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LLC Miramine
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TECHNICAL AND ECONOMIC CALCULATION OF THE EFFICIENCY OF THE KRASNY GOLD-ORE OCCURRENCE DEVELOPMENT

Original report is in Russian.
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2. Goals and Objectives of the Work

The work was conducted under Contract No MP-13 dated February 20, 2012, between LLC. Miramine (“Executor”) and LLC. Krasny (“Customer”).

The purpose of the work was also to develop technical and economic calculation materials in accordance with the Contract and the Terms of Reference.

2.1. Term of Reference for Development of Technical and Economic Calculations

Общество с ограниченной ответственностью «Копыловский»

Приложение № 1
к Договору **№ МР-13**
от 20 февраля 2013 г.

"Красный"

УТВЕРЖДАЮ:
Генеральный директор
ООО «Красный»
А.Я.Вамбольдт
20 февраля 2013.

ТЕХНИЧЕСКОЕ ЗАДАНИЕ
на разработку Технико-экономического расчета отработки золоторудного месторождения «Красное», в пределах ИРК 02804 БР.

1. ОБЩИЕ ДАННЫЕ	
1.1. Наименование разведанного геологического участка.	Месторождение «Красное» (лицензия ИРК 02804 БР)
1.2. Место размещения.	Иркутская область, Бодайбинский район, 84 км на север от города Бодайбо
1.3. Границы участка недр.	Границы участка недр принять в соответствии с границами, установленными лицензионным соглашением ИРК 02804 БР
1.4. Стадия работ и основная задача проектирования.	Разработка материалов ТЭР
1.5. Заказчик.	ООО «Красный».
1.6. Исходные данные для выполнения работ	Материалы по геологоразведочным работам: графические материалы (геологические планы и разрезы, колонки скважин). Материалы по вещественному составу, физико-механическим свойствам руд и пород, гидрогеологической характеристике месторождения с графическими приложениями. Топографо-геодезические материалы района работ (с привязкой к геологическим планам), исходные данные по инфраструктуре региона, климатическим условиям. Материалы по экологическим исследованиям. Основные исходные горнотехнические показатели проектируемого карьера, данные по устойчивости бортов карьера, предельные углы наклона откосов уступов. Исходные данные для выполнения сметной документации по перечню Исполнителя. Исходные данные для выполнения экономической части по

	перечню Исполнителя. Электронная база данных в формате xls: таблица координат скважин; таблица инклинометрии; таблица опробования.
2. ТРЕБУЕМЫЕ ПОКАЗАТЕЛИ И РЕКОМЕНДУЕМЫЕ ОСНОВНЫЕ РЕШЕНИЯ	
2.1. Номенклатура и объем производимой продукции	Оптимальную годовую производительность по добыче определить в ТЭР. Выбор вариантов производить по критерию технической и экономической эффективности.
2.2. Основные проектные решения	1. Разработать оптимальный горный календарь отработки с разбивкой поэтапного освоения месторождения (1 этап = 1 год). 2. Срок службы карьера определить проектом. 3. Транспортировку руды до обогатительной фабрики принять автомобильным транспортом. 4. Режим работы предприятия принять круглогодичный в две смены по 12 часов. 5. Расчеты капитальных вложений и эксплуатационных расходов выполнить в текущих ценах с использованием аналогов для данной климатической зоны и удаленности. 6. Технико-экономические расчеты выполнить на конечный продукт – сплав Доре. 7. Цены на товарную продукцию принять из расчета средней цены за прошедший календарный год. 8. Предусмотреть в разработке ТЭР мероприятия по снижению отрицательного влияния производственных процессов на окружающую среду и рекультивацию используемых земель после завершения отработки месторождения и переработки руд;
2.3. Инженерное обеспечение	По аналогам и укрупненными расчетами.
2.4. Административные и бытовые помещения	По аналогам и укрупненными расчетами.
3. ОБЪЕМ ПОРУЧАЕМЫХ РАБОТ.	
3.1.Объем поручаемых работ	1. Разработка ТЭР месторождения «Красное». Заказчик разрабатывает следующие материалы к ТЭР - оптимальный горный календарь отработки с разбивкой поэтапного освоения месторождения; - исходные технологические, технико-экономические данные на обогащение; - экономические показатели проекта. - оптимальную структуру рудника (перечень основных объектов строительства), предварительный расчет необходимой горнотранспортной техники и оборудования
4. ОРГАНИЗАЦИЯ РАБОТ	
4.1. Требования к оформлению	Работу выполнить в соответствии с нормативными

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материалов ТЭР	требованиями, а именно: - методическими рекомендациями по технико-экономическому обоснованию кондиций для подсчета запасов месторождений твердых полезных ископаемых - методическими рекомендациями по применению Классификации месторождений и прогнозных ресурсов твердых полезных ископаемых; - рекомендациями к комплексному изучению месторождений и подсчету запасов попутных полезных ископаемых и компонентов; - нормативами технологического проектирования (ВНТП 35-86) - рекомендациями единых правил безопасности: (ЕПБ при взрывных работах, ЕПБ при разработке месторождений полезных ископаемых открытым способом)
4.2. Сроки выполнения работ.	В соответствии с договором и приложениями к нему.
4.3. Другие существенные требования, исходя из конкретных условий.	- В ТЭР освоения месторождения на долгосрочную перспективу, дополнительно к обоснованию основного варианта обогащения руд и освоения, изложить общие соображения о возможности извлечения золота по другим технологиям (например, прямое цианирование) и возможности использования инфраструктуры предприятия для эксплуатации других близ расположенных объектов

5. РЕЗУЛЬТАТЫ РАБОТЫ.

Результатом работ (Продукцией) является ТЭР месторождения «Красное», разработанное в соответствии требованиями настоящего Технического задания.

От Заказчика Техническое задание подписал:

Генеральный директор
ООО «Красный»

_______ Вамбольдт А.Я.
«20» февраля 2013 г.

МП

От Исполнителя Техническое задание подписал:

Генеральный директор
ООО «Мирамайн»

_______ Никандров А.Н.
«20» февраля 2013г.

МП

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3. Qualification of Consultants

LLC. Miramine is an independent Russian company which includes more than 20 processional consultants providing expert services in a wide spectrum of technical disciplines associated with the mining and geological industry.

LLC. Miramine has experience in independent auditing and Mineral Resource and Reserve estimation.

LLC. Miramine also has significant experience in review of resource estimation reports prepared in Russia and CIS countries in compliance with the conditions and instructions used in these countries.

3.1. Working Program and Team Members

LLC. Miramine used the following specialists:

- Alexey Nikolayevich Nikandrov – geologist and Competent Person, project manager;
- Alexey Alexeyevich Roschin – leading mining engineer;
- Alexey Yevgenyevich Vikentyev – leading geologist;
- Yelena Sergeyevna Shmakova – geologist;
- Alexandra Vyacheslavovna Korsakova – geologist;
- Maria Dmitriyevna Ryabaya – Mining Engineer;
- Alexey Vyacheslavovich Rodin – economist, chief specialist;
- Vitaliy Nikolayevich Yevdokimov – technologist;
- Maria Igorevna Kholodova – ecologist.

Yevgeny Nikolayevich Bozhko was the contact person of the Company. He provided the input data for work.

He also presented all the information required afterwards.

The results of work conducted by the team were summarized in the report by the project manager and were examined by internal reviewers. Mr. Alexey Nikolayevich Nikandrov acted as a Competent Person.

4. Data Reliability

LLC. Miramine thinks that its opinion shall be assumed in whole, and that individual parts of the report, with no regard to all factors and analyses in whole, can give a misguiding pattern of processes serving as the basis for the opinions given in the report. Preparation of a review report is a complex process and is not related to any specific analysis or generalization.

Furthermore, LLC. Miramine is not obliged to and did not assume any responsibility to notify any person on any changes in the circumstances which become evident after preparation of this report, or edit, review and update the report or the opinion.

All reviews, comments and conclusions given in this report reflect the opinion of LLC. Miramine as of February 2013 and are based on the review of information and materials provided by the Company and discussions with Russian specialists.

These discussions were conducted during visits of the Moscow office. LLC. Miramine does not bear any responsibility, either direct or indirect, in relation to information reliability.

5. Declaration

Neither LLC. Miramine as a company, nor its director, employees or associates, own the securities of the Customer, its subsidiary companies or offices, and

- do not have any rights to receive Customer's securities either now or in the future;
- do not have any right fixed by the law for the property or participation in the concession belonging to the Customer; and
- were not promised or were not assumed to be given any such rights.

LLC. Miramine will receive the payment for preparation of this report in accordance with the regular practice of professional consulting. This payment does not depend on the outcome of any activity for raising funds, and LLC. Miramine will not receive any other profit from preparing this report. LLC. Miramine does not have any material or other interest which could be reasonably considered to be capable of influencing its ability to give neutral opinion in relation to the Mineral Resource estimate of the Company.

Therefore, LLC. Miramine, the Competent Persons and the Director of LLC. Miramine consider themselves independent from the Company.

This report includes the technical information which requires further calculations for making subtotals, totals and weighted average values. Such calculations can include some specific degree of roundoff and, thus, can lead to error. If such errors occur, LLC. Miramine does not consider them significant.

LLC. Miramine is responsible for this report and declares that it took all reasonable measures to provide correspondence of the information contained in this report to the facts, as far as they known, and no other significant facts known to the Customer, that could influence the conclusions, were omitted.

The Competent Person Alexey Nikolayevich Nikandrov, MSc, MAIG, LLC. Miramine full-time employee, is an independent Competent Person as defined by the JORC Code (2004). As a Competent Person, he has at least five years of experience in exploration of deposits of such type and assumes all responsibility for the information given below.

MICROMINE

This is to certify that
Данное свидетельство выдано

Nikandrov Alexey Nikolaevich

Никандрову Алексею Николаевичу

of JSC "Pokrovsky Mine"
Из ОАО "Покровский Рудник"
Has successfully completed the course
В том, что он успешно закончил курс

Advanced Resource Modelling
with MICROMINE Version 10
Подсчёта запасов в системе
Майкромайн версии 10

Training Course, held in MM Head Office, Perth, WA
Курс проводился в головном офисе ММ. г.Перт, Западная Австралия

Dmitry Pertel, Course Coordinator
Дмитрий Пертель, Координатор курса
Certificate No: MM.05.96

July 2005

MICROMINE PTY LTD
TRAINING CERTIFICATE

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The Australian Institute of Geoscientists

hereby certifies that

Alexey Nikolaevich Nikandrov

Membership Number **3084**

has satisfied the requirements for membership of this Institute, as stipulated in the Articles of Association, has agreed to accept the Institute's Code of Ethics and has been admitted as a

Member

of the Institute

Issued this **19th** day of **January** 20**05**

PRESIDENT

FEDERAL COUNCILLOR

6. Disclaimer

Opinions and estimates given in this report are based on the information provided by LLC. Krasny to LLC. Miramine.

LLC. Miramine accurately examined the information provided. Although LLC. Miramine compared the key data provided with the expected figures, the accuracy of the results and conclusions of this work wholly depend on the accuracy and fullness of the data provided. LLC. Miramine does not accept responsibility for any errors or omissions in the information provided and does not bear any further responsibility associated with the commercial decisions made or actions taken on their basis.

7. Location and Geological Understanding of the Krasnoye Gold Ore Occurrence

7.1. Landscape-Geographical, Climatic and Hydrological Conditions

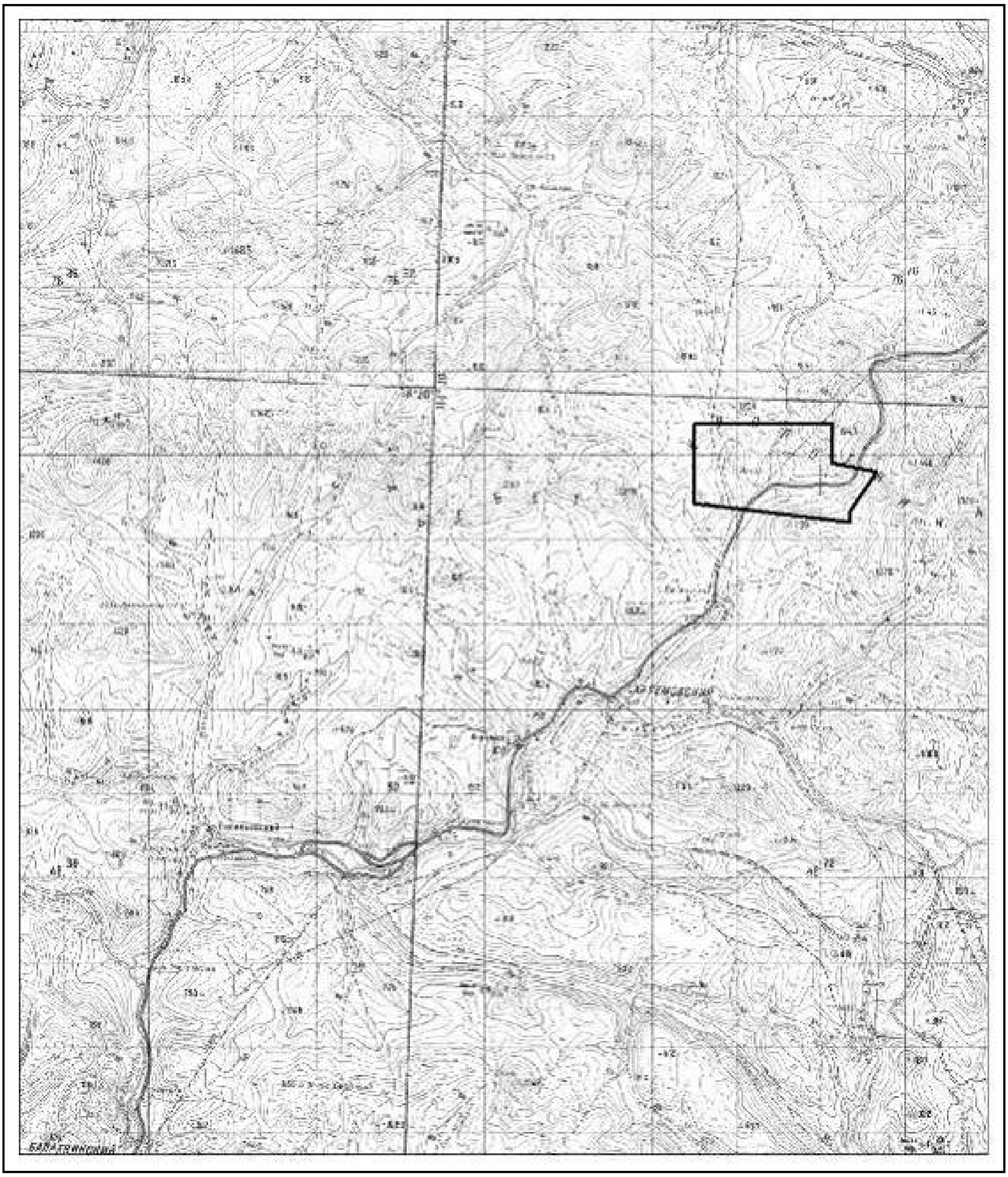
The Krasny site license area is located at the right bank of the Bodaibo River, in its upper reaches.

The climate of the district is extreme continental. The average annual air temperature is 6˚C. The average temperature variations are from 54˚C in January to +34˚C in July. The average annual amount of precipitation is 350 mm, most of it falling during a warm period. Snow falls in the mid or the end of September and thaws out completely in the end of June. The thickness of the snow cover in valleys reaches 2-4 m.

In the orographic relation the district is located within the Patom uplands with typical middle-mountain topography. The elevation varies between 800 and 1,200 m, with the local difference in elevation making 500-600 m. The topography is characterized by flattened watershed tops and steep, up to 30^{o} ($10\text{-}15^{o}$ on average), slopes covered with a slope deposits talus cone several meters thick. The degree of outcropping of the area is low. Bedrock outcrops are local. Glacial deposit residuals tens of meters thick are noted on watershed anticlinal folds and slope bottoms. River valleys are wide, frequently bogged. Most of valleys are complicated by technogenic deposits as a result of placer mining. The alluvium thickness reaches 50 meters. Technogenic dumps tens of meters thick, especially in the Teply creek, significantly complicate the exploration process.

The river network of the work area is presented by minor watercourses freezing out almost completely in winter. The Bodaibo River with the following tributaries is the main watercourse: Krasny, Teply and Mokry Creeks. Short springs drying out in summer after the end of the permafrost thawing period are found in minor creek valleys. The highest water rate falls on May-June – the periods of intensive snow thawing and rains. Permafrost is generally developed on slopes and watersheds, especially of the northern aspect. Thawed zones are noted under river beds. The seasonal thawing depth by the end of summer does not exceed 2 m on southern slopes and up to one meter on northern ones.

Figure 7-1. Review map of the work area



7.2. Level of Economic Development of the Area and Infrastructure Characteristic

The district is economically developed quite well and is located 70-80km away from explored ore deposits Sukhoy Log, Verninskoye and Vysochayshy. Gold is mined in the surroundings from placers of various types. The project is administratively subordinated by the Artemovsky municipal settlement of the Bodaibo District of the Irkutsk Region.

The nearest village - Artemovsky is located 15 km away. The distance to Bodaibo is 75 km. There is a motor road Bodaibo-Kropotkin-Perevoz passing across the site. There are gravel and dirt motor roads within the site in the lower reaches of creeks. They are currently used by placer miners. Roads require snow removal within the site in winter. Cargo delivery to the Krasny site from LLC. Kopylovskoye base (Bodaibo) is executed by cross-country vehicles. Most of cargos are delivered to the base from the nearest railway station Taximo (Baikal-Amur Railway) by motor road 220 km long. Water transportation along the Lena and the Vitim Rivers from the Osetrovo river port (Ust-Kut) to Bodaibo (750 km) is possible during the navigation period. In Bodaibo there is an operating airport which accepts cargo and passenger planes with medium capacity from Irkutsk (1200 km), Bratsk, Mirny, Kirensk and Ust-Kut.

The area enterprises are supplied with energy from the Mamakan hydro-electric power station and thermal plant by means of 110kV and 220kV overhead power lines Taximo-Bodaibo. 110kV and 36kV overhead power lines pass in proximate vicinity to the work area.

Telephone communication on site with the base and company's management is provided with the help of a satellite telephone.

7.3. Geological Structure

7.3.1. Stratigraphy

The central part of the Lenskaya gold-bearing province (Artemovsky ore cluster), within which the Krasny site is located, is positioned in the miogeosyncline baikalid belt in the northern limb of the Bodaibo complex syncline – one of the main structures in the central part of the Bodaibo synclinorium.

The Bodaibo complex syncline is composed of sedimentary-metamorphic rocks of the Bodaibo series of the Upper Proterozoic complex and is covered by loose deposits of the Quaternary system.

The **Bodaibo Upper Riphean series** is divided into two parts on the basis of its lithological and facial particularities. The lower one presented by Aunakit and Vacha suites is composed of carboniferous monomictic quartz sand-slate deposits. The upper one including the Anangr, the Dogaldyn and the Iligir suites, is characterized by the development of mainly sandy and terrigenic carbonate formations.

The rocks of the Aunakit, the Vacha and the Anagr suites crop out to the day surface within the Krasny site.

The **Aunakit suite (R_3au)** is the most ancient subdivision within the area under study. The suite deposits within the design area are developed in the northwest and compose the core parts of the Verkhne-Bodaibinskaya and the Rudnaya anticlines complicating a larger Bodaibo syncline. The suite is divided into three subsuites on the basis of lithological particularities. It should be noted that the lithological composition of sandstone varies from quartz-sericitic in the southeast of the area to carbonate in the northwest.

The lower subsuite ($R_3\ au_1$) is mainly composed (by 90%) of quartz sandstone with single carboniferous phyllite and calcareous sandstone interbeds. The thickness of the subsuite is 100-120 m. The lower subsuite rocks within the site do not crop out but are intersected in boreholes only.

The middle subsuite (R_3au_2) is presented by dark-grey, grey calcareous phyllites with thin quartz sandstone and siltstone interbeds. The thickness of the subsuite is 400 m.

The upper subsuite ($R_3\ au_3$) is most widely developed. The lower part of the section is presented by grey, dark-grey quartz metasandstone with dark-grey carboniferous aleurite slate interbeds which become dominating over carboniferous metasandstone in the middle part of the section. The upper part of the section is characterized by prevailing quartz metasandstone (70-

75%) with carboniferous chlorite-sericite slate interbeds and rare calcareous metasandstone interbeds. The thickness of the subsuite is 400-500 m.

Therefore, the rocks of two lithological groups participate in the suite structure: metasandstone and slate. Sandstone with calcareous cement is subordinated.

The total thickness of the Aunakit suite is 1000 m.

Vacha suite (R_3vc). The Vacha deposits occupy the central part of the area. In the structural relation they compose the limbs of anticline folds and the core of the Lozhkovaya syncline.

The suite is composed of black high-carbon and sericite-quartz slates (80-95%) with thin interbeds of carboniferous quartz sandstone. Coarse alternation of rocks with interbed thickness 0.5 m and above is typical for the section bottom. The thickness of the suite varies from 400 m to 600 m depending on the structural position.

The Vacha suite is clearly expressed in the section and is a reference stratigraphic unit of the region on the basis of lithological rock particularities (homogeneous composition, black colour, and high content of carboniferous substance).

The Vacha suite is divided into two subsuites: Upper and Lower.

The Lower Subsuite (R_3vc_1), first horizon ($R_3vc_1^1$) is characterized by high-carbon siliceous slates with dark-grey quartzitoid sandstone interbeds. *The second horizon ($R_3vc_1^2$)* is composed of black high-carbon phyllites and sericite-quartz slate with rare quartz-sericite sandstone interbeds.

Two horizons are also defined within the *Upper Subsuite (R_3vc_2)*: *the first horizon ($R_3vc_2^1$)* mainly presented by alternation of dark-grey quartzite and high-carbon phyllite, and *the second horizon ($R_3vc_2^2$)* composed of black high-carbon phyllites and sericite-quartz slates.

Anangr suite (R_3 an). The deposits of the Anangr suite are noted in the south of the area, composing the southern limb of the Rudnaya anticline, and in the northeast bedding conformably on the Vacha rocks. Two subsuites are defined on the basis of the particularities of the lithological rock composition: Lower – black-slate and Upper – light sandstone.

The *Lower Subsuite (R_3 an_1)* is presented by two-component rhythmic alternation of dark-grey carboniferous aleurite slates and grey small-grained metasandstone. The thickness of the subsuite is 150 m.

The *Upper Subsuite (R_3 an_2)* unites a complex of coarse-alternating light unequigranular metasandstone, carboniferous slates and metasiltstone, and rare polymictic metagravelite lenses. The thickness of the subsuite reaches 200-300 m.

According to the electric exploration data, the formations of the Vacha and the Anangr suites are characterized by a differentiated field of apparent resistance from units to 100 Ohm and natural field anomalies with the intensity -500 to 750 mA. Magnetic fields are calm, negative with the intensity 0.25-0.75 mOe, Typical for low metamorphism zones.

As for potential ore content, it is assumed that the rocks of the Aunakit and the Vacha suites are potential for mineralization localizing. According to their lithological composition, they are favourable for formation of sulfide-veinlet gold-ore projects. Quartz-vein mineralization (thickness and number of veins) generally grows with increase of the rock competency and is mainly subordinated to structural control.

Quaternary deposits covering Upper Riphean formations occupy almost the whole day surface. Upper Pleistocene (Q_{III}) and Holocene (Q_{IV}) deposits of the Quaternary system are identified within the work area.

Upper Pleistocene glacial deposits are presented by boulders, crushed stone, debris, rare lumps and pebble of local, rarely exotic rocks with aleurite, silt-clay and sand filling. The sizes of lumps and boulders reach 4 m. The total amount of crushed-stone-boulder and pebble-lump material makes about 30%.

Water-glacial deposits differ from glacial ones with belt-stratified structure due to prevailing silt-clay component and unequigranular sand and clay sand in the total mass. The upper parts of the section are frequently composed of pebble stone.

The total thickness of the complex of glacial and water-glacial deposits (Q_{III}) reaches 15-20m and locally falls till 4-6 m under terraces.

The current deposits (Q_{IV}) of the exogeodynamic series are presented by the following genetic types: bed alluvium, alluvium of lower above-floodplain terraces of 3-15 m and 20-40 m levels, and diluvium-solifluction and proluvium formations. Eluvium deposits of weathering crusts of mainly physical series are developed on watersheds.

A two-member structure is typical for alluvial deposits of the area: there are boulder-pebble formations and pebble stone of bed facies in the bottom section, covered by sand, clay sand and silt from the surface.

Differently rounded fragments of Upper Riphean metamorphic rocks dominate in the modern alluvium composition. Well rounded pebbles of granitoids of the Konkudero-Mamakan complex are present in an insignificant quantity.

Slope formations are presented by boulder streams, rockslides, temporary flow alluvial cones, deluvial talus cones in the bottoms. These are unrounded fragments of local rocks in the clay-silt matrix. The amount of the latter grows in the lower part of the section.

Eluvium of watersheds and flat slopes represents a fragmented cliff covered by poorly bound bedrock cryoclastites. These are debris-crushed stone-boulder fragments in the cryopelitic matrix formed on local substratum. Eluvium covers the above-mentioned geomorphologic taxonomic units blanketlike, obscuring the primary bedding of rocks and complicating observations. The fragmented material is frequently intensively weathered. The amount of the latter grows towards the bottom of the loose horizon where it transfers to the structural bedrock eluvium.

7.3.2. Intrusive, Metamorphic and Hydrothermal-Metasomatic Formations

Intrusive igneous rocks in the work area are not detected, although widely developed along the periphery of the Bodaibo zone. Thus, granitic massifs of the Konkudero-Mamakan complex limit the basin of the Bodaibo River from the south (Engazhimino-Vitim massif), the northeast (Dzhekdokarsky) and the northwest (Chumarkoysky). Granites are located at the distance of 40-50 km from the work area. The isometric boron anomaly covering also the work area and, as a result of acidic metasomatosis, presence of tourmaline in metamorphosed rocks of the area serve as one of indirect indicators for the granitic intrusives nearness.

Metamorphism. The Riphean deposits of the Bodaibo structural-facial zone generally suffer progressive zonal regional metamorphism with contact impact and regressive metamorphism indicators near granitoids of the Konkudero-Mamakan complex. Dislocation metamorphism formations are also developed within the area under study.

Zonal regional metamorphism. Mineral parageneses of regionally metamorphosed rocks correspond to the greenschist and the disthene-muscovite slate facies, formed in the middle-pressure conditions (Dobretsov et al., 1980). The greenschist facies is widely developed in the zone of near-latitudinal folds of the Bodaibo synclinorium. It is subdivided into low-temperature (chlorite) and high-temperature (biotite) zones located amidst the rocks of the Anangr, the Dogaldyn and the Aunakit, the Vacha, the Anangr, and the Dogaldyn suites respectively. The identified zones coincide with the strike of folded structures.

The zone of chlorite subfacies of the greenschist facies is characterized by:

- presence of the following minerals in the rocks of various lithological groups: actinolite, epidote, chlorite, quartz, albite, pyrite, pyrrhotite and other sulphides;
- a combination of folded and fault dislocations;
- a good degree of preservation of structural and textural particularities of sedimentary rock.

Dislocation metamorphism in the rocks of the site occurred in several stages on the rock and the mineral levels and led to cleavage and schistocity formation. Cleavage is widely developed in the limbs of folds of various orders, found in the rock area. Intensive cleavage is typical for more pelitic rocks and, to a lesser extent, for psammitic rocks. The rocks in the cleavage zone are dissected by near-parallel or crosscutting at an angle fractures into strips 1-5 m long and lenses 0.1-1 mm thick.

Dislocation metamorphism in the work area is characterized by:

- zoning reflecting temperature conditions of metamorphism (for the low-temperature process – cleavage; for a higher temperature one – schistocity);
- a complex configuration of zones depending on the nature of rock strata;
- conformity of tectonic structures;
- impact of the grain-size composition of rocks.

Hydrothermal-metasomatic formations. Hydrothermal-metasomatic rock alterations were expressed in general recrystallization of the rock cement, re-distribution of the carboniferous substance, intensive occurrence of brown feldsparization and sulphidization. These transformations are most evident in anticlinal structures, the zones of intensified fracturing and are widely developed in the area.

Calcite replacement with ankerite and siderite with simultaneous transportation and concentration of ferriferous-magnesial carbonates and sulphides into lithologically and geochemically favourable bands (carboniferous strata) occurred in the process of metamorphism. This led to the formation of veinlet-disseminated sulphide and carbonate mineralization zones. In addition to these metasomatosis occurrences, general rock silicification and formation of quartz vein fields extended in the near-latitudinal direction in the core parts of folds is observed. The development of shear deformations, schistosity and ore-metasomatic processes could occur in parallel. The formation of quartz veinlets is associated with the process of silica squeezing-out from compression sites to opening cleavage fractions.

The local re-distribution of “embryonal” gold in host rocks with its further concentration on polygenetic geochemical barriers could be associated with the metasomatosis process. The sites with increased gold grades are currently characterized by all metasomatic and hydrothermal processing indicators: increased content of sulphides, silicification, sulphide, quartz, and quartz-sulphide veinleting.

Metasomatically altered rocks within the Bodaibo syncline are localized mainly in the Bodaibo River basin, where their share is approximately 87% of the area. Most of previous researchers identify the following types metasomatites: beresites (according to researchers – listvenites), pyritized and pyrrhotitized rocks, silicification zones.

The beresite development halo includes the area of the Nakatami River basin adjoining from the west. In the east beresites are quite quickly replaced by the zones of pyrite-altered rocks. This is explained by the metasomatosis zoning. Beresites are localized in the hinge parts of anticlines of various orders, in contact zones of unequigranular layers. The Aunakit suite and the Lower subsuite of the Anangr suite are the most favourable stratigraphic levels for beresites in the region.

The gold grade in beresites usually makes thousandth fractions of g/t, an order higher in single cases. In the cases when the gold grade is higher in beresites, the latter always contain veinlet quartz and increased content of pyrite bearing most of gold-ore mineralization.

Pyrite- and pyrrhotite-altered rocks occupy limited spaces as compared to beresites, localizing in cleavage and schistosity zones. They are typical for the deposits of the Aunakit, the Vacha suites and the lower subsuite of the Anangr suite. They are widely developed within the work area. Rock pyritization is irregular and there area sites where the degree of saturation with pyrite reaches 3% and above. The gold grade in these rocks usually makes thousandth fractions of g/t, and increase of the gold grade up to industrial values is observed on local sites only.

Quartz veins, the formation of which is associated with local substance differentiation during metamorphism and the development of folding structures of the region, are widely developed.

As the results of previous works conducted within the ore fields of the district showed, the varieties of quartz-vein formations can be divided into the following types in the morphological relation: saddle-shaped veins; plate-shaped veins, lens-shaped veins; stockwork-like zones of adjacent quartz veins and veinlets (including zones of veinlet-disseminated quartz-sulphide mineralization); zones of ladder-type veins, lens-shaped and beaded veinlets and their zones, and folding veinlets.

The formation of quartz veins mainly occurred in tense dynamic conditions. Intensified schistosity and crushing of contact parts of veins, presence of tectonic hatch and fault polishes, frequent beaded vein shapes, presence of pseudo-boudines, deformation of veinlets, presence of several quartz generations testify to that.

The ore mineralization of quartz veins is quite poor (below 1%) and is presented mainly by sulphides: pyrite, pyrrhotite, rarely galena, sphalerite, chalcopyrite and silver. Veins concentrated in the lower-temperature zone of regional metamorphism have increase gold grades.

Precipitation beds of ironstone in the bed parts in the middle reaches of the Teply and the Krasny Creeks are one of the specific formations of the work area. The floodplain deposits of creeks are red-brown, ochreous due to “cold” metasomatosis. In addition to the bed parts of above-mentioned creeks, limonitization is observed in the form of local spots in local gullies of the topography.

7.3.3. Tectonics

The Bodaibo synclinorium is the largest structural unit of the district. It occurred at the place of the same-name Riphean paleodepression.

The Bodaibo synclinorium, as the structure of the first-order, is divided into a number of minor structures. The Bodaibo complex syncline is one of them. It joins the Kropotkin complex antiform in the north and the Tamarkan complex antiform in the south and hosts the research area.

Near-latitudinal fold strike is typical for the basin of the Bodaibo River in general. Anticlinal folds are frequently asymmetric with larger southern limbs. Synclines are mostly flatter, with wide hinges. The structural paragenesis is presented by cleavage and schistosity, minor folds, boudinage, mineral linearity, co-folding longitudinal faults, low-amplitude diaclases and fracturing. Deep faults played a significant role in the formation of area structures. Their activation preconditioned complication of earlier formed structures and formation of new ones.

Most intensive formations are typical for the axial zone of the Bodaibo syncline and, to a lesser extent, - for adjoining complex anticlines which evidently predetermined their various productivity in relation to gold. The folding stage of deformations, caused by general meridianal compression, led to the formation of linear folding of the Verkhne-Bodaibinskaya folding zone and associated co-folding upthrusts. Sublatitudinal fracturing zones formed during this period and zones of layer-by-layer movements were filled with metamorphogenic quartz-vein and sulfide mineralization. The formation of gold mineralization is associated with the final stage of folding. Spatial-genetic relation of the gold quartz-sulfide and vein-quartz mineralization with the zones of intensive occurrence of hydrothermal metasomatic processes (carbon dioxide and sulphuric metasomatosis) is established.

It shall be noted that the processes of metasomatic processing were most evident in steep limbs and the axial parts of anticlines of the 4^{th}-5^{th} order, where quartz-vein and sulphide mineralization were formed. Here, increase of intensity and sizes of ferriferous-magnesial carbonates a well as intensified sericite alteration of rocks is observed.

The Krasny site is located in the zone of impact of the Verkhne-Bodaibinskaya deep intra-block fault expressed in the form of linear folding of the regional folding zone at the surface. Tense different-scale folding is typical for the area. It is complicated by multiple faults, microfolding zones, boudinage and tectonic mélange.

Plicative structures identified within the site include the Verkhne-Bodaibinskaya (in the north) and the Rudnaya anticlines (in the south) and the Lozhkovaya syncline separating them.

The Verkhne-Bodaibinskaya anticline has the strike of the axial surface 285-290^0, with dipping to the north at 70-75^0, dipping of a hinge to ENE at 5-10^0. The fold is asymmetric, with dipping of the southern limb to the SW at 60-85^0 and the northern limb to the NW at 20-55^0. Limbs are complicated by additional folds of higher orders, and contrast gold halos are confined to them. As the most recent data of previous researchers shows (Melnik, 2006), folds of higher orders have isoclinal morphology in the area of periclinal closure of the anticline (area of the Mokry Creek). It shall be noted that the quartz-vein zone and the zones of dispersed sulphide mineralization are confined to the periclinal closure of the fold.

The Rudnaya anticline has a complex structure due to a dislocated northern limb. A wide development of folds varying in morphology and sizes, frequently located en echelon in relation to each other, is typical for it. The northern limb of the Rudnaya anticline is more compressed and steep (60-80^0) and is characterized by undulating alteration of the dipping of layers from 45^0 to 90^0. Additional folds are also found there with the amplitude from first meters to 50-100 m. The anticline hinge generally dips to the ENE at 10-5^0, but it beds near-horizontally at the watershed of the Teply and the Krasny creeks. The most intensive mineralization with increased gold content is confined to this part of the fold.

The Lozhkovaya syncline is characterized by an intensively dislocated core complicated by en echelon upthrust. Regular reduction of the width of the syncline from west to east is another particularity.

Disjunctive dislocations are presented by near-latitudinal upthrusts and downthrusts, near-longitudinal downthrusts-shifts and zones of increased fracturing and crushing.

Near-latitudinal faults are most widely developed. They are concentrated in the flat limb of the Rudnaya anticline and in the central parts of the Verkhne-Bodaibinskaya anticline. They are a part of the folding zone. The length of individual faults is 10-100 m to 4-5 km, rarely more. Their location is en echelon in relation to each other. Tectonic joints are usually presented by grinded fractures or fracturing zones filled with quartz, quartz-sulphide veinlets and quartz veins.

The orientation of near-latitudinal faults is conformable to the direction of main structures. The dipping is usually steep at 60-80^0. The amplitude of displacement along tectonic joints is low, making approximately first cm to first tens of meters.

Near-longitudinal downthrust-shifts are expressed in the form of series of adjacent steeply dipping shear fractures with fault polishes and channels. The displacement amplitude is insignificant – 0.5-1.0 m, rarely up to 5 m. The length of such faults does not exceed first hundreds of meters.

Minor structural forms include microfolding, boudinage, tectonic crushing and fracturing zones. The degree of occurrence of this or other type of minor folding forms depends on the lithological composition of intersected horizons. The zones of microfolding first cm to tens of meters thick are developed in thin rock alternation horizons. Boudinage is typical for coarse rock alternation; the boudine orientation is in all cases with their long axis along axial cleavage.

Zones of intensified fracturing and crushing have near-latitudinal and near-longitudinal orientation and form single zones. Rock fracturing is very intensive. Main fracture systems are defined within the Krasnoye ore field, similar to all studied ore occurrences and deposits of the Bodaibo ore-bearing district: cleavage 1 (dip azimuth 10-20^0, dip angle – 0-75^0); cleavage 2 (dip azimuth 90-100^0 and 280^0, dip angle – 70-80^0).

Moreover, there are fractures of NW orientation (dip azimuth 235^0-245^0, dip angle 80^0) and an insignificant number of flat ruptures (dip azimuth 110-120^0, dip angle 20^0).

Axial cleavage is generally developed and coincides at the areas of monoclinal bedding, with layer-by-layer cleavage developed at contacts of lithological varieties.

Near-latitudinal fracturing zones and zones of layer-by-layer shifts were formed during linear folding formation. These tectonically weakened zones were used by ore-bearing fluids for ore localization. At the ore occurrence the fracture tectonics controls the location of veinlet quartz-sulphide and quartz-vein mineralization.

7.3.4. Placer Gold Content

The gold placers of the Nearest Taiga (Blozhayshaya Taiga) unique in the grade have been developed, explored and studied for more than 100 years. The placers of the Bodaibo River and its tributaries are the richest. An exceptionally wide development of buried alluvial placers is typical for them. The placers of deep valley bottoms are dominating. Terraces of the low (up to 15 m), the middle (15-30 m) and the high (above 30 m) levels and bed placers are rarer.

Deep valley bottom placers are characterized by long extension, continuity and a high degree of saturation with metal (2-3 t/km). Low-level terrace placers are developed most widely and have the same degree of saturation as deep valley bottom placers. The placers of middle-level terraces are not very rich. High-level terrace placers are developed poorly.

River bed placers are most widely developed and are completely mined out to date.

Narrow placers are small in size and reserves and are characterized by irregular metal grades. Weak development of narrow placers is associated (Tischenko E.I.) with wide occurrence of solifluction processes in narrows.

Eluvial and deluvial-solifluction placers are unknown in the Bodaibo complex syncline, although concentrate gold halos are identified within all ore fields.

The placer gold content is known in the valleys of the Teply and the Krasny Creeks within the designed work area. These are river bed placers of minor watercourses. In this case the upper, head parts of placers are located orographically above the known ore occurrences. Previous researchers did not reveal any significant placer god resources in slope deposits of the territory.

7.3.5. Gold Mineralization

Two types of gold-ore mineralization are identified within the Krasny ore field: veinlet-disseminated quartz-sulphide type of mineralization with lithological and structural control, and quartz-vein type with limited development. These two types frequently spatially coincide.

The ***quartz-sulphide mineralization*** forms veinlet-disseminated zones amidst metamorphogenic dispersed zones of sulphide mineralization within the sites of structural complications. The internal structure of the zones is complex and represents a thick grid of veinlets, lenses and pockets oriented in different directions and found together with intensive (above 1-3%) dispersed dissemination (pyrite) in schistose and cleavaged host rocks. In general, increase of the number of veinlet-lens-shaped ore mineral association inclusions occurs in parallel with increase of disseminated sulphide mineralization.

A wide development of sulphide mineralization presented primarily by pyrite is a typical feature of the whole ore field. Pyrrhotite, chalcopyrite, sphalerite and galena are is subordinated quantities. Pyrite is presented by the following morphologic varieties: 1) dusty pyrite (pre-ore), 2) in the form of unclearly expressed inclusions; 3) porphyroblastic cubic pyrite, 4) lens-shaped pyrite, and 5) pyrite in quartz veinlets.

The gold grade in pyrite varieties is different. Lens-shaped aggregate cubic and cubic porphyroblastic pyrite has the highest gold grade.

Pyrite in quartz veinlets is most frequently observed within the ore field and is considered low-grade, with the gold grades 0.002-0.8 g/t. The thickness of quartz-pyrite veinlets is from fractions of mm to 2 cm and above; the degree of their saturation of host rocks varies and makes from 1 per 1 m^2 to 5-6 per 1 m^2. Quartz in veinlets is white, semi-transparent, amorphic.

Gold in veinlets is noted in the form of irregular grains 0.02 x 0.01 mm to 0.06 x 0.12 mm in size, present at the quartz contact with pyrite.

The axial part of the Rudnaya anticline (where the Krasnoye ore occurrence is located) and the axial part of the Verkhne-Bodaibinskaya anticline (Verkhne-Bodaibinskaya mineralized zone) are most mineralized. These are referred to as Southern and Northern mineralization zones below.

The ***quartz mineralization*** is localized in veins and veinlets, usually with lens-shaped morphology: thickness 0.2 cm to 0.5 m and length from first centimeters to tens of meters. Quartz-vein fields and zones are confined to the core and to the southern limb of the Verkhne-Bodaibinskaya anticline and the southern limb of the Rudnaya anticline. They are located within near-latitudinal zones (belts) of dispersed quartz and quartz-sulphide mineralization. The distribution of quartz-vein material in these belts is irregular and the sites with increased concentration alternate with barren ones. It is assumed that the ones which suffered additional hydrothermal processing are the only potential ones. The following vein and veinlet systems are defined: stratal near-latitudinal continuous in size and strike near-horizontal and near-longitudinal discontinuous in size.

The northern belt of quartz veins is confined to the core and the southern limb of the Verkhne-Bodaibinskaya anticline and stretches across the whole area with the width 0.8-1.0 km. The maximum degree of saturation with quartz is registered within the license area by previous researchers in the upper reaches of the Teply and the Krasny Creeks and at the watershed of the Krasny and the Mokry Creeks. Moreover, a zone of quartz vein development is noted, confined to the southern limb of the Rudnaya anticline. The most evident occurrences are defined at the Teply and the Krasny watershed and at the Teply and the Topky watershed.

8. METHODOLOGY, TYPES, SCOPES AND QUALITY CONTROL OF PERFORMED EXPLORATION

The purpose of works within the Krasny license area is completion of prospecting and appraisal for ore gold with forecast resource estimation with category P1 and reserve estimation with category C2. A complex of field, analytical and office works was executed for solution of these tasks. It included prospecting traverses, geochemical, geophysical studies, drivage of workings, drilling, sampling and topographic and geodetic works.

Since the works were conducted at the expense of the enterprise and had commercial purpose only, the design was implemented from the minimum sufficiency and the maximum profitability standpoints, with full observance of the work staging. Methodological works for testing a new method of geophysical studies – aerial tomography – were conducted. All works were conducted in stages.

8.1. Geophysical Work

8.1.1. Borehole Logging

In order to receive additional data on geological sections, gold mineralization parameters, the position of the bore and the condition of boreholes, a complex of geophysical studies was conducted. It included gamma logging, directional survey, electric logging, apparent resistance logging, and an electrode potential method (or a sliding contact method) in dry boreholes.

Borehole logging was conducted by a contractual organization Yuzhnaya Expedition of OJSC Krasnoyarskgeologia.

All types of logging were conducted in scale 1:200 for the whole depth of drilled boreholes.

Electric logging was conducted in the MSK modification with the purpose of lithological division of the section in boreholes, with identification of zones of hydrothermal rock alteration. Magnetic susceptibility logging was conducted for solution of the same tasks.

Directional survey was executed with the purpose of determining the actual trajectory of drilled boreholes. Directional survey was conducted with the devices ИММН 42, ИЭМ-36-80/20.

Gamma logging was used for the radiation-hygienic evaluation of rocks and ores composing the borehole section, and for identifying the zones of hydrothermal alterations and stratigraphic borehole reference.

All logging types were executed in accordance with the requirements of effective instructions.

A logging station Vulkan on the Ural truck chassis was used for borehole studies.

Geophysical studies of boreholes were conducted immediately after completion of the geological assignment by the borehole (ore zone intersection, reaching the design depth) and borehole preparation. After receiving the primary results of directional survey and the analysis of geological information, decisions on borehole acceptance and closure were taken.

The results of all logging types were recorded on a borehole geological column form for interpretation and reference to the geological documentation results.

8.1.2. Areal Geophysical Work

The task to prepare a geophysical basis for prospecting exploration, mapping of the ore zones known in individual intersections, and identification of sites potential for gold mineralization was set for geophysical studies at the Krasny site located in the Bodaibo District.

The task to study the structure on the basis of sections along profiles, the location of sulphidized zones, to integrate with the available geological and geophysical materials, and, if possible, to form a forecast was set for electric tomographic works with the induced polarization method.

In order to solve these tasks, 2 contracts were signed in 2011 between LLC. Kopylovsky and LLC. Geo Servis. The work sites and profiles for tomographic studies were identified by the Customer's geological service on the basis of available geological materials. Field works were conducted in August-September 2011. During the office work period office studies were conducted. Their results included plans of isolines and actual field graphs, sections, and schemes of complex interpretation results.

Field geophysical work in 2011 was conducted within the Krasny site in 1:10,000 scale. Electric tomography was executed on 6 profiles. The planned position of profiles was agreed with the geological service of LLC. Kopylovsky and was defined by the license agreement parameters.

The surface geophysical work of 1:10,000 scale (Table 8.1) included:

- magnetic exploration with the distance between profiles 100 m and with the space along the profile 5 m, with the use of a MINIMAG magnetometer;
- magnetic variation monitoring was conducted with the use of a MINIMAG magnetometer installed near the camp in a calm magnetic field, with the frequency 1 min;

- induced polarization survey in the symmetric electric profiling modification, with determination of a phase shift angle and apparent electric resistance with the use of a meter MERI-24 and a generator ASTRA-100, with the distance between profiles 100 m and with the space along the profile 20 m. Unit - A100M40N100B. Monitoring frequency – 2.44 Hz.
- electric exploration with the natural field method, in the potential modification, with the distance between profiles 100 m and with the space along the profile 20 m.

Table 8-1 Types and scopes of surface geophysical studies of 1:10,000 scale

No	Type of Work	UoM	Work Scope, km		Note
			Design	Actual	
1	2	3	4	5	6
Krasny Site					
1	Magnetic exploration (5 m space)	sq.km	3.8	3.8	Excluding 5% control
2	Magnetic variation monitoring			7 working days	
3	Induced polarization survey in the profiling modification (20 m space) Frequency 2.44 Hz	sq.km	3.8	3.8	Excluding 5% control
4	Electric exploration with the natural field method (20 m space)	sq.km	3.8	3.8	Excluding 5% control

Evaluation of the field accuracy of measurements was conducted on the basis of repeated monitoring in the volume of 5%.

Electric tomography with the induced polarization method (Table 8.2) was executed with a standard methodology along profiles with SyscalPro devices.

Table 8-2 Scope of surface electric tomography with the induced polarization method

Area	Number of Profiles	Profile Length	Total Length
Krasny	6	535	3210

8.1.3. Magnetic Survey. Methodology and Results

Methodology

Magnetic survey was used for solution of the following task:

- for division of different (by magnetic properties) formations;
- for localization of areas of metamorphic and hydrothermal transformation development;
- for mapping of tectonic faults on the basis of the magnetic field structure particularities.

The work was conducted in accordance with the norms with MIOMAG magnetometers. Reference profiles were developed prior to the beginning and after completion of works with the working magnetometer and the magnetometer used in the geomagnetic-variation system mode. A MINIMAG magnetometer was used as a geomagnetic-variation system.

The geomagnetic-variation system was equipped near the field camp in a calm, normal field. Variations were measured every minute during the whole working day. Prior to the beginning of work, the working capacity was measured every day for each device and the geomagnetic-variation system on the control point combined with the point where the geomagnetic-variation system was equipped. Simultaneously the hours of the regular survey operator and the geomagnetic-variation system were agreed. The survey space made 5 m.

The monitoring error is given in Table 8.3.

Table 8-3 Errors of physical field parameter monitoring

Method	Unit of Error	Permitted Error as per Geological Assignment	Error Received during Field Work
Krasny Site			
Magnetic field induction	nT	±5	±3.7
Apparent specific electric resistance	%	10	5.3
Phase shift angle	Degree	0.15	0.11
Natural electrical field potential	%	5	3.2

During office processing allowances were introduced for variations, the regular survey accuracy was estimated; the graphs of increase of the magnetic field inductance full vector (ΔT) were built for profiles with regard to allowances for variations, and also plans of isolines and plans of site graphs were prepared. Files cleaned from technogenic anomalies were used for building isoline plans. ΔT calculation the average value was assumed as the level of the normal field, namely:

- Krasny Site – 60,775 nT.

Results

The structure of the received magnetic field corresponds quite well to the structure of the magnetic field received as a result of previous works, on the basis of its generic particularities. But this statement is correct for general particularities. The values of **ΔT** within the survey site vary within a very narrow range – +50 to -50 nT.

The site can be conventionally divided into two parts on the basis of the magnetic field structure particularities:

- the area of relatively calm reduced (0 - -30 nT) values of the magnetic field;
- the area of relatively increased magnetic field values with local negative anomalies.

8.1.4. Electric Survey with Induced Polarization Method. Methodology and Results

Methodology

Electric survey was used for solution of the following tasks:

- for division of different (in terms of specific electrical resistance and polarizability) formations;
- for identification of silicification, sulphidization and carbonization zones;
- for mapping of tectonic faults on the basis of resistance field structure and phase shift angle distribution particularities.

The task of evaluating apparent specific electric resistances and detecting electronic conductors (negative phase shift angle), as well as dividing the section on the basis of specific electric resistances and polarizability was set for this work. The work was conducted with the use of the devices of the MERI-24 meter and a generator ASTRA-100, in accordance with the norms (Electric Survey Instruction, 1984). A portable generator ASTRA-100 was used as a current source. The work was conducted with a unit A100M40N100B at the frequency 2.44 Hz.

The space along the profile was assumed 20 m in accordance with the Terms of Reference. Steel studs were used as supply and metering electrodes. The unit was assembled of the GPSMP wire.

The following values were estimated during office processing: unit coefficient, apparent specific electric resistances (ρ_k), and a set of derivative parameters. The phase shift angle is studied during induced polarization measurements on the alternating current and is similar to polarizability which is studied during induced polarization measurements on the alternating current. Graph and isoline plans are built for the sites. The survey error is calculated.

Results

Since high conductivity and high polarizability of carbonized sulphidized rocks significantly influences the results of the phase shift angle determination, we used the so-called double frequency polarizability for analysis, recommended by many researchers. It allows to significantly avoid the impact of inductance preconditioned by the particularities of the geological structure of the area.

The main particularities of the polarizability field are actually preconditioned by the linear zones of the northeastern and the northwestern strikes, occurring in the resistance field as well. At that, the first of them are clearly seen on the background relief map, and the second ones are seen on the parameter isoline plan and on the map of anomalies automatic tracing results. By the way, this map clearly shows the anomalies preconditioned by surface inhomogenuities (results of industrial development of watercources).

At the same time, three anomalous zones are defined with certainty on the polarizability dispersion map. One of them – the largest and most intensive – is unambiguously associated with surface inhomogenuities. And the other two – western and eastern – correlate well with the anomalies identified in the magnetic field and in the resistance field. This makes them very interesting from the prospecting standpoint.

8.1.5. Electric Survey with Natural Electric Field Method. Methodology and Results

Methodology

Electric survey with the natural electric field method was used for solution of the following tasks:

- for identification of sulphidization and graphitization zones;
- for mapping of metasomatosis development zones.

Electric survey with the natural electric field method was executed with a combined method of potential – potential gradient, which was preconditioned by long profiles. The purpose of these studies was to identify electric conductors capable of forming natural electric fields.

Measurements were executed with an immobile electrode N with the help of a coil 200 m long marked every 20 m. Then the electrode N was moved to the last metered stake, and the survey was executed with the same methodology. The potential drop was measured with a digital all-purpose meter MASTECH MAS-345 No 20030808988 with input resistance above 10 MOhm. Standard non-polarizable electrodes of Polyakov's design filled with a concentrated copper sulphate solution were used as metering electrodes. Electrodes were prepared for work in accordance with the standard procedure, as per the Electric Survey Instruction, 1984. Prior to works, they were wetted for 2 days. Then they were filled with a concentrated copper sulphate solution prepared on the basis of distilled water. All electrodes were filled from the same copper sulphate portion. Prior to the beginning of a working day, a pair of electrodes with minimum and constant in time values of own polarization was selected.

A GPSMP wire was used for unit assemblage. The work was conducted in accordance with the norms (Electric Survey Instruction, 1984).

The field was georeferenced on the basis of main lines. The survey along the main line was executed with obligatory matching of main line and profile stakes.

During office treatment all received values were re-estimated to potential values, on the basis of which natural electric field potential graphs were built for profiles, and the regular survey error was calculated. Isoline plans and natural electric field graph plans were also built. Allowances were made for all estimated and referenced values of the field. The necessity of their introduction is associated with the fact that the natural electric field monitoring is not absolute but relative, and, according to the theory, the positive values of the natural electric field potential cannot exceed + 50 ÷ + 60 mV.

Results

It should be noted that the structure of the field received, especially in details, differs significantly from historical materials. The survey scale is the only reason for that.

Three zones with different morphology of the natural electric field are clearly defined within the site. In this regard, the western and the eastern (evidently) parts are characterized by very similar structure of the field, i.e. presence of intensive local anomalies of the west-north-western strike is typical for them. The natural electric field is flattened in the central part. It shows the same structures as in adjoining zones, their continuation to be exact. But the field intensity is significantly lower. This actually reflects the particularities described above for the magnetic field characteristics.

This evidently speaks of the fact that in the central part of the site we either have a block of different composition, or, which is more likely, the zone processed at later stages. The boundaries of this zone with the eastern and the western parts are evidently tectonic.

The observed anomalies of the natural electric field are characterized by very high intensity (up to -1000 mV and above). The anomalies of this intensity were earlier observed by us on the Olimpiada deposit and corresponded to the zones of intensive carbonization with hydrothermal sulphidization. The identified local anomalies agree quite well with the other physical fields.

One can generally speak of four such local anomalies – two in the west and two in the east. Only one – southern zone is more or less studied within the site, of the western zones. But its western boundary goes beyond the site. And the northern zone goes along the contour of the site and is evidently understudied. Low-intensive local anomalies of the magnetic field are also present there.

The eastern zones occur at the contours of the site only (southern and eastern), which does not allow to characterize them. Their presence can be only stated.

The tectonics in the natural electric field is reflected weakly. Faults of the northeastern strike can be more or less reliably visualized, and faults of the north-western and the west-north-western strike are less reliable.

8.1.6. Electric Survey Topography with Induced Polarization Method. Methodology and Results

Methodology

Electric tomography with the induced polarization method was used for solution of the following tasks:

- define presence or absence of sulphidized and well-conducting zones within profiles defined by the customer;
- characterize the particularities of the behaviour of sulphidization zones and conducting zones in the depth;
- correlate the electric tomography data with the available geological-geophysical data, with forecast formation.

The work was conducted with a high-efficiently 10-channel multi-electrode electric survey station Syscal-Pro Switch 72 of Iris Instruments production.

The built-in generator has the following characteristics: power - 250 W, maximum current intensity - 2.5 A; maximum output voltage – 800 V. Signal form – meander with the operating mode without induced polarization change and meander with a pause for metering the induced polarization drop curve. The typical accuracy of metering the passing current is 0.2%.

The built-in meter has the following characteristics: input resistance – 100 MOhm, accuracy of voltage metering – 0.2%, minimum metered voltage – 1 mkV, automatic compensation of linear drifting of intra-electrode polarization.

The meter allows to conduct measurements for 10 receiving dipoles simultaneously. This allows to achieve the speed of field work up to 200 measurements per minute. The memory allows to save 21,000 measurements at once.

The Syscal-Pro Switch 72 equipment allows to use electric survey strings for 72 electrodes in work. All electrodes can be used as receiving or supplying. A built-in switcher switches electrodes in accordance with the measurement protocol prepared in advance.

The work is executed in accordance with the requirements of the Electric Survey Instruction, on the basis of the multi-electrode sounding flowchart (electric tomography). This allowed to receive qualitative materials for interpretation within a 2D geological environment model.

Field work was executed in accordance with the Terms of Reference under contract, and also in accordance with instructive and methodological requirements to the applied equipment and technology. The term of field work was 14-31 July 2011.

An electric survey string for 72 electrodes with the space 5 m was used for work. The length of one arrangement was 355 m. All profiles were worked in several stages, with arrangement overlapping by 180 m (half of the arrangement length). A combined three-electrode unit Schlumberge (AMN+MNB) was applied. One of the supplying electrodes was moved to ‘eternity’, at the distance of at least 570 m perpendicular to the profile line.

Table 8.4 shows the grid of the spans used.

Table 8-4 Unit parameters for electric tomography survey

MN, m	5	5	5	5	5	5	5	5	25	25
AO, m	7.5	12.5	17.5	22.5	27.5	32.5	37.5	42.5	42.5	52.5
MN, m	25	25	25	25	75	75	75	75	75	75
AO, m	62.5	82.5	102.5	122.5	122.5	152.5	182.5	207.5	233.5	

The span interval used provides for the study of the geoelectrical section to the depth of approximately 85 m. Surface deposits to the depth of 1.5-3 m occur as a single layer of averaged resistance.

Transient resistance at the work site varied from 1 to 50 kOhm. The output voltage of the generator made 400 V. This allowed to form currents from 10 to 400 mA in the supplying line. The values of signals on receiving lines made 0.03 mV to 10 V. On some profiles the rocks with specific resistance below 100 Ohm.m (graphitized slates) were observed. This led to the amplitude of the metered signal below 0.03 mV, which the devices used could not measure reliably. The minimum signals for determination of the induced polarization parameters shall be above 0.1 mV, therefore, the induced polarization parameters could not be reliably defined on all profiles, especially in well-conducting zones.

The impulse length (minimum in terms of the time of a part of signal for one sign) and the pause made 1000 ms. Four impulses are an accumulation for each measurement.

In total, 6 profiles were worked. The total length of profiles is 3210 m. The lengths were defined by the string length and were metered along the terrain. The planned profile georeference was conducted with the help of GPS of Garmin 60Scx model. A built-in GPS altimeter was used for determining the relative difference in height along the profile.

The monitoring density made on average 6000 measurements per line kilometer of the profile.

Results

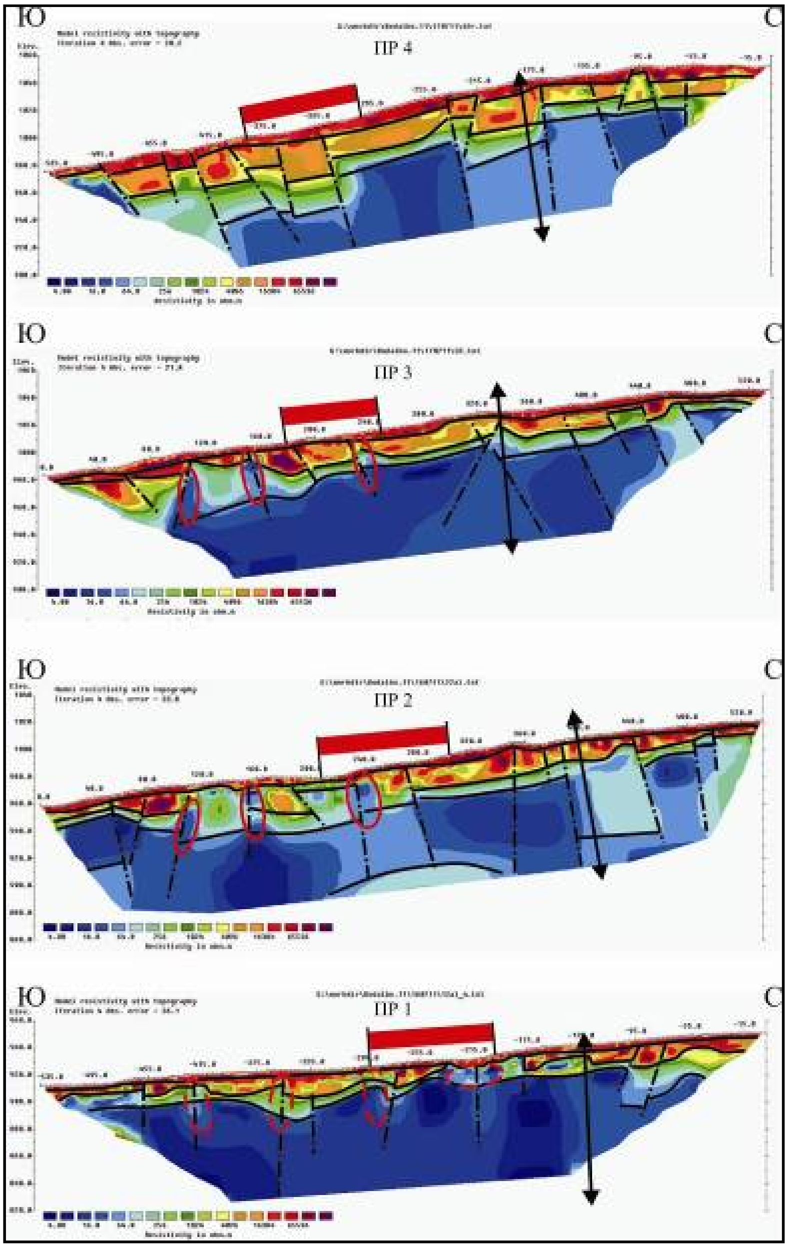
In total, electric tomography within the Krasny site was conducted on 6 profiles. Two of them (Profiles 6 and 7) were located in the northern part of the area, within the contour of surface geophysical works of 1:10,000 scale, and Profiles 1-4 are located in the south, within the Krasnoye ore occurrence.

Profile 7 was positioned along Trench 408, and Profile 6 – to the east, at the continuation of structures intersected in a trench. The analysis of the position of profiles in physical fields shows that they are located at the eastern continuation of structures potentially interesting from the ore content standpoint. In this regard, the structure intersected in the southern part of the trench and, respectively, in the southern part of profiles is most interesting. But it is weak within the profiles. And the northern zone, for the study of which the trench was driven, is hardly of any interest from the prospecting standpoint, because a combination of fields that do not follow the model ideas on the ore unit corresponds to it (in particular, it is characterized by more intensive magnetic field). Moreover, it is evident that the profiles are located in the conditions of a quite tense tectonic situation at the intersection of faults of different systems.

The results of the joint analysis of tomographic data with the observed physical field graphs (electrical ones primarily) shows their very satisfactory convergence (specifically when taking into account the difference in survey scales). The correlation scheme prepared for Profiles 6 and 7 speaks of very good periodicity of structures identified on the basis of resistance contrast on Profiles 6 and 7. This conclusion coincides with the ones made above – that both profiles, according to surface work results, are within the same structures. At the same time, local zones of reduced resistance occur within Profile 6, and this corresponds to the observed electrical fields quite well.

The structure identified on the basis of tomography results, represents (with regard to the antecedent geological information), the southern limb of a steep anticline fold, the axis of which is located in the northern part of profiles. The rocks composing this limb are characterized by abruptly reduced resistance, while the hinge part of the fold is characterized by increased resistance. The gold limb is complicated by folds of a higher order.

Figure 8-1 Results of interpretation of electric tomography data for Profiles 1, 2, 3 and 4 within the Krasny site



1 – geological boundaries assumed on the basis of geophysical data; 2 – tectonic faults assumed on the basis of resistance change; 3 – tectonic faults according to geological data; 4 –tectonic faults assumed on the basis of polarizability; 5 – local anomalous zones (in terms of resistance); 6 – local anomalous zones (in terms of resistance), not reliable; 7 – local polarizing units; 8 – ore zone projection to the day surface (according to geological data); 9 – boundaries of dispersed sulphidization zones; 10 – intensive polarizability zones (intensive graphitization and sulphidization); 11 – anticline fold axes defined reliably; 12 – anticline fold axes defined unreliably.

Since schistosity hardly changes rock resistance, and horizons are quite thick and their boundaries go beyond the profiles, it is very hard to estimate the dip angle correctly. However, the presence of near-vertical boundaries in the section allows to assess it as steep. A lot of tectonic faults identified on the basis of geophysical data shall be noted as well. This also agrees with the results of analysis of observed physical fields.

A local zone of reduced resistance is identified within Profile 6, in the area of Stake 575. It is most probably associated with a tectonic fault. The southern anomalous zone is also identified on the basis of surface work results. It allows to assess this anomalous zone as potential for gold mineralization detection.

The data interpretation for Profiles 1-4 is complicated with two main reasons – first, the profiles did not go beyond the structures. Thus, it is quite hard to characterize them and also define the position of the hinge unambiguously. Extremely low rock resistance is another interfering factor which significant reduces the resolution capability of the method.

Nonetheless, taking into account the antecedent geological information, one can speak of the fact that here a large anticline fold is developed (mainly its southern limb). The fold is complicated by very intensive fault tectonics with imposed hydrothermal alterations and evidently higher-order folds. The fold hinge most likely changes its strike and dip angle. The potential ore zone is located to the south from the hinge and mostly repeats its behaviour. In the section it appears in the form of a group of local anomalies with reduced resistance, confined to tectonic faults. It is most contrast on Profiles 2-3 and less contrast in Profiles 1 and 4, while it is hardly seen on Profile 1.

In 2012 pilot work on pole-dipole sounding was conducted. The work provided for the use of the methodology of monitoring with a two-sided 3-elecrode unit. The forth electrode – ‘eternity’ should have been moved at the distance of 4-5 km from the monitoring profile (combined multi-span electric profiling). During pilot work non-polarizing electrodes were used at the receiving dipole; groups of 20 steel electrodes and aluminum foil sheets with the area up to 10 sq.m were used as supplying electrodes A and B. A GPSMPO wire was used in the receiving line and the electrode A line, and a copper wire GPMP was used for an eternity line laying. The use of such structure of the supplying line allows to freely vary the current density without generator overload. The methodology of measurements on all profiles provided that the monitoring space and increase of spans are equal to the size of the receiving line. The initial span equaled to the size of the receiving line – 50 m as well. Measurements were conducted on 10-12 spans with the space 50 m. The effective length of the sounding profile made 1000 m.

Measurements were executed with a digital impulse device AIE-3 manufactured in 2011 with a 1 kW generator with the maximum load current 2 A. Measurements were conducted with 4-20 accumulations, with the charge time intervals 2 to 32 sec.

Pilot work was conducted on Profile 5, in the lines of boreholes 401,402,403,404,412,413 and 414. The total length of the geophysical profile made 3 line km.

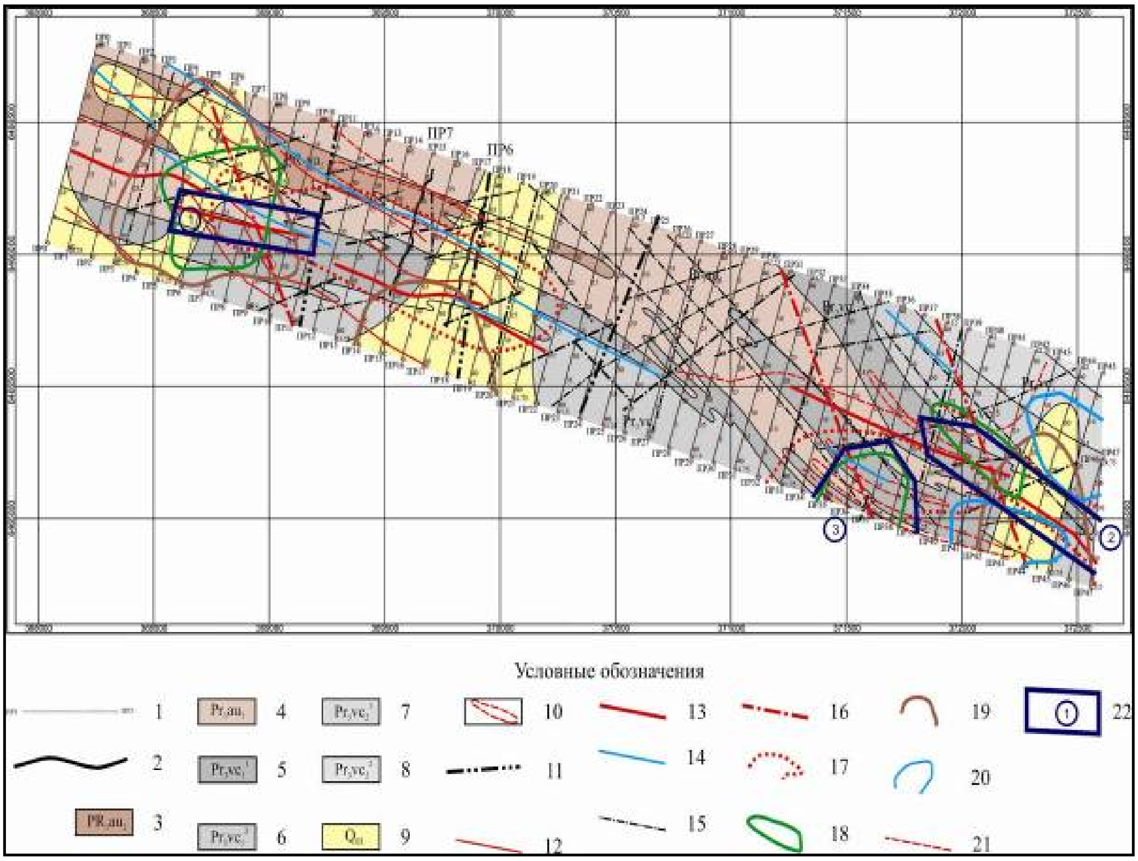
At the first stage, the minimum permissible level of the signal received by the meter was determined with wholly laid receiving and supplying lines. Unfortunately, the equipment did not register a stable signal. The indicator variety during repeated measurements reached 500 per cent and above, or the equipment failed to measure. The maximum current in Line AB did not exceed 150 MA, with the voltage up to 900 V.

In these conditions measures for changing unit parameters were taken. During their implementation the maximum sounding depth of 295 m was reached, but in this case only the apparent resistance parameter could be measured. The reproducibility of measurement results (even within 10%) was insufficient. During monitoring it was found out that only alluvial-delivual deposits are steadily and accurately registered in the conditions of the geological section of the Krasny site. The underlying rocks are characterized by the resistance of approximately 10 Ohm.m, falling to almost undetectable values (0.1 Ohm.m) due to high conductivity – above 200 siemens. The degree of rock polarizability could not be measured due to the presence of thick (up to 360 MV) high-frequency telluric noises, the trains of which blocked the useful signal. During pilot work the impossibility of reaching the purposes declared in the Terms of Reference, with the use of applied equipment, was defined unambiguously. Operating replacement of the hardware complex is impossible due to remoteness of the enterprise base from the work site. Pilot work was executed between May 30 and June 06, 2012.

8.1.7. Results of Complex Geological and Geophysical Interpretation of Surface Geophysical Data

It shall be noted that a very stretched shape of the site with a minor length of profiles predetermined some specific complexities for the usage of statistic methods for physical field analysis. Some weird shape of anomalies within individual transformant fields can be explained by that.

Figure 8-2 Results of complex geological and geophysical interpretation of surface geophysical work data for the Krasny site



Legend to Figure 3.3: 1 – design location of monitoring profiles; 2 – design location of tomography profiles; 3 – deposits of the middle subsuite of the Aunakit suite according to geological data; 4 - deposits of the upper subsuite of the Aunakit suite according to geological data; 5 - deposits of the first horizon of the lower subsuite of the Vacha suite according to geological data; 6 - deposits of the second horizon of the lower subsuite of the Vacha suite according to geological data; 7 - deposits of the first horizon of the upper subsuite of the Vacha suite according to geological data; 8 - deposits of the second horizon of the upper subsuite of the Vacha suite according to geological data; 9 – Upper Quaternary deposits according to geological data; 10 – gold-bearing zones according to geological data; 11 – tectonic faults identified on the basis of geological data; 12 – axes of linear negative anomalies of the magnetic field; 13 – axis of a large linear anomaly of the magnetic field; 14 – axes of linear anomalies of the natural field; 15 – tectonic faults identified on the basis of geophysical data; 16 – tectonic faults of the NNW strike; 17 – anomalous zones of the magnetic field according to statistical analysis results; 18 - anomalous zones of the resistance field according to statistical analysis results; 19 - anomalous zones of polarizability according to statistical analysis results; 20 - anomalous zones of the natural electrical field according to statistical analysis results; 21 – assumed boundary of the northern zone of the reduced magnetic field; 22 – recommended sites and their numbers

The purpose of the work was to study the geological structure and to assess the potential of a relatively narrow stretched zone located to the north from the Krasnoye ore occurrence within the license site for gold mineralization.

Several gold halos are known within the work area; a number of trenches were driven, within which several zones were identified, that can be characterized as mineralization points.

The work site is located within the areas of the Aunakit and the Vacha deposits development. As compared to the earlier studied sites, no geological boundaries of any suites or subsuites are identified within physical fields. Possible reasons for that:

- low contrast of deposits due to their physical properties;
- insufficient degree of geological study, as a result of which the boundaries are drawn conventionally and not always coincide with real ones;
- wide development of imposed processes with no clearly expressed lithological control.

In this regard the behaviour of local negative anomalies of the magnetic field, which the gold mineralization in the Bodaibo District is associated with, is interesting. It can’t be stated that they are confined to any stratigraphic horizon, unlike the other sites. They crosscut lithological boundaries and are located in different suites and subsuites (according to geological data). Such behaviour can be actually adequately described by the reasons above. Primarily the impact of hydrothermal processes developed on permeable tectonic structures can be stated. Also, the non-correspondence of the northern major negative zone of the magnetic field to any stratigraphic subdivision is evident.

A significant number of tectonic faults referred to different systems are evident within the site. They occur in almost all fields. The natural electric field is an exception. The faults of only north-eastern strike are more or less visualized in it, corresponding to the faults identified on the basis of geological data. The reason for that is the nature of the field – the natural electric field is potential (like the magnetic field), and the superimposition principle is typical for it. This means that the anomalies from local units are summed up, transforming to more intensive and flatter ones. But, as compared to the magnetic field, the natural field is characterized by a significantly lower resolution capacity. This leads to the fact that major tectonic faults changing the position of electronic conductors and located across the strike of major structures are well mapped in it, unlike the others.

In the other fields (especially on background relief maps), tectonic faults of NW, NNW and NE strikes are clearly seen. From the ore process standpoint, the most interesting are the sites of intersection of NW faults (red in the figure) mapped well in electric fields, and local anomalies of the negative magnetic field, in case of intensive natural electric field anomalies present, speaking of graphitization and hydrothermal sulphides.

Taking into account these criteria, three sites most potential from the mineralization standpoint were identified within the work area.

Site 1 is most interesting. It is located in the west of the area. Here, a combination of all above-listed abnormality criteria takes place. It is also identified on the basis of statistic analysis results. A gold halo was also detected on the basis of earlier conducted work. Placers of the Teply creek begin to the south. All these factors make the site interesting for mining confirmation.

We identified two more potential sites in the east. They are both located in favourable geological and geophysical conditions, but the problem is that they both pass along the contour of the site, go beyond it and are not characterized in full. The appropriateness of their identification is partially confirmed by the presence of gold halos and placers. However, their reliable identification (especially of their boundaries) in these conditions is impossible without additional information. Nonetheless, they are also recommended for mining confirmation.

8.1.8. Main Conclusions and Recommendations

Conclusions Made on the Basis of Work Results

1. The presence of geophysical zoning is the main prospecting indicator of gold mineralization. It consists in regular positioning of magnetic field, resistance field, natural electric field and polarizability anomalies in relation to each other. At that, the gold mineralization is confined to negative magnetic field zones. This is preconditioned by a negative angle of the hydrothermal pyrrhotite remnant magnetic induction vector inclination.

2. The Krasny site is characterized by quite tense structural and tectonic settings, while not all faults are similarly expressed in different physical fields.

3. Three sites potential for gold mineralization are identified within the Krasny site. Two of them are characterized incompletely and go beyond the work area contours.

4. The section structure is studied on the basis of the induced polarization electric tomography profiles.

The analysis of received results allows to give and confirm earlier given following recommendations:

1. The earlier formed geophysical optimal work complex within the research area for 1:10,000 scale is confirmed – magnetic exploration, induced polarization profiling, profile induced polarization and natural electric field tomography. This complex is recommended for further detailing work.

2. It might be feasible to carry out detailing of 1:2,000 scale with the same complex, with adding separate short profiles of electric survey tomography within detailing sites identified on

the basis of 1:10,000 scale work results. However, their development requires additional economic substantiation in each specific case.

3. It is feasible to carry out the following work on the sites with increased thickness of loose formations:

- areal magnetic survey;
- profile induced polarization tomography on some profiles.

The other methods will be either inconclusive due to great thickness of loose formations (gamma spectrometry, natural electric field, induced polarization profiling), or it will be impossible to carry out them correctly in terms of methodology, and they will not be able to solve the tasks set (electromagnetic sounding, VES-IP).

4. The problem of formation of petrophysic provision deserves special attention. We think that this necessity is preconditioned by the following. With comparatively low costs, the re-interpretation of the available and newly received geophysical materials on the basis of the received petrophysical model will allow to assess (or re-assess) the potential of work sites and find the criteria for detecting blind mineralization more correctly.

From the other side, the physical rock properties are very sensitive to hydrothermal metasomatic alterations, and changes in physical properties very frequently serve as clear indicators of these processes, while these changes cover a significantly larger area than mineral transformations. Therefore, the formation of petrophysic provision will allow to receive a more detail pattern of metasomatic alterations within exploration sites.

Finally, this data will allow to interpret GIS data more correctly, and some of it can be used in the future for deposit development design.

8.2. Trenching

The purposes of exploration trenching and historical trench clearing were to map the upper part of the ore zone, to specify the structures of the Krasny ore occurrence and to receive data on mineralization parameters. Trenching was conducted with a mechanized method, with the help of an excavator KOMATSU PC200 up to the bedrock, with extraction of a crushed and weathered part. The average depth of mechanized trench clearing made 2 m; length – 90-166.2 m. The finishing clearing of trench slopes and bottom was executed manually with broom and brush in order to avoid sample contamination.

The total scope of trenching is given in Table 8.5.

Table 8-5 Scope of trenching and sampling

No	Trench No	Bottom Length, m	No of Channel Samples, 10×5 cm in Section	No of Channel Samples, 3×5 cm in Section	No of Chip Samples	Bottom Logged, m
1	143501	90.00	-	99	-	90.00
2	143502	166.20	-	184	-	166.20
3	143503	154.00	17	151	10	154.00
4	143504	152.00	-	162	8	152.00
5	143505	125.50	-	135	-	125.50
6	143506	162.40	-	172	-	162.40
7	143507	158.00	-	168	-	158.00
8	143508	159.70	-	175	-	159.70
Total	8	1167.80	17	1246	18	1167.80

8.3. Core Drilling

Core drilling within the Krasny site was conducted with the purpose of tracing the ore zone to the depth, studying its morphology and material composition, specifying the geological structure of the ore occurrence and receiving data on the mineralization parameters.

Drilling was conducted by contractors LLC. Mama Exploration Team with drills SKB-5114, SKB-41 and LLC. Prikladnaya Geologia with drills SKB-5, SKB-51. The diameter of boreholes in loose technogenic and alluvial-diluvial deposits was 151, 132, 114 mm, and diameter in bedrock – 76 mm (single core pipe) and 95 mm (double core pipe). Core diameter in bedrock – 62.3 mm, core recovery in bedrock as per the Terms of Reference – at least 90%. The actual core recovery in bedrock made 98.2%.

The total scope of drilling for the period from 08.09.2011 to 25.09.2012 made 12802.70 m.

Table 8-6 Scope of core drilling and sampling

No	Borehole No	Depth, m	Core Samples for Fire Assay, pcs.	Samples for Gold Spectrometry and Spectral Analysis, pcs.
1	141401	301.35	300	300
2	141402	163.80	161	161
3	141403	290.55	288	288
4	141404	106.90	104	104
5	141405	300.00	300	300
6	141406	100.00	99	99
7	141407	100.00	97	97
8	141408	100.00	99	99
9	141409	99.70	97	97
10	141410	100.00	95	95
11	141411	301.20	298	0
12	141412	200.50	195	0
13	141413	408.00	411	0
14	141414	450.00	183	147
15	141415	450.40	206	126
16	141416	349.80	129	116
17	141417	403.00	142	137
18	141418	400.00	162	124
19	141419	427.00	303	96
20	141420	359.45	190	89
21	141421	340.90	250	69
22	141422	350.20	234	87

23	141423	381.60	226	93
24	141424	416.00	207	129
25	141425	200.75	108	64
26	141426	70.00	35	21
27	141427	141.95	108	24
28	141428	157.00	73	46
29	141429	105.60	53	30
30	141430	179.40	95	52
31	141431	84.40	42	25
32	141432	180.70	142	26
33	141433	95.50	95	0
34	141434	190.00	114	50
35	141435	86.00	62	15
36	141436	203.50	117	50
37	141437	381.00	198	115
38	141438	250.00	210	27
39	141439	300.20	269	27
40	141440	200.20	169	19
41	141441	403.40	416	0
42	141442	350.00	319	28
43	141443	300.20	319	0
44	141444	364.80	362	14
45	141445	351.70	266	51
46	141450	350.00	353	0
47	141460	250.00	246	0
48	141461	256.00	215	17
49	141462	225.80	220	0
50	141463	224.25	213	0
TOTAL for 50 boreholes		**12802.70**	**9595**	**3554**

Core laying in wooden boxes laid with polyethylene, was executed from left to right. The laid core was measured with a tape-measure on the basis of drilling runs with the purpose of defining the actual core recover. The laid and measured core was photographed in a wet condition. Geological logging was executed in 1:50 scale.

8.4. Topographic-Geodetic and Survey Work

Engineering geodetic research for the project: “Tomography Survey of 1:2,000 Scale, Topography Section 1 m, Krasny Site” within the exploration site in the area of the Artemovsky Village in the Bodaibo District of the Irkutsk Region was executed under Contract 28 dated 28 August 2012 and the Terms of Reference with the purpose of creating the topographic plan in 1:2,000 scale, in the coordinates system WGS-84UTM and the Baltic System of Heights.

Competency certificate for engineering research No 0068.01-2011-3812105147-И-003, issued by NP Tsentrizyskaniya on October 06, 2011.

The work was conducted in the coordinates system WGS-84UTM and the Baltic System of Heights of 1977 in accordance with the Terms of Reference, following the requirements of “Construction Norms and Rules” (SNiP 11.02.96), SP 11-104-97, “Legend for Topographic Plans of 1:5,000 – 1:500 Scales, 2004 edition, and “Rules of Drawing Symbols on Topographic Plans of Underground Utilities of 1:5,000-1:500 Scales”, 1981 edition, and “Instructions for Developing Surveyor’ Pickup and the Situation and Topography Survey with Applying GPS and GLONASS”.

- Field work conducted by: V.P. Babaytsev during the period from 11.09 to 25.09. 2012.
- Office work conducted by: V.P. Babaytsev during the period from 25.09 to 10.10. 2012.

The main scopes and types of completed work are given in Table 8.7.

Table 8-7 Main scopes and types of topogeodetic work

No	Work Type	UoM	Work Scope	
			as per Terms of Reference	Actual
1	Gathering and systematization of historical materials	Mapboard	No	No
2	Finding of polygonometric points	Polygonometric points	1	1
3	Complex of engineering-geodetic research, Scale 1:2,000, Cat. 1, Section 1.0 m	ha	170	170
4	Preparation of a technical report	Report	1	1

8.4.1. Level of Topographic-Geodetic Study of the Work Area

The work site is located 12 km away from the Artemovsky Village, in the Bodaibo River of the Irkutsk Region. The survey area is 170 ha, the length of the survey site – 3000 m, and the width – 560 m. The work site stretches from the Teply Creek to the Krasny Creek and the valley of the Bodaibo River. The topography of the work site is mountain. The Teply, the Krasny Creeks and the Bodaibo River mineral resources are mined. Vegetation within the work site varies in density and height. In the valley of the Teply Creek and the Bodaibo River it is presented by mixed forest. Slope hills are covered with cedar elfin wood 2-4 meters high.

A reference geodetic grid in the form of the 5th class triangulation and the 4th class leveling is formed in the research area by the East Siberian AGP. Materials for polygonometry points were received in the Geodesy and Cartography Department of Rosreestr in the Irkutsk Region, in the Svobodnenskaya Coordinates System, Request No 301 DSP. The work site is covered by topographic maps of Scale 1: 100,000 and 1:200,000, prepared by different organizations in different years in the local coordinates system and the Pacific Ocean System of Heights. The coordinates and heights catalogue is kept in the archive in the Geodesy and Cartography Department of Rosreestr in the Irkutsk Region.

Exploration is conducted within the Krasny site. Trenches and boreholes were georeferenced in the WGS84UTM Coordinates System, therefore, the Terms of Reference defined fulfillment of research in this coordinates system from the Ust Teply point with WGS84UTM coordinates calculated by the customer.

8.4.2. Survey Geodetic Grid

The setting and topography survey was conducted with the use of the satellite technology. In this case during its fulfillment there is no need to form geodetic thickening grids or a surveyor' pickup and its thickening, since the methods of satellite determinations in terms of the distance and accuracy principally provide for the possibility of surveying right on the basis of the state geodetic and leveling grid, cl. 2.19 (Instruction for Developing Surveyor' Pickup and Setting and Topography Survey with the Use of GPS and GLONASS).

All new surveys are usually executed in the earlier assumed coordinates system (Instruction…) in towns, the areas of industrial complexes, on operating enterprises of the mining and oil producing industry. Therefore, the point of the state geodetic grid "Ust Teply" in the WGS84UTM Coordinates System was used as a surveyor' pickup point.

8.5. Topographic Survey

According to the Terms of Reference for Tacheometric survey, the point of the reference geodetic grid was searched for. The triangulation point of the 5th class "Ust Teply" served as the basic point.

Field work for setting and topography survey was executed in accordance with normative requirements relating to topographic survey in 1:2,000 scale, with the topography sectioning 1.0 m.

The work was conducted with the help of a double-frequency geodetic satellite meter LEICA GS09 Nos. 166527, 166320 in the WGS84 Coordinates System. The survey was executed with the stop and go method – statics – dynamics. The survey of mining areas at the Teply Creek was also executed with geodetic satellite equipment LEICA GS09, survey of inaccessible contours of technogenic topography (peat overburden), dumps, the upper edge of the pit – with the help of an electronic tacheometer Set 530R3-385L No 159979, from the surveyor' pickup points, the coordinates of which are defined with geodetic satellite equipment.

According to the Terms of Reference, the survey results and the topographic map of 1:2,000 scale were prepared in the WGS84UTM coordinates system.

Office processing of the field information was conducted: satellite measurements – in the Leica Geo Office 8.2 program, in the WGS84UTM coordinates system, and observations conducted with a tacheometer Set 530R3-385L – in the CREDO-DAT software (Minsk).

All elements of the existing settling and engineering structures reflected in the plan scale and provided for the specified scales with effective symbols are reflected on the topographic map in 1:2000 scale. The area topography is reflected with 1.0 m sectioning and with stakes with elevation.

The symbols are supplemented with explanatory signs on plans.

The topographic plan is drawn in the DEMO version of nanoCad, as per the requirements of "Symbols for Topographic Maps of 1:5,000 – 1:500 Scales", issued in 1989, and in accordance with the "Rules of Drawing Symbols on Topographic Plans of Underground Utilities of Scales 1:5,000 – 1:500", issued in 1981.

A topographic map of scale 1:2000, with horizontal sectioning 1.0 m, was prepared on the basis of field and office processing work results within the land plot boundaries. A report on the work completed within the Krasny site was executed.

The work was conducted in accordance with the Terms of Reference in full volume, as per the requirements of norms for work execution.

The work methodology and accuracy meet the requirements of the “Instruction for Topographic-Geodetic and Navigation Provision of Exploration, Novosibirsk, 1997.

8.6. Sampling

8.6.1. Trench Sampling

Channel samples were taken at a previously developed grid upon documentation of the trench along the sampling line. In the trench samples were taken with a machine-manual method, with a channel 10×5 cm and 3×5 cm in section, taking into account the lithology. The channel was taken with a continuous method, for the whole length of the trench. Upon extraction of the sample material, the sampling section was cleaned up by brushing or sweeping and the cleaned up material was added to the sample material. Equipment for channel sampling: DES-5, an angular grinding machine (grinder), an electric chisel, diamond disks, and rock-breaking chisels. The samples were packaged in double bags with a firm inner polyethylene layer.

Testing was conducted for determining the optimal section of a channel. An adjacent channel sample with the section 5x10 cm was taken in addition to the main channel with the section 3x5 cm from the same interval (in total, 17 samples were taken with the section 5x10 cm).

The order was supplemented with blanks formed of silicate brick for sample processing quality control. The operating internal control was provided with the help of standards which were included into a lot of samples sent for analysis to the main laboratory.

Chip sampling was executed on quartz veins, dynamic slates, and clay gouge in a trench slope. The weight of samples was at least 4 kg.

8.6.2. Borehole Sampling

Core boreholes were sampled after photo logging and detail geological core logging in bedrock; the covering alluvial-diluvial deposits were not sampled. Sampling intervals were defined with regard to geological boundaries and the core recovery from drilling runs. The runs with the core recovery difference above 5-10% were sampled independently.

The average length of sampling intervals in the ore zone was 1 m, and in the barren zone – 2 m. In 2011 the whole borehole core was sampled for increase of the sampling representatively, except for the specimens for mineralogical and petrographic studies. Lack of coarse gold in the composition of ores of the site was proved later, and starting from 2012 only a half of core was sampled.

The core was split on site with a core cutter SREZ No 9 with diamond disks 350 mm in diameter. The samples crushed till fragments of irregular shape, 7-10 cm in size, were packaged into double bags (internal – polyethylene, and external – jute). The average weight of a sample in the ore zone made 4.0 kg, and in the barren zone – 8.0 kg. The taken samples formed registers and were sent to Bodaibo, to the sample preparation shop of LLC. Kopylovsky.

Blanks were added for sampling processing control to final registers for the sample preparation shop. They were made of silicate brick. Each 20th sample was a blank on average. The operating internal control was provided with the help of standards which were included into a lot of samples sent for analysis to the main laboratory. Each 45th sample was a standard on average.

8.6.3. Geochemical Sampling

In the boreholes which were not exposed to core sampling due to lack of visible mineralization, chip samples were taken on average from a 2 m interval, with the weight about 0.3 kg. These samples after preparation were sent to spectral and gold spectrometry analyses. Analyses were executed in the State Enterprise “Republican Analytical Center” in Ulan-Ude.

The semi-quantitative spectral analysis was executed for 18 elements (lead (Pb), copper (Cu), zinc (Zn), tin (Sn), silver (Ag), arsenic (As), antimony (Sb), aluminum (Al), tellurium (Te), molybdenum (Mo), barium (Ba), chrome (Cr), nickel (Ni), iron (Fe), manganese (Mn), bismuth (Bi), yttrium (Y), vanadium (V)) and a chemical spectral analysis for gold.

8.6.4. Sampling Quality Control

During exploration the sampling quality control was continuous.

In trenches, on average each 20th channel sample was duplicated with an adjacent channel sample of the same length and section 3x5 cm. 60 such samples were taken, which made 5% from the total number of 1186 samples. The comparative analysis of results on RoM and control samples showed good compatibility.

In boreholes, on average each 20th core sample was duplicated with a core sample taken from the second half of the core. 460 such samples were taken, which made 5% from the total number of 9135 samples. The comparative analysis of results on RoM and control samples showed good compatibility as well.

8.7. Sample Preparation

Samples crushed till the size 5x5 cm were delivered from the site to the sample preparation shop of LLC. Kopylovsky in polypropylene bags with polyethylene lining; a label with a number was added to the sample. The weight of a core sample was 3 to 5 kg, and the weight of a channel sample - from 3 to 5 kg (section 3x5 cm), and from 10.8 to 16.2 kg (Section 5x10 cm). Geological samples were stockpiled in the sample preparation shop under the shop manager's control.

Samples were delivered from the site to the shop together with a paper register (the electronic version was duplicated to the shop manager's e-mail).

The initial sample after registration was placed in an electric furnace (2 furnaces with the capacity 9 kW and 10 heating coils with the capacity 2 kW are operating). Type of furnace – Makar TV-9, time of drying – 4 to 8 hours. After sampling the samples were weighed on platform scales Acculab SVI 100/20.

After weighing, the sample is delivered to the crushing department where it is crushed in two stages:

Crushing in a jaw crusher ROSKLABS Boid Mark-3, with the clearance between crushing plates 5-7 mm.

1. Crushing in a roll crusher of Mechanobr production, machinery DG 200*125, with the clearance between rolls 1 mm.

After each crushing (repetition) stage, the material is sieved into classes +1 and -1 mm in order to reduce the percent of overgrinding. At that, Class +1 mm is ground at each following crushing (repetition) stage. A weighted portion 1 kg is taken from the material of a crushed sample of -1 mm class with a quartering method. Thorough mixing is executed during weight portion taking.

The weighted portion taken with the weight 1 kg is ground till the size -0.071 mm with two methods:

- on a vibrating grinder IV-4. At that, a weighted portion 200 g is taken from the ground sample material with a checkered method (for fire assay), or 150 g are taken for gold spectrometry and 50 g are taken for X-ray fluorescent analysis. Weighted portions are taken with the help of scales CAS MW-II 3000
- on a disk grinder of the mill ROSKLABS CRM Mark – 8, where a percent splitter is installed, for example 15% - 150 g. This grinder takes the weighted portion independently.

All data related to registration, crushing and grinding of samples is registered in the shop log in strict correspondence with the columns.

In the process of geological sample preparation, the head of the shop fills in the registers in an electronic format – enters all data from the sample log (laboratory No, sample weight, weight of a ground sample: 200 or 150 g depending on the analysis). Each 50^{th} sample of those sent for analysis is a standard. LLC. Krasny uses as an enterprise standard the 'composition of the gold-bearing ore' bought in OJSC Irgiredmet in Irkutsk. During work two types of standards were used: composition AuBL-ИАЦ-09 with the Au grade <0.005 g/t and composition ЗСР ИАЦ-10-11 with the Au grade 0.5±0.03 g/t.

The prepared samples are packaged into paper envelopes and, as per the register, are placed into polyethylene white bags and fixed with a strainer, with sealing. For the X-ray fluorescent analysis (50 g) the samples are packaged into gripper bags and are transported for analysis to the laboratory assistant in LLC. Krasny. The sample residue with the weight approximately 750-800 g is packaged into paper envelopes, is placed into polypropylene white bags, as per the register; the bag is tightened with a string, and a register No is indicated on each bag. Duplicates are kept in the storage and if necessary a weighted portion is taken and sent for control or other types of analyses (as per the instruction of the chief geologist).

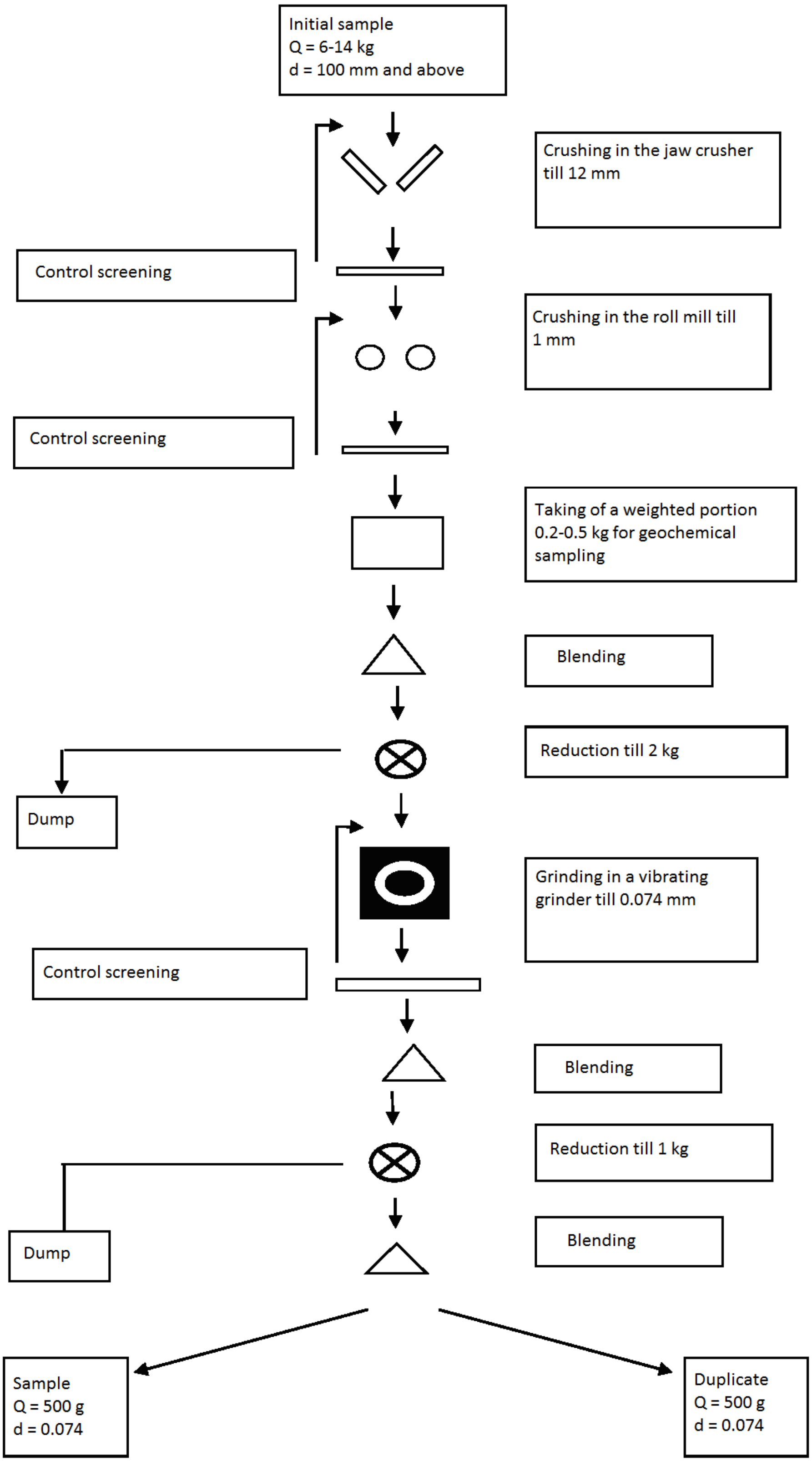
Sample transportation is executed under contracts signed with the following transport companies: LLC. Izumrud and LLC. Trans Logistic Service". Under contract signed with Izumrud, samples are transported by air from Bodaibo to Irkutsk. After that, under contract signed with LLC. Trans Logistic Service, the cargo delivered to Irkutsk is received by the representative of this transport company. LLC. Trans Logistic Service, on the basis of requests sent by e-mail, distributes the cargo by destination points. The analytical laboratory sample is delivered from Irkutsk by air, railway or motor transport, depending on its location.

The results for the samples are delivered in the form of protocols in an electronic format, and the originals are sent by post.

After receiving the results of analyses of samples, the exploration engineer enters the results to the data base and sends them to geologists for further data use and processing.

All sample movements are registered in the sample movement table.

Figure 8-3 Functional hardware sample preparation flowchart



8.7.1. Sampling Quality Control

All equipment used is sterilized by means of compressed air blowing after crushing and grinding of each sample. Chamotte is additionally supplied through the grinder ROSKLABS CRM Mark-8.

The head of the shop controls the grinding quality for each tenth sample by means of taking 100 g from the duplicate and sieving it during 5 minutes. After that the residue is weighed. The passage through the sieve shall make at least 85% of the material.

Blanks were added for sampling processing control to final registers for the sample preparation shop. They were made of silicate brick. Each 40th sample was a blank on average.

In general, the results testify to a good quality of sample processing. Of 297 blank samples, 20 samples showed the values above the sensitivity threshold 0.001 g/t, including 17 samples showing hundredth fractions of g/t. Only 5 samples of 297 (which is 1.7%) showed the grades above 0.1 g/t.

8.8. Process Sampling

In order to define the process properties of the ore and develop the processing technology, Sample No 1 made of ¼ of core was taken from the lower ore zone of the ore occurrence and sent for process testing.

The list of boreholes and sampling intervals with the average grades is given in Table 8.8.

Table 8-8 Boreholes and sampling intervals in them, taken into process sample No 1

Sampling Method	Borehole No	Sampling Interval, m			Average Au Grade, g/t
		from	to	total	
Core 1/4	141416	300.0	308.0	8.0	2.484
Core 1/4	141419	295.0	302.0	7.0	3.397
Core 1/4	141420	277.0	298.0	21.0	2.689

The actual weight of the sample made 60.9 kg. The weighted average grade was 2.781 g/t.

Process sample No 1 was sent for studies to OJSC Irgiredmet.

8.9. Laboratory Operations

Analytical sample studies are executed by the following contractors:

LLC. Stewart Geochemical and Assay, Moscow, carries out geochemical analysis: it defines gold by means of fire assay / AA finishing (in 10 ml of solution, lower threshold of defined grades 0.01 ppm) (Code Au4), analysis of the quantitative content: screen analysis – classification with the following fraction analysis – 1 kg (Code Au9).

The state enterprise Republican Analytical Center in Ulan-Ude carries out semi-quantitative spectral analysis was executed for 18 elements (lead (Pb), copper (Cu), zinc (Zn), tin (Sn), silver (Ag), arsenic (As), antimony (Sb), aluminum (Al), tellurium (Te), molybdenum (Mo), barium (Ba), chrome (Cr), nickel (Ni), iron (Fe), manganese (Mn), bismuth (Bi), yttrium (Y), vanadium (V)) and a chemical spectral analysis for gold.

8.9.1. Fire Assay

In the practice of Russian and foreign gold-ore companies, fire assay results are most reliable and preferable for substantiation of the reserve estimation materials and for developing investment programs (Bankable TEO).

Samples: core (9595); channel (1246); chip (18) were analyzed for gold with the fire assay method in the laboratory LLC. Alex Stewart Geo Analytics. The sample preparation methodology adopted for the samples of the Krasny site area assumes gravitation processing. Each sample after processing was presented by two independent samples: Gravitation concentrate (~ 150 g) and a weighted portion from gravitation tails (200-250 g). Samples are submitted for fire assay, where the whole gravitation concentrate is used as a weighted portion but is preliminarily ground and dry-weighed, and a weighted portion 50 g is taken from gravitation tails. The gold grade in the initial sample is defined on the basis of metal balance in the concentrate and gravitation tails. Therefore, 4 weighted portions, 50 g each (three weighted portions of a concentrate with gravimetric finishing and one weighted portion of gravitation tails with atomic adsorption finishing) are required for the RoM sample.

The total number of samples made 9,595+1246+18=10,859.

8.9.2. Spectral and Gold Spectrometry Analyses

Gold spectrometry and spectral analyses of 3554 geochemical samples were conducted in order to study primary gold and associated element halos and to define geochemical zoning, determine the level of erosion truncation and the environmental conditions of the area.

All samples were analyzed for gold with the gold spectrometry method, with the sensitivity threshold 0.002 g/t, and for 18 elements with the spectral semi-quantitative method: Ni, V, Cr, Mo, Cu, Pb, Ag, Mn, As, Zn, Sn, Y, Ba, Al, Fe, Te, Bi, Sb.

8.9.3. X-Ray Fluorescent Analysis

LLC. Krasny carries out X-ray fluorescent analysis by own means with the help of an X-ray fluorescent spectrometer Innov-X Systems. A ground sample with the weight 50 g, packaged into a gripper bag, is analyzed. The analysis is conducted in the soil mode with a triple source (ray). The program chooses different voltage, current and filter for each ray, therefore, the best possible analytical characteristics are achieved for each group of elements. The time of measuring one ray is 15 minutes, and the total time of analysis is 45 minutes. The total scope of analyses completed is 10,859.

8.9.4. Mineralogical Analysis

The mineralogical description of concentrate (crushed) samples was carried out under contract with the Institute of Mineralogy of the Uralian Department of the Russian Academy of Sciences. A semi-quantitative full mineralogical analysis with detail gold description was executed.

In total, 4 crushed samples were studied (2 sample from each of the lower and the upper ore zone of the Krasnoye ore occurrence).

8.10. Petrographic Characteristics of Host Rocks

During forecast and prospecting in 2011-2012 the following rock types were identified in the structure of the Krasny license area: quartz sandstone, quartz siltstone, carboniferous-clayey slates (phyllites) and their mutual alternation. The mineral composition of host rocks of the Krasnoye ore occurrence is given in Table 8.9.

Table 8-9 Mineral composition of carboniferous rocks of the Krasny site according to X-Ray phase analysis

No	Specimen No	Rock	Mineral Composition
1	263	Quartzitic sandstone	Quartz, illite impurity
2	3150-1	Quartzitic sandstone	Quartz, illite impurity
3	141412904	Siltstone	Quartz, illite-2M1, paragonite, tourmaline (dravite), pyrite, carbonate (breunnerite)
4	141412905	Oligomictic sandstone	Quartz, illite-2M1, pyrite, tourmaline (dravite), rutile, albite traces (?)
5	141412906	Siltstone	Quartz, illite-2M1, pyrite, tourmaline (dravite), albite traces (?)
6	141412907	Siltstone	Quartz, illite-2M1, pyrite, tourmaline (dravite), albite traces (?)
7	141412908	Siltstone	Quartz, illite-2M1, pyrite, tourmaline (dravite), carbonate (breunnerite), albite traces (?)
8	141412909	Siltstone	Quartz, illite-2M1, pyrite, tourmaline (dravite), carbonate (breunnerite), albite traces (?)
9	141412910	Siltstone	Quartz, illite-2M1, pyrite, tourmaline (dravite), rutile, albite traces (?)
10	141413901	Quartz sandstone	Quartz, illite-2M1, paragonite, tourmaline (dravite), pyrite (?)
11	141413903	Siltstone	Quartz, carbonate (breunnerite, siderite), illite-2M1, paragonite, tourmaline (dravite)
12	141413905	Carboniferous-clayey slate	Quartz, pyrite, illite-2M1, tourmaline (?), rutile, albite traces (?)
13	141413910	Carboniferous-clayey slate	Quartz, carbonate (siderite, breunnerite), illite-2M1, paragonite, tourmaline (dravite)
14	141413911	Carboniferous-clayey slate	Quartz, carbonate (siderite, breunnerite), illite-2M1, paragonite, tourmaline (dravite)
15	141413912	Carboniferous-clayey slate	Quartz, illite-2M1, paragonite, carbonate (breunnerite), pyrite, tourmaline (dravite)
16	141413913	Siltstone	Quartz, illite-2M1, pyrite, tourmaline (dravite), albite traces (?)
17	141413915	Siltstone	Quartz, illite-2M1, carbonate (dolomite), tourmaline (dravite), rutile, albite traces (?)

Note: DRON-2, 4° - 60°, space: 0.02°, anode: Cu. Analyst T.M. Ryabukhina (Institute of Mineralogy of the Uralian Department of the Russian Academy of Sciences).

Therefore, the host rocks at the Krasny site are presented by quartz sandstone and siltstone with high content of carboniferous substance (approximately 10 vol. %), carboniferous-clayey slate bodies are subordinated. Siltstones are composed of fine-grained quartz, are characterized by high content of thin-needle rutile and tourmaline, illite, and rarely paragonite (defined with X-ray phase analysis).

8.10.1. Indicators of Metamorphism, Metasomatic and Hydrothermal Transformations

In general, quartz rocks of the Krasny site, according to microscopic, X-ray phase and thermal gravimetric analyses, are metamorphosed in the conditions of muscovite-chlorite subfacies of the greenschist facies. This is primarily confirmed by the results of thermal gravimetric analysis, according to which the beginning of the effect of organic substance burn-off falls on 550 – 560 ^oC, and maximum – on 680 ^oC. Also, a shoulder of approximately 750 – 780 ^oC is observed, which testifies to presence of two structural graphitoid varieties. Based on temperature effects, the presence of bituminoids of the anthraxolite and the shungite types can be assumed in the composition of the carboniferous substance of host rocks. The amount of carboniferous substance in siltstone of the Krasny site, according to the weight loss curve recalculation results, can reach 6 mass. %.

Lower organic substance burn-off temperatures are noted in some samples. The maximum effect values are observed at 590 ºC, the beginning of the effect at 490 ^oC, and the end of the effect at 670 ºC. The beginning of the effect at 490 ^oC testifies to low-grade rock metamorphism at the schist stage. Such organics burn-off temperatures were observed in the rocks of the Prodolny site. The split nature of the effect speaks to the presence of two graphitoid varieties in the rock. Another indirect proof for the rock metamorphism in the greenschist phase conditions is regular presence of an organized structural variety of illite-2M_1, and in some specimens – paragnite-1M on X-ray patterns of siltstone and carboniferous-clayey slates of the site. The presence of structurally organized stratified silicates testifies to quite high temperatures and duration of their formation.

According to petrography studies, the hydromica-quartz cement of sandstone generally replaces quartz fragments. Fragment regeneration can be observed, up to formation of quartz pockets with coarse-grained construction and primary structure shadows. If there is carbonate in the cement composition, or carbonate poikilocrystals are observed in the rock, then fragments are replaced more intensively.

A high content of tourmaline, both of fragmented origin (mainly in sandstone) and newly formed (thin-needle tourmaline in siltstone) is typical for the rocks of the Krasny site. The X-ray patterns of rocks show regular tourmaline reflections. Fragmented tourmaline is generally overgrown by regeneration margins, is sometimes corroded, and thin-needle tourmaline contains carboniferous substance inclusions. This indirectly testifies to metamorphism processes.

A clearly expressed puckered structure is typical for siltstones of the Krasny site. It is stressed by fine-crystalline small quartz penetrations. No puckered structure is noted in carboniferous-clayey slates of the site. Carbonate metacrystals (according to X-ray structural analysis results presented by breunnerite and rarely dolomite) contains carboniferous substance inclusions and rock structure shadows. No traces of carbonate grain rotation are observed. Ruptures are formed around carbonate and pyrite metacrystals. They are filled with carbonate or parallel-columnar quartz aggregate.

The host rocks of the Krasny site are metamorphosed in the conditions of the greenschist facies of the muscovite-chlorite subfacies; metasomatosis (carbonatization) and hydrothermal activity processes (associated with formation of veinlets, pockets and lenses of various composition) occur in the rocks.

8.11. Ore Mineralization

The ore mineralization of the Krasny site is mainly presented by pyrite and forms the following varieties. Fine-crystalline dispersed and layer-by-layer pyrite dissemination (interbed thickness makes 2 mm – 5 cm) and pocket or lens-shaped dissemination of small-grained pyrite aggregates in the quartz margin are observed in the composition of high-carboniferous siltstone; pyrite lens and pocket chains can be united into veinlets and penetrations. Large, reaching 5-10 cm in size, pyrite metacrystals with quartz margin, and dispersed fine- and middle-grained dissemination of cubic crystals are typical for siltstone, sandstone and slate. Veinlet mineralization can be also presented by pyrite in the composition of quartz and quartz-pyrite veinlets and penetrations. Moreover, quartz veins with large galena, chalcopyrite pockets and crystals and small fahlore inclusions are noted.

Therefore, the main ore mineral of the Krasny site is pyrite, and chalcopyrite, sphalerite, galena, fahlore as well as rutile, goethite and covelline are secondary ones. Accessory and rare minerals include gold as well as gersdorffite and reliably undiagnosed nickel minerals, probably millerite, copper and cobalt mineral carollite, and presumably argentopyrite or eskebornite.

The chemical composition is studied only for gold-bearing porous fractured pyrite (Table 8.10). Some arsenic (up to 0.01 unit fractions) can be present in the composition of gold-bearing pyrite. A number of analyses shows insignificant amount of nickel. No impurities were defined in energy-dispersion spectra of crystalline pyrite, therefore, they were not analyzed.

Table 8-10 Chemical composition of pyrite

Spectrum	Sample	S	Fe	Ni	As	Σ	Formula
12227r	141411110	53.49	46.51			100.00	FeS_2
122227n		53.39	46.18		0.43	100.00	$(Fe_{0.99}As_{0.01})_{1.0}S_2$
12227f		53.05	45.77		0.83	100.00	$(Fe_{0.99}As_{0.01})_{1.0}S_2$
12,227g		53.39	46.61			100.00	FeS_2
12227b		53.25	45.99		0.76	100.00	$(Fe_{0.99}As_{0.01})_{1.0}S_2$
12228r	1414011285	53.26	45.94	0.23	0.57	100.00	$(Fe_{0.99}As_{0.01})_{1.0}S_2$
12,228t		53.79	46.21			100.00	$Fe_{0.99}S_2$
12228v		53.41	46.59			100.00	FeS_2
12228x		53.56	46.44			100.00	FeS_2
12,228g		53.92	46.08			100.00	$Fe_{0.98}S_2$
12228c		53.53	45.58	0.19	0.7	100.00	$(Fe_{0.98}As_{0.01})_{0.99}S_2$

Note: The data was received on a scanning microscope VEGA3 TESCAN, analyst I.A. Blinov.

Table 8-11 Chemical composition of chalcopyrite

Spectrum	Sample	S	Fe	Cu	Σ	Formula
12227o	141411110	35.58	30.63	33.79	100.00	$Cu_{0.96}Fe_{0.99}S_{2.00}$
12,227l		35.53	30.55	33.92	100.00	$Cu_{0.96}Fe_{0.99}S_{2.00}$
12228q	1414011285т	36.04	32.15	31.8	100.00	$Cu_{0.89}Fe_{1.02}S_{2.00}$
12228u		35.51	31.07	33.43	100.00	$Cu_{0.95}Fe_{1.00}S_{2.00}$
12228w		35.55	30.78	33.67	100.00	$Cu_{0.96}Fe_{0.99}S_{2.00}$
12228h		37.19	30.61	32.2	100.00	$Cu_{0.87}Fe_{0.95}S_{2.00}$
12228d		35.68	30.84	33.48	100.00	$Cu_{0.95}Fe_{0.99}S_{2.00}$
12228b		35.8	30.43	33.77	100.00	$Cu_{0.95}Fe_{0.98}S_{2.00}$

Note: The data was received on a scanning microscope VEGA3 TESCAN, analyst I.A. Blinov.

Table 8-12 Chemical composition of sphalerite

Spectrum	S	Fe	Zn	Cd	Σ	Formula
12,227t	33.16	0.26	64.1	2.47	100	$Zn_{0.95}Cd_{0.02}S_{1.00}$
12227p	32.73	2.17	59.88	5.22	100	$Zn_{0.90}Cd_{0.05}Fe_{0.04}S_{1.00}$
12227h	34.8	1.44	61.61	2.15	100	$Zn_{0.87}Cd_{0.02}Fe_{0.02}S_{1.00}$

Note: Sample 141411110. The data was received on a scanning microscope VEGA3 TESCAN, analyst I.A. Blinov.

Table 8-13 Chemical composition of galena

Spectrum	S	Pb	Σ	Formula
12227w	13.76	86.24	100.00	$Pb_{0.97}S_{1.00}$
12227m	13.97	86.03	100.00	$Pb_{0.95}S_{1.00}$
12227i	13.81	86.19	100.00	$Pb_{0.97}S_{1.00}$
12227j	14.39	85.61	100.00	$Pb_{0.92}S_{1.00}$

Note: Sample 141411110. The data was received on a scanning microscope VEGA3 TESCAN, analyst I.A. Blinov.

8.12. Native Gold

Gold is mainly found in aggregates with pyrite or in the form of inclusions in it. A "porous" pyrite variety is the most gold-bearing one. It also has chalcopyrite, galena, sphalerite, fahlore and pyrrhotite inclusions. Native gold inclusions in pyrite can form aggregates with galena, chalcopyrite or sphalerite. The size of gold grains is 1 to 150 micron, 30-70 micron on average. The surface of gold grains forming aggregates with pyrite is even; the edges can be slightly branching; skeleton gold crystals are rare.

Moreover, free gold is found as well, characterized by a flat surface and sizes up to 200 micron.

Some silver is observed in the chemical composition of gold (Table 8.14).. In general, the gold composition variations are insignificant.

Table 8-14 Chemical composition of gold

Spectrum	Sample	Ag	Au	Σ	Formula
12227v	141411110	13.25	86.75	100.00	$Au_{0.78}Ag_{0.22}$
12227u		13.93	86.07	100.00	$Au_{0.77}Ag_{0.23}$
12227q		12.85	87.15	100.00	$Au_{0.79}Ag_{0.21}$
12227d		12.79	87.21	100.00	$Au_{0.79}Ag_{0.21}$
12227e		12.98	87.02	100.00	$Au_{0.79}Ag_{0.21}$
12227a		13.43	86.57	100.00	$Au_{0.78}Ag_{0.22}$
12228z	1414011285	15.31	84.69	100.00	$Au_{0.75}Ag_{0.22}$
12228y		14.30	85.70	100.00	$Au_{0.77}Ag_{0.23}$
12228s		16.07	83.93	100.00	$Au_{0.74}Ag_{0.26}$
12228n		14.93	85.07	100	$Au_{0.76}Ag_{0.24}$
12228m		12.96	87.04	100	$Au_{0.79}Ag_{0.21}$
12228k		15.02	84.98	100	$Au_{0.76}Ag_{0.24}$
12228l		12.79	87.21	100	$Au_{0.79}Ag_{0.21}$
12228i		14.38	85.63	100	$Au_{0.77}Ag_{0.23}$
12228j		14.88	85.12	100	$Au_{0.76}Ag_{0.24}$
12228f		14.00	86.00	100	$Au_{0.77}Ag_{0.23}$
12228a		14.49	85.51	100	$Au_{0.76}Ag_{0.24}$

The shape of native gold was studied on the core sample material. The analysis of images was used for their quantitative characteristics. A selection for Sample 141411110 included 60 tests, and for Sample 1414011285 – 63 tests. It should be taken into account that

1) flat sections received from accidental slices were used for measurements; and

2) the analysis was conducted only for the particles, the linear sizes of which are available for microscopic studies (at least 2-3 micron).

Table 8-15 Main statistic characteristics for grain-size and morphologic gold parameters in Sample 141411110

	A_{max}, micron	A_{min}, micron	S, micron^2	P, micron	Feret Diameter, micron	Roundness	Elongation	Compactness
Minimum value	3.14	1.33	5.41	8.74	2.63	0.16	0.02	0.55
Maximum value	204.67	25.29	6228.44	501.06	89.07	1.00	0.84	1.00
Range	201.53	24.26	6223.03	492.32	86.44	0.84	0.82	0.45
Mean	39.81	8.15	809.84	106.99	25.09	0.65	0.29	0.84
Standard deviation	38.45	6.36	1247.54	97.67	22.22	0.19	0.20	0.10
Median	27.82	6.38	301.97	82.36	19.60	0.64	0.25	0.25

Table 8-16 Main statistic characteristics for grain-size and morphologic gold parameters in Sample 1414011285

	A_{max}, micron	A_{min}, micron	S, micron^2	P, micron	Feret Diameter, micron	Roundness	Elongation	Compactness
Minimum value	3.18	1.23	7.18	10.12	3.02	0.22	0.04	0.43
Maximum value	187.42	42.23	9934.6	755.35	112.50	1.00	0.82	1.00
Range	184.24	41.00	9927.42	745.23	109.48	0.80	0.78	0.60
Mean	37.31	8.19	861.12	109.81	24.4	0.65	0.33	0.72
Standard deviation	36.89	7.47	1695.11	127.1	22.58	0.20	0.22	0.15
Median	21.54	5.82	191.37	59.72	15.61	0.67	0.28	0.70

8.13. Laboratory Analysis Quality Control

The GKZ instruction provides for internal and external control of 5% from the total number of samples.

In total, 405 samples were sent for internal control to the laboratory LLC. Alex Stewart Geo Analytics in Moscow, where regular analysis was conducted, and 405 weighted portions of these samples will be sent for external control to the OJSC Irgiredmet laboratory after internal control. A slightly reduced total volume of control samples – 3.7% is explained by a small number of samples in the grade class above 5 g/t (45 samples in total). After all results are received, the samples with the grades above 1 g/t will be additionally sent for all control types.

The samples for internal and external geological control were sent coded by the geologists of LLC. Kopylovsky.

Calculation and estimation of systematic deviations in accordance with methodological NSAM instructions for statistic processing of analytical data are provided on the basis of internal and external geological control results.

Data processing for internal and external geological control will be executed by the employees of LLC. Kopylovsky, on the basis of grade classes, by the time of completion of regular analyses of three types:

- with determination of the mean square error of analyses;
- with comparison of regular and control samples on the basis of the Student criterion;
- with determination of the value of analyses deviation.

Arbitrary control will be executed in case of identification of systematic deviation between analytical results of the main and the control laboratories, on the basis of the external geological control data.

Moreover, operative internal control will be executed.

Each 50th sample of those sent for analysis is a standard. LLC. Krasny used as an enterprise standard the ‘composition of the gold-bearing ore’ bought in OJSC Irgiredmet in Irkutsk. During work two types of standards were used: composition AuBL-ИАЦ-09 with the Au grade <0.005 g/t and composition ЗСР ИАЦ-10-11 with the Au grade 0.5±0.03 g/t.

In total, 47 standards were sent. This is less than the designed 2% of all samples and is due to the fact that a part of the samples were sent for analysis without sample preparation at the base of the enterprise in Bodaibo.

The comparative analysis shows that the quality of completed analyses in the LLC. Alex Stewart Geo Analytics laboratory in Moscow meets all available requirements.

8.14. Material Composition and Process Properties of Ores

The process sample of the ore of the Krasnoye ore occurrence with the weight 60.9 kg was studied in OJSC Irgiredmet in August-October 2012.

The sample material represents carbon-carbonate-micaceous siltstone with quartz-pyrite mineralization.

The grain-size characteristic of the ore with gold distribution by size classes was studied on the basis of a 1kg weighted portion crushed till the size minus 2 mm (Table 8.17).

Table 8-17 Gold distribution by size classes

Size Class, mm	Recovery, %		Gold Grade, g/t		Gold Distribution, %	
	particular	total	particular	total	particular	total
+2	1.52	1.52	0.53	0.53	0.33	0.33
-2+1	21.85	23.37	2.07	1.97	18.53	18.86
-1+0.5	18.95	42.32	2.40	2.16	18.63	37.49
-0.5+0.315	8.74	51.06	3.33	2.36	11.92	49.41
-0.315+0.2	6.65	57.71	2.60	2.39	7.08	56.49
-0.2+0.16	5.51	63.22	2.63	2.41	5.94	62.43
-0.16+0.1	4.61	67.83	4.01	2.52	7.57	70.00
-0.1+0.071	2.81	70.64	6.0	2.66	6.90	76.90
-0.071	29.36	100.00	1.93	2.44	23.10	100.00
Total	100.00	-	2.44	-	100.00	-

The results of the table show that gold is distributed regularly in size classes.

The comparative data on the gold grade determined with various methods is given in Table 8.18.

Table 8-18 Comparative data on the gold grade in sample

Passport Data (Estimated Grade)	Direct Fire Assay	Data on Grain-Size Composition of Size Classes	Rational Analysis
Gold grade, g/t			
2.781	2.5±0.5	2.44	2.89

The gold grade in the sample defined with various analytical methods does not exceed the determination error.

8.15. Material Composition of the Ore Sample

Process sample of gold-bearing ore No 1 from ore zone No 2 of the Krasnoye ore occurrence was studied.

According to the passport, the ore sample was taken from ¼ borehole core. The size of fragments did not exceed 50 mm. According to the material composition, the sample material is presented by carbon-carbonate-micaceous siltstone with quartz-pyrite mineralization. The weight of a sample is 60.9 kg. The average estimated gold grade in the sample is 2.781 g/t.

8.15.1. Ore Description

The visual study of the sample showed that it is presented by quartz fragments the size of which is below 15 cm. The rock colour is dark-grey, almost black. The sample material consists of carbon-micaceous-quartz siltstones. The rock structure is massive, weakly schistose, locally puckered, unclearly banded, brecciated, and the texture is fine-grained. The ore mineralization is mainly presented by pyrite. It is observed in the form of irregular unequigranular dissemination and also lens-shaped, veinlet-like inclusions oriented in conformity with rock schistosity. The size of pyrite grains varies from hundredth fractions of millimeter to 5.0 mm, and the thickness of visible accumulations does not exceed 10.0 mm. Quartz margins with parallel-columnar construction are frequently noted around sulphide crystals and their accumulations. The thickness of margins is not continuous and varies from millimeter fractions to 10.0 mm. Moreover, quartz is observed in the form of thin veinlets and penetrations, the thickness of which is several millimeters.

Figure 8-4 Siltstone with rare pyrite dissemination (1). The structure is weakly schistose, and the texture is fine-grained. Grab sample

Figure 8-5 Siltstone with irregular unequigranular sulphide dissemination (1). Columnar quartz margins (2) around larger pyrite inclusions. Grab sample

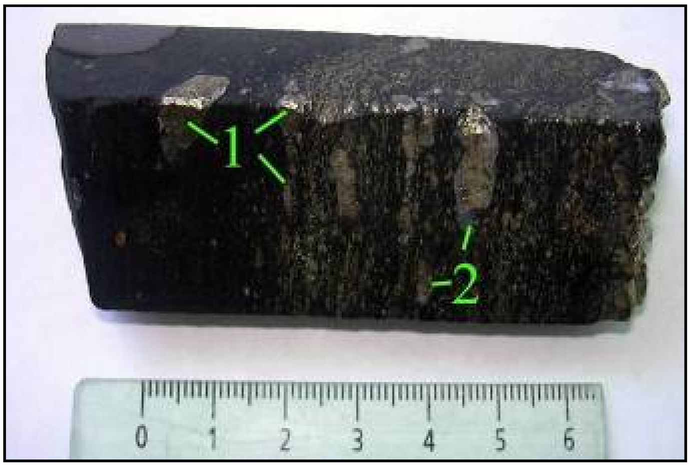
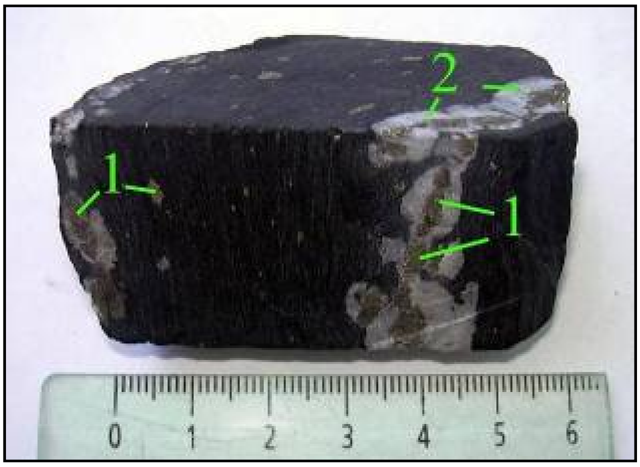


Figure 8-6 Pyrite (1) in siltstone forms dissemination, lens-shaped and veinlet-like inclusions with quartz margins, with some carbonate (2) Grab sample



Semi-quantitative, spectral, quantitative X-ray fluorescent, phase, atomic-absorption, gravimetric and ICP-AES analyses were conducted for studying the chemical composition of the ore sample. The weight fraction of carbon in the organic form was determined with the device Leco SC-114DR, in the central analytical laboratory of OJSC Pokrovsky Mine in Blagoveschensk, and carbon carbonate dioxide – with the methodology "titrimetric determination of carbon dioxide". The gold grade was defined with the assay melting method.

The chemical analysis results are reflected in Table 8.19. It was found out that the ore sample consists of 95.6% of lythophylous components with the silicon oxide (81.5%) prevailing significantly. The share of aluminum oxide is 5.8%. Ore components are mainly presented by iron and sulphur. The weight fraction of total iron is 2.61%, with the share of the element in a sulphide form – 1.66% and in an oxide form – 0.95%. The weight of total sulphur is 1.75% and it is mainly present in a sulphide form. The content of arsenic and zinc makes hundredths fractions, and antimony, copper and lead – below one thousandth of per cent.

Table 8-19 Chemical Composition of the Ore Sample

Components	Weight Fraction, %	Components	Weight Fraction, %
SiO_2	81.5	S_{tot}	1.75
Al_2O_3	5.8	S_{oxid}	0.04
TiO_2	0.38	S_{sulph}	1.71
CaO	0.20	As	0.014
K_2O	1.29	Sb	< 0.001
Na_2O	0.16	Zn	0.027
MgO	0.73	Cu	< 0.001
MnO	0.036	Pb	< 0.001
P_2O_5	0.042	$CO_{2\ carb}$	0.70
Fe_{tot}	2.61	C_{org}	2.8
Fe_{oxid}	0.95	Au. g/t	2.5
Fe_{sulph}	1.66	Ag. g/t	< 1.0

The share of carbon in an organic form is 2.8%, and the amount of carbon carbonate dioxide is about 0.7%.

The degree of ore oxidation estimated in terms of iron equals to 16%. Therefore, the ore sample is referred to an unoxidized primary type of ore. It should be noted that a part of oxidized iron is present due to carbonates and rock-forming minerals.

The grades of rare and dispersed elements, according to spectral analysis, are given in Table 8.20. It shows that their quantity is insignificant and is of no practical value.

Table 8-20 Results of spectral analysis

Elements	Weight Fraction, %	Elements	Weight Fraction, %
Ba	0.08	Y	0.004
Be	0.00015	Yb	0.0005
Zr	0.03	La	0.003
V	0.04	Nb	0.001
Cr	0.006	Sr	0.01
Ni	0.006	Ga	0.001
Co	0.002	Sc	0.001
Sn	0.0004	B	0.015
Mo	0.002		

Note: not detected: Li, Tl, Hf, Bi, W, Cd, Ce, Gd, Pt, In, U, Th, Ta, Te, Hg, In and Ge.

Gold is the main useful component of the ore sample. Its grade, according to the fire assay results, is 2.5 g/t. The silver grade, according to the atomic absorption analysis, is below 1.0 g/t.

8.15.2. Material Composition of the Ore Sample

The mineral composition of ore samples was defined qualitatively on the basis of the data of X-ray phase (difractometric) analysis conducted with the use of XRD-6000, Shimadzu, device, with Cu-filtered radiation. The following minerals were diagnosed in decreasing order – quartz, micaceous-hydromica minerals (sericite, hydrosericite, illite) and pyrite.

The quantitative mineral composition was studied on the crushed initial ore material with the size minus 2.0 mm, with the use of transparent and polished thin section microscopic data. Clayey-hydromicaceous formations were defined with the methodology of Yakovleva M.N. and Sidorenko G.A. The content of iron hydroxides was defined on the basis of difference in weight after sample treatment with 10% solution of ethane diacid in a water bath. The results of mineralogical analysis were corrected on the basis of the chemical and the X-ray structural analyses data. It was found out (Table 8.21) that the ore sample consists of 96.2% of rock-forming minerals mainly presented by quartz and micaceous formations. The size of grains and flakes of the main rock mass is thousandth and hundredth fractions of millimeter.

Table 8-21 Material Composition of the Ore Sample

Minerals, Groups of Minerals	Weight Fraction, %
Rock forming:	
Quartz	72.0
Micaceous-hydromicaceous	17.5
Carbonates (dolomite, ankerite, siderite)	1.7
Pyrite	3.3
Sphalerite, chalcopyrite, fahlore, galena	Rare and single grains
Iron oxides and hydroxides (hematite, limonite, goethite)	0.5
Carboniferous substance	5.0
Rutile, tourmaline, zircon, apatite	Rare and single grains
Total	100.0

The total weight fraction of ore minerals in the sample is 3.8%. Of them, 0.5% falls on iron oxides and hydroxides (hematite, limonite, goethite). Sulphides are presented mainly by pyrite, the share of which is 3.3%. Pyrite forms almost no aggregates with rock-forming minerals in the size class minus 0.2 mm. Sphalerite, chalcopyrite, fahlore and galena are noted in rare and single grains in the ore sample. By the amount of sulphides the sample is referred to a low-sulphide type of ore.

8.15.3. Gold Characteristics

The gold characteristics are given on the basis of its study in gravitation concentrates. It was found out that gold is native, free. The colour of gold grains varies from light-yellow to yellow. Discontinuous films of brown ferric hydroxides as well as leather coats of black ground carbon substance are noted at their surface. The surface of the main mass of gold particles is rough, small-lumpy, rarely flat. The most typical shapes of gold particles include: irregular compact with outgrowths, tabulate with crimps, lumpy, and flaky.

8.15.4. Gold Granulometry

The grain-size composition of gold was studied on the gravitation products material received after stage-by-stage reduction of the grinding size of ore. The grain-size characteristic of gold is given in Table 8.22. It shows that small, fine and fine-dispersed gold (size class minus 0.07) prevails in the ore sample – -86.4 %.

Table 8-22 Grain-size composition of gold in initial ore

Size class, mm	-0.5 +0.25	-0.25 +0.15	-0.15 +0.10	-0.10 +0.07	-0.07	Total:
Weight fraction, %	0.8	1.7	5.8	5.3	86.4	100.0

The share of coarse gold (size class +0.07 mm) is 13.6%, and most of it is concentrated in the size interval minus 0.15 + 0.07 mm. No gold particles more than 0.5 mm in size were found.

Most of fine and fine-dispersed gold (size class 3 to 25 micron) in the ore sample, according to the scintillation analysis, makes about 20% from the total gold grade and is presented by gold grains 15-25 micron in size (Table 8.23). The average estimated diameter of gold grades is 12.0 microns. The completed studies show that some part of gold in the ore sample is evidently below 3 microns in size. However, the particles of such size are not registered by scintillation analysis due to accumulation of a large statistic error.

Table 8-23 Distribution of fine and fine-dispersed gold by size classes

Size Class, micron	25-15	< 15-12	< 12-9	< 9-5	< 5-3	Total
Weight fraction of particles, %	50	-	-	-	50	100.0
Weight fraction of gold present by means of particles in this size class, g/t%	99.0	-	-	-	1.0	100.0
Average diameter of a gold grain	12.0 microns					

Gold fineness, according to the results of atomic-absorption analysis, varies from 777 to 851 units in the ore sample. This corresponds to the class of relatively low-fineness to moderately high-fineness according to Petrovskata N.V. Moderately high-fineness gold prevails in the mass.

A mineral monofraction was selected under a binocular in the size class minus 1.0 + 0.25 mm for determination of the gold grade in pyrite. According to the atomic-absorption analysis data, the gold grade in pyrite of this size reaches 62.4 g/t.

8.15.5. Rational Analysis

The rational analysis for gold in the initial ore sample from the Krasnoye deposit was conducted with a standard methodology used in OJSC Irgiredmet, on a weighted portion with the weight 1 kg, and the size of 96% minus 0.071 mm.

The following gold occurrence forms were defined: free (extracted by means of amalgamation), in the form of aggregates with ore and rock-forming components (cyanidable), extracted by means of cyanidation after treatment with chlorhydric and nitric acid, extracted by means of cyanidation after oxidation and reduction burning at T = 650^0 C, and also disseminated in rock-forming minerals.

The cyanidation operation was conducted with a sorbing agent present (resin АМ-2Б in CN-form) due to presence of a slurry-like mineral component and carboniferous substance.

The rational analysis results showed (Table 8.24) that the ore under study is resistant to the cyanidation process: gold recovery makes 79.9%. Most of precious metal falls on aggregates with ore and rock-forming components (49.3 %), and 30.6 % is free (amalgamable).

20.1% of precious metal is present in a form not accessible by cyanidation. Resistance is mainly preconditioned by gold association with sulphides (7.8%) and close association with rock-forming minerals (6.9%). To a lesser extent the resistance is influenced by: gold interrelation with a complex of minerals dissoluble in chlorhydric acid (ferric hydroxides, carbonates) and also opened by means of cyanidation after oxidation and reduction burning (1.8%).

Table 8-24 Rational analysis of the initial ore sample of the Krasnoye deposit for gold

Gold Occurrence Forms and its Relation to Ore and Rock-Forming Components	Gold Distribution	
	g/t	%
Free, extracted by means of amalgamation with the size class 0.071 mm (96%)	0.75	30.6
In aggregates (cyanidable), associated with ore and rock-forming components	1.21	49.3
Total in a cyanidable form	1.96	79.9

Gold Occurrence Forms and its Relation to Ore and Rock-Forming Components	Gold Distribution	
	g/t	%
Extractable by means of cyanidation after HCl treatment (with ferric hydroxides, carbonates etc.)	0.09	3.6
Extractable by means of cyanidation after HNO_3 treatment (associated with sulphides, mainly pyrite)	0.19	7.8
Extractable by means of cyanidation after oxidation and reduction burning at $T=650^0C$ (associated with carboniferous substance)	0.04	1.8
Fine-disseminated in rock-forming minerals	0.17	6.9
Total in the initial sample (according to the balance)	2.45	100.0

Conclusions

1. The ore sample is referred to the gold-quartz low-sulphide ore type.

2. The sample characterizes the primary ore type because the degree of oxidation estimated on the basis of iron is at the level 16%.

3. The main rock-forming minerals in it include quartz and micaceous-hydromicaceous formations.

4. Pyrite is a prevailing ore mineral. Its weight fraction is 3.3%. The share of ferric oxides and hydroxides is 0.5%.

5. Gold is the main useful component of the ore sample. The grade of the precious metal according to fire assay results is 2.5 g/t.

6. Gold is native. The shape of grains is irregular compact with outgrowths, tabulate with crimps, lumpy, and flaky.

7. Fine and fine-dispersed gold (size class 0.07 mm) prevails in the ore sample – 86.4%. The share of coarse gold (size class +0.07 mm) is 13.6%, and most of it is concentrated in the size interval minus 0.15 + 0.07 mm.

8. Gold is relatively low-fineness and moderately high-fineness. Its fineness varies from 777 to 851 units.

9. The ore studied is referred to hard-to-cyanate ores: gold recovery makes 79.9% (below 90%). The share of free gold is 30.6%. 49.3% of the precious metal is present in aggregates with ore and rock-forming minerals. Resistance is mainly preconditioned by gold association with pyrite (7.8%) and also fine dissemination in rock-forming minerals (6.9%). The presence of organic carbon 2.8% with regard to its sorption activity will significantly influence the degree of recovery of the previous metal from processing products.

8.16. Determination of Ore Density and Porosity

The density of the ore sample material was determined on the material ground till the size minus 0.071 mm.

The density is defined with the equation:

$$\sigma = \frac{(А - Б)\gamma_{ж}}{(С - Б) - (Д - А)}$$

where σ – density, g/cm^3;

А – weight of a dry flask with material, g;

С – weight of a flask with water, g;

Б – weight of a dry flask, g;

$\gamma_{ж}$ – density of water at test temperature (1 g/cm^3);

Д – weight of a flask with water and material, g.

The bulk weight (Δ, g/cm^3) was defined with the equation:

$\Delta=(P - P_0)/A$,

where P is the weight of a reservoir with material, g;

P_0– weight of a reservoir, g;

А – volume of a reservoir, cm^3;

Porosity (В) was defined with the equation:

В= (σ – Δ) /σ

The test results are given in Table 8.25.

Table 8-25 Results of determination of physical and mechanical properties

Item No	Weight of Empty Flask, Б	Weight of a Flask with Material, А	Weight of a Flask with Material and Water, Д	Weight of a Flask with Water without Material, С	Density, g/cub. cm	Density, g/cub. cm	Bulk Weight, Δ	Porosity, В
1	24.75	39.75	83.83	74.65	2.58	2.58	1.218	0.53
2	19.4	34.4	78.35	68.95	2.68			
3	20.14	35.14	78.84	69.73	2.55			
4	20.26	35.26	79.12	69.99	2.55			
5	26.07	41.07	84.97	75.93	2.52			
6	19.92	34.92	78.90	69.57	2.61			

Determination of rock competency for the Krasnoye deposit was conducted in accordance with GOST 21153.1-75 “Rocks. Method of Determination of the Strength Coefficient in Accordance with Protodyakonov”.

Pieces 50 mm in size were taken for testing.

The rock strength coefficient (f) is estimated with the equation:

$$f = \frac{20 \cdot n}{h},$$

where 20 is an empiric numerical coefficient providing for the reception of generally acknowledged strength coefficient values and taking into account the work spent for crushing;

n – number of weight drops for one weighted portion testing;

h – height of a column of fine fraction in a volumeter after testing five weighted portions, mm.

The arithmetic mean value of four test results is assumed as a final test result.

The strength coefficient test results are given in Table 8.26.

Table 8-26 Strength coefficient test results

Number of Weight Drops, times	Height of a Fine Fraction Column, mm	Rock Strength Coefficient (f)
50	31	6.45
50	50	4.00
50	55	3.64
50	31	6.45
Average value		5.14

The ore of the Krasnoye deposit is referred to the strength category IV – quite strong rocks on the basis of the strength coefficient as per the Protodyakonov’s scale.

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8.17. Gravitation and Flotation Processing

8.17.1. Gravitation Processing

The rational analysis results showed that there is 30.6% of free gold in the ore sample from the Krasnoye ore occurrence. It preconditioned the necessity of conducting gravitation processing studies.

Gold from the initial ore was processed in stages, with successive reduction of the grinding size. Tests were conducted in a jig, the black concentrate of which was processed on a concentration table and in a centrifugal concentrator Knelson KS-MDZ. The mode parameters of the jig and the concentrator Knelson KS-MDZ are given in Table 8.27. Test results are given in Table 8.28.

Table 8-27 Test mode parameters

Mode Parameters	Particle Size in the Feed, mm		
	-2 (-1)	-0.5	-0.2
Jig MOD-0.2			
Productivity, kg/hr	72	60	60
Water flow rate, l/hr, l/min	600/10-12	460/7-5	460/7-5
Length of the diaphragm run, mm	9-10	5-6	5-6
Shot size, mm	4-5	2-3	2-3
Sieve aperture, mm	2.0	0.5	0.2
Concentrator Knelson KS-MD3, mm			
	1.0	0.5	0.2
Initial weighted portion mass, kg	20.00	-	-
Feed rate, kg/hr	60	50	40
Rotating speed, rpm	1455 (60 G)	1455 (60 G)	1455 (60 G)
Fluidizing water flow rate, l/min	3.5	3.5	3.5

Table 8-28 Results of staged gravitation processing

Processing Products	Recovery, %	Gold Grade, g/t	Gold Recovery, %	Type of Gravitation Equipment
Table concentrate minus 2.0 mm	0.98	85.0	37.3	Jig – Concentration table
Table concentrate minus 1.0 mm	0.34	86.9	13.2	
Table concentrate minus 0.5 mm	0.31	112.0	15.6	
Table concentrate minus 0.2 mm	0.17	165.0	12.2	
Table concentrate minus 0.071 mm	0.05	184.0	4.1	
United concentrate	1.85	99.3	82.4	
Gravitation tails	98.15	0.4	17.6	
Ore as per balance/(analysis)	100.00	2.23/(2.5±0.5)	100.0	
Concentrate minus 1.0 mm	0.8	111.0	37.8	Centrifugal concentrator Knelson
Concentrate minus 0.5 mm	0.8	69.6	23.7	
Concentrate minus 0.2 mm	0.8	65.0	22.1	
United concentrate	2.4	81.8	83.6	
Gravitation tails	97.6	0.4	16.4	
Ore as per balance/(analysis)	100.00	2.35/(2.5±0.5)	100.0	

The comparison of the results shows that with the help of a jig and a concentration table 82.4% of metal can be extracted with the final size of minus 0.071 mm, and with the help of a centrifugal concentrator Knelson – 83.6%, with the final size of minus 0.2 m, with insignificant concentrate recovery growth. Therefore, it is evident that there is no demand in ore crushing till the size of 0.071 m. Meanwhile, the increase of the recovery from the size class minus 0.2 mm to the class minus 0-0.71 mm made 4.1%, with the total recovery of 0.05%.

8.17.2. Flotation of Gravitation Tails

The studies of the impact of the crushing size, the collecting agent rate and the duration of the flotation were conducted in order to receive optimal parameters of gravitation tails flotation processing, providing for maximum gold extraction with the gravitation-flotation processing flowchart.

At the first stage of studies, the crushing size was determined, which varied from 0.2 mm to 0.071 mm. Test conditions: chemical rate per 1 t of ore, g: BKK – 150 (100 to main and 50 to control), foaming agent (PM-2) – 150 (125 to main and 25 to control), and flotation duration – 25 minutes (10 – main and 15 – control), solid weighted fraction – 25%. The results received during tests on the impact of the crushing size on flotation processing parameters are given in the table below.

Table 8-29 Impact of the crushing size on flotation parameters

Processing Products	Recovery, %	Grade of Gold, g/t	Recovery Gold, %	Size of Crushing
Main flotation concentrate	5.96	3.16	35.53	60-65% of the class minus 0.074 mm (class -0.2 mm)
Control flotation concentrate	14.51	0.84	22.99	
Flotation concentrate	20.47	1.51	58.52	
Flotation tails	79.53	0.22	41.48	
Flotation feed as per balance/(analysis)	100.0	0.53 / (0.4±0.13)	100.0	
Main flotation concentrate	12.92	2.12	57.06	70-75 % of the class minus 0.074 mm (class -0.16 mm)
Control flotation concentrate	18.84	0.68	26.69	
Flotation concentrate	31.76	1.26	83.75	
Flotation tails	68.24	0.12	16.25	
Flotation feed as per balance/(analysis)	100.0	0.48 / (0.4±0.13)	100.0	
Main flotation concentrate	8.22	2.88	44.66	85-90 % of the class minus 0.074 mm (class -0.1 mm)
Control flotation concentrate	22.27	0.8	33.61	
Flotation concentrate	30.49	1.36	78.27	
Flotation tails	69.51	0.16	21.73	
Flotation feed as per balance/(analysis)	100.0	0.53/ (0.4±0.13)	100.0	
Main flotation concentrate	4.15	3.64	32.14	95 and > % of class minus 0.074 mm (class -0.074mm)
Control flotation concentrate	19.37	1.16	47.81	
Flotation concentrate	23.52	1.59	79.95	
Flotation tails	76.48	0.12	20.05	
Flotation feed as per balance/(analysis)	100.0	0.47 / (0.4±0.13)	100.0	

Table 9.28 shows that the optimal crushing size for flotation is 70-75% of the class minus 0.071 (-0.16) mm. All further tests were conducted for this size.

The next stage of investigations was to define the rate of a collecting agent – potassium butyl xantate required for maximum metal extraction from gravitation tails. The potassium butyl xantate rate varied from 120 to 250 g/t; the other parameters of flotation tails remained constant.

Test results are given in the table below and show that the optimal potassium butyl xantate rate made 150 g/t. With such a rate the losses with tails made only 3.36% with the gold grade in them 0.037 g/t.

The low gold grade in gravitation tails at the level of 0.037-0.09 g/t was reached as a result of high concentrate recovery 36.57 to 54.57%. Later the recovery was reduced for concentrate quality provision.

Table 8-30 Impact of the collecting agent rate on flotation parameters

Processing Products	Recovery, %	Grade of Gold, g/t	Recovery Gold, %	Potassium Butyl Xantate Rate per 1 t of ore, g:
Main flotation concentrate	15.36	2.32	74.24	70
Control flotation concentrate	21.21	0.4	17.67	50
Flotation concentrate	**36.57**	**1.21**	**91.91**	**120**
Flotation tails	63.43	0.065	8.09	
Flotation feed as per balance/(analysis)	100.0	0.48 / (0.4±0.13)	100.0	
Main flotation concentrate	24.43	1.44	78.17	100
Control flotation concentrate	21.88	0.38	18.47	50
Flotation concentrate	**46.31**	**0.94**	**96.64**	**150**
Flotation tails	53.69	0.037	3.36	
Flotation feed as per balance/(analysis)	100.0	0.45 / (0.4±0.13)	100.0	
Main flotation concentrate	23.64	1.68	81.05	150
Control flotation concentrate	22.65	0.32	14.79	50
Flotation concentrate	**46.29**	**1.01**	**95.84**	**200**
Flotation tails	53.71	0.043	4.16	
Flotation feed as per balance/(analysis)	100.0	0.49/ (0.4±0.13)	100.0	
Main flotation concentrate	37.98	1.18	86.18	200
Control flotation concentrate	16.59	0.2	6.38	50
Flotation concentrate	**54.57**	**0.88**	**92.56**	**250**
Flotation tails	45.43	0.090	7.44	
Flotation feed as per balance/(analysis)	100.0	0.52 / (0.4±0.13)	100.0	

Flotation duration was defined with the help of foam product removal every specific time intervals. The results are given in tables below.

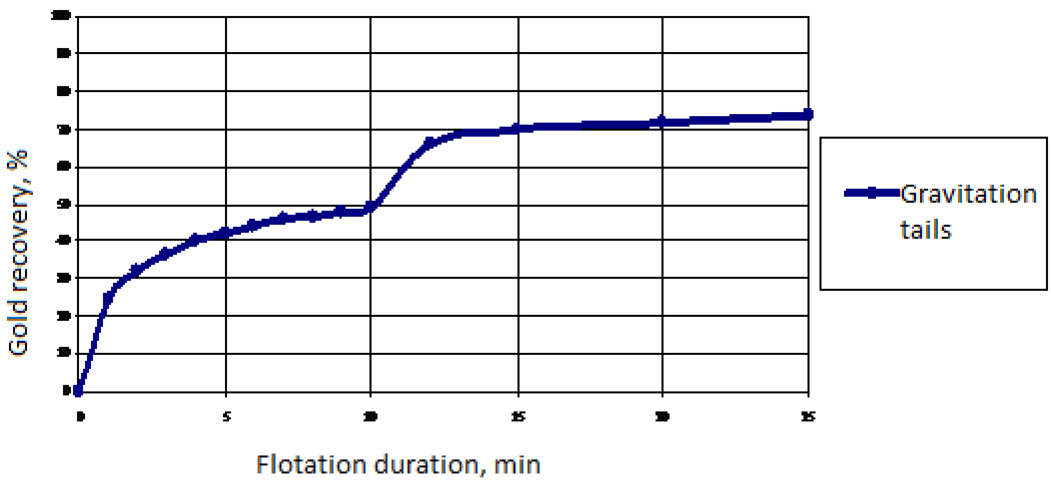
Process monitoring (foaming, degree of loading and foam structure) and the results of completed tests showed that the flotation duration made 25 minutes: 10 – main, 15 – control, 1 – aftertreatment.

Moreover, the main flotation shall be divided into two stages for reception of a higher-quality concentrate: the first main one – 2 minutes and the second main one – 8 minutes. The curve jump between 10 and 15 minutes corresponds to agents supply to control flotation and shows that fractional agent supply allows to increase the recovery.

Table 8-31 Determination of the flotation process duration

Flotation Duration, min		Product of Processing	Recovery, %		Au Grade, g/t		Au Recovery, g/t	
particular	total		particular	total	particular	total	particular	total
1	1	Concentrate 1	1.14	1.14	11.0	11.0	24.58	24.58
1	2	Concentrate 2	1.06	2.2	3.71	7.48	7.71	32.29
1	3	Concentrate 3	0.94	3.14	2.35	5.95	4.33	36.62
1	4	Concentrate 4	0.93	4.07	1.90	5.02	3.46	40.08
1	5	Concentrate 5	0.71	4.78	1.38	4.48	1.92	42.0
1	6	Concentrate 6	0.71	5.49	1.54	4.10	2.14	44.14
1	7	Concentrate 7	0.71	6.20	1.11	3.76	1.54	45.68
1	8	Concentrate 8	0.75	6.95	0.75	3.43	1.10	46.78
1	9	Concentrate 9	0.64	7.59	0.87	3.22	1.09	47.87
1	10	Concentrate 10	0.67	8.26	1.00	3.04	1.31	49.18
2	12	Concentrate 11	6.81	15.07	1.24	2.22	16.56	65.74
3	15	Concentrate 12	5.49	20.56	0.4	1.74	4.30	70.04
5	20	Concentrate 13	3.31	23.87	0.3	1.53	1.83	71.87
5	25	Concentrate 14	2.54	26.41	0.34	1.42	1.69	73.56
-	-	Tails	73.59	100.00	0.18	0.51	26.44	100.00
-	-	Feed - gravitation tails as per balance/(analysis)	100.00	-	0.51 (0.4±0.13)	-	100.0	-

Figure 8-7 Dependence of gold recovery on flotation duration



The flotation chart was chosen of two options: with the standard flotation chart and on the basis of earlier completed studies on similar types of ores of the Sukhoy Log and the Western-2 deposits.

Table 8-32 Optimal parameters of flotation processing of gravitation tails

Operation	Chemical Rate per 1 t, g		Flotation Duration, min	Weight Fraction of Solids, %
	Potassium Butyl Xantate	PM-2		
P70-75 % minus 0.074 mm (class -0.16 mm)				
I Main flotation	70	125	2	25-30
II Main flotation	30	12.5	8	
Middlings flotation	-	-	5	
Control flotation	50	12.5	15	
Aftertreatment flotation	-	-	1	

Figure 8-8 Gravitation-flotation ore processing chart, open cycle (Option 1)

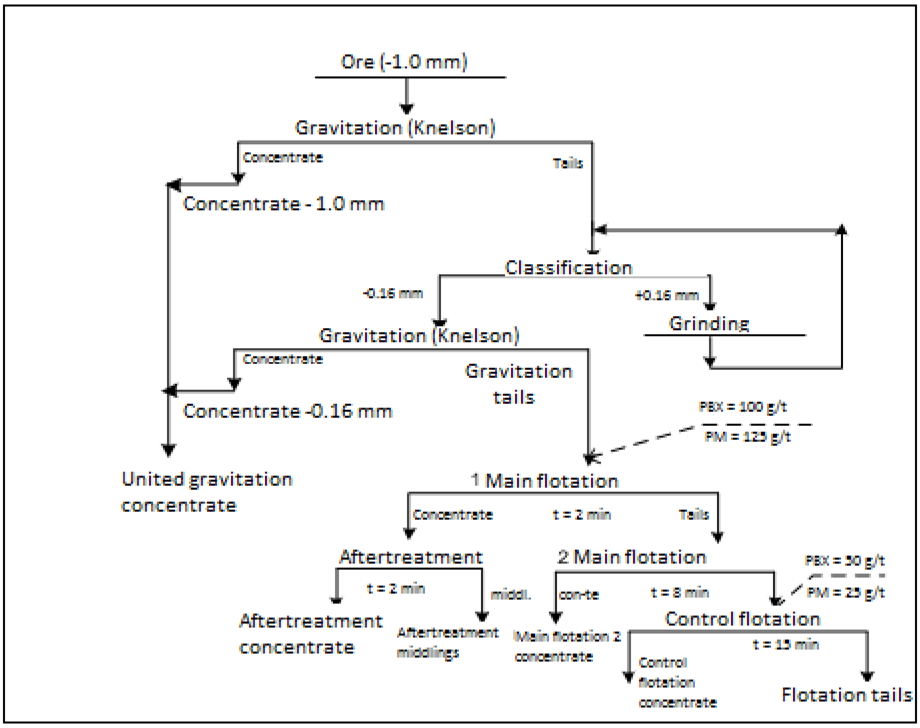


Figure 8-9 Gravitation-flotation ore processing chart, open cycle (Option 2)«

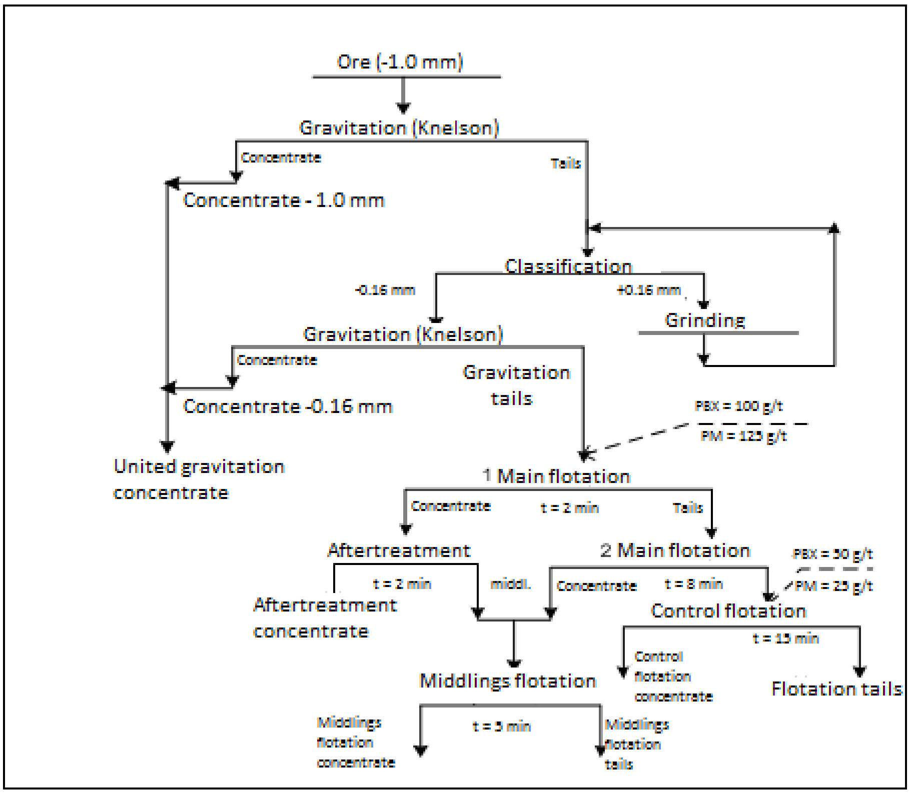


Table 8-33 Process parameters of gravitation-flotation ore processing in an open cycle (Option 1)

Processing Products	Recovery, %	Gold Grade, g/t	Gold Recovery, %
Gravitation concentrate minus 1.0 mm	0.83	109	37.08
Gravitation concentrate minus 0.16 mm	0.85	114	39.71
United gravitation concentrate	**1.68**	**111.5**	**76.79**
Aftertreatment concentrate	0.75	19.6	6.02
Aftertreatment middlings	3.30	1.27	1.70
II main flotation concentrate	7.16	1.01	2.96
Control flotation concentrate	15.49	0.90	5.70
Control flotation tails	71.62	0.25	6.83
Flotation feed (gravitation tails) as per balance/(analysis)	98.32	0.58/(0.6 ± 0.17)	23.21
Total RoM ore	100.00	2.44	100.00

Table 8-34 Process parameters of gravitation-flotation ore processing in an open cycle (Option 2)

Processing Products	Recovery, %	Grade of Gold, g/t	Gold Recovery, %
Gravitation concentrate minus 1.0 mm	0.83	109	37.08
Gravitation concentrate minus 0.16 mm	0.85	114	39.71
United gravitation concentrate	**1.68**	**111.5**	**76.79**
Aftertreatment concentrate	0.34	44.6	6.21
Middlings flotation concentrate	1.94	5.6	4.45
Control flotation concentrate	18.68	0.77	5.89
Middlings flotation tails	5.82	0.47	1.12
Control flotation tails	71.54	0.19	5.54
Flotation feed (gravitation tails) as per balance/(analysis)	98.32	0.58/(0.6 ± 0.17)	23.21
Total RoM ore	100.00	2.44	100.00

The gold recovery in case of gravitation-flotation ore processing in an open cycle to the united concentrate made 82.81 for the first option and 83.0% for the second option. Metal losses with flotation tails were at the level of 6.83 and 5.54% respectively.

The results of investigation showed that introduction of the middlings flotation fed by the low-grade concentrate of control flotation and the middlings of the main concentrate aftertreatment allows to remove a significant part of sludge to tailings, thus eliminating their return to the main flotation chart. This improves the quality of the flotation concentrate and reduces its recovery which is very important in this work.

In the future the time of this flotation shall be increased till 10 minutes in order to reduce the gold grade in the middlings flotation tails.

After establishment of the main mode parameters on gravitation tails, tests were executed with imitation of the closed cycle. The chemical rate for the 1st main flotation made: potassium butyl xantate 70 g/t, PM-2 – 125 g/t; and for the 2nd main flotation: potassium butyl xantate 30 g/t, PM-2 – 12,5 g/t; for control flotation - potassium butyl xantate 50 g/t, and PM-2 – 12,5 g/t. Starting from the second weighted portion the PM-2 rate for the main flotation fell till 112.5 g/t due to the use of recycled water and control flotation concentrate return.

Tests in a closed cycle were conducted on gravitation tails with the total mass 5,000 g for each sample with control flotation concentrate return to the main flotation operation on the following weighted portion and middlings flotation concentrate return to the aftertreatment operation on the following weighted portion etc. The parameters of flotation processing of gravitation tails with the closed cycle simulation are given in the table below.

Figure 8-10 Flotation processing chart with closed cycle simulation

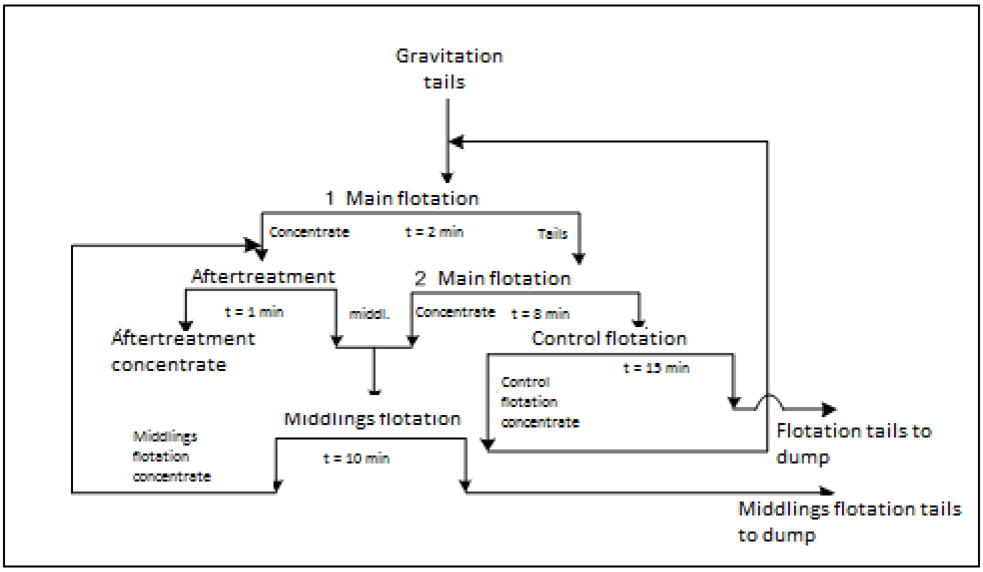


Table 8-35 Gravitation tails flotation with closed cycle simulation

Processing Products	Recovery, g	Gold Grade, g/t
Concentrate 1	4.6	34.2
Concentrate 2	9.9	35.7
Concentrate 3	8.8	26.6
Concentrate 4	14.1	19.3
Concentrate 5	14.0	19.4
Middlings flotation 1 tails	140.1	0.11
Middlings flotation 2 tails	175.3	0.19
Middlings flotation 3 tails	172.1	0.13
Middlings flotation 4 tails	145.7	0.20
Middlings flotation 5 tails	155.6	0.15
Flotation 1 tails	795.1	0.16
Flotation 2 tails	894.3	0.27
Flotation 3 tails	844.4	0.22
Flotation 4 tails	861.5	0.23
Flotation 5 tails	809.6	0.19
Control flotation concentrate	175.7	0.85
Middlings flotation concentrate	60.6	4.22
Flotation feed as per balance/(analysis)	5290.4	0.46/(0.6±0.17)

Table 8-36 Process parameters of gravitation processing tails flotation with a closed cycle simulation (by the last weighted portion)

Processing Products	Recovery		Gold Grade, g/t	Recovery of Gold, %
	g	%		
Flotation concentrate	14.0	1.43	19.4	60.3
Middlings flotation tails	155.6	15.89	0.15	5.18
Flotation tails	809.6	82.68	0.19	34.52
Total tails	965.2	98.57	0.18	39.7
Gravitation tails as per balance/(analysis)	979.2	100.0	0.46/(0.6±0.17)	100.0

The operating gold recovery as a result of gravitation tails flotation processing with closed cycle simulation made 60.3%; meanwhile, tails with the dump gold grade 0.18 g/t were received.

The flotation concentrate for metallurgic cyanidation studies was prepared with the gravitation and flotation flowchart.

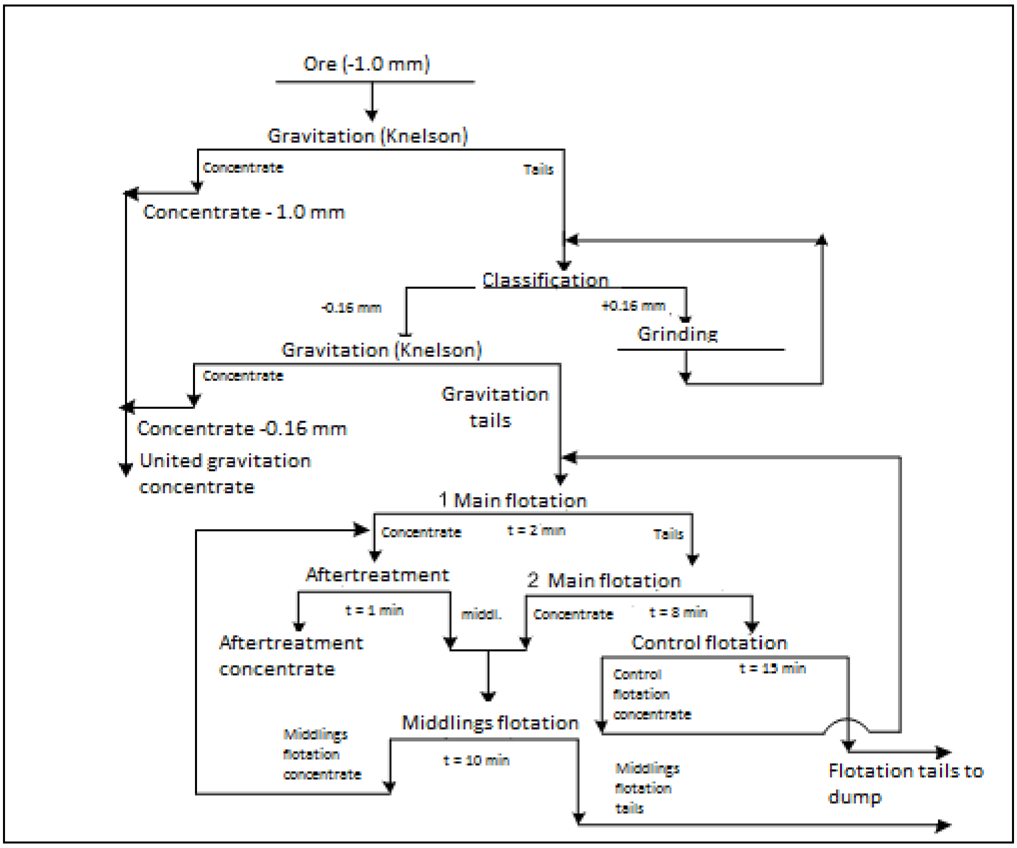
Table 8-37 Process parameters of gravitation and flotation ore processing

Processing Products	Recovery, %	Gold Grade, g/t	Gold Recovery, %
United gravitation concentrate	1.67	120	82.6
Gravitation tails (flotation feed) as per balance/(analysis)	98.33	0.43/ (0.6±0.17)	17.40
Flotation concentrate	1.25	20.2	10.22
United concentrate	**2.92**	**77.2**	**92.82**
Flotation tails	97.08	0.18	7.18
Ore as per balance/(analysis)	100.0	2.43/ (2.5±0.5)	100.0

Table results confirm that the gravitation-flotation processing chart will allow to receive high recoveries (at the level of 92.82%).

A large ore sample shall be studied for accurate confirmation of the recommended flowchart results.

Figure 8-11 Gravitation and flotation processing flowchart



9. Reserve Estimation Methodology

Resources and reserves were estimated with the computer geostatistic modeling methodology.

Computer modeling of deposits with the use of statistic and geostatistic methods reflects spatial regularities in distribution of a wide complex of mineralization parameters for mineral deposits more accurately and fully. The quantitative estimate of mineral resources on the basis of computer models defines higher accuracy as compared to traditional methods since it allows to take into account the arbitrary number of parameters which influence reserve estimation (direct and indirect).

The modeling process mainly consists of the following stages:

- Development of a data base structure for storage of the primary exploration data;
- Basic information analysis and input to the geological workings data base.
  - *preparation of geological information for its entering to the system;*
  - *base replenishment with geological sampling, geophysical and other measurement data;*
  - *statistic analysis of primary geological data, correction of errors, data grouping, base confirmation, detection of regularities;*
- Interpretation of geological exploration data:
  - *building of boreholes in the model space, grouping by profile lines;*
  - *determination and contouring of ore and waste intervals on the basis of the stratigraphic principle and lithology, specification of intervals on the basis of cutoff grade values (geological data interpretation);*
  - *specification of the boundaries of spatial rock distribution with account of tectonic faults and in accordance with the data of geophysical studies (seismic, electric exploration, magnetic and gravitation survey);*
- Formation of volumetric wireframe models in space:
  - *wireframe modeling of the deposit (modeling of ore bodies and overburden rocks, seams, anomalies, traps etc.);*
  - *wireframe modeling of surfaces and underground workings;*
- Geostatistic studies of the deposit (analysis of spatial data, variography, determination of spatial variability (anisotropy) laws of geological component characteristics;

- Block modeling of deposits:
  - *building of empty block models;*
  - *interpolation of the grade of components with mathematic methods – IDW, kriging (in modifications) etc.;*
  - *determination of geological reserves and resources by categories (classes);*
- Reserve estimation.

9.1. Analysis of Borehole and Trench Databases

The Customer provided databases on boreholes and channels in .mdb and .dat formats. The files contains in bases were renamed in accordance with LLC. Miramine internal standards, with preservation of field names in files. The input files, their content and renamed files are given in the table below.

Table 9-1 Borehole database (DH_NEW.dhdb)

Original File	File Content	Renamed File
KRA_Ustia_core.DAT	Collar file	DH_collar_new.DAT
KRA_Inklin_core.DAT	Directional survey file	DH_survey_new.DAT
KRA_Proba_core.DAT	Borehole sampling file	DH_assay_new.DAT
KRA_Geoldat_core.DAT	Lithology file	DH_geo_new.DAT

Table 9-2 Channel database (TR_NEW.dhdb)

Original File	File Content	Renamed File
KRA_Channel_MarkPiket.DAT	Trench coordinates file	TR_collar_new.DAT
KRA_Channel_proba.DAT	Channel sampling file	TR_assay_new.DAT
KRA_Channel_GeoldatDAT	Lithology file	TR_Geo_new.DAT

The database DH_NEW.dhdb contains the following fields in files:

1. Collar file: DH_collar_new.DAT.

Table 9-3 Field name: Collar file: DH_collar_new.DAT.

Field	Field Content
BHID	Borehole No
X,Y,Z	Borehole collar coordinates
EOH	Borehole depth (m from collar)

2. Directional survey file: DH_survey_new.DAT.

Table 9-4 Field name: Directional survey file: DH_survey_new.DAT.

Field	Field Content
DHID	Borehole No
FROM	Measurement depth from, m
TO	Measurement depth to, m
AZ	Borehole inclination azimuth
DIP	Borehole dip angle

3. Sampling file: DH_assay_new.DAT.

Table 9-5 Field name: Sampling file: DH_assay_new.DAT.

Field	Field Content
BHID	Borehole No
SAMPLE	Sample No.
FROM	Beginning of the sampling interval (m from collar)
TO	Ending of the sampling interval (m from collar)
LENGTH	Interval length (m)
CONTROL	
AU	Grade field, g/t

4. Lithology file: DH_geo_new.DAT.

Table 9-6 Field name: Lithology file: DH_geo_new.DAT.

Field	Field Content
BHID	Borehole No
FROM	Beginning of the sampling interval (m from collar)
TO	Ending of the sampling interval (m from collar)
NAME	Lithological description
SOSTAV	
COLOR	
STRUKTURA	
TEXTURA	
METASOM	
PROJGILK	
TRESCH	
VIVETR	
OKISL	
TXT_CTS	
TXT_SOSTAV	
ROCK-FORM	
TXT_ORE	
TXT_PROJGI	
SUPERGENE	
TXT_TRESCH	
TXT_METASO	
TXT_TEKTON	
CONTACT	
COMMENT	

The database TR_NEW.dhdb **contains the following fields in files:**

1. Collar file: TR_collar_new.DAT.

Table 9-7 Field name: Collar file: TR_collar_new.DAT.

Field	Field Content
BHID	Trench No.
X,Y,Z	Channel stake coordinates
meter	Distance between channel stakes

2. Sampling file: TR_assay_new.DAT.

Table 9-8 Field name: Sampling file: TR_assay_new.DAT.

Field	Field Content
BHID	Trench No.
SAMPLE	Sample No.
FROM	Beginning of the sampling interval (m from zero stake)
TO	Ending of the sampling interval (m from zero stake)
LENGTH	Interval length (m)
CONTROL	
AU	Grade field, g/t

3. Lithology file: DH_geo_new.DAT.

Table 9-9 Field name: Lithology file: DH_geo_new.DAT.

Field	Field Content
BHID	Trench No.
FROM	Beginning of the sampling interval (m from zero stake)
TO	Ending of the sampling interval (m from zero stake)
NAME	Lithological description
SOSTAV	
COLOR	
STRUKTURA	
TEXTURA	
METASOM	
PROJGILK	
TRESCH	
VIVETR	
OKISL	
TXT_CTS	
TXT_SOSTAV	
ROCK-FORM	
TXT_ORE	
TXT_PROJGI	
SUPERGENE	
TXT_TRESCH	
TXT_METASO	
TXT_TEKTON	
CONTACT	
COMMENT	

The Customer also provided graphical materials (geological sections along drilling profiles) and a topographic surface file Krasniy_topo.dwg, which was converted to the wireframe file DTM.TDB.

9.2. Verification of Borehole and Trench Databases

The received borehole coordinates and sampling files were verified in GGIS Micromine with the help of special processes for presence of the following possible errors:

- The working number is absent in the coordinates file but is present in the analytical results file.
- The working number is absent in the analytical results file but is present in the coordinates file.
- The working number is present more than once in the analytical results or the coordinates files;
- One or more coordinates of the working collar is omitted or wrong in the coordinates file;
- FROM or TO is missing in the analytical results file;
- FROM is more or equals to TO in the analytical results file;
- Sampling intervals are not adjacent in the analytical results file (there is a gap between analyses);
- Sample intervals overlap in the analytical results file.
- The total depth of the borehole is less than the depth of the last sample.

No errors were detected as a result of verification in the DH_NEW.dhdb database.

Verification of the TR_NEW.dhdb database revealed the following errors given in Table 9.10.

Table 9-10 Database TR_NEW.dhdb errors

File	Working No.	Error
TR_collar_new.DAT	143501	Total length of the trajectory < total length of intervals
TR_collar_new.DAT	143502	Total length of the trajectory < total length of intervals
TR_collar_new.DAT	143505	Total length of the trajectory < total length of intervals
TR_collar_new.DAT	143506	Total length of the trajectory < total length of intervals
TR_collar_new.DAT	143507	Total length of the trajectory < total length of intervals

Errors were corrected by means of extension of the channel trajectory in the general direction to the total sampling length.

A repeated check detected no errors.

Moreover, a united sampling file was created for boreholes and trenches all_assay_new.dat. The structure of the file was preserved, a field AU_correct was added, and zero gold grades detected in the Au grade field were replaced in the field AU_correct with 0.01 (half of the detection limit of the device).

9.3. Statistic Analysis of Sampling Data

The statistic analysis was conducted several times, both for the whole sampling database and the selection in the wireframe.

The purposes of the analysis were:

- determination of the natural cutoff grade of the mineralization for further interpretation;
- determination of the type of distribution in ores.

The processes *Statistics/distribution, Statistics/Normal/Log* were used in ***Micromine*** for statistic analysis.

Figure 9-1 Data on classic statistics Normal/Log for Au grades for all samples

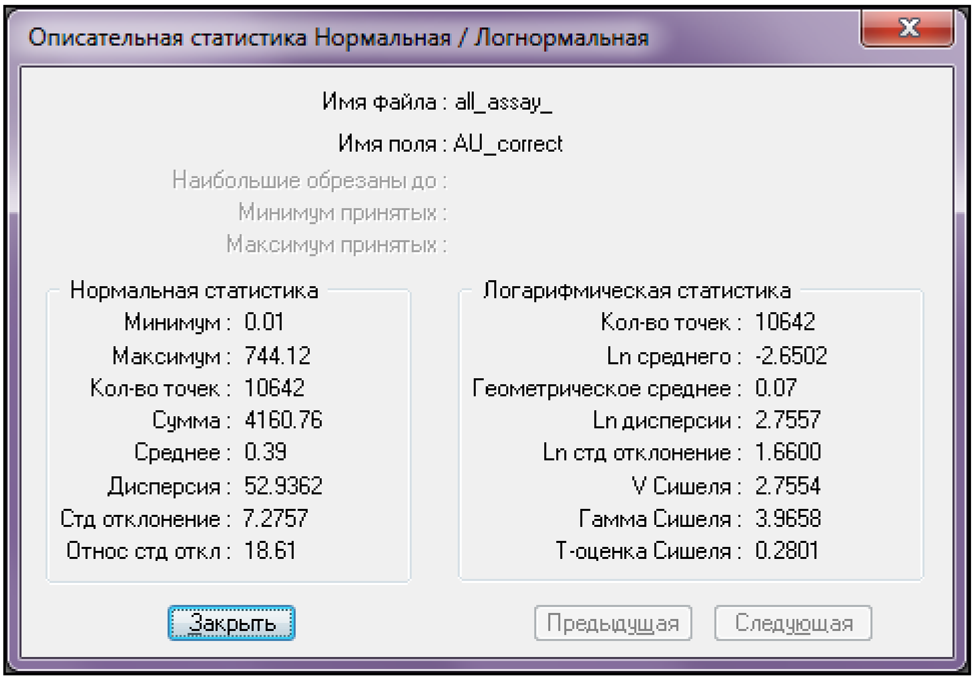


Figure 9-2 Data on classic statistics (Median/Mode) for Au grades for all samples

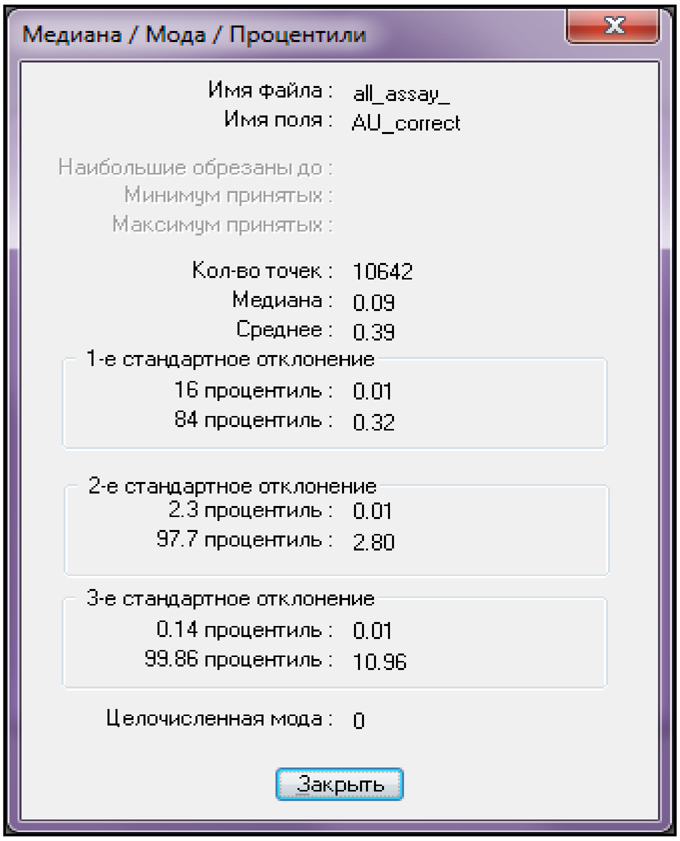


Figure 9-3 Histogram for all samples of the deposit for Au grades

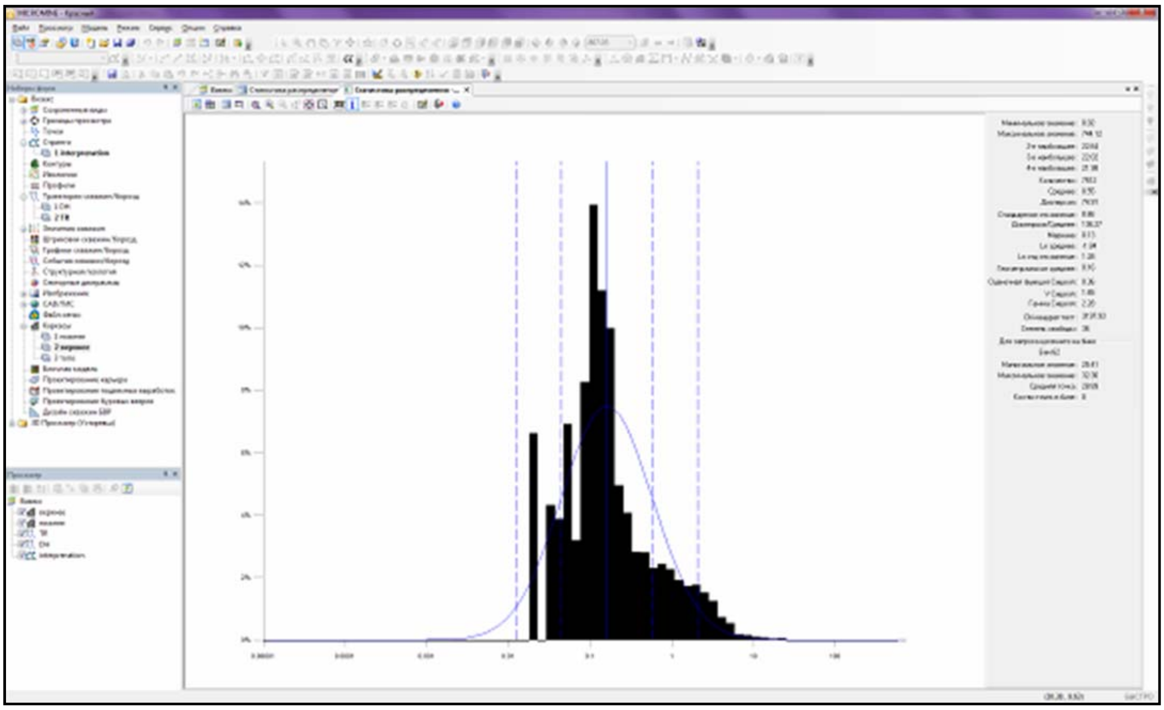
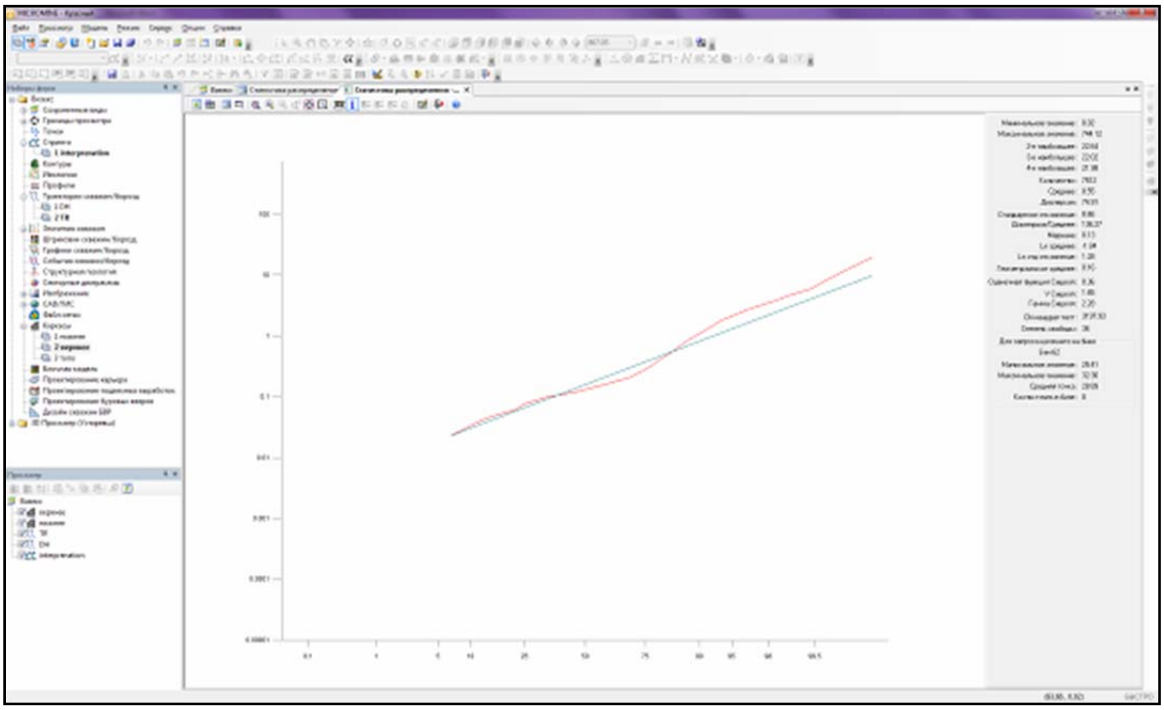


Figure 9-4 Probability diagram for all samples of the deposit for Au grades



The histogram and the cumulative probability diagram show lognormal gold distribution and presence of one population. A natural cutoff grade equal to 0.2 g/t was used for Au mineralization interpretation. It is seen at the bend of the cumulative probability curve.

9.3.1. Statistical Analysis for Different Workings

Figure 9-5 Data on classic statistics Normal/Log for Au grades for boreholes

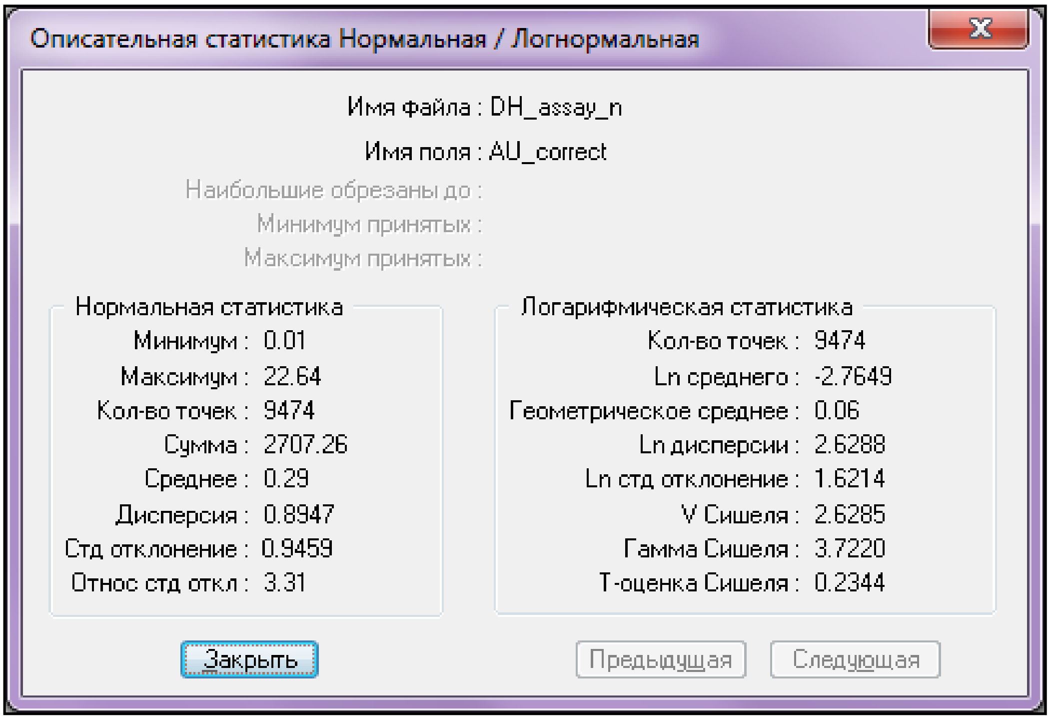


Figure 9-6 Data on classic statistics Normal/Log for Au grades for trenches

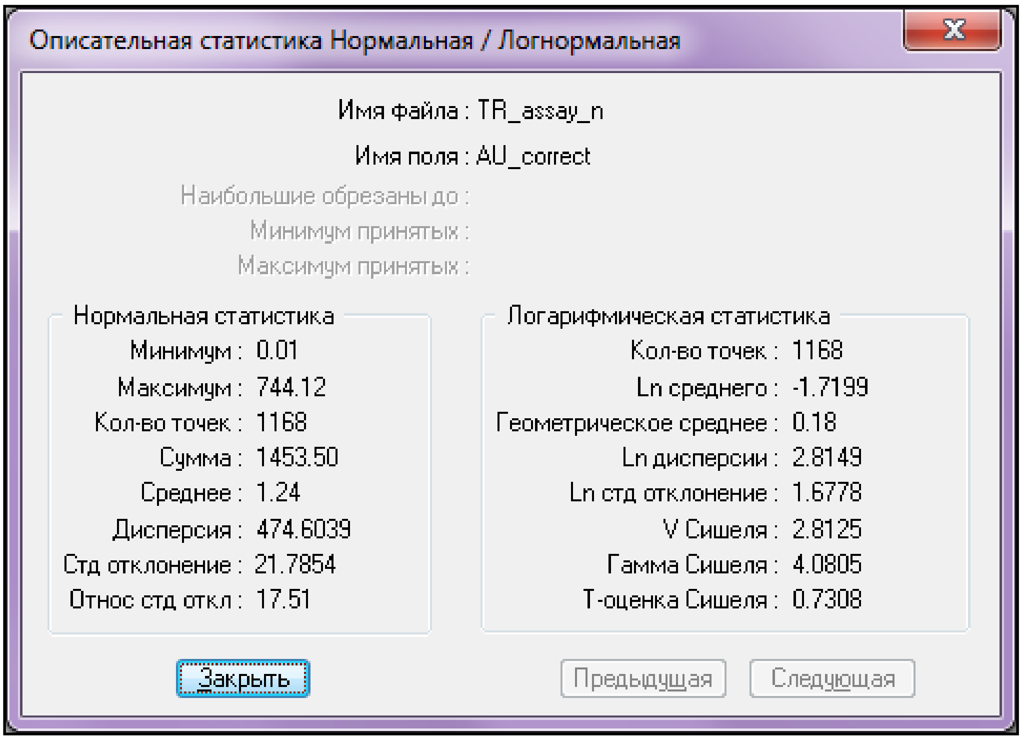


Figure 9-7 Data on classic statistics (Median/Mode) for Au grades for boreholes

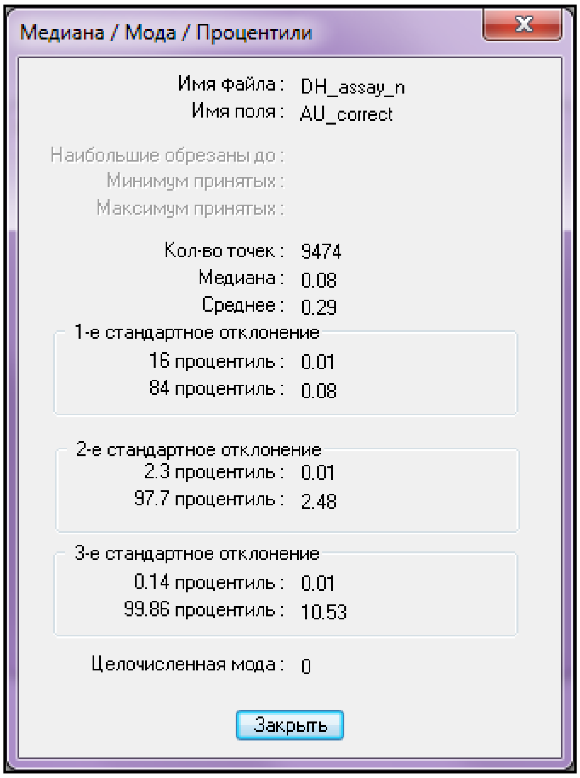


Figure 9-8 Data on classic statistics (Median/Mode) for Au grades for trenches

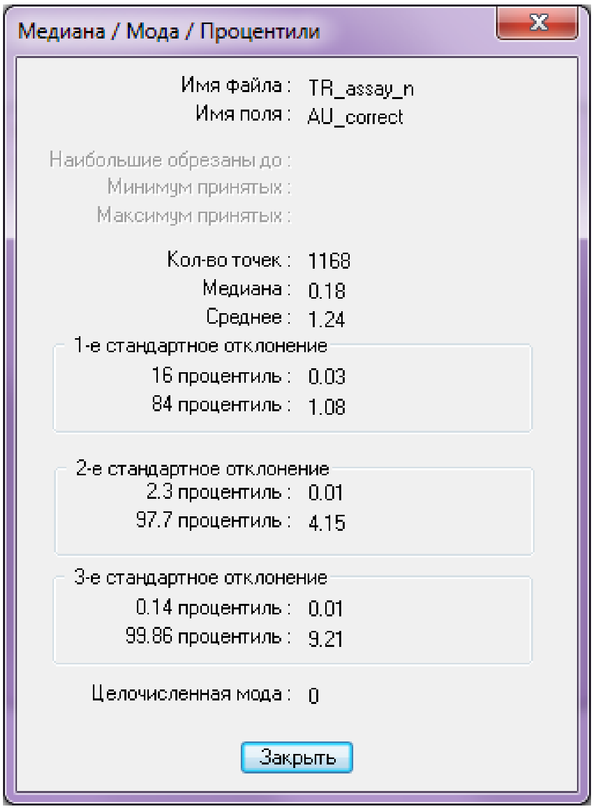


Figure 9-9 Histogram of Au grades in boreholes

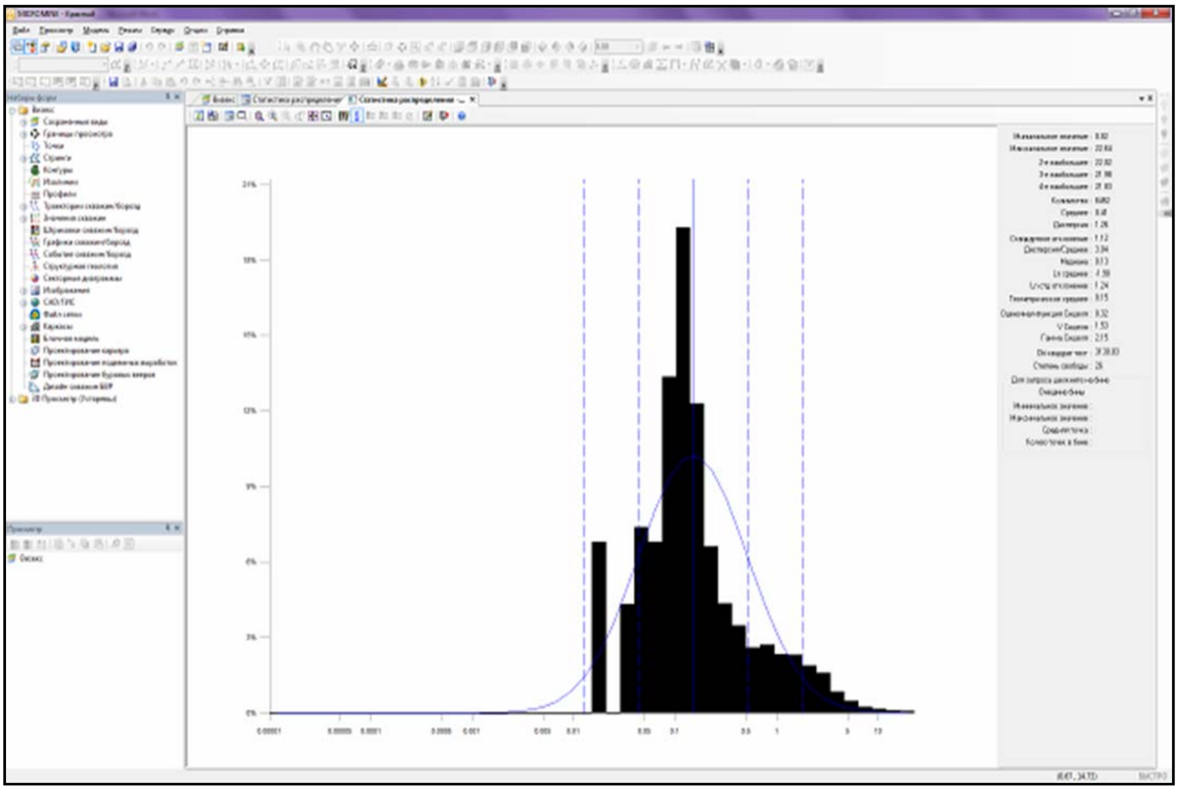


Figure 9-10 Probability diagram of Au grades in boreholes

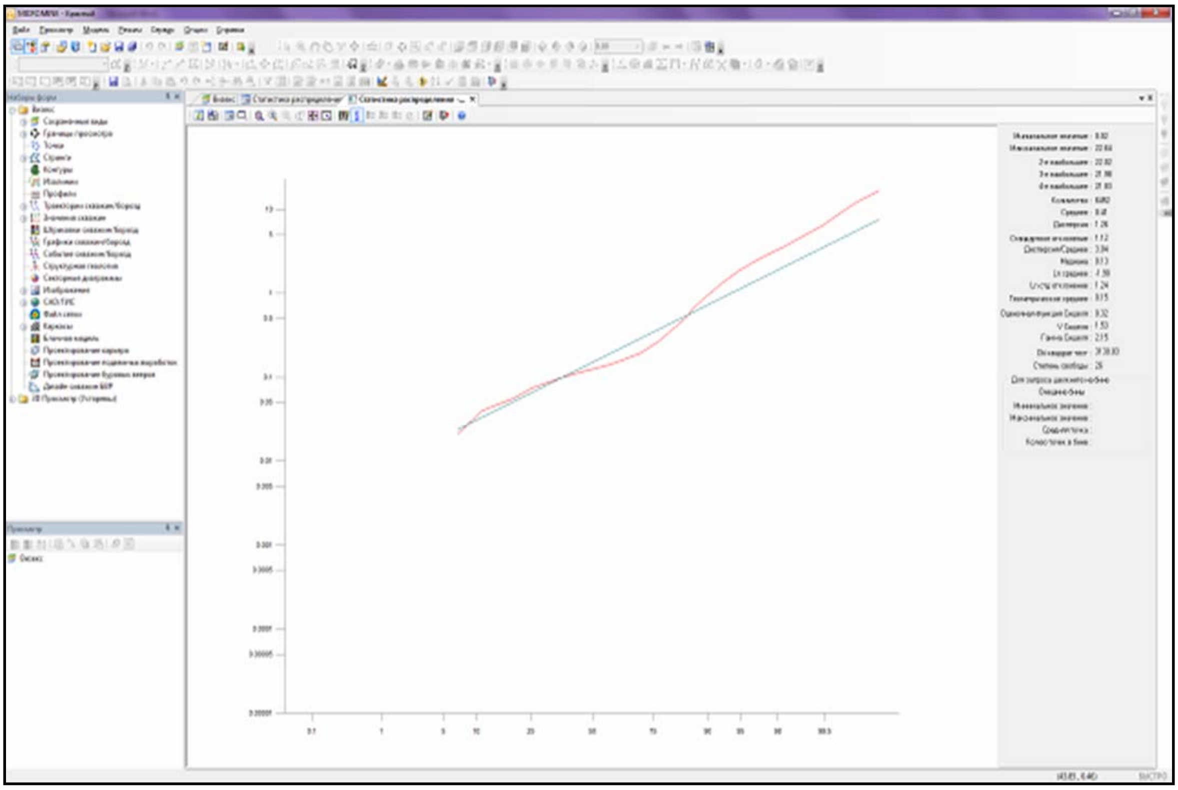


Figure 9-11 Histogram of Au grades in trenches

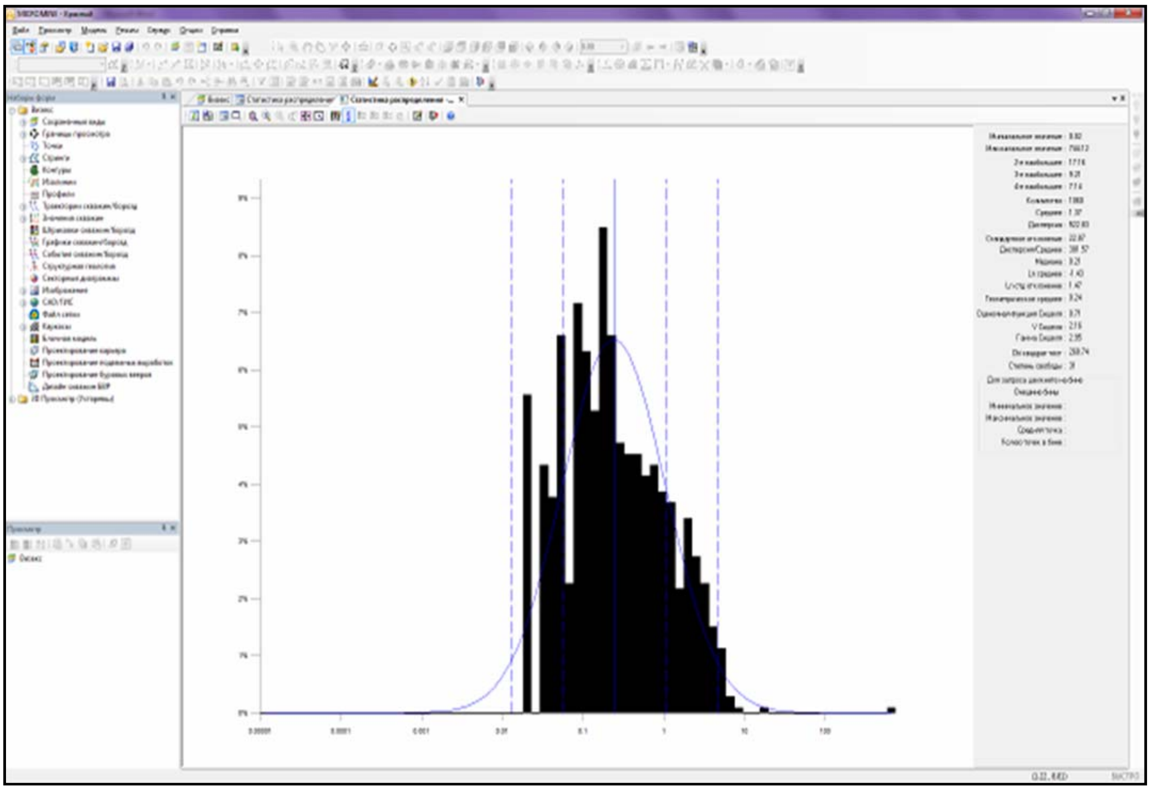


Figure 9-12 Probability diagram of Au grades in trenches

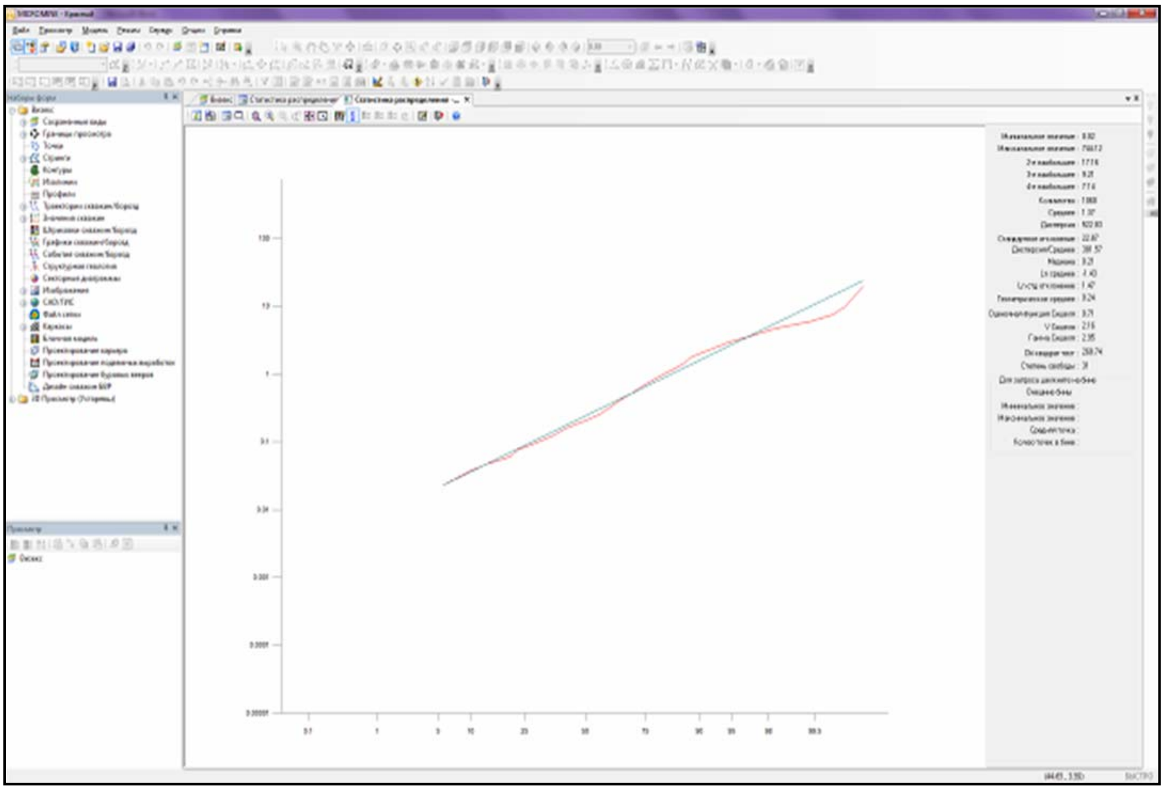


Table 9-11 Summary table of classic statistics data on Au grades

	Number of Samples	Minimum	Maximum	Mean	Coefficient of Variation	Median	Variation	Standard Deviation
For all workings	10642	0.01	744.12	0.39	18.61	0.09	52.9362	7.2757
For boreholes	9474	0.01	22.64	0.29	3.31	0.08	0.8947	0.9459
For trenches	1168	0.01	744.12	1.24	17.51	0.18	474.6039	21.7854

The table shows that the grades according to channel sampling are higher than those according to core sampling.

Channel sampling is also characterized by high variability.

9.4. Mineralization Interpretation

Geological modeling of mineralization is commonly conducted on the basis of natural indicators, such as natural cutoff grade of the mineralization and geological boundaries. In this case the received model will reflect natural distribution of mineralization which remains unchanged until the analytical base is replenished or the geologist's idea on the mineralization morphology is changed. Such mineralization model will be independent from any economic parameters and the latter can be applied to the model at any moment. For example, if the deposit is modeled on the basis of the natural gold cutoff grade 0.2 g/t, the model received can be used for reserve estimation with any Au cutoff grade above 0.2 g/t.

Natural Au cutoff grades 0.2 g/t identified by means of classic statistic analysis were used for ore body modeling.

Digital interpretation of the Krasnoye occurrence ore bodies was conducted in the following way:

Prior to the beginning of interpretation, new string files were created for the contours of mineralization of corresponding bodies (process *File* | *New*). The string file for the interpreted body was uploaded to *Vizex (Vizex/Strings*), and a corresponding section was opened, using one of the forms of *Display Contours.* Interpretation was conducted by means of reference of the interpreted contour points to borehole or trench sampling intervals, i.e. the created string of the ore body contour was not in the section plane but connected corresponding boreholes and/or trenches.

The orientation of sections for interpretation was chosen mainly across the strike of the main ore bodies.

The general methodology of interpretation (digitization) of ore bodies looks as follows:

1 A new string file is created in Vizex: an upload form Strings was opened, the string window was right-clicked and the option New was selected. The structure of the new file is standard, containing point and string coordinate fields. A separate string file was created for each ore body.

2 Prior to interpretation, it is recommended to install an Edit Request for a new string upload: Options/Vizex/Request editing prior to new string upload? In this case the sequence of interpretation can be controlled while creating new strings and string parameters can be entered to the file created. It is recommended to enter values for each new string.

3 Strings were initially referenced to sampling intervals, and then additional points were added to them.

4 All digitized strings were closed.
5 If the ore body was not continued, the string was closed at a half of the distance between boreholes (ore trenches).

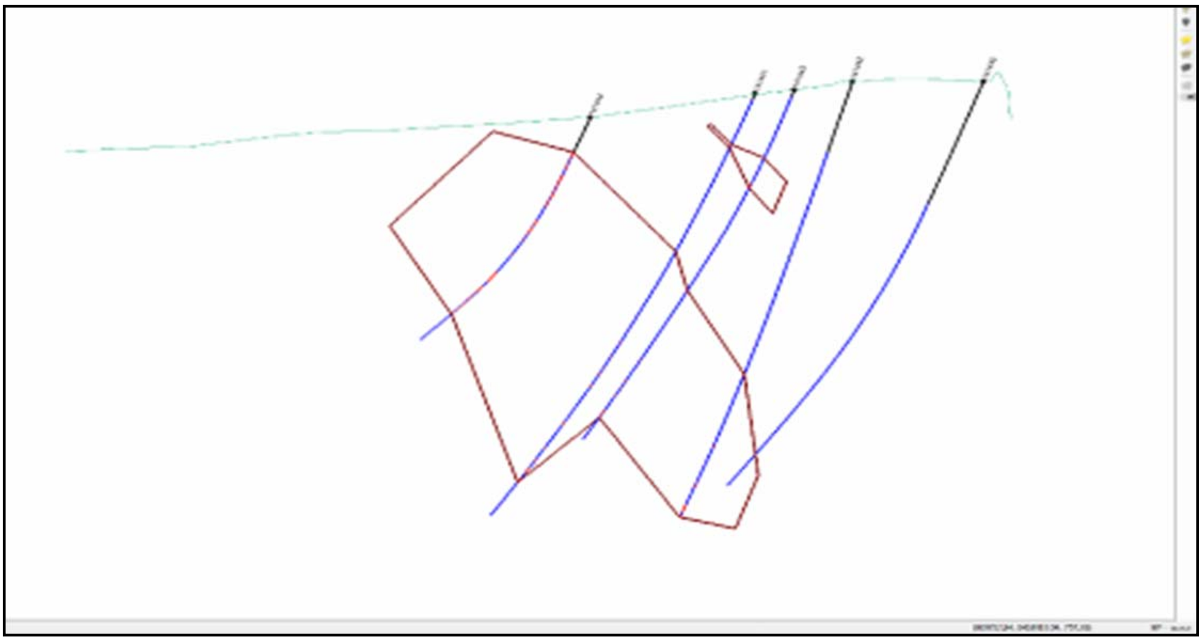
Mineralization interpretation was conducted separately on the basis of sections, for each available profile with boreholes (trenches). A string file interprenation_NEW.STR was created for interpretation lines (strings). Intervals with insignificant length with the grade below the cutoff grade inside the ore body were also included into this contour. These intervals were assumed with KP equal to zero (field KP_NEW).

While choosing the orientation of the strike (dipping angle) of the ore body, graphical materials provided by the Customer (geological sections) were taken into consideration.

According to the Customer's requirements, the results for boreholes on Profiles 16 and 22 were not included into the resource estimation. Moreover, Borehole 141450 was excluded from resource estimation. It was drilled in the east of the ore occurrence in Profile 43, and the Customer thinks that the driven profiles do not reflect the actual location of the zones of mineralization intersected in the central part of the deposit because they were driven to the north from mineralized zones. Mineralization intersected in Profile 16 reflects the ore bodies not related to the main deposit.

Therefore, in accordance with the customer's requirements, the resource estimation of the Krasny ore occurrence was limited by Profiles 26 and 35, with extrapolation of modeling results to a half of the distance between profiles.

Figure 9-13 Example of mineralization interpretation



9.5. Wireframing

Closed wireframe models (solids) of ore bodies and cavities were built using digitized strings. Therefore, the following wireframe models were built:

Table 9-12 Ore body wireframes

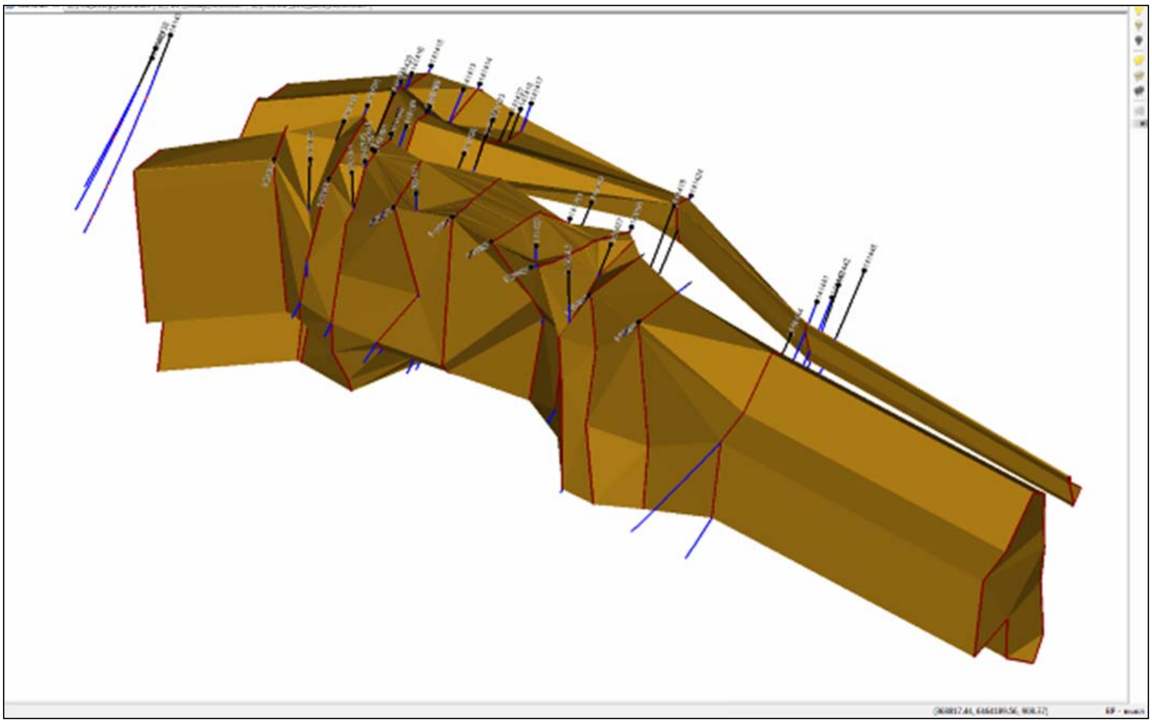
Type of Wireframe	Name of Wireframe	Volume, m^3
Ore_NEW	upper	57905153.32
Ore_NEW	lower	3903248.41
ALL	TOTAL	61808401.73

A standard MICROMINE methodology was used for creating digital wireframes. Wireframes are technically built in the following way:

- Start Vizex.
- Upload the string file: Upload | Strings.
- Start building a digital wireframe: Wireframe/New.
- Edit binding lines: Add.
- Build a wireframe: Build a wireframe.
- Validate the wireframe for continuity, intersections and open sites: Right-click/Validate. Validation colours as well as optimization options are chosen in options: Compress points and/or Compress triangles.

Closed wireframe models of ore bodies were built by means of successive creation of a continuous surface between strings, from section to section. Boreholes and trenches were uploaded to a 3D environment for building wireframe models. This allowed to check which borehole intervals fall into the wireframe model. The last string in a row was projected to a half of the distance between sections for creation of the ore body pinch-out. Then this string was closed.

Figure 9-14 Wireframing of ore bodies

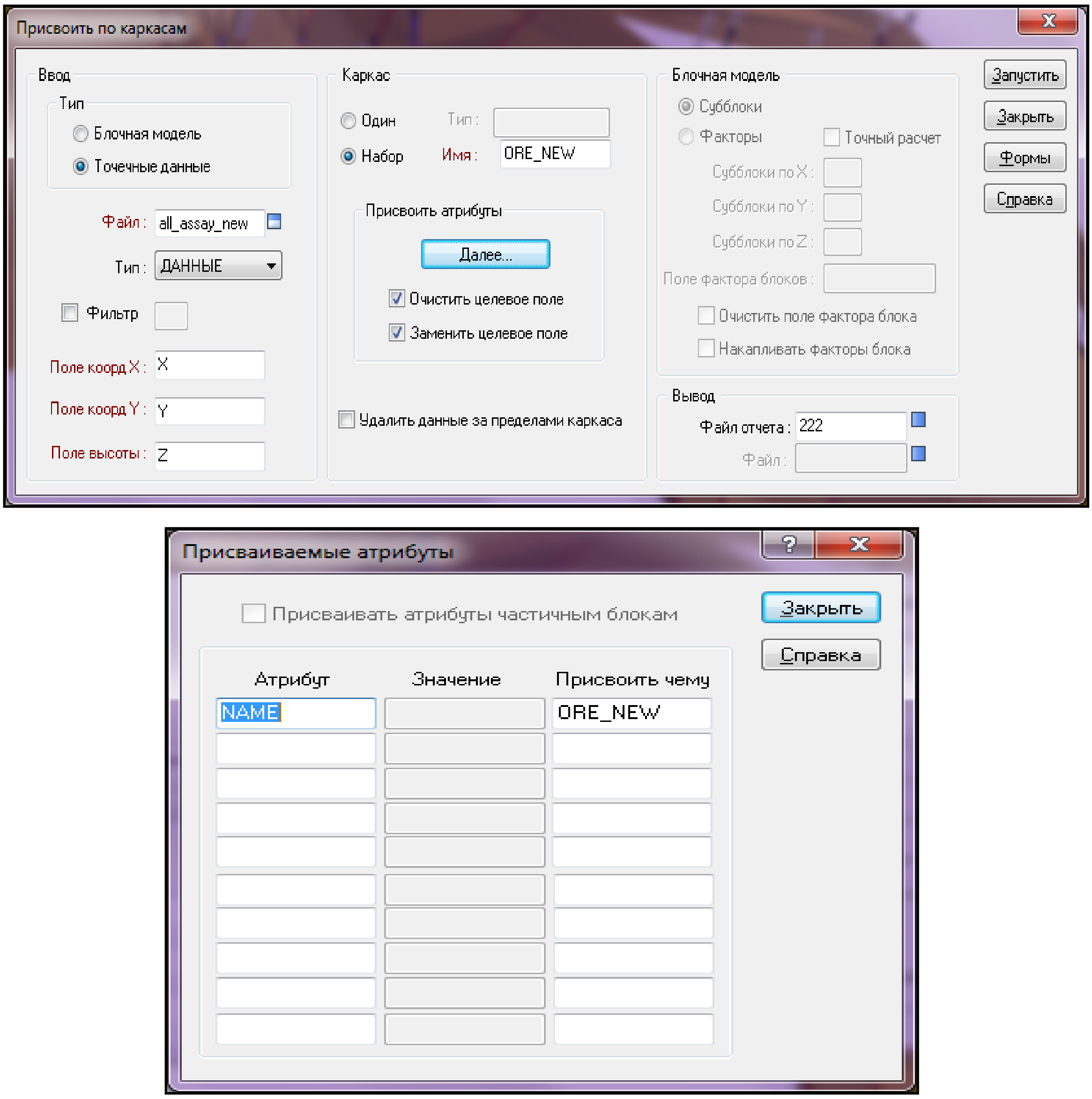


In some cases the ore body could be interpreted for only a half of the section. In such cases adjoining sections were used for full interpretation of these sections. This means that in Vizex the view window was enlarged so that the ore intervals could be seen in adjoining sections. After that strings in the interpreted section were digitized but without reference to boreholes in adjoining sections.

9.6. Sample Selection

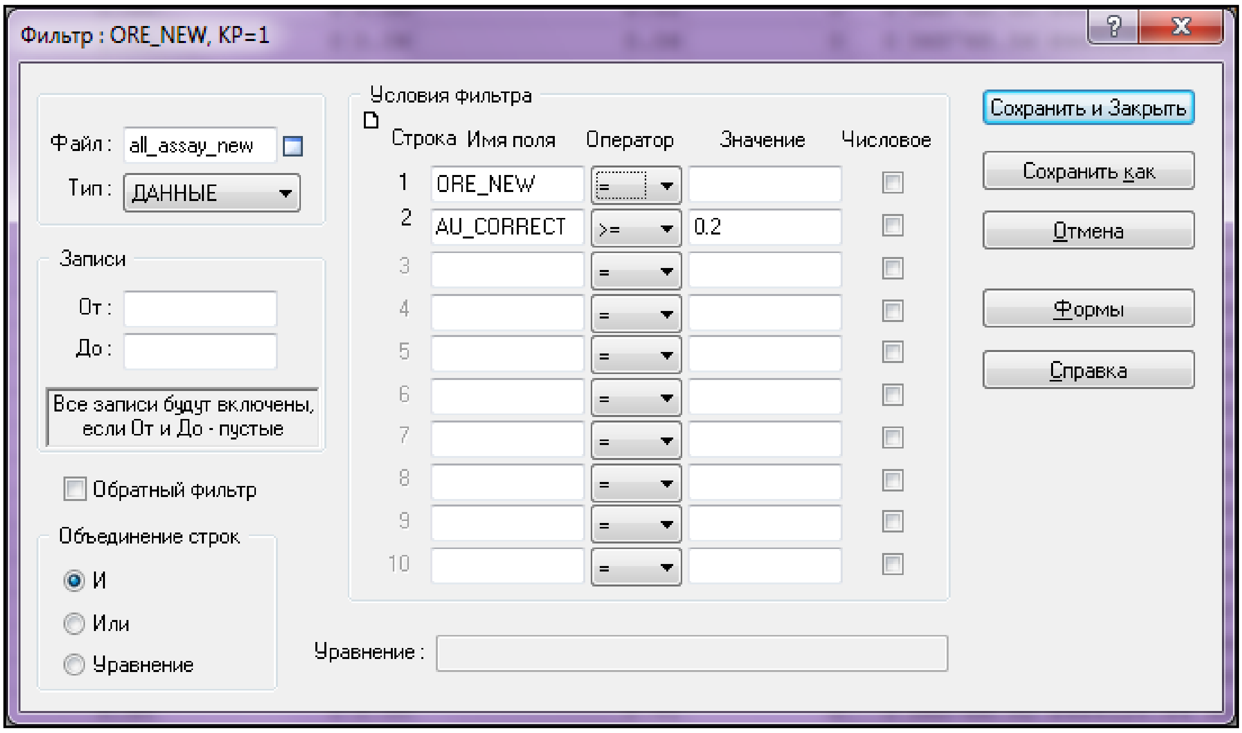
Data selection for wireframes is a standard procedure guaranteeing usage of correct samples for classic statistic and geostatistic analyses, and for the grade interpolation process. Samples were selected on the basis of closed wireframes of mineralized zones. At that, the names of wireframes (ore bodies) coded were entered to the field ORE_NEW.

Figure 9-15 Coding of samples with wireframe models of ore bodies



After all borehole and trench samples were coded, all samples and strings were uploaded for validation to Vizex; visual check as well as check with the help of filters were conducted. No errors were detected.

Figure 9-16 Filter for coding verification



9.7. Statistic Analysis of Sampling Data Selected

Repeated statistic analysis was conducted with the purpose of evaluation of statistic parameters for useful component grades. This data was later used for validation of interpolated average grades in block models.

The processes described in Chapter 10 were used for reception of statistic data (Statistics | Descriptive | Normal/Log and Statistics | Descriptive | Median/Moda).

Samples which fell into wireframes of the ore bodies of the deposit were statistically analyzed.

Figure 9-17 Data of classic statistics Normal/Log for all workings in ore body wireframes

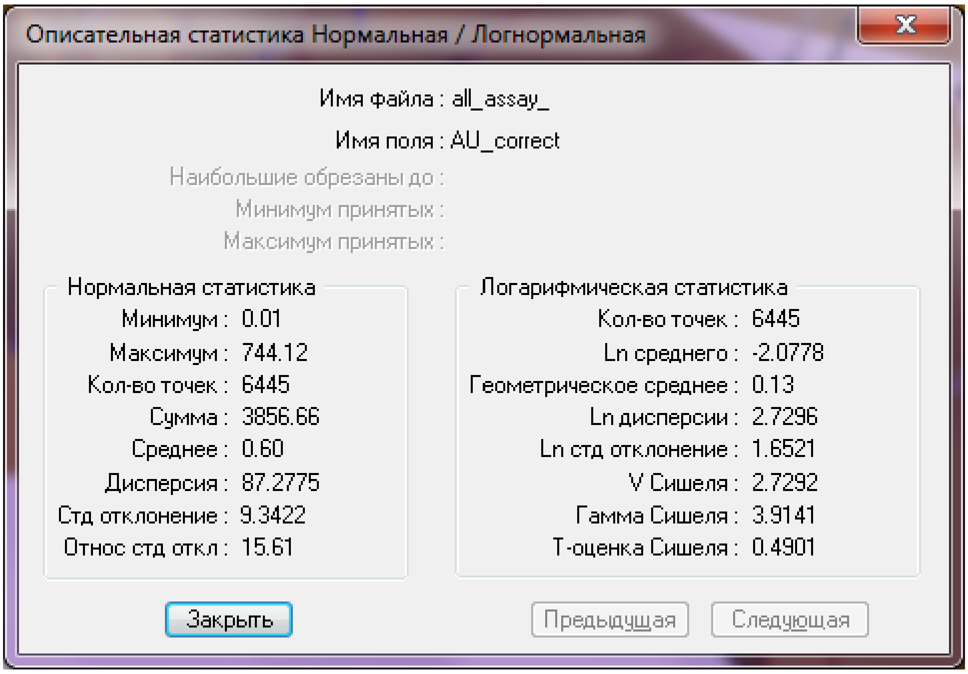


Figure 9-18 Data of classic statistics Median/Mode for all workings in ore body wireframes

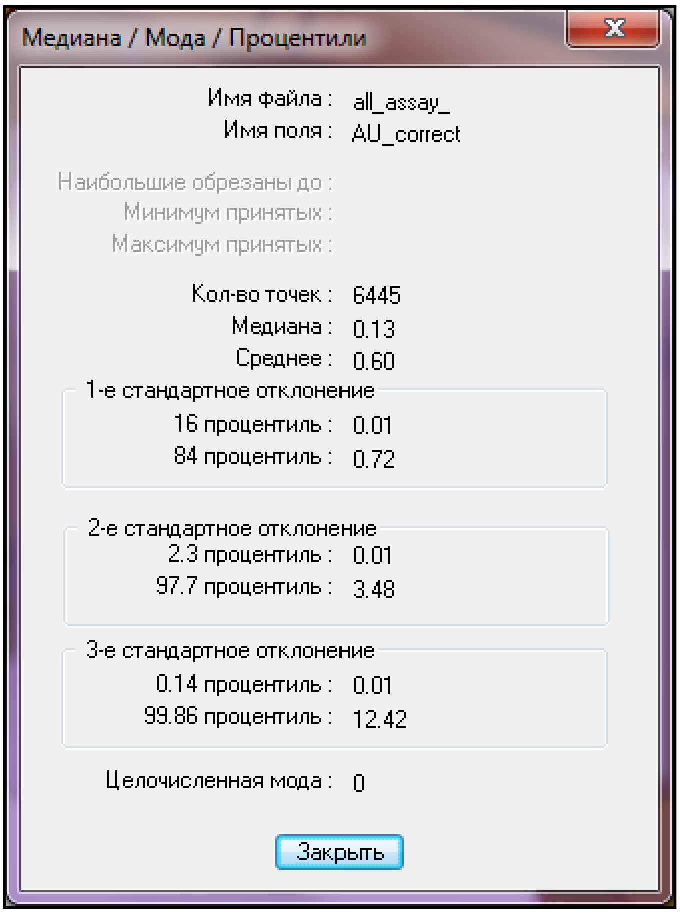


Figure 9-19 Histogram of Au grade distribution by all workings within ore body wireframes

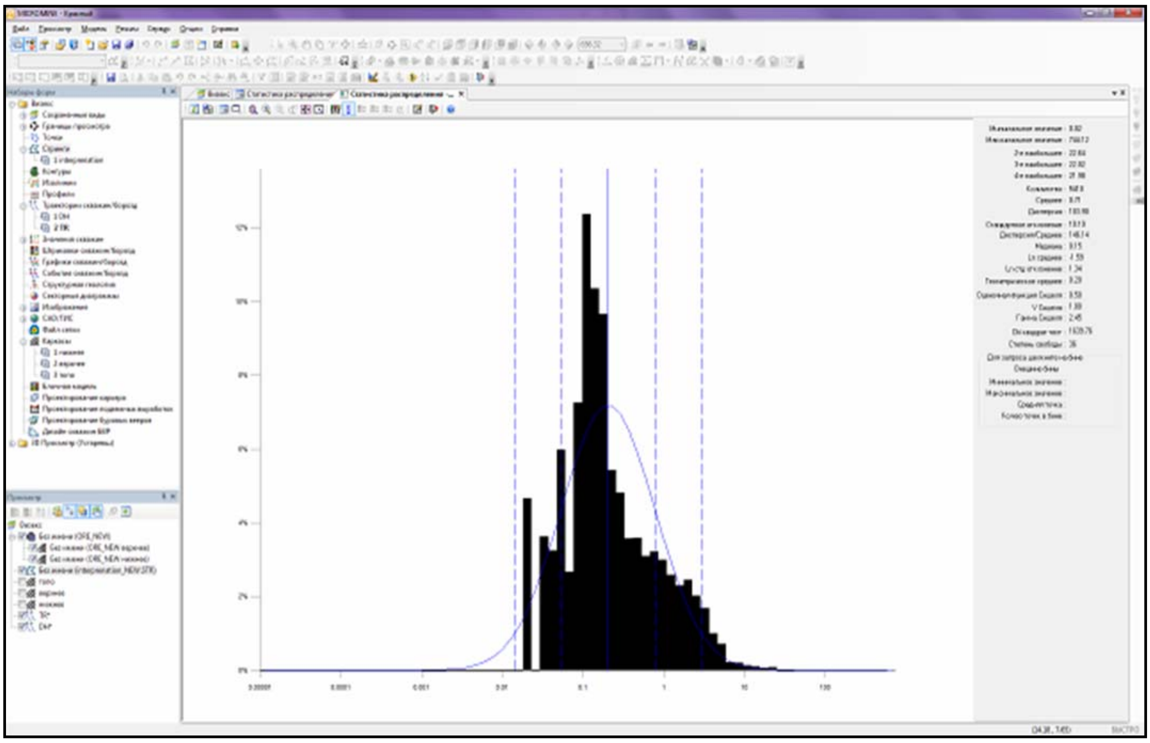


Figure 9-20 Probability diagram of Au grades for all workings within ore body wireframes

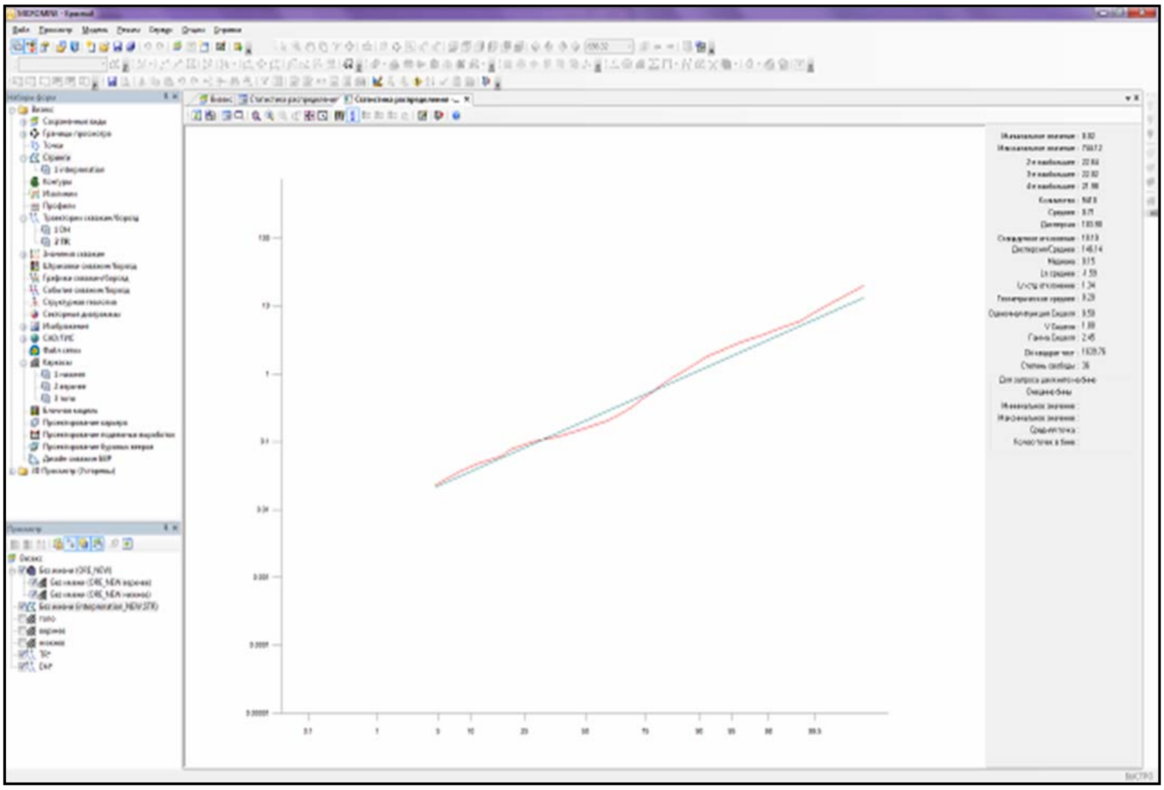


Figure 9-21 Data of classic statistics Normal/Log for Au grades in boreholes within ore body wireframes

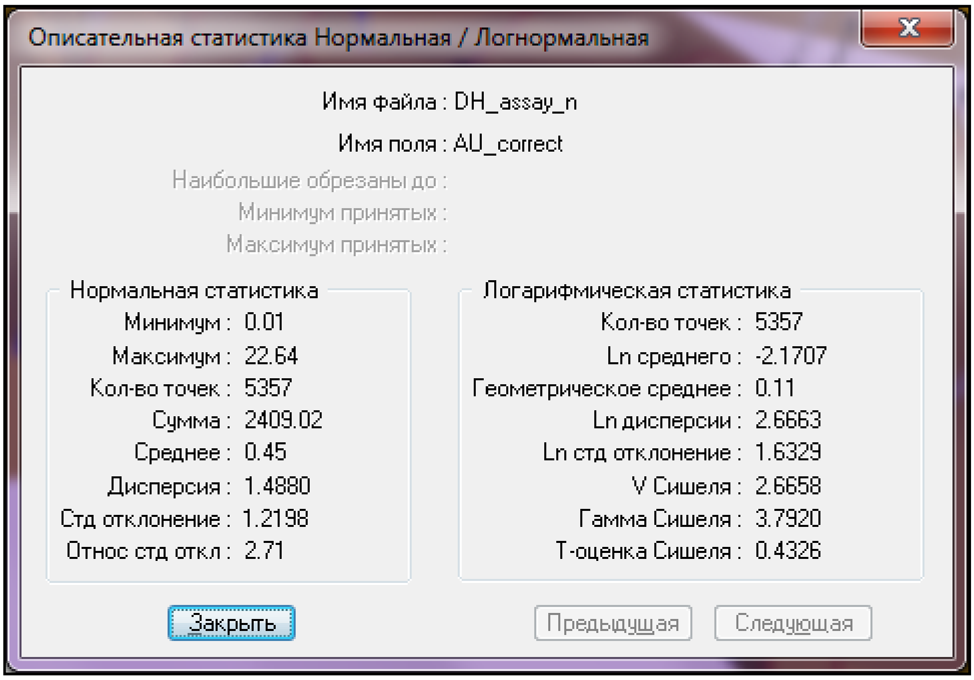


Figure 9-22 Data of classic statistics Median/Moda for boreholes within ore body wireframes

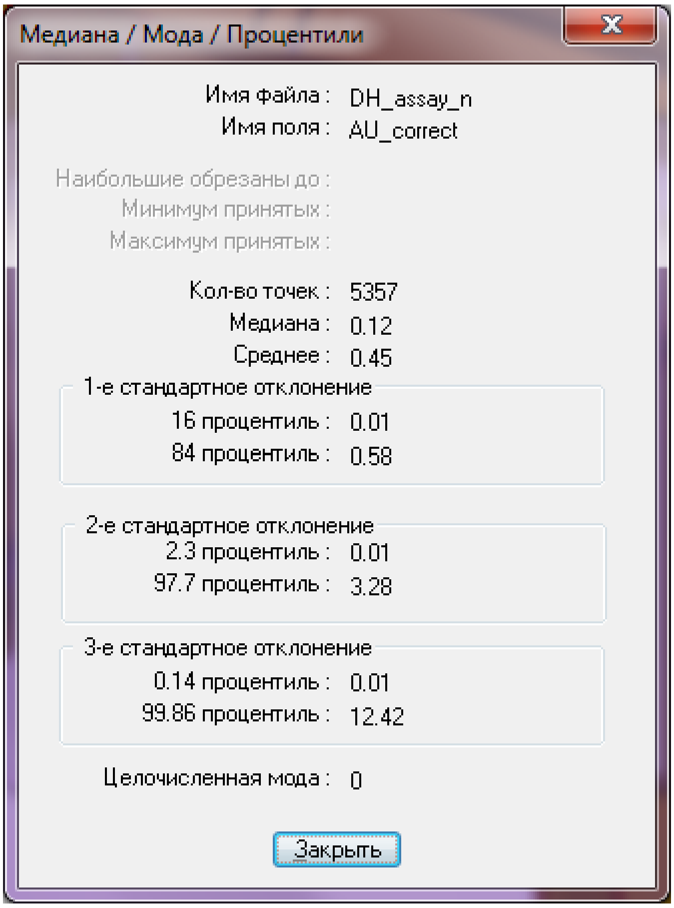


Figure 9-23 Histogram of Au grade distribution in boreholes within ore body wireframes

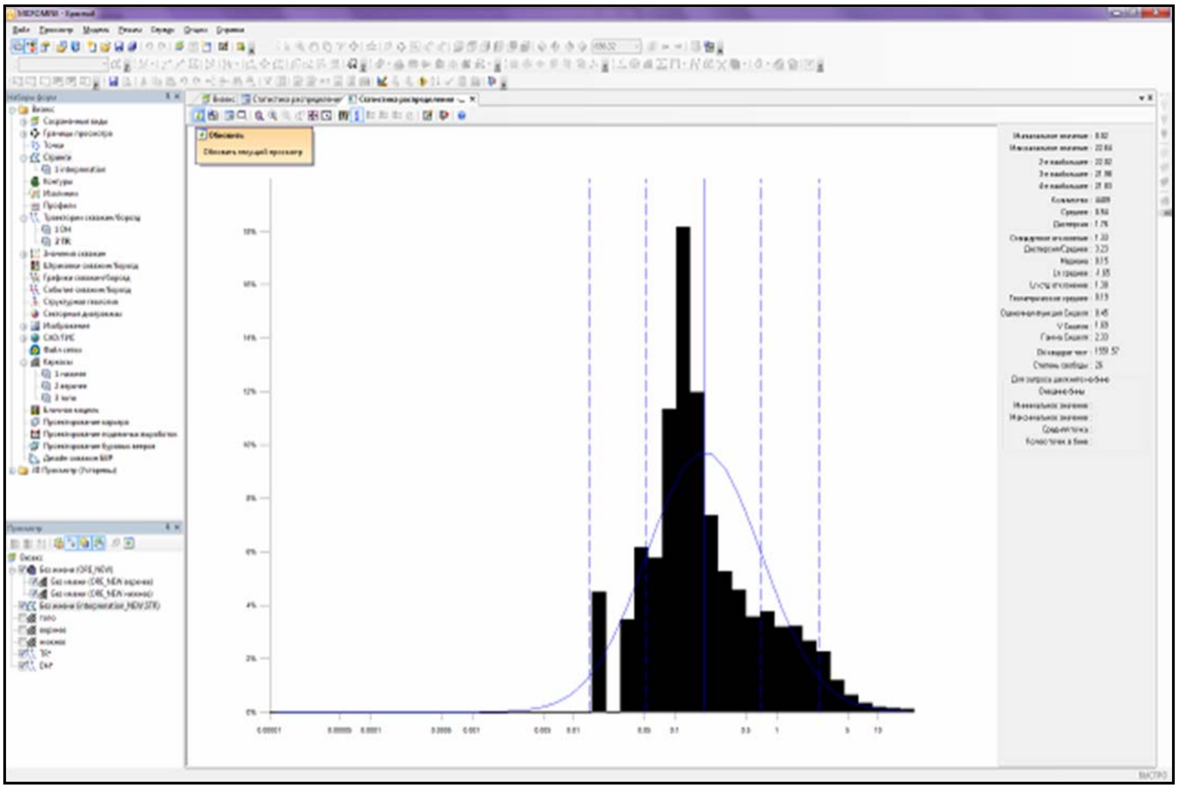


Figure 9-24 Probability diagram of Au grades for boreholes within ore body wireframes

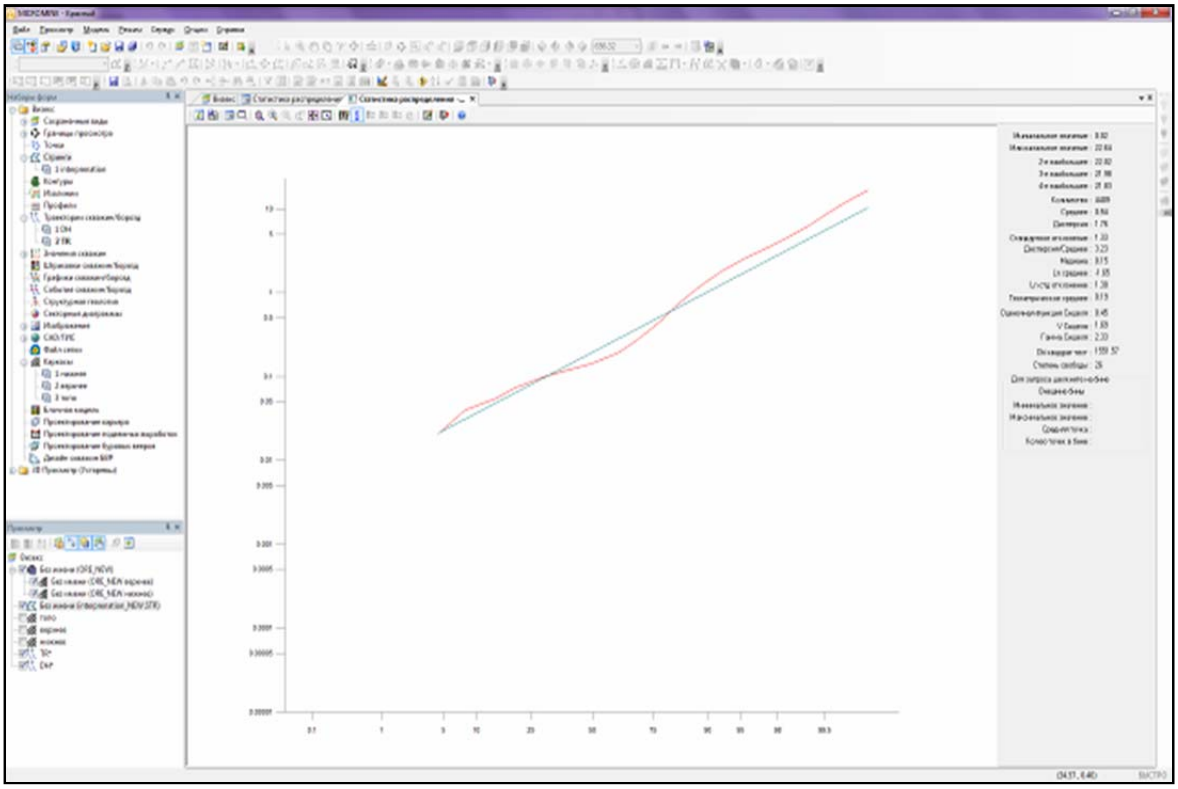


Figure 9-25 Data of classic statistics Normal/Log for Au grades in trenches within ore body wireframes

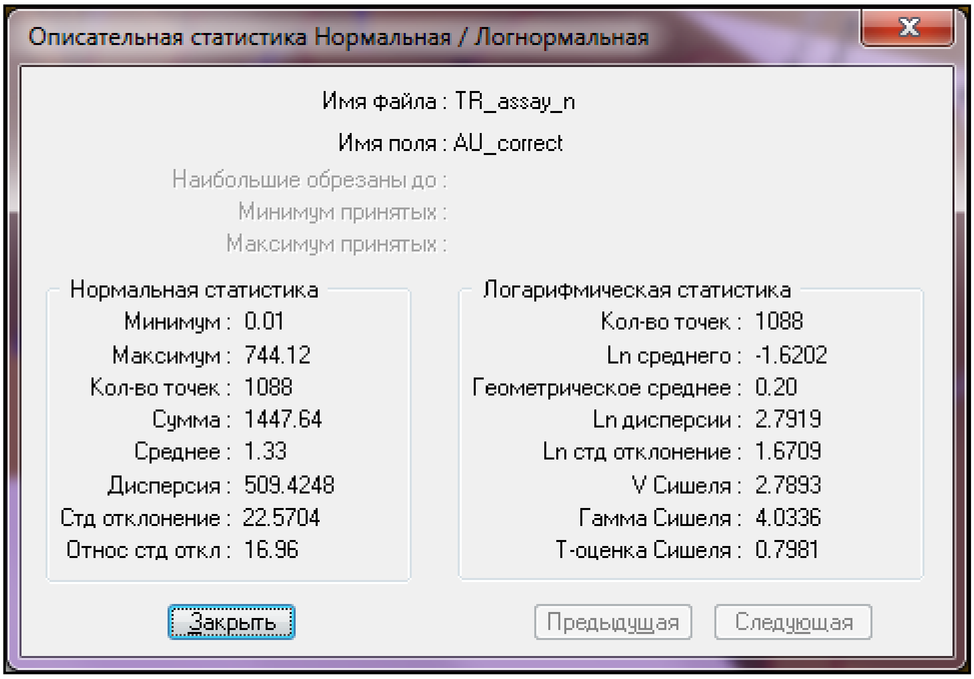


Figure 9-26 Data of classic statistics Median/moda for trenches within ore body wireframes

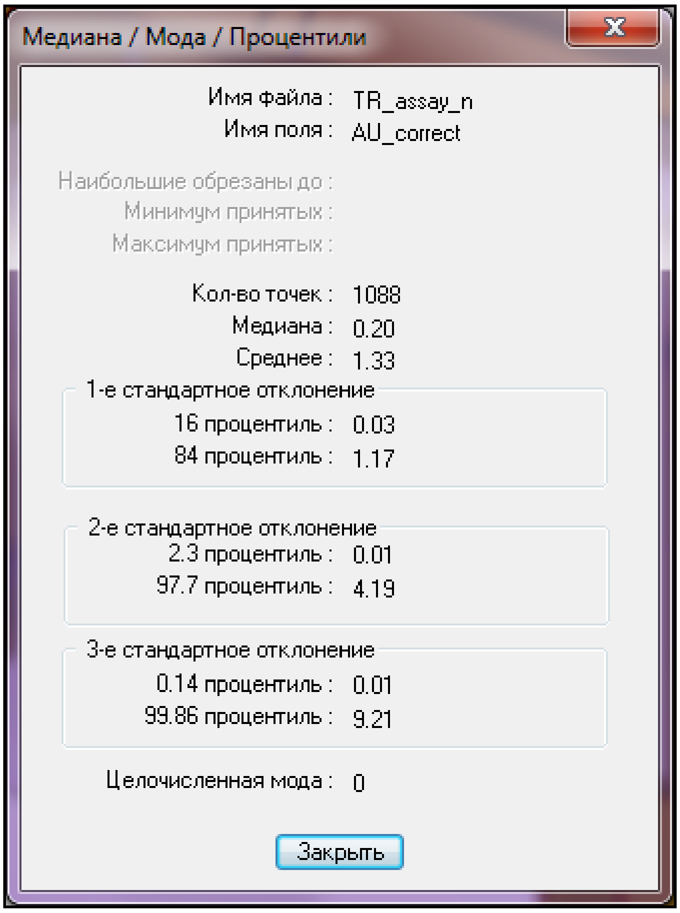


Figure 9-27 Histogram of Au grade distribution in trenches within ore body wireframes

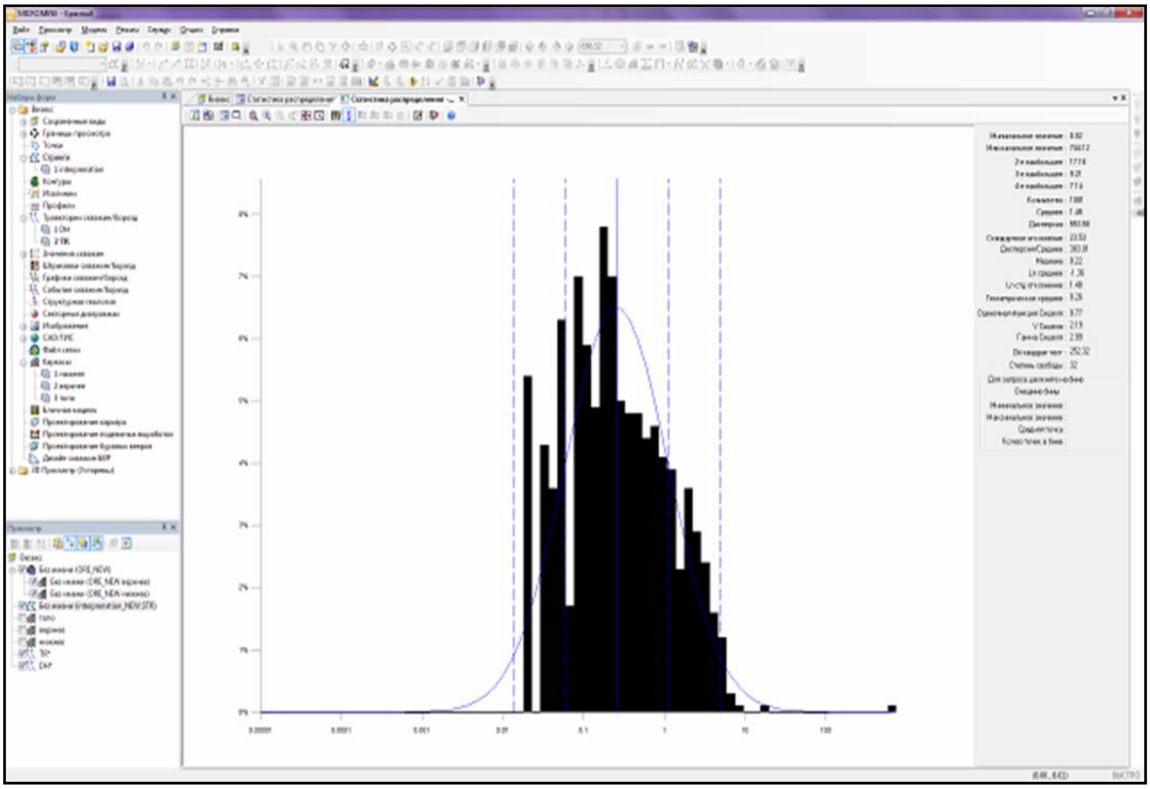


Figure 9-28 Probability diagram of Au grades for trenches within ore body wireframes

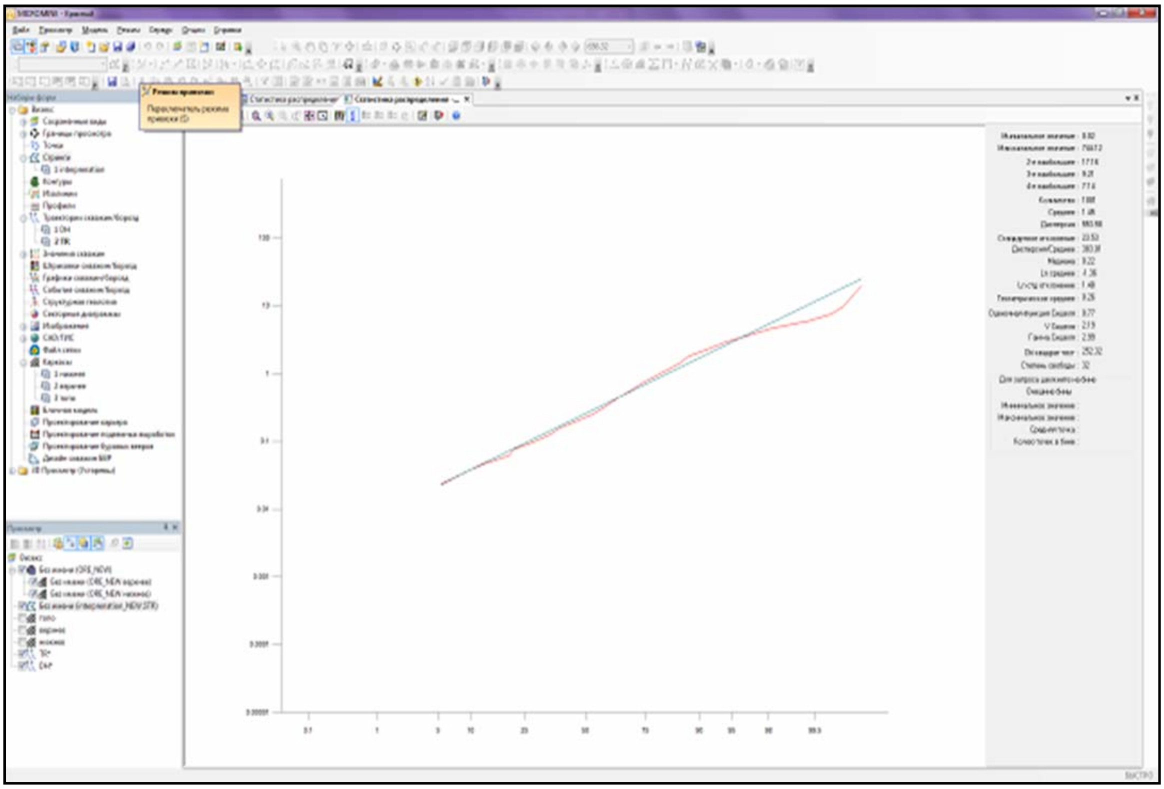


Figure 9-29 Data of classic statistics Normal/Log for Au grades in the upper ore body wireframe

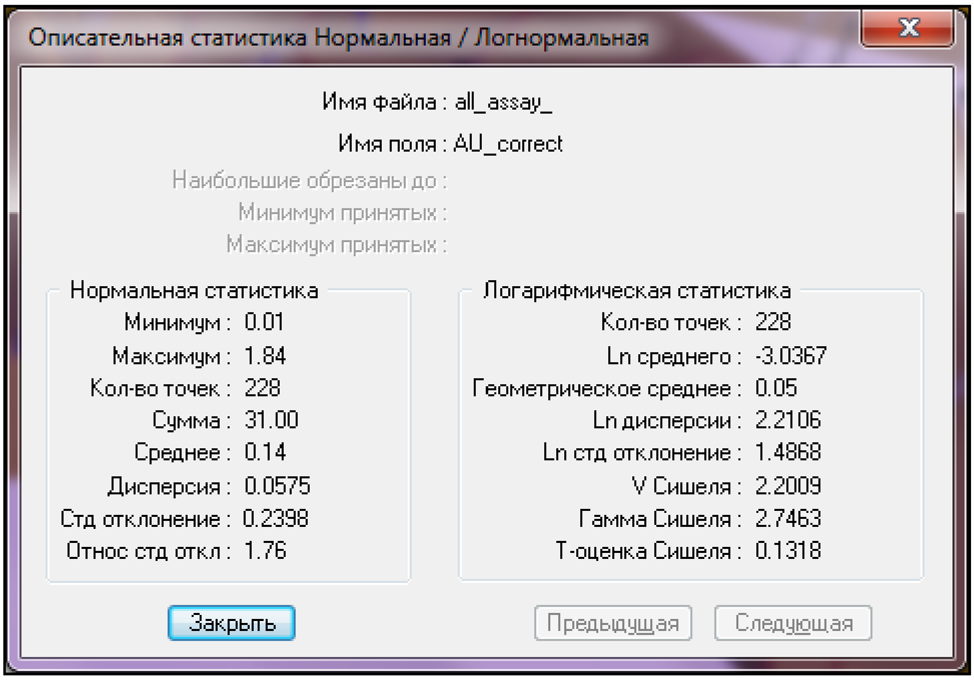


Figure 9-30 Data of classic statistics Median/Moda for Au grades in the upper ore body wireframe

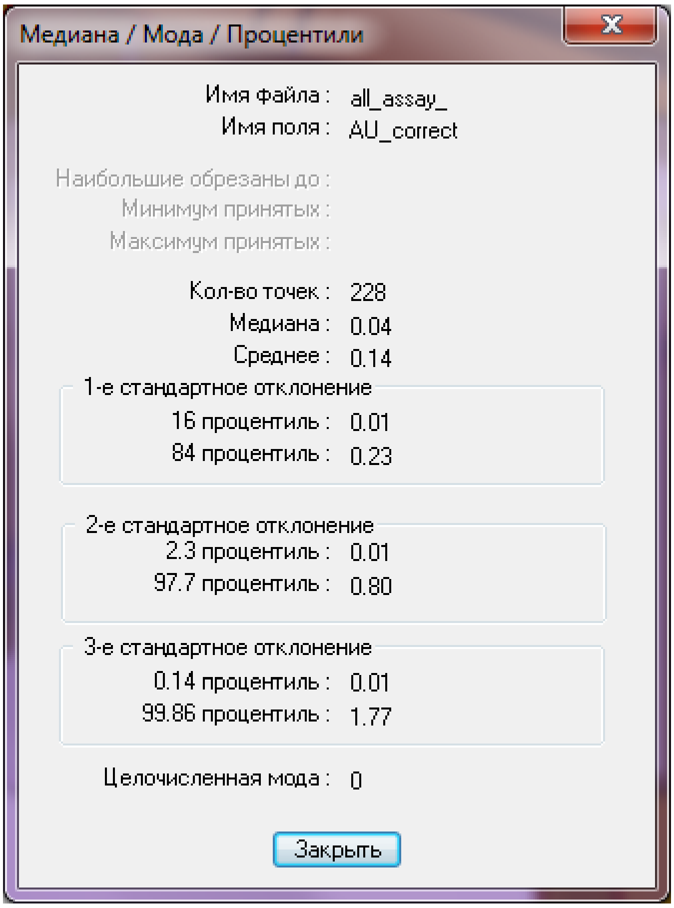


Figure 9-31 Histogram of Au grades distribution within the upper ore body wireframe

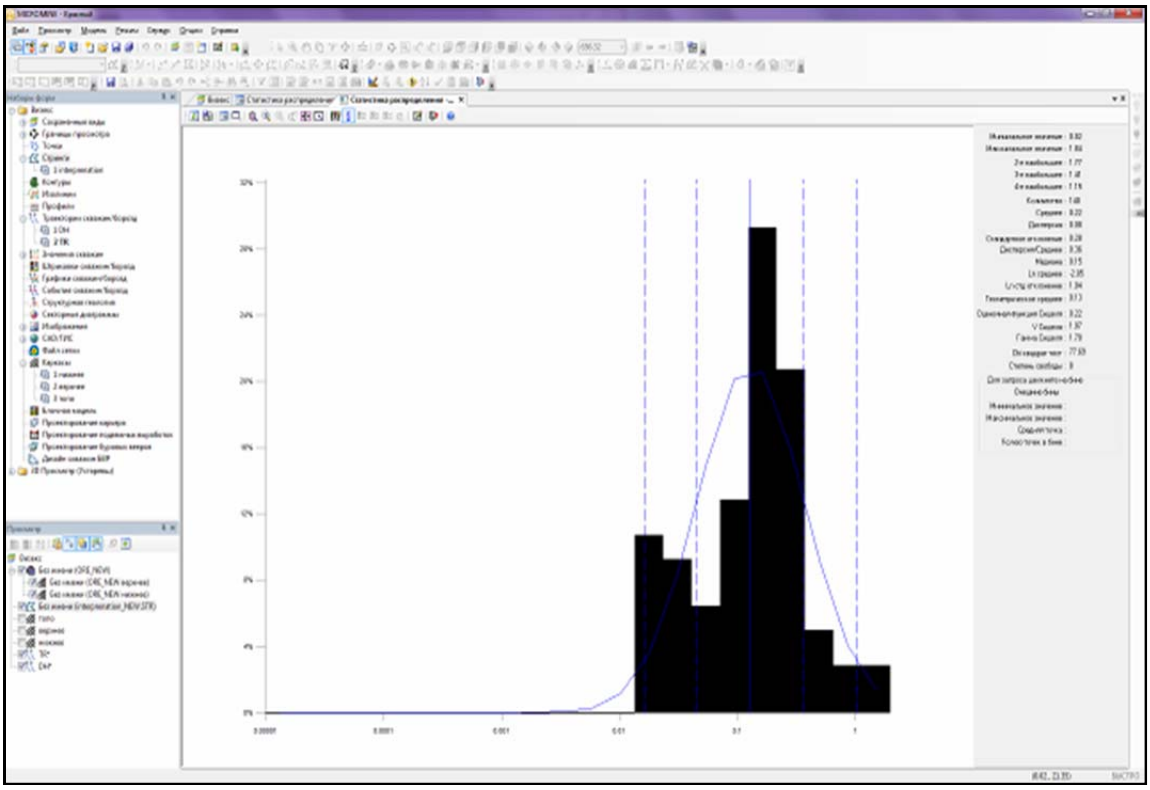


Figure 9-32 Probability diagram of Au grades within the upper ore body wireframe

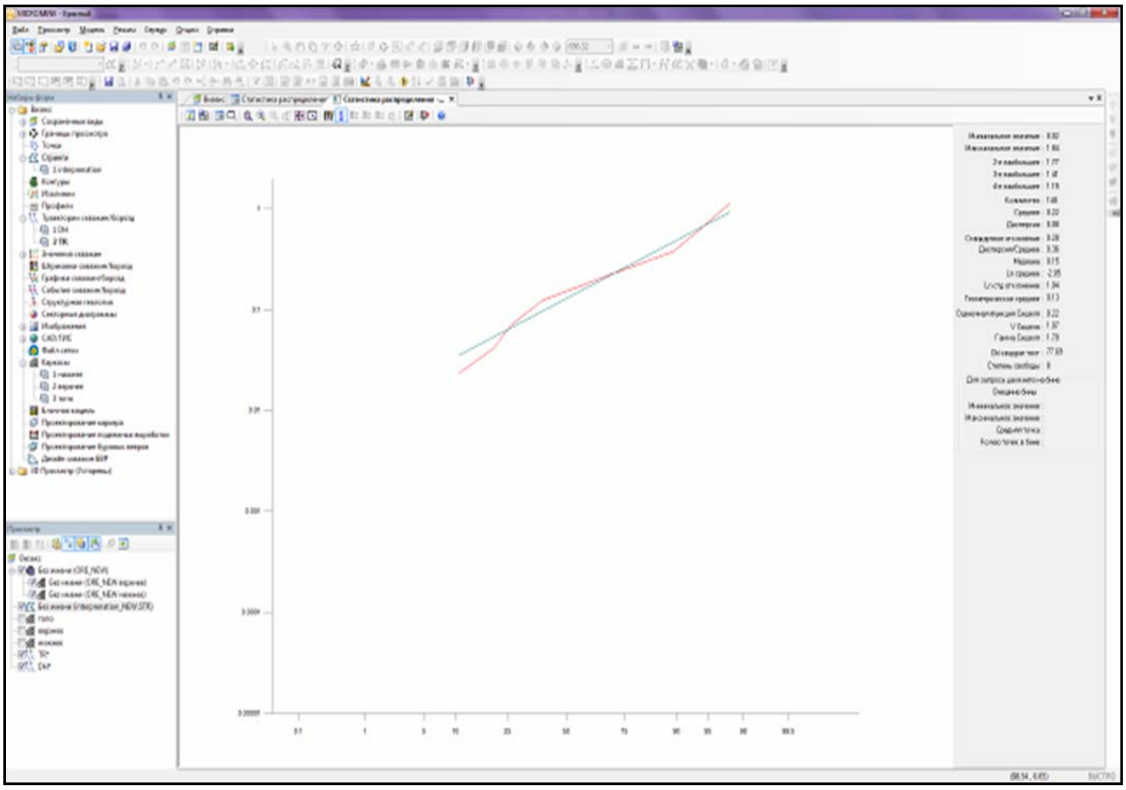


Figure 9-33 Data of classic statistics Normal/Log for Au grades in the lower ore body wireframe

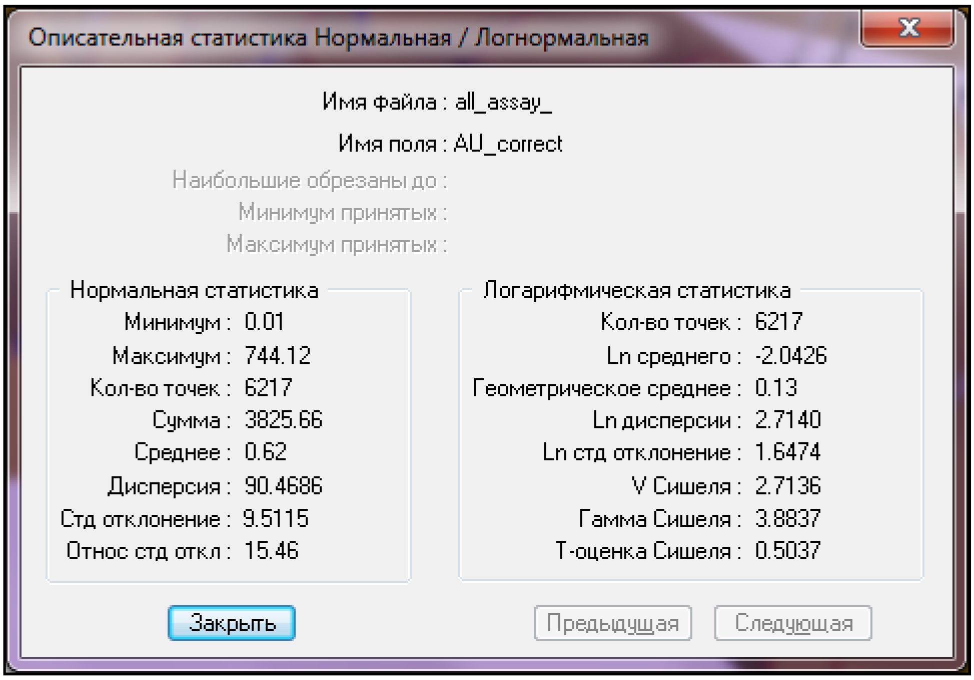


Figure 9-34 Data of classic statistics Median/Moda in the lower ore body wireframe

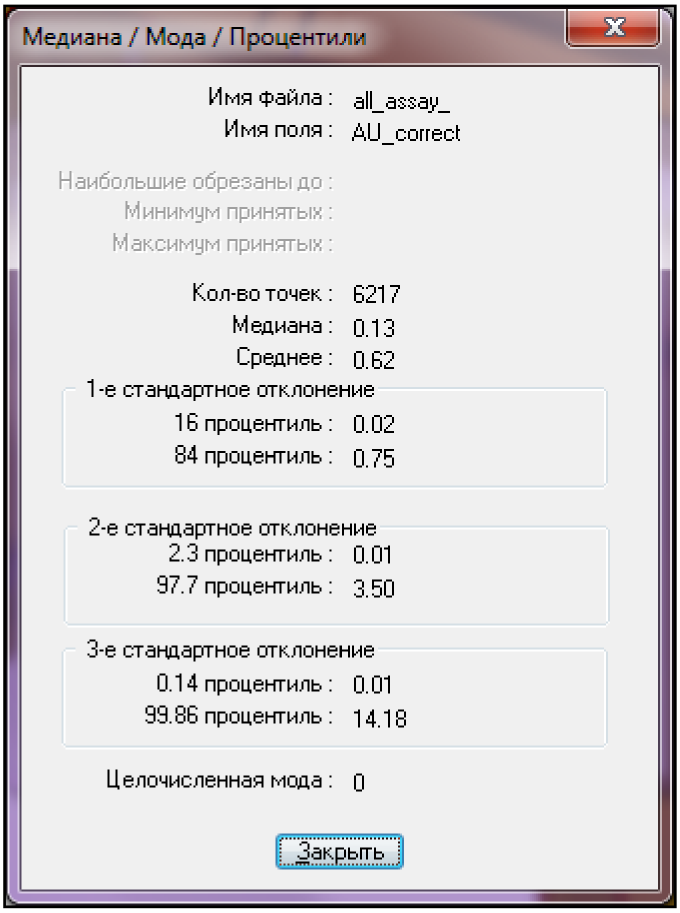


Figure 9-35 Histogram of Au grades distribution within the lower ore body wireframe

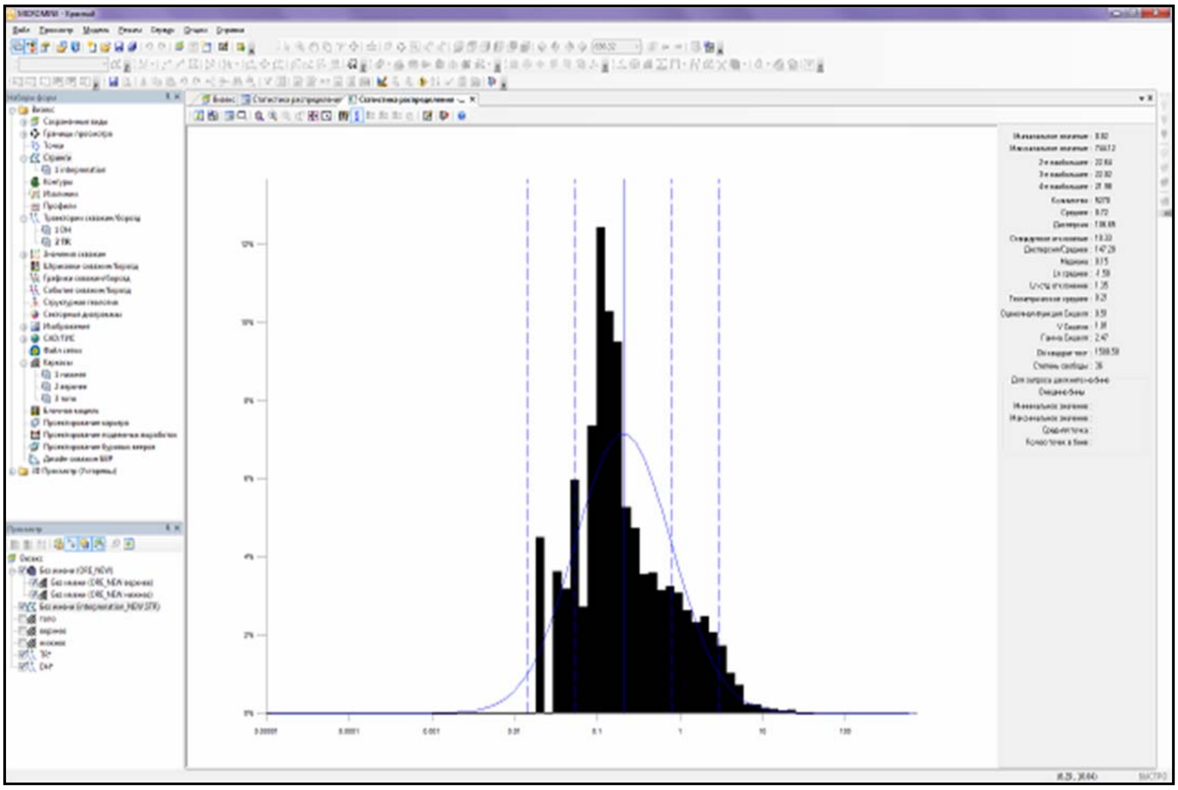


Figure 9-36 Probability diagram of Au grades within the lower ore body wireframe

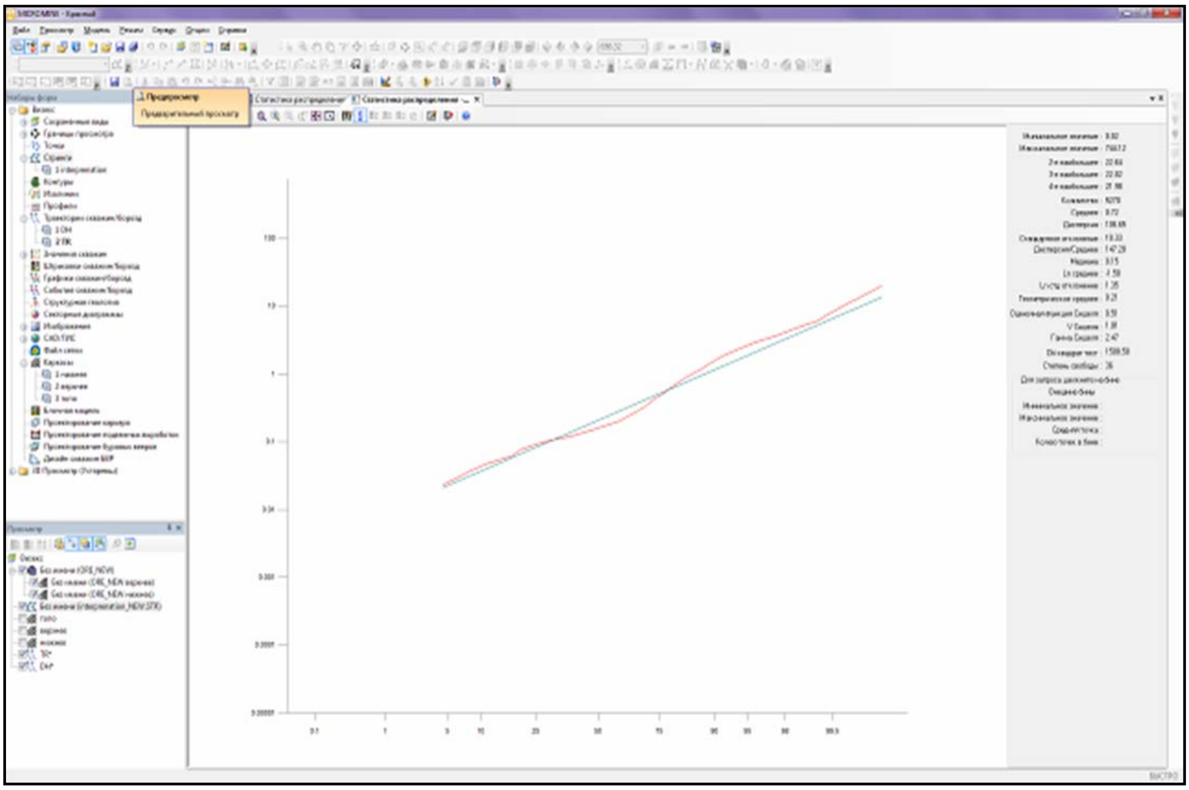


Figure 9-37 Data of classic statistics Normal/Log for Au grades in ore body wireframes (KP=1)

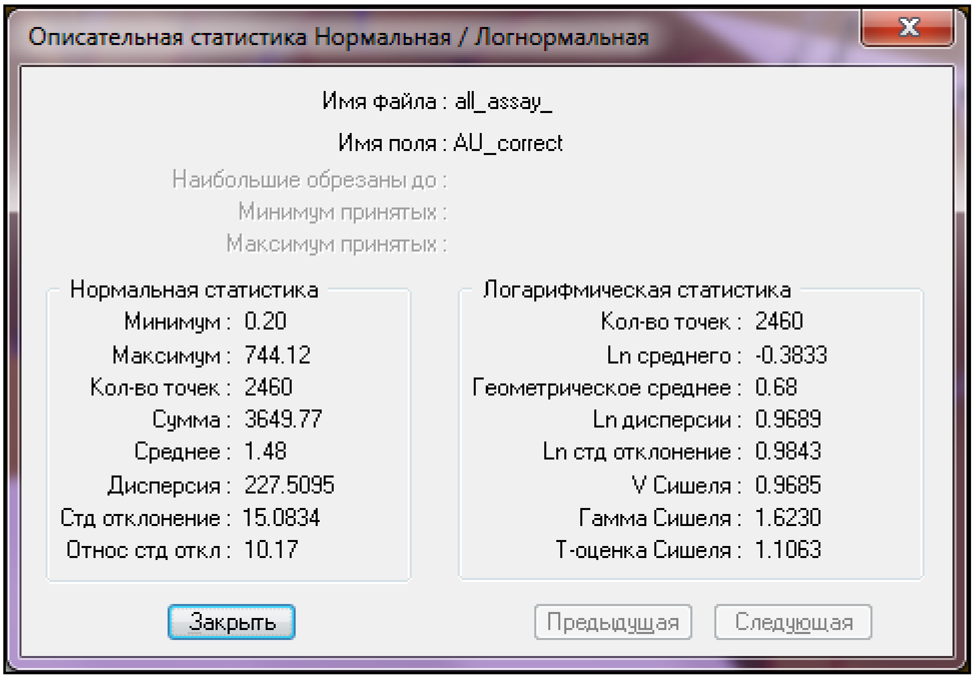


Figure 9-38 Data of classic statistics Median/Moda for Au grades in ore body wireframes (KP=1)

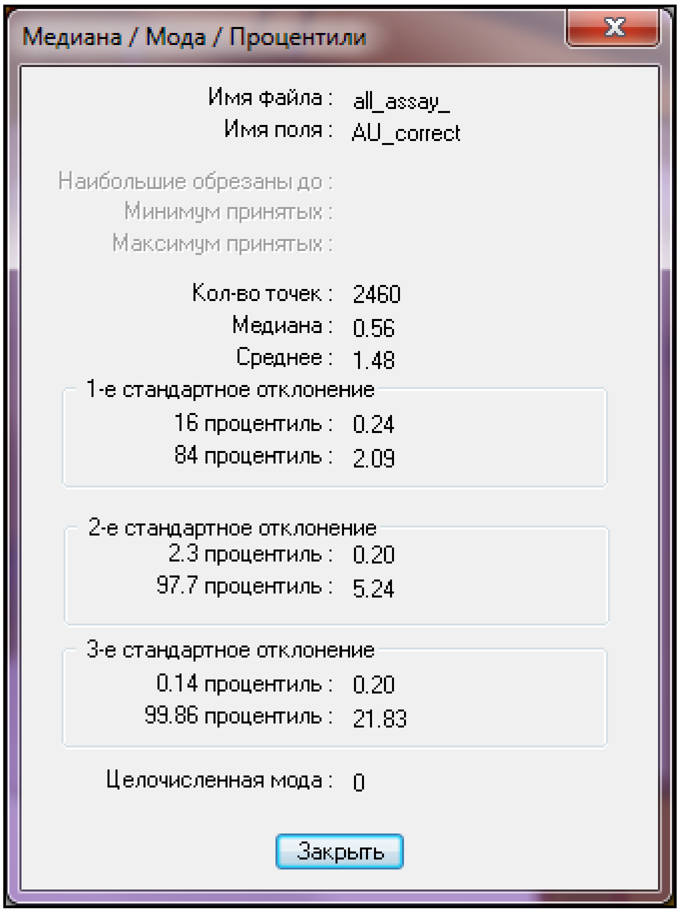


Figure 9-39 Histogram of Au grade distribution within ore body wireframes (KP=1)

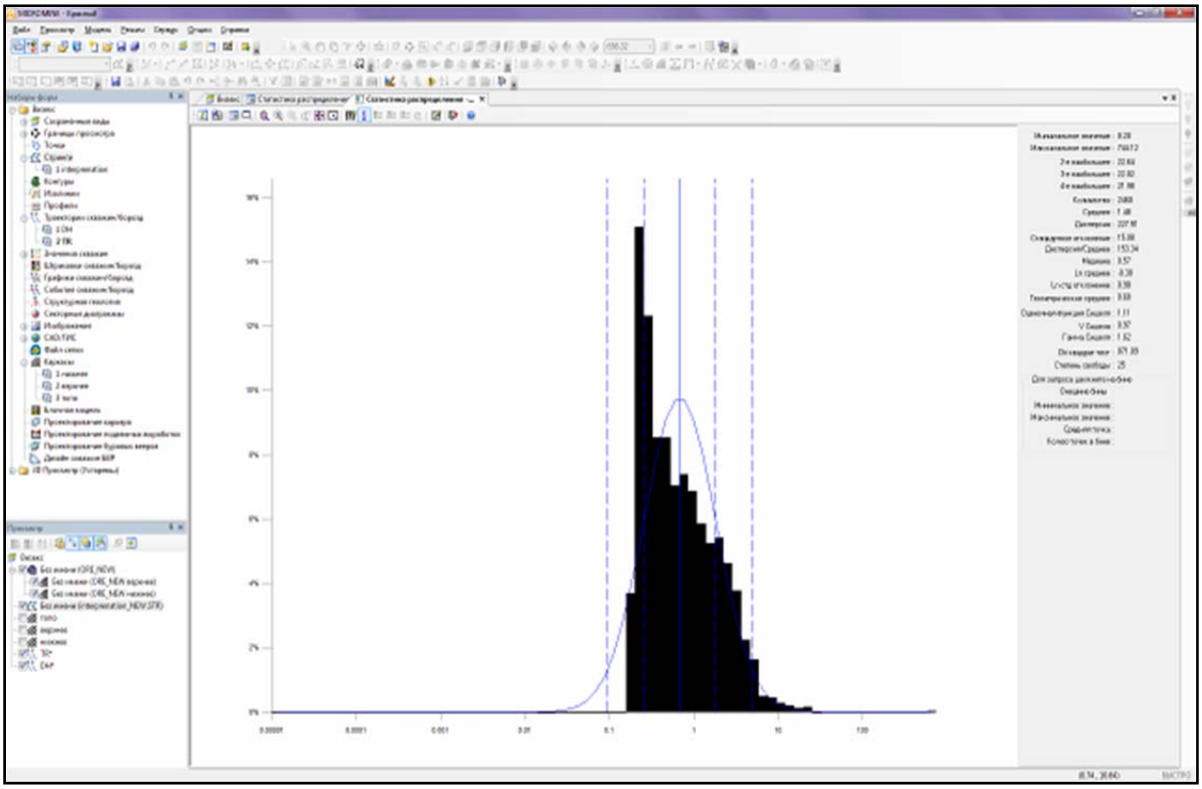


Figure 9-40 Probability diagram of Au grades within ore body wireframes (KP=1)

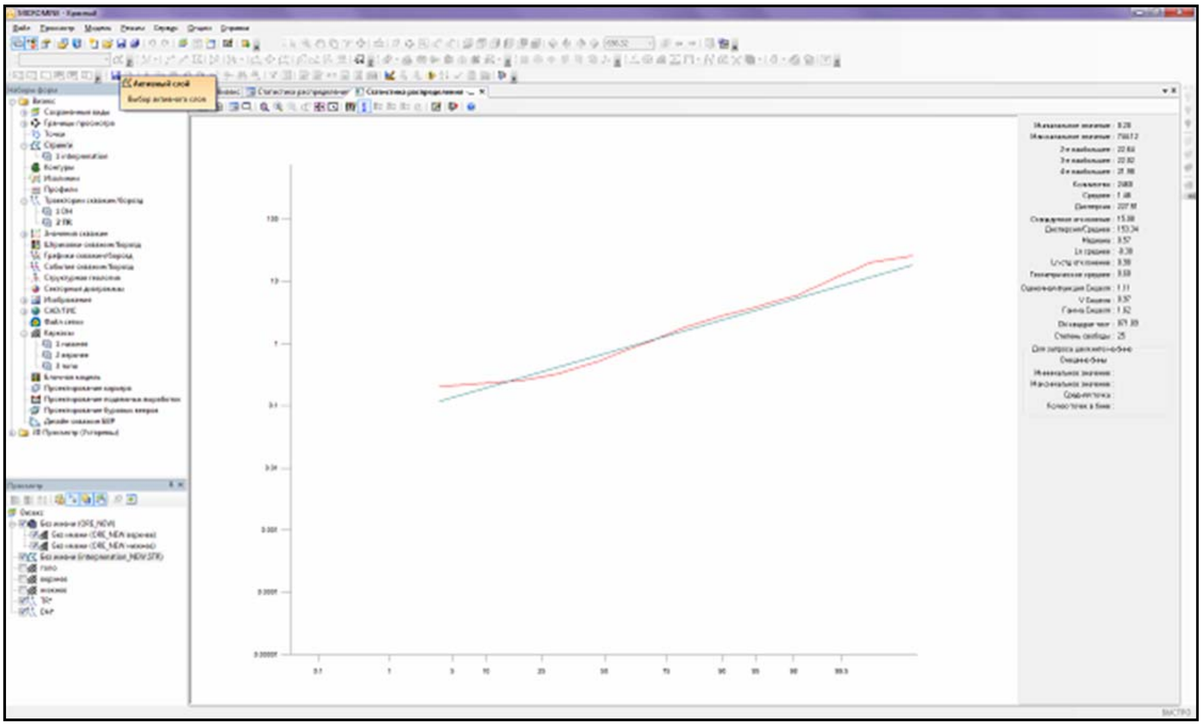


Table 9-13 Summary table of Au grade classic statistics data for ore body wireframes

Statistics Data	Number of Samples	Minimum	Maximum	Mean	Coefficient of Variation	Median	Variation	Standard Deviation
Within the contour of the upper ore body	228	0.01	1.84	0.14	1.76	0.04	0.0575	0.2398
Within the contour of the lower ore body	6217	0.01	744.12	0.62	15.46	0.13	90.4686	9.5115
Within the contour of the ore bodies as per all workings	6445	0.01	744.12	0.60	15.61	0.13	87.2775	9.3422
Within the contour of the ore bodies as per boreholes	5357	0.01	22.64	0.45	2.71	0.12	1.4880	1.2198
Within the contour of the ore bodies as per trenches	1088	0.01	744.12	1.33	16.96	0.20	509.4248	16.96
Within the contour of the ore body KP=1	2460	0.20	744.12	1.48	10.17	0.56	227.5095	15.0834

It follows from the figures and the table that higher grades are typical for channel samples of the lower ore body.

9.8. Restriction of Top Cut Grades

A classic scheme for determination of the values of gold top cut grades based on the statistic analysis of sampling data was applied in this work.

A cumulative probability diagram was built with a statistic method for gold grades within all ore bodies, using the process *Statistics* | *Distribution*, for determining the top cut grade limit.

Figure 9-41 Cumulative Probability diagram of Au grades within ore body contours

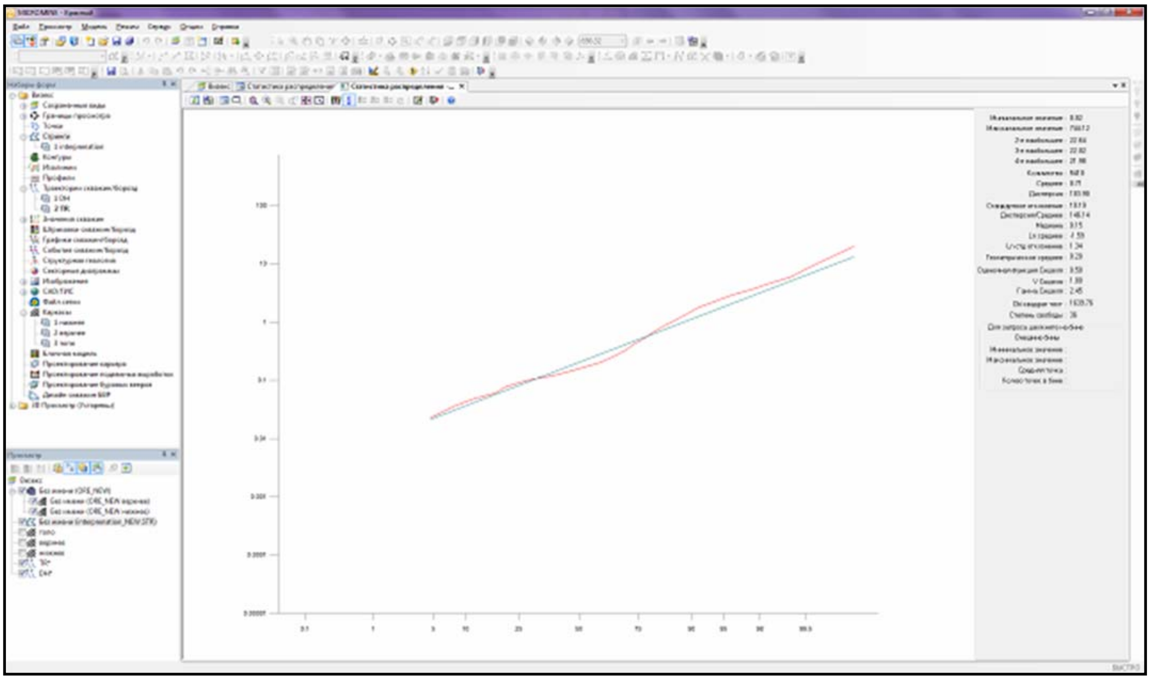
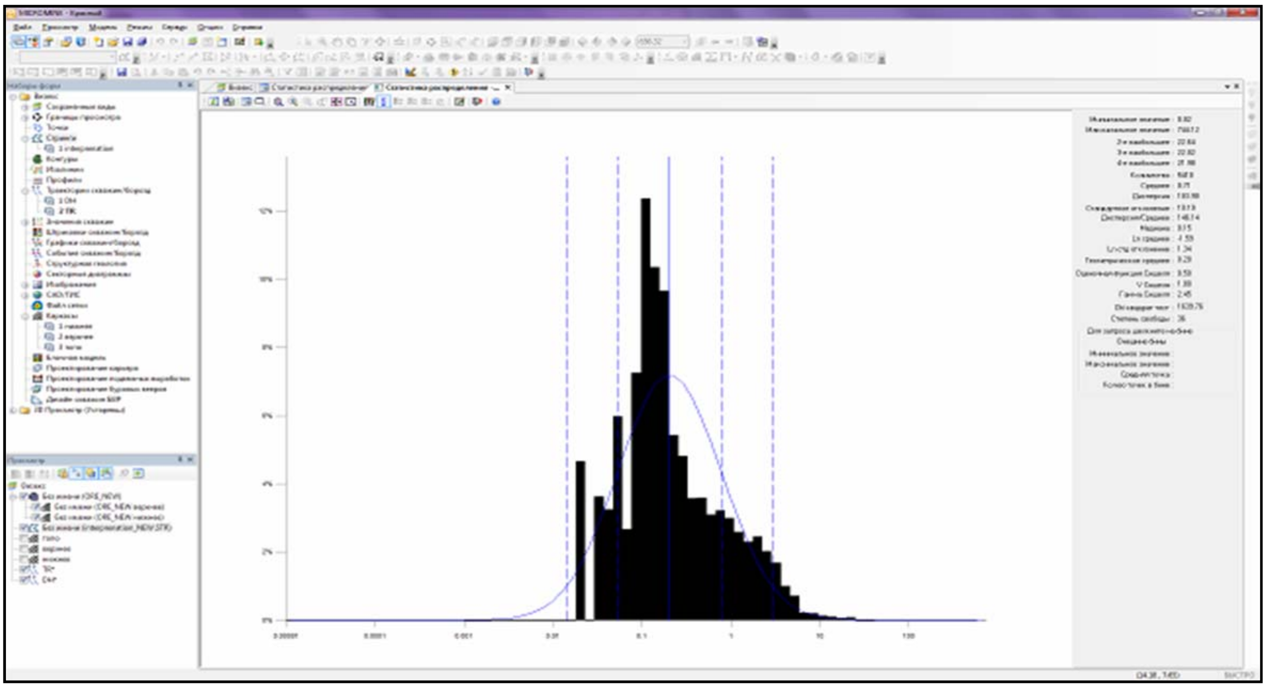


Figure 9-42 Histogram of distribution for Au grades within ore bodies



The cumulative Probability diagram of Au (Figure 9.42) shows that there are not evident bends in the high-value zone. However, a sample with an outstanding grade 744.12 g/t was detected. Therefore, it was decided to additionally create a grade field with restricted top cut values in the sampling file. The value of 22.64 g/t was assumed as a limit value for top cut grade restriction (the curve in the area of this grade has a bend in the cumulative probability diagram) (note: it is enough to place a cursor on the bend and read the grade value in the information window in the bottom of the screen).

After determining the top cut grade limit, the top cut grade sample was assigned the grade 22.64 g/t. The following operation was used for that:

- In the process *File* | *Fields* | *Calculate* an additional field was created in the sample file: *AU_CUT*. The process copied the gold grade to this field from the field *Au_correct*, applying a corresponding grade limit to it with the help of the function *Reduce till.*

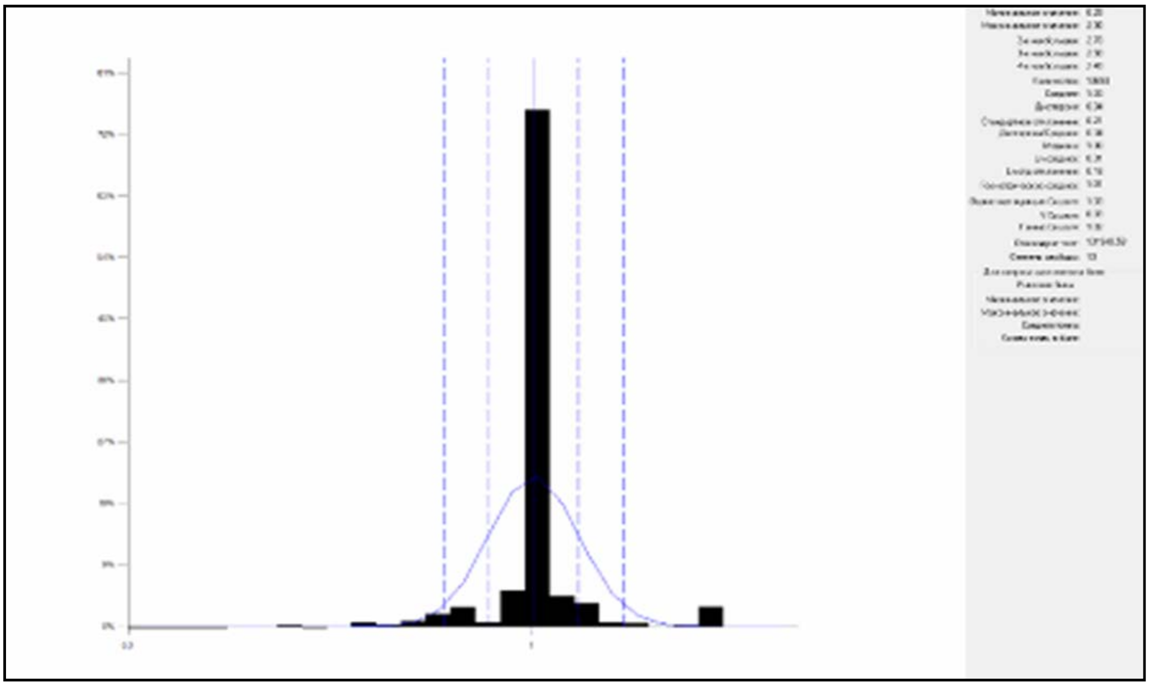
The created field *AU_CUT* was later used for geostatistic analysis and grade interpolation.

9.9. Compositing

Compositing is a standard procedure allowing to bring all sampling intervals to identical length. This allows all samples to have the same weight for grade interpolation. The length of composite intervals usually equals to the standard or the average sampling length.

In order to determine the optimal length of composite intervals, a distribution histogram of interval lengths was built, using the process *Statistics* | *Distribution.* The histogram (Figure 9.43) showed that most of sampling intervals have the length close to 1 m. The mean and the median values of sampling interval lengths within ore bodies were estimated in the process Statistics | Descriptive | Median/Moda as 1.0 and 1.03 m respectively. Based on these studies, it was decided to create a file of composite intervals 1 m long.

Figure 9-43 Histogram of distribution of sampling interval lengths



The process Borehole | Composite calculation | Along borehole was used for creation of the composite intervals file. Meanwhile, the process of creation of composite intervals with the set length occurred from the borehole/trench collar to the bottom. The field ORE_NEW was used as the constant field, i.e. the merging process stopped and began again when the values of these fields were changed, and the samples within ore bodies and outside ore bodies did not mix.

New coordinates of interval middles were calculated for compositing.

Thus, the composite files were created:

- KOMP_1m__ALL_NEW.DAT.

- KOMP_1m__TR_NEW.DAT.
- KOMP_1m__DH_NEW.DAT.

9.10. Geostatistical Analysis

The purpose of geostatistic analysis is creation of a series of oriented diagrams which could be used as a weighting mechanism for the ordinary kriging algorithm. Variogram parameters contribute a lot to determination of the search ellipse sizes and resource categorization.

Therefore, the main purposes of geostatistic analysis were as follows:

- Evaluate the presence of oriented mineralization anisotropy.
- Assess spatial continuity of the mineralization in main anisotropy directions. Mineralization continuity can be assessed by means of usage of the variogram impact zone, i.e. the distance at which the variogram reaches the absolute limit (plateau). Respectively, the grades cannot be reliably evaluated if the search radius for grade interpolation is larger than the variogram magnitude. When the variogram reaches the limit, there is no correlation between a pair o samples within the distance separating these samples.
- Receive variogram parameters (nugget effect, absolute limit – sill, variability) which are used as input parameters for interpolation with the ordinary kriging method.
- Evaluate the presence of oriented mineralization anisotropy. This can be done by means of study of oriented variograms. Oriented variability exists if the variogram reaches the absolute limit within different distances in different directions.
- Receive an ideal distance for the exploration grid.

Geostatistic analysis (building of variograms) was conducted.

The process *Statistics | Semivariograms* was used for modeling of omnidirectional variograms, and only for those samples in the composite interval file, which were selected within the ore body. A variogram was modeled for the whole selection of composite samples within ore body wireframes with log transformation, since the sample distribution is close to logarithmically normal one.

Semivariograms for the borehole with the space 1 to 7 m were used for determination of the nugget effect.

Figure 9-44 Variogram for the borehole

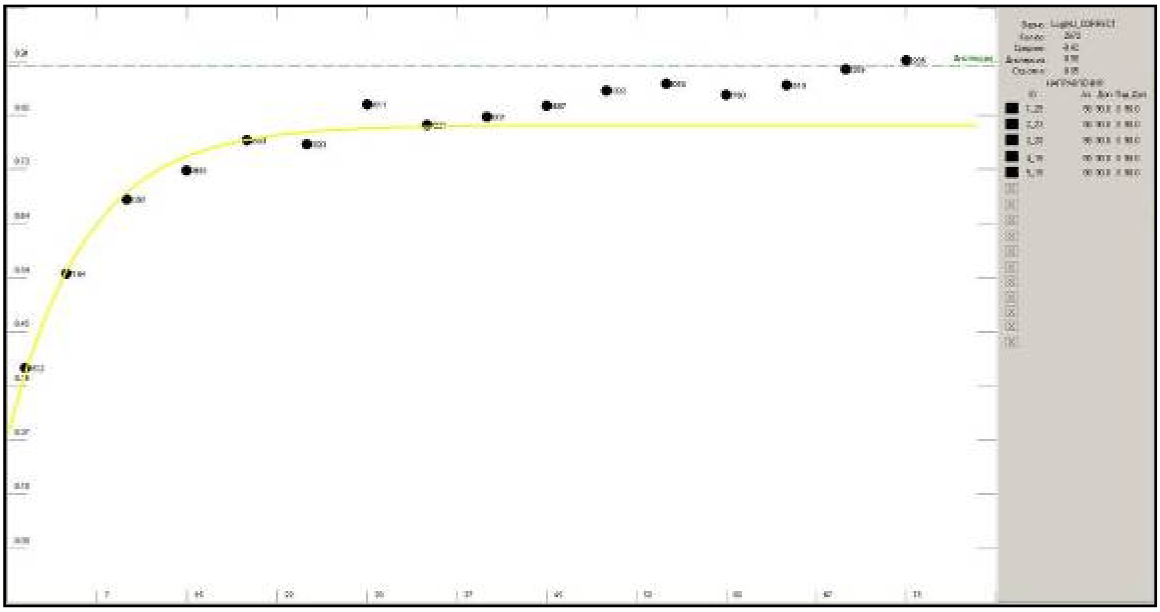


Figure 9-45 Omnivariogram

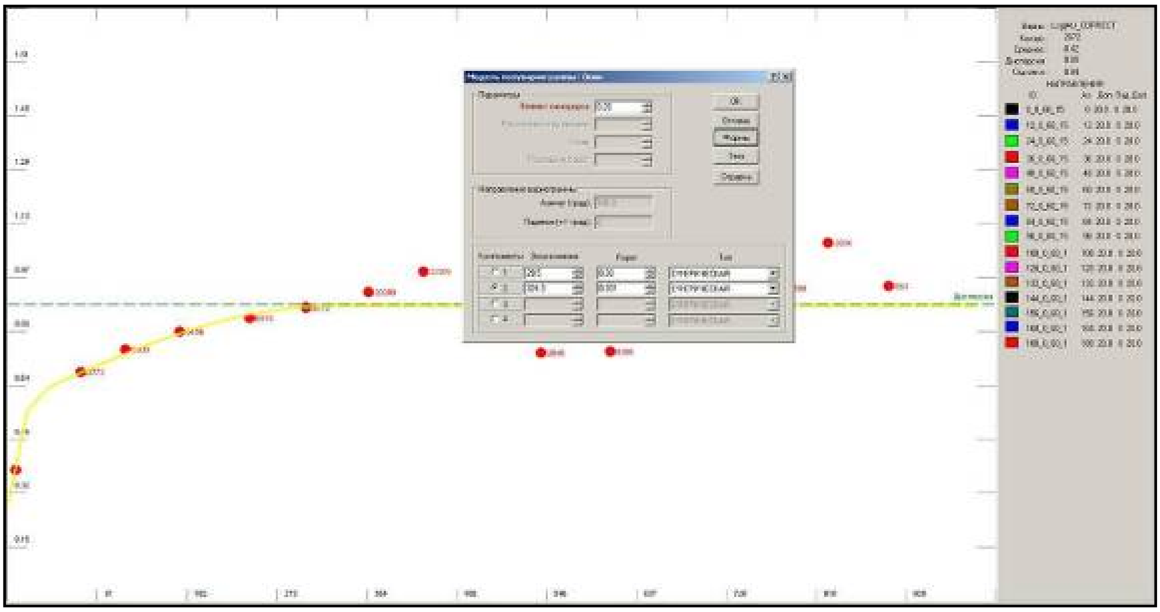


Figure 9-46 Variogram for gold, main axis - azimuth 108, dipping 0°

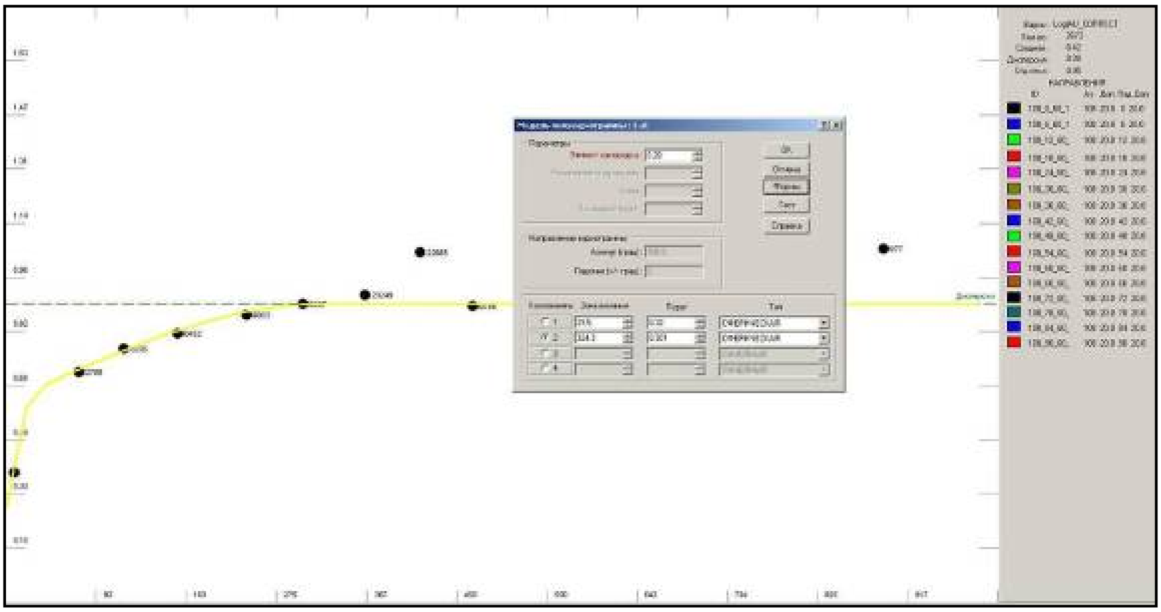


Figure 9-47 Variogram for gold, second axis - azimuth 1980, dipping 66°

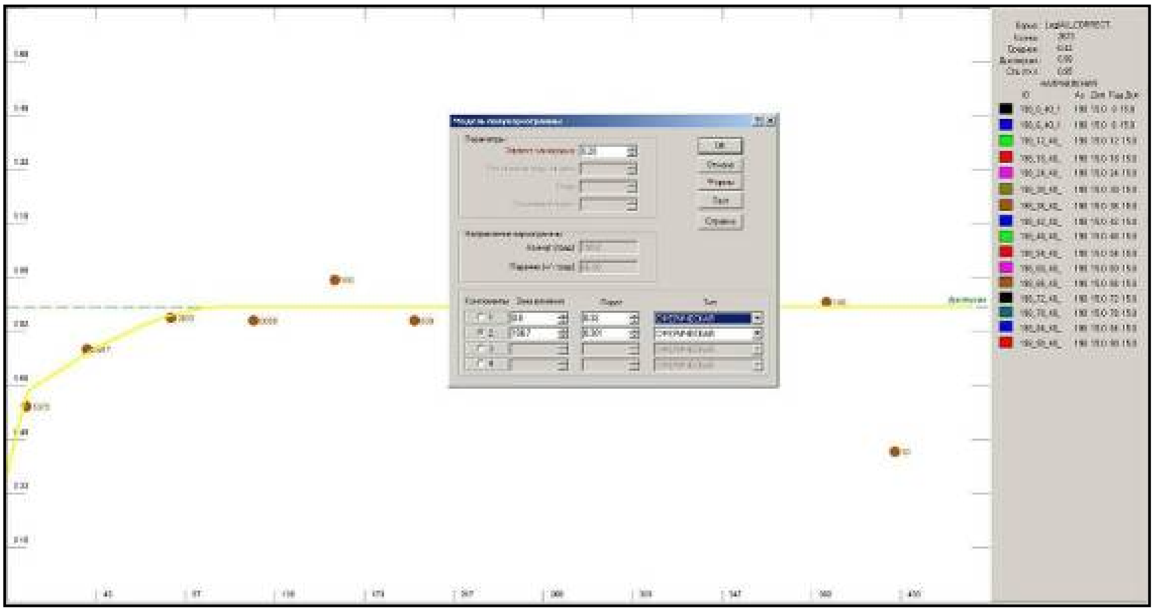
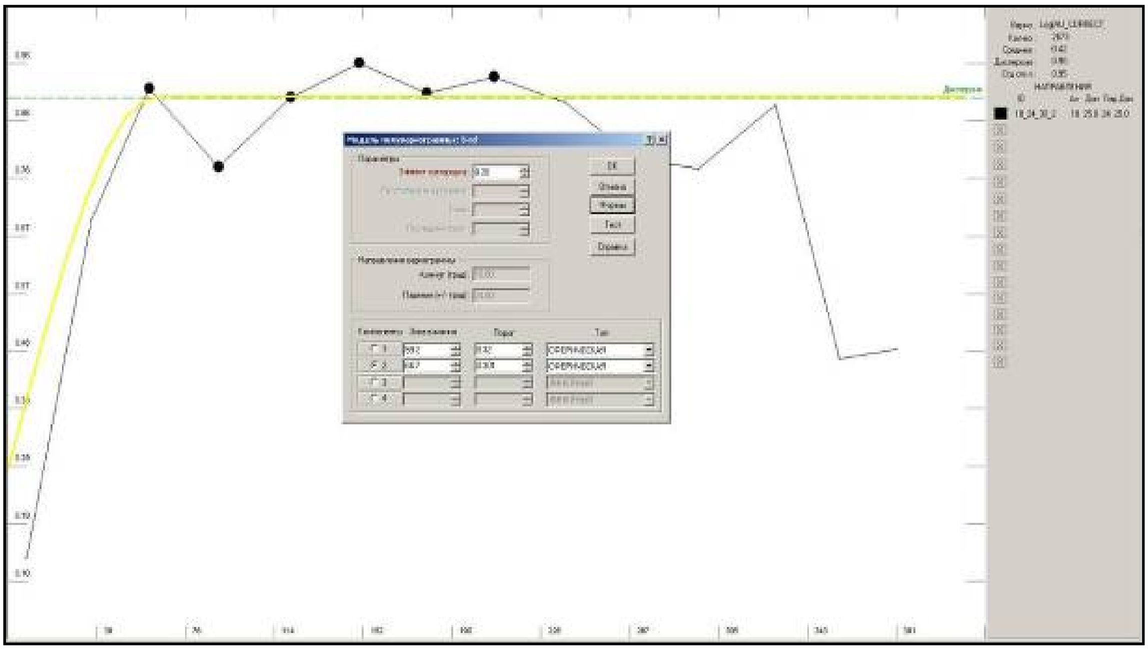


Figure 9-48 Variogram for gold, third axis - azimuth 180, dipping 240



9.11. Block Modeling

Block modeling consisted of the following stages:

a. Building of empty block models for all ore bodies.
b. Coding of the block model with wireframe models of ore bodies.
c. Coding of the block model with wireframe sets.

9.11.1. Building of Empty Block Models

Prior to building an empty block model for each ore body it was necessary to determine minimum and maximum coordinates of the ore body boundaries. These parameters were defined in the following way:

- A set of wireframe models of ore bodies was exported to two files – a triangle file and a point file, using the process *File* | *Export* | *Wireframes.*
- The created point file contained the coordinates of all points used for building a wireframe model. It is enough to place the cursor on the file, right-click and choose the option *Min/Max* for determination of minimum and maximum values of coordinates.

The size of cells was dictated by the height of the working bench, the thickness of the exploration grid (more than ¼ and less than ½ of the distance between exploration workings with the grid at detailed sites 80 m x 60 m) and the sizes of the block model confirmed by means of declustering (rarefication of the sites with thickened sampling) of sampling data. The height of the working bench of the pit for such type of a deposit is usually assumed equal to 10 m. Therefore, it was decided to use the sizes of cells 30x25x10 m.

An empty block model was created using the process Modeling | 3D block evaluation | Create an empty block model.

Table 9-14 Block model characteristics

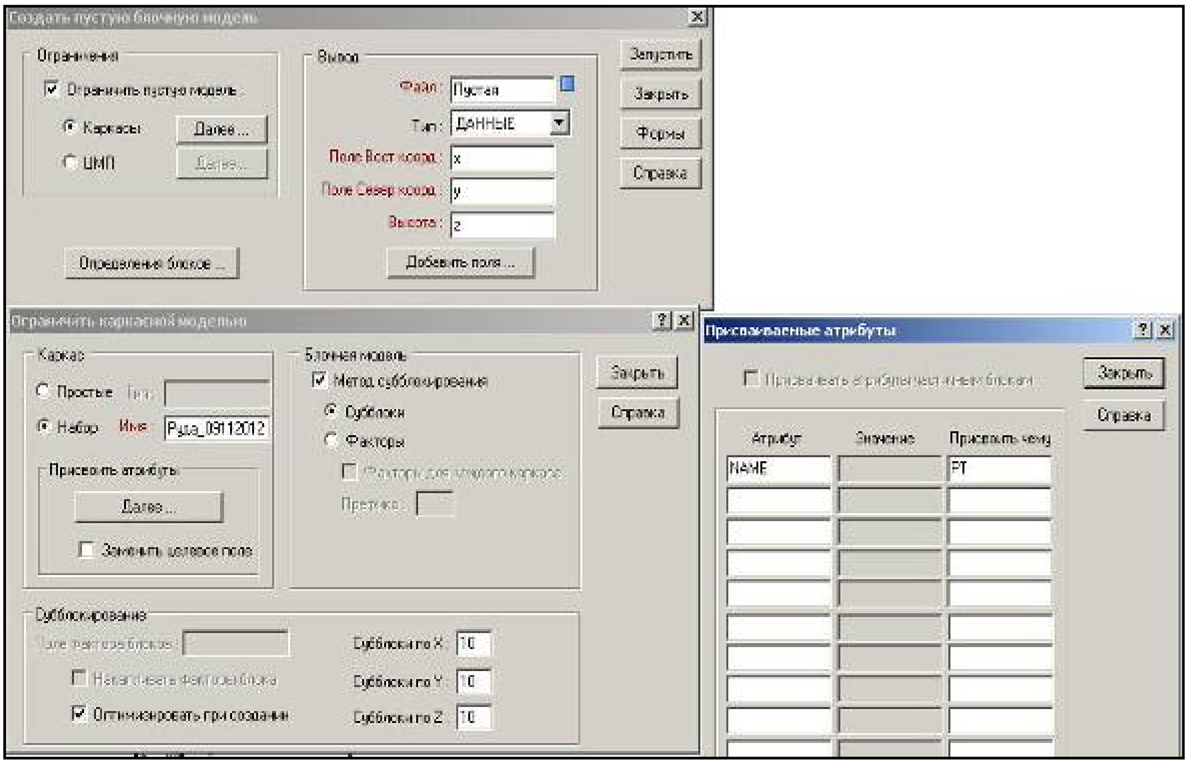
Direction	Length (m)		Grade Models	
	Minimum	Maximum	Size of Block (m)	Number of Subblocks
X	367502.50	370631.50	30	189525
Y	6462486.25	6464766.25	25	
Z	593.5	1018.5	10	

In the process of creation, empty block models were coded with wireframe models of ore bodies so that it were possible to choose only those blocks which fell within the corresponding ore bodies.

The subblock parameter was used for assigning (coding) wireframe models to block models. In this case this parameter was set as 10 by 10 by 10, i.e. each block was divided into 1000 parts (10 * 10 * 10 = 1000).

Prior to coding, a field was created in the block model file for each closed wireframe model – a field for the wireframe model flag. The name of the wireframe was recorded in this field. A block model was assigned to this wireframe.

Figure 9-49 Form for building an empty block model



9.11.2. Grade Interpolation

Grades were interpolated to block models from the samples selected within all ore bodies.

Each ore body was interpolated separately, with sample division by ore bodies, because during interpolation the search ellipse in most cases could reach neighbouring ore bodies. Grades above 0.2 g/t were interpolated.

Since the ore bodies included a significant number of samples with grades below the cutoff one, an ore content coefficient was calculated for each block of the model (KP_NEW). In this regard all samples with the grade above or equal to 0.2 g/t in the KP_NEW field were assigned the value 1 and the samples with the grades below the cutoff grade – value 0. These were later interpolated to the block model.

Since geostatistic analysis was conducted and variograms were modeled in three main directions, the ordinary kriging method was chosen as the main interpolation method. Kriging parameters for gold grades were used for interpolation of the ore content coefficient.

The search ellipse parameters indicated in Table 9.15 were used for the interpolation process. Variogram ranges and variogram model parameters are shown as well.

Table 9-15 Grade interpolation parameters

No of Interpolation Cycle	1	2	>2
Search radii	Less or equal to 2/3 of the variogram range	Full variogram range	Increase by the variogram range
Minimum number of samples	3	3	1
Maximum number of samples	12	12	12

Several points located in different parts of the block were interpolated for grades interpolation, for substantiation of the average grade estimate in the block within its whole volume. I.e. the Discretization function was used (5 points by 5 points by 5 points). These point estimates were simply averaged for evaluation of the average value of the blocks.

The declusterization process was conducted for interpolation by means of usage of four sectors within the search ellipse, with limitation up to 3 points maximum per sector. The maximum total number of samples used for interpolation was assumed equal to 12. Points were selected on the following basis: the point located farthest from the evaluated point was discarded (basis for evaluation). If the block did not receive the grade estimate, the search ellipse was increased until each block in the model received the interpolated grade. Two fields with gold grades and the ore content coefficient were interpolated separately, but all interpolations used the parameters of the search ellipse and variograms for gold.

A macro Интерполяция.mcr (Interpolation) was written for work simplification and process automation. It creates empty block models, codes them, interpolates gold, the ore content coefficient, and classifies reserves.

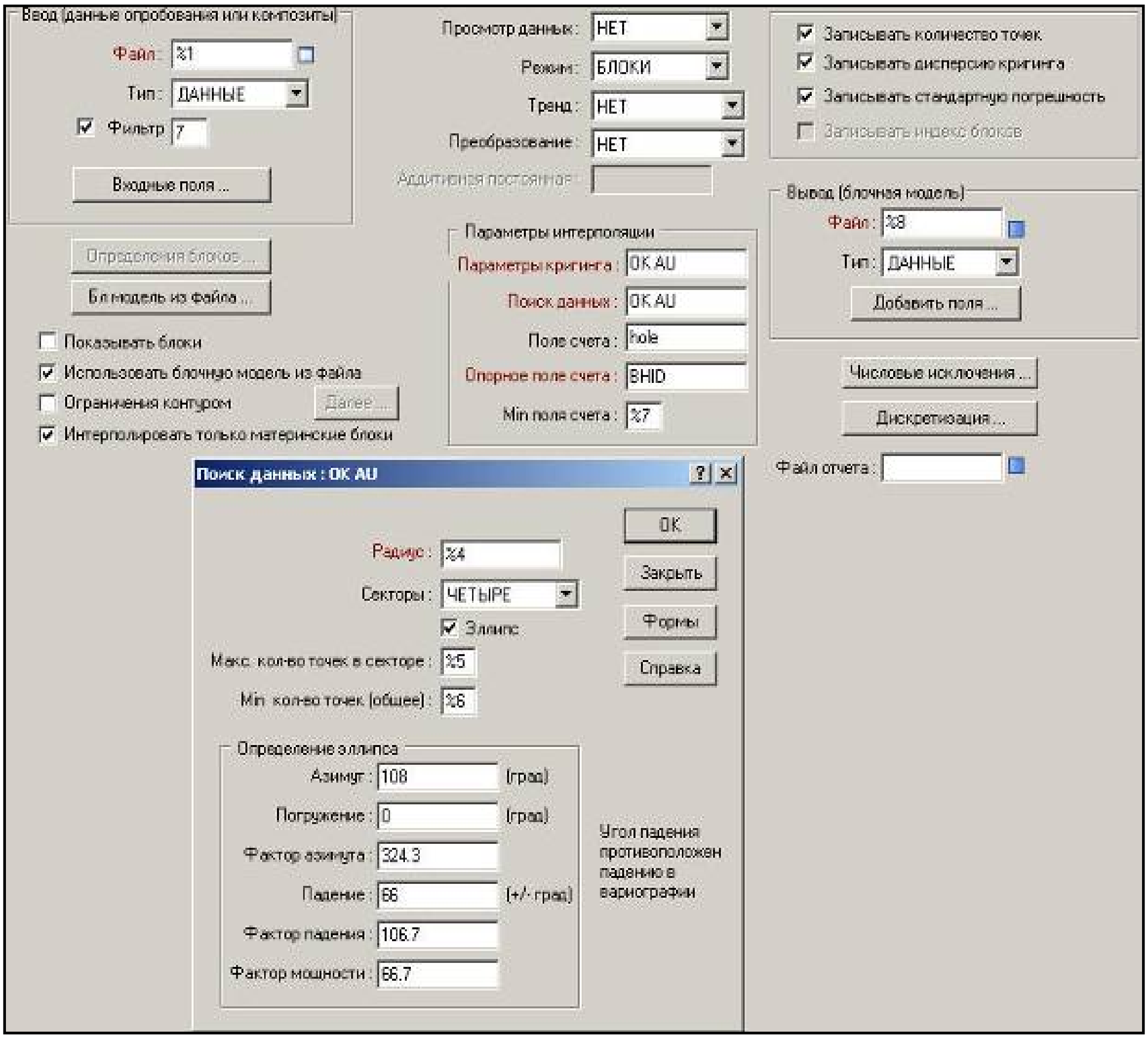
The process Modeling | 3D block evaluation | Ordinary/Universal kriging was used for interpolation.

In order to use the processes in macros, the following variables were created in process forms:

- %1 – composite file,
- %2 – interpolated element,

- %3 – input block model,
- %4 – multiplier for search ellipse axes
- %5 – maximum number of composite intervals
- %6 – minimum number of composite intervals
- %7 – number of workings participating in interpolation,
- %8 – output block model,
- %9 – number of interpolation cycle.

Figure 9-50 Form for grade interpolation with the kriging method (for macro)



The macro was run in the process Service | Macro | Start (or using the blue arrow icon on the control panel). The macro created block model files with interpolated Au grades and ore content coefficients, with corresponding fields of the interpolation number, the specific weight field and all codes. Macro run is possible starting not from the first process but from the one indicated by the user, which makes it possible to successively run macro blocks, with validation of temporary files created by the macro.

Figure 9-51 Fragment of a macro building block models and interpolating grades

NBBLANK	2											
Comment		OK по Au_correct	верхнее PT									
Comment				файл композитов	элемент	модель на входе	радиус	макс точки	мин точки	выработки	файл на выходе	заход	PT
OKRIG	1			copm_lm_all_new	AU_CORRECT	Пустая	0.67	3	1	2	xxx1	1	верхнее
OKRIG	1			copm_lm_all_new	AU_CORRECT	xxx1	1	3	1	2	xxx2	2	верхнее
OKRIG	1			copm_lm_all_new	AU_CORRECT	xxx2	2	3	1	1	xxx3	3	верхнее
OKRIG	1			copm_lm_all_new	AU_CORRECT	xxx3	10	3	1	1	xxx4	4	верхнее
Comment		OK по Au_correct	нижнее PT										
OKRIG	1			copm_lm_all_new	AU_CORRECT	xxx4	0.67	3	1	2	xxx5	1	нижнее
OKRIG	1			copm_lm_all_new	AU_CORRECT	xxx5	1	3	1	2	xxx6	2	нижнее
OKRIG	1			copm_lm_all_new	AU_CORRECT	xxx6	2	3	1	1	xxx7	3	нижнее
OKRIG	1			copm_lm_all_new	AU_CORRECT	xxx7	10	3	1	1	xxx8	4	нижнее
Comment		OK по KF	верхнее PT										
OKRIG	2			copm_lm_all_new	KF	xxx8	0.67	3	1	2	xxx9	1	верхнее
OKRIG	2			copm_lm_all_new	KF	xxx9	1	3	1	2	xxx10	2	верхнее
OKRIG	2			copm_lm_all_new	KF	xxx10	2	3	1	1	xxx11	3	верхнее
OKRIG	2			copm_lm_all_new	KF	xxx11	10	3	1	1	xxx12	4	верхнее
Comment		OK по KF	нижнее PT										
OKRIG	2			copm_lm_all_new	KF	xxx12	0.67	3	1	2	xxx13	1	нижнее
OKRIG	2			copm_lm_all_new	KF	xxx13	1	3	1	2	xxx14	2	нижнее
OKRIG	2			copm_lm_all_new	KF	xxx14	2	3	1	1	xxx15	3	нижнее
OKRIG	2			copm_lm_all_new	KF	xxx15	10	3	1	1	BI_модель_OK	4	нижнее
Comment		IDW по Au_correct	верхнее PT										
OIMBI	1			copm_lm_all_new	AU_CORRECT	Пустая	0.67	3	1	2	xxx1	1	верхнее
OIMBI	1			copm_lm_all_new	AU_CORRECT	xxx1	1	3	1	2	xxx2	2	верхнее
OIMBI	1			copm_lm_all_new	AU_CORRECT	xxx2	2	3	1	1	xxx3	3	верхнее
OIMBI	1			copm_lm_all_new	AU_CORRECT	xxx3	10	3	1	1	xxx4	4	верхнее
Comment		IDW по Au_correct	нижнее PT										
OIMBI	1			copm_lm_all_new	AU_CORRECT	xxx4	0.67	3	1	2	xxx5	1	нижнее
OIMBI	1			copm_lm_all_new	AU_CORRECT	xxx5	1	3	1	2	xxx6	2	нижнее
OIMBI	1			copm_lm_all_new	AU_CORRECT	xxx6	2	3	1	1	xxx7	3	нижнее
OIMBI	1			copm_lm_all_new	AU_CORRECT	xxx7	10	3	1	1	xxx8	4	нижнее

9.11.3. Setting of Specific Weight in Block Models

The specific weight value in block models can be defined with several methods:

1) Direct assignment of the specific weight value to all model blocks.

This method is used when the specific weight of ores is more or less constant or the data for specific weight determination is not sufficient.

2) Specific weight calculation with regression equations.

When the specific weight of ores depends on useful element grades, it can be calculated for each cell of the block model with regression equations. This method is frequently applied for iron or manganese ores, for instance.

3) Interpolation of specific weight values.

The specific weight can be interpolated to each cell of the block model in the same way as sampling values. This method is frequently used when the database has enough measurements of specific weight values.

4) Usage of the geological model for setting specific weight values.

A geological model of lithologies occurred within the deposit can be used for setting the specific weight. Built wireframes of each lithology can be used for that. It should be noted that it is desirable to have the depths of weathering crust development or an additional field with the degree of oxidation of sampling intervals or lithology in the geological database for correct assessment of the density of unconsolidated (weathered) rocks. Wireframe models on the basis of lithology are used for block model coding and further assignment of specific weight values to each cell of the block model. This method is most reliable if the specific weight does not depend on useful element grades.

In this work the specific weight was assigned with a direct method. The Customer provided with the tables of bulk weight studies with the derived average value 2.6 t/m^3 which was used in all calculations.

9.12. Reserve Classification

When kriging is used for grade interpolation, the reserve classification strategy is usually mainly based on prospecting and interpolation parameters as well as the minimum number of samples and workings participating in the interpolation process.

The following methodology is commonly used for reserve (resource) classification.

1. Model cells can be classified as *Measured Resources* if they were interpolated with the use of prospecting radii not exceeding 2/3 of variogram ranges, under condition that at least two or three samples and two boreholes (or trenches) participated in the block interpolation process (model cell). If the variograms had several structures, then prospecting radii equal to long variogram structures can be used for interpolation of Measured Resources.
2. Model cells can be classified as Indicated Resources if they were interpolated with the use of prospecting radii not exceeding full variogram ranges, under condition that at least two or three samples and at least two boreholes (or trenches) participated in the block interpolation process (model cell). If variograms had several structures, long variogram strictures are used for setting the interpolation search ellipse.
3. All other cells interpolated with search radii exceeding variogram ranges or using less than three samples or less than two boreholes can be classified as Inferred Resources.

Resources cannot be classified as Measured and the block class is reduced till Indicated in case of possible risk of reliability of input information such as borehole coordinates data, sampling quality data, borehole directional survey data etc.

The analytical information received for the Krasny deposit is based on channel and core sampling, which is very reliable information, but with the thickness of sampling not sufficient for assigning the categories of Measured and Indicated in compliance with international requirements to resource classification and the JORC Code.

The purpose of work at the following stages included the study of such parameters important for reserve reliability assessment as drilling quality, channel sampling methodology, sample preparation methodology, quality and methods of analyses, core recovery, results of internal and external sampling control and other data required for preparation of bankable TEO (feasibility study). Therefore, the received categories of Mineral Resources can be used in the future in bankable TEO (feasibility study) without additional studies.

This work used the methodology of reserve classification for models with grades interpolated with the Ordinary Kriging method.

9.12.1. Classification of OK Models

The classification executed by the author is conditional and reflects only possible resource reference to a specific category.

The data of geostatistic analysis and grade interpolation parameters were used for classification of gold mineralization in models with grades based on interpolation with the ordinary kriging method. Specific requirements were also taken into account, for example, the minimum number of samples used for grade interpolation to each block. The blocks which contain interpolated grades were classified as *Measured Resources* assuming that at least 3 samples were selected during interpolation, and the search radii equaled to or were less than two thirds of variogram ranges in all directions. If the cells of the block model contained interpolated grades with the use of search radii equal to or smaller than the full ranges of semi-variograms in all directions, such blocks were classified as Indicated Resources (unless they were classified as Measured Resources prior to that). It was assumed that at least 3 samples were used for grade interpolation. If the grades were interpolated to the block model using the radii greater than variogram ranges in all directions, such blocks were classified as *Inferred Resources* assuming that at least one sample was used for interpolation.

The RUN field was created in models for grade interpolation, corresponding to the interpolation cycle. The first interpolation was conducted with parameters corresponding to the Measured category. At that the *RUN* field contained the value 1. During the second interpolation run corresponding to the Indicated category, this field contained the value 2. All other model blocks interpolated with greater search radii received the value 3 and above in the *RUN* field (Inferred).

Such resource classification meets the requirements of the JORC Code (Australasian Code for Reporting of Mineral Resources and Ore Reserves) in relation to the degree of reliability, the sampling grid density and grades continuity.

To take a final decision on the resource classification, the authors took into account the whole complex of the information provided and the results of statistic and geostatistic studies.

A sparse and irregular sampling grid does not allow to refer resources to the category above the Inferred one based on the reliability degree.

9.13. Block Model Validation

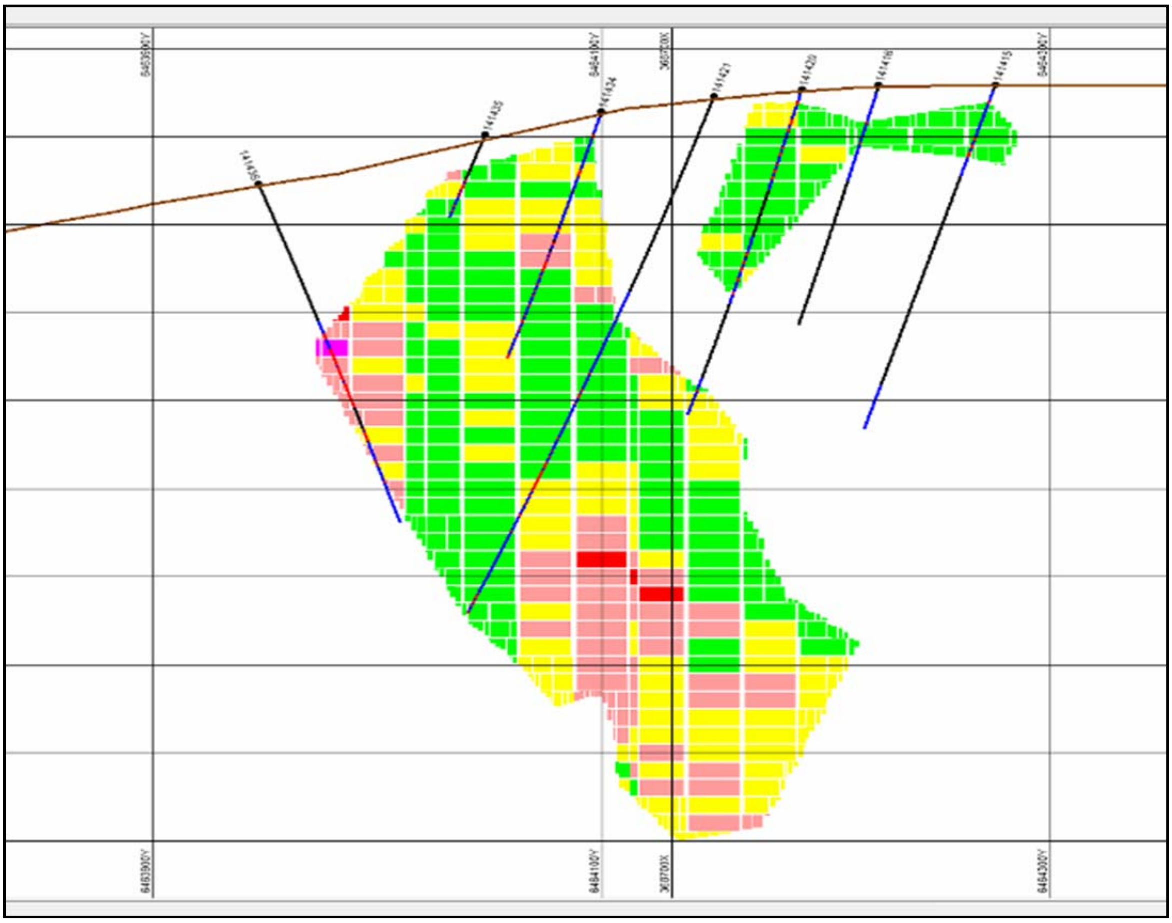
The block model can be validated with several methods:

- visual validation;
- digital validation;
- grade interpolation with an alternative method,
- cross validation,
- quantile plot,
- model comparison with actual data.

9.13.1. Visual Validation

For visual validation the block model was uploaded to *Vizex* together with sampling data. At that, samples and model blocks were coloured on the basis of gold grades and compared visually. The figure shows that the grades in the block model generally correspond to the samples in boreholes but are smoothed to greater extent than in samples, which is normal.

Figure 9-52 Block model and boreholes



9.13.2. Digital Validation

Digital validation of the block model is commonly conducted with the following methods:

The average gold grade in the block model was compared with average grades in the sampling database, chosen within the ore bodies (process *Statistics* | *Descriptive* | *Normal/Log*). This analysis showed that the average grades in the block model are slightly lower than according to sampling data and equal to 1.38 g/t as compared to samples - 1.75 g/t for gold. This is explained by the fact that the sampling grid is sparser in those parts of ore bodies where the average grades are relatively low, and block models have regular filling with blocks within the whole volume of ore bodies. This preconditions lower general average grades in models than in sampling data.

9.13.3. Grade Interpolation with an Alternative Method

The Inverse Distance Weighting (IDW) method, degree 3, was chosen as an alternative grade interpolation method. At that, the block model coding, sample selection and grade interpolation methodology was similar to the main interpolation method described above. The difference is that the process for this interpolation did not use variogram models for weighting of samples participating in grade estimation in the block. Same search radii corresponding to variogram ranges were used.

The gold grade without restricted top cut grades was also used for alternative grade interpolation.

The search ellipse parameters indicated in Table 10.16 were used for the interpolation process.

Table 9-16 Parameters of grade interpolation with the IDW method

No of Interpolation Cycle	1	2	>2
Search radii	Less or equal to 2/3 of general variogram ranges within the deposit	Full variogram range	Increase by the variogram range
Minimum number of samples	3	3	1
Maximum number of samples	12	12	12

A macro Интерполяция.mcr (Interpolation) was written for work simplification and process automation. It creates empty block models, codes them, interpolates gold, the ore content coefficient, and classifies the resources.

The process Modeling | 3D block evaluation | Ordinary/Universal kriging was used for interpolation.

In order to use the processes in macros, the following variables were created in process forms:

- %1 – composite file,
- %2 – interpolated element,
- %3 – input block model,
- %4 – multiplier for search ellipse axes
- %5 – maximum number of composite intervals
- %6 – minimum number of composite intervals
- %7 – number of workings participating in interpolation,
- %8 – output block model,
- %9 – number of interpolation cycle.

The table of comparison of average gold grades received as a result of interpolation with OK and IDW methods is given below.

The table shows that ordinary kriging estimated the average grade slightly higher than the inverse distance method. Nonetheless, the correlation between grades received with different methods is very high which confirms the accuracy of grade estimation in block models.

As a result, block models were created with the ordinary kriging method and with the inverse distance method for validation, both with and without restriction of top cut grades.

Table 9-17 Comparison of average gold grades for OK and IDW methods, without top cut grade restriction

CUT_OFFf	OK			IDW			Difference, %		
	Ore, Kt	Average Grade, g/t	Metal, t	Ore, Kt	Average Grade, g/t	Metal, t	Ore	Grade of	Metal
0.20	54,523.1	1.08	58.9	57,993.1	1.05	60.96	-5.98	2.80	-3.35
0.40	44,729.2	1.24	55.7	43,584.9	1.29	56.22	2.63	-3.50	-0.97
0.60	35,776.5	1.43	51.2	33,816.3	1.52	51.48	5.80	-5.92	-0.46
0.80	26,841.1	1.68	45.0	26,783.9	1.74	46.55	0.21	-3.49	-3.29
1.00	20,779.4	1.91	39.6	20,692.6	1.99	41.10	0.42	-4.01	-3.61
1.20	16,009.0	2.15	34.4	16,256.4	2.23	36.24	-1.52	-3.61	-5.07
1.40	12,638.8	2.38	30.0	12,644.8	2.50	31.57	-0.05	-4.83	-4.87
1.60	9,396.5	2.68	25.2	9,696.9	2.80	27.16	-3.10	-4.22	-7.19
1.80	7,321.6	2.96	21.7	7,507.1	3.12	23.44	-2.47	-5.08	-7.43
2.00	5,614.9	3.29	18.5	5,877.0	3.47	20.37	-4.46	-5.14	-9.37

Table 9-18 Comparison of average gold grades for OK and IDW methods, with top cut grade restriction

cut_off	OK			IDW			Difference, %		
	Ore, Kt	Average Grade, g/t	Metal, t	Ore, t	Average Grade, g/t	Metal, kg	Ore	Grade of	Metal
0.20	54,523	1.04	56.6	54,921	1.04	57.2	-0.72	-0.35	-1.07
0.40	44,729	1.19	53.3	42,360	1.25	53.1	5.59	-4.90	0.42
0.60	35,776	1.37	48.9	33,138	1.47	48.6	7.96	-6.82	0.60
0.80	26,841	1.59	42.7	26,277	1.67	43.8	2.15	-4.59	-2.55
1.00	20,779	1.79	37.3	20,415	1.89	38.5	1.79	-4.96	-3.26
1.20	16,009	2.00	32.1	15,970	2.11	33.7	0.24	-5.00	-4.77
1.40	12,639	2.19	27.7	12,410	2.34	29.1	1.85	-6.44	-4.71
1.60	9,397	2.43	22.9	9,519	2.60	24.7	-1.29	-6.35	-7.55
1.80	7,322	2.65	19.4	7,389	2.86	21.1	-0.91	-7.48	-8.32
2.00	5,615	2.87	16.1	5,762	3.14	18.1	-2.56	-8.34	-10.69

9.13.4. Quantile Plot

Fitting of theoretic distribution to observed data can be visually validated on the Q-Q plot (also 'quantile plot'). This plot reflects the relation between observed values of variable and theoretical quantiles. If the observed values fall on a straight line, then theoretical distribution fits well to the observed data. In order to build the Q-Q plot, the program firstly regulates n points of the observed data in ascending order:

$$x_1 \leq x_2 \leq \ldots \leq x_n$$

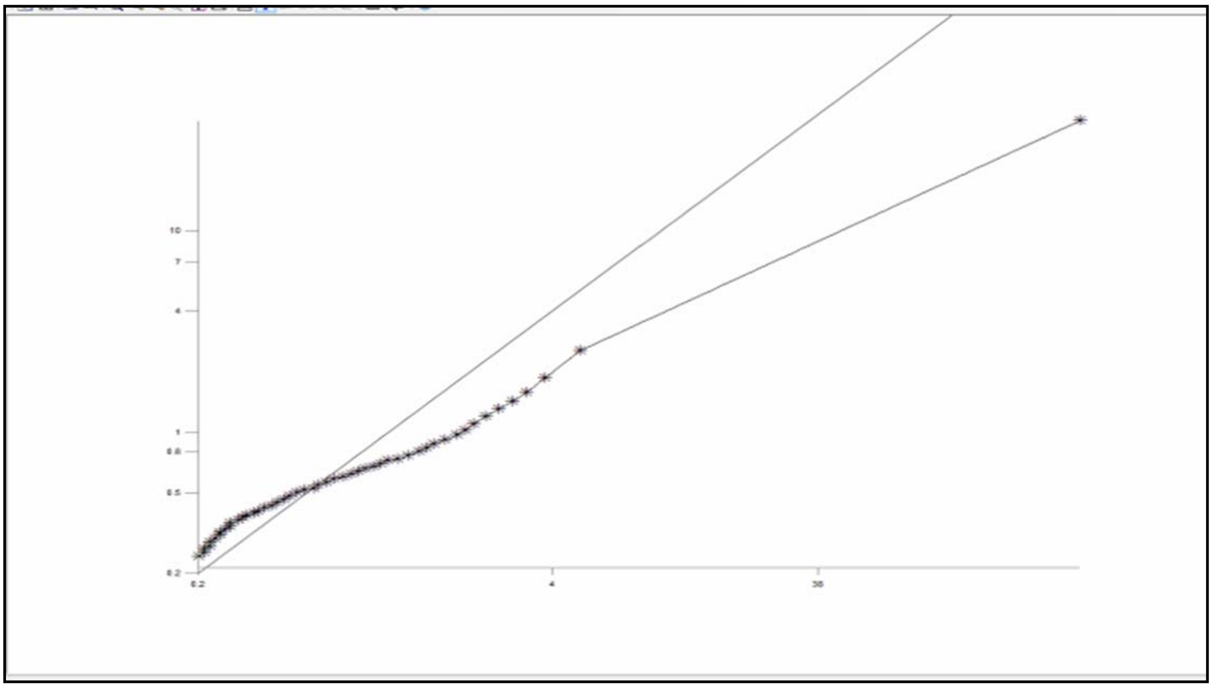
These observed values are reflected on one of the plot axes; and the following values are reflected on the other axis:

$$F^{-1}((i-r_{adj}) / (n+n_{adj})),$$

where i is the rank of the corresponding observation, r_{adj} and n_{adj} are corrections (≤ 0.5), and F^{-1} is reverse probabilistic integral for the corresponding standardized distribution. The plot received represents a diagram of dispersion of observed and expected (standardized) values with the corresponding set distribution.

In order to check model correctness quantile plots were built. They allow to compare and evaluate the correlation between input data and the data received as a result of interpolation to the block model.

Figure 9-53 Quantile diagram for models received with the ordinary kriging method



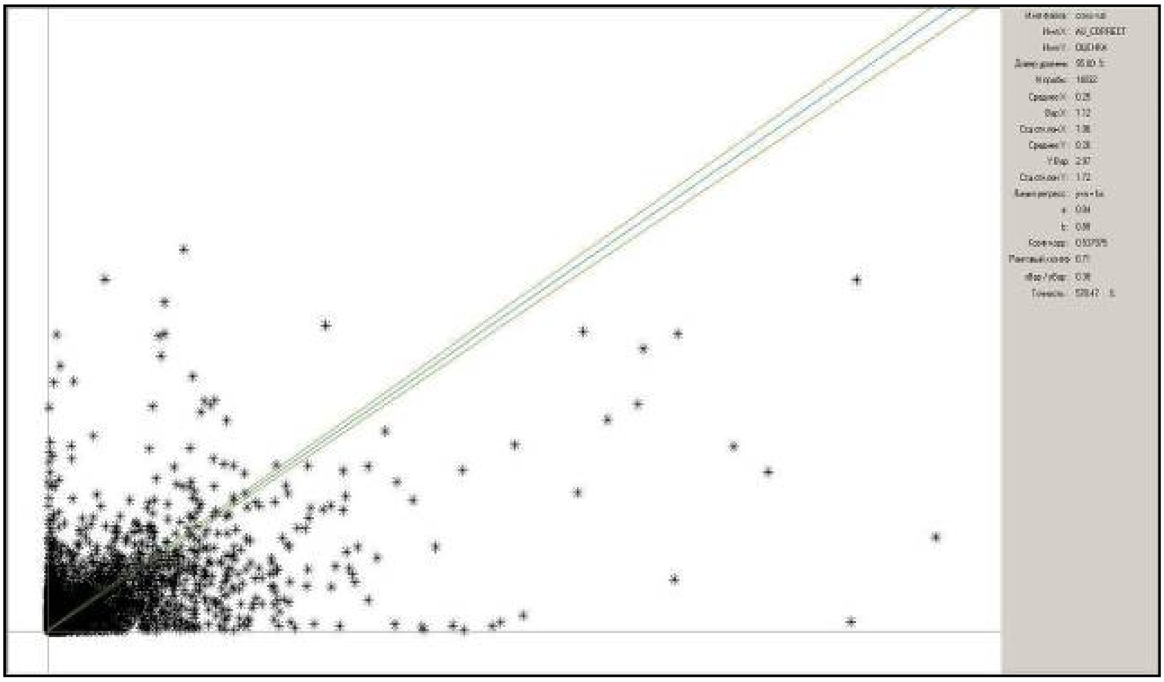
The plot shows that overestimation in low grade classes and underestimation in high grade classes occurred during interpolation to the block model. This effect is particularly typical for ore zones with lower number of samples for interpolation. But in general the Spearman and Pearson correlation coefficients close to 100% speak to a high degree of compatibility of input and received values.

9.13.5. Cross Validation

Cross validation is validation of the reliability of the model, with the help of which researchers study whether the model is applicable for analysis of comparability of data not used for building the basic model.

The validation showed a comparatively high degree of convergence of the model without top cut grade restriction with basic data. The Pearson correlation coefficient equals to 54%, and the Spearman rank correlation coefficient is 71%. This speaks to correct geostatistic analysis, on the basis of which search ellipsoid and kriging parameters were defined (Figure 9.54-2).

Figure 9-54 Diagram of model dispersion based on the results of cross validation of the model received with the ordinary kriging method



9.14. Mineral Resources Report

The models of the ore bodies within the Krasnoye gold-bearing ore occurrence were developed based on the natural cut-off grade. Thus the Mineral resources report is based on a number of cut-off grades shown by dependence diagrams of the average grade throughout the ore bodies upon the tonnage, using various cut-off grades.

Table 9-19 Resources, obtained by ordinary kriging, avoiding reduction of top-cut grade

Cut-off Grade, %	Ore Resources		Bulk Weight, t/m^3	Average Grade, g/t	Metal, t
	M m3	Kt			
0.2	21.0	54,523.1	2.60	1.08	58.9
0.4	17.2	44,729.2	2.60	1.24	55.7
0.6	13.8	35,776.5	2.60	1.43	51.2
0.8	10.3	26,841.1	2.60	1.68	45.0
1.0	8.0	20,779.4	2.60	1.91	39.6
1.2	6.2	16,009.0	2.60	2.15	34.4
1.4	4.9	12,638.8	2.60	2.38	30.0
1.6	3.6	9,396.5	2.60	2.68	25.2
1.8	2.8	7,321.6	2.60	2.96	21.7
2.0	2.2	5,614.9	2.60	3.29	18.5

Figure 9-55 Resources and quality of the deposit ore depending on Au cut-off grade.

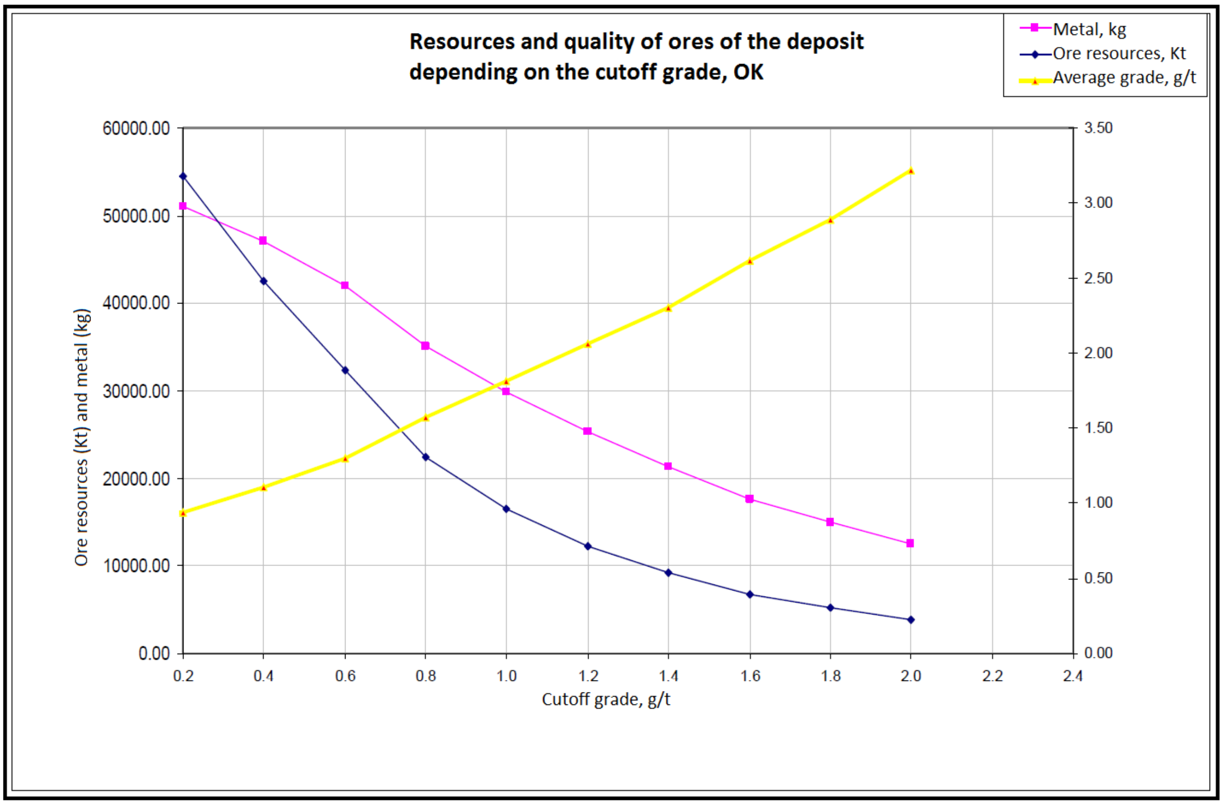


Table 9-20 Resources, obtained by ordinary kriging, using reduction of top-cut grade

Cut-off Grade	Ore Resources		Specific Weight t/m^3	Average Grade, g/t	Metal, t
	M m^3	Kt			
0.20	21.0	54,523.07	2.60	1.04	56.6
0.40	17.2	44,729.18	2.60	1.19	53.3
0.60	13.8	35,776.47	2.60	1.37	48.9
0.80	10.3	26,841.07	2.60	1.59	42.7
1.00	8.0	20,779.38	2.60	1.79	37.3
1.20	6.2	16,009.04	2.60	2.00	32.1
1.40	4.9	12,638.82	2.60	2.19	27.7
1.60	3.6	9,396.51	2.60	2.43	22.9
1.80	2.8	7,321.62	2.60	2.65	19.4
2.00	2.2	5,614.93	2.60	2.87	16.1

Figure 9-56 Resources and quality of the deposit ore depending on Au cut-off grade

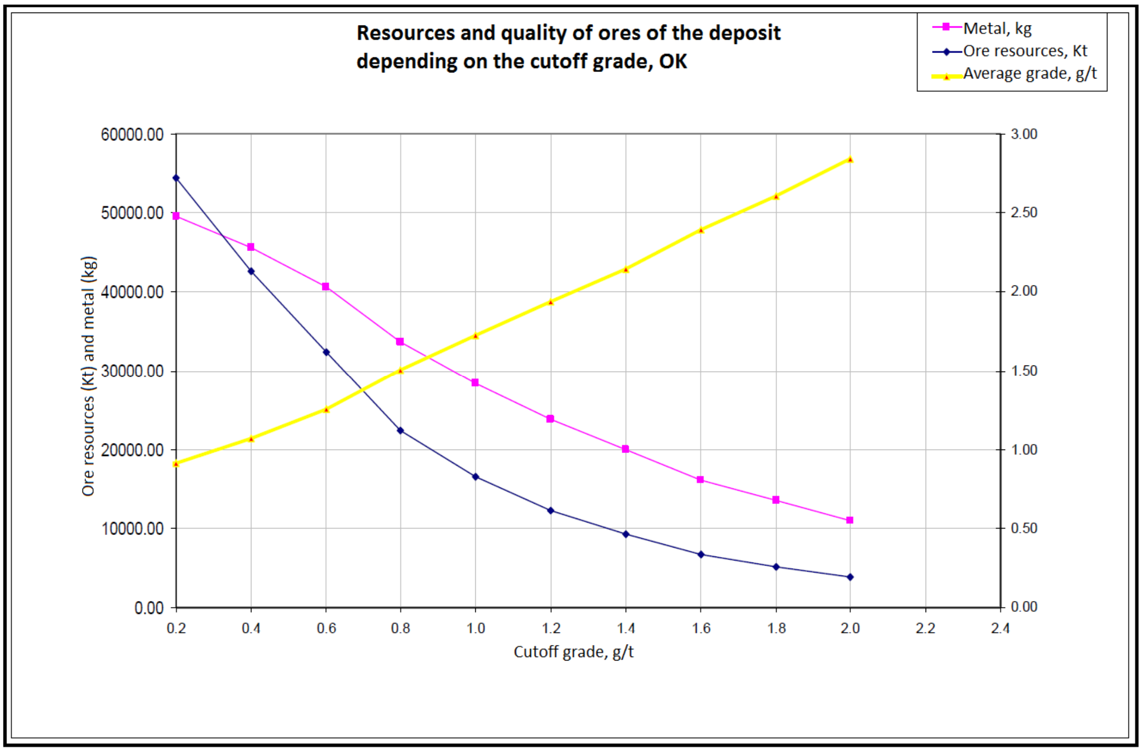


Table 21-3 Resources by categories with top-cut grades excluded

Volume, th. cub.m	Tonnage, Kt	Average Grade, g/t	Metal, t	Share of Indicated, %	Category
17,879	46,485	1.03	48.0		Inferred
3,091	8,038	1.06	8.5		Indicated
20,970	54,523	1.04	56.6	15.1	Total
14,525	37,764	1.20	45.1		Inferred
2,679	6,965	1.18	8.2		Indicated
17,204	44,729	1.19	53.3	15.4	Total
11,567	30,075	1.37	41.3		Inferred
2,193	5,702	1.33	7.6		Indicated
13,760	35,776	1.37	48.9	15.5	Total
8,657	22,508	1.60	36.1		Inferred
1,667	4,333	1.53	6.6		Indicated
10,323	26,841	1.59	42.7	15.5	Total
6,703	17,427	1.81	31.5		Inferred
1,290	3,353	1.71	5.7		Indicated
7,992	20,779	1.79	37.3	15.4	Total
5,178	13,462	2.02	27.2		Inferred
980	2,547	1.91	4.9		Indicated
6,157	16,009	2.00	32.1	15.2	Total
4,113	10,693	2.21	23.6		Inferred
748	1,945	2.10	4.1		Indicated
4,861	12,639	2.19	27.7	14.8	Total
3,091	8,036	2.45	19.7		Inferred
523	1,361	2.37	3.2		Indicated
3,614	9,397	2.43	22.9	14.1	Total
2,447	6,362	2.64	16.8		Inferred
369	960	2.65	2.5		Indicated
2,816	7,322	2.65	19.4	13.1	Total
1,898	4,934	2.86	14.1		Inferred
262	681	2.97	2.0		Indicated
2,160	5,615	2.87	16.1	12.5	Total

10. Mine Engineering

10.1. Method of Development

The deposit consists of a single site which has been traced by tranches and exploration holes deep to 370 m from the surface.

The ore bodies outcrop under the quaternary deposits having thickness up to 10 m. Relatively shallow distribution of the ore bodies, their considerable thickness and bedding in the proximate vicinity with the surface, as well as low content of gold predetermine the open mining of the deposit deep to the economically feasible level.

Taking into consideration the fields' orientation the project considers the development of a single open pit.

10.2. Open Pit Mining

10.2.1. General Provisions and Determination of the Optimal Boundaries of the Open Pit Mining

Consolidated indexes as well as the ones obtained by the previous developments of similar deposits shall be used at this stage of project operations. For the present moment the studies of the pit slopes stability in terms of the ore and rocks have not been executed yet.

The height of ore and waste benches were determined on the analogy and equals to 10 m. Under the ultimate positioning of the slope (reclamation) the benches shall be tripled; the height of the tripled bench amounts to 30 m.

The slope angle of working benches also amounts to 75°, in the ultimate position - 70°.

The general slope angle of the open pit amounts to 47°-50°.

According to the Unified safety regulations the width of the safety bench (at least 1/3 of the height of the bench), and taking into consideration the height of the tripled benches (H=30 m) would be 10 m.

According to the current international practice the feasibility study of the open pit boundaries is made on the basis of results obtained by the block modeling of the deposit using special software.

Calculation of reserves based on block modeling was used under the development of the given TER. Determination of the open pit boundaries in the TER was also made on the basis of the block model and the process to develop the optimal open pit envelope using Micromine software (License MM1163).

Table 10-1 Basic data to develop the optimal open pit envelope

Parameters	Unit	Index
Ore mining	RUR/t	50
Rock mining	RUR/m3	120
Bulk weight of ore/rock	t/m^3	2.6
Dilution during mining	unit fraction	1.03
Extraction under mining (losses)	unit fraction	0.97
Recovery under processing	unit fraction	0.85
Cost of processing	RUR/t	475
Slope angle of the open pit	deg.	50
Au cost	RUR/t	1700

Modeling of the open pit contour with account of the mining parameters was made based on the chosen boundaries. (Graphical annex No.1, “The open pit as of the end of development”, Scale 1:2,000).

Figure 10-1 Appearance of the optimal open pit envelope and block model

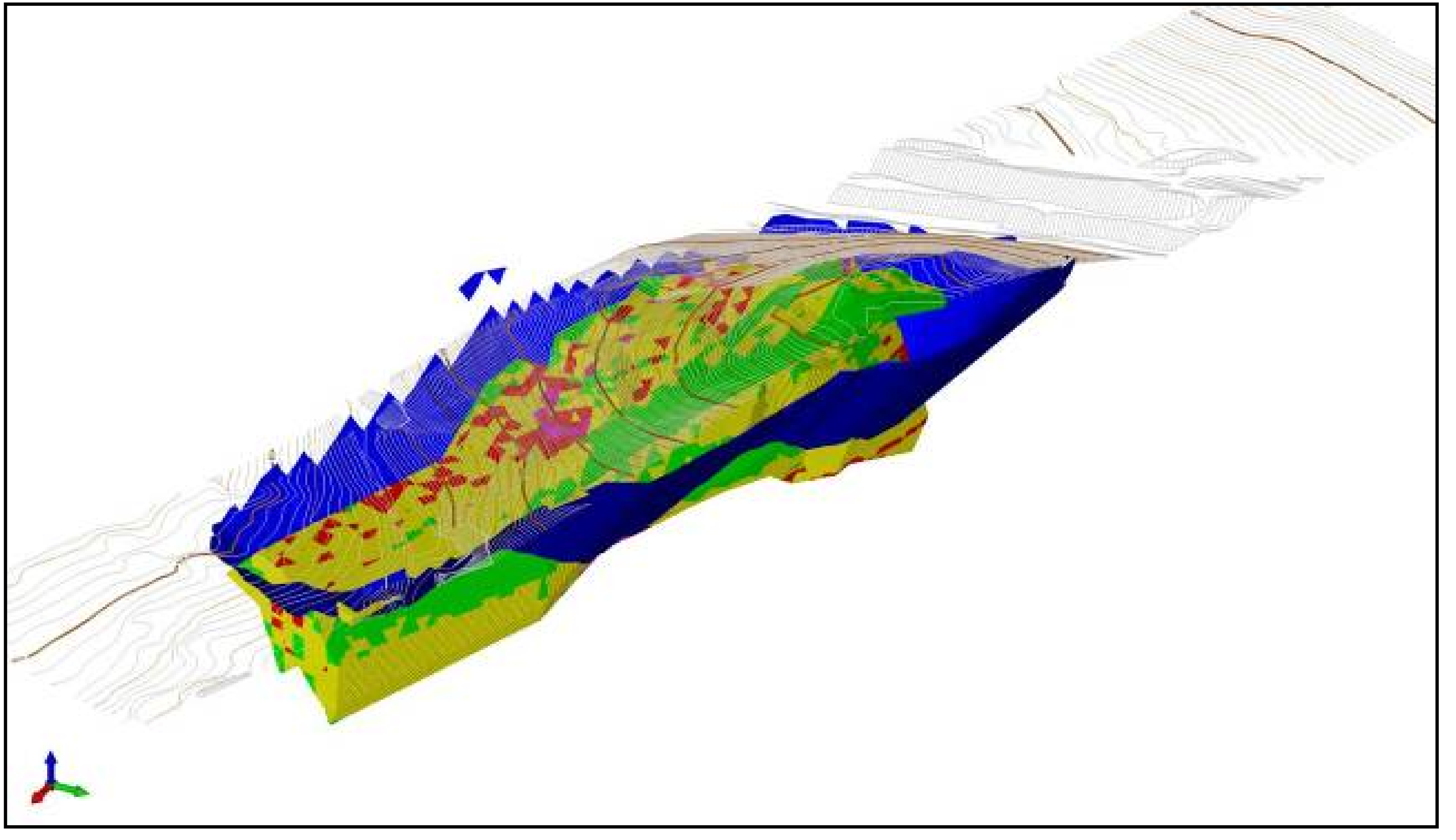


Figure 10-2 Open pit as of the end of development. 3D view

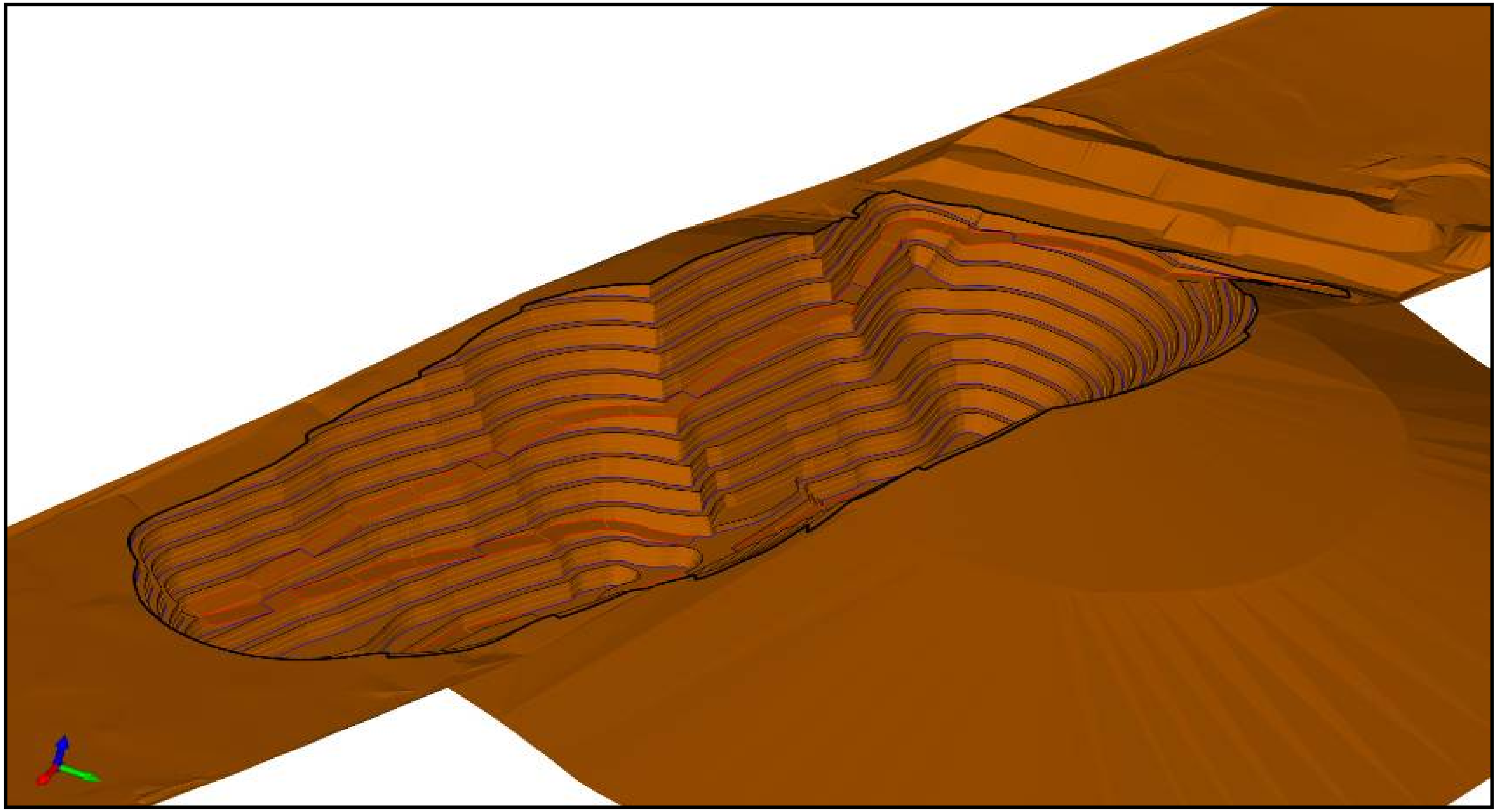


Figure 10-3 Plan of the open pit as of the end of development.

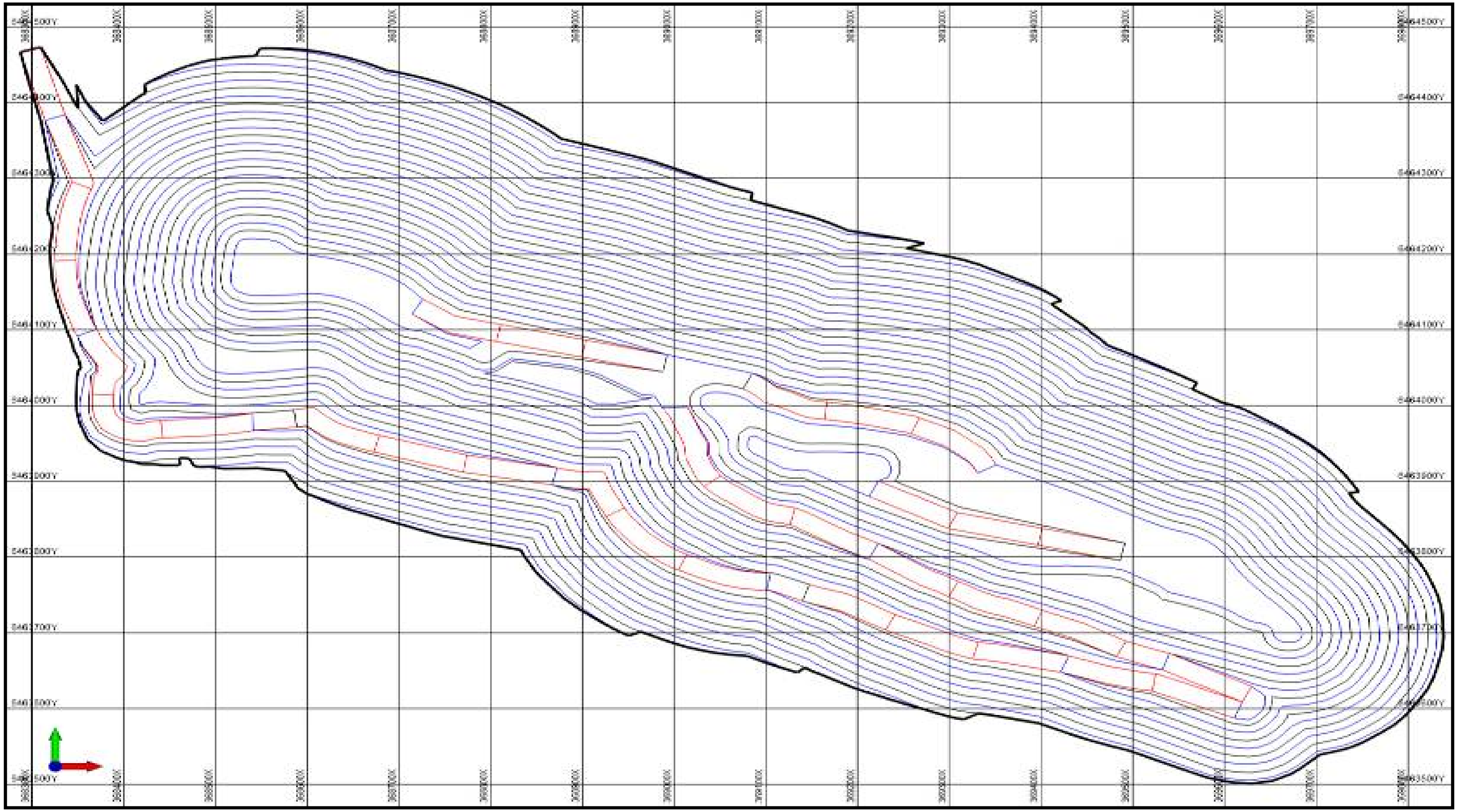


Table 10-2 Basic parameters of the open pit

Parameters	UoM	Values
Length of the open pit on the surface	m	1650
Width of the open pit on the surface	m	600
The maximum elevation of the slope on the surface	m	1025
The minimum elevation of the slope on the surface	m	870
Open pit bottom level	m	620
The depth of the open pit (maximum)	m	405
Area	ha	81.4
Height of the working ore bench	m	10
Height of the working ore rock bench	m	10
Working bench slope angle	deg.	75
Height of the reclamation bench	m	30
Slope angle of the reclamation bench	deg.	70
General slope angle of the reclamation bench	deg.	47-50
Run-of-mine rock within the pit envelope	th. m^3	132,550.0
Stripping rock volume	km^3	122,280.8
Ore reserves within the pit envelope	Kt	26,700.0
The reserves Gold grade	g/t	1.5
Gold reserves within the pit envelope	kg	40,746.7
Average cut-off grade throughout the open pit	m^3/t	4.6

Parameters of the open pit would be adjusted in the course of its development using the data of prospecting and further study of physical and mechanical properties of rocks and geological survey observations of the slopes' condition.

Under determination of the ultimate development depth to model the designed open pit the revision of the optimal form of the pit was executed by means of comparison of the ultimate (K_{gr}) and average (K_{sr}) stripping ratio.

The maximum permissible in terms of economical feasibility the depth of the open mining would be determined on the basis of calculation of the ultimate maximum permissible stripping ratio "K_{gr}", which would be determined based on the equation of the cost of the saleable product recovered from 1 ton of ore "Ts_i", all the product costs "Z" per its extraction and processing, considering indirect costs as well.

Under "Ts_i" $\geq$ "Z" extraction of ore above the pit bottom, determined by the ultimate maximum permissible stripping ratio "K_{gr}" would be break even. The recovered gold value per 1 ton of ore would amount to:

$Ts_i = Ts_{z3} \times S_3 \times K_{izvl} \times K_{aff} \times K_{nalog}$, RUR.

In which:

Ts_{z3} – gold price per 1 gram at the precious metal market, RUR;

S_3 – ore gold grade, g/t;

K_{izvl} – gold recovery ratio under processing of ore at the gold processing plant (ZIF);

K_{aff}– ratio, Considering refining costs;

K_{nalog} – ratio, considering mineral extraction tax.

Total costs per 1 ton of ore would be calculated as follows:

$Z= Z_d+Z_p+Z_{okhr}+Z_{am}$, RUR

In which:

Z_d- mining costs per 1 ton of ore, excluding the rock stripping, RUR;

Z_p – processing costs per 1 ton of ore at the ZIF, RUR;

Z_{okhr} – general duties costs, RUR;

Z_{am} – depreciation costs, RUR.

Upon calculation of “Ts_i" and "Z" values the maximum permissible ratio ("K_{gr}") would be determined:

$K_{gr}=\dfrac{Ци-З}{Св}$, m^3/t,

In which: S_v – costs per 1 m^3 of stripping, RUR.

According to the calculations K_{gr} would amount to 9.48 m3/t, which is more than the average stripping ratio calculated under determination of the pit boundaries modeled using the optimal open pit envelopes. This fact is indicative of the deposit promising potential.

10.2.2. Industrial Capacity and Life of the Open Pit

According to the project mining operations at the open pit would be executed on a year-round basis.

Taking into consideration a 7-days work week, the number of work days per year would amount to 350, a number of work shifts per day – 2, the shift continuity – 12 hours (working shift – 11 hours). The annual productive capacity was calculated using the Taylor formula (preliminary estimation of the potential annual productivity of the open pit was made considering the optimal continuity of the enterprise operation) and with regard to geotechnical possibilities.

The continuity of the mine operation would be determined by Taylor empirical formula:

$$T=0{,}2\cdot A^{0{,}25}$$

or

$$T = 6{,}5 \cdot B^{0{,}25}$$

in which: T – life of the mine, years;

A – ore reserves, t;

V – ore reserves, Mt.

The calculation of the LoM using the Taylor formula is given in Table 10.3

Table 10-3 Calculation of the LoM using the Taylor formula

Parameters	UoM	Formula, Designation	Value
Ore reserves within the pit envelope	t	A	26,699,910
Ore reserves within the pit envelope	Mt	B	26.7
Period of the open pit development using the Taylor formula (1)	year	T=0.2*A,0.25	14.8
Period of the open pit development using the Taylor formula (2)	year	T=6.5*V,0.25	14.6
Period of the open pit development	year	Tsr	14.1

Moreover, the productivity and LoM were calculated on the basis of geotechnical conditions.

The capacity of the open pit in terms of ore on the basis of geotechnical conditions was calculated according to the formula:

$$Ag = h_r \cdot S \cdot \eta_o \cdot (1+r_o),\ m^3$$

in which: hr – average annual reduction of mining operations, m;

S - the ore body average area, m2;

η0 – ore extraction ratio, represented as a fraction.

ro – ore dilution coefficient, represented as a fraction.

The average annual reduction of mining was determined by formula:

$$h_r = h_b + \Delta h,\ m/year$$

in which: h_b –average annual rate of mining reduction, m/year;

Δh – excavator class correction, m/year.

Basing on the calculated annual productivity by separate sites and accounting engineering potential the following values of the project capacity are accepted

Table 10-4 Determination of the annual productivity in terms of the annual reduction

Parameters	Formula, Designation	Value
Basic average annual rate of mining operations reductions, mpa	hб	19.5
Excavator class correction, mpa	Δh	1.6
Estimated average annual reduction of mining operations, m/year	hr= hб+Δh	21.1
Bulk weight of ore in situ, t/m^3	γ	2.6
The ore body average area, m^2	S	27,460
Losses, %	П	3.00
Dilution, %	P	3.00
Oil Recovery Factor, unit fraction	ηo	0.970
Ore dilution factor, unit fraction	ro	0.030
Estimated average annual capacity of the open pit in terms of the ore mining, m^3/year	Ar = hr · S · ηo · (1+ro)	578,885
Estimated average annual capacity of the open pit in terms of the ore mining, t/year	Qp= Ar · γ	1,505,101
Accepted average annual capacity of the open pit in terms of the ore mining, t/year	Qп	1,500 .0
Reserves within pit contours, Kt		26,699.9
LoM, years		17.7

The calculations show that the LoM and the annual productive capacity are different. Since the annual reduction calculation uses such parameters as equipment and the structure of ore beds, we will assume the LoM equal to 18 years in this report, taking into consideration reaching the design capacity and reduction of mining operations.

10.2.3. Stripping of the Deposit

Development of land (forest clearing, stump and shrub removal) would be executed prior to stripping operations within the frames of the open pit development. According to GOST 17.4.3.02-85 “Environmental protection. Land use”, removal and stockpiling of the said layer is not projected due to its low thickness (0.05-0.10 m).

Stripping of production levels of the open pits would be made by inclined and horizontal trenches. Open trenching would be made along the strike of the ore bodies per each site.

The stripping method would be determined with account of geotechnical conditions based on the adopted development system at the deposit and the type of transport. The deposit stripping considers drifting of mine workings facilitating access to the ore bodies. Since the ore bodies outcrop onto the surface extraction of ore would be commenced since the first year of the open pit operation.

According to the project preparation of rocks would be made by drilling and blasting.

Drilling of boreholes would be made by SBSh-250 rig (boreholes Ø - 250 mm) both through the ore and waste rock intervals.

Arrangement of benches and slopes of the open pit on to the final contour pre-splitting would be made by means of preliminary slotting.

Front-shovel excavators EKG-5 with bucket capacity $5m^3$ would be utilized for loading the rock mass. Transportation of the blasted ore would be made by pit motor dump trucks BelAZ 7540 with capacity 30t. EKG-10 (bucket capacity $10m^3$) and BelAZ 7555E (55 t) would be used for stripping operations.

Permanent and temporal inclined ramps would be arranged to develop the lower levels of the open pit. Berm limiting gradient was accepted to be equal to 100‰.

Under the divided motion the width of the berm (ramp) was accepted to be equal to 23m.

Capital mining development would facilitate access to the ore body at the beginning of development. Since the ore bodies outcrop onto the surface capital mining considers stripping operations to provide with the opened reserves which are registered by the calendar schedule of the operations. For the period of the open pit commissioning its provision with striped, prepared and ready for extraction reserves of ore would meet the VNTP 35-86 standards.

Table 10-5 Standards of reserves availability

Period of the Open Pit Operation	Reserves Availability, months		
	Stripped	Prepared	Ready for Extraction
Putting into operations	12.0 - 6.0	6.0 - 4.0	1.5 - 0.5
Design capacity operation	7.0 - 4.5	3.0 - 2.0	1.5 - 1.0
Reduction of mining	4.5 - 3.5	3.5 - 1.5	1.0 - 0.5

Primary mining would be executed within the period of the open pit operation and include:

- construction of motor access roads (inclined trenches) onto the opened levels of the high part of the open pit;
- arrangement of stripping trenches on to the working levels of the open pit;
- stripping operations to prepare reserves for extraction.

The scope of capital mining would provide the enterprise with the opened reserves for the period of at least 6 months, with the prepared reserves – for the period of at least 4 months, ready for extraction reserves – for the period of at least 0.5 months. Within the period of the design capacity operation these parameters reduce.

The designed open pit is referred to the low-upland type, a part of the upper levels outcrop directly onto the relief. The open pit will be developed within the frames of a single site.

Transportation of stripping rock from all the levels of the open pit would be made by motor dump trucks through the capital and temporary ramps, transportation berms with permanent and temporary motor roads.

Capital ramps down the pit have internal laying. Formation of permanent ramps in the course of the open development is arranged along with ultimate positioning of the open pit slope. The main exit from the open pit on the waste dump is located in the north-north-western flank of the pit, the main ramp to the ZIF is located at the same place.

The width of permanent ramps is 23 m, transportation berms 23 m. Limiting gradient of the permanent ramps is 0.1. According to SNiP 2.05.07-91* “Industrial transport” the angle of temporary ramps may increase the said value.

The opening of the consequent working levels of the open pits would be executed by open trenching along the ore bodies.

The procedure of the open pit development considers simultaneous development of all the sites. The development will be arranged in the manner avoiding concentration of mining operations at the isolated site of the open pit.

Since the ore bodies outcrop onto the surface the scope of capital mining includes stripping operations facilitating thus transportation of ore to the plant in the first year of the open pit development already. Stripping rocks would be utilized to arrange the floodslope at the tailings pond and water reservoir dam, as well as to fill the roads. These scopes will be developed by the principal process equipment and considered by the calendar schedules of mining operations.

10.2.4. Pit Performance Schedule and Approximate Schedule of Mining Operations

In accordance with norms of process design VNTP-35-86 the schedule of the open pit performance was adopted to be year-round:

Table 10-6 Pit performance schedule

Working days per year	350
Working days per week	7
Working shifts per day	2
Working shift continuity	11 hours

Synchronous operation of all the pit sites is considered by the calendar scheduling of the pit development.

The open pit output in terms of stripping during the entire operation period would facilitate stable ore production and irreducible normative of prepared reserves.

Table 10-7 Calendar schedule of operations

Description	Year of Development																		Total
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	
Rock mass, Mm^3	7.30	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	5.25	**132.6**
Stripping, Mm^3	6.72	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	6.92	4.79	**122.3**
Open pit mining, Mt	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.20	**26.7**
Stripping ratio, m^3/t	4.5	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.0	**4.6**
Average gold grade, g/t	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.4	**1.5**
Metal, t	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30	1.65	**40.7**

10.2.5. Development System

The given TER considers transport-and-dumping system of development with external dumping.

The stripping and mining operations consider involvement of the following principal equipment:

- Excavators EKG-10 – excavation of stripping rocks;
- Excavators EKG-5 – excavation of ore mass;
- Motor dump truck BelAZ-7555 with capacity 55 t – transportation of stripping rocks onto the waste dumps;
- Motor dump truck BelAZ-7540A with capacity 30 t – transportation of ore mass to the ZIF.

First loosening of the rock massif is executed by blasting. The height of the ore bench is accepted to be 10 m, considering minimal losses and dilution of mineral, the height of the rock bench is also accepted to be 10 m.

The major parameters of the development system are given in Table 10.8.

Table 10-8 Parameters of the development system

Parameters	UoM	Values
Height of ore bench	m	10.0
Height of rock bench	m	10.0
Rock and ore working bench slope angle	degree	75
Width of the opening trench	m	23.0
Width of the work area by ore and rock	m	30
Width of safety benches in the reclamation area	m	10
Width of transportation berm (ramp) under both-way traffic	m	23

10.2.6. Blasting Operations

Analysis of physical and mechanical properties of rocks determined the necessity of blasting within the frames of the open pits development.

Prior to holes drilling the block is subjected to preparation. Using a bulldozer the site is cleaned from snow, vegetation, rock piles and leveled for drilling operations.

For the purpose of drilling operations the company shall issue a drilling certificate for each block designed for the consequent blasting, specifying the following data: depth of the boreholes, drilling grid, quantity of boreholes and scope of drilling. Upon the drilling completion check measurement of the boreholes, survey and filling the project for huge blast would be made.

Mining and geological conditions of the deposit facilitate usage of simple explosives. Igdanite of local production (a mixture of ammonia nitrate and motor fuel) is used in the dry holes. Water will be accumulated in the blast holes during seasonal melting of the permafrost layer at the working levels of the open pit.

Considering igdanite water irresistibility, considerable depth of the boreholes, as well as potential flooding blasting agent in the watered boreholes will be waterproofed in the PE hose.

The project considers capless and short-delay blasting.

Detonating cord DShE-12 would be used as a demolition means. Stick powder ammonite primer No. 6 – ZhV placed in the blasting material charge acts as an initiator of the process. DSh circuit is electrically blasted at least by two electro-detonators ED-8-Zh.

Pyrotechnical relays RP-8 are used for the required number of delays. According to GOST 6285-74 water pipeline VP-0.8 with core diameter 0.8 mm is utilized as the main cable, blasting gear KPM-3 is used as a current supply source.

KZV circuit is transverse with alternate sequence. Blasting materials would be delivered to the site by MZ-3B automobile.

The calculation of drilling and blasting parameters considering the outcropping rock properties was executed according to the methodology of FGUP Sojuzvzryvprom. This calculation is preliminary and would be considered as an estimation.

In the course of the project development all the calculations of drilling and blasting operations would be adjusted with respect to the properties of the encountered rock type at the given deposit, the results of pilot and production blasts, reach and technologies, best experience in the sphere of blasting operations and similar conditions, as well as requirements of the Unified safety regulations for blasting operations and other regulatory instruments, confirmed by or coordinated with Gosgortechnadzor of Russia.

Table 10-9 Calculation of parameters of drilling and blasting

	Parameters	Unit	Values
1	Hardness coefficient of rocks according to M.M.Protodyakonov		4
2	Group of soils according to SNiP		5
3	Annual volume of blasted rock mass	ths.m^3	7,500
4	Height of bench	m	10.0
5	Slope angle of bench	deg	75
6	Ore hole diameter	mm	250
7	Bulk weight of the blasted	ths.m3	2.60
8	Type of the utilized blasting agent (specific action of blast) Igdanite		0.96
9	Specific consumption of blasting agent (see normative reference book)	kg/m3	0.36
10	Quantity of blasted charges		6
11	Ratio of automatic charging of the boreholes (1.1), of manual charging (0.9)		1.1
12	Days per year of the drilling rig operation	week	52
13	The interval between blasting operations, days	days	7
14	length of the line of the lower resistance along the bench bottom (W)	m	8.4
15	Test parameter LNS (would be < or = W)	m	5.7
16	Distance between boreholes in a row	m	7.7
17	Distance between rows	m	7.7
18	Relative length of the charge	m	0.35
19	Length of the borehole	m	11.4
20	Length of the continuous charge in the borehole	m	3.9
21	Stemming length	m	7.4
22	Subdrill size	m	1.4
23	Weight of the blasting material in the borehole	kg	212.6
24	Blasting material per 1m of the borehole	kg/l.m	54.0
25	Rock mass output per 1lm of the borehole	m3/lm	51.9
26	Width of the rock pile under a single-row blast (from the bench toe)	m	13
27	Width of the rock pile under the multi-row positioning of the boreholes	m	52
28	Ultimate width of the face slope when the blast does not form the rock pile	m	9
29	Quantity of boreholes per one row	Pcs	60
30	Mass of blasting material simultaneously blasted (without delay)	kg	1,063
31	Total exploded material per a single blasting action	kg	58,384
32	Volume of the blasted block	m3	165,385
33	Height of the rock pile under 1-3 row blasting	m	8.0
34	Height of the rock pile under 4 and above row blasting	m	10.7
35	Maximum height of the dirt pile in the clamped environment	m	12.4
36	Length of the blasted block	m	422.1
37	Width of the blasted block	m	39.2
38	Quantity of the blasted blocks	block	52.0
39	Total lm of the boreholes	Thsd lm	213.0
40	Timing rational interval between charges	ms	32

The basic data for calculation of the drilling rig performance are the following:

- Working shift continuity – 11 hours.
- Number of working shifts per year – 700
- Required quantity of linear meters of boreholes – 213 klm
- Required design number of drilling rigs with account of maintenance time – 4 units

Calculation of safe distances was made in accordance with the methodology contained by section VIII of the "Unified safety regulations for blasting operations". The calculations considered the borehole relieving charges.

The calculations were made with account of identification of zones, dangerous for people, dispersion of separate lumps during blasting of the holes charges, seismic influence on buildings and constructions, as well as the effect of the shock wave and are given in Table 10.10.

Table 10-10 Calculation of safe distances

Parameters	Unit	Values
Distance hazardous for people in terms of dispersion of separate lumps of rock (the minimal acceptable radius of zone hazardous for people under boreholes charges 200m. See the Unified safety regulations for blasting operations (EBP))	m	110
Radius of zone hazardous for mechanisms	m	55
Radius of hazardous zone on slope hazardous for people (under the 30m slope, the minimum permissible radius of zone, hazardous for people would be at least 300m. See EBP).	m	181
Seismically safe distance under non-simultaneous blasting with timing at least 20ms between blasts of each charge	m	205
Safe distance in terms of poisonous gases affect, regardless wind	m	621

The radius of hazardous zone during drilling and blasting operations with respect to the relief was determined by calculation and amounts to 300m. The access of people to the open pit and the area of blasting operations would be permitted in accordance with the procedure adopted by the chief engineer of the enterprise.

10.2.7. Major and Auxiliary Equipment for Stripping Operations

To load the rock mass in the open pit excavators EKG -10 with bucket capacity 10 m^3 would be utilized, excavators EKG-5 with bucket capacity 5 m^3 are considered by the project for loading the extracted ore mass, SBSh-250 with drilling diameter 250 mm would be used for drilling blast holes.

The complete description and number of machinery used at the open mining are given in table:

Table 10-11 List of mining machinery for the open pit.

Item	Description	Quantity, pcs.
Major mining equipment		
1	Excavators EKG-10	3
2	Excavators EKG-5	1
3	Motor dump trucks BelAZ-7555E, capacity 55 t	24
4	Motor dump trucks BelAZ-7540A, capacity 30t	8
5	Drilling rig SBSh-250-MIA-32KP ø250 mm	4
6	Bulldozer T.35-01	3
7	Bulldozer T0.25-01	4
Auxiliary mining equipment		
1	Charging machine MZ-3B based on the Ural-4320 Machinery plant «Zvezda» city of Kaprinsk	1
2	Bench edging machine machine, auto hydraulic hoist VS-22.05 based of the Ural - 4320-40 chassis	1
3	All-season combined road machine VMKD-6983, based on Ural 4320	1
4	Maintenance machine PARM-4784 based on Ural-4320-1112-41 (OOO Avtomaster)	1
5	Auto grader DZ-98V (ZAO Chelyabinsk road-building machines, city of Chelyabinsk)	1
6	Mobile flood light tower EFA 650 with integral DPP lights capacity 4 x 1500 W (Endress)	1
7	Motor for crew transportation Ural Vakhta 3255-0010-01	1
8	Crane truck KS-557722, capacity 25 tones based on Ural-4320 chassis, OAO Avtokran, city of Ivanovo	1
9	Fuel truck ATZ-12 based on Ural-4320-40 OAO GrAz, p/o Grabovo	1

Maintenance of the open pit roads would be executed by road repair service.

The principal mechanisms to maintain and repair roads are a bulldozer, a motor grader and flushing machine for the summer period.

The structure of external intra-site motor roads is predetermined by permafrost conditions. The roads would be filled in a moderate regime, i.e. frontwards from the filled bed in winter period. The upper soil layer would not be removed, since it is used as a thermal protection and secures the roads from prethawing and erosion. Stripping rocks are projected to be used for motor roads arrangement.

The auxiliary process equipment was chosen with account of the following operational factors:

- technical characteristics of the equipment would correspond to physical and mechanical properties of developed rocks, conditions of their bedding and facilitate safety of mining operations;

- the equipment would meet the adopted mining technology, size of the open pit and its output.

Fuelling of the open pit machinery would be executed using fuel truck ATZ-12 based on Ural vehicle.

A crane truck would be utilized for handling operations, maintenance and repairs. Mobile motor workshop PARM-4784 based on Ural would be used for maintenance operations. Motor dump trucks and automobiles based on the Ural chassis would be utilized for auxiliary operations.

10.2.8. Dump Formation

The given TER considers bulk stockpiling of stripping rocks into the external dumps, located beyond the open pit contours. The internal dump formation will not be arranged at this stage of the deposit prospecting. The location of dumps is predetermined by the local relief and optimizing of the transportation leg. The dumps location is projected is the proximity of the open pit. The volume of dumps, calculated for the open pit with account of the loosening residual coefficient Kr.ost. = 1.3 would be 160 MMm3. Prior to the dumps formation preparation of the territory is executed, i.e. forest clearing, stump and shrub removal, and if required – removal of the upper loose layer of the soil.

The dumps formation would be made by bulldozers T.35-01. Transportation of ore to the dumps is projected along the process motor roads, parameters of which were determined basing on the involvement of heavy open pit motor dump trucks BelAZ-7555E with capacity 55t.

10.2.9. Personnel Number at the Mining Operations Efficiency

The open pit is one of production units of the mine.

The staff specification considers the maximum cooperation of such subdivisions as motor transport facility, logistics department, consumer service establishment, central bases etc.

The major production is represented by the following types of operations:

- drilling operations;
- excavation, loading of rock mass onto the motor dump trucks;
- transportation of rock mass by the motor dump trucks;
- leveling operations on the dumps and within the pits;
- man sledging;
- dump formation.

The adopted production procedure facilitates the best utilization of the equipment, optimal periodicity of the personnel involvement and reduction of the operational personnel.

The manpower at the open pit is given in the table:

Table 10-12 Manpower at the open pit

Item	Profession, Capacity, Specialty	Effective Number per Day, man
Engineering and technical personnel		
1	Pit superintendent	1
2	Chief Engineer	1
3	Resident geologist	1
4	Resident surveyor	1
5	Resident power engineer	1
6	Crew captain	2
Total I		7
II Industrial-production personnel		
1	SBSh-250 drilling rig operator	8
2	EKG-10 excavator operator	6
3	EKG-5 excavator operator	2
4	BelAZ-7555 operator	48
5	BelAZ-7540 operator	16
6	Hole man	4
7	T-35 bulldozer operator	6
8	T-25 bulldozer operator	8
Total II		98
III Auxiliary motor transport facility		
1	Water sprayer operator VMDK-6983	1
2	MDZ-3B charging machine operator	1
3	Operator of the Ural Vakhta for crew transportation	2
4	Operator of the maintenance machine PARM-4784 -1112-41	1
5	DZ-98V motor grader operator	1
6	ATZ-12 fuel truck operator	2
7	Hydraulic hoist VS-22.05 operator	1
8	Crane truck KS-557722 operator	1
Total III		10
Total for the mine		115

10.2.10. Protection of the Pit from Flooding and Surface Discharge Dewatering

Normally watering of the pits occurs by means of ground waters and atmospheric precipitations. For the current period no hydro-geological studies of ground waters presence have been executed.

To exclude watering by atmospheric precipitations from the water collection area of the open pits and surface discharge during snow melting the project considers the arrangement of surface trenches to drain the precipitations down the relief with jigging of suspended material in the collection ponds. After that the drain would be directed to the brooks. The volume of the rock mass extraction under the surface trench would amount to 3.0 km^3. Thus the water inflow into the open pits by means of the atmospheric precipitations is formed by means precipitations fallen directly onto the surface of the open pits and would be determined by formula:

$Q_{OC}=[O_C*\eta*F]:t$

in which:

O_C – volume of atmospheric precipitations in the warm period of the year (April - October) - 402 mm;

F – estimated area of the open pit throughout the surface, km^2- 800;

η - coefficient of surface run-off, unit fraction – 0.7;

t – period of atmospheric precipitations draining, days. - 160.

Watering of the open pits by means of atmospheric precipitations excepting evaporation and transpiration losses (“engineering load factor”) would amount to 59 m^3/hour.

To drain the daily maximum of 5% availability of the project one pumping unit of TsNS - 60-300 type is considered by the project. The apparent wattage of the latter is 76.00 kW. In this period the power capacities switch from excavators to pumping units of the open pit drain system. Flooding of the lower levels is accepted.

10.3. Site Plan

The required complex of production and social infrastructure will be arranged on the sites of the designed enterprise. Excepting the principal objects (mining operations site, mill site with a complex of hydro-technical facilities) it will contain the objects of power-, heat- and water supply, sewage system, communication and maintenance facilities as well as a camp.

For the present period no topographic survey of the entire site and surface plan have been made. Determination of areas to arrange the principal sites of the enterprise is impossible either so far. The given chapter of the technical and economic calculations is of generic character.

The designed objects would be connected with each other by motor roads and located on the four major sites:

1. **A site of mining operations** within the pit, waste rocks dump, settling pond for the pit and under-dump waters, surface and drainage trenches. Explosives storage located at the remote site.
2. **Mine site** within the gold beneficiation plant, objects of maintenance and storage facility and support systems.
3. **Site of the camp.**
4. **Site of the tailings pond.**

The site of mining operations contains the open pit, waste dump, settling pond for pit and under dump waters, located in the vicinity of the Krasny and the Teply Creeks drainage channel in the area of +860 +920 m elevation points.

The storage of explosives will be located 1.5km from the open pit exit.

To drain the channel of the Krasny and Teply rivers from the pit sites and dumps construction of drainage trench 1km long is designed. Upwards along the slope relative to the open pit a surface trench will made in order to prevent drainage of surface waters into the pit.

To allocate the major **mine site** the area with typically low relief is preferable, beyond the safety zones and as close to the pit site as possible.

The mine site hosts the following facilities:

1. Gold beneficiation plant;
2. Repairing and maintenance workshops for reparation of heavy duty trucks;
3. Repairing and parking box;
4. Auxiliary garage and parking for vehicles;
5. Storage for firefighting materials;
6. Pumping station of fire fighting water supply with reservoirs;
7. Materials storage with gantry crane;
8. Fuel and lubricants storage;
9. Checkpoint;

10. Solid wastes landfill.

The area to allocate the **camp** would also be chosen at the minimum distance from the pit site but beyond the safety zones of blasting operations

The camp contains:

1. Administration and amenities with a canteen, clinic and shower rooms;
2. Building of the operation assignment issue;
3. Dormitories of block-module type per 60 persons, each containing 5 buildings;
4. Biological treatment facilities BIODISK-350.

The tailings pond would be of surface type. Hydraulic stockpiling of tailings onto the dump would be made by means of dispersed discharge from the major dam throughout the entire length. The distribution slurry pipeline would be laid along the ridge of the head dam. The main slurry pipeline is laid in two rows, the distribution one – in a single row. The tailings pond reservoir is formed by a protective dam.

The feeding of tailings to the tailings pond is projected to be gravity-flowing.

The structure of the protective dam would exclude potential filtration of waters into the beds of rivers and brooks. The dam would be filled using the local clayey soils with arrangement of the impervious screen. The ridge of the screen (upon the final settlement of the dam) would be higher than the maximum water surface in the headwater with account of the height of the wave and wind-induced water level. Arrangement of the apron deep to 4 meters is projected to prevent the water drainage through the filtrating basement of the water reservoir bottom.

10.4. Accident Prevention, Occupational Sanitation, Fire and Explosion Safety Measures to Provide with Stability of the Pit Slopes

To meet the standards of industrial health under the open pit operations the project considers the following measures:

- the workers would be serviced in the amenity complex of the mine and provided with all the proper closing and shoes;
- the trailers at the open pit would be furnished with first aid kits;
- the workers at the open pits would be subjected to medical and preventive examination in due time;
- under dust content exceeding the sanitary norms, the workers would be equipped with the individual protection means;
- persons diagnosed for diseases which restrict their further involvement into the operations would be promptly transferred to other position in accordance with the conclusion issued by the medical examination commission.

Drilling and blasting operations would be executed in the open pit in accordance with the "Unified safety regulations for blasting operations" (PB-13-407-01). Mining operations in the open pits would be executed in accordance with requirements of the "Unified Safety Regulations for Open Pit Mining of Subsurface Resource (PB-03-498-02) and Sanitary Norms and Regulations". To enhance fire safety of mining operations the project considers the following measures:

- regular control of fire fighting means, their timely renewal and addition;
- upkeep of water reservoirs and pipelines;
- training programs by rules of behavior upon occurrence of fire, and use of fire extinguishers;
- extinguishing of potential fires in the open pit is carried out in accordance the accident response.

To facilitate preservation of the pit slopes' stability and continuity of their life (8 years) the following measures are considered:

- using of short-delay blasting at the stripping and mining operations, reducing the general seismic affect of the blast onto the massif;

- weight limitation of simultaneously blasted charge of the blasting agent by means of reduction of the charge weight to 2400 kg per a single delay along with the ultimate pit contour approach, under calculation per the total volume of rock mass in the open pits.

The study of geo-technical conditions determined the existence of unequal stability conditions at the various areas of the pit slopes. Control for the slopes' stability is executed by geological-survey service according to the following parameters:

- observation of the slopes deformations in the unstable areas;
- along with the mining down progressing the structural peculiarities of the massif are adjusted;
- the most deformed areas of the slopes are formalized;
- pre-splitting of benches by inclined boreholes would be developed experimentally;
- loosened areas of slopes would be identified timely and the optimal measures and parameters of the slopes reinforcement would be determined.

10.5. Mine Technical Reclamation

The object of reclamation is the mined-out area of the open pit, opening the workings, rock dumps and water drain tranches. Designated purpose: Future use of the reclaimed soil areas to preserve the landscape or for economic purposes.

The process flowsheet of mine technical reclamation and basic types of operations:

- selective extraction, stockpiling and storage of top soil in the temporary dumps;
- stockpiling of soil, formation of optimal parameters of dumps or fields of developed areas;
- leveling of the surface of the dumps, slope flattening of the pit slopes' slopes and dumps;
- arrangement of exits and roads;
- meliorative antierosion measures;
- filling and leveling of the top soil at the reclaimed sites.

11. Technology

Technology of the ore processing of the ore occurrence Krasnoye is recommended on the basis of the data, studies and tests arranged by OAO Irgiredmet with involvement of similar ores. Process flowsheets were calculated for the ZIF capacity by ore 1500 Kt per year under the year-round performance and gold grade 1.5 g/t and the equipment utilization factor 0.9. Saleable product is the Dore gold bars TU 117-2-7, production wastes are the tailings of flotation beneficiation and neutralized tailings of sorption cyaniding of concentrates.

Gold recovery to the bar amounted to 85.0%.

11.1. Results of Studies

Material composition, physical and mechanical properties and processability of ore by various methods was executed by OAO Irgiredmet in 2012 with involvement of a gold-containing sample No. 1 taken from the ore zone No.2 of the ore occurrence Krasnoye.

The ore of the ore occurrence Krasnoye is referred to the gold-quartz low-sulfate type of ores and characterize the primary ore type, since the oxidization degree calculated by iron amounts to 16%. Quartz and micaceous-hydromicaceous matters are the basic rock forming elements. Pirate with mass weigh 3.3% prevail among the ore minerals. The share of iron oxides and hydroxides is 0.5%.

The major commercial component of the ore sample is gold. The content of precious metal in the studied sample amounted to 2.5 g/t. The gold is native. The shape of gold grains is: Irregular compact with slips, plated with constrictions, lumpy with scales. Fine and fine dispersed gold prevail in the ore. It is relatively low- and moderately-high grade, its fineness varies from 777 to 851 units.

The studied ore is referred to hard cyaniding ores: the gold recovery amounts to 79.9% (less than 90%). The share of free gold amounts to 30.6%, the share of noble metal amounts to 49.3% and occurs in joints with ore and rock formation components. Obstinateness is basically conditioned by the association of gold with pyrite (7.8%) as well as by fine impregnations to rock-formation minerals (6.9%). Possibly the occurrence of carboniferous matter (2.8%) will require utilization of special methods during metallurgical processing.

To process the ore gravity – flotation beneficiation is recommended. It would facilitate recovery of gold at the level of 92.8 % (82.6 % - by gravitation and 10.2 % - by flotation).

Table 11-1 shows the results of ore testing using gravity-flotation beneficiation flowsheet, the results obtained by cyaniding of concentrates with and without pressure leaching are given in Table 11.2.

Table 11-1 Technological indexes of gravity-flotation beneficiation of ore

Process Products	Recovery %	Gold Grade, g/t	Recovery of Gold, %
Mixed gravity concentrate	1.67	120	82.6
Gravity tailings (flotation feeding) according to the balance/ (according to analysis)	98.33	0.43/ (0.6±0.17)	17.40
Flotation concentrate	1.25	20.2	10.22
Mixed concentrate	**2.92**	**77.2**	**92.82**
Flotation tailings	97.08	0.18	7.18
Ore according to the balance	100.0	2.43/ (2.5±0.5)	100.0

Table 11-2 Results of sorption cyanidation of flotation concentrate of the Krasnoye deposit

Item #	Au Content in Cake, g/t	Au Concentrate Content, g/t	Au Recovery to Solution, %	Chemicals Consumption, kg/t	
				NaCN	CaO
Sorption cyanidation of basic concentrate					
1	5.4	18.8	71.3	1.1	1.4
Sorption cyanidation of concentrate upon pressure leaching					
2	2.84	19.52	85.5	1.5	1.1

Pilot testing using direct cyanidation and pressure leaching of flotation concentrate with following cyanidation of cakes taken from the Krasnoye deposit demonstrated that the flotation concentrate is a refractory product in respect to the process of cyanidation. The recovery of gold under sorption leaching of basic concentrate amounts to 71.3%.

Pressure leaching (220°C, 3.0 MPa oxygen, 2 hours) facilitates almost complete oxidizing of sulphides (the degree of oxidizing is above 97 %);

The recovery of gold from the oxidizing products under sorption cyaniding amounted to 85.4 %.

The product of autoclave opening has sufficient sorption activity. Processing of this product can be arranged by sorption cyanidation only.

To open refractory concentrates of Krasnoye ore occurrence processes of bacterial leaching, heat leaching ore extra-fine grinding can be recommended.

The anticipated end-to-end gold recovery from ore will amount to 85.0 - 89.0 % under the gold grade 2.4 g/t.

11.2. Ore Processing

Basing on the studies data for the purpose of processing the ore of the Krasnoye ore occurrence the project considers gravitation-flotation flowsheet of beneficiation with sorption cyanidation of ground flotation concentrate and intense cyaniding of gravity concentrate.

11.2.1. Process Flowsheet of Ore Processing

Process flowsheet of the Krasnoye ore occurrence includes the following principal operations: crushing of base ore of size class 0-700 mm to the final class 80-85% minus 0.071 mm in two stages. The first stage is executed by semi-autogenous grinding mills operating in closed cycle with classifiers, the second stage is executed by ball mills with discharge through the screen, operating in closed cycle with hydro-cyclones. Water-slurry and quality-quantitative beneficiation flowsheet are shown in figures 11.1 and 11.2 correspondingly, the mode parameters of process flowsheet are given in table 11.3, metal balance – in tablee 11.4, water balance – in table 11.5, agents and materials consumption – in table 11.6.

Gravity concentration is executed in jigging machines and on the concentration tables. Gravity concentrates are subjected to intense cyaniding. The cake of intense cyaniding is mixed with middlings of gravity beneficiation and fed into the third stage of grinding into the ball mill with central discharge, performing in a closed cycle with hydro cyclone.

The tailings of gravity concentration, ground to size class 80-85% minus 0.071 mm are fed into flotation beneficiation by flotation machines of mechanical type (basic, test and middlings operations). Re-cleaner flotation is executed by flotation machines of mechanical type. The concentrate of re-cleaner flotation is mixed with further ground to size class 80-85% -0.071mm cake of intense cyaniding, thickened in the radial thickener and is fed into the mill of extra fining to size class 10 micron. The ground product is fed onto the agitators for lime treatment and further for sorption cyaniding. The tailings of sorption cyaniding are neutralized using calcium hypochlorite in the mechanical mixers.

Desorption of gold and electrolysis are executed by the equipment of All-China gold corporation. Upon desorption carbon is subjected to reactivation by the drum furnace.

Cathode deposits obtained by electrolysis are dried in the resistance furnace and heated in the induction furnace with obtaining of Dore gold bars.

Tailings of flotation beneficiation and sorption cyaniding are directed for storage in the proper tailings facilities.

Figure 11-1 Water-slurry flowsheet

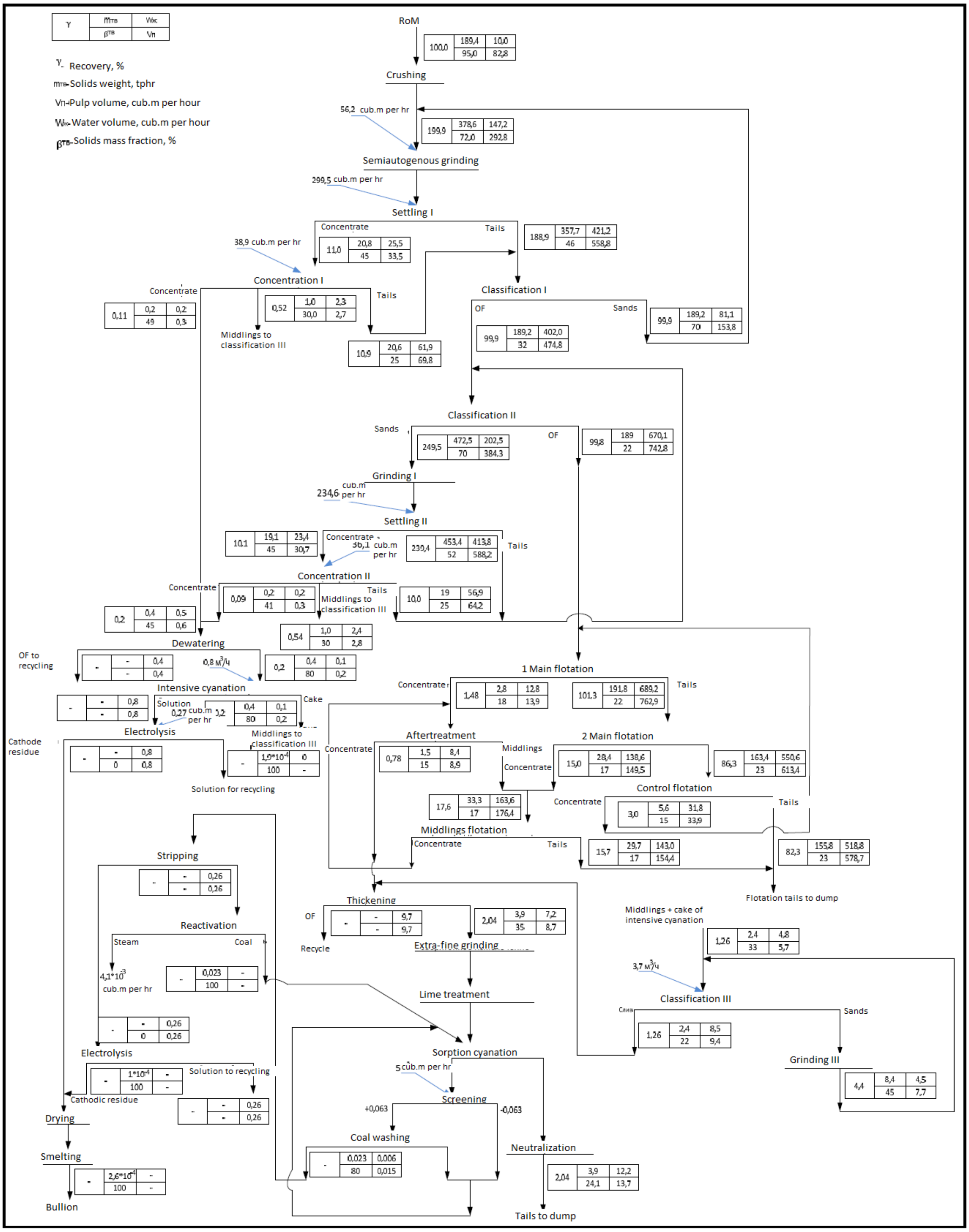


Figure 11-2 Qualitative-quantitative flowsheet

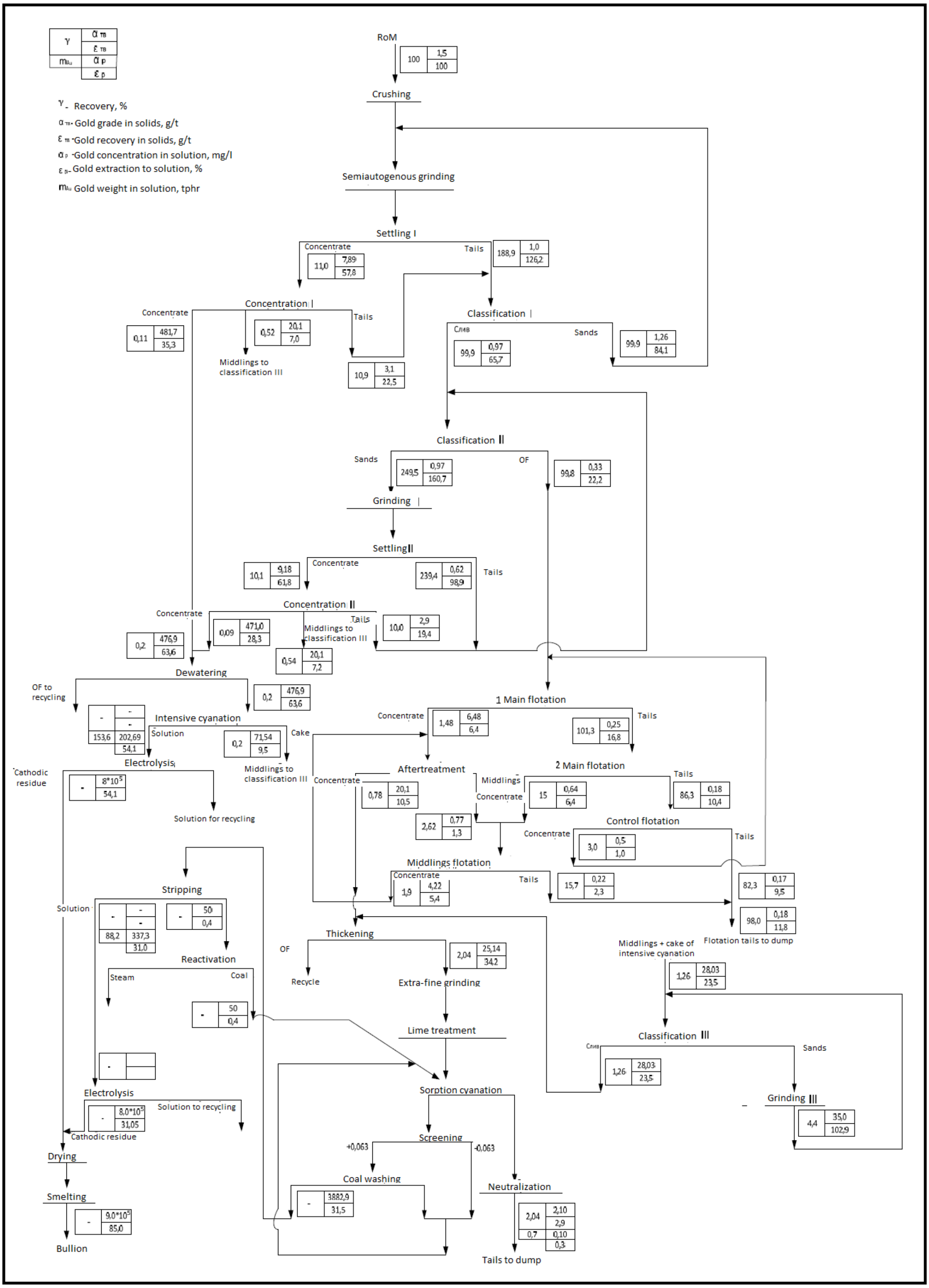


Table 11-3 Mode parameters of process flowsheet

Parameter	Parameter Value
General data	
Capacity of beneficiation plant	
t/year	1500000
t/day	4545.5
t/h	189.4
Ore gold grade, g/t;	1.5
Specific ore weight, g/cm^3	2.6
Humidity of base ore,%	5
Size class of base ore, no more than, mm	700
Mode of ore delivery	24 hours/day
Operation mode, days/year	330
Shifts/hours per shift	2/12
Obtained products	Dore gold bars TU-117-2-7, Beneficiation tailings Cyanide concentrates tailings Non-standard carbon
Amount of metal in ore (gold), kg/year	2300.0
Recovery, % To Dore gold bar; To beneficiation tailings; To tailings of cyanide concentrates.	 85.0 11.8 3.2
Ore preparation and beneficiation	
Crushing:	
Apron feeder	BW140-8 (China)
Electric engine capacity, kW	15
Quantity, pcs.	1
Vibration feeder	ZSW-380X95II
Engine capacity, kW	15
Number, units	2
Crushing stages	1
The biggest size of lumps in feeding, mm	700
Type of crusher: Jaw crusher with complex jaw motion Size of discharge slot, mm Grain size, mm	 ShchDS 9x12 130 0-700
Electric engine capacity, kW	100
Quantity	1
Lining consumption , kg/t of ore	0.1
Grinding:	
Number of grinding stages	2
Stage I: Mill type: semiautogenous grinding mill:	MZ 64x33
Working Volume of the mill, m^3	107
Diameter of balls, mm Filling of balls with respect to the mill capacity, % Weight of balls, t	100-120 10 67
Electric engine capacity, kW	2000

Parameter	Parameter Value
Quantity, pcs.	2
Weight class minus 0.074 mm in the mill discharge, %	20-25
Efficiency of the mill with respect to the basic feeding, t/h: Section 1 Section 2	 94.7 94.7
Specific capacity by size class minus 0.074mm, th/m^3 Section 1 Section 2	 0.5-0.6 0.5-0.6
Size class of base ore, mm	100-160
Solid mass fraction at the balls' discharge, %	70
Circulating load, %	100-120
Specific consumption of balls, kg/t of ore	0.9
Specific consumption of lining, kg/t of ore	0.1
Stage II: Mill type: Ball mill with discharge through the screen:	 MQG 4060
Working mill capacity, cub.m	69.8
Ball diameter, mm Filling of balls with respect to the mill capacity, % Weight of balls, t	60-80 40-45 120-130
Electric engine capacity, kW	2000
Quantity, pcs.	2
Weight class minus 0.074 mm in the mill discharge, %	35-40
Specific capacity by size class minus 0.074mm, th/m^3 Section 1 Section 2	 0.5-0.6 0.5-0.6
Size class of base material, mm	- 0.5
Solid mass fraction at the balls' discharge, %	70
Circulating load, %	250-260
Specific consumption of balls, kg/t of ore	1.0
Specific consumption of lining, kg/t of ore	0.1
Stage III: Mill type: Ball mill with central discharge:	 MQY 1530
Working Volume of the mill, m^3	4.0
Diameter of balls, mm Filling of balls with respect to the mill capacity, % Weight of balls, t	40-60 35 6.0
Electric engine capacity, kW	110
Quantity, pcs.	1
Weight class minus 0.074 mm in the mill discharge, %	45-50
Specific capacity by size class minus 0.074mm, th/m^3	 0.4-0.5
Size class of base material, mm	- 2
Solid mass fraction at the balls' discharge, %	60
Circulating load, %	300-350
Specific consumption of balls, kg/t of ore	0.7
Specific consumption of lining, kg/t of ore	0.1
Extra fining of concentrates	

Parameter	Parameter Value
Mill type - IsaMill	M1000
Capacity, m^3	1.0
Electric engine capacity, kW	500
Quantity, pcs.	1
Efficiency of the mill with respect to the feeding, t/h:	3.9
Electricity consumption, kWh/t of concentrate	57.7
Final size class 95-98 %, micron	10
Solid mass fraction at the balls' discharge, %	35-40
Diameter of grinding bodies, mm	1.5-6
Coefficient of filling the mill with grinding bodies, %	70-80
Specific consumption of grinding bodies, kg/kWh	0.07
Jigging:	
Jigging 1	
Input: Output of the ball mill, t/h	378.6
Type of equipment Jigging machine	MOD-3M
Electric engine capacity, kW	2x2.2
Quantity, pcs. Section 1 Section 2	6 3 3
Specific efficiency, t h/m^2	20
Stroke length of deck, mm	15-20
Stroke per minute	240-260
Jig shot size, mm	5-6
Jig shot thickness, mm	50
Jig shot consumption per ton of material, kg	0.08
Maximum size of grains in the basic feeding, mm	10
Consumption of under screen water (to the process), l/sec including:	10
Jigging II	
Input: Output of the ball mill, t/h	472,5
Type of equipment Jigging machine	MOD-3M
Specific efficiency, t h/m^2	25
Stroke length of deck, mm	8-10
Stroke per Minute	240-260
Jig shot size, mm	5-6
Jig shot thickness, mm	50
Jig shot consumption per ton of material, kg	0.08
Maximum size of grains in the basic feeding, mm	1.0
Consumption of under screen water (to the process), l/sec	65
Concentration on the tables	
Concentration I	

Parameter	Parameter Value
Type of table	SKO-7.5
Electric engine capacity, kW	1.1
Quantity, pcs. Section 1 Section 2	6 3 3
Effective area of beneficiation, m^2	7.5
Specific efficiency, t h/m^2	0.5
Consumption of water per 1t of feeding, m^3	1.8
Stroke length of deck, mm	16-20
Stroke frequency of deck, min^{-1}	280
Feeding size, mm	2
Transverse pitch of deck, degrees	3-5
Concentration II	
Type of table	SKO-7.5
Electric engine capacity, kW	1.1
Quantity, pcs. Section 1 Section 2	6 3 3
Effective area of beneficiation, m^2	7.5
Specific efficiency, t h/m^2	1.3
Consumption of water per 1t of feeding, m^3	1.8
Stroke length of deck, mm	20-26
Stroke frequency of deck, min^{-1}	350
Feeding size, mm	1
Transverse pitch of deck, degrees	3.5-5
Classification	
Classification I:	
Type of equipment Spiral classifier	1KSN-30
Electric engine capacity, kW	30
Quantity, pcs. Section 1 Section 2	2 1 1
Solid mass fraction, % In the overflow, In the underflow.	 45-55 80
Weight class minus 0.074 mm in the mill overflow, %	50-55
Classification II	
Type of hydro cyclone	GTs 500
Quantity, pcs. Section 1 Section 2	20(10 working +10 backup) 10(5 working +5 backup) 10(5 working +5 backup)
Solid mass fraction in the overflow, %	20-25
Weight class minus 0.074 mm in the mill overflow, %	80-85
Solid mass fraction in the underflow, %	70
Weight class minus 0.074 mm in the underflow, %	15-20

Parameter	Parameter Value
Classification III	
Type of hydro cyclone	GTs -250
Quantity, pcs.	2(1 working +1 backup)
Solid mass fraction in the overflow, %	22-25
Weight class minus 0.074 mm in the mill overflow, %	80-85
Solid mass fraction in the underflow, %	60
Weight class minus 0.074 mm in the underflow, %	15-20
I Principal flotation	
Mechanical mixer	KCh 16
Quantity, pcs.	2
Electric engine capacity, kW	15
Type of equipment Flotation machine (q-ty of vessels, pcs.) Section 1 Section 2	 FMP-16 (2) FMP-16 (1) FMP-16 (1)
Frothing agent consumption, g/t	70
Electric engine capacity, kW	20
PM-2 consumption	125
Flotation time, min	2
II Principal flotation	
Mechanical mixer	KCh 16
Quantity, pcs.	2
Electric engine capacity, kW	15
Type of equipment Flotation machine (q-ty of vessels, pcs.) Section 1 Section 2	 FMP-16 (8) FMP-16 (4) FMP-16 (4)
Frothing agent consumption, g/t	30
Electric engine capacity, kW	20
PM-2 consumption	12.5
Flotation time, min	8
Recleaner flotation	
Mechanical mixer	KCh 16
Quantity, pcs.	2
Electric engine capacity, kW	15
Type of equipment Flotation machine (q-ty of vessels, pcs.) Section 1 Section 2	 FMP-16 (10) FMP-16 (5) FMP-16 (5)
Frothing agent consumption, g/t	50
Electric engine capacity, kW	20
PM-2 consumption, g/t	12.5
Flotation time, min	15
Middlings flotation	

Parameter	Parameter Value
Type of equipment Flotation machine (q-ty of vessels, pcs.) Section 1 Section 2	 FMP-8.5 (2) FMP-8.5 (1) FMP-8.5 (1)
Electric engine capacity, kW	18.5
Flotation time, min	5
Re-cleaner flotation	
Type of equipment Flotation machine	FMP-10
Electric engine capacity, kW	1.1
Quantity of vessels, pcs.	2
Chip separation	
Type of equipment Inertia screen	GIL-21
Electric engine capacity, kW	7.5
Quantity, pcs.	1
Water consumption, m^3/h	2.0
Mesh size, mm	0.5
Concentrates thickening	
Type of equipment Radial thickener	 S-9
Electric engine capacity, kW	4
Quantity, pcs.	1
Specific efficiency of thickening, t/m^2h	0.1
Solid mass fraction in the thickened product, %	35-40
Intense and sorption cyaniding	
Lime treatment	
Required continuity, h	5
pH of pulp	11
Consumption of lime per treatment, kg/t	10
Pulp flow, m^3/h	8.8
Required capacity of tanks for lime treatment, m^3	30
Quantity of machines , pcs.	2
Recommended type of machine	Agitator with mechanical mixer (China)
Useful capacity of a single machine, m^3	25
Electric engine capacity, kW	5
Sorption leaching	
Sodium cyanide consumption, kg/t of concentrate	5.0
Solid mass fraction in the pulp, %	35
Pulp efficiency, m^3/h	8.7
Required continuity of cyaniding in sorption, h	24
Required continuity of sorption process, h	300-320
Proper capacity of sorption machines, m^3	30

Parameter	Parameter Value
Proper quantity of sorption stages, at least	10
Recommended type of machine	Agitator with mechanical mixer (China)
Useful capacity of a single machine, m^3	25
Actual quantity of sorption machines, m^3	10
Electric engine capacity, kW	5
Gold extraction with the CIP process, %	31.5
Drainage mesh size, mm	0.63
Air consumption per one sorption machine, m^3/min	1.0
Au concentration in a liquid phase of sorption tailings, mg/l	0.1
Solid mass fraction in the sorption tailings, %	30-35
Au content in the solid phase of sorption tailings, g/t	3.77
Recommended type of coal	China or Malaysia origin
Coal size class, mm	+1.6
Coal bulk weight, g/cm^3	0.52
Coal flow, l/h	44-45
Total weight of coal, fed into desorption, t/day	0.6
m^3/day	1.2
Total feeding of coal into the machines, t	6.5
Concentration of coal in the pulp, g/l	5.0
Coal losses per 1t of concentrate, g	200
Cleaning of coal from pulp	
Screen type	GIL-21
Screening area, m^2	2
Electric engine capacity, kW	7,5
Mesh size in the screen, mm	0.63x0.63
Water consumption, m^3/h	5
Cleaning of coal from silts and chips	
Type of equipment recommended the coal cleaning	Washing column
Linear speed of water through output, m^3/h	25,0-30,0
Expansion of coal layer, %	Up to 30
Volume of water per 1 m^3 of coal, m^3	5
Test screening of sorption tailings	
Type of test screen	GIL-21
Mesh size of screening surface, mm	2
Electric engine capacity, kW	7.5
Specific load by pulp, $m^3/m^2 \cdot h$	0.63x0.63
Screening area, m^2	2
Water consumption, m^3/h	0.4
Desorption, electrolysis and heat	
Capacity by coal, m^3/day.	1.1

Parameter	Parameter Value
Acid treatment, desorption, electrolysis	Equipment of All-China gold corporation.
Capacity by coal per 1 cycle, m^3	2
Quantity of the equipment sets, pcs.	1
Operating mode	Periodic
Acid treatment	
Type of equipment	Feeding bunker with drainage unit
HCl concentration in the base solution, g/l	30-35
HCl consumption (37 %) per 1t of coal, l	up to 100
Treatment continuity, h	1
Coal neutralizing	
Number of solution volumes per the coal volume, m^3	1.5
Alkalis concentration in the neutralizing solution, g/l	2,5-3
NaOH consumption, kg/t of coal	130
Desorption of gold and electrolysis	
Useful capacity of a column of desorption, m^3	2
Temperature in the desorption machine and electrolyte unit, ^oC	145
Tank of recovered carbon by gold, mg/g	0.05
Continuity of desorption and electrolysis, h	12
Recovery of metals under desorption, %, at least	98
Residual gold concentration in the solution after electrolysis, mg/l	3-5
Total continuity of 1 cycle of acid treatment, coal neutralizing, desorption and electrolysis, h	16-24
NaOH consumption per 1 t of coal, kg	50
Total electric power of the unit, kW	200
Thermal reactivation	
Capacity by coal, t/day	0.6
Recommended furnace type	ZSL-125 (China) - Efficiency 125 kg/h
Maximum internal temperature of the furnace, ^oC	900
Operational internal temperature of the furnace, ^oC	650
Electric power of the unit, kW	63
Coal screening after reactivation	
Screen type	GIL-21
Screening area, m^2	2
Volume of water per cleaning, 1 m^3 of coal, m^3	0.4
Sieve Size (mm)	0.63x0.63
Losses of sorbent under reactivation and screening, %	10
Dewatering and intense cyanidation of gravity concentrate, electrolysis	
Efficiency, t/h	0.4
Type of equipment	CS 500
Sodium cyanide consumption, kg/t of gravity concentrate	10

Parameter	Parameter Value
Electric power of the unit, kW	40
Heating section	
Resistance furnace	SN3-8.15.5/12I4
Electric power, kW	75
Induction furnace	PPI-25
Electric power, kW	25
Recovery of gold to the bar, %	85.0
Neutralization	
Unit type	Agitator China
Volume,m^3	15
Electric engine capacity, kW	10
Quantity, pcs.	3
Active chlorine consumption (100%), kg/t of cyanide tailings	33.8
CaO consumption (100%), kg/t of cyanide tailings	11.6

Table 11-4 Metal balance

Input		Output	
Parameter	Value	Parameter	Value
RoM, Kt/year Gold grade, g/t Gold weight, kg	1500.0 1.5 2300.0	Dore gold bars, kg/year Gold grade, g/t Gold weight, kg/year Recovery, %	2,172.0 900,000.0 1,955.0 85.0
-	-	Output of beneficiation tailings, % Beneficiation tailings, Kt/year Gold grade, g/t Gold weight, kg/year Losses with beneficiation tailings, %	98.0 1469.4 0.18 265.5 11.8
-	-	Output of cyanidation tailings, % Beneficiation tailings, Kt/year Gold grade, g/t Gold concentration, mg/l Gold weight, kg/year Losses with cyanidation tailings, %	2.0 30.6 2.1 0.1 72.0 3.2

Table 11-5 Water balance for beneficiation flowsheet

Water Input, m^3/day		Water Output, m^3/day	
With base ore	10	With beneficiation tailings	661.8
Into grinding	56.2	With cyanidation tailings	12.2
Into jigging machines	534.1	Thickeners overflow	10.1
On to the concentration tables	75.1	Spillage solutions	1.0
For classifying	3.7	-	-
Screening	5	-	-
Intense cyaniding etc.	1.0	-	-
Total	685.1	Total	685.1

Process water consumption per 1t of ore in the process of preparation and concentration amounts to 3.6m^3. Considering the consumption of water for transportation of concentrates, middlings, washing of floors and machinery (10% of the total consumption for technology), the total consumption of process water would amount to 4.1m^3/t of ore, including fresh water 0.4 m^3/t.

Table 11-6 Consumption of reagents and materials

Parameter	Value
Consumption of mills' lining, kg/t of ore	0.4
Hammered balls consumption, kg/t of ore	2.6
Consumption of grinding bodies for extra-fine grinding, kg/t of ore	0.1
Consumption of jig shots, kg/t of ore	0.08
Butyl xanthate consumption, g/t of ore	150
Frothing agent PM-2 consumption (100%), g/t of ore	150
Sodium cyanide consumption (100%), g/t of ore	130
CaO consumption (100%), g/t of ore	500
NaOH consumption, g/t of ore	22
Active chlorine consumption (100%), g/t of ore	700
Chlorhydric acid consumption (37%), ml/t of ore	15
Activated charcoal consumption, g/t	17,0
Electricity consumption, kW/h per 1 t of ore+10% for the unrecorded equipment	37,3
Total consumption of fresh water, m^3/t of ore	0,4

11.2.2. Machine Flowsheet

Sample preparation and ore processing is performed in two sections of the gold processing plant with similar equipment. Regrinding, concentrates' densification and their hydrometallurgical processing are done in a separate section.

The machine flowsheet for the ore processing technology of the Krasnoye ore occurrence is given in Figure 11.3, specification of the equipment is given in Table 11.7.

Figure 11-1 Machine flowsheet

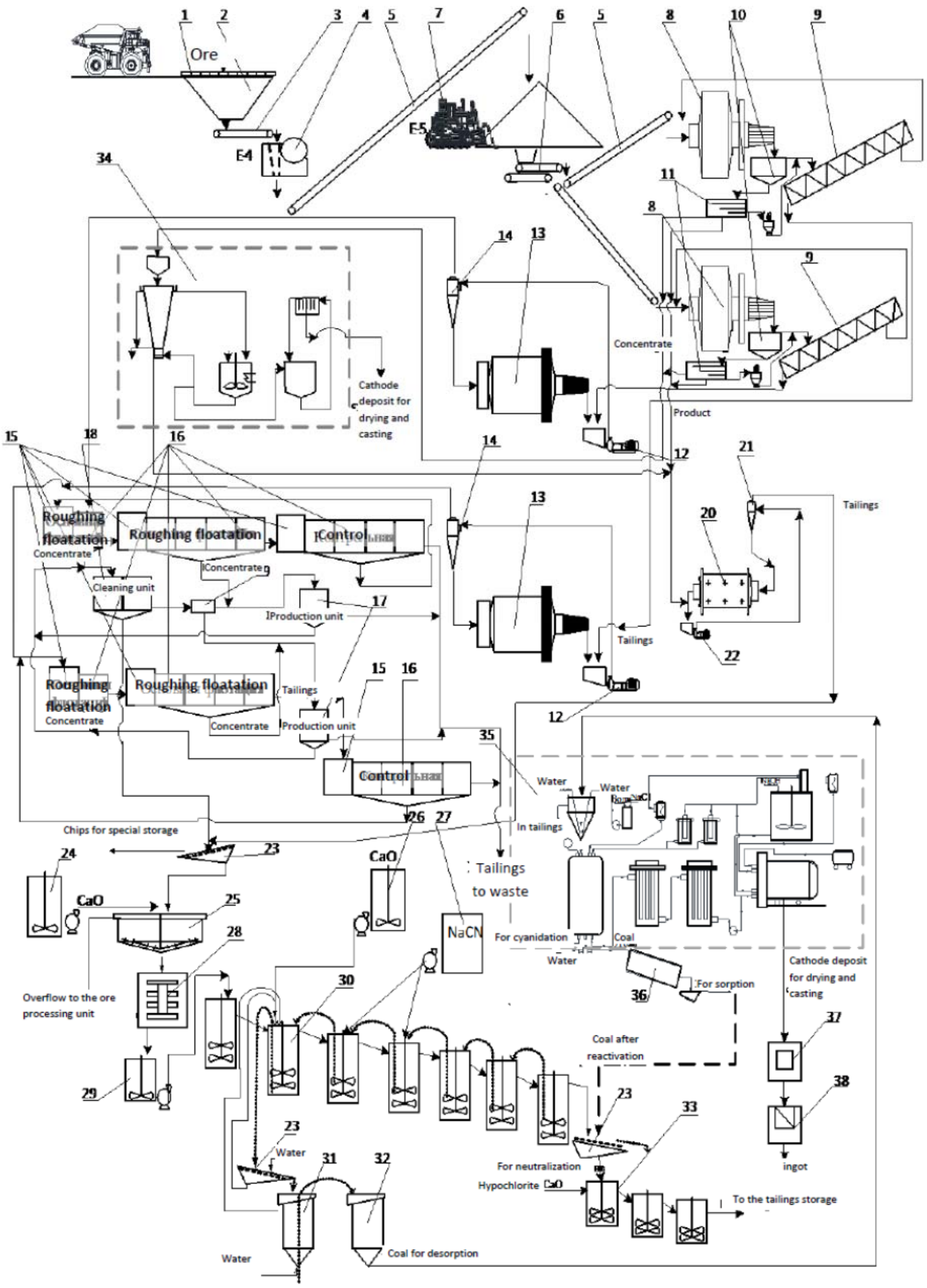


Table 11-1 Equipment specification

Position No.	Equipment Type	Amount, units	Electric Power, kW	Total, kW
1	Lattice	1	-	
2	Box 20 m3	1	-	
3	Apron feeder BW140-8 (China)	1	15	15
4	Jaw crusher ЩДС 9x12	1	130	130
5	Belt conveyor	3	5	15
6	Vibrating feeder ZSW-380X95II	2	15	30
7	Bulldozer or loader	1	-	-
8	Mill MZ 64×33	2	2,000	4,000
9	Spiral classifier 1КСН30М	2	30	60
10	Jolting machine МОД-3М	6	2x2.2	26.4
11	Concentration table СКО-7,5	6	1.1	6.6
12	Centrifugal pump	6	300	1800
13	Ball mill MQG 40x60	2	2,000	4,000
14	Hydrocyclone ГЦ 500	20	-	-
15	Conditioning tank КЧ-16	6	15	90
16	Floatation machine ФПМ-16	20	20	400
17	Floatation machine ФПМ-8,5	2	18.5	37
18	Floatation machine ФМР-10	2	1.1	2.2
19	Slurry flow divider	2	-	-
20	Mill MQY 1530	1	110	110
21	Hydrocyclone ГЦ 250	2	-	-
22	Centrifugal pump	2	50	100
23	Unbalanced throw screen ГИЛ 21	2	7.5	15
24	Conditioning tank for lime milk КЧР-25А	1	15	15
25	Thickener С-9	1	4	4
26	Conditioning tank for lime milk for output per shift КЧР-25А	1	15	15
27	Tank for a strong solution of cyanide КЧР-25А	1	15	15
28	IsaMill M1000	1	500	500
29	Conditioning tank КЧР-25А	1	15	15
30	Agitators of sorption and lime processing V=30 m3, overall dimensions	12	8	96
31	Column for washing out silts and chips	1	-	-
32	Accumulative column	1	-	-
33	Destruction agitators 15 m3	3	7	21
34	Installation of intensive cyanidation reactor ConSep ACACIA CS-500	1	40	40
35	Equipment set of the China National Gold Corporation (China)	1	200	200
36	Reactivation furnace ZSL-125 (China)	1	63	63
37	Resistance furnace СНЗ-8.15.5/12И4	1	75	75
38	Inductive furnace ZSL-125 (China) – productivity 125 kg/h	1	25	25

11.3. Amount and Professional Composition of Employees

Operation of the gold processing plant requires participation of engineers and other professionals. Real number of employees and their professional composition is given in Table 11.8.

Table 11-2 Real number of employees and professional composition

№	Job Positions and Professions	Amount of Employees per Shift		Total
		1 (day)	2 (night)	
	A. Workers			
1	Weighmaster	1	1	2
2	Bulldozer driver T-170M.01.E	1	1	2
3	Front loader operator	1	1	2
4	Metalworker for the containers maintenance and repair	1	1	2
5	Bunker servicing worker: operator of the rock breaker	1	-	1
6	Operator of the mills and classifying equipment	2	2	4
7	Crusher operator	2	-	2
8	Concentrator operator	2	2	4
8	Floatation operator	2	2	4
9	Thickeners' operator	1	1	2
11	Compressing units' operator	1	1	2
12	Sampler, Quality Control Department laboratory worker	2	2	4
13	Reagent operator	2	2	4
14	Repairman on duty	1	1	2
15	Electric repairman on duty	1	1	2
16	Repairman for repair and maintenance of ventilation and heating systems	1	1	2
17	Repairman for instrumentation and control (on duty)	1	1	2
18	Electric and gas welder	2	1	3
19	Repairman of the process equipment	2	-	2
20	Mechanic-electrician for the repair of electrical equipment	2	-	2
21	Reagents warehouse worker	2	-	2
22	Carpenter, tinsmith	1	-	1
	Total	**32**	**21**	**53**
	Assay laboratory			
23	Engineer, laboratory assistant	1	1	2
24	Operator of crushing and grinding equipment	1	1	2
25	Furnaceman	1	1	2
26	Chemist	1	1	2
27	Laboratory assistant of the sanitary laboratory	1	1	2
	Total	**5**	**5**	**10**
	Total amount of workers	**37**	**26**	**63**
	B. Engineers and Technicians			
28	Head of the Gold Processing Plant	1	-	1
29	Senior master	2	-	2
30	Technologist	1	-	1
31	Traffic controller	1	1	2
32	Head of the technical monitoring department	1	-	1
33	Head of the assay laboratory	1	-	1
34	Shifts masters	1	1	2
35	Mechanic of the Gold Processing Plant	1	-	1
36	Electrical mechanic the Gold Processing Plant	1	-	1
37	Engineer for instrumentation and control	1	-	1
38	Labour safety engineer	1	-	1

№	Job Positions and Professions	Amount of Employees per Shift		Total
		1 (day)	2 (night)	
39	Storage facilities supervisor	1	-	1
40	Cleaner of the working premises	1	-	1
41	Environmental engineer	1	-	1
42	Master of the tailings facilities	1	-	1
	Subtotal	**16**	**2**	**18**
	TOTAL	**53**	**28**	**81**

12. Environmental Protection

12.1. General Provisions

Assessment of the anticipated environmental consequences of the proposed activities for the components of the environment is performed on the basis of identification of potential sources of the anthropogenic impact and determination of the impacts of these sources, their qualitative and quantitative characteristics.

Characteristics of the Proposed Activities

The following key factors have been considered during the selection of the project fulfillment method:

- rational compact location of the facilities;
- safe operation of the plant;
- minimum impact on the environment;
- favourable engineering and geological conditions for the construction;
- economic efficiency.

The project of the mining and processing plant at the Krasny site includes:

- complex of the open mining facilities;
- complex of the ore processing facilities;
- complex of the facilities of production and social infrastructure.

Development of the Krasny site assumes construction of the pit and waste dumps, production sites with the disposal of the processing plant and production infrastructure facilities, rotation camp, tailings site, facilities of electric, heat and water supply, intersite engineering communications.

The staff is recruited organizationally in the area of the site location and in the central regions of Russia.

The plant facilities are located in accordance with the natural bedding of the deposit with consideration of the terrain, the hydrographic network, climate peculiarities of the area, technology interrelations between the facilities, spreading of dangerous areas of the designed production, as well as observance of sanitary protection zones.

The mode of the plant operation has been adopted as all-year-round, 350 days, with rotation (1 month working, 1 month on vacation), a continuous working week in two shifts of 12 hours.

The suspect environmental consequences of the designed activities for the components of the environment are assessed on the basis of establishing the sources of the anthropogenic impact and determination of the impact of these sources. Analysis of the decisions recommended by the project regarding the technology of ore mining and processing, organization of subsidiary production, and ensuring energy and material resources supply allows to identify the following main sources of the anthropogenic impact on the environment components:

- Workings excavation is the source of disturbing the geological unit, the ground surface, transformation of the landscape leading to contamination of underground waters and the ground level atmosphere.
- Organized and unorganized dust-gas emissions are the source of contamination of the ground level atmosphere with toxic components, and indirectly the ground surface.
- Loss of ore during transportation is the source of ground surface contamination.
- Work sites under construction and communications are the source of alienation, disturbance and pollution of lands, transformation of the natural landscape.
- Water consumption is the source of water resources depletion.
- Industrial and household wastewaters are the source of surface waters pollution, and, indirectly, of the bottom sediments and soils.
- Wastes storage is the source of contamination of underground and surface waters, and soil.
- Noise and vibration.

The set of the designed plant sources will influence all the components of the environment, which in this or that way will experience the following effects: mechanical, gas-aerosol and dust, hydrodynamic; hydrochemical, chemical, noise, land alienation and disturbance of the natural landscape.

Location of the designed sites at the deposit was made with consideration of the technological interrelation of the facilities, the terrain, wind patterns and emissions of industrial hazards, orientation by the cardinal points, enlargement and blocking of buildings, sanitary protection zones of industrial facilities, barren areas, sanitary and fire safety requirements.

There are no objects limiting the construction within the deposit (archaeological monuments, nature reserves, national parks, natural landmarks). The anthropogenic load in the area is low and is associated mainly with geological mapping and prospecting.

The main types of impacts on land and soil during the period of construction and operation of the plant's designed facilities are land alienation, mechanical effects on the soil, chemical impacts and disturbance of the natural landscape.

The chemical effect involves soil pollution with dust emissions falling from the air. The sources of the chemical effect on the soil are: the pit, the processing plant, the diesel electric power station, ore transportation roads.

The nature of the chemical effects of pollutants influences the complex of soil factors: particle size distribution of the soil, organic matter content, cation exchange capacity, presence of geochemical barriers and drainage.

Disturbance of the natural landscape. Territories occupied by the man-made terrain will experience the following conversion: surface sediments will be removed or relocated, meso- and micro terrain will be changed, and a large area of topsoil will be disturbed. As a result of industrial activities the aesthetics of the natural landscape is lost, which will be partially restored after a long period of time.

The impact on the subsoil. Mining operations will have a direct impact on the subsurface area of the deposit. The main type of the effect the site of the geological unit will experience (subsoil and surface), is ***the mechanical effect,*** which leads to disruption of the integrity of the geological unit's physical properties, due to the irreversible removal of ore and host rocks from the natural lands.

A permit for the deposit mining is the license that certifies that its holder has the right to use the subsoil area.

Geological and underground surveying services of the mine are assumed to ensure complete mining of the ore balance reserves from the subsoil. These services monitor operational exploration and control completeness of mining the ore balance reserves from the subsoil, record the state and movement of the reserves, losses and dilution of the ore.

12.1.1. Minimization of the Negative Effects

To reduce the negative impact on soils during the period of construction and operation of the designed sites provide the following environmental measures:

1. Construction works strictly in the contours of the land acquisition to prevent further disturbance of the vegetative soil cover.

2. In order to avoid getting fuel on the ground the project assumes the following organizational and engineering measures:

- The permanent facilities for fuel and lubricants storage;
- Refueling of machinery in-situ at the pit with the fueling tankers equipped with hoses and fueling nozzles, constant monitoring of tightness of the shutoff equipment at the fueling tankers, in case of failure it should be immediately removed;
- Organized collection of waste oil products;
- Repair and maintenance of machinery and equipment should be carried out in a specially equipped site of the mining machinery parking.

3. Strict observation of the fire safety rules during construction works and in life.

4. To protect the soil an impervious screen of loam is installed in the tailings storage.

5. Organization of temporary storage of the production and consumption wastes in the specially designated areas, in specially equipped areas and containers.

6. For the collection and removal of the surface drainage from the pit a diverting ditch is provided.

7. Traffic of transport and heavy equipment should appear only on haul roads and passages.

8. Selective removal of the vegetative soil cover, its storage to a temporary dump and use in full at revegetation of the disturbed land.

9. In accordance with GOST 17.4.3.02-85 "Environmental Protection. Soils. Requirements to the Protection of the Topsoil During Excavations" removal and preservation of the vegetative soil cover is provided at special sites for the subsequent use during land reclamation.

Land reclamation is assumed in accordance with GOST 17.5.1.02-85 "Environmental Protection. Lands. Classification of Disturbed Lands for Reclamation" and GOST 17.5.3.04-83 "Lands. General Requirements to Land Reclamation".

Since the entire adjacent territory is intended solely for forestry purposes, the focus of reclamation of the lands disturbed by the industrial use is the sanitary revegetation with the prospect growing of the normal forest.

The technical stage of land reclamation is performed in all areas of disturbed land during and after completion of the plant operation. During the technical stage of land reclamation the following activities should be performed at the sites:

Reclamation of the pit. Technical activities undertaken during reclamation of the pit assume the output of the mining equipment from the pit, formation of optimal parameters of dumps or fields of the mined spaces. The diverting ditch is filled up. During reclamation activities the following is provided:

- protection embankment in the entry trench of the pit;
- fencing of the pit's perimeter with a barbed wire in 3 threads.

The major part of the pit will be naturally flooded after mining is completed.

Reclamation of the waste dump. Mining reclamation of the waste dump means sheltering only the horizontal site of the dump (S = 10.2 ha) with the fertile layer, where the main part of the fine and dust fraction of the rock is accumulated. No topsoil is put on the slopes of the dump, because its slope is 30°. A strong washout of the applied soil appears at the surface with such a slope. Slopes' surface is exposed for self-vegetation.

Reclamation of tailings storage. After water drainage from the pond the distributing slurry line and the circulation water supply pipe are dismantled. The bund slope is broken. The developed soil is ground leveled to the bottom of the tailings storage with the slope towards the tail bay. To avoid dusting the planned surface is covered with the layer of the Quaternary sediments, and the topsoil from the temporary dump formed during the construction of the tailings storage.

Reclamation of the site of the processing plant, the rotation camp, the fuel and lubricants storage. Buildings and structures are dismantled, the surface is planned for the drainage of the rainfall. The topsoil is put to the planned surface.

12.2. The Effect on the Atmosphere

The main type of the effect of the facilities on the condition of the atmosphere is contamination of the air with the emission of hazardous matters from various sources of the mine, the rotation camp and the industrial site of the processing plant.

The main sources of the mining site pollution are as follows.

- waste dumps;
- bulldozers;
- vehicles;
- refueling point;
- drilling equipment;
- loading/off-loading, drilling and explosive works.

The main sources of pollution in the rotation camp are as follows:

- fuel tanks.

The main sources of the processing plant site pollution are as follows:

- crushing section;
- Furnaces for concentrates drying;
- fuel tanks;
- repair and maintenance boxes for large loading vehicles with wood processing site;
- garage;
- repair and parking box.

Grinding and processing of the ore is performed in the presence of water, then no emission of hazardous substances appears.

Contamination of the atmosphere appears as the result of the emission of:

- exhaust gases from the road transport, road construction machinery, drying furnaces;
- vapours from the fuel storage tanks;
- dust from the surface of the dumps, from loading and off-loading operations;
- dust and gases from explosive works;
- combustion products in the diesel electric station;
- dust from grinding equipment of the processing plant;

- dust from the metal and woodworking machines;
- harmful welding.

As the result of the above mentioned actions the contamination of the air increases in the area of the plant location.

The main feature of mining companies, in terms of the impact on the atmosphere, is the large number of unorganized and non-stationary sources with very uneven emission of polluting substances during the shift. Explosive works and emissions during unloading of loose materials have a volley character. The impact of such volley emissions on the atmosphere is insignificant by time: the duration does not exceed 10 minutes.

Design experience of mining facilities in different climatic regions of Russia (Yakutia, Chukotka, Kamchatka, the Irkutsk region, the Khabarovsk Territory, the Altai Territory, the Republic of Komi, etc.) allow to make a reasonable conclusion that for mining facilities the concentration of pollutants in the surface layer of the atmosphere at the boundary of the sanitary protection zone almost never exceeds health standards.

At the stage of the project documentation development for the construction of capital construction facilities a detailed calculation of the pollutants' dispersion in the surface layer of the atmosphere will be carried out.

The noise levels will not exceed the limit values by noise indicators at the border of the regulatory sanitary protection zone.

It is offered to set the area of the sanitary protection zone in accordance with the sanitary regulations and standards SanPiN 2.2.1/2.1.1.1200-03, i.e. 1,000 m, resulting from the total expected harmful impacts on the environment. The emissions will be considered as marginal.

12.2.1. Minimization of the Negative Effects

To reduce the emission of pollutants and to improve their dispersion a set of technological and special air protection measures is provided:

- the use of the equipment with built-in dust collection systems for drilling;
- the use of motor vehicles and self-propelled machinery equipped with catalysts for the exhaust gas cleaning;
- the routine maintenance and repair of motor vehicles and mining equipment;
- irrigation of the sites of loading and off-loading activities and ore storage in summer;
- systematic watering of roads and access routes;

- self-monitoring of compliance with the standards of the maximum permissible emissions.

12.3. The Effect on the Surface and Underground Waters

The objects of the direct and indirect effects of the man-made facilities during the development of the deposit are surface and underground waters of the area.

The main types of impact on the water basin are as follows:

- hydrodynamic disturbances;
- water basin pollution (hydrochemical impact);
- damage to fisheries.

Disturbances that occur as a result of hydrodynamic effects are connected with changes in the regime and dynamics of surface and underground waters. Hydrodynamic disturbances may lead to changes in hydrological parameters of waterways, depletion of aquifers, reduction (formation of the depression sinkhole) or, conversely, increase (repression cone) of the underground water level.

To reduce the impact on the underground waters the project includes measures on drainage of the surface runoff from the sites of hydrotechnical structures and reduction of the filtering of their reservoirs, thus eliminating floods and bogging of the area below the dams of these facilities.

Thus, the hydrodynamic impact on the underground waters of the deposit area during the operation of the proposed facilities will take place. It is necessary to monitor the development of the depression sinkhole, the level and the quality of the underground waters.

Household and drinking water supply may be provided by local sources after prior agreement with the local authority for sanitary and epidemiological supervision (Gossanepidnadzor).

Waters of the rivers located in the vicinity of the designed sites do not cross the territory of any industry, residential villages, so it is assumed that the quality of the water should meet sanitary and epidemiological requirements.

To reduce the consumption of water for production at the gold processing plant a water recycling system with water return to the process after cleaning in the pond will be organized. Melted snow and rain water that fell on the industrial site, are fully involved in the technological process at the closed cycle. Moreover, it is provided that settled pit and dump waters will be used for eliminating dusting of the roads and filling of the fire tanks.

During construction of the pit it is necessary to make a diversion of beds of the creeks Krasny and Teply. The length of the disturbed areas is around 1 km, hydrodynamic disturbances will have a local character.

In general, the hydrodynamic effects on the surface waters of the deposit area are expected to fit the permissible limits and will not lead to the loss of waterways, irreparable harm, or disturbance of the surface waters regime.

The hydrochemical impact involves contamination of the surface and underground waters with chemical components and is associated with waste products of the plant entering the aquatic environment. The main sources of pollution are usually discharges of household and industrial waters, filtering of the pollutants out of the storage facilities, washing out of the pollutants deposited at the territory of the plant with dust emissions with precipitation.

For the collection and drainage of wastewaters separate sewerage is organized at the production site and the rotation camp.

All household wastewaters are treated before discharge at the compact treatment plant BIODISK-350 located near the site of the camp. These treatment facilities are manufactured by the Research and Production Enterprise “Ekotekhnika” (Moscow). The treatment technology applied allows to reduce the content of pollutants to the regulatory requirements for the discharge into creeks and rivers.

The pit and dump drainage channels are contaminated mainly with suspended solids and oil products. Wastewaters’ treatment with natural sedimentation is done in the ponds. The slurry is deposited at the bottom, the oil film is captured by oil collecting booms. To reduce the filtering capacity of the ponds they are equipped with screens made of loam. Treated wastewater is partially used for production needs of the plant, the excess is discharged into the creeks.

The chemical composition of wastewaters must be regularly monitored. Upon the results of the monitoring data analysis, the volume and mode of discharge into the water body can be adjusted, measures on reduction of the hydro-chemical impact are developed.

Overall, as a result of the wastewaters discharge into the surface water bodies there may be some deterioration in the quality of the river below the wastewater inflow, but the concentration of pollutants is expected within the maximum permissible amounts, or close to the background.

Pollution of the surface waterways with dust emissions and oil products can also cause the quality loss of the surface and underground waters.

The dust part of the emissions in the form of solid particles of suspended solids, heavy metals compounds will be deposited on the ground by gravity, cooling and catching with precipitation. Concentrating on the ground and accumulating in the topsoil, a drip-dust part of emissions may form technogenic geochemical anomalies around the production sites. In the warmer months these materials will be washed off from the surface by melted snow and rain waters, entering the underground waters and, partially, waterways. Pollutants' concentrations in the melted snow and rainwater may exceed the maximum permissible concentrations, usually on metals in the ore.

Thus, as a result of surface washing off during the flood period and air transfer of a small fraction of rocks some variation of the microelement composition of natural waters may occur. Moreover, they may contain oil products due to the exhaust gases from the working machinery.

Reducing the extent of the impact of this type is usually achieved by implementation of gas-cleaning equipment, collection and treatment of rainwater from the most contaminated areas, performance of activities for dust suppression.

Thus, the operation of the designed plant under the condition that the environmental measures are taken, will not cause significant changes in the regime and quality of the natural waters; the level of the impact on the aquatic environment can be considered permissible.

12.3.1. Minimization of the Negative Effects

The project assumes the following activities providing reduction of the hazardous impacts of the designed mining activities on the water basin and rational use of water resources at the plant:

1. Activities within the area or line of drainage. Location of the designed facilities sites beyond the borders of the water protection areas of the water bodies.

2. Introduction of the recirculation system of water supply and repeated use of wastewaters.

3. Absence of oil and lubricants storage and refueling of the machinery within the boundaries of the floodplain. Organization of regular cleaning of the territory.

4. Antifiltration activities during construction of hydrotechnical structures.

5. To prevent pollution of the surface runoff the diverting ditch is designed for the pit site.

6. During the construction of the water intake the areas of the sanitary protection of the household, drinking and industrial water supply source should be formed and protected.

7. Application of the efficient methods of wastewaters' treatment providing reduction of the pollutants' content to the level corresponding to the requirements for the water basin protection from pollution.

8. Activities on prevention of the territory cluttering with production wastes, providing their organized storage and recycling during the period of the plant operation.

9. Organization of dust catching at the mining site and roads in the dry warm period of the year to prevent dust contamination of the atmosphere and settlement of the dust on the surface of water sources.

10. Activities on exclusion of oil and lubricant entering into the soil and water bodies, which provide:

- organization of stationary facilities for oil and lubricants storage;
- organized collection and recycling of waste oil products;
- systematic monitoring of the equipment, tanks, fueling systems of the machinery and mechanisms.

11. Measures to prevent accidental discharge of wastewaters, including:

- organization of the accumulating tanks for wastewaters;
- elevation of the apex of the hydrotechnical structures' walling dams above the maximum water level not less than 1.0 m;
- organization of regular monitoring of the structures' conditions, correct operation of the equipment;
- monitoring of the compliance of the technological processes parameters.

12. Organization of the production environmental monitoring of the condition and quality composition of the natural waters, quality of the wastewaters entering the environment.

Fulfillment of the protective and restorative measures provided by the project will allow to reduce the negative impacts on the water basin and will ensure its protection from contamination and depletion.

Permits for the use of water resources and wastewaters' discharge are received by the subsoil user in the established order after detailed development of the project documentation for the construction of the capital construction facilities.

12.4. The Impact of the Production and Consumption Wastes on the Environment

Wastes formed at the plant during its operation may be potentially one of the main negative impacts on all components of the environment, first of all, on the soil reserves and the topsoil.

Production and consumption wastes do not only require large areas for storage, but contaminate the atmosphere, the soil, the surface and underground waters with hazardous substances, dust and gas emissions.

During the operation period the following types of wastes appear at the plant.

12.4.1. Production Wastes

1. *Wastes, which appear during mineral resources mining (waste rocks),* are formed during mining.

According to the paragraph 13 of the "Criteria for Classifying Wastes By Their Hazardous Impacts on the Environment" approved by the Decree of the Ministry of Natural Resources of the Russian Federation dated June 15, 2001, waste rocks have been classified as almost nonhazardous wastes.

Waste rocks are stockpiled. Upon completion of mining stockpiles should be recultivated in accordance with the engineering conditions for recultivation.

2. Processing wastes are floatation tailings, which are stored in the tailings storage. Floatation tailings are mineral products. Upon the results of investigation of the processing tailings of the similar plants, they refer to the almost nonhazardous wastes of the 5th hazard class.

During the first year of operation the hazard class of the mining and processing plant wastes should be confirmed experimentally (using the method of biotesting). Upon completion of the deposit mining the tailings storage should be recultivated.

12.4.2. Subsidiary Production Wastes

This type of wastes includes wastes formed at the subsidiary production facilities, during operation and repair of equipment and machinery used at the plant, wastes resulting from wastewaters treatment. Here we consider wastes formed in the process of the employees' daily life and work, and functioning of the social infrastructure facilities.

1. S*ynthetic and mineral waste oils* are formed as a result of oils replacement in the engines of vehicles and mining machinery, equipment of the processing plant.

The following types of waste oils are formed at the plant: engine, transmission, hydraulic.

Waste oils are collected in sealed metal barrels, which are placed in the open air storage of oils in the fuel storage containers. The site is equipped with the concrete cover. Upon accumulation, waste oils will be passed under a contract to a special company that has a license for wastes' management: for neutralization.

A part of waste oils can be applied to lubricate the friction surfaces of lifting equipment of the mining machinery that do not require pure oils.

2. *Sludge from pipelines and oil tanks cleaning* is produced during the operation of the fuel storage equipment. The sludge is collected in sealed metal barrels with lids for temporary storage, and then transported for recycling along with waste oils.

3. *Cleaning material contaminated with oils (oil content less than 15%)* is produced due to the use of rags in the maintenance of the machinery and the equipment which require lubrication. Oiled rags are collected in closed metal containers placed in the repair sites of the processing plant, the repair shop, the garage, and upon accumulation of a considerable amount the materials are periodically removed for recycling together with waste oils.

4. *Waste lead batteries, not disassembled, electrolyte not drained,* are formed in case of failure and replacement of accumulative batteries of the vehicles and mining machinery. They are disposed in a special container for temporary storage in the garage, with subsequent transportation to a licensed company for neutralization.

5. *Waste or rejected mercury lamps and mercury-containing fluorescent tubes* are formed as a result of failure of the lighting equipment installed in the buildings and the sites of the plant. For temporary storage waste lamps are placed in the manufacturer's package to be stored in closed warehouses. They are periodically transported to the nearest regional center to be given to a specialized company for neutralization.

6. *Unsorted ferrous scrap* includes metal scrap of the processing plant formed in the process of crushing equipment operation in grinding balls, liners, etc., as well as during the process of the replacement of parts and components during the repair of equipment and vehicles. Ferrous scrap is stored for temporary storage at a special open site with waterproof coating. It is implemented as recycled materials by specialized recycling companies.

7. *Scrap and waste containing non-ferrous metals* are formed during replacement of parts and components during the repair of equipment and vehicles. Metal wastes are implemented as recycled materials by specialized enterprises "Vtormetresursy". The wastes are temporarily stored at a specially designated site.

8. *Uncontaminated ferrous metal chips* are formed during the processing of parts on the machines in the repair shop. Chips are deposed together with ferrous metals wastes. It is accumulated in metal containers placed at the work site.

9. *Residues and scoria of steel welding electrodes* are formed as a result of repair welding. Temporary scoria accumulation is done in the place of formation in metal containers. Recycling is provided along with ferrous metal wastes as secondary raw materials.

10. *Uncontaminated containers and packaging from steel, which have lost consumer properties* are formed during the use of reagents at the processing plant. Steel packaging wastes are stored at an open site with waterproof coating for temporary storage of the mining and processing complex metal wastes. Further a common set of metal wastes is formed for transportation.

11. *Waste brake shoes* are formed during the repair of machinery. Collection and recycling of brake shoes is implemented together with ferrous scrap.

12. *Wastes of solid industrial materials contaminated with oil and mineral fat products (used oil filters*) are produced by vehicles and equipment. Filters are collected into metal tanks installed at the place of repair. Periodically, upon accumulation, used filters are given for recycling to a licensed organization together with other wastes contaminated with oil products.

13. *Waste abrasive discs, scrap of waste abrasive discs* are formed during processing of the parts on machine tools at the repair shop. They are disposed at the solid wastes landfill of the mining and processing complex. They are accumulated in metal containers placed at the work site.

14. *Waste bark* results from the treatment of sawn wood in the workshops for wood processing. It is recycled by burial at the solid wastes landfill. It is accumulated in metal containers placed at the work site.

15. *Natural pure wood sawdust* is formed during processing of timber in the workshops for wood processing. The accumulation and disposal of wastes is done together with the waste bark.

16. *Wastes of polypropylene in the form of a film* are formed during the use of reagents at the processing plant and unloading of explosives in the explosive storage facilities. Upon

accumulation the containers with reagents are transported to the solid wastes landfill of the mining and processing complex. Containers with explosives are burned at the site for explosive destruction.

17. *Polyethylene wastes in the form of a film* are produced and recycled similar to the polypropylene wastes.

18. *Uncontaminated wastes of the packaging cardboard* are formed during unloading of the explosives at the explosives storage site. They are utilized by burning at the site for explosives destruction.

19. *Waste pneumatic tires* result from the wear of the vehicles' tires. Waste tires are placed on the solid wastes landfill. Worn tires can also be used to support the needs of the plant (for example, to strengthen the slope areas, roads).

20. *Uncontaminated rubber products, which have lost consumer properties* (a used conveyor belt) result from stretching and change of the belt on the belt conveyors of the processing plant. Wastes are collected in metal containers, storage is provided at the solid wastes landfills.

21. *Trimmed rubber* is produced during curing works and is disposed at the solid wastes landfill. Wastes are temporarily stored in the container at the site of the repair shop.

22. *Wastes (sediments) of mechanical and biological treatment of wastewaters* appear during the treatment process of melted snow and rainwater of production sites and sewage. Sediments at the sites of the deposit are taken for burial to the solid wastes landfill.

23. *Wastes of solid industrial materials contaminated with oil and mineral fat products (oil sorbing booms).* Capturing of oil from wastewaters is done with the use of polypropylene oil sorbing booms, which the cleaning facilities are equipped with. Oil sorbing booms are replaced 1-2 times a year. Saturated oil booms are stacked in sealed metal barrels, and upon accumulation are transported to a licensed organization for recycling together with other wastes contaminated with oil products.

24. *Unsorted wastes from households (excluding large waste)* are formed during living of the staff in the rotation camps of the mining and processing complex. Standard rubbish bins are placed for collection of solid household wastes. The solid wastes are taken to the solid wastes landfills according to the schedule.

To reduce the impact of the wastes on the environment the following measures should be taken:

- Special facilities are designed for storage and disposal of production and consumption wastes: waste rock dumps, the tailings storage of the processing plant, the landfill of solid wastes and industrial wastes.
- Centralized collection and sending for further use or neutralization in accordance with the contracts signed with specialized enterprises licensed for the treatment of hazardous wastes. The range of specialized companies and contracting is planned for the next stage of design, after a detailed calculation of production and consumption wastes.
- Organization of temporary accumulation and storage of production and consumption wastes is provided in designated areas, specially equipped sites, in containers that do not allow their pouring out or winding out on the surface. Sites for temporary storage of wastes are equipped with waterproof coatings.
- Transportation of wastes for permanent storage or disposal must be carried out upon accumulation with the intervals eliminating the formation of unorganized landfills. The maximum amount of temporary storage of wastes on the territory of the designed production is determined by the availability of free space for temporary storage under the conditions of storage and conditions of free access for loading and removal to disposal facilities.
- Provision of technological control of the production processes, compliance with the rules of operation and safety will help to prevent accidents and, as a result, contamination of soil with wastes of the plant.
- Safety measures for handling of hazardous wastes. To comply with the safety measures for handling of hazardous wastes, instructions for collection, storage and transportation of wastes of different hazard classes in containers are developed. The staff working at the plant is informed about these instructions, observance of the rules is monitored.
- Upon completion of production the recultivation of the wastes storage and burial structures is provided.

In compliance with the design solutions for collection, storage, temporary storage and disposal of wastes, no cluttering of lands, contamination of the air, natural waters and soil will happen. Land contamination can be of a pointed character limited by the wastes storage sites.

12.5. Production Environmental Monitoring

Production environmental monitoring is aimed at information provision for the management of the environmental protection and safety. Production environmental monitoring should be fulfilled during the whole operation period of the plant. Its main objectives are to monitor the compliance with the standards of the environment quality and the requirements of the environment protection legislation, measures on using and reproduction of natural resources.

Production environmental monitoring is fulfilled within the borders of the mining license and the land license of the plant. The programme of environmental and production monitoring provides continuous monitoring of the intensity and character of the man-made impacts on the environment to take operative and/or long-term measures on prevention and/or elimination of the negative impacts.

Monitoring and control activities include assessment of the parameters of influencing sources and results of the plant's impacts on the objects of the natural environment. Herewith, not only the plant as whole should be controlled, but also its individual facilities, sources of emissions, pollutants discharge and formation of production and consumption wastes.

Production environmental monitoring of the environment condition should be ensured by the service of the production environmental monitoring of the plant in accordance with the existing environmental legislation.

The environmental service records and controls the use of water resources, emissions of hazardous substances into the atmosphere and production wastes by taking the following measures:

1. Organization of the primary monitoring of pollutants emission into the atmosphere:
   - monitoring the fixed sources of pollution and their characteristics;
   - monitoring the work of gas treatment and dust collection units;
   - monitoring the vehicles use and fuel consumption.
2. Organization of the primary monitoring of water supply and drainage:
   - Recording of the amount of the consumed water for drinking, household needs and production.
   - Recording of the volume of the household and production drainage.
3. Organization of the documentation recording regarding the amount and types of wastes at the disposal.

4. Annual reporting to the authorities on environmental protection using the forms 2TP-air, 2TP-water, 2TP-wastes.

To perform laboratory monitoring specialized certified laboratories are engaged on a contract basis. Laboratory monitoring is conducted in accordance with the charts of production environmental monitoring consistent with the requirements of the environmental authorities, with implementation of the approved procedures. For special investigations as a part of the production control programme specialized research organizations are engaged.

With the stabilization of technological processes the environmental monitoring programme is adjusted with the focus on the parameters, which have or may have a negative impact on natural components.

The programme of the production control is focused on the normal mode of the plant operation. In emergency situations, the programme of control is adjusted in the direction of increasing the frequency of control (hourly/daily/weekly sampling) at all stages of development and elimination of the emergency situation.

Wastewaters discharge. Production control of wastewaters delivered for discharge into the natural water bodies is made with the frequency of once per month with determination of the consumption characteristics, properties and indicators of the chemical composition at all outflows of regulatory treated wastewaters into water bodies.

Table 12-1 List of Controlled Indicators in the Samples of Wastewaters

Name	List of indicators
Amount	Expenditure of wastewaters
Physical properties	Temperature, colour, transparency, smell
Hydrochemical indicators	pH, biochemical oxygen consumption, dissolved oxygen, suspended solids, oil products, surfactants, chlorides, sulfates, ammonium ions, nitrates, nitrites, phosphates, total iron, copper, zinc, lead, manganese, arsenic, aluminum.

13. Financial and Economic Analysis

13.1. General Provisions

The current technical and economic report contains the estimate of the Krasnoye site with annual ore production and processing volumes 1,500 Kt.

The key solutions adopted in this report include:

The use of open-cast mining method with drilling and blasting, and a transport method of overburden transportation to dumps.

The annual productivity is 1,500 Kt of ore.

All ore mined will be processed at the gold processing plant with the gravitation-flotation and cyanidation circuits, with production of the gold concentrate. The transient gold recovery is assumed equal to 85.0%.

The enterprise will adopt a rotational work method, with formation of the required scope of production infrastructure and social and household facilities at the mine site.

13.2. CAPEX

The direct CAPEX for the development of the Krasnoye deposit reserves, assumed in economic calcuations, represent the sum of the following items:

- construction costs including costs for purchasing mining equipment, construction of the gold-processing plant and production and social infrastructure facilities;
- design costs.

Capital expendistures will be defined on the basis of the production nature and scopes determining the demand in process equipment, and expenditures required for construction of general utilities.

The cost of mining equipment was assumed on the basis of the data of equipment manufacturers.

The calculated cost of mining equipment is given in the table below.

Specific capital expenditures for equipment of the gold processing plant are assumed in the report equal to USD 25.0 per tonne, on the basis of the average industry data.

CAPEX for tailings facilities are assumed in accordance with the data of similar enterprises, on the basis of USD 0.8 per tonne of tails disposed. Construction will be executed in two stages. The expenditures for construction of the first stage of the tailings pond are included into the initial CAPEX estmate, and the costs for the second stage of the tailings pond are a part of the sustaining capital.

Specific CAPEX for construction of production and social infrastructure facilities are assumed in this report on the basis of the average industry data equal to USD 20 per tonne of the ore processed.

Design, engineering and geological research costs are defined equal to 2% from the total cost of construction.

Based on the average industry data the report assumes specific costs for mining-capital work execution equal to USD 2.0 per m^3 of the rock mass. The mining capital work scope is assumed in the report based on 50% from the designed average annual scope of work execution in terms of the rock mass.

Moreover, the CAPEX estimate includes indirect costs[1] - 15% from direct construction costs, and contingencies representing the backup sum associated with probable increase in the work cost which is hard to predict at the moment of estimate due to insufficient or missing information, equal to 3% from the sum of direct and indirect expenditures.

The CAPEX calculation for the development of the Krasnoye deposit reserves is given in the table below.

Moreover, the combined CAPEX includes floating capital provided for formation of the required material resource backup, both in the sphere of production and the sphere of circulation (final product at the enterprise warehouse; product shipped but not paid; debit float, advanced payments to suppliers).

In order to ensure stable work of the entreprise, this feasiiity study provides for the foating capital equal to 30% from the cost of ore production, processing and general running costs.

[1] Expenditures for preparation of the construction site, construction of temporary buildings and facilities; expenditures for construction equipment and means for construction work provision, logistics and construction control etc.

Tabe 13-1 Calculation of the cost of mining equipment purchased

No	Description	Cost of Unit, th. RUR	No of Units, pcs.	Total, th. RUR
Main mining equipment				
1	Excavators EKG-10	120,000	3	360,000
2	Excavators EKG -5	40,000	1	40,000
3	Truck BelAZ-7555E, capacity 55 t	12,000	24	288,000
4	Truck BelAZ -7540A, capacity 30 t	4,500	8	36,000
5	Drill SBSh-250-MIA-32KP ø250 mm	30,000	4	120,000
6	Dozer T.35-01	18,000	3	54,000
7	Dozer T.25-01	8,000	4	32,000
Auxiliary mining equipment				
1	Charging machine MZ-ZB on Ural-4320 chassis, Machine-building plant Zvezda, Karpinsk	3,500	1	3,500
2	Scaing machine, autohydrauic hoist BC-22,05 on Ural-4320-40 chassis	3,500	1	3,500
3	All-season combined road machine VMKD on Ural- 4320 chassis	3,500	1	3,500
4	Repair machine PARM-4784 on Ural- 4320-1112-41 chassis (LLC. Avtomaster)	3,500	1	3,500
5	Grader DZ-98B (CJSC Chelyabinsk Construction Road Machines, Chelyabinsk)	5,200	1	5,200
6	Mobile floodlight tower EFA 650 with built-in diesel power plant, floodlight power 4 x 1500 W (Endress)	375	1	375
7	Passenger car, Ural Vakhta 3255-0010-01	2,300	1	2,300
8	Cranmobile KC-557722, capacity 25 t on Ural-4320 chassis, OJSC Avtokran, Ivanovo	3,500	1	3,500
9	Fuel Charger ATZ-12 on Ural-4320-40 chassis. OJSC GraAz», Grabovo	3,500	1	3,500
Total				**958,875**

Table 13-2 Determination of the specific cost of equipment for the gold processing plant on the basis of data of similar projects

Deposit	Location	Capacity of the Enterprise, Kt	Cost of Equipment of the Processing Plant, th. USD	Specific Cost of Equipment of the Processing Plant, USD/t	Conversion Factor to Current Prices	Specific Cost of Equipment for 2013, USD/t	Processing Circuit	Design Organization and Year of Design
Pokrovskoye	Amur Region	1000	7 933.00	7.93	2.96	23.44	Direct sorption cyanidation	Actual data of OJSC Pokrovsky Mine, 2002
Verninskoye	Irkutsk Region	800	8 071.40	10.09	2.64	26.62	Gravitation-cyanidation of concentrates (combined semidry stockpiling of processing and hyodrmetallurgy tails)	OJSC Irgiredmet, 2003
Kupol	Chukotka Autonomous Area	400	11 300.00	28.25	2.11	59.69	Gravitation-cyanidation	CJSC NBLzoloto, 2005
Kun-Manye	Khabarovsk Territory	1000	11 528.65	11.53	1.71	19.68	Gravitation-flotation	Feasibility study of mining parameters, 2007. Sibtsvetmet niiproyekt
Maly Taryn and Drazhny	Republic of Sakha (Yaponia)	1000	9 958.97	10.0	1.19	11.83	Gravitation-cyanidation of gravitation tails and gravitation concentrates	Feasibility study of temporary mining parameters, 2011. FGUP Yakutskgeologiya
Specific cost of equipment of the gold-processing plant assumed in calculations, USD/t							**25.0**	

Table 13-3 Calculation of CAPEX for the development of Krasnoye deposit reserves

Items, facilities, costs	Value, th. USD
Design, engineering and geological research	2,562
Construction	128,116
Mining capital work	6,723
Mining transport equipment	31,963
Processing plant	48,750
Tailings facilities	10,680
Other social and production infrastructure facilities	30,000
Total direct expenditures	**130,678**
Indirect expenditures (15% from construction costs)	14,423
Total direct and indirect expenditures	**145,101**
Contingencies (3%)	4,353
Total, including VAT	**149,454**
including VAT	22,798

13.3. Production Costs

The combined OPEX of the enterprise represents the sum of the following items:

- ore production costs;
- ore processing costs;
- general running costs.

The prime cost of production of 1 cub.m of the rock mass is assumed on the basis of the average branch parameters for medium-capacity pits equal to USD 3.0 per cub.m.

The prime cost of procesing of the ore is assumed on the basis of the average branch parameters equal to USD 20.0 per tonne.

The general running costs are assumed equal to 15% from the sum of ore production and processing expenditures.

The value of annual depreciation is defined on the basis of combined CAPEX excluding VAT, with the average branch norm of depreciation 8.0%.

The depreciation value calculation is given in the Income and Expenditure Summary for the entertrpise.

Credit interest costs are also included into the prime cost.

The effective credit interests are assumed on the basis of the subsurface user's data; the rate of interest for the investment credit is assumed equal to 8% per annum in USD.

Tax deductions include royalty making 6.0% for the gold deposit from the cost of the resource mined, the property tax equal to 2.2% from the cost of remaining fixed assets of the enterprise, and environmental payments assumed on the basis of average branch parameters equal to 0.5% from the cost of the marketable product of the enterprise.

The profit tax is 20% from the taxable positive profit of the company and is calculated in the Income and Expenditure Summary for the project.

13.4. Income and Expenditure Summary

Rounded average (for the recent year) cost of gold at LBMA equal to USD 1,670 per troy ounce serves as the basis for determination of the estimated price for 1 g of chemically clean gold.

The dollar rate is assumed equal to RUR 30.0 per USD.

Therefore, the price of 1 g of chemically clean gold is USD 53.69 per gram.

The calendar schedule of ore mining given in the table below serves as the basis for calculation of the annual average volume of marketable product.

Profit from marketable product sales is the difference between the cost of marketable gold extracted during processing and the cost for its refining (1.5% from the marketable product cost).

The income and expenditure summary incudes the following indicators:

- profit from marketable product sales;
- OPEX of the enterprise;
- depreciation of fixed assets of the enterprise;
- royalty;
- property tax;
- credit interest;
- taxable base representing the difference between the income of the enterprise and OPEX, depreciation, taxes and credit interest;
- profit tax calculated on the basis of the taxable base with the rate established by the RF legislation;
- net profit representing the taxable base minus profit tax.

The use of 30% of loan capital and 70% of own capital is assumed in the financial model of evauation of the investment attractiveness for the Krasnoye deposit reserves development project for provision of the probabiity of finanical project implementation.

The bank credit interest is assumed equal to 8%. The project financing chart is given in Table 13.6.

The Income and Expenditure Summary for the enterprise for the reserves mining period is given in Table 13.6.

The calcuation of the net present vaue is given in Table 13.8.

The main technical and economic parameters of the project are given in Table 13.9.

Table 13-4 Determination of the prime cost of ore processing on the basis of the data of similar projects

Deposit	Location	Production Capacity, Kt	Prime Cost of Processing of 1 t of Ore at the Date of Report Preparation, USD/t	Processing Circuit	Comments	Conversion Factor to Current Prices	Prime Cost of Processing of 1 t of Ore for 2013, USD/t
Blagodatnoye	Krasnoyarsk Territory	5000	8.77	Gravitation-cyanidation	Design, 2005, OJSC Krasnoyarskgeologiya	2.03	17.8
Vysochayshi	Irkutsk Region	1300	13.12	Gravitation-cyanidation of concentrates	Plan, 2008, data of OJSC Vysochayshi	1.46	19.2
Pokrovskoye	Amur Region	1000	8.14	Direct cyanidation	Plan, 2003, data of OJSC Pokrovsky Mine	2.53	20.6
Titimukhta	Krasnoyarsk Territory	1700	10.05	Direct cyanidation	Feasibility study of permanent exploration parameters, OJSC Krasnoyarskgeologiya, 2007	1.64	16.5
Berezitovoye	Amur Region	1500	7.87	Gravitation-cyanidation of gravitation tails and gravitation concentrates	Feasibiity study of construction of the Berezitovy Mine, OJSC Sibgiprozoloto, 2003	2.53	19.9
Zegen-gol	Repubic of Buryatiya	800	13.40	Gravitation-flotation	Preliminary feasibiity study, 2008, CJSC NBLzoloto	1.46	19.6
Kyuchevskoye	Chita Region	3000	12.70	Gravitation-flotation-cyanidation of concentrates	OMS, 2008	1.46	18.6
Prime cost of ore processing assumed in calculations, USD/t							20.0

Table 13-5. Schedule of final product issue

Description	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	Total
Stripping, th.m^3	3,362	3,362	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	6,923	4,788	122,281
Open-cast mining, Kt		1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,200	26,700
Average grade of																				
gold, g/t		1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.37	1.53
Metal																				
gold, kg		2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	1,647	40,747
Ore, th.m^3		577	577	577	577	577	577	577	577	577	577	577	577	577	577	577	577	577	462	10,269
RoM, th.m^3	3,362	3,938	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	5,250	132,550
Strip ratio, m^3/t		2.2	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.0	4.6

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Table 13-6 Project financing

Total sum of investments, th. USD	208,647
CAPEX of the construction period	146,892
Exploration and design expenditures	2,562
Sustaining capital	41,080
Floating capital	18,113
Own capital, %	70%
Borrowed capital, %	30%
Payment for borrowed capital, %	8.0%

Indicators		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	Total
CAPEX, total, incuding	46,367	62,443	58,757					5,340	5,340		4,343	4,343	4,343	4,343	4,343	4,343				204,304
Construction expenditures	29,378	58,757	58,757																	146,892
Exploration expenditures	2,562																			2,562
Sustaining capital								5,340	5,340		4,343	4,343	4,343	4,343	4,343	4,343				36,737
Floating capital	14,426	3,686																		18,113
Borrowed capital	13,910	6,117	6,712																	26,740
Debt by the end of the year	13,910	20,027	26,740																	
Own capital	32,457	14,273	15,662																	62,392
Repayment of financial debt				26,740																26,740
Interest payment		1,113	1,602	2,139																4,854

Table 13-7 Project Cash Flow

Parameters		Years of mining of the deposit																		Total
	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	
Ore mining, Kt		1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,200	26,700
Open-cast mining		1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,200	26,700
RoM, th.m^3		3,938	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	5,250	129,188
Ore processing, Kt		1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,200	26,700
Metal grade in processed ore																				
Gold		1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.37	1.53
Metal in processed ore, kg																				
Gold, kg		2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	2,300	1,647	40,747
Recovery after processing, %																				
Gold		85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	
Marketable product, kg		1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,400	34,635
Gold		1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,400	34,635
Price of marketable product																				
Gold, USD/g		53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	53.69	
Cost of marketable product, th. USD		**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**104,966**	**75,151**	**1,859,566**
Gold		104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	104,966	75,151	1,859,566
Refining costs, th. USD		**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,574**	**1,127**	**27,893**
Gold		1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,574	1,127	27,893
Total profit from sales		**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**103,391**	**74,024**	**1,831,673**
OPEX, th. USD																				
Ore mining costs		11,815	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	22,500	15,750	387,565
Ore processing costs		30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	24,000	534,000
General running costs		6,272	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	5,963	138,235
Royalty		6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	6,298	4,509	111,574
Environmental payments		525	525	525	525	525	525	525	525	525	525	525	525	525	525	525	525	525	376	9,298
Interest payment		1,113	1,602	2,139	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4,854
Total expenditures		**56,023**	**68,800**	**69,337**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**67,198**	**50,597**	**1,185,526**
Total profit		47,368	34,591	34,054	36,193	36,193	36,193	36,193	36,193	36,193	36,193	36,193	36,193	36,193	36,193	36,193	36,193	36,193	23,426	646,146
Depreciation		**1,737**	**5,211**	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	**8,686**	8,686	**8,686**	8,686	8,686	**4,343**	**16,760**	**149,653**
Property tax		936	1,815	2,140	1,949	1,758	1,567	1,435	1,361	1,229	1,085	990	894	799	703	608	512	416	184	20,382
Profit prior to taxation		44,694	27,564	23,228	25,558	25,750	25,941	26,073	26,147	26,279	26,422	26,518	26,613	26,709	26,804	26,900	26,996	31,434	6,482	476,112
Profit tax		8,939	5,513	4,646	5,112	5,150	5,188	5,215	5,229	5,256	5,284	5,304	5,323	5,342	5,361	5,380	5,399	6,287	1,296	95,222
Net profit		**35,755**	**22,051**	**18,583**	**20,447**	**20,600**	**20,753**	**20,858**	**20,917**	**21,023**	**21,138**	**21,214**	**21,291**	**21,367**	**21,444**	**21,520**	**21,596**	**25,147**	**5,185**	**380,890**
VAT returned from CAPEX		4,560	9,119	9,119	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	22,798
Free capital of the enterprise (net profit+depreciation+ VAT returned from CAPEX)		**42,052**	**36,382**	**36,388**	**29,133**	**29,285**	**29,438**	**29,544**	**29,603**	**29,709**	**29,824**	**29,900**	**29,976**	**30,053**	**30,129**	**30,206**	**30,282**	**29,490**	**21,946**	**553,340**
Calculation of the cost of property of the enterprise (th. USD)																				
Initial cost by the beginning	0	21,715	65,144	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	
Average initial cost for the year	10,857	43,429	86,858	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	
Initial cost by the end	21,715	65,144	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	108,573	
Remaining cost by the beginning	0	21,715	63,406	101,624	92,938	84,252	75,566	66,881	63,535	60,189	51,503	47,160	42,817	38,475	34,132	29,789	25,446	21,103	16,760	
Average remaining cost for the year	10,857	42,560	82,515	97,281	88,595	79,909	71,224	65,208	61,862	55,846	49,332	44,989	40,646	36,303	31,960	27,617	23,274	18,932	8,380	
Remaining cost of fixed assets by the end	21,715	63,406	101,624	92,938	84,252	75,566	66,881	63,535	60,189	51,503	47,160	42,817	38,475	34,132	29,789	25,446	21,103	16,760	0	
Floating capital (30% from OPEX)	0	14,426	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	18,113	13,714	

Table 13-8 Calculation of integral parameters of project efficiency

Indicators	Years of mining of the deposit																			Total
	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	
Influx of funds																				
Net profit		35,755	22,051	18,583	20,447	20,600	20,753	20,858	20,917	21,023	21,138	21,214	21,291	21,367	21,444	21,520	21,596	25,147	5,185	380,890
Depreciation		1,737	5,211	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	8,686	4,343	16,760	149,653
VAT returned		4,560	9,119	9,119	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	22,798
Change in floating capital	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4,399	13,714	18,113
Total	0	**42,052**	**36,382**	**36,388**	**29,133**	**29,285**	**29,438**	**29,544**	**29,603**	**29,709**	**29,824**	**29,900**	**29,976**	**30,053**	**30,129**	**30,206**	**30,282**	**33,889**	**35,659**	**529,401**
Outlow of funds																				
CAPEX to construction	29,378	58,757	58,757	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	146,892
Exploration and design expenditures	2,562																			2,562
Sustaining capital	0	0	0	0	0	0	0	5,340	5,340	0	4,343	4,343	4,343	4,343	4,343	4,343	4,343			41,080
Replacement of mining equipment, motor transport and partially plant equipment											4,343	4,343	4,343	4,343	4,343	4,343	4,343			30,400
Tailings pond expansion								5,340	5,340											10,680
Change in floating capital	14,426	3,686	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	18,113
Total	**46,367**	**62,443**	**58,757**	0	0	**0**	**0**	**5,340**	**5,340**	**0**	**4,343**	**4,343**	**4,343**	**4,343**	**4,343**	**4,343**	**4,343**	0	0	**208,647**
Cash balance	-46,367	-20,391	-22,374	36,388	29,133	29,285	29,438	24,204	24,263	29,709	25,481	25,557	25,634	25,710	25,786	25,863	25,939	33,889	35,659	362,806
Same, cumulative fund	**-46,367**	**-66,757**	**-89,132**	**-52,744**	**-23,612**	**5,674**	**35,112**	**59,316**	**83,579**	**113,288**	**138,769**	**164,326**	**189,960**	**215,670**	**241,456**	**267,319**	**293,258**	**327,147**	**362,806**	
Net profit value by years, th. USD																				
Investments	46,367	62,443	58,757	0	0	0	0	5,340	5,340	0	4,343	4,343	4,343	4,343	4,343	4,343	4,343	0	0	208,647
Free cash	0	42,052	36,382	36,388	29,133	29,285	29,438	29,544	29,603	29,709	29,824	29,900	29,976	30,053	30,129	30,206	30,282	33,889	35,659	571,453
CF	-46,367	-20,391	-22,374	36,388	29,133	29,285	29,438	24,204	24,263	29,709	25,481	25,557	25,634	25,710	25,786	25,863	25,939	33,889	35,659	
Discount rate		10%																		
Discount coefficient	1.00	0.91	0.83	0.75	0.68	0.62	0.56	0.51	0.47	0.42	0.39	0.35	0.32	0.29	0.26	0.24	0.22	0.20	0.18	
DCF	-46,367	-18,537	-18,491	27,339	19,898	18,184	16,617	12,421	11,319	12,599	9,824	8,958	8,168	7,447	6,790	6,191	5,645	6,705	6,414	101,123
DCF, cumulative fund	-46,367	-64,904	-83,395	-56,057	-36,159	-17,975	-1,358	11,063	22,382	34,981	44,805	53,763	61,931	69,378	76,168	82,360	88,005	94,710	101,123	
Discount investments	46,367	56,766	48,559	0	0	0	0	2,740	2,491	0	1,674	1,522	1,384	1,258	1,144	1,040	945	0	0	165,890
NPV		101,123																		
IRR		23.9%																		
PI		1.61																		
DPB		7.2																		
PB		5.8																		

Table 13-9 Main technical and economic parameters

Indicators	UoM	Value
1. Operational ore reserves	Kt	26,700
Operational metal reserves		
gold	kg	40,747
2. Average metal grade in operational reserves		
gold	g/t	1.53
3. Mining losses	%	3.0%
4. Dilution		3.0%
5. Annual capacity of the enterprise		
Pit in terms of ore	Kt	1,500
Plant in terms of ore		1,500
Pit in terms of waste	th.m^3	6,923
Pit in terms of RoM extraction		7,500
In terms of marketable product issue		
gold	kg	1,955
6. Term of mine provision with reserves	years	18
7. Transient recovery		
gold	%	85.0%
8. Issue of marketabe product during the whole operational period		
gold	kg	34,635
9. Investment expenditures	th. USD	208,647
design		2,562
CAPEX of the construction period		146,892
Floating capital		18,113
Investments during the operational period		41,080
10. Specific CAPEX to construction per 1 t of production capacity of the plant	USD/t	98
11. Cost of marketable product for the whole operational period	th. USD	1,859,566
12. Prime cost of 1 t of ore	USD/t	50.8
Expenditures for ore mining		14.5
Expenditures for ore processing		20.0
Running expenditures		5.2
Royalty		4.2
Environmental payments		0.3
Interest payment		0.2

Depreciation		5.6
Property tax		0.8
13. Full reserves for the whole period of development of the deposit	th. USD	1,355,561
Expenditures for ore mining		387,565
Expenditures for ore processing		534,000
Running expenditures		138,235
Royalty		111,574
Environmental payments		9,298
Interest payment		4,854
Depreciation		149,653
Property tax		20,382
14. Price of marketable product		
gold	USD/g	53.69
15. Profit		
– taxable profit for the estimated year (annual average)	th. USD	26,451
– taxable profit for the operational period		476,112
– net profit for the estimated year (annual average)		21,161
– net profit for the operational period		380,890
16. Depreciation		
– for the estimated year (annual average)	th. USD	8,314
– for the operational period		149,653
17. Net profit + depreciation		
– for the estimated year (annual average)	th. USD	29,475
– for the operational period		530,542
18. discount rate	%	10.0%
19. NPV	th. USD	101,123
20. IRR	%	23.93%
21.PI	-	1.61
22. PB	years	5.8
DPB		7.2

13.5. Project Sensitivity, Risk Assessment

At the early stage of investment research of mining projects the risks of non-confirmation of reserves are most significant.

Process risks (underreception of marketable product due to non-achievement of resource mining and processing plans, reduction of the value of the planned process recovery, lower-grade ore mining than planned etc.) are mostly derivatives from the problems of reliability and predictability of the resource base of the enterprise.

At the same time the obligatoriness of all pre-investment studies of the porject (process studies, detail engineering research etc.) shall be noted, and at this stage they haven’t been completed which leads to increase of process risks.

The analysis of the project sensitivity is conducted on the basis of alteration of three main parameters from -25% to +25% with increment 5%:

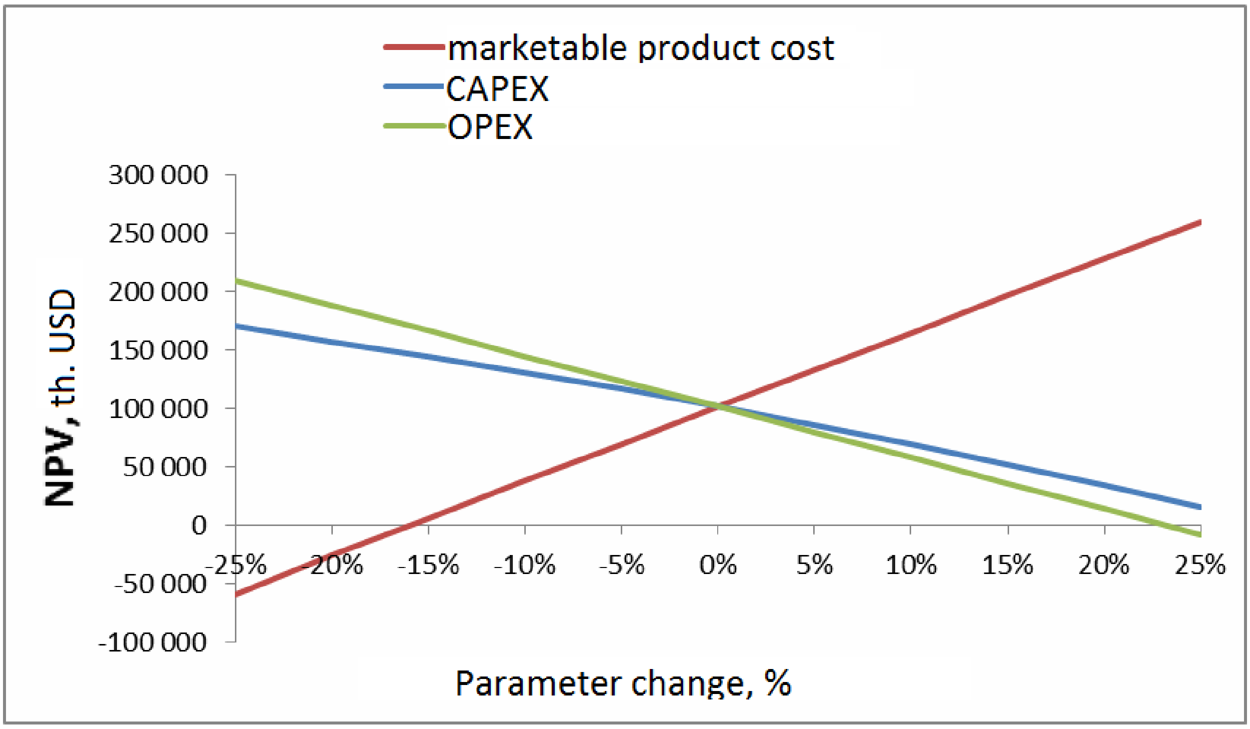
- volume of investments;
- sales price;
- total costs.

The results of the sensitivity analysis of the financial modeling key parameters (metal price, OPEX and CAPEX) are given in Figure 13-1.

As the diagrams show, this variant of calculations has quite a sufficient margin safety, both in terms of CAPEX and OPEX. The integral parameters of the project efficiency are most sensitive to change of the price of the marketable product.

However, quite a high current level of prices for gold as well as the probability of such prices remaining at this level in the short- and middle-term perspective allow to speak of some margin of financial safety.

Figure 13-1 Project sensitivity analysis



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14. Conclusions

The completed technical and economic analysis testifies to economic profitability of the project.

Indicators	UoM	Value
Annual productive capacity of the enterprise	Kt	1,500
Production of marketable gold during the whole operational term	kg	34,635
CAPEX	th. USD	208,647
Cost of marketable product during the whole operational term	th. USD	1,859,566
Prime cost of 1 t of ore	USD/t	50,8
NPV	th. USD	101,123
IRR	%	23.93%
PI	relative units	1.61
PB	years	5.8
DPB	years	7.2

15. Qualified Specialist's Declaration

This report was prepared and signed by Qualified Specialists who, while having the required experience in the study of mineralization of the research type at this deposit, are recognized Qualified Specialists in accordance with this definition in the JORC Code.

While signing this document we thus confirm that the report terminology, qualification of mineral resources and the results received during this investigation correspond to the policy and rules required and expained in the JORC Code for the control and quality of reporting Mineral Resource Statements.

Signature:

Nikandrov A.N.

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17. Appendixes

Appendix 1. Glossary of Technical Terms and Abbreviations

Term	Description
%	percent
azimuth, azimuth angle	borehole deviation (from north)
binary	digital file containing characteristics readable by the computer only
channel sampling	in exploration, a sampling method by means of cutting-out of continuous samples from the soil or the working wall
cut-off grade	the threshold value in exploration and geological resources estimation above which ore material is selectively processed or estimated
variation	in statistics, the measure of variance around the mean value of a data set
variogram	graph showing variability of an element by increasing spacing between samples
variography	the process of constructing a variogram
year	year
g	gram
histogram	diagrammatic representation of data distribution by calculating frequency of occurrence
geometric mean	the antilog of the mean value of the logarithms of individual values. For a logarithmic distribution, the geometric mean is equal to the median. A mean geometric value equals to median in case of lognormal distribution.
probability curve	diagram showing cumulative frequency as a function of interval size on a logarithmic scale
declustering	in geostatistics, a procedure allowing bounded grouping of samples within the octant sectors of a search ellipse
wireframe model	3D surface defined by triangles
km	kilometre
core sampling	in exploration, a sampling method of obtaining ore or rock samples from a borehole core for further assay
JORC Code	Australasian Code for Reporting Mineral Resources and Ore Reserves
compositing	in sampling and resource estimation, process designed to carry all samples to certain equal length
Kriging	method of interpolating grade using variogram parameters associated with the samples' spatial distribution. Kriging estimates grades in untested areas (blocks) such that the variogram parameters are used for optimum weighting of known grades. kriging weights known grades such that variation of the estimation is minimised, and the standard deviation is equal to zero (based on the model)
coefficient of variation	in statistics, the normalised variation value in a sample population
coefficient of correlations	statistical measure of the degree of similarity between two parameters
lag	the chosen spacing for constructing a variogram
lognormal	relates to the distribution of a variable value, where the logarithm of this variable is a normal distribution
m	meter
M	million or mega (10^6)
macro	a set of commands written as a computer program for reading and handling data
Median	value occupying the middle position in a database
mm	millimetre

Term	Description
OK	ordinary kriging
omni	in all directions
population	in geostatistics, a population formed from grades having identical or similar geostatistical characteristics. Ideally, one given population is characterised by a linear distribution
sample	specimen with analytically determined grade values for the components being studied
reserves	mineable geological resources
resources	geological resources (both mineable and unmineable)
range	with increase of the distance between pairs, the value of the corresponding variogram generally grows However the value of the average square of the difference between pairs of values does not change starting from the specific value of the distance, and the variogram reaches its plateau. The horizontal spacing at which a variogram reaches its plateau is called the range. Above this spacing there is no correlation between samples.
sill	a value of variation at which the variogram curve starts flattening
cm	centimeter
Average	arithmetic mean
standard deviation	the statistical value of data variance around the mean value
string	series of 3D points connected in series by straight lines
t	ton
t/m^3	ton per cubic meter
DTM	digital terrain model, a 3D wireframe model of the surface, for example, topography.
nugget effect	measure of the variability during repeat analysis of a sample due to a measurement error or the presence of natural, small-scale variability. Although the variogram value at 0 spacing should be equal to zero, these factors may affect the values of samples taken at a very short distance from each other such that their values may vary. A vertical jump from the zero value at the origin of a variogram with very small spacing is called the nugget effect.
ASCII	digital computer format containing text data
IDW	inverse distance weighting method
3D	volume (three-dimensional) model or data