces a solid continuous core sample of the rock.
"diamond drilling"	Drilling method which obtains a cylindrical core of rock by drilling with an annular bit impregnated with diamonds.
"diatreme"	A volcanic vent piercing sedimentary strata, usually the result of an explosive eruption.
"dilution"	The amount of barren or low-grade material that has to be extracted to recover the ore.
"diopside"	A mineral of the pyroxene group, $CaMgSi_2O_6$.
"diorite"	Coarse-grained igneous rock with composition of andesite (no quartz or orthoclase), composed of 75% plagioclase feldspars and balance ferromagnesian silicates.
"dip"	The true dip of a plane is the angle it makes with the horizontal plane.
"discount rate"	Is the value used in accounting procedures to determine the present value of future cash flows arising from a project, i.e. the discounted value of all future cashflows.
"dislocation"	Displacement, general term to describe a break in the strata.
"disseminated"	Of a mineral deposit in which the desired minerals occur as scattered particles in the rock, but in sufficient quantity to make the deposit an ore.
"dolerite"	*See diabase*
"dolomitized"	A limestone which has been wholly converted to dolomite rock or dolomitic limestone by the replacement of calcium carbonate (calcite).
"doré"	Unrefined silver that contains a variable but usually appreciable percentage of gold.
"drawpoint"	A spot where gravity fed ore is loaded into a hauling unit.
"drift and fill"	A type of cut and fill mining method that uses fill to develop the stope upwards.
"drive"	A horizontal underground tunnel.
"drusy"	A cavity in a rock or mineral vein into which crystals of the minerals forming the rock or vein project.
"dump leach"	Similar to heap leach except ore is not crushed.
"dunite"	Peridotite in which the mafic mineral is almost entirely olivine.
"dyke"	A sheet-like body of igneous rock which is discordant, generally steeply dipping.
"EA"	Environmental assessment.
"eclogite"	Metamorphic rocks of gabbroic composition, consisting primarily of pyroxene and garnet.
"EIA"	Environmental Impact Assessment.
"electrum"	Argentiferous gold containing more than 20% silver.

"EMP"	Environmental management plan.
"en echelon"	Said of geologic features that are in an overlapping or staggered arrangement, e.g. faults.
"enargite"	An orthorhombic mineral, Cu_3AsS_4; in vein and replacement copper deposits as small crystals or granular masses; an important ore of copper and arsenic; may contain up to 7% antimony.
"enrichment"	Process by which the relative amount of one constituent mineral or element contained in a rock is increased.
"Eocene"	Geologic epoch (relatively short period of time) from 55 to 38Ma.
"epidote"	A basic silicate of aluminium, calcium and iron.
"epigenetic"	Applied to mineral deposits of later origin than the enclosing rocks or to the formation of secondary minerals by alteration.
"epithermal"	Said of a hydrothermal mineral deposit formed within about 1km of the of the Earth's surface.
"EPS®"	Enhanced Production Scheduler allows the user to input specific production rates for individual activities and manipulate, in detail, the various mining activities by applying mining resources.
"ESIA"	Environmental and Social Impact Assessment.
"euhedral"	Term applied to grains displaying fully developed crystal form.
"evaporites"	A nonclastic sedimentary rock composed primarily of minerals produced from a saline solution as a result of extensive or total evaporation of the solvent.
"exploration"	Method by which ore deposits are evaluated.
"extrusive"	Said of igneous rock that has been erupted onto the surface of the Earth. Extrusive rocks include lava flows and pyroclastic material such as volcanic ash.
"fahlore"	Form of tetrahedrite; isometric mineral, $(Cu,Fe)_{12}Sb_4S_{13}$, copper replaced by zinc, lead, mercury, cobalt, nickel, or silver; forms a series with tennantite; metallic; occurs in hydrothermal sulphide veins and contact metamorphic deposits.
"Famennian"	Geological stage of the Late Devonian from 367 to 362Ma.
"fault"	Surface of rock fracture along which has been differential movement.
"fauna"	The entire animal population, living or fossil, of a given area, environment, formation or time span.
"Fe"	Chemical symbol for iron.
"feasibilty study"	An extensive technical and financial study to assess the commercial viability of a project.
"feldspar(s)"	The most important group of rock forming silicate minerals, with end-members, alkali feldspar $KalSi_2O_8$, sodium feldspar $NaAlSi_2O_8$ and calcium feldspar $CaAlSi_2O_8$.
"feldspathoids"	A group of rock-forming minerals containing sodium and potassium silicates.
"felsic"	Refers to silicate minerals, magmas, and rocks which are enriched in the lighter elements such as silica, oxygen, aluminum, sodium, and potassium.
"felsite"	A general term for any light coloured, fine grained or aphanitic extrusive or hypabyssal rock, composed chiefly of quartz and feldspar with a felsitic texture.

"ferromagnesian"	Consisting of iron and magnesium.
"fineness"	Referring to gold: purity.
"fire assay"	A dry thermal technique for gold analysis.
"flexure"	General term for a fold, warp, or bend in rock strata.
"flocculant"	A chemical used to aggregate fine particles to improve settling rates.
"flora"	The entire plant population of a given area, environment, formation or time span.
"flotation"	A mineral processing technique used to separate mineral particles in slurry, by causing them to selectively adhere to a froth and float to the surface.
"flow-banded"	The structure in a volcanic rock arising when the directional movement of a lava (for example) containing crystals causes these crystals to take up a parallel orientation.
"flysch"	A marine sedimentary facies of thinly bedded marls, sandy and calcareous shales, muds interbedded within conglomerates, course sandstones, and graywackes.
"fold"	A flexure in rocks.
"footwall"	One of blocks of rock involved in fault movement. One that would be under feet of person standing in tunnel along or across fault; opposite hangingwall.
"free gold/free-milling gold"	A metallurgical term, gold that has a clean surface so that it readily amalgamates with mercury after liberation.
"fuchsite"	Green chromium rich variety of muscovite.
"g/t"	Gramme per metric tonne.
"Ga"	Chemical symbol for gallium.
"gabbro"	Coarse-grained igneous rock with composition of basalt.
"gabbrodiorite"	Fine to medium-grained igneous rock between gabbro and diorite.
"galena/galenite"	Important sulphide ore of lead, PbS.
"gamma spectrometry"	The technique of measuring the spectrum, or number and energy, of gamma rays emitted as natural radioactivity by the formation.
"gangue"	General term for minerals that are not considered to be of economic significance. That part of the mineral deposit from which a metal or metals is not extracted.
"GDP"	Gross Domestic Product; total value of goods produced and services provided in a country in one year.
"Ge"	Chemical symbol for germanium.
"Gemcom®"	Complex mining software used primarily for orebody modelling, resource estimation and pit optimisation.
"geochemical"	Prospecting techniques which measure the content of specified metals in soils and rocks; sampling defines anomalies for further testing.
"geologic block"	The defined boundaries of an ore resource.
"Geologic Resource"	An approximate estimate of a mineral resource, based mainly on geological information, of less reliability than an Inferred Resource.
"geophysical"	Prospecting techniques which measure the physical properties (magnetism,

	conductivity, density, etc.) of rocks and define anomalies for further testing.
"geostatistics"	Complex method of resource estimation using regionalised variables i.e., grade and thickness.
"geothermal"	Pertaining to the heat of the interior of the Earth.
"GijimaAST Mine2-4D®"	Complex mining software used primarily for mine design.
"glauconitic"	Containing sufficient grains of glauconite.
"gneiss"	A term applied to banded rocks formed during high-grade regional metamorphism.
"goslarite"	A mineral formed though the decomposition of sphalerite.
"gossan"	Decomposed rock, usually reddish or ferruginous (owing to oxidised pyrites), forming the upper part of a metallic vein.
"gouge"	A layer of soft, earthly or clayey fault-comminuted rock material along the wall of a vein.
"GPS"	Global Positioning System.
"graben"	Elongated, trench-like, structural form bounded by parallel normal faults created when block that forms trench floor moves downward relative to blocks that form sides.
"grade"	Relative quantity or the percentage of ore mineral or metal content in an ore body.
"granite gneisses"	Metamorphosed (altered) granites.
"granite"	Coarse-grained igneous rock dominated by light-coloured minerals, consisting of about 50% orthoclase, 25% quartz, and balance of plagioclase feldspars and ferromagnesian silicates.
"granitoids"	Pertaining to or composed of granite.
"granodiorite"	Coarse-grained igneous rock intermediate in composition between granite and diorite.
"granulometry/ granulometric"	Relating to the size distribution or measurement of grain sizes in sand, rock or other deposits.
"gravimetric analysis"	Quantitative chemical analysis in which the different substance of a compound are measured by weight.
"greenschists"	Is a general field petrologic term applied to a low grade metamorphic and/or altered mafic volcanic rock.
"greenstone"	A term applied to any compact dark-green altered or metamorphosed basic igneous rock that owes it colour to the presence of chlorite, actinolite or epidote.
"grizzly"	A device for the coarse screening or scalping of bulk materials.
"ground water"	Water occupying openings, cavities, and spaces in rocks and soils.
Group 1 deposits	Large deposits, simple in form with uniform distribution of minerals (examples: coal, some iron and disseminated copper deposits). A normal density of drillholes allows the definition of a high level of A and B reserves.
Group 2 deposits	Large deposits with different and sometimes complicated forms and uneven

	distribution of minerals (examples: some iron and sedimentary copper deposits). Only B category reserves may be defined with a normal grid of drillholes. A combination of drilling and underground workings may be necessary to define the reserves. Category A reserves can be established only by close spaced drilling and underground workings.
Group 3 deposits	Smaller sized deposits with uneven distribution of minerals (examples: some veins, skarns, dykes, and pegmatite deposits). Drillholes can only establish C_1 reserves. A and B reserves can be established only with underground workings.
Group 4 deposits	Smaller sized deposits similar to Group 3 deposits or with even more complex shapes (examples: some veins, skarns, dykes, pegmatite deposits and gold placers). Category A reserves cannot be established with drilling or a normal grid of underground workings. Drilling in combination with underground workings is necessary to establish category B reserves.
Group 5 deposits	Small "pocket" type deposits. Category A and B reserves cannot be established by systematic prospecting. Only category C reserves can be established.
"gusano texture"	Silicified rock having a wormy texture composed of ill-defined masses of alunite and/or pyrophylite.
"gypsum"	A calcium sulphate mineral.
"h"	Hours.
"H_2SO_4"	Sulphuric acid.
"halo"	Circular or crescent distribution pattern about the source or origin of a mineral, ore, mineral association, or petrographic feature.
"hangingwall"	The overlying side of an orebody, fault, or mine working, especially the wall rock above an inclined vein or fault.
"HDPE"	High Density Polyethylene.
"heap leach"	Process used for the recovery of metal ore from weathered low-grade ore. Crushed material is laid on a slightly sloping, impervious pad and uniformly leached by the percolation of the leach liquor trickling through the beds by gravity to ponds. The metals are recovered by conventional methods from the solution.
"hematite"	Important ore mineral of iron, Fe_2O_3, found as an accessory in igneous rocks, in hydrothermal veins and replacements, and in sediments.
"hemimorphite"	A sorosilicate mineral found in the upper parts of zinc and lead ores, chiefly associated with smithsonite.
"Hercynian"	Is a geologic mountain-building event recorded in the European mountains and hills. This occurred in Paleozoic times (from ~390 to ~310Mya) and reflects continental collision.
"Hg"	Chemical symbol for mercury.
"hornfels"	A fine-grained rock composed of a mosaic of equidimensional grains without preferred orientation and typically formed by contact metamorphism.
"hydrocarbon"	Any organic compound, gaseous, liquid, or solid, consisting solely of carbon and hydrogen.

"hydrogeology"	The study of the water cycle.
"hydrogoethite"	A fibrous red mineral formed from hematite and goethite.
"hydrohematite"	A fibrous red mineral formed from hematite and goethite.
"hydrothermal"	Refers in the broad sense to the process associated with alteration and mineralization by a hot mineralised fluid (water).
"hydrozincite"	A secondary mineral in weathered zones of zinc deposits commonly associated with smithsonite or sphalerite. A source of zinc.
"hypogene"	Formed or crystallised at depths below the earth's surface; said of granite, gneiss, and other rocks.
"igneous"	Said of a rock or mineral that solidified from molten or partly molten material, i.e., from a magma.
"ignimbrite"	Rock formed by the widespread deposition and consolidation of ash flows and nuée ardentes.
"ignimbritic"	Consisting of fragmental volcanic material which has been blown into the atmosphere by explosive activity.
"illite"	Clay mineral.
"In"	Chemical symbol for indium.
"inclinometry"	Measurement of the angle of a drillhole below surface.
"inclusion"	Any size fragment of another rock enclosed in an igneous rock. A particle of nonmetallic material retained in a solid metal or alloy.
"*Indicated* Resource"	As defined in the JORC Code, is that part of a Mineral Resource which has been sampled by drill holes, underground openings or other sampling procedures at locations that are too widely spaced to ensure continuity but close enough to give a reasonable indication of continuity and where geoscientific data are known with a reasonable degree of reliability. An *Indicated* Mineral resource will be based on more data and therefore will be more reliable than an Inferred resource estimate.
"Indicated"	An estimate of mineral resources made from geological evidence as defined by the JORC Code for reporting ore reserves and resources. Means a mineral resource that has been sampled by drill holes or other sampling procedures at locations too widely spaced to ensure continuity but close enough to give a reasonable indication of continuity.
"*Inferred* Resource"	As defined in the JORC Code, is that part of a Mineral Resource for which the tonnage and grade and mineral content can be estimated with a low level of confidence. It is inferred from the geological evidence and has assumed but not verified geological and/or grade continuity. It is based on information gathered through the appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.
"Inferred"	An estimate of mineral resources made from geological evidence as defined by the JORC Code for reporting ore reserves and resources. It is inferred from geological evidence and assumed but not verified geological and/or grade

	continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.
"intercalated"	Said of layered material that exists or is introduced between layers of a different character.
"intermediate"	The composition of igneous or volcanic rocks whose composition lies between those of basic and acid rocks.
"intermontane"	Between mountain ranges.
"intrusive"	Of or pertaining to intrusion – both the processes and the rock so formed.
"IOM_3"	The Institute of Materials, Minerals and Mining.
"IP"	Induced Polarisation; geophysical method whereby an induced electrical polarisation is manifested by a decay of voltage in the Earth following the cessation of an excitation current pulse.
"IRR"	Internal Rate of Return is a capital budgeting method used by firms to decide whether they should make long term investments. The IRR is defined as any discount rate that results in a net present value of zero, and is usually interpreted as the expected return generated by the investment.
"jamesonite"	Ore mineral of lead antimony sulphide.
"jarosite"	An iron oxide mineral.
"jasper"	A red variety of chalcedony.
"jaw crusher"	A primary crusher designed to reduce large rocks or ores to sizes capable of being handled by any of the secondary crushers.
"jog"	An offset in a flat plane consisting of two parallel bends in opposite directions by the same angle.
"JORC Code/JORC"	Joint Ore Reserve Committee Code. The Committee is convened under the auspices of the Australasian Institute of Mining and Metallurgy.
"Jurassic"	Geologic period of time from 190 to 135Ma.
"JV"	Joint Venture.
"kaolinite/kaolin"	A monoclinic mineral, $2[Al_2Si_2O_5(OH)_4]$; kaolinite-serpentine group; soft; white; formed by hydrothermal alteration or weathering of aluminosilicates, esp. feldspars and feldspathoids; formerly called kaolin.
"karstic"	An area of irregular limestone in which erosion has produced fissures, sinkholes, underground streams and caverns.
"kersantite"	Rock type of lamprophyre containing biotite and plagioclase, with or without clinopyroxene and olivine.
"km(s)"	Kilometres.
"km^2"	Square kilometres.
"km^3"	Kilometres cubed.
"Kriging"	Geostatistical estimation of ore reserves though the application of a weighted, moving average accounting for estimated values of spatially distributed variables and to assess the probable error associated with the estimates.
"kt"	Kilo tonnes (1,000 tonnes).

"kV"	Kilo-volt.
"kW"	Kilo-watt.
"lacustrine"	Pertaining to, formed in, growing in, or inhabiting lakes.
"ladderway"	Mine shaft, raise or winze between two main levels, equipped with ladders.
"lamprophyre"	Group of dark-coloured, porphyritic, hypabyssal igneous rocks with a high percentage of mafic minerals.
"lapilli"	Pyroclastic debris of fragments or clasts measuring between 4 - 32mm.
"leached"	A rock that is in the process of being broken down by the action of substances dissolved in water.
"leaching"	*See cyanidation*
"leucoxene"	Fine grained, opaque white alteration products of ilmenite, mainly finely crystalline rutile. A variety of sphene.
"limestone"	Sedimentary rock composed largely of mineral calcite, $CaCO_3$, formed by either organic or inorganic processes.
"Limonite"	Limonite is an ore consisting in a mixture of hydrated iron (III) oxide-hydroxide of varying composition. The generic formula is frequently written as $FeO(OH)nH_2O$, although this is not entirely accurate as limonite often contains a varying amount of oxide compared to hydroxide.
"lineament"	A large scale linear structural feature.
"lenticular"	Resembling in shape the cross section of a lens.
"listric"	A curved downward-flattening fault, generally concave upward.
"lithocap"	Rock unit conspicuously covering another, often formed by alteration.
"Lithology"	A term usually applied to sediments, referring to their general characteristics. A macroscopic hand-sample or outcrop-scale description of rocks.
"lithological"	A term usually applied to sediments, referring to their general characteristics.
"lithostratigraphic"	The variation of rock types in a sedimentary sequence.
"littoral"	Or pertaining to a shore. A coastal region.
"loam"	Soil containing sand, silt and clay in roughly equal parts.
"lode"	A mineral deposition consisting of a zone of veins, veinlets, disseminations, or planar breccias.
"low-sulphidation"	Quartz veins, stockworks and breccias carrying gold, silver, electrum, argentite and pyrite with lesser and variable amounts of sphalerite, chalcopyrite, galena, rare tetrahedrite form in high-level to near-surface environments. The ore commonly exhibits open- space filling textures and is associated with volcanic-related hydrothermal to geothermal systems.
"m"	Metre.
"m^3"	Metres cubed.
"Ma"	Million years.
"mafic"	A dark-coloured igneous rock which has a high proportion of pyroxene and olivine minerals.
"mafics"	Generally dark coloured, iron and magnesium rock forming minerals.
"magnesium"	A light, silvery-white, and fairly tough metal. Chemical symbol, Mg. Found in

	large deposits in the form of magnesite, dolomite, and other minerals.
"magnetics"	A geophysical technique used to measure the magnetic susceptibility of rocks.
"magnetite"	An iron ore mineral, Fe_3O_4.
"malachite"	A source of copper. Bright green monoclinic mineral, occurs with azurite in oxidized zones of copper deposits.
"manganese"	A grey white, hard, brittle metallic element. Chemical symbol Mn.
"MapInfo GIS®"	Professional 'Geographic Information System' mapping software.
"marble"	Metamorphic rock of granular texture, with no rock cleavage, and composed of calcite, dolomite, or both.
"massif"	A very large topographic or structural feature, usually of greater rigidity than the surrounding rock.
"MCRP"	Mine Closure and Rehabilitation Plan.
"Measured Resource"	Defined in the JORC Code, as that part of a Mineral Resource for which the resource has been intersected and tested by drill holes, underground openings or other sampling procedures at locations which are spaced closely enough to confirm continuity and where geoscientific data are reliably known. A measured resource estimate will be based on a substantial amount of reliable data, interpretation and evaluation which allows a clear determination to be made of the shapes, sizes, densities and grades.
"Measured"	An estimate of mineral resources from geological data as defined by the JORC Code for reporting ore reserves and resources. This is part of a mineral resource where exploration data are distributed in sufficient density and are of sufficient reliability to allow the estimation of the resource, volume, shape, tonnage and grade to a level of confidence in their accuracy to allow a detailed mining Feasibility Study to be carried out.
"melanterite"	A mineral formed from the decomposition of pyrite.
"Mesozoic"	An era of geologic time, from the end of the Paleozoic to the beginning of the Cenozoic, or from about 225 million years to about 65 million years ago.
"metalliferous"	Containing metal.
"metallogenic province"	A belt of rocks, often structurally controlled, that are host to a specific selection of minerals.
"metallogenic"	Study of the genesis of mineral deposits, with emphasis on its relationship in space and time to regional petrographic and tectonic features of the Earth's crust.
"metallurgical"	Describing the science concerned with the production, purification and properties of metals and their applications.
"metamorphism"	Process whereby rocks undergo physical or chemical changes or both to achieve equilibrium with conditions other than those under which they were originally formed (excluding process of weathering). Agents of metamorphism are heat, pressure, and chemically active fluids.
"metamorphosed"	Rocks which have been altered by temperature and pressure.
"metasomatic"	Process whereby rocks are altered when volatiles exchange ions with them.

"metasomatism/ metasomatic"	Metamorphic change which involves the introduction of material from an external source.
"Metavolcanics"	Metamorphosed volcanic rocks.
"Mg"	Chemical symbol for magnesium.
"micro"	Prefix that divides a basic unit by 1 million or multiplies it by 10^{-6}. A prefix meaning small. When modifying a rock name, it signifies fine-grained, as in microgranite.
"Micormine®"	Complex mining software used primarily for ore body modelling, resource estimation and pit optimisation.
"migmatite"	Rock formed under extreme temperature-pressure conditions during metamorphism, where partial melting occurs in pre-existing rocks.
"mill"	Equipment used to grind crushed rocks to the desired size for mineral extraction.
"mineral resource"	A concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such a form that there are reasonable prospects for the eventual economic extraction. The location, quantity, grade geological characteristics and continuity of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral resources are sub-divided into *Inferred*, *Indicated* and *Measured* categories.
"mineralisation"	Process of formation and concentration of elements and their chemical compounds within a mass or body of rock.
"Miocene"	Geological epoch of time from 23 to 5Ma.
"mixite"	A hexagonal mineral.
"mm"	Millimetre, one thousandth of a metre.
"mm^3"	Millimetres squared.
"Mo"	Chemical symbol for molybdenum.
"moisture content"	The percentage moisture content equals the weight of moisture divided by the weight of dry soil multiplied by 100.
"molybdenite"	Mineral compound of molybdenum and sulphur, MoS_2.
"molybdenum"	Silvery-white, very hard, metallic element. Chemical symbol, Mo. Does not occur native, but is obtained principally from molybdenite.
"monoclinic"	Of or denoting a crystal system or three-dimensional geometrical arrangement having three unequal axes of which one is at right angles to the other two.
"monzonite"	Intrusive igneous rock that contains abundant and approximately equal amounts of plagioclase and potash feldspar.
"Moz"	Million troy ounces.
"MPa"	Unit to measure rock strength.
"Mt"	Million tonnes.
"mW"	Mega-watt.
"NaCn"	*See sodium cyanide*
"nearest neighbour"	A method of assigning a sample value to a point in space.
"Neoproterozoic"	A geological era from 1000 to 542.0 Ma.
"NO_2"	Nitrogen Dioxide.

"NPV"	Net Present Value is a standard method in finance of capital budgeting – the planning of long-term investments. Using the NPV method a potential investment project should be undertaken if the present value of all cash inflows minus the present value of all cash outflows (which equals the net present value) is greater than zero.
"nuée ardente"	An incandescent cloud of gas and volcanic ash violently emitted during the eruption of certain types of volcano.
"nugget effect"	Anomalously high precious metal assays resulting from the analysis of samples that may not adequately represent the composition of the bulk material tested due to non-uniform distribution of high-grade nuggets in the material to be sampled.
"NUP"	Natural Use Permit.
"oceanic"	Of or relating to the ocean.
"ochre"	Natural pigments, usually iron hydroxides and hydrated iron oxides.
"Oligocene"	Geological epoch of time from 35 to 23Ma.
"open-pit"	A large scale hard rock surface mine.
"open stope"	An unfilled cavity.
"ophiolite"	Group of mafic and ultramafic igneous rocks, whose origin is associated with an early phase of the development of a geosyncline.
"Ordovician"	A period of geologic time from about 500 to 435Ma.
"ore body"	Mining term to define a solid mass of mineralised rock which can be mined profitably under current or immediately foreseeable economic conditions.
"ore reserve"	The economically mineable part of a *Measured* or *Indicated* mineral resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments, which may include feasibility studies, have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could be reasonably justified. Ore reserves are sub-divided in order of increasing confidence into *Probable* and *Proven*.
"ore"	A mineral deposit that can be extracted and marketed profitably.
"orebody"	Mining term to define a solid mass of mineralised rock which can be mined profitability under current or immediately foreseeable economic conditions.
"ore-field"	A zone of concentration of mineral occurrences.
"orepass"	A vertical or inclined passage for the downward transport of ore.
"oreshoot"	An elongate pipelike, ribbon-like, or chimney-like mass of ore within a deposit (usually a vein), representing the more valuable part of the deposit.
"orogenic"	Mountain building.
"orthoclase"	A series of potassium feldspars.
"ounce/oz"	Troy ounce (= 31.1035 grammes).
"outcrop"	The area over which a particular rock type occurs at the surface – whether visibly

	exposed or not.
"OVOS"	Soviet Environmental Impact Assessment (EIA).
"pa"	Per Annum.
"palaeo"	A prefix common in geological terminology, meaning ancient, of past times, and sometimes suggesting an early or primitive nature.
"paleogene"	A geological time period from 65.5 to 23.03Ma. Part of the Cenozoic Era.
"paleosurface"	Original land surface.
"Paleozoic Era"	The first of the three eras of the Phanerozoic, spanning 570-248 million years.
"Paleozoic"	Geological era from 570 to 245Ma.
"paragenesis"	The relationship of minerals expressed in terms of a time sequence.
"paste backfill"	Cement and waste rock mixture used to backfill underground stopes.
"Pb"	The chemical symbol for lead.
"pelitic"	Pertaining to or characteristic of pelite.
"peridotite"	General term for a coarse-grained plutonic rock composed chiefly of olivine with or without other mafic minerals such as pyroxenes, amphiboles, or micas, and containing little or no feldspar.
"Permian"	Is a geologic period that extends from about 299.0Ma to 248.0 Ma.
"pH"	The measure of the acidity or alkalinity of a solution.
"Phanerozoic"	Rocks younger than 590 million years.
"phengite"	A variety of muscovite having high silica.
"phenocrysts"	Relatively large crystals which are found set in a finer-grained groundmass.
"phlogopite"	A manganese rich end member of the biotite crystal solution series. Found as an alteration mineral in sulfur rich hydrothermal assemblages.
"phosphorite"	Any rock containing calcium phosphate of sufficient purity and quantity to permit its commercial use as a source of phosphatic compounds or elemental phosphorus.
"phyllite"	A fine grained low-grade metamorphic rock.
"pillar"	A column of ore left to support the overlying strata or hanging wall in a mine.
"placer"	A mineral deposit formed by the winnowing action of either water, or air to concentrate minerals of different mass by gravity separation.
"plagioclase"	A series of sodium/calcium feldspars, plagioclase feldspars are common rock-forming minerals.
"plications"	Intense small scale folds.
"plumbojarosite"	A mineral of the alunite group. $PbFe_6(SO_4)_4(OH)_{12}$.
"plunge"	A fold is said to plunge if the axis is not horizontal.
"pluton"	An igneous intrusion.
"plutonic"	Pertaining to igneous rocks formed at great depths.
"polymetallic"	Refers to a mineral deposit or occurrence with several metal sulphides, common metals include Cu, Pb, Zn, Fe, Mo, Au and Ag.
"polymict"	Term describing detrital rock consisting of fragments of many different materials.
"porosity"	The space available to fluid penetration.
"porphyrites"	Porphyries that contain plagioclase phenocrysts.

"porphyritic"	A medium coarse-grained intrusive or volcanic rock which is conspicuous by containing more than 25% large well-formed crystals by volume.
"porphyry"	Igneous rock containing conspicuous phenocrysts (crystals) in fine-grained or glassy groundmass.
"potassium"	Highly reactive metallic element of the alkali group; it is soft, light, and silvery. Chemical symbol, K. Occurs abundantly in nature.
"ppb"	Parts per billion.
"ppm"	Parts per million.
"Precambrian"	Era before 590 million years.
"precious metal"	Gold, silver and platinum group minerals.
"propylitic"	Plagioclase in an igneous rock is altered to epidote, sericite and secondary albite, and ferro-magnesian minerals are altered to chlorite-calcite-epidote-iron oxide assemblages.
"Proterozoic"	Most recent geological Eon of three sub-divisions of the Precambrian, from 2,500 to 570Ma.
"pyrargyrite"	A trigonal mineral; chemical symbol Ag_3 SbS_3; soft; deep red; in late-primary or secondary-enrichment veins, and an important source of silver.
"pyrite"	A mineral compound of iron and sulphur, sulphide mineral, iron sulphide, chemical symbol FeS_2.
"pyrobitumen"	Naturally occurring hydrocarbon that is formed or affected by heat, or has a fiery colour.
"pyroclastic"	Rock consisting of fragments of volcanic material which has been blown into the atmosphere by explosive activity.
"pyroxene"	A group of chiefly magnesium-iron minerals.
"pyrrhotite/pyrrhotine"	A mineral compound of iron and sulphur found in basis igneous rocks.
"QA/QC"	Quality Assurance/Quality Control.
"quartz"	A mineral composed of silicon dioxide.
"quartzites"	Is a hard, metamorphic rock which was originally sandstone.
"Quaternary"	Geological period of time from 2Ma; youngest period of the Cenozoic.
"radiometry"	Measurement of gamma radiation within rocks.
"radon"	A heavy, radioactive, gaseous, inert element.
"raise"	A vertical or near-vertical opening driven upward form a level to connect with the level above.
"RC drlling"	Drilling method where samples are cut by percussion roller or blade drill bits and flushed to the surface using compressed air or water.
"recovery"	Proportion of valuable material obtained in the processing of an ore, stated as a percentage of the material recovered compared with the total material present.
"relief"	Terrain, or relief, is the third or vertical dimension of land surface.
"reserves"	**Proven**: measured mineral resources, where technical economic studies show that extraction is justifiable at the time of the determination and under specific economic conditions. **Probable:** measured and / or indicated mineral resources which are not yet

	proven, but where technical economic studies show that extraction is justifiable at the time of the determination and under specific economic conditions.
"resistivity"	A geophysical technique to measure the electrical resistance of rocks.
"resources"	**Measured**: a mineral resource intersected and tested by drill holes, underground openings or other sampling procedures at locations which are spaced closely enough to confirm continuity and where geoscientific data are reliably known. A measured mineral resource estimate will be based on a substantial amount of reliable data, interpretation and evaluation which allows a clear determination to be made of shapes, sizes, densities and grades. **Indicated**: a mineral resource sampled by drill holes, underground openings or other sampling procedures at locations too widely spaced to ensure continuity but close enough to give a reasonable indication of continuity and where geoscientific data are known with a reasonable degree of reliability. An indicated resource will be based on more data, and therefore will be more reliable than an inferred resource estimate. **Inferred**: a mineral resource inferred from geoscientific evidence, underground openings or other sampling procedures where the lack of data is such that continuity cannot be predicted with confidence and where geoscientific data may not be known with a reasonable level of reliability.
"rhenium"	A rare mineral which occurs in very small quantities in platinum ores and in columbite, gadolinite and molybdenite.
"rhodanine"	An organic compound derived from thiazolidine.
"rhodochrosite"	A manganese carbonate mineral.
"rhyolite"	A group of extrusive igneous rocks, typically porphyritic and commonly exhibiting flow texture, with phenocrysts of quartz and alkali feldspar in a glassy to cryptocrystalline groundmass.
"riffle splitter"	A device used to reduce the volume or weight of a sample.
"rift"	A regional-scale strike-slip fault, with offset measured up to hundreds of kilometres or a trough or valley formed by faulting.
"Riphean"	An era of the Middle Proterozoic, lasting from about 1675-825Ma.
"roasting"	The treatment of ore by heat and air, or oxygen-rich air, in order to remove sulphur and arsenic.
"rock chip"	A chip sample taken from one or more points within a restricted area.
"rod mill"	A rotating drum containing steel rods used to grind ore.
"run-of-mine"	Average grade of mineralisation to be extracted from a mine.
"rutile"	A titanium dioxide mineral tetragonal in nature.
"s"	Seconds.
"S"	Chemical symbol for sulphur.
"Sb"	Chemical symbol for antimony.
"sandstone"	Detrital sedimentary rock in which particles range from 1/16 to 2mm.
"saturated"	A rock or soil where all its interstices are filled with water, holding as much water or moisture as can be absorbed.

"scandium"	A metallic element which is extracted as a by-product from uranium mill tailings. It is also the principal component in the rare mineral thortveitite.
"scheelite"	A tungsten ore mineral, chemical symbol $CaWO_4$.
"schist or schistocity"	Metamorphic rock dominated by fibrous or platy minerals with parallel alignment that splits readily. Has schistose cleavage and is product of regional metamorphism.
"schistose"	Said of a rock displaying schistocity.
"schists"	Metamorphic rock dominated by fibrous or platy minerals.
"scoriaceous"	A term used to describe a pyroclastic rock containing cavities.
"Se"	Chemical symbol for selenium.
"sedimentary"	Rocks formed from material derived from pre-existing rocks by processes of denudation.
"selenium"	A non-metallic element and member of the sulphur family.
"self-potential"	A geophysical method used in mineral exploration to locate and delineate conductive mineral deposits.
"sericite"	A white mica.
"sericitization"	The formation of sericite, usually at the expense of other minerals in the rock.
"serpentinite"	A rock consisting almost wholly of serpentine-group minerals.
"shaft"	Vertical or inclined excavation into mine workings.
"shotcrete"	Is mortar or concrete conveyed through a hose and pneumatically projected at high velocity onto a surface for stabilisation purposes.
"shrinkage stope"	A stope in which part of the severed ore is removed during stoping, the balance being temporarily available as support of workings.
"siderite"	An iron mineral.
"Sigma-3D"	A plastic simulation software product.
"silica cap"	The chemically resistant dioxide of silicon, SiO_2. Resistant to erosion, capping often less resistant rocks below.
"silicate"	Is a compound consisting of silicon and oxygen (Si_xO_y), one or more metals, and possibly hydrogen.
"siliceous"	Of relating to or derived from silica.
"silicification"	The introduction of silica into a rock, either filling pore spaces or replacing pre-existing minerals.
"sill"	A tabular mass of igneous rock that has been intruded laterally between layers of older rock.
"siltstone"	Detrital sedimentary rock in which particles are less than 1/16mm.
"Silurian"	A period of geologic time from about 435 to 395 million years.
"sinistral"	A term applied to a fault to describe the apparent direction of relative movement of each side, in this case to the left.
"sinter"	A process for agglomerated ore concentrate in which partial reduction of minerals may take place and some impurities be expelled prior to subsequent smelting and refining.
"skarn"	An old Swedish mining term for silicate gangue (amphibole, pyroxene, garnet,

	etc.) of certain iron ore and sulfide deposits of Archean age, particularly those that have replaced limestone and dolomite. Its meaning has been generally expanded to include lime-bearing silicates, of any geologic age, derived from nearly pure limestone and dolomite with the introduction of large amounts of Si, Al, Fe, and Mg.
"slag"	Material formed during smelting operations through the combination of a flux and gangue/waste portion of the ore.
"slimes"	Extremely fine sediment produced in the processing of ore or rock.
"sludge"	Rock cuttings produced by a drill bit.
"smectite"	A swelling clay: originally applied to fuller's earth and later to montmorillonite; also to certain clay deposits that are apparently bentonite, and to a greenish variety of halloysite.
"smelting"	A metallurgical operation (at a smelter) in which metal is separated from impurities by a process that includes melting.
"smithsonite"	A variety of zinc carbonate and source of zinc.
"SO_2"	Chemical formula for sulphur dioxide.
"sodium cyanide"	An inorganic highly toxic chemical compound.
"specific density"	The ratio of the density of a substance to the density of a given reference material.
"sphalerite"	A zinc sulphide mineral.
"sphene"	An accessory mineral found in acid igneous rocks and in metamorphosed limestones.
"steppe"	An extensive, treeless grassland area in the semiarid mid-latitudes of southeastern Europe and Asia.
"stibnite"	An orthorhombic mineral, chemical symbol Sb_2S_3; may contain gold and silver; in low-temperature veins and around hot springs; the chief source of antimony.
"stock"	An intrusive mass of igneous rock.
"stockpile"	An accumulation of ore or mineral built up when demand slackens or when the treatment plant or beneficiation equipment is incomplete or temporarily unequal to handling the mine output.
"stockwork"	A mineral deposit consisting of a three-dimensional network of planar to irregular veinlets.
"stoping"	A mining method of extract ore by digging tunnels along the trend of the ore and extracting ore from above and below the tunnel.
"stratiform"	Having the form of a layer, bed, or stratum; consisting of roughly parallel bands or sheets.
"stratigraphy"	Study of the stratified rocks, sedimentary and volcanics, especially their sequence in time, the character of the rocks and the correlation of beds in different localities.
"strike length"	The longest horizontal dimension of an ore body or zone of mineralisation.
"strike"	Direction in which a horizontal line can be drawn on a plane, and determines the direction in which to measure the true dip.

"stringers"	Mineral veinlet or filament, usually one of a number, occurring in a discontinuous sub parallel pattern in host rock.
"stripping ratios/ strip ration"	A ratio of the waste relative to ore in a mining operation.
"sub-volcanic"	Pertaining to an igneous intrusion, or to the rock of that intrusion, whose depth is intermediate between that of abyssal or plutonic and the surface.
"sulphide"	Mineral containing sulphur in its non-oxidised form.
"sulfur/sulphur"	Soft yellow mineral.
"supergene alteration"	Near surface alteration.
"supergene"	Said of a mineral deposit or enrichment formed near the surface, commonly by descending solutions, used almost exclusively for processes involving water.
"syncline"	A basin shaped fold.
"syngenetic"	Said of a mineral deposit formed contemporaneously with, and by essentially the same processes as, the enclosing rocks
"synorogenic"	Said of a geological process or event occurring during a period of orogenic activity.
"t"	A metric tonne.
"tabular"	Said of a feature having two dimensions that are much larger or longer than the third.
"tailings"	Material that remains after all metals/minerals considered economic have been removed from the ore.
"Te"	Chemical symbol for tellurium.
"tectogenesis"	Structural history of an area.
"tectonic"	Said of or pertaining to the forces involved in, or the resulting structures or features of, tectonics: branch of geology dealing with the broad architecture of the outer part of the Earth; i.e., the regional assembling of structural or deformational features.
"tectono-magmatic"	Structural and intrusive history of an area.
"telluride"	Mineral containing tellurium.
"tellurium"	Soft semimetallic mineral associated with pyrite, sulfur, or in the fine dust of gold-telluride mines.
"tellurobismuthite"	A trigonal mineral from the tetradymite group.
"tenorite"	A copper oxide found in oxidised zones of copper deposits, a source of copper.
"tennantite"	Isometric mineral, $(Cu,Fe)_{12}As_4S_{13}$; forms a series with tetrahedrite; may contain zinc, silver, or cobalt replacing copper; in veins; an important source of copper.
"terrigenous"	Derived from the land or continent.
"Tertiary"	Geologic period of time from 65 to 2Ma; first period of Cenozoic.
"tetrahedrite"	A copper, iron, antimony sulphide mineral.
"thorium"	Element which is slightly radioactive.
"thrust"	Overriding movement of one crustal unit over another, such as in thrust faulting.
"tin"	The metal extracted from cassiterite.

"Tl"	Chemical symbol for thallium.
"TMF"	Tailings management facility.
"tonalite"	Alternative name for diorite.
"tonalities"	An igneous, plutonic (intrusive) rock, of felsic.
"top-cut"	Process applied to grade evaluation to eradicate "nugget-effect".
"topography"	The general configuration of a land surface or any part of the earth surface including its relief and the position of its natural and manmade features.
"tourmaline"	Is a complex silicate of aluminium and boron, its composition varies widely with sodium, calcium, iron, magnesium, lithium and other elements entering into the structure.
"Tournaisian"	Geological epoch from 362 to 349Ma.
"tpa"	Tonnes per annum.
"trachyte"	A fine grained igneous rock.
"trachytes"	Fine-grained, intermediate igneous rocks.
"transgressive"	Discrepancy in the boundary lines of continuous strata; i.e., unconformity.
"treatment plant"	A plant where ore undergoes physical or chemical treatment to extract the valuable metals/minerals.
"tremolite"	Monoclinic mineral, $2[Ca_2Mg_5Si_8O_{22}(OH)_2]$; amphibole group with magnesium replaced by iron, and silicon by aluminium toward actinolite; in low-grade metamorphic rocks such as dolomitic limestones and talc schists.
"trench sampling"	Sampling of a trench cut through the rock, generally in the form of a series of continuous channels (channel samples).
"Triassic/Trias"	Geological period of time from 250 to 200 Ma.
"trigonal"	Of or denoting a crystal system having three equal axes separated by equal angles that are not at right angles.
"truncation"	To shorten by or as if by cutting off.
"tuffite"	A tuff containing both pyroclastic and detrital material, but predominantly pyroclasts.
"tuffs"	Rock consolidated from volcanic ash.
"tungsten"	Hard, brittle, white or grey metallic element. Chemical symbol, W. Also known as wolfram.
"turbidite"	Means a sedimentary rock usually formed in deep water from a turbidite current (a slurry of sediment). It is typically graded in grain size from coarse at the bottom to very fine at the top.
"ultramafic"	An igneous rock composed chiefly of mafic minerals.
"unconformity"	Surface separating two rock masses, older exposed to erosion before deposition of younger. If older rocks were deformed and not horizontal at time of subsequent deposition, surface of separation is angular unconformity. If older rocks remained horizontal during erosion, surface separating them from younger rocks is called disconformity. Unconformity that develops between massive igneous or metamorphic rocks exposed to erosion and then covered by sedimentary rocks is called nonconformity.

"underground working"	Mine openings for evaluation for ore extraction excavated beneath the ground surface.
"UNDP"	United Nations Development Project.
"uranium"	A radioactive metallic element. Chemical symbol U.
"variogram"	A plot of the variance of paired sample measurements as a function of the distance between samples.
"variography"	A geostatistical method of determining the spatial variations in the grade and nature of mineralisation within a particular ore body.
"vein swarm"	Multiple and composite veining.
"vein"	A tabular deposit of minerals occupying a fracture, in which particles may grow away from the walls towards the middle.
"vesicular"	Small spherical or ellipsoidal cavities found in volcanic lavas, which are produced by bubbles of gas trapped during the solidification of the rock.
"VMS"	Volcanogenic Massive Sulphide ; accumulation of sulphide minerals which may include iron, copper, lead, zinc sulphides in varying proportions, together with gold and silver minerals, formed on the sea floor by mineralised fluids rising from the earth's crust and cooling by contact with sea water.
"volcano-clastic"	Volcanic rocks which have been at least slightly worked by the sedimentary process.
"vug"	A hole or cavity in a rock.
"W"	Chemical symbol for tungsten.
"weathering"	The breakdown of rocks and minerals in the near-surface environment by the action of physical and chemical processes, in the presence of air and water.
"willemite"	A trigonical mineral found in zinc deposits. A source of zinc.
"wireframed"	A technique to convert ore body intersections in a 3D computer model to assist interpretation.
"wollastonite"	A mineral of the pyroxene group, $CaSiO_2$, formed by the thermal metamorphism of impure limestone.
"xanthate"	Common specific promoter used in flotation of sulphide ores.
"xenoliths"	Rock fragment foreign to igneous rock in which it occurs.
"Zn"	The chemical symbol for zinc; bluish-white, lustrous metal.
"µm"	Micron (one millionth of a metre).
"zircon"	The chief source of zirconium ($ZrSiO_4$) widely occurs in granite.
ZVOS	Soviet EIA

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