CENTERRA GOLD INC.

TECHNICAL REPORT ON THE GATSUURT GOLD PROJECT, MONGOLIA

NI 43-101 Report

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CAUTIONARY NOTE REGARDING FORWARD-LOOKING INFORMATION

Information contained in this Technical Report and the documents referred to herein which are not statements of historical facts, may be "forward-looking information" for the purposes of Canadian securities laws. Such forward looking information involves risks, uncertainties, and other factors that could cause actual results, performance, prospects and opportunities to differ materially from those expressed or implied by such forward looking information. The words "expect", "target", "estimate", "may", "will", and similar expressions identify forward-looking information. These forward-looking statements relate to, among other things, mineral reserve and mineral resource estimates, grades and recoveries, development plans, mining methods and metrics including strip ratio, recovery process and production expectations including expected cash flows, capital cost estimates and expected life of mine operating costs.

Forward-looking information is necessarily based upon a number of estimates and assumptions that, while considered reasonable by Centerra Gold Inc. ("Centerra") are inherently subject to significant political, business, economic and competitive uncertainties, and contingencies. There may be factors that cause results, assumptions, performance, achievements, prospects, or opportunities in future periods not to be as anticipated, estimated, or intended. These factors may include following risks relating to the Gatsuurt Gold Project and/or Centerra: (A) strategic, legal, planning and other risks, including political risk; resource nationalism including the management of external stakeholder expectations; the impact of changes in, or to the more aggressive enforcement of laws, regulations and government practices; the impact of changes to and the increased enforcement of, environmental laws and regulations; potential defects of title to the property that are not known as of the date hereof; the inability of Centerra to enforce its respective legal rights in certain circumstances; risks related to anti-corruption legislation; potential risks related to kidnapping or acts of terrorism; the terms pursuant to which the Mongolian Government will participate, or to take a special royalty, in the Gatsuurt Project, the ability of the Company to successfully negotiate agreements for the development of the Gatsuurt Project and the risk to the Gatsuurt licences from a claim made by the Movement for Saving Mt. Noyon NGO and of adverse decisions or actions taken by Mongolian courts or governmental agencies relating thereto; (B) risks relating to financial matters, sensitivity of the business to the volatility of metal prices; the imprecision of Mineral Reserves and Mineral Resources estimates, and the assumptions they rely on; the accuracy of the production and cost estimates; and (C) risks related to operational matters and geotechnical issues, including the adequacy of insurance to mitigate operational risks; mechanical breakdowns; the occurrence of any labour unrest or disturbance; the ability to accurately predict decommissioning and reclamation costs, including closure costs; and the ability to attract and retain qualified personnel.

There can be no assurances that forward-looking information and statements will prove to be accurate, as many factors and future events, both known and unknown could cause actual results, performance or achievements to vary or differ materially, from the results, performance or achievements that are or may be expressed or implied by such forward-looking statements contained herein or incorporated by reference. Accordingly, all such factors should be considered carefully when making decisions with respect to Centerra and prospective investors should not place undue reliance on forward-looking information. Forward-looking information in this Technical Report is as of the issue date, December 14, 2017. Centerra and the Qualified Persons who authored this Technical Report assume no obligation to update or revise forward-looking information to reflect changes in assumptions, changes in circumstances or any other events affecting such forward looking information, except as required by applicable law.

Non-GAAP Measures

This Technical Report contains certain non-GAAP financial measures. These financial measures do not have any standardized meaning prescribed by GAAP and are therefore unlikely to be comparable to similar measures presented by other issuers, even as compared to other issuers who may be applying the World Gold Council ("WGC") guidelines, which can be found at http://www.gold.org.

Centerra believes that the use of these non-GAAP measures will assist analysts, investors and other stakeholders of the Company in understanding the costs associated with producing gold, understanding the economics of gold mining, assessing our operating performance, our ability to generate free cash flow from current operations and to generate free cash flow on an overall Company basis, and for planning and forecasting of future periods. However, the measures do have limitations as analytical tools as they may be influenced by the point in the life cycle of a specific mine and the level of additional exploration or expenditures a company has to make to fully develop its properties. Accordingly, these non-GAAP measures should not be considered in isolation, or as a substitute for, analysis of our results as reported under GAAP.

Definitions

The following is a description of the non-GAAP measures used in this Technical Report. The definitions are similar to the WGC's Guidance Note on these non-GAAP measures:

- All-in sustaining costs per ounce sold include adjusted operating costs, the cash component of capitalized stripping costs, corporate general and administrative expenses, accretion expenses, and sustaining capital. The measure incorporates costs related to sustaining production.
- All-in costs per ounce sold include all-in sustaining costs and additional costs for growth capital, global exploration expenses, business development costs, project development costs and social development costs not related to current operations.
- All-in cost per ounce sold with or without tax exclude the following:
  - Working capital (except for adjustments to inventory on a sales basis).
  - All financing charges (including capitalized interest).
  - Costs related to business combinations, asset acquisitions and asset disposals.
  - Other non-operating income and expenses, including interest income, bank charges, and foreign exchange gains and losses.
- Adjusted operating costs include cost of sales (cash component), regional office administration, mine standby costs, community costs related to current operations, refining fees.
- Sustaining capital is a capital expenditure necessary to maintain existing levels of production. The sustaining capital expenditures maintain the existing mine fleet, mill and other facilities so that they function at levels consistent from year to year.
- Growth capital is capital expended to expand the business or operations by increasing productive capacity beyond current levels of performance.
- Free cash flow (unlevered) is calculated as cash provided by operations less additions to property, plant and equipment.

A reconciliation of the non-GAAP measures used in this Technical Report is contained in Centerra's Management's Discussion & Analysis for the year ended December 31, 2016, and for the quarter ended September 30, 2017, which is available on SEDAR at www.sedar.com.

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1 SUMMARY

BACKGROUND AND PROJECT DESCRIPTION

The Gatsuurt Gold Project (Gatsuurt or the Project or the Property) is located approximately 200 kilometres (km) south of Mongolia's northern border with the Russian Federation and 90 km north of Mongolia's capital city of Ulaanbaatar in the Mandal Soum (district) of Selenge Aimag (province), north-central Mongolia. Centerra Gold Inc. (Centerra or the Company) has prepared this Technical Report on the Gatsuurt Gold Project which supports the disclosure of the Project's Mineral Resource and Mineral Reserve estimates at October 31, 2017. The Technical Report has been prepared in accordance with National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101). All dollar figures in this Technical Report refer to US dollars (US$ or $), unless otherwise noted.

Exploration in the Project area was carried out during various intervals from the 1970s to 2016 and evolved from an initial gold placer discovery and mining to the discovery of gold in bedrock and resource delineation at Gatsuurt. In 1996, Cascadia LLC, a Mongolian subsidiary of Cascadia Mining Inc. (together Cascadia), obtained three major exploration licences in Mongolia, including the Kharaagol Licence that covers a significant part of the Yeroogol gold trend, including the Gatsuurt placer area. Cameco Gold Inc. (Cameco, later Centerra Gold Inc.) funded Cascadia's exploration programs from 1997 to 2001. In 2001, Cameco acquired Cascadia (now Centerra Gold Mongolia LLC (CGM)).

The Project is situated 35 km from Centerra's Boroo Project in Mongolia. It is connected to the Boroo mine site by a 52 km road which was completed in 2010.

The mineral rights to the Project tenements are held under four mining licences covering 2,937 ha. The Property is subject to a sliding scale royalty fee payable to the Mongolian Government on gold sales pursuant to the Minerals Law (as amended in 2010), which starts at 5% and increases to a maximum of 10%, depending on the price per ounce of gold (the maximum being reached at a gold price of $1,300 per ounce or above). However, for gold sales to Mongolbank, the rate of royalty payable to the Mongolian Government is set at a flat rate of 2.5%. This 2.5% royalty for gold sales to Mongolbank is scheduled to expire on January 1, 2019 and there are no assurances that it will be continued. Licences MV-00371A and MV-00431A are encumbered by an underlying 3% Net Smelter Return (NSR) royalty in favour of Gatsuurt LLC, an arm's length Mongolian limited liability company.

In January 2015, the Project was designated as a mineral deposit of strategic importance by Mongolian Parliament. Such a designation entitled Mongolia, pursuant to the Minerals Law, to take a 34% ownership interest in the Project. The Government of Mongolia and CGM have entered into a non-binding memorandum of understanding to exchange Mongolia's 34% interest in the Project for a 3% special royalty on the Project, though that arrangement is subject to the negotiation and execution of definitive agreements.

Gold mineralization at the Project occurs in two zones, the Central Zone and the Main Zone, separated laterally from each other by the Sujigtei fault.

At the Central Zone, continuous gold mineralization has been traced over a strike length of 900 metres (m) over horizontal widths that vary from two metres to greater than 70 m. It comprises a broad lower grade shell (over 1.0 gram of gold per tonne (g/t Au)) containing higher-grade (over 3.0 g/t Au) lenses with variable lateral and vertical continuity. Gold mineralization has been traced by drilling to a maximum depth of 360 m and is open at depth. The Central Zone has oxide and transitional ore, which will be processed through the existing Boroo mill, and refractory ore, which is expected to be processed through the bio-oxidation facility to be constructed at Boroo.

The Main Zone contains fairly continuous gold mineralization over a 400 m strike length. The gold mineralization is limited along strike but remains open at depth. The altered and mineralized zone trends parallel to the Sujigtei fault and dips sub-vertically. At the Main Zone, the gold mineralization is almost entirely refractory although leach recovery testwork was limited.

The Project is planned as a conventional truck and shovel open pit mine. A total of approximately 15.4 million tonnes of ore at a grade of 2.7 g/t Au, containing a total of approximately 1.3 million ounces of gold, is planned to be mined. Processing of the oxide ore will occur through an existing Carbon-in-Pulp (CIP) facility and the sulphide ore through the BIOX® process followed by a Carbon-in-Leach (CIL) facility over a total mine life of ten years from two open pits, the Central Zone pit and the smaller Main Zone pit.

Table 1-1 provides a summary of Mineral Reserves and Mineral Resources exclusive of Mineral Reserves with an effective date of October 31, 2017. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves adopted on May 10, 2014 (CIM definitions) were followed for this estimate.

TABLE 1-1 GATSUURT MINERAL RESERVE AND MINERAL RESOURCE SUMMARY

Mineral Reserves[1,2]

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable	15,356	2.7	1,316

Mineral Resources (Exclusive of Mineral Reserves)[1,3]

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Indicated	10,988	1.9	678

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Inferred	3,812	2.1	263

Notes:

1. The effective date of the Mineral Reserve and Mineral Resource estimates is October 31, 2017.
2. The Mineral Reserve has been estimated using a gold price of US$1,250 per ounce and have been validated by the positive project NPV that uses a 5% discount rate.
3. The Mineral Resources are exclusive of Mineral Reserves and have been estimated on the basis of a gold price of US$1,450 per ounce. The Mineral Resources have been constrained by an economic open pit in accordance with CIM best practices.

PROJECT LOCATION, DESCRIPTION, AND CLIMATE

The Project is located approximately 200 km south of Mongolia’s northern border with the Russian Federation and 90 km north of Mongolia’s capital city of Ulaanbaatar in the Mandal Soum of Selenge Aimag, north-central Mongolia.

Access to the Project is, in part, via the highway that connects Ulaanbaatar to Sükhbaatar (Suhbaatar) near the north border with Russia, which roughly takes one and a half hours. From the highway, there is a two lane all-weather road through the general countryside which leads to the site, approximately 10 km from the highway. The new Gatsuurt process plant facilities will be constructed adjacent to Centerra’s existing Boroo process plant. The access road connecting the Gatsuurt site with the Boroo Mine is 52 km in length measured from the ore stockpile pad at the Gatsuurt site to the run-of-mine (ROM) pad at the Boroo crusher located in the Boroo site. The road is of all-weather, earth and stone construction designed to accommodate 40 t tractor trailers.

The mineral rights to the Project tenements are held under four mining licences covering 2,937 ha. The four mining licences were granted between July 1995 and November 2005 and may be maintained over the 70 year term of the licences by making annual payments of US$10/ha to the Mongolian Government.

Climate in the Project area is mid-continental, temperate to cold. Average annual temperature is approximately 0ºC. Temperatures may drop to minus 40ºC during the months of December through February. Summer temperatures may exceed 40ºC, however, the average daytime summer temperature is approximately 20ºC. The area receives approximately 25 cm of precipitation per year, most of which is rainfall from July through August. Winters are relatively dry, with a moderate amount of snow that may persist on the ground from early November to April due to prevailing low temperatures.

PROJECT HISTORY AND OWNERSHIP

Exploration in the Project area was carried out during various intervals from the 1970s to 2016 and evolved from an initial gold placer discovery and mining to the discovery of gold in bedrock and resource delineation at Gatsuurt.

In 1996, Cascadia obtained three major exploration licences in Mongolia, including the Kharaagol Licence that covers a significant part of the Yeroogol gold trend, including the Gatsuurt placer area. Cameco funded Cascadia's exploration programs from 1997 to 2001.

In 1997, quartz veins with visible gold were discovered in the placer bedrock floor and within boulders of altered granite, with disseminated arsenopyrite and pyrite noted in placer debris. This discovery was evidence that a significant gold-bearing system and alteration assemblage existed in bedrock under the placer deposit.

In 1998, Cascadia carried out further exploration to identify the bedrock source of the alluvial gold deposit in the Gatsuurt area. This work included detailed mapping of the Gatsuurt placer floor, soil sampling, topographic and ground magnetic surveys, and an induced polarization (IP) survey. The survey located major chargeability and resistivity targets over the Central and Main Zones and the South Slope. Four core drill holes were completed to test the IP anomalies and the bedrock targets identified in the Central and Main Zones. Drill hole GT-06 intersected broad widths of gold mineralization, including 1.57 g/t Au over 110 m, and is considered to be the Project discovery hole.

Limited drilling programs were completed from 1999 to 2000 with 16 core drill holes totalling 2,138 m completed at the Central Zone, and eight core drill holes totalling 1,174 m completed at the Main Zone. During this time, preliminary metallurgical testing of drill samples determined that gold mineralization in both zones is largely refractory.

In 2001, Cameco acquired Cascadia in exchange for the surrender of Cameco's 42% share ownership interest in Cascadia and $2.5 million cash to Cascadia. Cascadia is now Centerra's wholly-owned subsidiary, CGM. At the time of acquisition, the Project licences included 46 exploration licences. CGM was now the operator and, in late 2001, 25 drill holes were completed in the Central Zone and one hole in the Main Zone.

Additional exploration work was carried out from 2002 through 2010 including topographic surveys, stream, soil, and rock grab sampling on grids, ground magnetometer geophysical surveys, IP surveys, core drilling, and reverse circulation drilling.

In May 2005, Centerra retained SNC-Lavalin Engineers & Constructors Inc. (SNC-Lavalin) to complete a feasibility study on the Central Zone based on a stand-alone operating mine and processing facility on the Gatsuurt Mine site. The SNC-Lavalin study indicated that the size of the Gatsuurt resource was insufficient to justify the cost of building a stand-alone processing facility and infrastructure for the Project. A combined development plan and feasibility study was then carried out by Centerra in order to identify the scope for improvement of Project economics by utilizing portions of the existing facilities at the Boroo Mine. The study was completed in December 2005. It was determined that the coordinated development of the Gatsuurt and Boroo properties would create opportunities to maximize the use of existing assets and reduce the Project capital costs. Under this approach, this combined development plan was used as the basis for advancing Project planning, based on a sequential development of the Central Zone and Main Zone deposits, with the ore to be transported and processed at a modified Boroo processing plant.

GEOLOGY AND MINERALIZATION

The Project mining licences, including the Gatsuurt Deposit, are located in the North Khentei gold belt, as is Centerra's Boroo Deposit, which is located approximately 35 km northwest of Gatsuurt.

The Gatsuurt Deposit is identified as an orogenic gold deposit, also termed low-sulphide quartz gold.

The Gatsuurt Deposit comprises two significant zones of fairly continuous mineralized rock, the Central Zone (including the GT60 and South Slope areas) and the Main Zone, as well as significant prospects referred to as the 49-er and SW Ext areas. The Central and Main Zones are separated laterally from each other by the Sujigtei fault, possibly in a major dilational step-over along its fault trace. The Central Zone lies on the southeast, hanging wall side (east block down) of the Sujigtei fault, whereas the Main Zone is on the northwest side approximately 800 m to the southwest of the Central Zone.

In the Central Zone, located southeast of the Sujigtei fault, rocks are mostly early Paleozoic in age and are the oldest present in the general area of the deposit. Early Paleozoic Kharaa Series meta-sedimentary rock (generally meta-sandstone, some calcareous) has been intruded by early Paleozoic Boroogol Complex granodiorite, granite, and diorite. Rocks in the Main Zone, located within the northwest block of the Sujigtei fault are the Permian Dzuun Mod massive quartz-potassium feldspar phyric rhyolite, which regionally once formed part of a semi-circular body but was later wrenched apart in a left-lateral sense by displacements along the Sujigtei fault.

The combined length of the two zones from the GT-60 and South Slope areas at the northernmost edge of the Central Zone to the southernmost end of the Main Zone is approximately 2.0 km, and the combined width is approximately 300 m. The Gatsuurt

Deposit is open at depth, as mineralized drill intercepts have been encountered approximately 1,000 m below the ground surface.

MINERAL RESOURCES

Mineral Resources for the Project were estimated by Roscoe Postle Associates Inc. (RPA) using a block model constrained with three dimensional (3D) wireframes of the principal mineralized domains and incorporating all the drilling completed to the date of the resource estimate. Values for gold were interpolated into blocks using an inverse distance estimator to the power of three (ID^3) methodology. The Mineral Resource estimate, exclusive of Mineral Reserves, is summarized in Table 1-1.

Leapfrog Geo software (version 4.0) was used to construct the geological solids. GEOVIA GEMS software (version 7.6.3) was used to prepare assay data for geostatistical analysis, construct the block model, estimate gold and other metal grades, and tabulate Mineral Resources.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

MINERAL RESERVES

RPMGlobal (RPM) has estimated the Mineral Reserves for the Project based on the engineering studies and other work performed by RPM and other experts. The Mineral Reserve estimate is summarized in Table 1-1.

In order to define the economic mining area, pit limit optimization was completed using Geovia Whittle 4X Software. This provided a series of pit shells that were used for detailed mine design, which in turn were used for estimation of mineable quantities and NI 43-101 Mineral Reserves.

Although the QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate, the estimate should be considered within the context of the outcomes stated in this report. Of note, the Project’s base case economic modelling results indicate a positive net present value (NPV) of $39.5 million at an annual discount rate of 5%. Should any of the key revenue drivers (price, grade, recovery) incur any adverse variance to the plan, this may place the above estimate at risk.

MINING METHODS

The Project will be operated using a traditional Contractor-operated truck and shovel/excavator mining method. This is considered to be the most appropriate method as it is robust, commonly understood, and provides the required flexibility for the Project with regard to capacity increases and decreases, and shifts of mining location.

The Project comprises two pits, the Central Pit and the Main Pit; a ROM ore storage area near each of the two pit exits; four dumps for ex-pit storage of waste material; a diversion bench through the Central Pit to prevent interruption of the Gatsuurt River water flow through the Project; surface haul roads connecting the pits to the dumps and ROM areas; and a haulage road from the Project to the Boroo processing facility 52 km away.

The expected life of the Project is approximately nine years of mining (106 months duration) with ore feed, post-mining, extending the operation out to 10 years (119 months). Total ore mined and fed to process during this period is 15.4 million tonnes at an average grade of 2.7 g/t Au; requiring the removal of 72.6 million tonnes of waste and sub-grade material for a total material movement of 87.9 million tonnes. Table 1-4, below, shows the life of mine (LOM) mining and processing schedules.

METALLURGY

Mineralization within the Central Zone is categorized as oxide, transition, or sulphide. The Main Zone is defined as having predominantly sulphide mineralization. Lower transition and sulphide mineralization in both zones is refractory. Therefore, the metallurgical properties of this mineralization bear important implications for the design of a suitable processing facility to recover the gold and the economic significance of the Project.

Mineralogical and metallurgical studies were completed on samples from the Project to investigate the metallurgical characteristics of the reportedly refractory gold mineralization and to develop design criteria for evaluating processing options. This testing culminated with a large scale concentrate biological oxidation pilot plant in 2005. Testing has also been completed on oxide and upper transition samples to define recovery characteristics of this free-milling ore.

Testing completed to 2006 included mineralogical investigations, comminution testing, gravity recovery of gold, flotation testing, biological oxidation using BIOX® technology, and leach testing. Since 2006, additional testing has been performed to define cyanide destruction parameters on BIOX® leach tailings, leach testing on newly defined oxide resources, and variability testing of the BIOX® process using small batch tests.

RECOVERY METHODS

Gatsuurt ore will be treated at Centerra's Boroo Mine. Processing of the oxide ore will occur from Year 1 to Year 4 through the existing CIP facility at Boroo and the sulphide ore will be treated through a BIOX® process followed by a CIL facility which will be operational by Year 4.

The oxide flowsheet is a standard layout that consists of crushing, grinding, gravity concentration, cyanide leaching, and gold recovery in a CIP circuit.

Sulphide ore is milled through the existing crushing, grinding, and gravity circuit with gravity gold recovery estimated at 15% gold.

Grinding circuit cyclone overflow is fed to a new bank of six 40 m^3 capacity mechanically agitated flotation cells to recover approximately 6% to 8% by weight of flotation feed as a rougher concentrate. Flotation concentrate is thickened in a new high-rate thickener prior to biological oxidation. Most of the flotation reagents contained in the flotation concentrate slurry are removed by this thickener.

Flotation recoveries, residence times, concentrate grades, and flotation reagent consumptions were selected by SNC-Lavalin based on the metallurgical testing carried out as part of the feasibility study on the Central Zone in 2004 and 2005.

The design for the BIOX® circuit was supplied by Outotec in an engineering package based on pilot scale testwork conducted by Biomin South Africa Pty Ltd (now part of Outotec) during 2005. Reagent consumptions, process operating conditions, reactor size, and installed agitator power were also provided by Outotec.

The oxidation of the sulphide minerals occurs due to the action of the bacteria *thiobacillus ferrooxidans* and other species adapted to Gatsuurt concentrate. Essential nitrogen, phosphorus, and potassium required for bacterial growth are added as a solution. Sufficient carbon dioxide is assumed to be available from decomposition of the carbonate minerals in the ore and from the finely ground limestone slurry added to the reaction tanks to maintain solution pH in the optimum range for bacterial growth, typically between 1.2 and 1.5.

TAILINGS MANAGEMENT FACILITY

The Boroo site currently contains a tailings management facility (TMF) located approximately 5 km south of the process plant and 2 km south of the camp. This existing TMF contains a Main Cell at elevation 934.5 m above sea level (MASL), which will be expanded to an elevation of 937.7 MASL for the additional 3.6 million m^3 volume requirement for the oxide ore with an added capacity of 0.44 million m^3 for the BIOX® CIP tailings. The BIOX® CIL tailings will be stored in a new East Cell, which will occupy 9.8 million m^3 of tailings and constructed from an elevation of 893 MASL to an elevation of 924 MASL.

ENVIRONMENT AND PERMITTING

CGM has secured many of the operating permits required to operate the Project in accordance with Mongolian regulatory requirements. Approved permits and licences include Land Use Permit for the Gatsuurt Mine, Gatsuurt Mineral Reserve Report Approval (2013), Gatsuurt Mining Licence, Gatsuurt Feasibility Study (2014), BIOX® Plant Environmental Impact Assessment (EIA) (2010), Gatsuurt – Boroo Haul Road EIA (2006), Water Reserve Approval (2010), and Gatsuurt Landfill Permit (2007).

Outstanding permits and licences include Hazardous Chemicals Permit, Gatsuurt Gold Mine Detailed EIA (submitted), Water Use Permit, and Mine Closure Plan.

COMMUNITY SUSTAINABILITY

There are unique demographic challenges confronting Mandal and Bayangol soums. Due to its large population of youth between the ages of 20 and 24, the key challenge for Mandal Soum is driving vocational educational, economic opportunities, and work spaces for youth. In Bayangol Soum, the key challenge, based on its large population of young children, is providing sufficient educational facilities.

Statistically migration is declining in the Project area, although there is a perception among communities in the Project area that there is an influx of migrant herders who are viewed as a key source of pastureland degradation and land use conflict.

The levels of trust in public institutions in the Project area are generally low due to community dissatisfaction with the activities of public organizations, especially political parties. Community perceptions on corruption are also a key source of discontent with the political situation in both Mandal and Bayangol.

Welfare and community upliftment projects have generally low levels of awareness in the Project area. The program with the highest level of awareness is the “restocking livestock program” run by the Government.

Steep price inflation of goods and services is the major concern for households in the Project area, as are the poor quality of goods. Tunkhel bagh households experience these issues most acutely in the Project area due the bagh’s more remote location. A related issue is that households in the Project area appear to be increasingly living on credit, and struggling to pay back their loans.

There is also a general lack of awareness about the formal mining sector in the Project area, the processes and technicalities thereof, and the potential benefits and risks associated with large scale mining.

There are limited opportunities in the Project area to access tertiary and vocational education and the facilities that do exist in Mandal Soum provides vocational training for a predominantly male student population in traditionally male professions.

The main sources of drinking water in the Project area are wells, rivers, and springs, and mobile distribution points. Bayangol sources more water from unprotected wells (at a higher risk of contamination than protected wells) than Mandal Soum.

Overgrazing and the accompanying pastureland degradation is a central issue in the Project area, due to existing herders and livestock owners having increased the number of animals in recent years. Overgrazing problems have resulted in land use disputes among herders in some areas, and between herders and crop farmers.

The social baseline survey found that 60% of Mandal Soum households were involved in Artisanal and Small-scale Mining (ASM) – to supplement their incomes/livelihoods, as a hobby, or as a poverty reduction strategy. A total of 228 soum residents were members of the 30 officially registered ASM cooperatives (three large NGOs, with around 30 to 100 members and 27 small cooperatives with 3 to 10 members).

In September 2015, over 5,000 miners were recorded conducting trespass and illegal mining (TIM) at the Project site. These miners gained access to the site in large vehicle convoys and overcoming safety and environmental measures in place at the site. Since September 2015, the illegal activity at Gatsuurt has escalated and the security at site in cooperation with the Mongolian Police have undertaken a total of 28 operations during which over 5,000 intruders and more than 200 vehicles have been expelled from the area.

There is a lack of health care personnel (doctors and nurses) in the Project area of influence. Specifically, Mandal Soum has a significant shortage of doctors, but a relatively better supply of nurses, while Bayangol Soum lags behind both World Health Organization (WHO) indicators for doctors and the national average for nurses.

Child labour most often takes the forms of herding, artisanal mining, or working for forestry companies.

Worship of Noyon Uul (mountain), located immediately north of the Project area, is a traditional heritage practice near the Project area. According to local households, worship of the mountain occurred before the Socialist era, but was then forbidden during communist rule (1920s to 1990s).

CAPITAL COSTS

The Project consists of two phases: refurbishment of the existing mill at the depleted Boroo Mine to process leachable ore from the Gatsuurt pits, and construction of a BIOX® plant to process the sulphide ore. Pre-production costs for the first phase have been assumed to occur in Year 0 and Year 1, and pre-production costs for the second phase have been assumed to occur in Years 2 to 4, with mining for the first phase commencing in Year 1 and for the second phase commencing in Year 4. Total initial direct and indirect capital costs and pre-production operating costs are estimated to be $244.9 million, excluding the closure deposit. This total includes all applicable Owner's costs, Boroo refurbishment, new sulphide process circuit (Flotation, BIOX®, and CIP plant), infrastructure, engineering, mining, and project management costs, as well as taxes, with a contingency of 10% upon commencement of construction. The estimates are summarized in Table 1-2.

TABLE 1-2 PRE-PRODUCTION EXPENDITURES SUMMARY

Item	Total ($ x 1000)
Mining and Mine Construction	2,815
Refurbishment of Boroo Mill	7,728
Upgrade of Haul Road	14,000
Utilities (Powerlines and Substations)	11,501
Tailings Management Facility	21,342
Water Diversion and Treatment	9,981
Sulphide Circuit (Flotation, BIOX® and CIP)	95,397
Owner's Costs	27,464
Contingency (10%)	26,989
Taxes	**27,639**
TOTAL	**244,856**

Sustaining capital requirements for the Project are minimal, primarily due to the short life of mine, as well as contracting out of the mining tasks, obviating the need for allocating sustaining capital for mobile mining equipment.

Closure costs have been estimated at $15.4 million for the Project and will be incurred after mining is complete.

OPERATING COSTS

The LOM operating cost estimates on a per tonne processed basis are: mining $10.97, ore haulage $5.37, oxide processing $11.90, sulphide processing $26.42, and general and administration $8.92 for a total operating cost (excluding royalties) of $47.09 per tonne processed (Table 1-3). A cost model was prepared to estimate the operating cost for site mining activities and the ore haulage to the Boroo plant. The level of detail of the operating costs are commensurate with a feasibility study (+/-15%) as the key mining cost items are supported by third party tender submissions for contract mining and a first principles model to account for sufficient technical staff supervision and management of the mining contractor.

TABLE 1-3 LIFE OF MINE OPERATING COST SUMMARY

Cost Centre	Unit Value $/t Total Material	Unit Value $/t Ore	Value $M
Mining Costs			
Contract Mining	1.62	9.26	142
Overheads	0.22	1.25	19
Water Management	0.08	0.46	7
Total Mining	**1.92**	**10.97**	**168**
Ore Haulage	**0.94**	**5.37**	**82**
Total Mining and Haulage	**2.85**	**16.34**	**251**
Processing Costs			
Oxide Processing	0.66	11.90	58
Sulphide Processing	3.15	26.42	277
Total Processing	**3.81**	**21.83**	**335**
General and Administration	**1.56**	**8.92**	**137**
Total Operating Cost	**8.22**	**47.09**	**723**

ECONOMIC ANALYSIS

The material economic assumptions such as operating and capital cost estimates used for the calculations presented in this section are summarized in Tables 1-2 and 1-3.

CASH FLOW FORECAST

The LOM production plan is presented in Table 1-4 and the LOM cash flow forecast is summarized in Table 1-5.

Considering the Project on a stand-alone basis, the undiscounted after-tax cash flow totals $115.1 million over the mine life and simple payback occurs during Year 8 following the start of production.

The on-site cash operating cost averages $651 per ounce of gold, excluding royalties. Including royalties, the cash operating cost rises to $839 per ounce of gold. The All-in Sustaining Cost (AISC), including sustaining capital cost, is $870 per ounce of gold, and when construction and closure costs are included, the All-In-Cost equals $1,103 per ounce of gold before taxes and $1,143 per ounce of gold after tax. The average annual gold production during operation is approximately 111,000 ounces per year.

The NPV, at an annual discount rate of 5%, is $39.5 million, and the Project's IRR is 9.3%.

TABLE 1-4 LIFE OF MINE PRODUCTION PLAN SUMMARY

		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Ore mined	kt	**15,356**	1,631	3,113	2,846	558	1,188	1,005	1,363	2,044	1,607	
Gold grade	Au g/t	**2.7**	2.2	2.4	2.9	2.7	2.5	2.8	3.4	2.7	2.7	
Waste mined	t x 1000	**72,588**	8,597	9,646	9,948	12,202	11,572	7,501	7,167	4,335	1,620	
Total Tonnes mined	t x 1000	**87,944**	10,229	12,759	12,794	12,759	12,759	8,506	8,530	6,380	3,227	
Average Ore mined	kt/d		4.0	8.5	7.8	1.5	3.3	2.7	3.7	5.6	4.4	
Average mined	kt/d		28.0	34.9	35.1	35.0	35.0	23.2	23.4	17.5	8.8	
Stripping Ratio	w/o	**4.7**	5.3	3.1	3.5	21.9	9.7	7.5	5.3	2.1	1.0	
Tonnes Milled	**kt**	**15,356**	**442**	**1,752**	**1,757**	**1,319**	**1,752**	**1,752**	**1,757**	**1,752**	**1,752**	**1,322**
Head grade gold	Au g/t	**2.7**	2.4	2.2	2.2	3.1	3.2	2.7	3.1	2.8	2.6	2.1
Tonnes Milled per Day	kt/d		1.21	4.79	4.81	3.61	4.80	4.79	4.81	4.80	4.80	3.61
Recovery	%	**84.4%**	79.4%	78.4%	79.7%	83.3%	85.5%	85.6%	86.2%	86.9%	88.5%	84.3%
Gold Ounces Produced	**oz x 1000**	**1,111**	**26**	**97**	**99**	**109**	**155**	**131**	**151**	**136**	**130**	**75**

TABLE 1-5 LIFE OF MINE CASH FLOW FORECAST

		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11+
Sales and Revenue													
Gold Price	US$/oz	**1,250**	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Gold Produced (Sold)	oz x 1000	**1,111**	26	97	99	109	155	131	151	136	130	75	-
Gold Sales	**$ million**	**1,386.1**	**33.0**	**121.2**	**124.0**	**136.2**	**193.4**	**164.0**	**188.0**	**170.1**	**162.6**	**93.6**	**0.0**
Operating Costs													
Mining and Haulage(1)	$ million	**250.9**	21.2	33.4	33.7	30.2	32.6	25.9	25.5	22.9	17.2	8.0	0.0
Processing(1)	$ million	**335.3**	5.3	20.8	20.9	28.9	46.3	46.3	46.4	46.3	46.3	27.8	0.0
General and Administration(1)	$ million	**137.0**	8.6	13.8	13.8	13.8	13.8	13.8	13.8	13.8	13.8	11.5	0.0
Royalties(1)	$ million	**208.9**	4.8	17.8	18.2	20.6	29.4	24.9	28.6	25.8	24.7	14.1	0.0
Cash Operating Costs	**$ million**	**932.0**	**39.9**	**85.8**	**86.6**	**93.5**	**122.1**	**110.9**	**114.3**	**108.9**	**102.0**	**61.5**	**0.0**
Capital and Other Costs													
Construction Capital	$ million	**217.9**	17.3	18.6	30.1	91.2	0.6	0.3	0.4	0.3	0.0	0.0	0.0
Contingency	$ million	**27.0**	1.9	2.2	4.3	9.7	0.4	0.3	0.5	0.4	0.0	0.0	1.4
Sustaining Capital(2)	$ million	**36.6**	0.1	3.1	12.4	5.8	3.5	3.1	4.2	3.6	0.5	0.5	0.0
Reclamation & Closure	$ million	**15.4**	1.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	13.8
Capital Expenditure	**$ million**	**296.9**	**20.8**	**23.8**	**46.8**	**106.7**	**4.4**	**3.7**	**5.1**	**4.3**	**0.5**	**0.5**	**15.2**
Working Capital	$ million	**0.2**	2.9	9.3	0.2	2.5	7.6	(2.2)	2.0	(1.4)	(0.4)	(20.2)	0.0
Discount Post Closure Cost	$ million	**(1.6)**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	(1.6)
Constr.+Contingency	*$ million*	***244.9***	*19.2*	*20.7*	*34.3*	*100.9*	*1.0*	*0.6*	*0.8*	*0.7*	*0.0*	*0.0*	*1.4*
Cashflow													
Pre-tax cashflow	$ million	**158.6**	(30.6)	2.2	(9.6)	(66.4)	59.4	51.6	66.6	58.3	60.5	51.9	(13.6)
Income Tax Payable	$ million	**43.5**	0.0	0.7	5.0	5.0	8.5	4.4	8.6	5.6	5.5	0.2	0.0
Free Cash Flow (FCF)(2)	**$ million**	**115.1**	**(30.6)**	**1.5**	**(14.6)**	**(71.5)**	**50.9**	**47.3**	**58.0**	**52.7**	**55.0**	**51.6**	**(13.6)**
Cumulative FCF	$ million		(102.1)	(100.7)	(115.3)	(186.8)	(135.9)	(88.6)	(30.7)	22.0	77.1	128.7	115.1
All In Sustaining Cost(2)	US$/oz	**870**	1,512	915	996	909	810	867	786	825	786	825	
All In Cost before tax(2)	US$/oz	**1,103**	2,295	1,129	1,341	1,833	816	871	792	830	786	825	
All In Cost after tax(2)	US$/oz	**1,143**	2,295	1,129	1,341	1,833	816	871	792	830	786	825	

Notes:
(1) Values do not include depreciation.
(2) Non-GAAP measure, see discussion under "Non-GAAP Measures" in Centerra's Management's Discussion and Analysis for the year ended December 31, 2016, and for the quarter ended September 30, 2017.

INTERPRETATION AND CONCLUSIONS

Based on the information contained herein, the QPs, as authors of this Technical Report, offer the following interpretations and conclusions:

GEOLOGY

- The Gatsuurt deposit is located in the northeast-trending Mongol-Okhotsk belt in north-central Mongolia along the regional Sujigtei fault. Mineralization is hosted in early Paleozoic meta-sediments and early Paleozoic granodiorites, granites, and diorite that have intruded the meta-sedimentary rocks.

- Mineralization is structurally controlled along first, second, and possibly third-order faults that are generally steeply dipping and parallel to sub-parallel to the Sujigtei fault.

- Mineralization is concentrated in two distinct areas that are sinistrally separated by approximately 800 m along the Sujigtei fault.

- Mineralization occurs in: (a) quartz-sericite-pyrite-arsenopyrite-iron carbonate (siderite) veinlets; (b) quartz-native gold veinlets; and (c) black silica or pervasive quartz silica rock, in which disseminated sulphides and native gold make up one of the highest-grade components of the deposit.

- The geology of the Project area has shown to contain considerable amounts of gold mineralization in two laterally distinct zones; the geology and geological controls on mineralization are sufficiently well understood for resource modelling and estimation.

MINERAL RESOURCES

- The updated Mineral Resource Model described in this report considers 630 core boreholes (87,261 m) completed between 1998 and 2016 and is based on a new interpretation of geology and grade distribution and considers three nested grade domains with increasing cut-off grades of 0.4 g/t Au, 1.0 g/t Au, and 3.0 g/t Au.

- Estimation parameters largely follow those employed in the previous resource model; however, block sizes were reduced to 5 m x 5 m x 5 m to conform the block geometry to the thin and local variations found primarily within the high-grade domains.

- The current model considers high grade domains in the Central Zone in areas where none were considered previously. This change has led to a slight overall decrease in reported Mineral Resources. However, the QP considers the current model to be more robust than previous models.

MINING AND MINERAL RESERVES

- Although the QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate, the estimate should be considered within the context of the outcomes stated in this

Report. Of note, as outlined in Section 22, the Project's base case economic modelling results indicate a marginally positive NPV@5% of $39.5M. Should any of the key revenue drivers (price, grade, recovery) incur any adverse variance to the plan; this in turn places the above estimate at risk.

MINERAL PROCESSING

- Gold mineralization is categorized as oxide, transition, or sulphide.
- The metallurgical properties of this mineralization bear important implications for the design of a suitable processing facility to recover the gold and the economic significance of the Project.
- Gatsuurt ore will be treated at Centerra's Boroo Mine. Processing of the oxide ore will occur through the existing CIP facility at Boroo and the sulphide ore will be treated through a BIOX® process followed by a CIL facility which will be operational by Year 4.

ECONOMIC ANALYSIS

- The base case results indicate a marginally positive NPV at an annual discount rate of 5%. However, there is considerable down-side risk should any of the key revenue drivers (price, grade, recovery) present any adverse variance to plan.
- Analysis of the alternative scenarios suggests that the increase in production rate is unlikely to result in any significant reduction in unit costs. Nevertheless, the CIP Only case (mining and treating 6,000 t/d by CIP over a three year period) generates a higher IRR (14.5%) than the base case (9.3%) and slightly greater NPV at a discount rate of 8% or more, suggesting that this option may be worthy of further investigation.
- It is apparent that under base case conditions, the treatment of oxide ores in the existing process plant is profitable. However, it does not, by itself, generate sufficient cash flow to fund the development of the BIOX® circuit required to treat refractory ore from Gatsuurt, which therefore requires further investment during the first four years of the operating period.
- Once the BIOX® plant is in place, the Project begins to generate better returns (and on a simple, undiscounted basis pays back that investment within four years). However, the relatively short mine life then curtails operations before significant value is created.

RECOMMENDATIONS

Based on the information contained herein, the QPs, as authors of this Technical Report, offer the following recommendations.

GEOLOGY AND MINERAL RESOURCES

- The Project merits further exploration expenditures to improve overall Project economics.
- While the overall geology is well understood, additional investigations through oriented core drilling is warranted especially in the South Slope area of the Central Zone to better understand the structural controls on mineralization and overall geometry of the mineralized zone.
- Additional core drilling is warranted to test for a down-dip extension of mineralization in the Central Zone.
- Borehole spacing should be no less than currently available throughout the deposit area.
- Additional drilling should be considered in mineralized areas along strike that have not been included in the current Mineral Resource Model in order to include mineralization in those areas in future resource updates

GEOTECHNICAL ASPECTS

- The interpretation of faulting with the rhyolites is not well understood and will require further investigation by a qualified structural geologist.
- The weathering profile (in particular, weathering along joints), as well as existence of microfractures noted in the core, may influence rock mass stability at a bench and inter-ramp scale. Discontinuum numerical modelling should be considered once rock exposures have been mapped during early mining.
- The base of the high-weathered material should be determined from all available drill hole data and a wireframe developed for this surface during the detail design phase of this project.
- Structurally controlled failure could be sensitive to the presence of groundwater pressures. The most recent hydrogeological data should be assessed to see if it impacts any recommendations made in this report and a hydrogeological program should be initiated to include monitoring of water pressures near the pit slopes during early pit development. This program can then be used as a basis for assessing potential impacts of groundwater pressures on pit slopes and any need to provide dewatering measures for geotechnical purposes.
- Additional drilling should be completed on the waste dump footprints to confirm subsurface conditions assumed as part of this study.

MINING AND MINERAL RESERVES

- The material flagging process should be reviewed and updated once more accurate cost information becomes available and at any time there is a notable change in either costs or metal prices. This should be done with consideration of the potential effect on reported Mineral Reserves.

- Further refinement of the sulphur feed grade can be achieved at an operational level with appropriate grade control and stockpile management. Investigations into the practicality and effectiveness of further classification of the ore into sulphur graded stockpiles should be carried out as an optimization exercise.

- The non-acid generating (NAG) and potential acid generating (PAG) waste classification scheme should be further investigated and refined as part of any detailed engineering phase, after which the dump designs and schedule would need to be revised to ensure appropriate dump sizing, NAG cover, and timing issues are addressed.

- Once approvals for the dump locations are finalized, the dump designs should be modified such that their design capacity matches the scheduled movements to those dumps.

- Clarification should be sought on the status of the land on which the North Dump is to be situated.

- Further analysis of ore loss and dilution should be carried out once operational data becomes available.

MINERAL PROCESSING

- Conduct a thorough review prior to commencement of detailed engineering phase to assess optimum building and equipment layout for Flotation and BIOX® areas to identify cost reduction potential, some of which include:
  - Evaluate opportunity for reduction in size of BIOX® and Neutralization containment area walls to potentially reduce size and concrete requirements.
  - Review current BIOX® Utility Building size and equipment layout for optimal layout, maintenance, and potential reduction of overall building size.
  - Evaluate potential for Flotation Building height reduction when equipment details are available for the Limestone Mill, cyclones, and flotation cells.

INFRASTRUCTURE

- Due to the geographic proximity and potential for combining of shared utilities and services, it is recommended that the CIL and Cyanide Detox circuits be integrated into the BIOX® circuit and detailed engineering and construction be executed simultaneously.

- This would provide a more complete cost estimate and take advantage of synergies such as improved layout, reduced materials, and vendor packaging to lower cost, construction flexibility, and maximization of schedule. The execution schedule and construction strategy was prepared on this basis.

- Evaluate the option to use pre-engineered building structures for buildings. The layout is suitable to pre-engineered building. There may be an opportunity for cost reduction, improved construction efficiency and reduction in overall schedule.

OPERATING COST

- A contingency of 10% should be applied to the contract mining cost in any future economic modelling, although the inclusion of a 10% contingency does not impact the Mineral Reserve estimate.

2 INTRODUCTION

Centerra Gold Inc. (Centerra or the Company) has prepared a Technical Report on the Gatsuurt Gold Project (Gatsuurt or the Project or the Property) located in the Selenge Aimag (district) of Mongolia. The Technical Report supports the disclosure of the Project's Mineral Resource and Mineral Reserve estimates at October 31, 2017 and has been prepared in accordance with National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101). All dollar figures in this Technical Report refer to US dollars, unless otherwise noted.

Centerra is a gold mining company focused on operating, developing, exploring, and acquiring gold properties in North America, Asia, and other markets worldwide. Centerra is a leading Canadian-based gold producer and is one of the largest Western-based gold producers in Central Asia. Centerra's principal operations are the Kumtor Mine located in the Kyrgyz Republic and the Mount Milligan Mine located in British Columbia, Canada. The Company also has agreements to earn interests in joint venture exploration properties located in Canada, Mexico, Sweden, and Nicaragua. Centerra's shares are listed on the Toronto Stock Exchange under the trading symbol "CG". The Company is headquartered in Toronto, Ontario, Canada.

The Project is planned as a conventional truck and shovel open pit mine. A total of approximately 15.4 million tonnes of ore at a grade of 2.7 g/t Au, containing a total of approximately 1.3 million ounces of gold, is planned to be mined. Processing of the oxide ore will occur through an existing Carbon-in-Pulp (CIP) facility and the sulphide ore through the BIOX® process followed by a Carbon-in-Leach (CIL) facility over a total mine life of ten years from two open pits, the Central Zone pit and the smaller Main Zone pit.

SOURCES OF INFORMATION

This Technical Report was prepared by Centerra, T.R. Raponi Consulting Limited, Roscoe Postle Associates Inc. (RPA), RPMGlobal (RPM), Adiuvare Geology and Engineering Ltd. (AdiuvareGE), Micon-International (Micon), and Centerra Gold Mongolia LLC (CGM) personnel. The dates of personal inspections of the Project by the Qualified Persons are provided in Section 29 of this Technical Report.

The Qualified Persons and their responsibilities for this Technical Report are listed in Table 2-1.

This Technical Report is based on published material and data, professional opinions, and unpublished materials available to Centerra or prepared by its employees. In addition, certain information used to support this Technical Report was derived from previous technical reports on the Project and from reports and documents listed in Section 27 References. Other sources of data include geologic and block model reports, drill hole assay data, the block model, and mine plans, which were prepared by employees of Centerra.

TABLE 2-1 QUALIFIED PERSONS AND RESPONSIBILITIES

Qualified Person	Title/Company	Primary Areas of Responsibility	Report Sections Authored
Gordon D. Reid, P.Eng.	Vice-President and COO, Centerra	Overall responsibility.	All sections.
Boris Kotlyar, M.Sc., P.Geo., AIPG	Chief Geologist, Global Exploration, Centerra	Geology, exploration, drilling, sample preparation and analysis, data verification.	Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, and 21.
Tommaso Roberto Raponi, P.Eng.	President, T.R. Raponi Consulting Limited	Metallurgical testing and mineral processing.	Sections 13 and 17.
Lars Weiershäuser, Ph.D., P.Geo.	Associate Senior Geologist, Roscoe Postle Associates Inc. (RPA)	Mineral Resource estimate.	Section 14 and parts of sections 10, 11, and 12.
Igor Bojanic, FAusIMM, MIQA	General Manager, RPMGlobal (RPM)	Mineral Reserve estimate and mining methods.	Section 15 and parts of sections 16 and 21.
Chris Sharpe, P.Eng.	Director, Technical Services, Centerra	Geotechnical analysis.	Part of section 16.
William Pitman, M.Sc., P.Eng., ACSM	Director and Principal Geotechnical Engineer, Adiuvare Geology and Engineering Ltd.	Geotechnical slope design.	Part of section 16.
Kevin P.C.J. D'Souza, MEng, CEng, ARSM, FIMMM, FRGS	Vice-President, Security, Sustainability & Environment, Centerra	Environmental studies, permitting, and social and community impact.	Section 20.
Christopher Jacobs, CEng, MIMMM, MBA	Vice President & Mining Economist, Micon-International	Economic analysis.	Section 22.

Standard professional procedures have been followed in preparing the contents of this Technical Report. Data used in this Technical Report have been verified, where possible, and there is no reason to believe that the data was not collected in a professional manner.

UNITS

A list of abbreviations is provided in Table 2-2. The units of measurement used in this report conform to the metric system unless otherwise indicated. The currency used in this report is US dollars (US$) unless otherwise noted. An exchange rate of 2,200 MNT : US$1 was used to convert Mongolian Tugrik to US dollars.

Grades are in grams per tonne (g/t) for gold grades. Tonnages are metric tonnes of 2,204.6 lbs. Gold sales in units of troy ounces with a conversion of 31.1 grams per troy ounce. Within this text, “kt” means 1,000 metric tonnes and “Mt” means 1,000,000 metric tonnes.

TABLE 2-2 LIST OF ABBREVIATIONS

a	annum	LIMS	laboratory information management system
A	ampere	L/s	litres per second
AGP	acid-generating potential	m	metre
AISC	all-in sustaining cost	M	mega (million); molar
ANP	acid-neutralization potential	Ma	million years
ARD	acid rock drainage	m^2	square metre
ASM	Artisanal and Small-scale Mining	m^3	cubic metre
bbl	barrels	µ	micro
btu	British thermal units	MASL	metres above sea level
°C	degree Celsius	µg	microgram
C$	Canadian dollars	m^3/h	cubic metres per hour
cal	calorie	MDI	micro defects intensity
CCD	counter current decantation	MDS	micro defects strength
cfm	cubic feet per minute	ML	metal leachate
CIL	carbon-in-leach	mi	mile
CIP	carbon-in-pulp	min	minute
cm	centimetre	µm	Micrometre/micron
cm^2	square centimetre	mm	millimetre
CN_{WAD}	weak acid dissociable cyanide	mph	miles per hour
CCP	Conceptual Closure Plan	MAA	Mutual Assistance Agreement
CV	coefficient of variation	NGO	non-governmental organization
d	day	Mt/a	million tonnes per year
DEIA	Detailed Environmental Impact Assessment	MVA	megavolt-amperes
D-D	dipole-dipole	MW	megawatt
dia	diameter	MWh	megawatt-hour
dmt	dry metric tonne	NAG	non-acid generating

dwt	dead-weight ton
EIA	environmental impact assessment
EDM	electronic measuring device
EPC	Engineering, Procurement, and Construction
EPCM	Engineering, Procurement, and Construction Management
°F	degree Fahrenheit
FF	fracture frequency
ft	foot
ft^2	square foot
ft^3	cubic foot
ft/s	foot per second
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
Gpm	Imperial gallons per minute
GPS	global positioning system
g/t	gram per tonne
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
ha	hectare
hp	horsepower
hr	hour
Hz	hertz
ID^3	inverse distance cubed
ILO	International Labour Office
IP	induced polarization
in.	inch
in^2	square inch
IRR	internal rate of return
IS(50)	point load strength index
J	joule
k	kilo (thousand)
K (m/sec)	permeability
kcal	kilocalorie
kg	kilogram
km	kilometre
km^2	square kilometre
km/h	kilometre per hour
kPa	kilopascal
kV	kilovolt
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
lb	pound
NNP	net neutralization potential
NPV	net present value
OK	ordinary kriging
oz	Troy ounce (31.1035g)
oz/st, opt	ounce per short ton
PAG	potential acid generating
ppb	part per billion
ppm	part per million
pH	hydrogen ion concentration
psia	pound per square inch absolute
psig	pound per square inch gauge
QA/QC	quality control and quality assurance
RC	reverse circulation
RL	relative elevation
RMR	rock mass rating
RMS	Rapid Mineral Scan
RQD	rock quality designation
s	second
SAG	semi-autogenous
SI	strength index
SOX	sulphide oxidation rate
st	short ton
st/a	short ton per year
st/d	short ton per day
SRCE	Standardized Reclamation Cost Estimator
t	metric tonne
TCR	total core recovery
TIM	Trespass and Illegal Mining
TMF	tailings management facility
t/a	metric tonne per year
t/d	metric tonne per day
UCS	unconfined compressive strength
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
VPSHR	Voluntary Principles on Security & Human Rights
W	watt
WAD CN	weak acid dissociable cyanide
WHO	World Health Organization
WRL	waste rock landform
WI	weathering index
wmt	wet metric tonne
wt%	weight percent
XRD	X-ray Diffraction
yd^3	cubic yard
yr	year

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by Centerra. The information, conclusions, opinions, and estimates are based on:

- Information available at the time of this report, including Centerra's internal FEL-3 Study of the Project, and
- Assumptions, conditions, and qualifications as set forth in this report.

The authors have relied, and believe they have a reasonable basis to rely upon the following individuals who have contributed to the legal, political, environmental, and tax information stated in this report, as noted below:

- Indranila Ishbaljir, Legal Counsel, Centerra Gold Mongolia LLC (CGM) legal matters in Section 4.
- Lkhamsuren Baasandolgor, Manager, Environmental, CGM, with respect to environmental matters in Section 20.
- Rajeev Hampole, Director, Taxation, CGI with respect to taxation matters in Sections 4 and 22.

The date of these contributions is October 31, 2017.

The authors of this Technical Report have reviewed the information provided by the other experts as listed above and, based on the authors' review of this information, believe it to be reasonable and reliable.

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

4 PROPERTY DESCRIPTION AND LOCATION

The Project is located approximately 200 km south of Mongolia's northern border with the Russian Federation and 90 km north of Mongolia's capital city of Ulaanbaatar in the Mandal Soum (district) of Selenge Aimag (province), north-central Mongolia. The Property is located at approximately latitude 48º 30' N and longitude 106º 45' E. Tunkhel Bagh (village) is located 14 km to the east, and the nearest town is Zuunkharaa, the capital of Selenge Aimag, located approximately 34 km northwest of the Project with a population of 15,000 (2004). The Project is located 52 km by gravel road southeast of Centerra's Boroo Gold Mine and Mill (Figure 4-1).

The Project contains two major deposit areas, the Central Zone and the Main Zone. In UTM Zone 48N coordinates (WGS 84), the Central Zone area is centred at 616,600 East and 5,387,000 North, with the centre of the Main Zone lying less than 800 m southwest of the Central Zone.

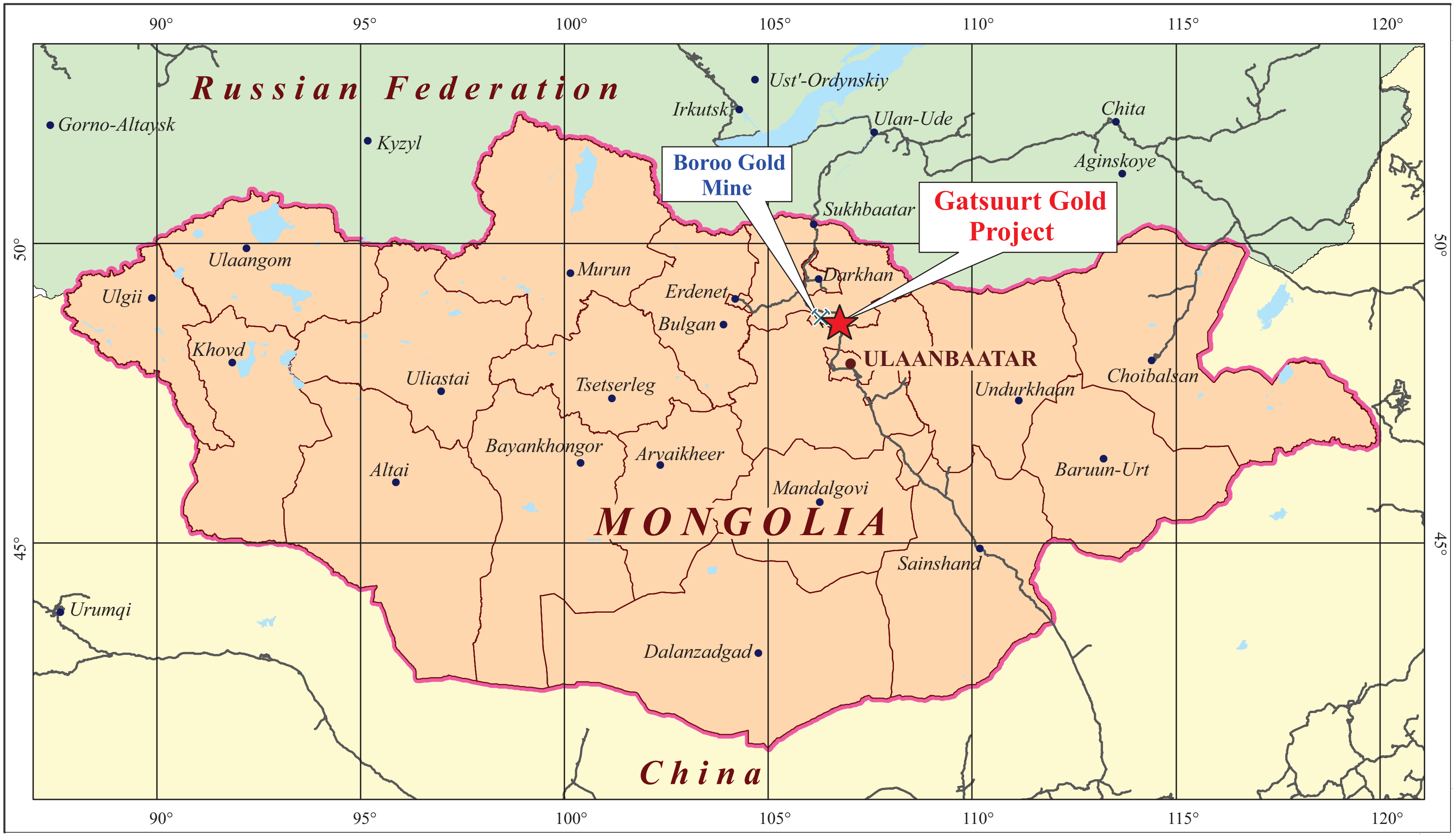

Figure 4-1

Legend:

- Capital city
- Province town
- Province boundary
- Railway
- Lake

0 250 500 750 1000
kilometres

centerraGOLD

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

centerraGOLD mongolia

Location Map

Date: December 2017

File: Location_map.cdr

LEGAL FRAMEWORK

The main legislation that governs the mining sector in Mongolia is the Constitution of Mongolia, the Subsoil Law, the Minerals Law, the Common Minerals Law, the Land Law, the Investment Law, the Environmental Protection Laws, Law on General Taxation, Corporate Income Tax Law, and the Law on National Security.

While the Constitution of Mongolia and the State Policy on Minerals lay out the foundation and general state policy and principle towards subsoil and mineral wealth, the Minerals Law governs reconnaissance, exploration, and mining of all types of minerals except water, petroleum, natural gas, radioactive minerals, and common minerals which are regulated by other specific laws.

The Minerals Law regulates, among other things, the ownership of minerals, classification of mineral deposits, requirements for minerals licence holders, state involvement and participation in minerals sector, requirements for taking back a licenced area for reserve, limitations and prohibitions of minerals prospect, exploration, and mining, regulations for licensing term, fee, exploration and mining activities and their requirements and the rights and obligations of licence holders.

The mining industry is regulated by the Ministry of Mining and Heavy Industry and the Mineral Resources and Petroleum Authority of Mongolia (MRPAM).

Under the Minerals Law, the holder of an exploration licence has rights to conduct exploration activities in the licence area, to construct temporary structures within the licence area related to its exploration activities, and if gaining access to its exploration licence area requires passing over land which is owned or possessed by others, to traverse such land subject to terms and conditions negotiated with such owners or possessors. If a mineral resource is identified by exploration activities, the exploration licence holder has the right to apply for a mining licence for any part of the exploration licence area. Pursuant to the Minerals Law, exploration licences granted on or after August 26, 2006 have an initial term of three years. The holder of such an exploration licence can apply for an extension of the licence for two successive additional periods of three years each. Annual fees payable per hectare escalate from US$0.10/ha in Year 1, US$0.20/ha in Year 2, US$0.30/ha in Year 3, US$1.00/ha in Years 4 to 6, to US$1.50/ha in Years 7 to 9. Work requirements escalate from

$Nil in Year 1, to US$0.50/ha in Years 2 and 3, US$1.00/ha in Years 4 to 6 to US$1.50/ha in Years 7 to 9.

Exploration licence holders are also subject to various environmental protection obligations. Within 30 days of receiving an exploration licence, the holder must prepare, and submit to the relevant authorities, an environmental protection and reclamation plan. Once the plan has been approved by the relevant authorities, the holder of the exploration licence must deposit funds equal to 50% of its environmental protection budget for that particular year in a bank account established by the governing authority of the soum in which the exploration licence area is located. Holders of exploration licences must also submit to relevant authorities an exploration plan and annual reports of exploration activities.

The Parliament of Mongolia passed Mineral Laws in July 2014, which introduced several changes and new provisions that have the aim of improving the existing legal framework relating to mining. Important provisions pertaining to exploration licences included allowing exploration licences (except those for radioactive minerals) to be extended to a total duration of 12 years and reducing the maximum area of a single exploration licence to 150,000 ha from 400,000 ha. In return for the additional three-year extension, a licence holder is subject to exploration expenditures of not less than US$10/ha in years 10-12 and a fee of US$5/ha of licence area in licence years 10-12.

If a commercially viable Mineral Resource is defined within the area of an exploration licence, the holder of the exploration licence is entitled to apply for a mining licence covering the relevant portion of the licence area defined by specific longitude and latitude coordinates in the mineral exploration licence. A mining licence holder has the right to conduct mining activities throughout the licence area and has the surface rights to construct structures within the licence area that are related to its mining activities. All such activities must be conducted in compliance with the 2006 Minerals Law and relevant Mongolian laws pertaining to health and safety, environment protection, and reclamation. Mining licences are granted by the MRPAM for an initial term of 30 years and are renewable for two successive periods of 20 years each based upon remaining reserves, for a maximum overall period of 70 years. Upon the expiration of a mining licence, the licence and the rights under such licence revert to the Government of Mongolia. In the case of all minerals other than coal and common construction minerals (e.g., sand and gravel), annual licence fees of US$15.0 are payable per hectare of the relevant mining licence area. A mining licence is subject to cancellation if applicable licence fees are not paid on time or other requirements under the 2006 Minerals Law or other relevant laws are not satisfied.

To receive a mining licence, an exploration licence holder must submit an application to the MRPAM together with, among other documents, an environmental impact assessment (EIA) and a resource report. Holders of mining licences must also prepare environmental protection and reclamation plans and satisfy various reporting and security deposit requirements. Obligations of a mining licence require submitting a feasibility study on the development of the deposit prepared by an accredited technical expert within one year of obtaining the mining licence; ensuring that feasibility studies include detailed information on the transportation of mining products, development of infrastructure, and funds required for mine restoration and closure work.

WATER FOREST LAW

The Gatsuurt Project is subject to the Mongolian Law on the Prohibition of Minerals Exploration in Water Basins and Forested Areas (Water Forest Law) which is more commonly referred to as the “Long Named Law”. The Water Forest Law was enacted in July 2009 and prohibits mining in Mongolian forest areas and near river basins; however, there is an exemption for mineral deposits of strategic importance.

In March 2010, Centerra received a letter from the Mineral Resources Authority of Mongolia stating that certain of its mining and exploration licences, including the Gatsuurt mining licences, could be revoked under the Water and Forest Law. The letter requested Centerra to submit an estimate of expenses incurred in relation to each licence and compensation it would expect to receive if such licences were revoked.

In November 2010, Centerra announced that mining operations were to cease at Boroo as of December 1, 2010, and 250 workers, previously scheduled to be deployed to Gatsuurt, were laid off. They were terminated because of the uncertainties of future operations at Gatsuurt as a result of the then enacted Water and Forest Law.

As of 2011, all the exploration licences in the Gatsuurt Gold Project area were dropped and four mining licences were retained.

From 2011 to 2014, all exploration activities at Gatsuurt were limited pending clarification of Gatsuurt's status with regard to the Water and Forest Law of Mongolia. During this period, the Gatsuurt Deposit was placed under care and maintenance.

In January 2015, the Project was designated as a mineral deposit of strategic importance by Mongolian Parliament. Such a designation entitled Mongolia, pursuant to the Minerals Law, to take a 34% ownership interest in the Project. As noted below under "Royalties" the Government of Mongolia and CGM have entered into a non-binding memorandum of understanding to exchange Mongolia's 34% interest in the Project for a 3% special royalty on the Project, though that arrangement is subject to the negotiation and execution of definitive agreements.

MINERAL TENURE

The mineral rights to the Project tenements are held under four mining licences covering 2,937 ha (Figure 4-2). The four mining licences were granted between July 1995 and November 2005 and may be maintained over the 70 year term of the licences by making annual payments of US$10/ha to the Mongolian Government.

Mining licences at the Boroo Mine, whose existing facilities and infrastructure will be used for processing ore from the Gatsuurt deposit, were granted between 1997 and 1999. In total seven Boroo licences are currently in good standing and expected to stay in the same status until at least 2037, provided that the economic rationales of keeping the licences remain positive.

The corner coordinates of the Gatsuurt licences are shown in Table 4-1.

TABLE 4-1 MINING LICENCE CORNER COORDINATES IN UTM ZONE 48N (WGS84)

Mining Licence No.	Hectares	Easting (UTM)	Northing (UTM)
MV-00372A	415.9	621,153.92	5,388,398.99
		621,169.22	5,387,688.54
		621,599.01	5,387,697.82
		621,610.82	5,387,142.05
		621,099.12	5,387,131.01
		621,088.48	5,387,625.02
		617,261.26	5,387,543.92
		617,270.91	5,387,080.78
		616,800.35	5,387,070.99
		616,820.02	5,386,113.53
		616,308.44	5,386,102.93
		616,282.91	5,387,338.27
		616,794.58	5,387,348.87
		616,784.96	5,387,812.32
		617,501.25	5,387,827.24
		617,490.92	5,388,321.25
MV-00431A	1,818.65M	623,194.34	5,389,648.05
		623,248.48	5,387,177.70
		621,610.82	5,387,142.05
		621,599.01	5,387,697.82
		621,169.22	5,387,688.54
		621,153.92	5,388,398.99
		617,490.92	5,388,321.25
		617,485.12	5,388,598.82
		616,257.37	5,388,573.30
		616,273.32	5,387,801.72
		616,784.96	5,387,812.32
		616,794.58	5,387,348.87
		616,282.91	5,387,338.27
		616,308.44	5,386,102.93
		616,820.02	5,386,113.53

Mining Licence No.	Hectares	Easting (UTM)	Northing (UTM)
		616,800.35	5,387,070.99
		617,270.91	5,387,080.78
		617,261.26	5,387,543.92
		621,088.48	5,387,625.02
		621,099.12	5,387,131.01
		617,721.86	5,387,059.31
		617,747.70	5,385,824.29
		616,621.71	5,385,800.84
		616,634.50	5,385,183.32
		614,587.00	5,385,141.26
		614,550.11	5,386,962.94
		615,675.87	5,386,985.98
		615,637.13	5,388,869.40
		616,557.70	5,388,888.41
		616,544.89	5,389,506.24
MV-005082A	359.77	616,537.45	5,388,887.99
		614,511.75	5,388,846.36
		614,482.18	5,390,297.84
		614,870.85	5,390,305.77
		614,862.65	5,390,706.85
		616,499.04	5,390,740.54
Mv-010810A	342.26	623,248.48	5,387,177.70
		623,985.07	5,386,267.48
		623,121.69	5,385,475.95
		621,871.16	5,386,468.10
		620,401.24	5,386,251.11
		620,382.73	5,387,115.63
Total	2,936.58		

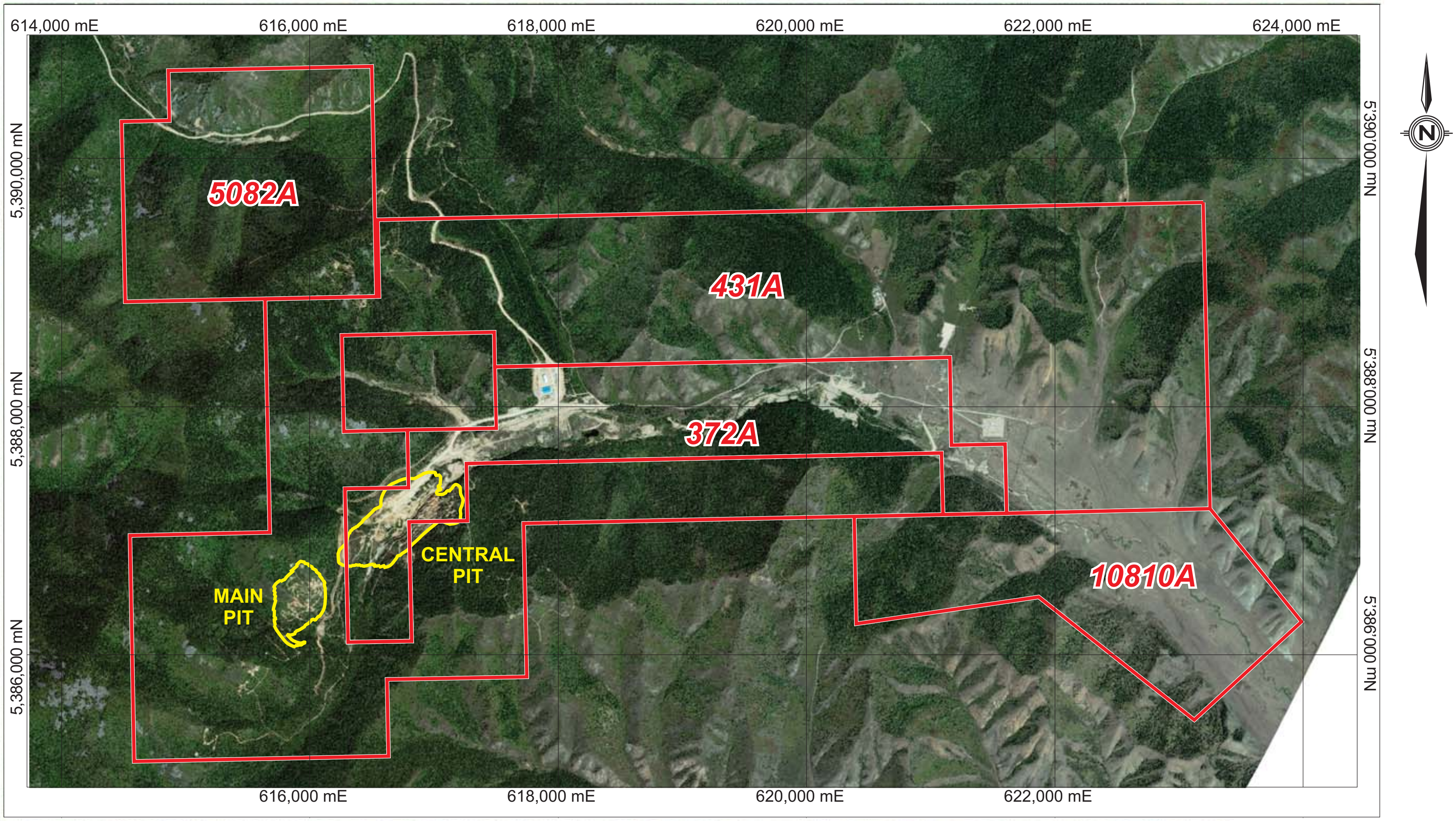

Figure 4-2

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
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Mining Licence Location Map

Date: December 2017
File: Mining_license_location.cdr

Legend:
Mining license

0 0.5 1 1.5 2
kilometres

ROYALTIES

The Property is subject to a sliding scale royalty payable to the Mongolian Government on gold sales pursuant to the Minerals Law, which starts at 5% and increases to a maximum of 10%, depending on the price per ounce of gold (maximum reached at a gold price of US$1,300 per ounce or above). However, for gold sales to Mongolbank the rate of royalty payable to the Mongolian Government is set at a flat rate of 2.5%. This 2.5% royalty for gold sales to Mongolbank is scheduled to expire in January 1, 2019 and there are no assurances that it will be continued. Licences MV-00372A and MV-00431A are encumbered by a 3% net smelter return (NSR) royalty in favour of Gatsuurt LLC, an arm's length Mongolian limited liability company.

As noted above, the Project was designated as a mineral deposit of strategic importance by the Mongolian Parliament in January 2015 which entitled Mongolia to take a 34% ownership interest in the Project. In October 2015, the Mongolian Government and CGM entered into a non-binding memorandum of understanding pursuant to which Mongolia would exchange its 34% ownership interest in the Gatsuurt Project for a 3% special royalty on the Project. On February 4, 2016, the Mongolian Parliament approved the level of Mongolian state ownership in the Project at 34% which allows the Government to substitute the 34% state ownership with a special royalty. The final ownership in the Gatsuurt Gold Project is subject to signing definitive agreements with the Mongolian authorities. CGM expects to enter into the following agreements (among others) prior to the commencement of Project development:

- A Deposit Development Agreement under the Minerals Law to set the level of the special royalty on the Project and certain operational parameters for the Project;
- An Investment Agreement under the Investment Law to stabilize the fiscal regime applicable to the Project; and
- A Community Development Agreement to set out certain community relations matters to be undertaken in connection with the Gatsuurt Project.

ENVIRONMENTAL LIABILITIES

The Project area is presently characterized by numerous shallow excavations, piles of alluvial sands, and trenches of various dimensions, all within the proposed mine area as a result of past placer-mining activities. Artisanal placer gold mining has occurred within the

Project area where it was concentrated within the Gatsuurt River Valley. This placer mining, well prior to Centerra's activities, has significantly altered the landscape, drainage, and water quality in the valley; it also has permanently damaged the top 5 m to 15 m of the land surface of the valley. Most soil, vegetation, aquatic and subaquatic ecosystems were disrupted and (or) permanently lost as a result of this alluvial placer mining. These losses to the ecosystem will restrict the overall extent of eventual reclamation that can be performed on the Property. Mine closure and reclamation planning by Centerra proposes to use all available soils and materials to reclaim the site.

Water quality within the Gatsuurt River Valley has also been impacted by previous mining activities carried out by other groups, with increased sedimentation of surface water and higher than normal concentrations of some heavy metals and other elements, including arsenic, following exposure to air, of previously buried and saturated soil and rock layers. Exposure of these rocks, which were previously buried during past mining activities, causes chemical reactions (oxidation of sulphide-bearing rock) that release naturally occurring metals into the environment through acid-rock drainage.

Examination of numerous drill holes at the Project has not identified the presence of any permafrost that could be affected by mining and associated mine development activities. It is not expected that any permafrost will be disturbed by these mining operations, however, should permafrost be identified in the future in the Project area, Centerra will consult with relevant authorities and the Mongolian Academy of Science to ensure that any adverse impacts to environmental, social, or cultural values are promptly appropriately mitigated.

PERMITS

CGM has secured many of the operating permits it needs to operate the Project in accordance with Mongolian regulatory requirements. Outstanding permits and licences include Hazardous Chemicals Permit, Gatsuurt Gold Mine Detailed EIA (submitted), Water Use Permit, and Mine Closure Plan. The status of current permits and approvals is summarized in Table 20-1.

KNOWN RISKS

To the QP's knowledge, except as set out in this Technical Report, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESS

Access to the Project is, in part, via the highway that connects Ulaanbaatar to Sükhbaatar (Suhbaatar) near the north border with Russia, which roughly takes one and a half hours. From the highway, there is a two lane all-weather road through the general countryside which leads to the site, approximately 10 km from the highway. The access road connecting the Gatsuurt site with the Boroo Mine is 52 km in length measured from the proposed ore stockpile pad at the Gatsuurt site to the run-of-mine (ROM) pad at the Boroo crusher located in the Boroo site (Figure 5-1). The road is of all-weather, earth and stone construction designed to accommodate 40 t tractor trailers.

CLIMATE

Climate in the Project area is mid-continental, temperate to cold. Average annual temperature is approximately 0ºC. Temperatures may drop to minus 40ºC during the months of December through February. Summer temperatures may exceed 40ºC, however, the average daytime summer temperature is approximately 20ºC. The area receives about 28 cm of precipitation per year, most of which is rainfall from July through August. Winters are relatively dry, with a moderate amount of snow that may persist on the ground from early November to April due to prevailing low temperatures. Although the climate may present some challenges to heavy equipment operation, the nearby Boroo Gold Mine and Mill have demonstrated the ability to operate effectively year round. Field exploration is most efficient during the months of March through November, although drilling operations can continue almost year round.

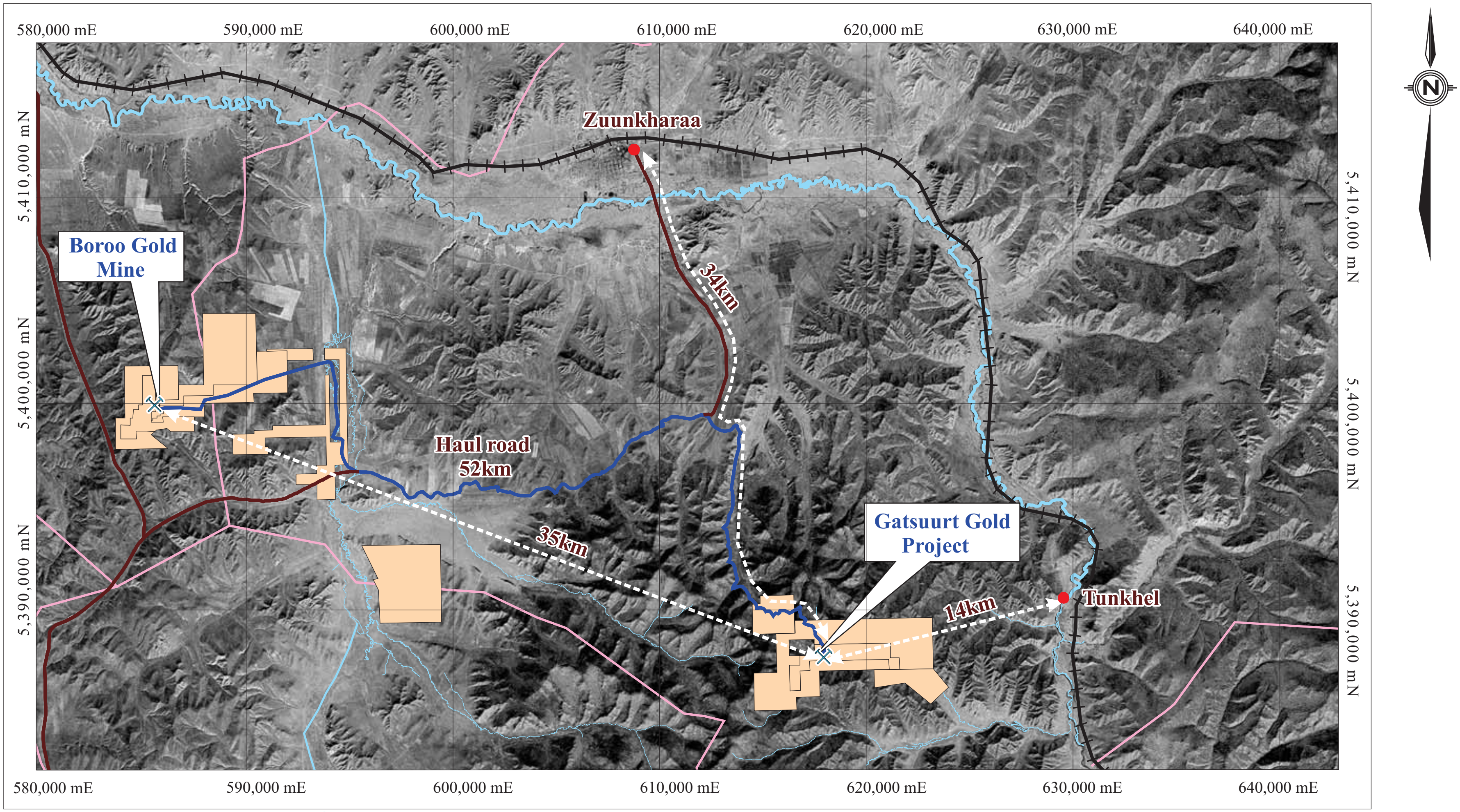

Figure 5-1

Legend:

- Village
- Paved road
- Haul road
- Railway
- River
- Soum boundary
- Mining license

0 3 6 9 12 15
kilometres

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
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centerragold
centerragold mongolia

Detailed Location and Access

Date: December 2017	File: Road_map.cdr

LOCAL RESOURCES

The Gatsuurt area is sparsely populated and is inhabited by mainly nomadic herdsmen living in small camps and villages. Labour and other support services are available in Ulaanbaatar, the capital city, with a population of over 1,300,000 inhabitants in 2013, roughly half of the country's entire populace. Ulaanbaatar is also Mongolia's central hub for road, rail, and international air services. The Zuunkharaa farming community, the largest settlement near Gatsuurt with approximately 14,000 inhabitants in 2004, lies approximately 30 km north of the Project area. The Project is expected to be staffed from Ulaanbaatar and the local region.

Mandal Soum (district) has the largest population of any soum in Mongolia at just over 25,000 people. The soum represents a prime agricultural area in Mongolia. In addition, 70% of the soum is covered by forested land, which supports forestry-based enterprises. In addition to the impact of past mining activities, the forest surrounding the Gatsuurt Valley has been, and continues to be, subjected to timber harvesting.

INFRASTRUCTURE

Project infrastructure is described in Section 18 of this report.

PHYSIOGRAPHY

The geography of Mongolia is characterized by a great diversity of topographic landforms. From north to south, the country is categorized by the presence of four major terrains: mountain forest steppe, mountain steppe, and in the extreme south, semi-desert, and desert. North-central Mongolia includes at least two major landscape types of upland steppe, termed the "Boroogol terrain" and "Dzuun Mod".

The Boroogol terrain is typified by gentle relief and moderately steep, rolling hills covered by grasslands with small discontinuous forests on north-facing slopes. The average elevation is 1,200 MASL. Streams are moderately or poorly developed and the higher order streams, in large part, are ephemeral. Loess (windblown silt) soil dominates the top of the soil profile.

The “Dzuun Mod” terrain in the Project area has rolling to steep mountains with moderately forested northern and eastern facing slopes generally devoid of outcrop. Forest cover generally consists of birch, pine, and larch species. The dryer south and west facing slopes are generally grass covered and, where very steep, 15% to 20% of the slopes have exposed outcrops. The average elevation is 1,300 MASL. Major river valleys are just under 1,000 MASL and the highest mountains are less than 2,500 MASL. Podzol soils dominate, and streams are moderately to well developed. Solifluction is common and evident as “contour” lines on grassy slopes in both the Dzuun Mod and Boroogol terrains.

The Gatsuurt Deposit lies in the Gatsuurt River Valley, which is occupied by recent alluvium flanked by older terraces. The alluvium is 10 m to 20 m thick, except where alluvium has been stripped by prior placer mining operations or buried under placer waste and tailings. Both recent alluvium and older terraces provided feed for placer sluices. Drainage from Gatsuurt follows the Gatsuurt River, the Sujigtei River, and the Kharaa River to eventually empty into Lake Baikal in the Russian Federation.

6 HISTORY

HISTORIC EXPLORATION ACTIVITIES

The following significant historical exploration and drilling highlights of the Gatsuurt Gold Project are modified from Hendry et al. (2006), and compiled from a number of CGM annual exploration reports, as well as Centerra press releases and internal documents.

Exploration in the Project area was carried out during various intervals from the 1970s to 2016 and evolved from an initial gold placer discovery and mining to the discovery of gold in bedrock and resource delineation at Gatsuurt.

Pan concentrates collected during a Mongolian government mapping program in 1970 identified gold in samples along a length of 1.5 km in the Gatsuurt Valley. Trenches and shallow pits were sampled, however, no further work was completed because exploration focused mainly on mercury.

In 1989, two cable tool placer drill hole fences detected low gold concentrations in the lower part of the Gatsuurt Valley, however, it was not until 1991 that detailed exploration in the Dzuun Mod district discovered the Gatsuurt placer deposit. The initial gold placer resource at that time was estimated at 2,668 mg/m^3, or 2,500 kg contained gold (80,000 ounces). This estimate is historical in nature and should not be relied upon. A qualified person has not completed sufficient work to classify the historical estimate as a current Mineral Resource or Mineral Reserve and Centerra is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

In 1995, Mining Licences 372A and 431A were issued, under the pre-1997 Mining Law, to the Mongolian company Gurvan Gol Co. to cover the Gatsuurt gold placer and the surrounding area. A Russian operator was hired to carry out placer mining from 1996 to 2000. Gurvan Gol Co. later became Gatsuurt LLC and continued operations until 2002, in the last years mainly reprocessing placer tailings. According to official and available reports, placer production by Gurvan Gol Co./Gatsuurt LLC from 1992 to 2000, totalled 4,400 kg gold (Centerra, 2009).

In 1996, Cascadia LLC, a Mongolian subsidiary of Cascadia Mining Inc. (together Cascadia), obtained three major exploration licences in Mongolia, including the Kharaagol Licence that covers a significant part of the Yeroogol gold trend, including the Gatsuurt placer area.

In August 1997, Cameco Gold Inc. (Cameco, later Centerra Gold Inc.) entered a subscription and earn-in agreement with Cascadia in which Cameco acquired an ownership interest in Cascadia by funding Cascadia's exploration programs from 1997 to 2001. By the spring of 1999, Cameco's share interest in Cascadia totalled 42%.

In 1997, quartz veins with visible gold were discovered in the placer bedrock floor and within boulders of altered granite, with disseminated arsenopyrite and pyrite noted in placer debris. This discovery was evidence that a significant gold-bearing system and alteration assemblage existed in bedrock under the placer deposit.

In 1998, Cascadia carried out exploration to identify the bedrock source of the alluvial gold deposit in the Gatsuurt area. This work included detailed mapping of the Gatsuurt placer floor and soil sampling. Strong gold and arsenic soil anomalies were detected in the South Slope of the Central Zone and over the Main Zone. Placer floor mapping identified the northeast extension of the Central Zone. Topographic and ground magnetic surveys were then completed. The Canadian geophysical contractor, Quantec Geoscience Inc. (Quantec), completed an induced polarization (IP) survey at Gatsuurt in September and October 1998. The survey located major chargeability and resistivity targets over the Central and Main Zones and the South Slope. Four core drill holes were completed to test the IP anomalies and the bedrock targets identified in the Central and Main Zones. Drill hole GT-06 intersected broad widths of gold mineralization, including 1.57 g/t Au over 110 m, and is considered to be the Project discovery hole.

Limited drilling programs were completed from 1999 to 2000 with 16 core drill holes totalling 2,138 m completed at the Central Zone, and eight core drill holes totalling 1,174 m completed at the Main Zone. During this time, preliminary metallurgical testing of drill samples determined that gold mineralization in both zones is largely refractory.

In 2001, Cameco acquired Cascadia LLC, which held the Project licences, in exchange for the surrender of Cameco's 42% share ownership interest in Cascadia and $2.5 million cash to Cascadia. Cascadia is now Centerra's wholly-owned subsidiary, CGM. At that time, the Project licences included 46 exploration licences. CGM was now the operator and, in late 2001, 25 drill holes were completed in the Central Zone and one hole in the Main Zone.

In 2002, more extensive exploration was carried out at Gatsuurt and other Project areas. Work consisted of topographic surveys, stream, soil, and rock grab sampling on grids and on reconnaissance traverses, along with IP (174.5 km) and ground magnetometer (9 km) geophysical surveys. The gradient IP followed up and extended the coverage at Gatsuurt of the chargeability IP survey previously completed by Quantec. An additional 52 core drill holes (NTW and BTW) for 4,814 m were completed in the Central Zone, on its extensions, and on the South Slope. Information was compiled and digitized, with data used for a preliminary resource estimate (Table 6-1).

TABLE 6-1 2002 IN-HOUSE RESOURCE ESTIMATE FOR GATSUURT CENTRAL ZONE

Cut-off Grade	Tonnes (Mt)	Grade (Au g/t)	Contained Gold (koz)
1.2 g/t Au	6.14	3.1	615
3.0 g/t Au	2.24	4.9	351

This estimate is historical in nature and should not be relied upon. A qualified person has not completed sufficient work to classify the historical estimate as a current Mineral Resource or Mineral Reserve and Centerra is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

PREVIOUS FEASIBILITY STUDY

In May 2005, Centerra retained SNC-Lavalin Engineers & Constructors Inc. (SNC-Lavalin) to complete a feasibility study on the Central Zone based on a stand-alone operating mine and processing facility on the Gatsuurt Mine site. As part of that study, SNC-Lavalin carried out field studies to acquire environmental, geotechnical, and metallurgical data, from which the design of the plant and mine infrastructure was completed. The SNC-Lavalin study indicated that the size of the Gatsuurt resource was insufficient to justify the cost of building a stand-alone processing facility and infrastructure for the Gatsuurt Gold Project. A combined development plan and feasibility study was then carried out by Centerra in order to identify the scope for improvement of Project economics by utilizing portions of the existing facilities at the Boroo Mine. The study was completed in December 2005. It was determined that the coordinated development of the Gatsuurt and Boroo properties would create opportunities to maximize the use of existing assets and reduce the Project capital costs. Under this approach, this combined development plan was used as the basis for advancing Project planning, based on a sequential development of the Central Zone and Main Zone deposits, with the ore to be transported and processed at a modified Boroo processing plant.

In March 2006, Centerra completed an updated feasibility study based on the estimates and factors for the combined Boroo/Gatsuurt operation. The updated study utilized the same development sequence for the Central Zone and Main Zone deposits, with the ore transported and processed at a modified Boroo processing plant.

In 2010, detailed engineering for the additional processing facilities at Boroo was initiated, however, in was suspended in May 2010 as a result of the uncertainties of future operations at Gatsuurt due to the then enacted Water Forest Law (see Section 4, Water Forest Law).

PREVIOUS MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Upon acquisition of the Project, Centerra completed geological and resource models for the Central and Main Zones. The Mineral Resource estimate was announced by Centerra in a press release of January 23, 2006 and is summarized in Table 6-2.

TABLE 6-2 MINERAL RESOURCE ESTIMATE - CENTERRA GOLD INC. (DECEMBER 31, 2005)

Zone	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Central Zone Indicated	11,543	3.5	1,314
Inferred	2,460	3.3	263
Main Zone Indicated	7,054	2.4	540
Inferred	1,520	2.4	116
Combined Totals Indicated	18,597	3.2	1,854
Inferred	3,980	3.0	378

Notes:
1. Cut-off grade of 1.6 g/t Au for both zones
2. Assays cut to 20 g/t Au maximum for Main Zone
3. Composites cut at various levels in the Central Zone depending on the grade-shell:
   - o 45 g/t Au for the 3 g/t Au grade shell
   - o 40 g/t Au for the 1 g/t Au grade shell
   - o 30 g/t Au for the 0.5 g/t Au grade shell
   - o 20 g/t Au for the Stockwork zone shell
4. Minimum composite length of 1.5 m

Later in 2006, RPA carried out an audit of the existing resource and reserve estimates at Gatsuurt and prepared an updated resource estimate for the Main Zone. RPA was of the opinion that the Gatsuurt Gold Project represents a valuable asset to Centerra and that the estimates of Mineral Resources and Mineral Reserves met the requirements of NI 43-101 and the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (Hendry et al., 2006). The estimates, with an effective date of December 31, 2005, superseded the estimates previously announced by Centerra in the January 23, 2006 press release.

TABLE 6-3 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATE - CENTERRA GOLD INC. AND ROSCOE POSTLE ASSOCIATES INC. (DECEMBER 31, 2005)

	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable Reserves			
Central Zone			
Oxide (0.93 g/t Au Cut-off)	1,818	2.3	133
Sulphide (2.00 g/t Au Cut-off)	5,133	4.1	681

	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Main Zone			
Sulphide (2.00 g/t Au Cut-off)	2,008	2.7	172
Total Probable Reserves	8,959	3.4	986
Indicated Resources			
Central Zone			
Oxide (0.93 g/t Au Cut-off)	216	2.0	14
Sulphide (2.00 g/t Au Cut-off)	2,738	3.8	332
Main Zone			
Sulphide (2.00 g/t Au Cut-off)	2,630	2.6	219
Total Indicated Resources	5,584	3.1	565
Inferred Resources			
Central Zone			
Oxide (0.93 g/t Au Cut-off)	119	2.9	11
Sulphide (2.00 g/t Au Cut-off)	1,643	4.0	211
Main Zone			
Sulphide (2.00 g/t Au Cut-off)	987	2.6	83
Total Inferred Resources	2,749	3.5	305

Notes:

1. Indicated Mineral Resources are in addition to Mineral Reserves
2. Assays cut to 20 g/t Au maximum for Main Zone
3. Composites cut at various levels in the Central Zone depending on the grade-shell:
    - 45 g/t Au for the 3 g/t Au grade shell
    - 40 g/t Au for the 1 g/t Au grade shell
    - 30 g/t Au for the 0.5 g/t Au grade shell
    - 20 g/t Au for the Stockwork zone shell
4. Minimum core length of 1.5 m
5. The Mineral Reserves have been estimated based on a gold price of US$400 per ounce

The December 31, 2005 Mineral Resource and Mineral Reserve estimates were updated in 2006 to reflect a higher gold price of US$475 per ounce (Table 6-4).

TABLE 6-4 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATE, AS OF DECEMBER 31, 2006

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable	9,101	3.4	1,005
Indicated	6,238	3.0	607
Inferred	2,437	3.3	256

Notes:

1. CIM (2004) definitions were followed for Mineral Resources.
2. Indicated Mineral Resources are in addition to Mineral Reserves
3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability
4. Mineral Resources occur outside the ultimate pits which have been designed using a gold price of US$475 per ounce.
5. Mineral Resources were estimated using a 1.2 g/t Au or 1.4 g/t Au cut-off grade depending on the type of material and the associated recovery.
6. The Mineral Reserves have been estimated on a gold price of US$475 per ounce.
7. Numbers may not add due to rounding.

The Mineral Resource and Mineral Reserve estimates were updated as of December 31, 2009 due to an expanded pit as a result of the higher gold price of US$825 per ounce and a resulting lowering of the cut-off grade (Table 6-5).

TABLE 6-5 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATE, AS OF DECEMBER 31, 2009

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable	13,850	2.9	1,280
Indicated	5,751	2.6	480
Inferred	2,260	2.4	177

Notes:

1. CIM (2004) definitions were followed for Mineral Resources.
2. Indicated Mineral Resources are in addition to Mineral Reserves
3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability
4. Mineral Resources occur outside the ultimate pits which have been designed using a gold price of US$825 per ounce.
5. Mineral Resources were estimated using a 1.2 g/t Au or 1.4 g/t Au, or 1.5 g/t Au cut-off grade depending on ore type and processing method.
6. The Mineral Reserves have been estimated on a gold price of US$825 per ounce.
7. Numbers may not add due to rounding.

The Mineral Resource and Mineral Reserve estimates were updated as of December 31, 2010 due to the successful exploration drilling in the South Slope area which added oxide, transition, and sulphide material (Table 6-6).

TABLE 6-6 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATE, AS OF DECEMBER 31, 2010

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable	16,349	2.8	1,489
Indicated	5,533	2.4	426
Inferred	5,926	2.6	491

Notes:
1. CIM (2010) definitions were followed for Mineral Resources.
2. Indicated Mineral Resources are in addition to Mineral Reserves
3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability
4. Mineral Resources occur outside the ultimate pits which have been designed using a gold price of US$1,000 per ounce.
5. Mineral Resources were estimated using a 1.2 g/t Au or 1.4 g/t Au, or 1.5 g/t Au cut-off grade depending on ore type and processing method.
6. The Mineral Reserves have been estimated on a gold price of US$1,000 per ounce.
7. Numbers may not add due to rounding.

The Mineral Resource and Mineral Reserve estimates were updated as of December 31, 2013 as a result of an updated block model and en expanded pit design (Table 6-7).

TABLE 6-7 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATE, AS OF DECEMBER 31, 2013

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable	17,129	2.9	1,603
Indicated	5,098	2.4	398
Inferred	5,475	2.5	440

Notes:
1. CIM (2010) definitions were followed for Mineral Resources.
2. Indicated Mineral Resources are in addition to Mineral Reserves
3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability
4. Mineral Resources occur outside the ultimate pits which have been designed using a gold price of US$1,300 per ounce.
5. Mineral Resources were estimated using a 1.4 g/t Au cut-off grade.
6. The Mineral Reserves have been estimated on a gold price of US$1,300 per ounce.
7. Numbers may not add due to rounding.

Centerra is not treating the previous estimates as current Mineral Resources or Mineral Reserves and these have been superseded by the Mineral Resource and Mineral Reserve estimates in Sections 14 and 15 of this report.

PAST PRODUCTION

To date, alluvial mining has been conducted on the Property, and according to official and available reports, alluvial production by a previous owner from 1992 to 2000 totalled 4,400 kg gold (Centerra, 2009).

No hardrock mining has been carried out on the Property.

7 GEOLOGICAL SETTING AND MINERALIZATION

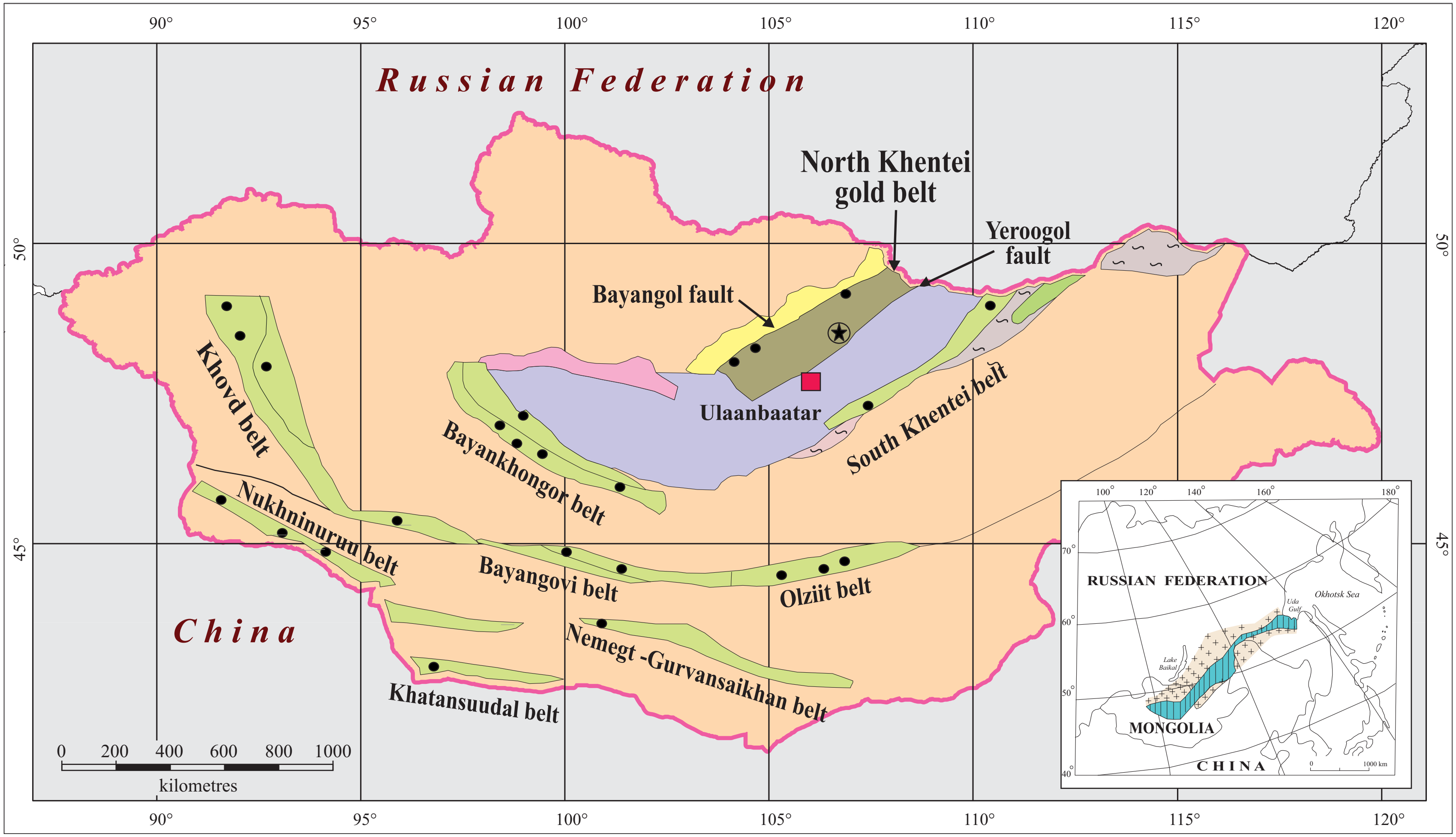
This section is largely taken from Theodore and Kotlyar (2007), Tosdal (2010), Starling (2006, 2010a, 2010b), Kotlyar et al. (1999), Kotlyar et al. (2002), Cluer et al. (2003, 2004), Chapman (2004), Cluer et al. (2005), Hendry et al. (2006), Khishgee et al. (2014), and a number of Centerra internal reports.

REGIONAL GEOLOGY

The Gatsuurt Deposit is located within the northeast-trending Mongol-Okhotsk belt of north-central Mongolia, which is surrounded by areas dominated by magmatic rocks associated with the overall evolution of the belt (Figure 7-1). As currently defined, a number of geologic terranes make up the Mongol-Okhotsk belt including:

1. the early Paleozoic Kharaa terrane (comprising back-arc to fore-arc basinal rocks that delineate the North Khentei gold belt),
2. Devonian to Carboniferous turbidites,
3. several accretionary metallogenic belts on the southeastern and southwestern flanks of the Mongol-Okhotsk belt,
4. an island arc terrane and accretionary wedges, and
5. a narrow east-west trending block of cratonic rock.

Figure 7-1 is a simplified map showing the tectonic setting of Mongolia, and the location of the Gatsuurt Deposit (circled star) and gold-quartz vein deposits.



Legend:

- Mongol - Okhotsk belt (Middle Paleozoic Early Mesozoic folded marine sequences)
- Area of magmatism related to evolution of the Mongol - Okhotsk fold belt
- Kharaa terrane backarc/forearc basin, contains North Khentei gold belt (NKGB)
- Devonian - Carboniferous turbidite basin
- Major accretionary metallogenic belts
- Islands arc
- Accretionary wedge
- Cratonic block
- Gatsuurt gold deposit
- Au-quartz vein deposits and occurrences

Figure 7-1

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Gatsuurt Gold Project, Mongolia
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North Khentei Tectonic Map

Date: December 2017	File: Tectonic_map.cdr

The Project mining licences, including the Gatsuurt Deposit, are located in the North Khentei gold belt, as is Centerra Gold's Boroo Deposit (Kotlyar et al., 1999; Cluer et al., 2005), which is located approximately 35 km northwest of Gatsuurt. The northeast trending North Khentei gold belt is bounded by two continental-scale fault systems with an overall sinistral sense of movement: the Bayangol fault on the northwest and the Yeroogol fault on the southeast (Figure 7-2). The Bayangol fault separates the North Khentei belt from the Taryato-Selenge Terrane, a stable craton overlain by Paleozoic carbonate platform rocks to the northwest. The Yeroogol fault borders the Kharaa terrane along its southeastern margin and the Khentei Terrane to the southeast. In addition, a number of linear accretionary metallogenic belts are present in the southern part of the country and include several gold-quartz vein deposits and occurrences.

The North Khentei tectonic belt includes three predominant lithotectonic components.

- Late Precambrian-Early Paleozoic flysch and subsequent plutonism. Miogeosynclinal flysch includes the Precambrian Yeroo Series greenschist grade metamorphic rocks adjacent to the northwest of the Yeroogol fault and the Lower Paleozoic Kharaa Series sandstone, shale, siltstone, conglomerate, phyllite, quartz-sericite and sericite-chlorite schist, and some compositionally intermediate volcanic tuff. The Early Paleozoic Boroogol Complex (520±20 Ma to 410±30 Ma; Kampe and Gotsman, 1966) biotite and biotite-hornblende granodiorite and granite have intruded the Yeroo and Kharaa Series east of the Bayangol fault zone. These intrusive rocks have been offset at Gatsuurt by protracted lateral separations along faults such as the Sujigtei fault, which represents a major splay of the Yeroogol fault system.

- Permian continental volcanic and sedimentary rocks lie unconformably on the Yeroo, Kharaa, and Boroo rocks and crop out in two small areas proximal to the Yeroogol fault system, where they have been offset by subsequent displacements along the fault. Volcanic rocks include subvolcanic phyric rhyolite, tuffaceous andesite lava, and breccia. Sedimentary rocks include shale, sandstone, and conglomerate.

- Jurassic-Cretaceous and Tertiary coal-bearing sedimentary rock and conglomerate represent the youngest assemblages in the region.

Over the years, much of the mining occurred along the traces of these northeast-striking terrane bounding faults. Major placers and placer districts include those at Bukhlei, Sharingol, Tolgoit, Ikh Alt, Yalbag, and Ikh Tashir in the North Khentei gold belt. Also, some lode gold occurrences in the belt have been active for many decades, while other lode occurrences remain as undeveloped, yet significant, gold prospects (i.e., Ulaanbulag; Baavgait, Ereen, Sujigtei, Tsagaanchuluut, and Urt). Figure 9-1 in Section 9 of this report shows the location of these occurrences along the Sujigtei fault.

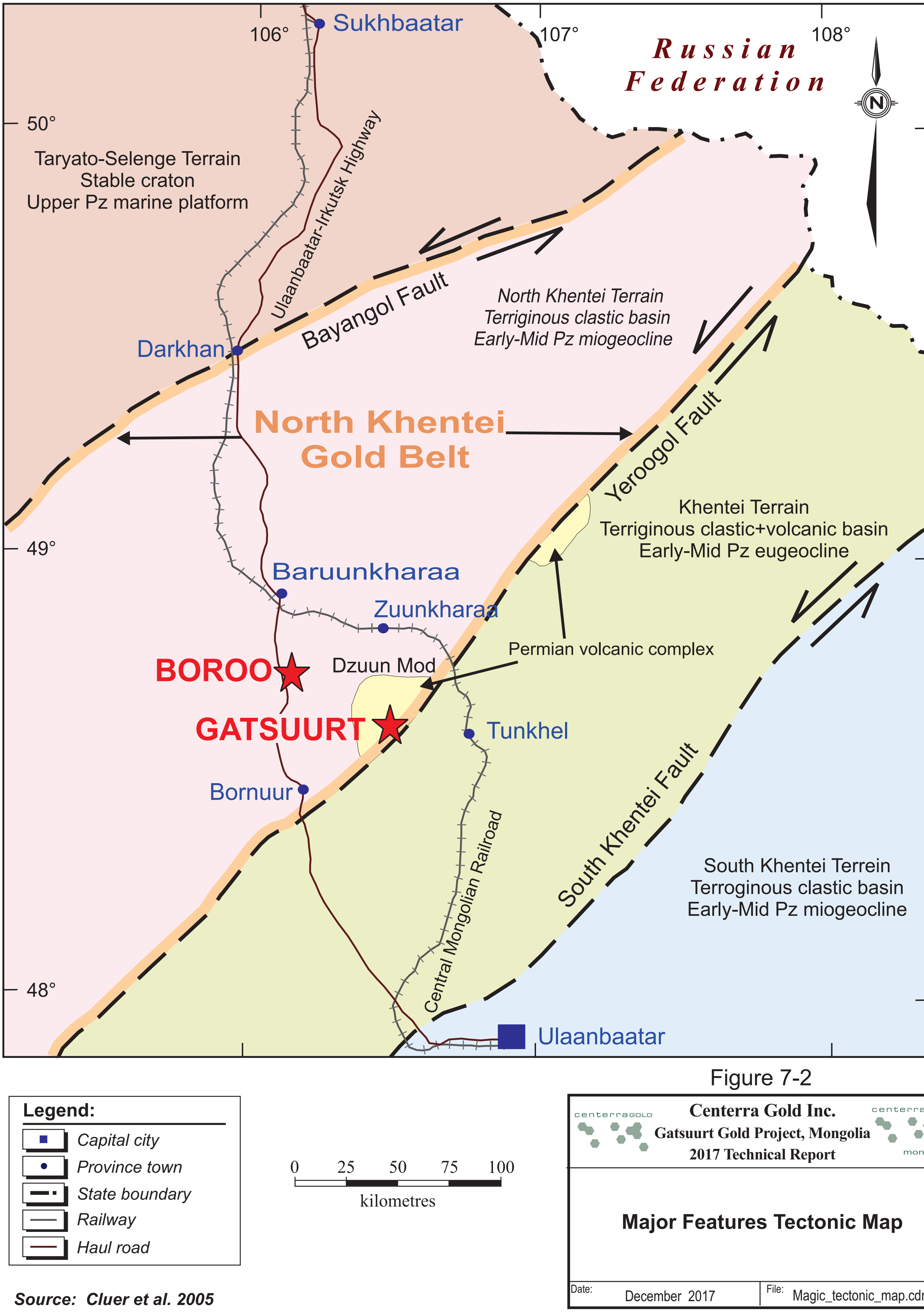

Figure 7-2

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Major Features Tectonic Map

Date: December 2017 | File: Magic_tectonic_map.cdr

LOCAL AND PROPERTY GEOLOGY

In the Central Zone of the Gatsuurt Deposit, located southeast of the Sujigtei fault, rocks are mostly early Paleozoic in age and are the oldest present in the general area of the deposit. Early Paleozoic Kharaa Series meta-sedimentary rock (generally meta-sandstone, some calcareous) has been intruded by early Paleozoic Boroogol Complex granodiorite, granite, and diorite. Rhyolite dykes and sills of low abundance are also present.

Rocks in the northwest block of the Sujigtei fault are Permian Dzuun Mod massive quartz-K feldspar phyric rhyolite, which regionally once formed part of a semi-circular body but was later wrenched apart in a left-lateral sense by displacements along the Sujigtei fault. The phyric rhyolite is interpreted by Tosdal (2010) and other internal Centerra authors as largely representing a caldera complex formed at approximately 280 Ma, though extensive presence of welded tuff in the caldera has yet to be described.

The Kharaa meta-sedimentary rocks, Boroogol Complex intrusions, and Dzuun Mod phyric rhyolite outcrop along the Sujigtei fault that marks a major structural boundary between phyric rhyolite and the latter two packages of rock. Intrusive rock of the Boroogol Complex also contains fault-bounded “xenoliths” of Kharaa Series meta-sedimentary rock and hosts volumetrically minor diorite dykes and xenoliths of other Boroogol Complex rocks. Additionally, near the Central Zone at Gatsuurt, abundant blocks of Kharaa hornfels are enveloped by granite suggesting that exposed outcrops may represent border phases of a pluton, and probably even that part of the intrusion near or at the roof of the pluton.

Overall, Jurassic sinistral displacement along the Sujigtei fault is evidenced by separation of a portion of the Permian Dzuun Mod phyric rhyolite in the east block of the fault approximately 60 km to the northeast (Cluer et al., 2005). Because of its scale of displacement, the Sujigtei fault unquestionably represents a major crustal feature. As such, a broad spectrum of various types of fault rock formed along it, ranging from early high temperature mylonitic and blastomylonitic facies rocks near the trace of the Sujigtei fault. In the Central and Main Zones much of the fault is marked by a healed contact without any fault gouge. However, farther to the southwest along the Sujigtei fault in the general area of the 49-er, Ganaa, and Sujigtei prospects, low temperature, clay-gouge cataclastic facies rocks are present. Based on the 800 m separation of the Central and Main Zones at Gatsuurt, which is much less than the overall 60 km separation recorded by the phyric rhyolite caldera, mineralization at Gatsuurt must have occurred after the bulk of the displacement had occurred along the Sujigtei fault and its nearby fault strands.

As reported by Tosdal (2010), no solid constraints can be established for vertical movement along the Sujigtei fault to accompany its approximately 800 m of horizontal offset between the Central and Main Zones. Nonetheless, some data are compatible with minor vertical offset also being present. Two other prospects, Arangat and Sujigtei, lie to the southwest of the Central and Main Zones and are similarly separated by the Sujigtei fault by approximately 1,200 m.

A considerable fraction of the mylonitic rocks in the Gatsuurt district is also related to displacements along the Sujigtei fault. Yet, much of the mineralized rock in both the Central Zone and Main Zone lacks a shear fabric and instead contain of a dense concentration of stockwork veinlets. It is important to note that mineralized mylonitic fault rocks that are present in the two zones are syn-mineral, as they host arsenopyrite-pyrite-gold-white mica (sericite)-carbonate (siderite) assemblages along shear planes. Additionally, a large displacement shear along the Sujigtei fault also appears to be responsible for development of elongate packages of rocks oriented at slightly oblique angles to the main trace of the fault in the South Slope area near the northeast end of the Central Zone.

Figure 7-3 is a geologic map of the Gatsuurt area.

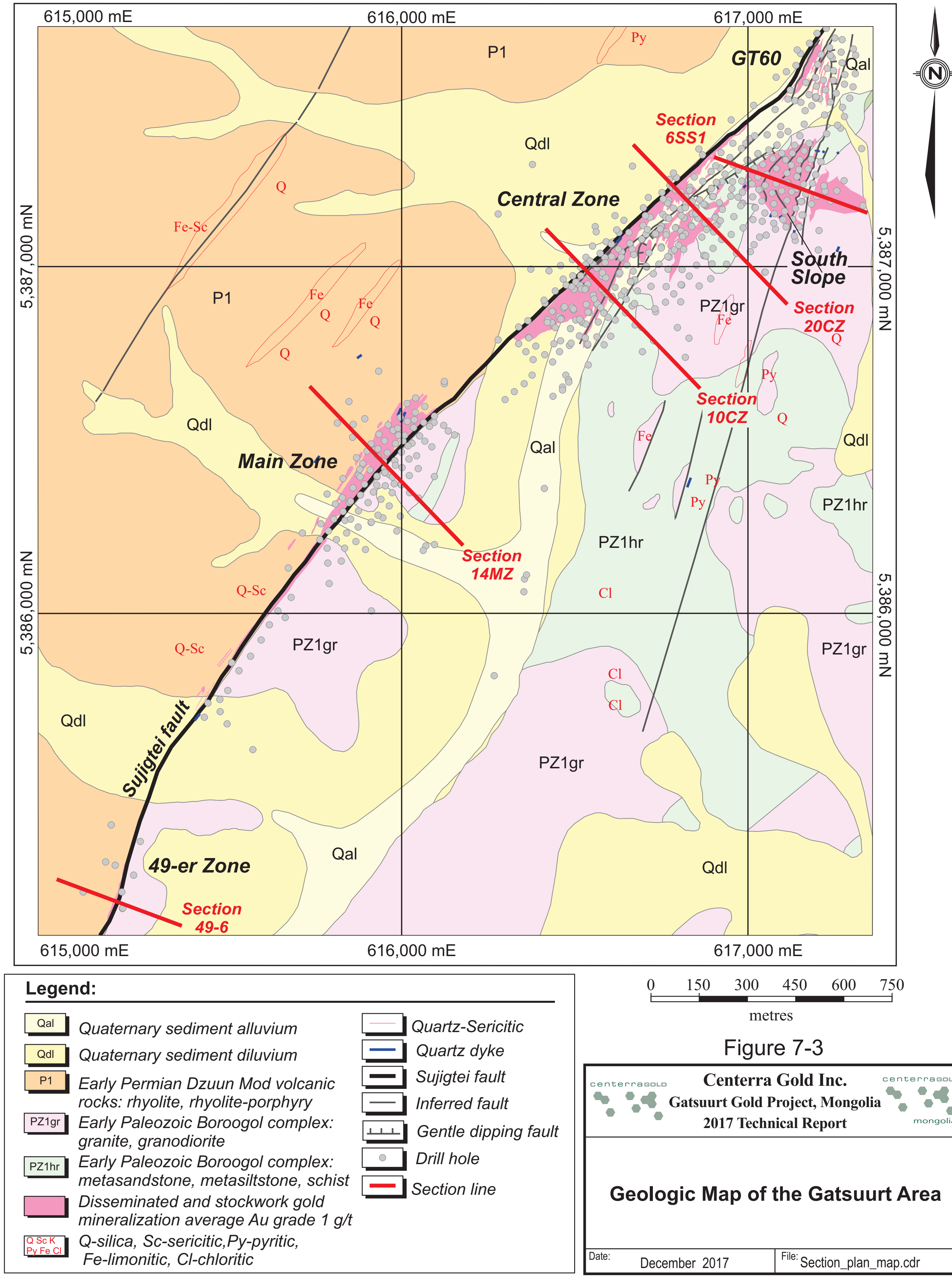

Figure 7-3

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Gatsuurt Gold Project, Mongolia
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Geologic Map of the Gatsuurt Area

Date: December 2017 | File: Section_plan_map.cdr

STRUCTURE

Structural elements control the distribution of gold mineralization and prominent alteration zones at the Gatsuurt Deposit.

The Sujigtei fault is a north-easterly striking (N 40° E), high-angle fault system that has been traced over 200 km along strike. The two major mineralized zones of the Gatsuurt Deposit, Central Zone and Main Zone, are offset from each other by displacements along the Sujigtei fault. In addition to the Sujigtei fault, the Gatsuurt Deposit is controlled by a number of parallel, higher order, steeply dipping splay faults that generally strike N 20° E. Further complicating are gently dipping, pre-mineralization faults.

As described earlier, the Sujigtei fault marks a major structural boundary between the Dzuun Mod phyric rhyolite and the Boroogol Complex intrusions and Kharaa meta-sedimentary rocks. Phyric rhyolite crops out on the northwest side of the Sujigtei fault, whereas intrusive rock belonging to the Boroogol Complex is present on the southeast side, where it intrudes the Kharaa Series (Figure 7-4). The Sujigtei fault is marked by a narrow, healed zone of mineralized black silica to the north of the southernmost Main Zone, whereas south of the Main Zone, the fault trace is marked by cataclastically formed fault gouge. The Main and Central Zones appear to abut against an earlier active fault segment, whereas the Sujigtei and Arangat prospects to the south lie on a later subparallel but close-by en-echelon fault. The 800 m of offset on the Sujigtei fault in the Main-Central segment represents an accommodation of approximately two thirds of the 1,200 m of offset recorded along the Arangat-Sujigtei segment. The remaining 400 m can be accommodated on various N 20° E striking splays that cut through the Central Zone in the South Slope area on the east side of the Sujigtei fault and are important mineralizing conduits to high-grade mineralization. In this scenario, the N 20° E faults would largely represent reactivated lithologic contacts or faults that bound north-northeast–trending packages of rock on the east side of the Sujigtei fault.

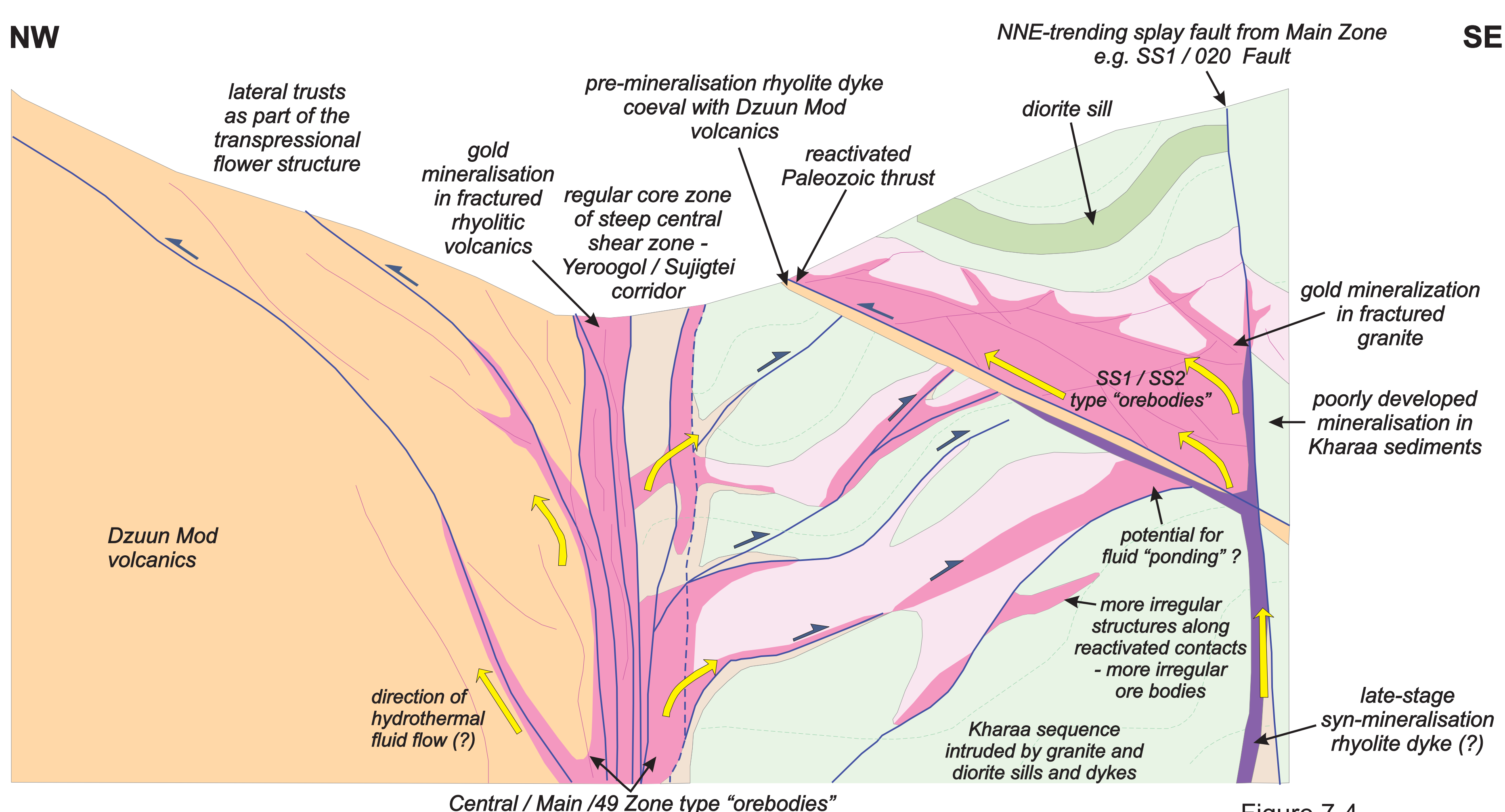

Figure 7-4

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Composite Schematic Cross-Section of the Central and Main Zones of the Gatsuurt System

Date: December 2017	File: Schematic_Cross_Section.cdr

Modified from Starling (2010a)

Three important structural domains that are important from a modelling perspective can be described as follows:

- As the step-over is dilational, the N 20° E faults have both normal and strike-slip components, and hence are normal oblique-slip faults (Tosdal, 2010). The net effect of oblique slip on these mineralized faults is to displace shallowly dipping contacts, leading to complex morphologies of mineralized rock at Gatsuurt.

- As mineralization occurred under a phase of broadly north-south to north-northeast regional compression, a number of steep N 20° E structures also developed in the South Slope area as tensional or Riedel shears due to sinistral transpressional motion along the main Yeroogol fault zone (Starling, 2010a). The overall extensional nature of these N 20° E structures allowed hydrothermal fluids associated with mineralization to migrate preferentially into these second order structures from the main northeast-striking shear zone and faults.

- In addition, shallow southeast dipping faults are present in the South Slope area of the Central Zone, and they probably represent reactivated Paleozoic thrust surfaces along which Permian Dzuun Mod rhyolite dykes and much later Jurassic mineralization also occurred (Starling, 2010a). These second order structures off of the main northeast-striking Sujigtei shear zone acted as feeder structures to fractured granite and diorite in the Central Zone.

ALTERATION

The various kinds of alteration recognized at the Gatsuurt Deposit include silica, potassic, sericitic, and chloritic alteration (Figure 7-5).

Deposit-scale alteration patterns in the Central Zone are extremely complicated and include a complex interfingering of silica, potassic, and chloritic alteration. Potassic alteration (a) coincides fairly closely with distribution of greater than 1 ppm Au in the partly oxidized South Slope area, and (b) is largely coincident with silica plus sericitic alteration where the southern part of Central Zone is aligned north-easterly along the Sujigtei fault. Pervasive quartz silica alteration including “black” silica alteration, or intense flooding by introduced veinlets of quartz, is fairly well constrained to the trace of the Sujigtei fault through much of the Central Zone. Chloritic alteration mineral assemblages are present only slightly on the periphery of sericitic alteration in the footwall block of the Sujigtei fault where Lower Paleozoic granitoids and metasedimentary rock crop out. Many chloritic mineral assemblages also include purple fluorite. Much of the chloritic-altered rocks in the deposit interfinger with areas of all other types of alteration. In the GT60 area, distribution of all four types of alteration is controlled by the prominent N 20° E striking faults.

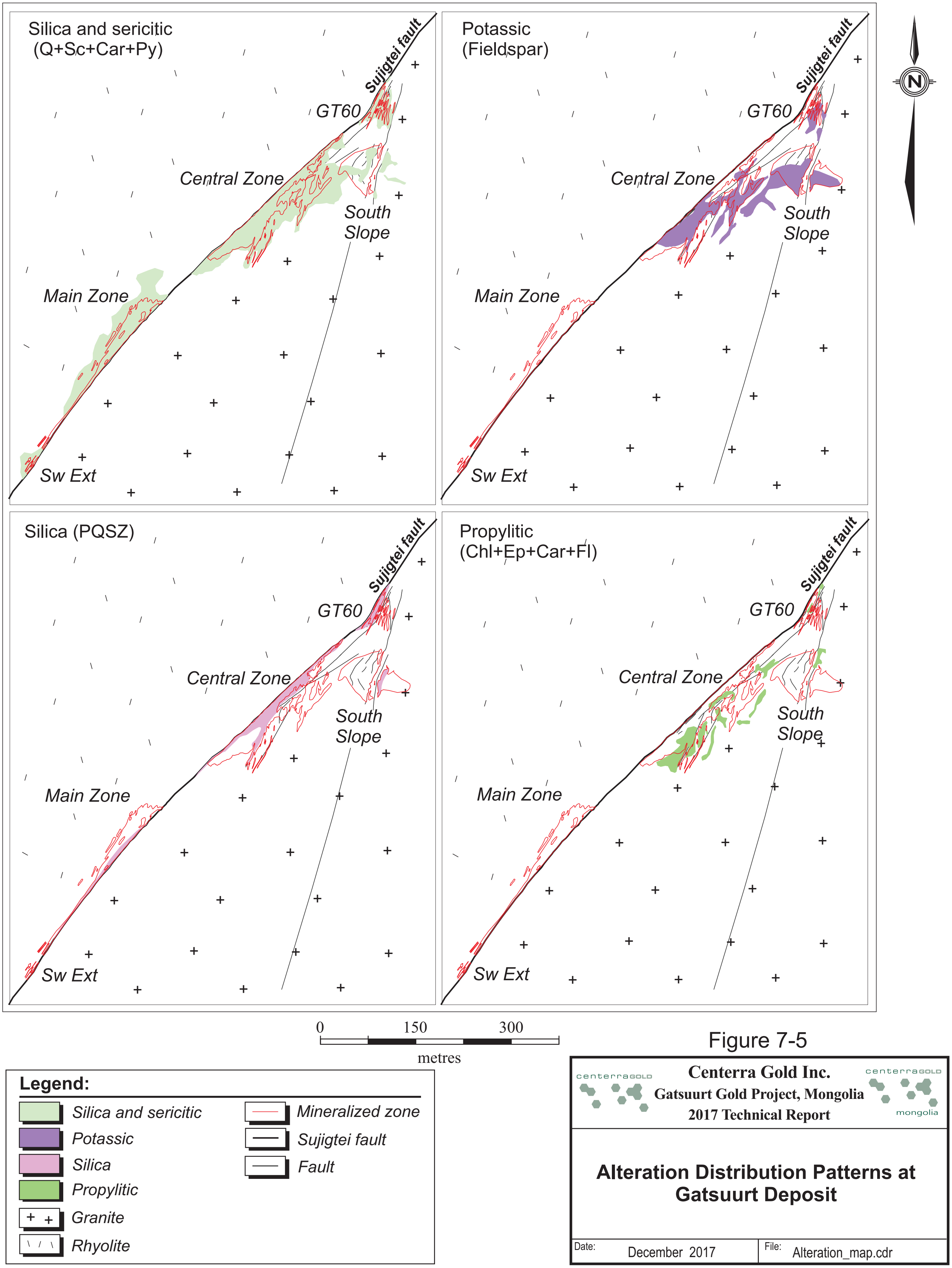

Figure 7-5

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Alteration Distribution Patterns at Gatsuurt Deposit

Date: December 2017 File: Alteration_map.cdr

The dominant type of alteration in the Main Zone is quartz sericitic, including carbonate minerals together with sulphide minerals (mainly arsenopyrite), as well as a narrow band of “black” silica alteration along the Sujigtei fault. The extent of potassic alteration in the Main Zone is limited, and is recognized in only one small area outward of sericitic alteration, a relationship similar to much of its distribution in the South Slope area. However, distribution of potassic alteration during initial stages of mineralization may have been much more widespread in the Main Zone prior to the development of subsequent sericitic mineral assemblages.

Alteration at the surface of Permian Dzuun Mod phyric rhyolite in the 49-er prospect area consists of quartz sericite that is fairly tightly constrained to the trace of the Sujigtei fault. However, in the general area of the SW Extension (SW Ext) prospect, quartz sericitic alteration is more widespread in surrounding phyric rhyolite where alteration has been mapped as much as 150 m from the Sujigtei fault.

MINERALIZATION

The Gatsuurt Deposit comprises two significant zones of fairly continuous mineralized rock, the Central Zone (including the GT60 and South Slope (SS) areas) and the Main Zone, as well as significant prospects referred to as the 49-er and SW Ext areas. The shape of the mineralized zones is essentially defined on the basis of analytical sampling that determined that the highest gold grades are associated with flooding by silica, stockwork veinlets, and discrete veins.

The Central and Main Zones are separated laterally from each other by the Sujigtei fault, possibly in a major dilational step-over along its fault trace (Tosdal, 2010). The Central Zone lies on the southeast, hanging wall side (east block down) of the Sujigtei fault, whereas the Main Zone is on the northwest side approximately 800 m to the southwest of the Central Zone. The combined length of these two zones from the GT-60 and South Slope areas at the northernmost edge of the Central Zone to the southernmost end of the Main Zone is approximately 2.0 km, and the combined width after restoration of the Central Zone and the Main Zone is approximately 300 m. The Gatsuurt Deposit is open at depth, as mineralized drill intercepts have been encountered approximately 1,000 m below the ground surface.

Three morphologically distinct mineralized zones are present at Gatsuurt:

1. Predominant vertical high-grade siliceous rocks (black silica and pervasive quartz silica) in the Central Zone, Main Zone, and 49-er prospect along the N 40° E striking Sujigtei fault;

2. Zones related to N 20° E striking, steeply dipping splay faults in the South Slope area of the Central Zone;

3. Shallow southeast dipping and near horizontal mineralization along pre-mineralization post-Boroogol igneous complex thrusts that structurally prepared granite in their hanging wall.

In these three zones, gold-mineralized rock consequently includes three fairly discrete populations: (a) quartz-sericite-pyrite-arsenopyrite-iron carbonate (siderite) veinlets; (b) quartz-native gold veinlets; and (c) black silica or pervasive quartz silica rock, in which disseminated sulphides and native gold make up one of the highest-grade components of the deposit.

High gold grade silicified zones in the Gatsuurt Deposit generally form vertical bodies parallel to the Sujigtei fault, where the attitude of the fault controls the morphology of the surrounding mineralized rock. The intensity of mineralization decreases laterally away from the fault, yet the vertical component of mineralization continues to as much as one kilometre below the surface in both the Central and Main Zones where mineralization is open at depth. Pervasive quartz silica rock in the deposit, in places, also includes various types of mylonitic rock compatibly together with sulphides, with relationships indicative of syn-deformation mineralization. Furthermore, much of the highly quartz-veined mineralization components in the deposit include three or more generations of quartz ± sulphide mineral introduction.

Native gold is present in two distinct size populations at Gatsuurt: submicroscopic refractory gold in pyrite and arsenopyrite, and relatively coarse gold in veinlet quartz and silica-replaced rock. Mineralization in the Main Zone is entirely sulphidic, whereas in the Central Zone, especially in the GT60 and South Slope areas, it is composed of a mixture of oxides and sulphides.

Cross sections along three section lines, 10CZ, 20CZ, and 6SS1 (Figures 7-6, 7-7, and 7-8) through the Central Zone, one section line, 14MZ, through the Main Zone (Figure 7-9), and one section line, 49-6, through the 49-er prospect (Figure 7-10) are used to illustrate the morphology of continuously mineralized zones through the Gatsuurt Deposit. The location of the section lines and drilling is shown in Figure 7-3.

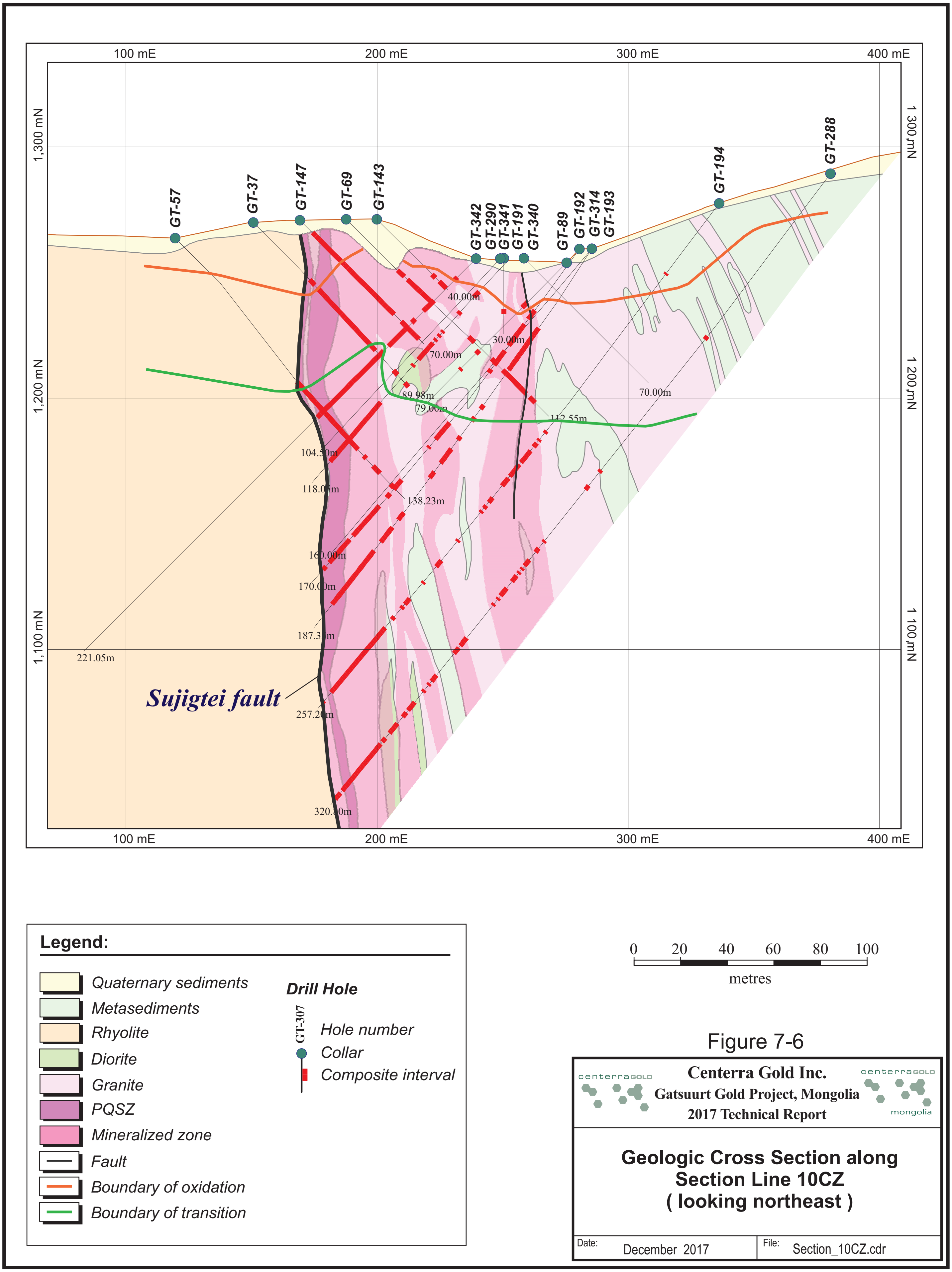

Figure 7-6

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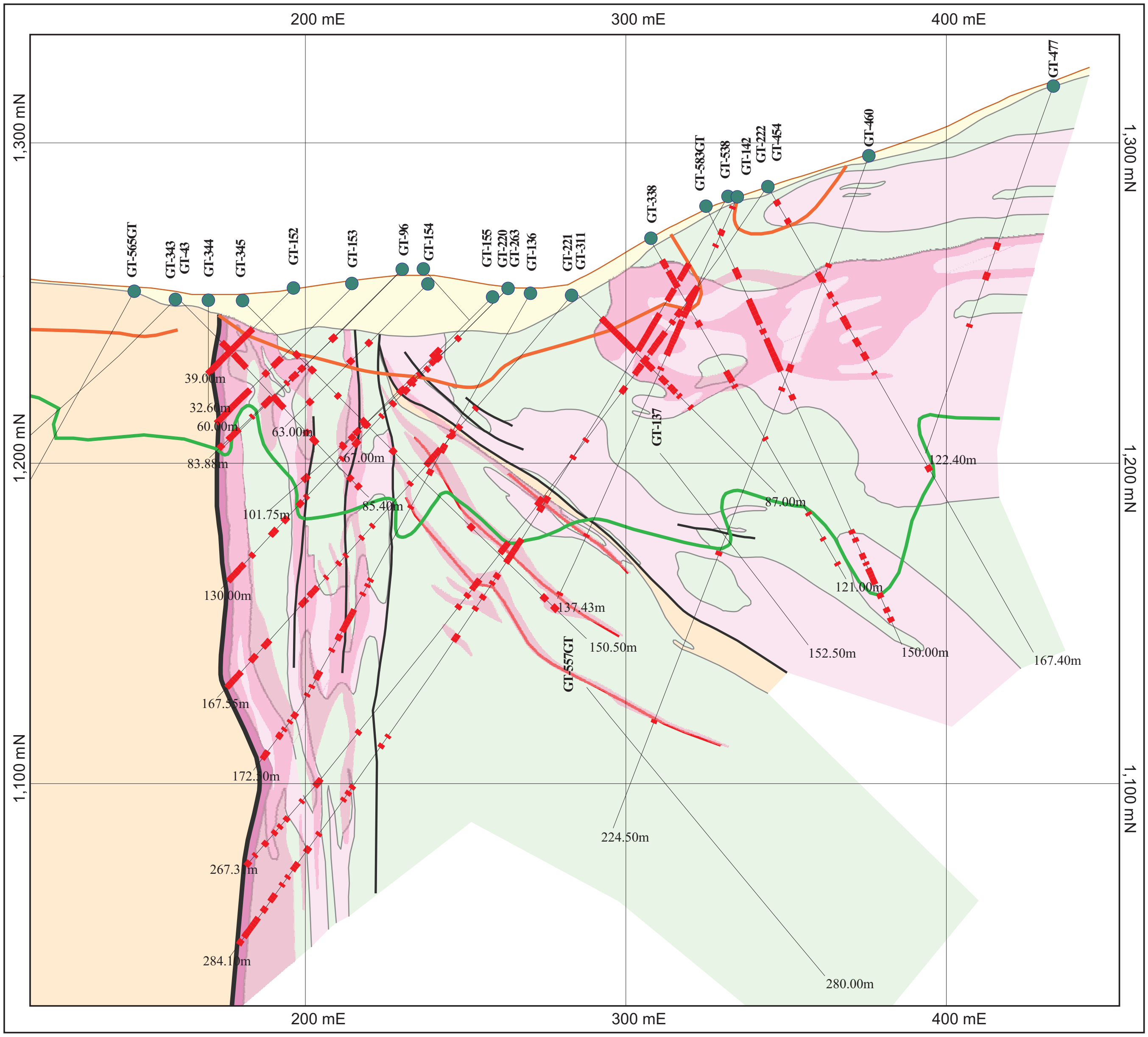
Geologic Cross Section along Section Line 10CZ (looking northeast)

Date: December 2017

File: Section_10CZ.cdr



Legend:

- Quaternary sediments
- Metasediments
- Rhyolite
- Diorite
- Granite
- PQSZ
- Mineralized zone
- Fault
- Boundary of oxidation
- Boundary of transition

Drill Hole

- GT-307 Hole number
- Collar
- Composite interval

0 20 40 60 80 100
metres

Figure 7-7

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Geologic Cross Section along Section Line 20CZ (looking northeast)

Date: December 2017	File: Section_20CZ.cdr

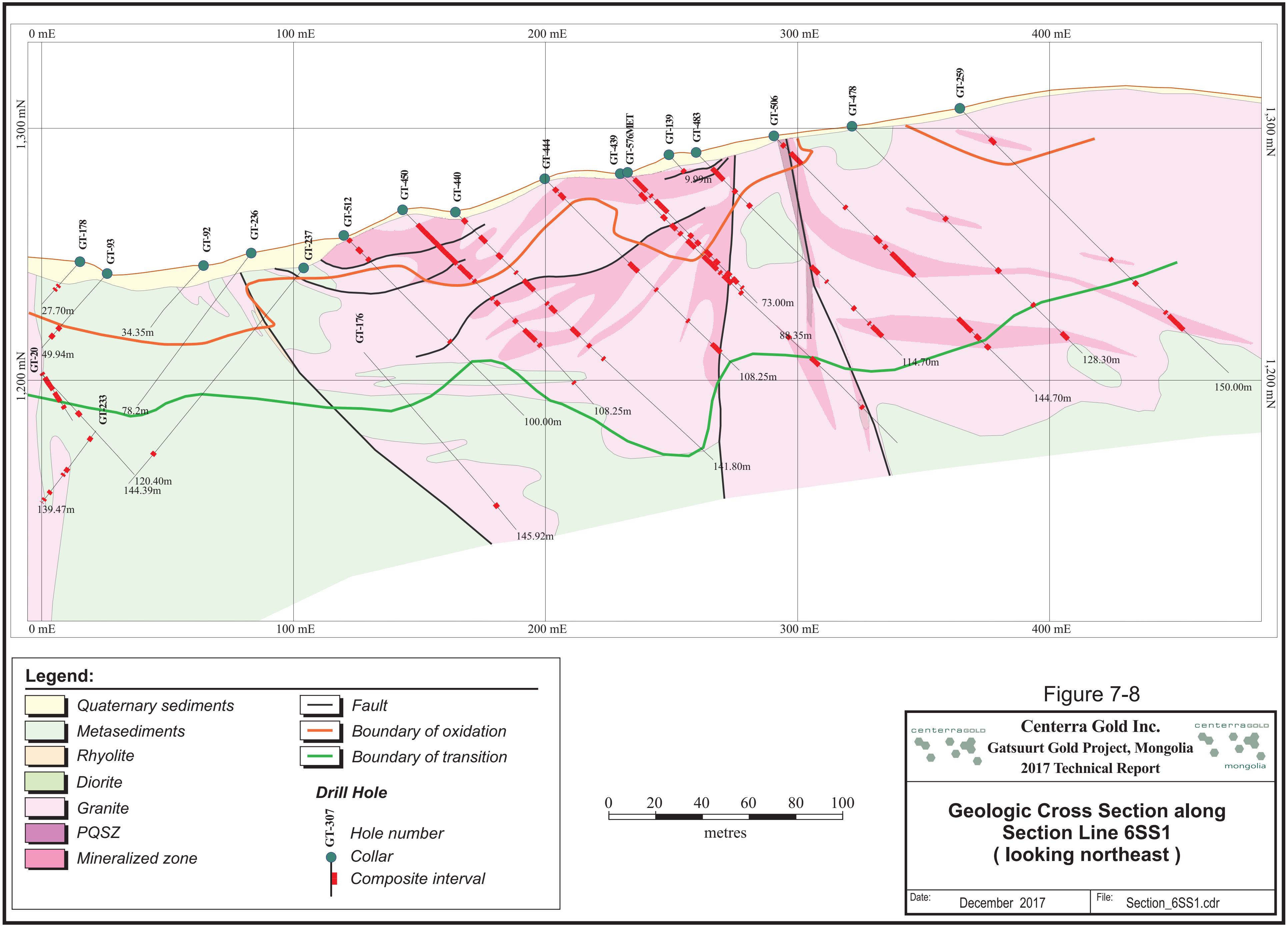
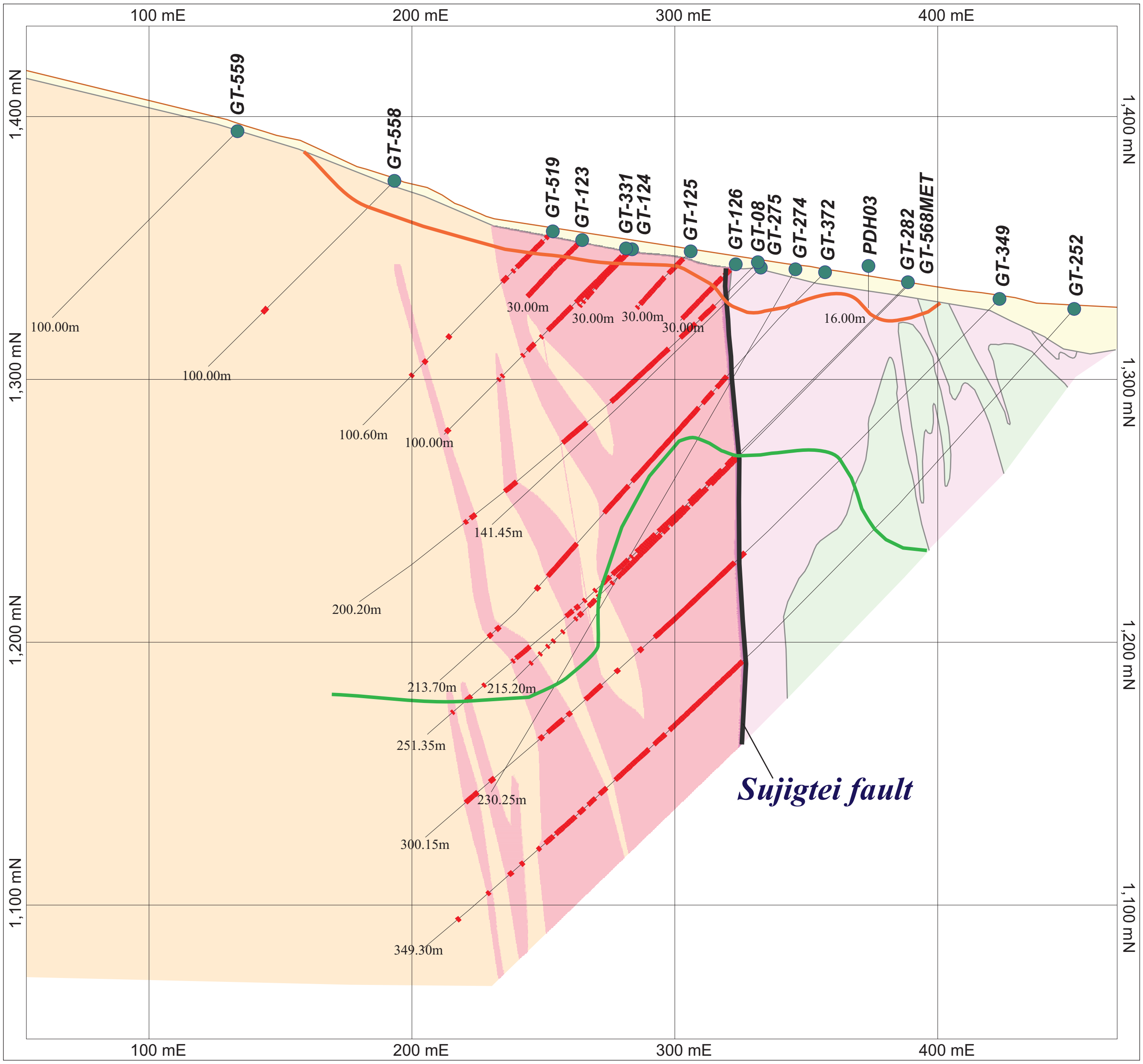


Figure 7-8

Geologic Cross Section along Section Line 6SS1 (looking northeast)



Legend:

Quaternary sediments
Metasediments
Rhyolite
Diorite
Granite
PQSZ
Mineralized zone
Fault
Boundary of oxidation
Boundary of transition

Drill Hole

GT-307
Hole number
Collar
Composite interval

0 25 50 75 100 120
metres

Figure 7-9

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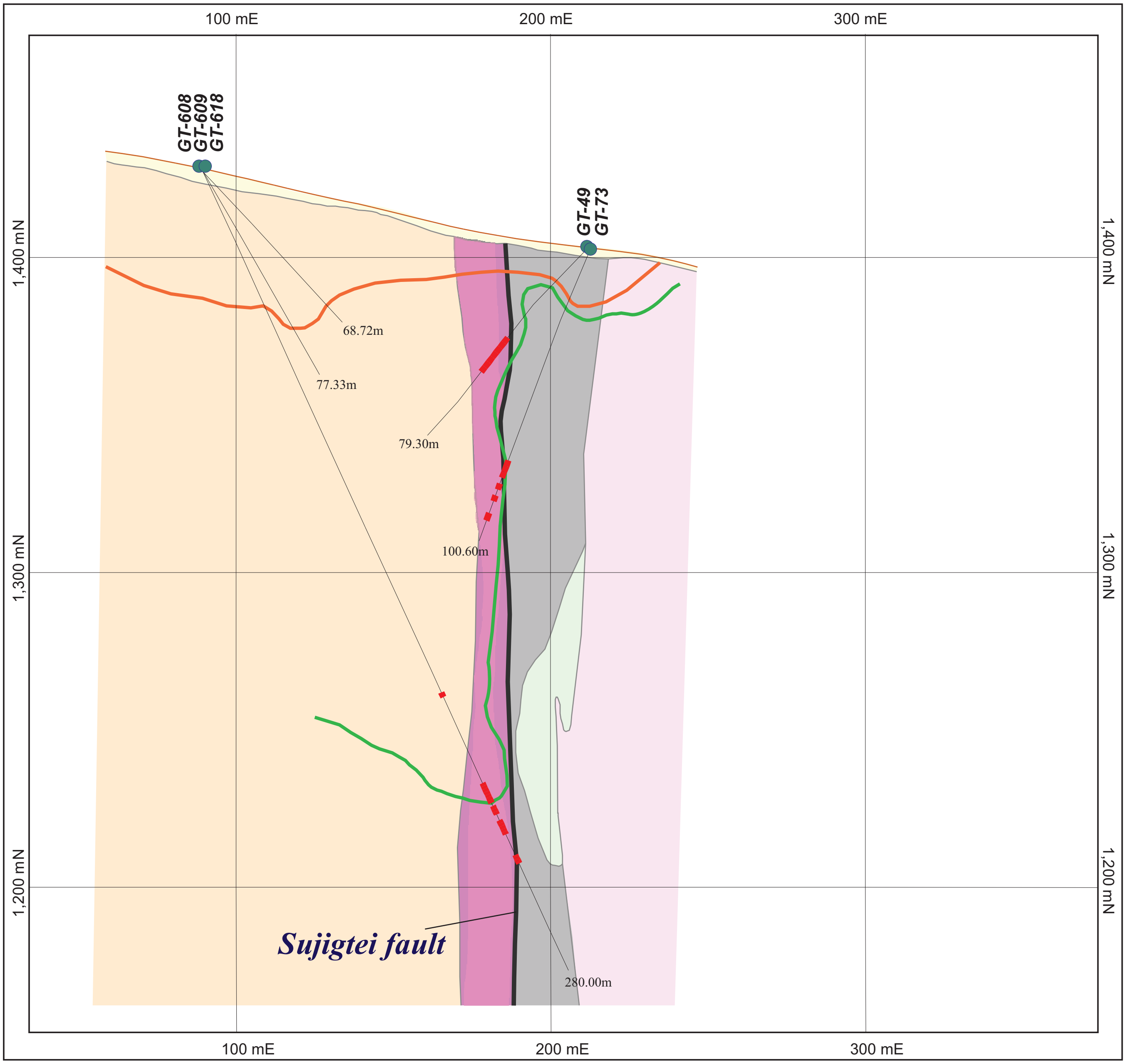
Geologic Cross Section through the Main Zone of Gatsuurt Deposit along Section Line 14MZ (looking northeast)

Date: December 2017	File: Section_14MZ.cdr



Legend:

- Quaternary sediments
- Metasediments
- Rhyolite
- Diorite
- Granite
- PQSZ
- Mineralized zone
- Fault
- Boundary of oxidation
- Boundary of transition

Drill Hole

GT-307

- Hole number
- Collar
- Composite interval

0 20 40 60 80 100

metres

Figure 7-10

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Geologic Cross Section through 49-er Prospect of Gatsuurt System along Section Line 49-6 (looking north-northeast)

Date: December 2017	File: Section_49-6.cdr

8 DEPOSIT TYPES

The Gatsuurt Deposit is identified as an orogenic gold deposit (Böhlke, 1982; Groves et al., 1998; also termed low-sulphide quartz gold by Drew, 2003).

Many other bedrock gold deposits and occurrences in the North Khentei gold belt are orogenic gold-type occurrences (Khishgee et al., 2014). These include Boroo, Ulaanbulag, Baavgait, Ereen, Sujigtei, Tsagaanchuluut, and Urt (Kotlyar et al., 1999; Hendry et al., 2006; Goldfarb et al., 2013). Locally, these occurrences comprise quartz veins with relatively coarse gold and overall low-sulphide content, as well as the presence of disseminated fine gold in nearby highly sulphidized rock. Orogenic gold deposits are commonly closely associated with economically important gold placers, and this relationship is also observed in the Gatsuurt Deposit area.

Mineralizing fluids responsible for generating the Gatsuurt and Boroo deposits are most likely related to deep sources in the earth's crust. These fluids emanated from either metamorphic or magmatic devolitilization reactions in the surrounding orogen and then were channeled by major continental-scale fault systems. They eventually focused in upper level (sub-greenschist facies) residencies at their present crustal positions (Goldfarb et al., 1995; Drew, 2003; Groves et al., 2005; Salier et al., 2005).

Goldfarb et al. (1995) present a schematic geologic cross section through a typical orogenic gold quartz system. Primarily on the basis of the overall presence of minimal amounts of tungsten and antimony in the Gatsuurt and Boroo deposits and the absence of mercury, the two deposits must represent orogenic-style mineralization at intermediate paleo-levels of an idealized orogenic gold system (Figure 8-1). Inferred fluid pathways of deeply sourced mineralizing fluids associated with orogenic gold systems also are depicted in Figure 8-1.

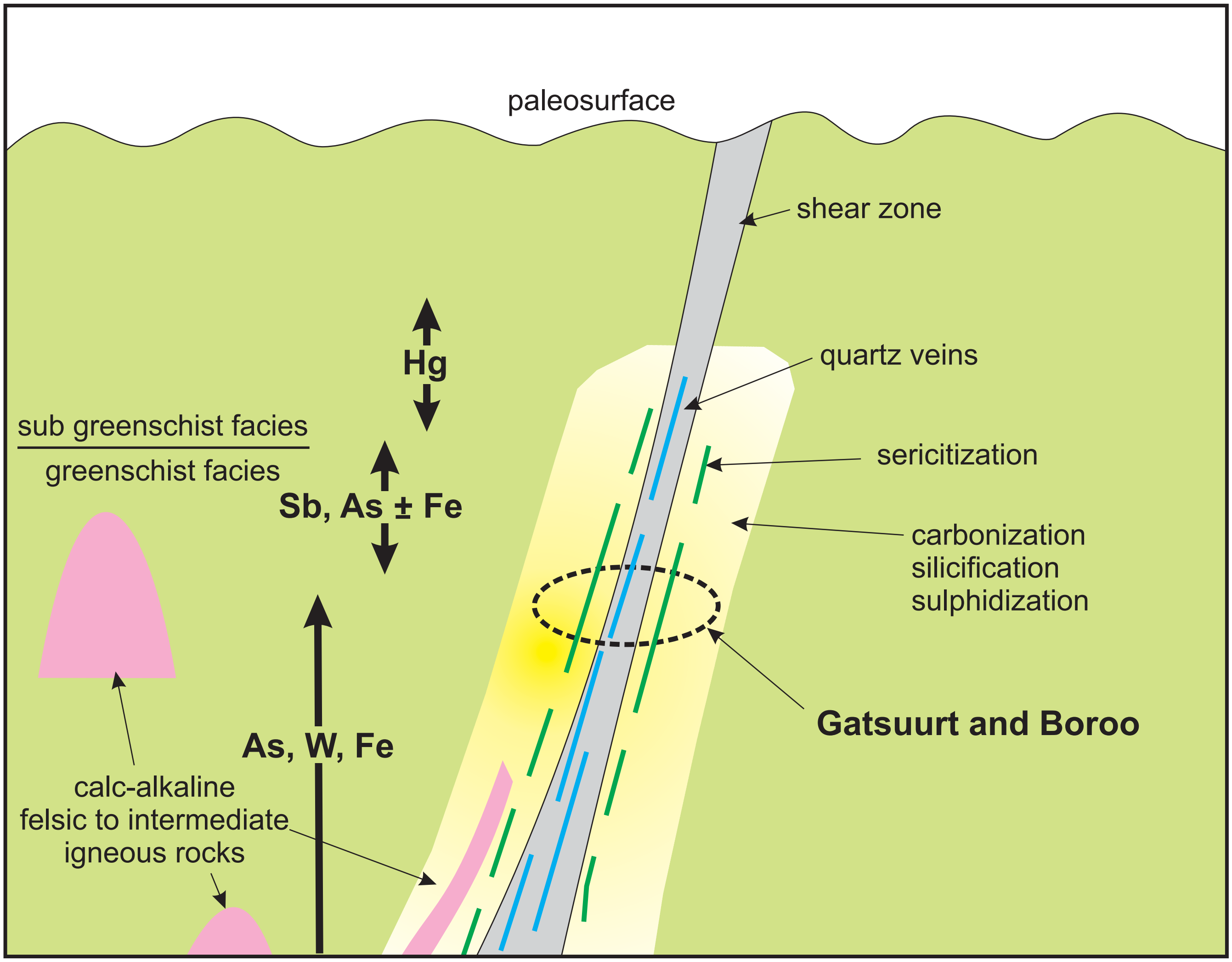


Figure 8-1

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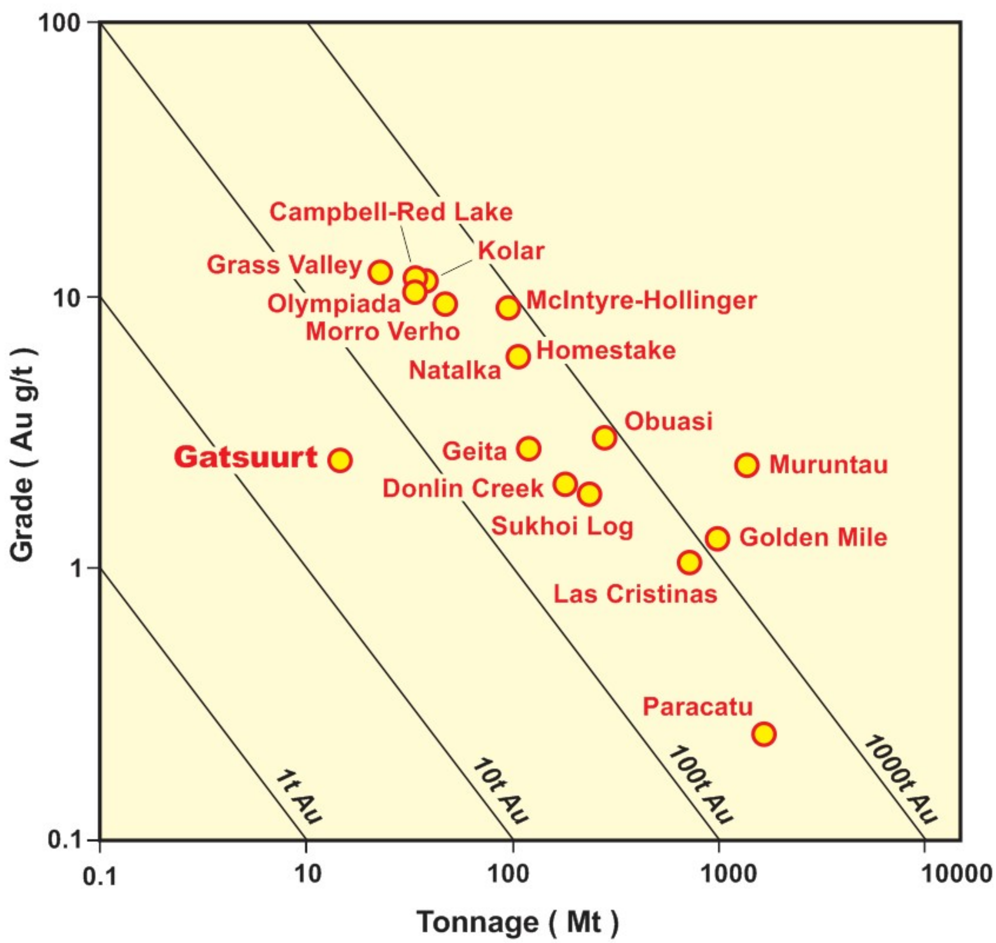
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Schematic Geologic Cross Section of an Orogenic Low Sulphide Gold Quartz Vein System

Date: December 2017	File: GeologicCrossSection.cdr

Though the Gatsuurt Deposit certainly is presently known to contain less gold than the 14 giant orogenic gold camps worldwide (Figure 8-2), Gatsuurt, nonetheless, is situated in a province similar to those of the giant deposits wherein the fundamental control of the deposit location is attributed to continental-scale shear zones related to compressional or transpressional tectonism (Groves et al., 2015). While it contains less gold than these giant orogenic camps, its grade (about 2.74 g/t Au) is generally greater than or equal to the grade in eight of the 14 giant orogenic gold camps. The current tonnage at Gatsuurt is also approximately at the 90+ percentile for tonnage and approximately at the 60 percentile for grade in grade-tonnage cumulative frequency plots for 73 orogenic gold deposits in Finland, Sweden, and Northern Territory Australia (Eilu et al., 2015).

FIGURE 8-2 PLOT SHOWING ORE GRADE VERSUS TONNAGE SHOWING GATSUURT COMPARED TO 14 GIANT OROGENIC GOLD DEPOSITS



9 EXPLORATION

Exploration carried out within the Project area includes diamond and RC drilling, trenching, soil and stream geochemical data, geophysical survey data, and geological mapping. The spatial location of most of this data is usually defined with reference to the UTM grid system.

A detailed description of the drilling programs at Gatsuurt is provided in Section 10 Drilling.

MAPPING

Mapping of the bedrock surface in the Central Zone area has been limited to trenches and rock exposed by placer operations, since most of the valley is under alluvium and placer mining debris. Mapping has been completed at 1:5,000 scale for the entire Project area. Between 1999 and 2010, 10,533.15 m of trenching was completed, including 7,174.05 m at Gatsuurt and 3,359.1 m at Biluut.

GEOCHEMISTRY

From the very onset of field examinations, the geochemistry of rock and soil played an important role in exploration efforts leading to the eventual delineation of the Gatsuurt Deposit. During the 1998 field season, Cascadia geologists collected 44 grab samples of rock in the Dzuun Mod district to accompany their geologic traverses. All these rock samples were analyzed only for gold. These grab samples also incorporated select mineralized rock from the placer pit floor overlying the area now known as the Central Zone that had as much as 119.7 g/t Au. In the intervening years of exploration at Gatsuurt through 2016, overall, 628 grab samples were collected in the area of Gatsuurt's four mining claims.

Many more rocks were analyzed during an extensive trenching program by backhoe that was an integral part of the overall exploration effort at Gatsuurt. A total of 104 trenches totalling 9,850 m, which ranged from 5 m to 200 m in length, were excavated since exploration began by Centerra in the area of the four mining claims. A total of 4,269 channel samples were analyzed from these trenches, amounting to an average channel-sample length of approximately 2.1 m. During the years 1998-2006, channel samples from the trenches were analyzed for gold in the Analab Pty Ltd laboratory and for arsenic by Alex Stewart Assayers. During the years 2009-2010, the channel samples were analyzed for gold and arsenic at Alex Stewart Assayers.

In the nearby dropped exploration claims surrounding the four mining claims, 5,045 m of trenches were also excavated. In all, 1,758 channel samples were analyzed from these trenches for gold and arsenic. This amounted to an average channel-sample length of about 2.9 m. Trench lengths mostly were 20 m to 200 m, though some were as long as 400 m.

All trenches dug by Centerra have since been reclaimed, including being covered by soil saved from the original ground surface along the trenches.

In addition, 9,379 geochemical soil samples were collected by Centerra in the Dzuun Mod district. The grids used were either 100 m x 200 m or 100 m x 100 m over the bulk of the district, however, tighter grids of 25 m x 25 m, 50 m x 50 m, and 50 m x 100 m were utilized over the deposits and areas containing anomalous gold. Approximately 1.5 kg to 2.0 kg soil samples were collected from "B" horizons at 20 cm to 50 cm depths. Samples were dried out at ambient air temperatures in the field, and then sieved retaining the minus 80 mesh fractions for analysis. The soil samples were analyzed for various elements at different laboratories between 1998 and 2010 (Table 9-1).

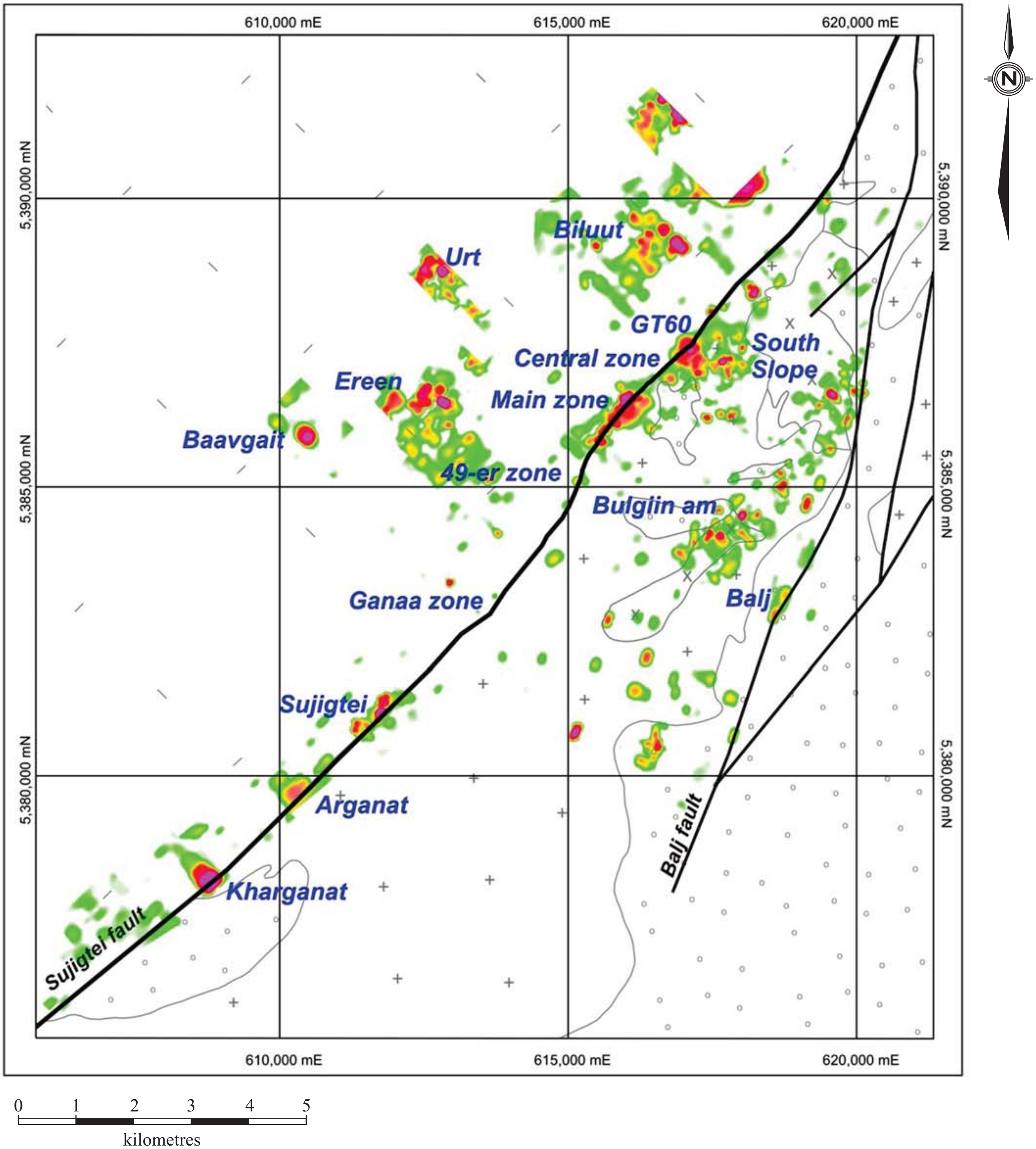
TABLE 9-1 SUMMARY OF SOIL SAMPLES ANALYZED GEOCHEMICALLY AT GATSUURT AND LABORATORIES CONDUCTING ANALYSES

Years	Number Analyses	Elements Analyzed	Laboratory
1998	444	Au, As, Ag, Cu, Pb, Sn, Sb, W Zn	Analabs Pty Ltd Laboratory
1999-2000	4,464	ICP–33 element	ACME Analytical Laboratories
2005-2006	2,080	ICP–72 element	AAL American Assay Laboratory
2009-2010	2,391	Au, As	Actlabs Asia LLC
Total	9,379		

Figure 9-1 shows gold in soil anomalies in the Dzuun Mod district, demonstrating close spatial relation of > 50 ppb Au and an approximately 12 km long trace of the Sujigtei fault (Baasandolgor, 2003). The two gold in soil anomalies at the Central and Main Zones are among the strongest throughout the district, however, soil gold anomalies are also present as much as four kilometres away from the Sujigtei fault. To the northwest of the Sujigtei fault, these anomalies are present in the Biluut, Ereen, and Baavgait areas where the host rock is Permian rhyolite of the Dzuun Mod rhyolitic complex. Anomalous soil gold at Biluut has an area of approximately four square kilometres. In the Balj and Bulginam areas, to the southeast of the Sujigtei fault where the host is Early Paleozoic Kharaa meta-sandstone and Paleozoic Boroogol Complex granite and diorite, anomalous soil gold has a generally N 40° E trend that parallels the Sujigtei fault, and is bounded on the east by a N 20° E–striking fault.

Figure 9-2 shows the distribution of arsenic anomalies in soil over the entire Dzuun Mod district, which demonstrates a strong concentration of arsenic clustered at the Central and Main Zones (≥ 300 As), as well as an alignment of somewhat more areally restricted anomalous arsenic along the Sujigtei fault at the Sujigtei and Arangat prospects (Figure 9-2). In addition, the Biluut, Ereen, and Baavgait areas in the Dzuun Mod rhyolite also have anomalous clusters of arsenic somewhat mimicking the presence of anomalous soil gold. To the southeast of the Sujigtei fault, at Bulgiinam and Balj, a broad 30 ppm to 50 ppm As area has a N 40° E elongation parallel to the trace of the Sujigtei fault.

In the QP's opinion, the samples are representative and there are no factors that may have resulted in bias.



Legend:

Rhyolite	Sandstone
Granite	Sujigtei fault
Diorite	Fault

Soil Au in ppm

<10 30 50 100 >300

Figure 9-1

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Gold in Soil Anomalies Map in Dzuun Mod District

Date: December 2017 | File: Soil_Au_map.cdr

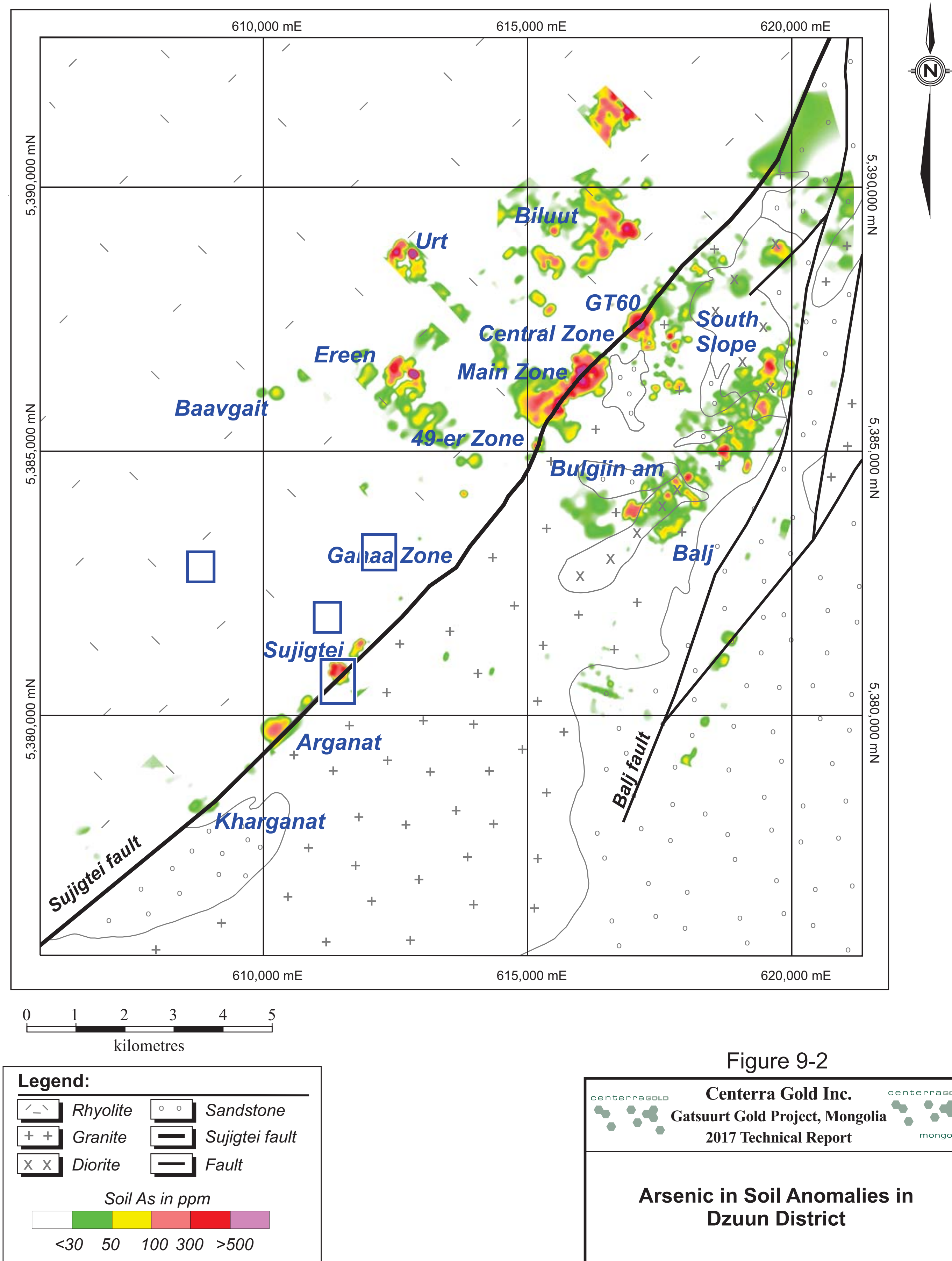

Figure 9-2

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Gatsuurt Gold Project, Mongolia
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Arsenic in Soil Anomalies in Dzuun District

Date: December 2017 | File: Soil_As_map.cdr

GEOPHYSICS

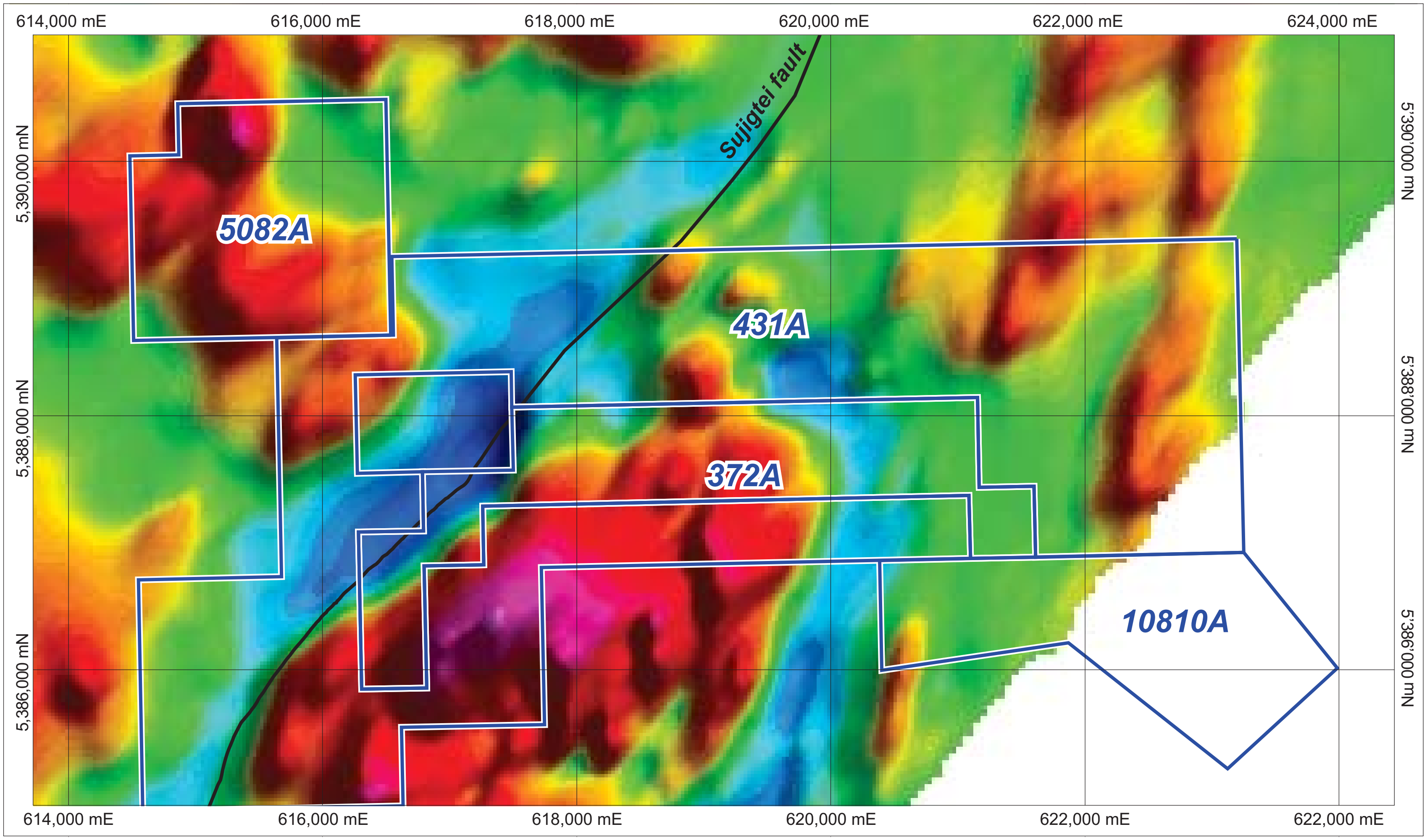
From 1998 to 2002, geophysical surveying within the Project area included gradient IP and magnetometer surveys. In 1998, the IP and magnetic surveys were performed by Quantec Geoscience Inc. and Geosan Co. Ltd., respectively. Between 1998 and 2006, gradient IP surveys covering 7.9 km^2 were performed by GeoMaster Engineering Ltd. to delineate sulphide zones and related gold mineralization in the Central Zone, Main Zone, and South Slope area. The IP and resistivity survey outlined the two gold zones and a number of anomalous zones mostly southeast of the Central Zone, on the south valley slope, providing additional drill targets there. A distinct area of high resistivity and chargeability, coinciding with low magnetic response, is present in the northeast part of the Central Zone and outlines silicification and sulphides. The IP coverage straddling the Sujigtei fault at the Central Zone is approximately 1,000 m northwest and southeast. In addition, 159.65 m of dipole-dipole IP geophysical surveys were conducted in the Central Zone, Main Zone, and South Slope area during 2009 and 2010 to further define the gold zones.

MAGNETICS

Figure 9-3 shows an airborne magnetic map over the Gatsuurt Deposit. Though regional airborne magnetics appear to coincide well with the trace of the Sujigtei fault as a continuous narrow band of low magnetic susceptibility, this low actually results from major mineralization alteration cells that yield the low susceptibility band. Ground magnetics, in fact, provide much better resolution of the impact of alteration on pre-existing magnetite-bearing host rocks.

Figure 9-4 is a map showing ground magnetics at the Central Zone (including the GT 60 and South Slope areas), Main Zone, Southwest Extension (SW Ext), and 49-er area of Gatsuurt mineralized system. At Gatsuurt, especially in the Central Zone, ground magnetics define lobate-shaped areas of low magnetic susceptibility marking lateral penetration of sulphidizing fluids away from the Sujigtei fault and into surrounding pre-mineralization host rocks. Alteration cells that formed along the Sujigtei fault are made up mostly of gold mineralization-associated, quartz-sericite-pyrite (± K feldspar) mineral assemblages that have destroyed pre-existing magnetite. This is especially pronounced in Kharaa Series meta-sedimentary rock and Boroogol Complex intrusive rock in the Central Zone southeast of the Sujigtei fault. As shown in the ground magnetic map, these lobes of low magnetic susceptibility project southeasterly into a broad area of high magnetic susceptibility reflecting Boroogol Complex granitoids and Kharaa metamorphic rocks.

Susceptibility contrast is not as pronounced in the Main Zone where gold-mineralized rock is hosted by Dzuun Mod phyric rhyolite on the northwest side of the fault. A decrease in values of magnetic susceptibility southeast of the Main Zone and in the eastern block of the Sujigtei fault partly may result from near-surface presence of oxidized rock. Also, a combination of a general lack of pre-mineralization magnetite in phyric rhyolite also may have combined with intense sulphidation accompanying mineralization to produce the narrow northeast elongate magnetic susceptibility low centred on the Sujigtei fault.



Legend:

Sujigtei fault

Mining license

RTP Magnetic (nT)

0 0.5 1 1.5 2

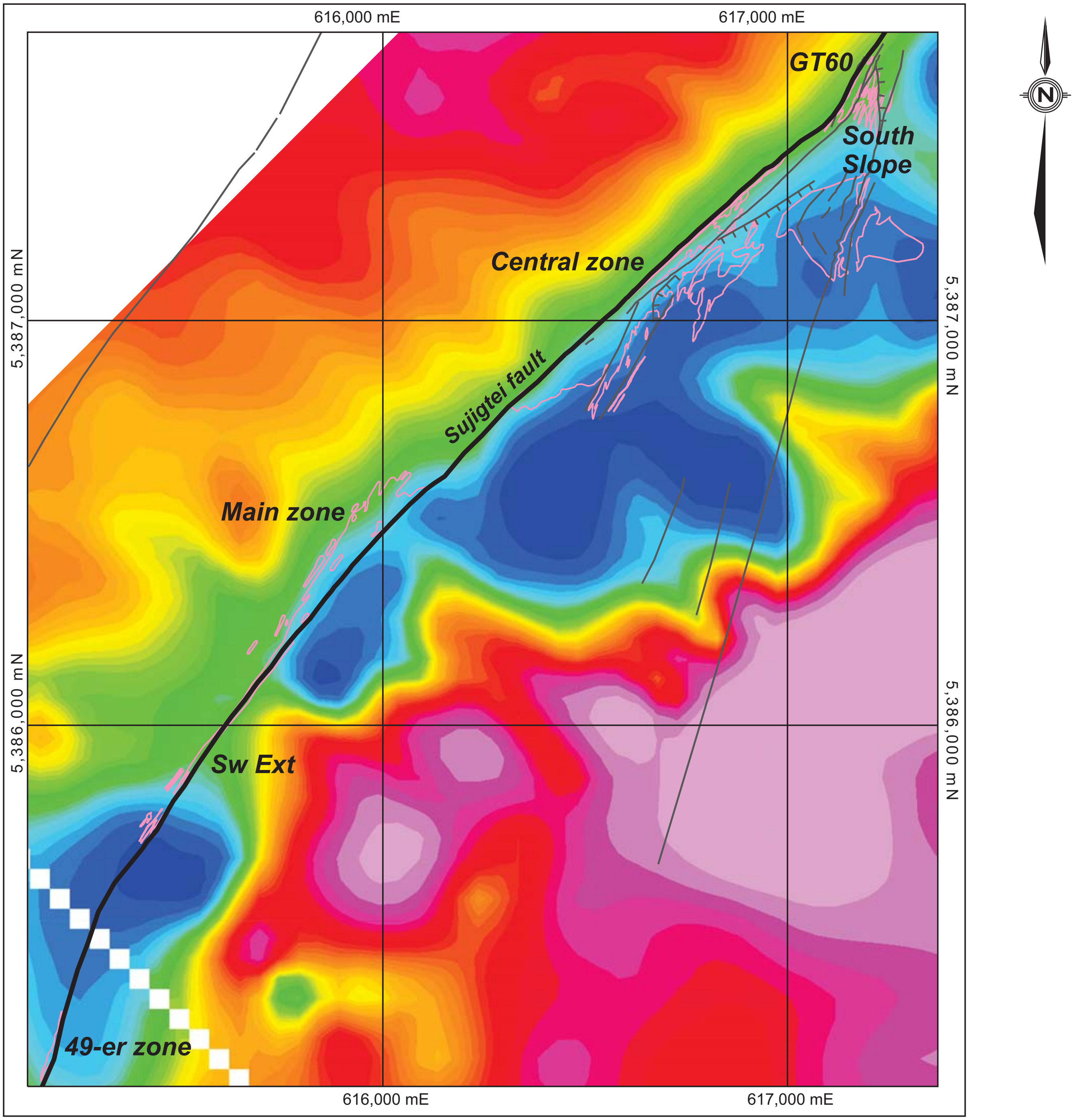
kilometres

Figure 9-3

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Airborne Magnetic Map over Gatsuurt Mining Licenses

Date: December 2017	File: Airborne_magnetic_map.cdr



0 150 300 450 600 750
metres

Legend:

- Sujigtei fault
- Inferred fault
- Gentle dipping fault
- Mineralized zone

Magnetic Field %

Figure 9-4

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Gatsuurt Gold Project, Mongolia
2017 Technical Report

Ground Magnetic Map

Date: December 2017 | File: Ground_magnectic_map.cdr

CHARGEABILITY AND RESISTIVITY

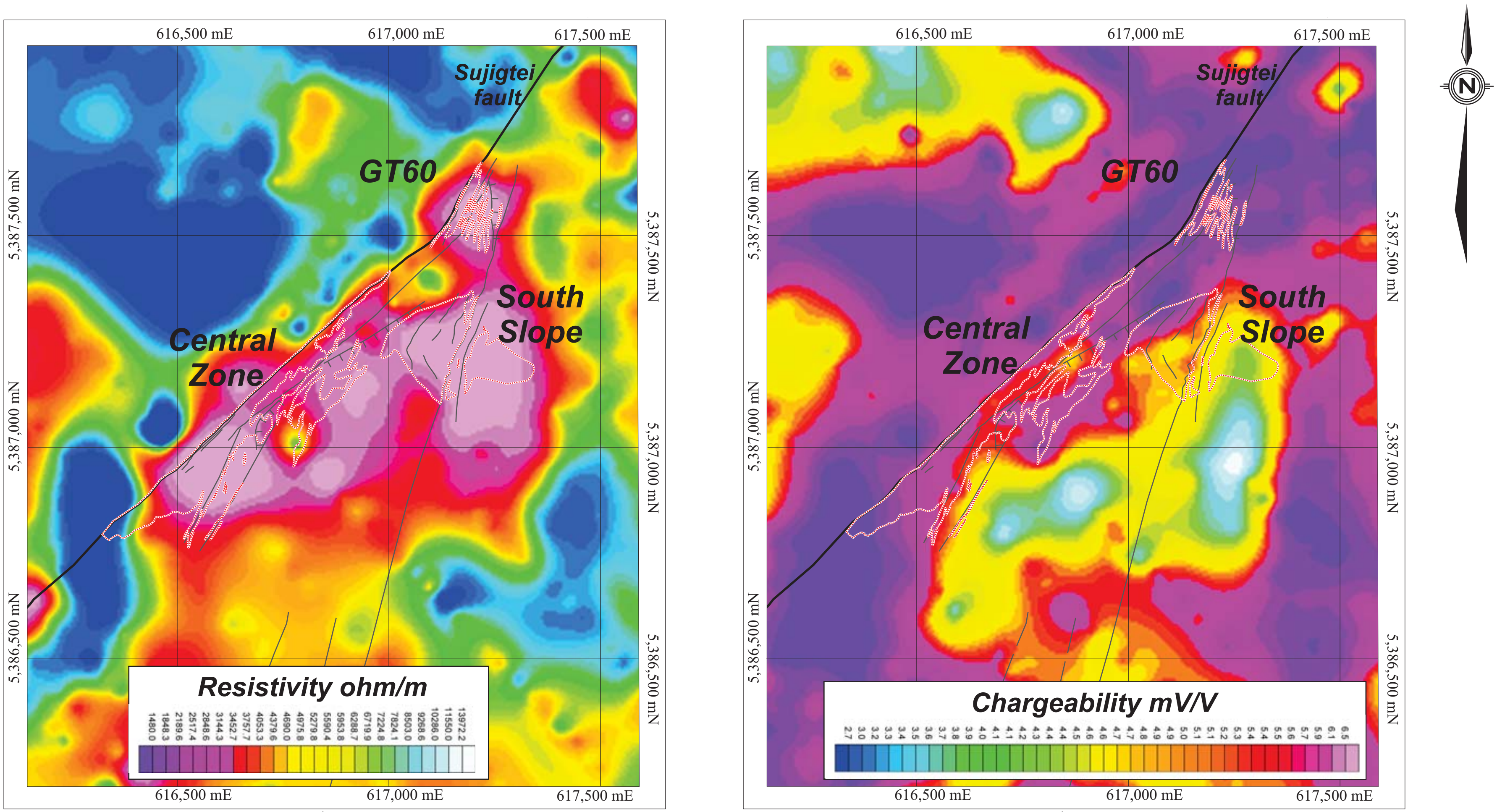
Figure 9-5 shows a plan view of the dipole-dipole induced polarization (DDIP) resistivity (left) and chargeability (right) at the Central Zone, including the South Slope and GT60 areas. The mineralized zone outlines are also shown. Gradient array chargeability data identify the Central Zone as a chargeability high, located immediately southeast of the trace of the Sujigtei fault. The Sujigtei fault has a marked chargeability contrast along much of its trace. The chargeability high over the Central Zone is bi-lobate. One lobe parallels the Sujigtei fault and extends N 40° E generally within the planned outer pit limit, while the other lobe extends N 20° E and is present mostly outside the planned pit limit. The latter lobe is bounded on its west by a N 20° E–striking fault and is centred on an approximately 200 m wide zone with gold grades ≥ 1 ppm.

Gradient array resistivity results at the Central Zone also indicate the presence of two prominent resistivity highs generally on the periphery and south of the two lobate chargeability highs (Figure 9-5 right). These two resistivity highs are also elongated along the N 40° E and N 20° E trends. They most likely represent areas of increased sulphide-poor silicification that mantle sulphide-rich stockwork veinlets in the core of the mineralized system.

Figure 9-6 presents plan views showing the DDIP resistivity (left) and chargeability (right) at the Main Zone deposit and the SW Ext prospect. The mineralized zone outlines are also shown.

Gradient array chargeability data identify the Main Zone as a chargeability high, elongated N 40° E along the Sujigtei fault and mostly northwest of the trace of the Sujigtei fault in the Dzuun Mod rhyolite. The chargeability high perfectly encompasses most precisely the 1 ppm Au envelope at the Main Zone. Overall intensity of chargeability in the Main Zone is similar to that in the Central Zone.

Gradient array chargeability data in the area of the SW Ext is somewhat more ovoid than that in the Main Zone, and is not parallel to the Sujigtei fault.



Legend:
- Mineralized zone
- Sujigtei fault
- Inferred fault
- Gentle dipping fault

0 150 300 450 600 750
metres

Figure 9-5

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Gatsuurt Gold Project, Mongolia
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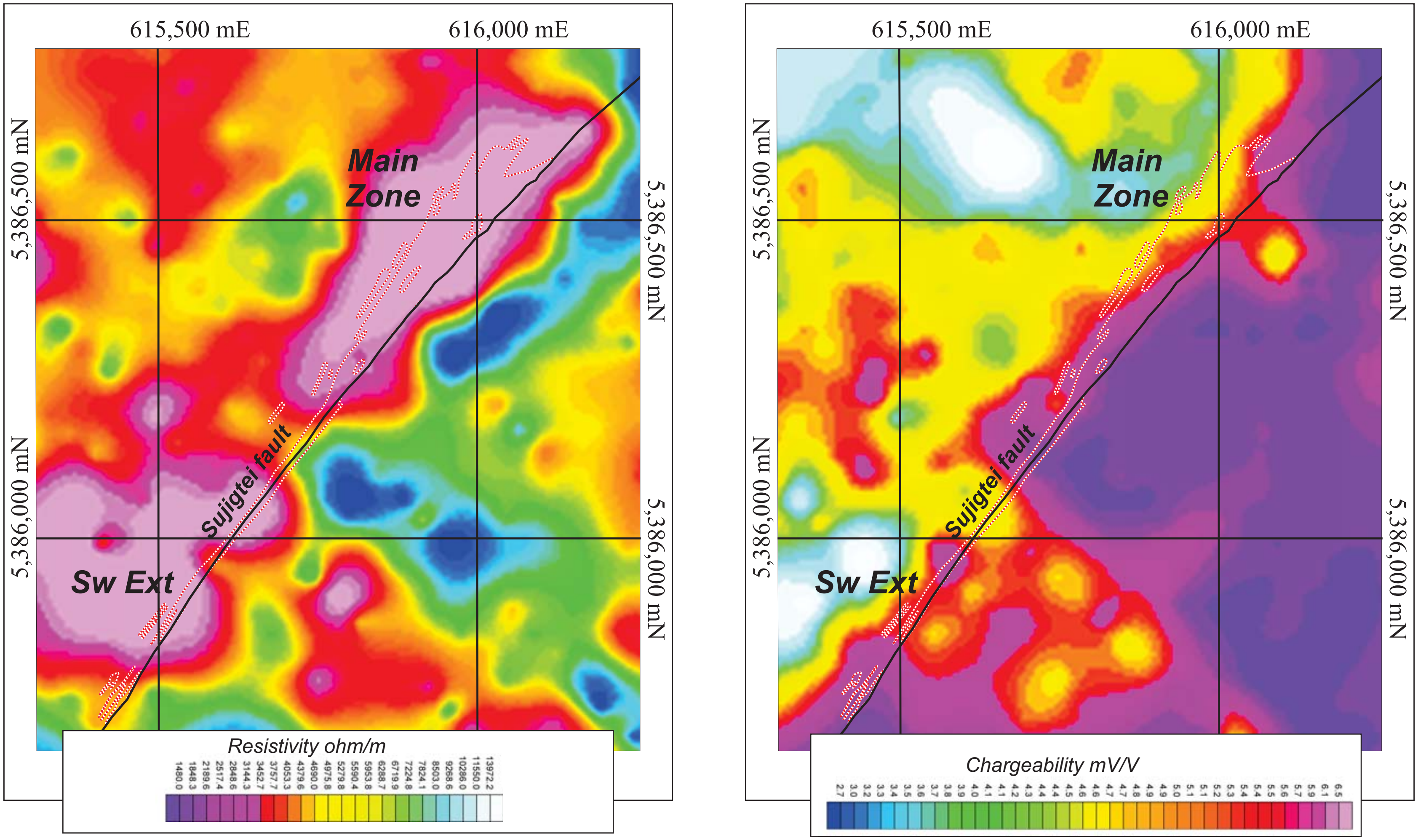
Central Zone DDIP

Date: December 2017	File: DDIP_CZ_map_updated.cdr



Legend:

- Mineralized zone
- Sujigtei fault
- Inferred fault
- Gentle dipping fault

0 100 200 300 400 500
metres

Figure 9-6

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Gatsuurt Gold Project, Mongolia
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Main Zone DDIP

Date: December 2017	File: DDIP_MZ_map.cdr

10 DRILLING

Exploration drilling at the Property has been a combination of both core and RC drilling with drill campaigns spanning the years from 1998 to 2016:

- 1998: Geological and IP geophysical targets were drill tested by four drill holes, one of which intercepted a broad zone of mineralization yielding 1 g/t Au to 2 g/t Au at the Main Zone target.

- 1999 to 2000: Limited drilling programs were completed. The Main Zone was tested by an additional seven holes totalling 1,151 m, and the Central Zone target was tested by 16 holes totalling 2,138 m. Preliminary metallurgical testwork determined that the gold mineralization at both zones was refractory.

- 2001 and 2002: An additional 77 core drill holes totalling 4,534 m were completed at the Central Zone. Much of the deposit was tested systematically to vertical depths of 75 m to 100 m, generally coinciding with the contact between the transitional oxide and fresh sulphide zone.

- 2003: Twelve RC and core drill holes totalling 1,755 m were completed at the Central Zone to test for strike extensions to the mineralization, and to test IP anomalies at the South Slope target. Eleven shallow RC holes totalling 435 m were also completed at the Main Zone to determine cyanide leach characteristics of shallow, oxidized mineralized rock.

- 2004: A total of 130 drill holes for 21,697 m were completed at the Central Zone, largely comprising in-fill drilling and systematic testing of the deposit to greater depths. Five holes totalling 1,497 m were also completed at the Main Zone during 2004 to test for mineralization at depths below known mineralization.

- 2005: Exploration drilling at the Central Zone focused on in-fill drilling and testing for strike extensions to the mineralization; 33 holes totalling 4,877 m were completed. In the Main Zone, 59 exploration holes totalling 10,916 m were completed as well.

- 2007 to 2008: No drilling took place.

- 2009: Drilling included 18 drill holes totalling 2,337 m at the Central Zone. Two geotechnical holes were also completed in the Central and Main Zones for a total of 343m.

- 2010: Drilling included 80 drill holes totalling 9,874 m that were focused on the South Slope GT-60 target area. The probable reserves increased by 209,000 contained ounces of gold due to the successful exploration drilling in the South Slope area.

- 2011 to 2015: No drilling occurred.

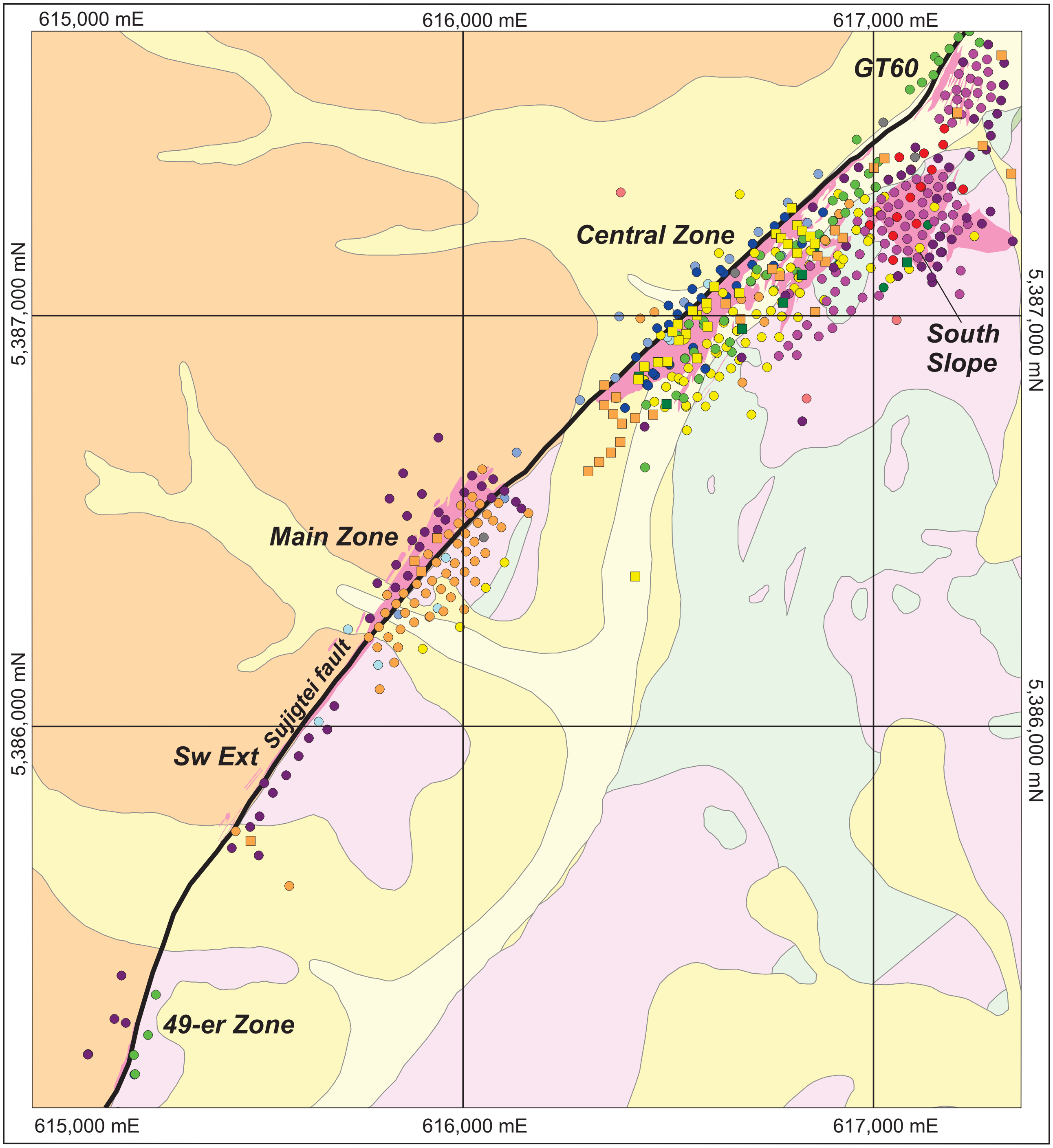
- 2016: Drilling comprised 47 holes totalling 6,903 m at the Central Zone and 35 holes totalling 3,973 m at the Main Zone.

Table 10-1 is a summary of drilling conducted between 1998 and 2016. Figure 10-1 is a map showing drilling coded by year.

TABLE 10-1 DRILLING CONDUCTED FROM 1998 THROUGH TO DECEMBER 2016

Year	Zone	Exploration				Metallurgy				Geotechnical			
		Core	m	RC	m	Core	m	RC	m	Core	m	RC	m
1998	CZ	3	595	-	-	-	-	-	-	-	-	-	-
	MZ	1	277										
1999	CZ	3	342	-	-	-	-	-	-	-	-	-	-
	MZ	5	749	-	-	-	-	-	-	-	-	-	-
2000	CZ	13	1,796	-	-	-	-	-	-	-	-	-	-
	MZ	3	426	-	-	-	-	-	-	-	-	-	-
2001	CZ	28	2,738	-	-	-	-	-	-	-	-	-	-
	Other	1	79	-	-	-	-	-	-	-	-	-	-
2002	CZ	49	4,594	-	-	-	-	-	-	-	-	-	-
	Other	7	508	-	-	-	-	-	-	-	-	-	-
2003	CZ	5	988	7	767	-	-	-	-	-	-	-	-
	MZ	-	-	-	-	-	-	11	435	-	-	-	-
	Other	2	139	1	100	-	-	-	-	-	-	-	-
2004	CZ	100	18,886	25	1,314	-	-	-	-	-	-	-	-
	MZ	4	1,359	1	138	-	-	-	-	-	-	-	-
2005	CZ	9	1,870	24	3,007	14	1,693	-	-	8	796	-	-
	MZ	50	9,025	9	1,229	2	372	-	-	3	662	-	-
2006	CZ	4	2,580	-	-	-	-	-	-	-	-	-	-
2007	-	-	-	-	-	-	-	-	-	-	-	-	-
2008	-	-	-	-	-	-	-	-	-	-	-	-	-
2009	CZ	17	2,217	-	-	-	-	-	-	1	223	-	-
	Other	3	292	-	-	-	-	-	-	1	120-	-	-
2010	CZ	80	9,874	-	-	-	-	-	-	-	-	-	-
2016	CZ	47	6,903	-	-	11	1,278	-	-	7	1,336	-	-
	MZ	35	3,973	-	-	2	350	-	-	5	920	-	-
	Other	6	1,095	-	-	-	-	-	-	-	-	-	-
Total		475	71,305	67	6,555	29	3,693	11	435	25	3,937	-	-

Notes. CZ – Central Zone, MZ – Main Zone
1998 to 2000 – drilling by Cascadia; 2001 onwards – drilling by CGM



Legend:

- Qal — Quaternary sediment alluvium
- Qdl — Quaternary sediment diluvium
- P1 — Early Permian Dzuun Mod volcanic rocks: rhyolite, rhyolite-porphyry
- PZ1gr — Early Paleozoic Boroogol complex: granite, granodiorite
- PZ1hr — Early Paleozoic Boroogol complex: metasandstone, metasiltstone, schist
- Sujigtei fault

Core Drill Holes

1998, 1999, 2000, 2001, 2002, 2003, 2004, 2005, 2006, 2009, 2010, 2016

Reverse Circulation Drill Holes

2003, 2004, 2005

0 150 300 450 600 750

metres

Figure 10-1

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Gatsuurt Gold Project, Mongolia
2017 Technical Report

Drill Plan Map

Date: December 2017	File: Drillhole_plan_map.cdr

CORE LOGGING

The methods for core logging were standardized for drilling campaigns starting with Cascadia and continuing with Centerra.

Prior to logging, core was aligned such that broken core was reassembled. Then, geotechnical and geologic logging were completed and sample intervals were marked on the core. The heads of core boxes were marked with the hole number, box sequence number, and downhole interval, in accordance with standard industry practice. Aluminum sample tags were stapled to the core box at sample intervals. Core, in core boxes, was photographed before and after sawing. Information collected during Cascadia operations was recorded on paper drill logs and included observations on lithology, alteration, structure, and mineralization. A review of the logs indicates that they were complete and of high quality. Handwritten logs were later input to digital format in Excel.

Starting in the 2001 drill campaign, all subsequent core logging was directly recorded to digital format using CoreView/Interdex software.

COLLAR SURVEYS

Collar locations were surveyed by theodolite, electronic measuring device (EDM), and global positioning system (GPS) in 2001. Errors were discovered in the EDM results, and all collars were resurveyed with a differential GPS in 2002. A differential GPS has been used to survey all hole collars in subsequent drilling campaigns. Coordinates were recorded in UTM projection using the WGS 84 Datum.

DOWNHOLE SURVEYS

The downhole surveying for the Cascadia and Centerra drilling through 2001 consisted of acid dip inclination tests at 50 m intervals. Downhole measurements in 2002 were taken by a Tropari survey instrument that reads both inclination and azimuth by use of a magnetic compass. The Tropari was replaced by Sperry Sun single shot surveys in 2003. A Reflex Instrument AB (Reflex) single shot digital device was in use for the 2004 program. Tests were recorded every 50 m downhole distance, with azimuth readings referenced to magnetic north. A four degree adjustment was applied in collar surveying to account for the local declination. The magnetic declination at the Central Zone was 4°W in 2004.

Downhole surveying during drill campaigns after 2004 was also carried out at 50 m intervals. Inclination azimuth and dip of drill holes continued to be measured using a Reflex EZ-SHOT Instrument, a self-contained, single shot instrument providing borehole direction, temperature, and magnetic measurements.

FIGURE 10-2 REFLEX EZ-SHOT INSTRUMENT USED IN DOWNHOLE SURVEYING

A review of the survey database indicated that less than 10% of the holes have no recorded downhole surveys, leading to some uncertainty regarding the core position at depth. Less than 1% of the holes had the same azimuth and dip downhole, which indicates that these holes either have no deviation or were not surveyed. Less than 10% had the same azimuth or the same dip downhole, which indicates that these holes have no deviation or were only partially surveyed.

CORE RECOVERY

Core recovery, as well as rock quality designation (RQD), is generally 90% to 100% within the Gatsuurt mineralized zones. Localized, minor poor core recovery and low RQD occur in faulted intervals.

SAMPLING METHOD AND APPROACH

Sampling of gold mineralization at Gatsuurt has included surface diamond drilling and channel sampling in surface trenches. The Central Zone Mineral Resource estimate is supported by diamond drill hole core samples. The Main Zone Mineral Resource estimate is supported by diamond drill hole core and channel samples.

From 1998 to 2001, sampling of mineralization was carried out at two metre to three metre intervals. This procedure was revised by Centerra after 2001 to sampling at one metre intervals in mineralization and two metre intervals in unaltered host granites or waste. The one metre interval is considered by Centerra to be the standard interval length for mineralization amenable to open pit mining method. Centerra has continued to use the sampling protocol established in 2001 for all drilling through 2016. In the QP's opinion, this interval length is suitable and practical for the mineralization observed at Gatsuurt.

Core intervals identified for sampling are moved to a separate ger (Mongolian tent) adjacent to the core logging ger. A sample technician uses a diamond saw to cut the core lengthwise into two equal halves. One half is placed into a numbered plastic sample bag along with a sample tag. The other half core is placed back in the core box and then the core box is covered for transport to storage at the Boroo Mine site. Samples marked for duplicate assay have one of the core halves sawed. One quarter of the core is returned to the core box and the other is bagged for duplicate assay. The diamond saw blade is cleaned by cutting 5 cm in barren siliceous rock before each sample. Saw coolant water is continuous flow to ensure contamination free cutting. The sample technician maintains core alignment when sawing and replacing core in the core box. To maintain good core alignment and simplify this in practice, a china marker is used to mark a line along the core indicating the cut line during geotechnical core logging. This ensures consistent sampling, particularly if an experienced technician has to be replaced. Such procedures were applied whenever high-grade individual veins were encountered during the drill campaigns in 2009-2010 and in 2016, however, the overwhelming bulk of mineralized rock in the Central and Main Zones included silica flooding and development of stockworks.

Individual sample bags are prepared for transport and packed into rice bags, which are closed with plastic cable locks. The sample dispatch is carried out by technicians in a separate ger.

In the QP's opinion, the sampling method and approach at the Gatsuurt Gold Project follows industry standard practice.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

SAMPLE PREPARATION AND ASSAYING

Most of the drill core samples collected by Cascadia and Centerra were submitted to SGS Analabs (SGS Mongolia LLC) for assaying. SGS Analabs are accredited mineral laboratories that are independent from Centerra. Some of the early Cascadia samples were assayed by Dunn Analytical, also independent from Centerra. Table 11-1 lists the primary assay laboratories by year.

RC samples were prepared by Alex Stewart Assayers Mongolia LLC (Alex Stewart) in Ulaanbaatar for assay in Bishkek, Kyrgyz Republic. Alex Stewart was not accredited but participated in laboratory round robin assay studies. Alex Stewart is a laboratory independent from Centerra.

Centerra personnel have visited the SGS Analabs and Alex Stewart facilities on numerous occasions over the years.

TABLE 11-1 PRIMARY AND UMPIRE ASSAY LABORATORIES BY YEAR

Year	Operator	Primary Assay Laboratory	Location	Umpire/Secondary Laboratory	Location
1998	Cascadia	Dunn Analytical	Ulaanbaatar	ALS Chemex	Ulaanbaatar
1999	Cascadia	Analabs	Ulaanbaatar	-	-
2000	Cascadia	Analabs	Ulaanbaatar	-	-
2001	Centerra	Analabs	Ulaanbaatar	ALS Chemex	Ulaanbaatar
2002	Centerra	Analabs	Ulaanbaatar	ALS Chemex	Ulaanbaatar
2003	Centerra	Analabs	Ulaanbaatar	Alex Stewart	Ulaanbaatar
2004	Centerra	SGS	Ulaanbaatar	Alex Stewart, AAL	Ulaanbaatar
2005	Centerra	SGS	Ulaanbaatar	-	-
2006	Centerra	SGS	Ulaanbaatar	-	-
2007	Centerra	SGS	Ulaanbaatar	-	-
2008	Centerra	-	-	-	-
2009	Centerra	SGS	Ulaanbaatar	Actlabs	Ulaanbaatar
2010	Centerra	SGS	Ulaanbaatar	-	-
2016	Centerra	SGS	Ulaanbaatar	ALS Mongolia	Ulaanbaatar

SGS ANALABS

The SGS Ulaanbaatar laboratory is accredited by the Mongolian Agency for Standardization and Metrology (MASM) to standards MNS 5373:2004 and CS11-0201:2013 for the analysis of gold by fire assay. The laboratory is also accredited to ISO 17025:2007; however, it could not be confirmed that the latter accreditation includes analytical packages used during this study. The current accreditation is valid from December 16, 2015 to December 16, 2019. Information about accreditation prior to this time frame was unavailable. All samples received by SGS (previously named SGS Analabs) are registered with an internal sample control number and entered into the laboratory information management system (LIMS). The LIMS program eliminates the need for manual data entry and reduces the chances of human transcription error. Samples are sorted and dried prior to sample preparation. Sample preparation begins with comminution in a Rhino/Terminator jaw crusher to a nominal 2 mm to 3 mm size. The crushed material is reduced in a Lab Tech Essa 201 8-bin rotary splitter to 750 g to 1,500 g and pulverized to 90% passing 75 µm in a LabtechLM1or LM2 bowl and ring pulverizer. Two splits of 300 g each are taken, with one allocated for assay and the other for archiving.

The crusher and pulverizer are flushed by compressed air after each sample. SGS Analabs assaying employs fire assay digestion on one assay ton aliquots and an atomic absorption finish on a 10 mL volume by a Varian Spectra 50A or 55B unit. The detection limit is 0.01 g/t or 10 ppb Au. The laboratory uses conventional flux and crucibles, fires 50 sample batches, and dissolves the prills by HNO_3/HCl.

ALEX STEWART ASSAYERS (ALS)

No information was available regarding the certification to ISO standards of the Alex Stewart laboratory. The process at the Alex Stewart preparation facility is generally similar to that at SGS Analabs. Samples are dried at 105ºC and crushed in a 160 mm by 100 mm jaw crusher to 95% passing -2 mm. The sample is reduced by an 11 gate riffle splitter to approximately 300 g before grinding in a Lab Tech Essa LM2 bowl and ring pulverizer to 95% passing -75 µm. Routine cleaning of equipment by compressed air between samples and silica cleaning between batches is performed to prevent contamination.

ALS fires batches of 50 samples. A 30 g or 50 g sample, depending on drill campaign and year, is fused at approximately 1,100°C with alkaline fluxes including lead oxide. During the fusion process, lead oxide is reduced to molten lead which acts as a collector for gold. When the fused mass is cooled the lead separates from the impurities (slag) and is placed in a cupel in a furnace at approximately 900°C. The lead oxidizes to lead oxide, being absorbed by the cupel, leaving a bead (prill) of gold, silver (which is added as a collector), and other precious metals. The prill is dissolved in aqua regia with a reduced final volume. Gold content is determined by Varian 55B flame AAS using matrix matched standards. For samples which are difficult to fuse, a reduced charge may be used to yield full recovery of gold.

SECURITY

The moderately difficult site access and distance from population centres offers security from the general public. Drill contractors provide their own camp facilities that are removed from the core handling and sample areas. The core handling and sampling facilities are adjacent to one another in a centralized area of the camp and thus are under observation of security personnel at all times. Processed core is prepared for transport and storage by placing plastic packing material under secured core box lids. Boxed drill core and bagged core samples are transported by a Centerra truck to a storage site at Boroo and to SGS in Ulaanbaatar for analysis under the supervision of a company geologist. Core, as well as assay coarse rejects and pulps returned from the assay laboratories, is stored under lock and key in the warehouse at the Boroo Mine site.

In the QP’s opinion, sample preparation, security, and analytical procedures utilized during drilling programs were conducted according to industry standards and are adequate for use in Mineral Resource estimation.

SAMPLING AND ANALYTICAL QUALITY CONTROL AND QUALITY ASSURANCE

An industry standard quality control and quality assurance (QA/QC) program has been implemented since the onset of drilling by Cascadia and has been continued under Centerra project management. Procedures for quality control have included the use of standard reference materials, duplicates (resampling of core), blanks submitted by Centerra, and checks on pulps at other laboratories. Quartered core duplicates have been submitted for assay (one quarter core is retained in the core box for reference and archive). Core duplicates are taken at 20 m to 40 m intervals down hole. Approximately one in every 50 pulps are re-assayed and many samples have an assay from a second pulp.

CENTERRA QA/QC

In 2001, field QA/QC comprised the inclusion of commercial blanks and reference standard pulps purchased from American Assay Lab and Minerals Exploration and Environmental Geochemistry in the sample stream at a rate of one in 10 for standards and one in 20 for blank material. Gold grades of the four standards ranged in grade from 0.068 g/t to 9.80 g/t.

In 2002, 22 pulp standards grading from 0.02 g/t Au to 25 g/t Au were used. Some were obtained from Rocklabs, Auckland, New Zealand, and Lovstrom & Associates of Tucson, Arizona, USA. Others were prepared by Shea Clark Smith from Minerals Exploration and Environmental Geochemistry of Reno, Nevada, by doping the standards with gold chloride. Some of the standards were created synthetically (Thalenhorst et al., 2004). A total of 161 pulps from the 2002 Gatsuurt drilling program were sent to American Assay Labs in Nevada, USA, where the check assays used a gravimetric finish for samples, yielding greater than 5 g/t Au. Thalenhorst (2004) reports that average values were comparable but the assay pairs did show some scatter.

Centerra reviews results from the analyses of the standards, blanks, and core duplicates. Centerra's protocol calls for batch re-assay if batch results are unacceptable for either two standards or one standard and one core duplicate. Since batches correspond to drill holes, core samples for the entire hole are rerun. In general, SGS Analabs has returned mainly acceptable results, with no systematic error or bias evident. In the few cases where batches have been rejected, error appears to have been random.

SGS ANALABS LABORATORY QA/QC

A certified standard, a blank sample, and a replicate assay are run with each sample batch. In addition, duplicate (second split) assays are run on every tenth sample to be used for checking the reproducibility of the assays. All results of control samples are graphed to monitor the performance of the laboratory. The warning limit used is two times the standard deviation and the control limit is three times the standard deviation. Any work order with a standard running outside the warning limit will have the selected re-assays performed, and any work order with a standard running outside the control limit will have the entire batch of samples re-analyzed. The level of bias and relative standard deviation (%RSD) are also reported. All QC data run with each work order are kept with the client's file. All QC graphs are available upon request and QA/QC reports are issued quarterly. The laboratory logs and reports on sample turnaround time.

QA/QC Protocols at SGS include:

- 10% QC run on analysis, so each batch of 50 runs consists of:
  - 43 samples
  - 1 duplicate (sample prep duplicate)
  - 1 blank (sample prep blank)
  - 2 standards
  - 2 replicates
  - 1 blank (method blank)

SGS Analabs uses four certified gold standards with grades of 0.55 g/t Au, 0.65 g/t Au, 3.68g/t Au, and 7.615 g/t Au. For October 2004, bias ranged from -1.82% to 2.83% (acceptable level) and %RSD varied from 3.08% to 4.80% (<-5% is acceptable).

SGS Analabs assay data for 1999 to 2001 were reviewed on behalf of Centerra by Ms. Lynda Bloom of Analytical Solutions Ltd. of Toronto in 2002. Data included both Gatsuurt and Boroo samples. Bloom (2002) concluded that results for gold standards assayed in 1999 were biased low; however, the performance in 2000-2001 improved. There was reasonable agreement between SGS Analabs and other Canadian laboratories (Lakefield Research, XRAL, Acme, and Swastika) for pulps at gold grades of 0.2 g/t Au to 8.0 g/t Au. A limited number of coarse rejects available at SGS Analabs, however, averaged 4% to 10% higher for gold assays in the range of 2.0 g/t Au to 40.0 g/t Au. This bias may not be significant due to the small number of samples and the high percentage of values > 40 g/t Au that may indicate problems in reject splitting, i.e., nugget effect. The latter may have been exacerbated by SGS Analabs using smaller samples for assaying high grade, a practice that was discontinued. Bloom (2002) made a number of recommendations in connection with QA/QC protocols and procedures by SGS Analabs and Centerra, laboratory reporting, and screen fire assays for high-grade samples (metallics) that were implemented.

ALEX STEWART QA/QC

Dr. Barry Smee of Smee and Associates Consulting Ltd., Sooke, B.C., audited work at Alex Stewart Assayers' laboratories, in both Ulaanbaatar and Bishkek. The Alex Stewart manager in Ulaanbaatar states that the Smee recommendations have been implemented. ALS QC protocol requires that each batch of 40 samples analyzed include a reagent blank, two replicate determinations, and two standard materials. Samples exhibiting anomalous concentrations (high or low) are routinely re-analyzed using either the original pulp or a second split. All routine replicate analyses are reported to the client.

SAMPLING VARIANCE

In order to assess overall sampling variance from core sampling to assaying, a comparison of assay data for original assays and core duplicate samples has been made for sampling from 2004 to 2016 results. A previous analysis of variance was completed by RPA in 2006 for the drilling from 1999 to 2004 (Hendry et al., 2006). RPA observed that there was a fair amount of sampling variability for the 2004 data with the observed trend of higher duplicate values relative to original sample values. RPA did note that duplicates were on quarter core, whereas the original sample is larger half core. The conclusion was that sample size difference may affect precision where free gold is present, since the smaller size may amplify the inherent nugget effect.

A comparison of duplicate samples to original assay samples from 2006 to 2016 show a progressive improvement in precision through the different drill campaigns (Figure 11-1); however, RPA's observation in 2006 with regard to sampling variability is still valid.

FIGURE 11-1 SCATTERPLOT OF ORIGINAL VERSUS DUPLICATE CORE SAMPLES

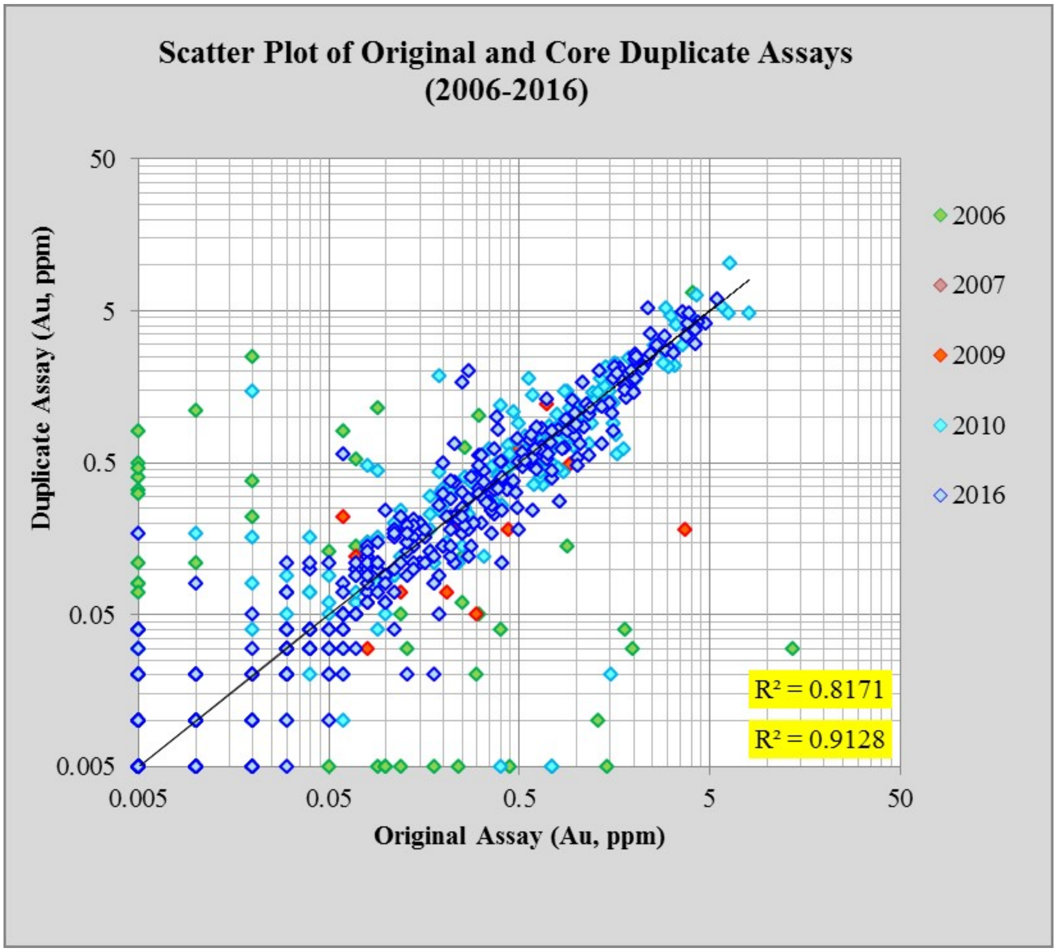
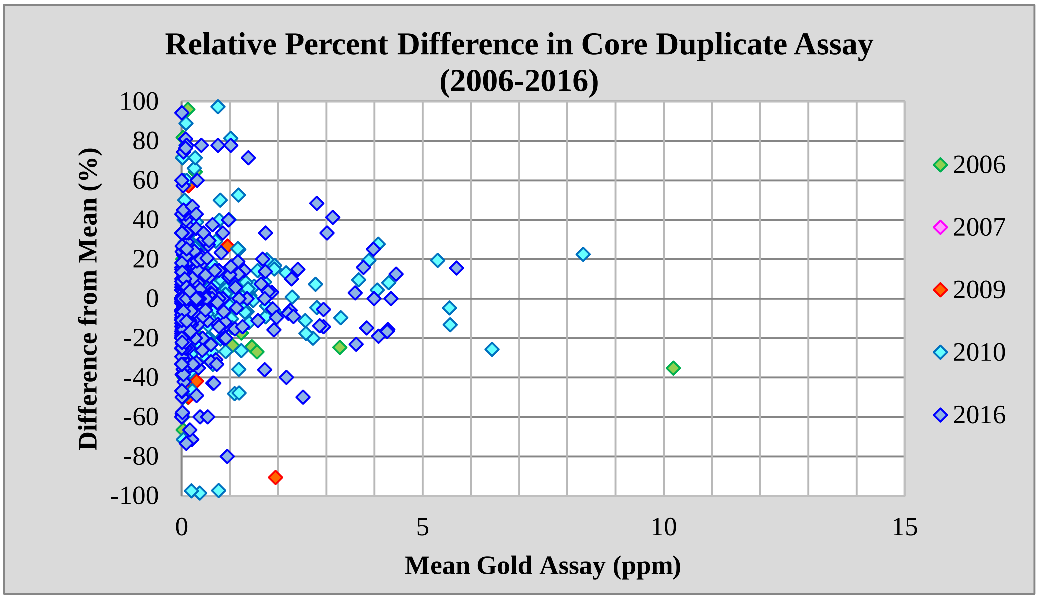
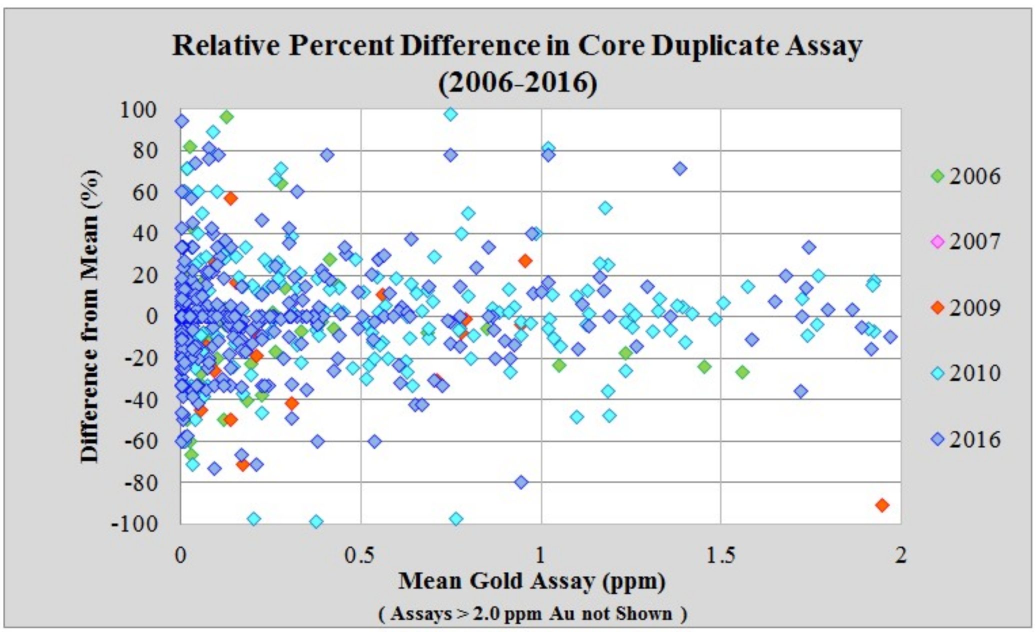


FIGURE 11-2 RELATIVE PERCENT DIFFERENCE IN CORE DUPLICATE ASSAYS





In the QP's opinion, the QA/QC program as designed and implemented by Centerra is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.

12 DATA VERIFICATION

Previous external data verification was carried out by RPA in preparation of the 2006 Technical Report (Hendry et al., 2006).

All drill hole data are initially stored and entered in Excel files and subsequently used in MapInfo and Geovia GEMS software. Both these software packages have data validation tools which include checks on survey data for spurious information, sample interval overlaps, missing sample intervals, and duplicate sample identification numbers. Additional data verification procedures are performed on a routine basis by Centerra personnel which include the check of the assay analyses for erratic or out of range values and 3D visual inspection.

It is Centerra's and RPA's opinion that the data used are valid and adequate and no limitations for the Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

INTRODUCTION

At the Gatsuurt Gold Project, gold mineralization occurs in the Main Zone and the Central Zone. Mineralization within the Central Zone is categorized as oxide, transition, or sulphide. The Main Zone is defined as having predominantly sulphide mineralization. Lower transition and sulphide mineralization in both zones is refractory. Therefore, the metallurgical properties of this mineralization bear important implications for the design of a suitable processing facility.

During 2005, as part of a feasibility study carried out by SNC-Lavalin, mineralogical and metallurgical studies were completed to investigate the metallurgical characteristics of the refractory gold mineralization and to develop design criteria for evaluating processing options. As part of these studies, a large scale concentrate biological oxidation pilot plant was initiated. Testing has also been completed on oxide and upper transition samples to define recovery of this free – milling ore.

Testwork completed up to 2006 is described in the “Technical Report on the Gatsuurt Gold Project, Northern Mongolia” dated May 9, 2006 (Hendry et al., 2006).

Testing completed to 2006 included:

- Mineralogical investigations
- Comminution testing
- Gravity recovery of gold
- Flotation testing
- Biological oxidation using BIOX® technology
- Leach testing

Since the publication of the May 9, 2006 report, additional testing has been performed to define cyanide destruction parameters on BIOX® leach tailings, leach testing on newly defined oxide resources and variability testing of the BIOX® process using small batch tests.

SAMPLES

2009 CYANIDE DESTRUCTION PROGRAM

A composite sample of available drill core (18 Central Zone holes and seven Main Zone holes) was assembled to provide approximately 165 kg of feed for the program. With the passage of the Water and Forest Law in 2009, no additional drilling was possible at Gatsuurt to obtain additional sample for testing. The grade of the composite sample for this program was higher than the average reserve grade as a result. The head grades for this sample are shown in Table 13-1.

TABLE 13-1 2009 CYANIDE DESTRUCTION PROGRAM SAMPLE FEED ASSAYS

Analysis	Unit	Central Zone/Main Zone Composite
Au	g/t	5.25
S_T	%	1.82
$S^{=}$	%	1.66
S^{o}	%	<0.5
Fe	%	3.11
As	%	1.11
Cu	g/t	20
Pb	g/t	<40
Zn	g/t	91
Carbonate	%	2.24
C (organic)	%	<0.05

2016 BIOX® VARIABILITY PROGRAM

In 2016, a core drill program was conducted to provide samples for metallurgical testing to assess the spatial and mineralization variability of the refractory areas of the Central Zone and the Main Zone. Thirteen HQ size holes were drilled that twinned previously drilled core drill holes. A total of 735 samples were submitted, representing four distinct lithologies.

Initially 79 composite samples were prepared for the testing program. These were then used to assemble 16 composite samples for metallurgical testing. The 16 composite samples are considered representative of the major lithological units and mineralization types in the

Central and Main Zones. The test program, including all assaying, was completed by SGS Lakefield Research (SGS Lakefield 2017b). The samples are shown in Table 13-2.

TABLE 13-2 SAMPLES FOR 2016 BIOX® VARIABILITY PROGRAM

Zone	Sample No.	Grade Range (g/t Au)	Mineralization	Rock Type	Percentage of Zone Resource
Central	9	>3	Upper Transition	Granite	6%
Central	16	>3	Upper Transition	Sandstone	1%
Central	14	>3	Fresh	PQSZ	3%
Central	1, 2, 3	>3	Fresh	Granite	16%
Central	12	>3	Fresh	Sandstone	4%
Central	5	1>/<3	Upper Transition	Granite	28%
Central	15	1>/ <3	Upper Transition	Sandstone	3%
Central	4	1>/ <3	Fresh Upper	Granite	30%
Central	13	1>/<3	Fresh Lower	Sandstone	7%
Main	10	>3	Transition	Rhyolite	12%
Main	6, 7	>3	Fresh	Rhyolite	19%
Main	8	1>/<3	Fresh	Rhyolite	35%
Main	11	1>/<3	Transition	Rhyolite	34%

Assays for the samples are shown in Table 13-3. Several of the Central Zone samples (1, 3, 12, and 16) returned head grades below expectations.

TABLE 13-3 ASSAYS FOR 2016 BIOX® VARIABILITY PROGRAM SAMPLES

Determination		Variability Composite No.															
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Au (Cut A)	(g/t)	2.04	3.37	2.82	2.11	1.36	3.52	3.99	0.41	3.50	19.00	1.22	2.22	1.88	6.33	1.49	1.48
Au (Cut B)	(g/t)	2.09	4.01	2.55	2.12	1.29	3.44	3.91	1.97	3.68	4.01	1.33	0.93	2.09	6.07	1.72	1.50
Au (Cut D)	(g/t)	2.13	1.91	2.71	1.90	1.48	3.62	4.20	1.97	4.33	5.02	1.32	1.11	1.88	6.83	1.61	2.07
Au (Cut E)	(g/t)	2.15	1.83	2.82	1.90	1.59	4.34	4.16	2.07	4.64	4.77	1.36	0.88	3.65	9.62	2.09	1.64
Au (Cut F)	(g/t)	2.20	2.32	2.72	1.76	1.32	3.61	4.04	1.99	4.32	4.80	1.38	0.93	1.35	9.92	1.56	2.76
Au (average)	(g/t)	2.12	2.69	2.72	1.96	1.41	3.71	4.06	1.68	4.09	7.52	1.32	1.21	2.17	7.75	1.69	1.89
Ag (Cut A)	(g/t)	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Ag (Cut B)	(g/t)	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Ag by AA Cut C)	(g/t)	0.8	1.0	0.6	0.8	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	1.2	0.8	1.4	1.2	<0.5
Fe	(%)	2.83	1.83	2.97	2.61	2.69	2.45	2.50	2.47	3.61	2.25	2.47	3.69	3.57	2.55	4.40	3.97
As	(%)	0.78	0.58	0.67	0.55	0.40	0.88	0.97	0.57	0.91	1.05	0.55	0.42	0.32	0.73	0.88	0.43
S(t)	(%)	2.42	1.37	1.74	1.70	0.90	1.58	1.59	1.03	2.09	1.45	1.00	0.62	0.73	1.65	1.06	0.90
S(=)	(%)	2.16	1.36	1.68	1.55	0.90	1.56	1.51	0.99	2.08	1.39	0.94	0.59	0.72	1.60	1.01	0.83
SO_4	(%)	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Hg	(g/t)	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3

MINERALOGY

Flotation concentrates, from four of the samples tested, were submitted for mineralogical evaluation. The samples selected were Nos. 3, 4, 6, and 8. The objectives of this evaluation were to determine:

(1) the bulk mineralogy of each sample,

(2) the gold deportment, including gold mineral speciation, grain size, liberation, and association, and

(3) estimation of overall gold extraction by a comprehensive mineralogical analysis, including assays.

The bulk mineralogy of each concentrate sample was determined with QEMSCAN (RMS, Rapid Mineral Scan) technology and X-ray Diffraction (XRD) bulk analysis including clay mineral speciation. The key findings from the bulk modal mineralogy by QEMSCAN are shown in Table 13-4. The results confirm that pyrite is the predominant sulphide mineral although significant arsenopyrite is found in each sample. The predominant mineral in the non – sulphide gangue is potassium feldspar.

TABLE 13-4 BULK MINERALOGY OF SELECTED FLOTATION CONCENTRATE SAMPLES

Sample ID		Comp. No. 3 Concentrate	Comp. No. 4 Concentrate	Comp. No. 6 Concentrate	Comp. No. 8 Concentrate
Mineral Mass (%)	Pyrite	14.6	27.2	20.4	15.9
	Arsenopyrite	6.91	11.1	18.2	14.9
	Other Sulphides	0.05	0.07	0.14	0.09
	Quartz	13.9	11.0	24.3	22.8
	K-Feldspar	38.0	30.0	13.0	25.2
	Plagioclase	9.56	9.41	0.76	2.28
	Muscovite/Illite	9.52	6.64	18.1	13.3
	Other Micas/Clays	1.72	0.49	0.67	0.64
	Other Silicates	0.20	0.16	0.14	0.12
	Oxides	1.15	0.97	1.49	1.33
	Carbonates	4.03	2.46	2.48	3.07
	Other	0.31	0.52	0.27	0.43

Gold deportment studies confirmed the refractory nature of the sulphide mineralization in the Central and Main Zones. Gold particles observed were all microscopic in size, with an average size of <5 µm. Shown in Table 13-5 are the results of gold particle analysis from each concentrate. Full results of the mineralogical investigation are found in SGS Lakefield Report “Gold Deportment Of Four Composite Samples From The Gatsuurt Project”, Project 15586-001A – Final Report dated May 9, 2017 (SGS Lakefield, 2017a).

TABLE 13-5 CONCENTRATE GOLD PARTICLE ANALYSIS

Sample ID	Au Grade (g/t)	Association	# of Observed Gold Grains	Size Range (µm)	Average Size (µm)	Au-Mineral Abundance	Minerals Associated with Exposed and Locked Au-Minerals
Comp No 3 Conc	**14.2**	Liberated Exposed Locked	67 142 243	0.6 - 29.9 0.6 - 20.6 0.6 - 12.9	4.7 3.9 2.8	Gold (99%), other (1%)	pyrite 87.8%, pyrite/silicate 5.91%, arsenopyrite 3.94%, and <2% pyrite/arsenopyrite and sphalerite.
			452	**0.6 - 29.9**	**3.4**		
Comp No 4 Conc	**17.5**	Liberated Exposed Locked	32 102 162	0.6 - 117.2 0.6 - 18.8 0.6 - 13.8	10.3 4.5 2.9	Gold (100%)	pyrite 81%, silicate/pyrite 13.2%, sphalerite/pyrite 2.24%, and <2% arsenopyrite, silicate, arsenopyrite/pyrite, and other elements.
			296	**0.6 - 117.2**	**4.3**		
Comp No 6 Conc	**33.2**	Liberated	1	4.7 - 4.7	4.7	Gold (100%)	pyrite 93.1%, arsenopyrite 3.59%, and pyrite/calcite 3.35%.
		Exposed	14	0.6 - 4.9	2.3		
		Locked	35	0.6 - 7.5	2.0		
			50	**0.6 - 7.5**	**2.2**		
Comp No 8 Conc	**22.8**	Liberated	5	1.6 - 8.7	3.6	Gold (100%)	pyrite 55.2%, arsenopyrite 44.7%, and pyrite/arsenopyrite <1%.
		Exposed	7	0.6 - 13.7	3.3		
		Locked	18	0.6 - 10.1	2.5		
			30	**0.6 - 13.7**	**2.8**		

OXIDE TESTING

2010 OXIDE LEACH TESTS

Additional leach tests were completed on exploration drill core in 2010 to provide additional recovery data for the South Slope part of the Central Zone. This area was a fairly recent discovery and no recovery data was previously available. The additional leach tests were completed at Actlabs in Ulaanbaatar. The leach test procedure previously used for the 2004 Alex Stewart leach tests was maintained so that the results would be comparable. The program included 252 samples from 29 drill holes.

The 400-g samples were ground to 80% passing 75 µm and cyanide leached in bottles for 72 hours at 33% solids. The sodium cyanide (NaCN) concentration that was used was 1.0 g/L with pH maintained between 11.5 and 12.5. The solution values were assayed and the leach residue was analyzed by fire assay. The head assay was calculated from leach solution assay and tailings residue assay. NaCN and lime (CaO) consumption were monitored during the tests.

2016 OXIDE LEACH PROGRAM

Additional leach tests were completed on exploration drill and metallurgical drill hole core in 2016 and 2017 to provide additional recovery data for the Main Zone, North Pit, and the GT-49 Zone. The last two areas are fairly recent discoveries and no recovery data was available. Historically, the Main Zone was considered to have minimal oxide resources. A re-evaluation of the drill core and resource model showed some oxide resources, but only minimal oxide recovery data. The additional leach tests were completed at SGS in Ulaanbaatar. The leach test procedure previously used for the 2004 Alex Stewart leach tests and the 2009 Actlabs procedure was maintained so that the results would be comparable. The program included 837 samples from 63 drill holes.

OXIDE RECOVERY MODEL

Oxide recoveries (1,969 results) were included in the block model. Wireframes at a 65% leach recovery cut-off were created to capture the subset of data that had similar recovery characteristics. The recoveries were then interpolated into the block model, however, the short estimation ranges required the use of default values. As a result, recovery defaults were applied both internal to the wireframes for blocks beyond the estimation ranges as well as outside of the wireframes to ensure that every oxide block with estimated gold grades also had a companion leach recovery.

SULPHIDE TESTING

2016 BIOX® VARIABILITY PROGRAM AND RECOVERY MODEL

In 2016, a metallurgical test program to assess the variability of recovery in the Central and Main Zones was completed. The objective was to produce a recovery model that could be incorporated into the block model to better estimate recovery than the previous single recovery applied to both the Central Zone and the Main Zone. Details from this program are found in the SGS Lakefield Research Report "An Investigation into Gold Recovery From

Gatsuurt Deposit Samples", Project 15586-001A – Progress Report 1, June 16, 2017 (SGS Lakefield, 2017b).

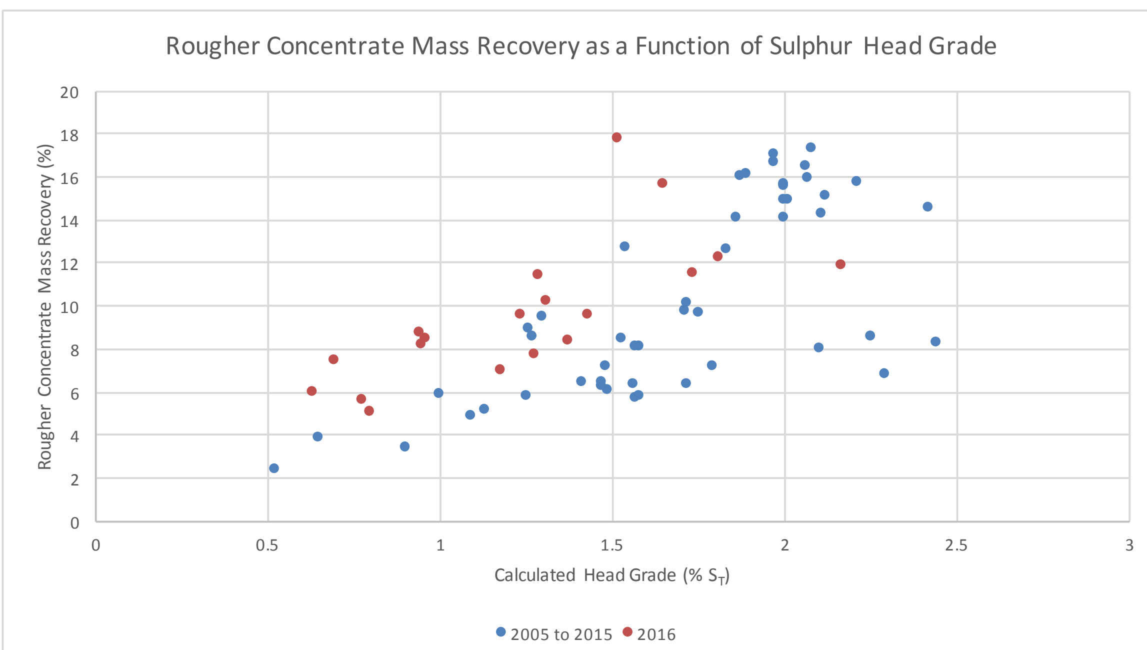
The testing program consisted of bench scale grinding, flotation, and batch BIOX® on flotation concentrates followed by leaching of the BIOX® products. Flotation and BIOX® procedures used were from previous programs completed in 2005, 2006, and 2009.

The test program included:

- Batch grinding of samples to 80% passing 75 μm.
- Flotation to produce a rougher concentrate.
- Batch BIOX® of each flotation concentrate. Tests were run for 5, 12, and 20 days. The 20 day tests were done in duplicate.
- CIL testing of each BIOX® product, including baseline tests on flotation concentrates. CIL tests were run for 48 hours with no intermediate sampling.

The flotation results show some variability, as would be expected. The results of the 2016 tests were analyzed in comparison to all previous tests. Figure 13-1 shows concentrate mass recovery as a function of sulphide sulphur head grade for the 2016 flotation tests and all previous tests (2005 to 2015).

FIGURE 13-1 BENCH SCALE FLOTATION CONCENTRATE MASS RECOVERY AS A FUNCTION OF SULPHUR HEAD GRADE



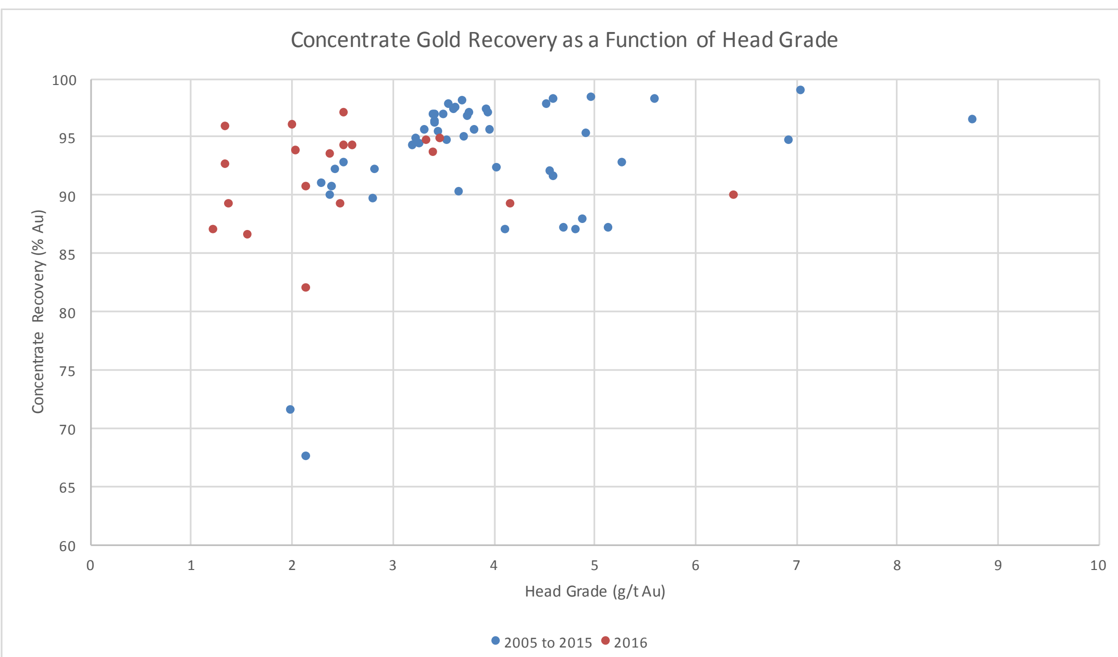
The 2016 results are at the upper end of mass recovery. The mass recoveries align with the design conditions selected for BIOX® design with a head grade of 0.84% $S^=$ which were:

- Design = 6.8%
- Maximum = 8.9%
- Minimum = 4.5%

The average concentrate $S^=$ grade of 12.2% essentially matches the BIOX® design feed grade of 12.0% established during engineering completed in 2010.

In terms of the recovery model, the impact of the low head grades from some of the Central Zone samples was assessed by looking at concentrate gold recovery as a function of head grade. As shown in Figure 13-2, the relationship is not that strong. The results from the variability testing would be suitable for use in the recovery model as is, without adjustment.

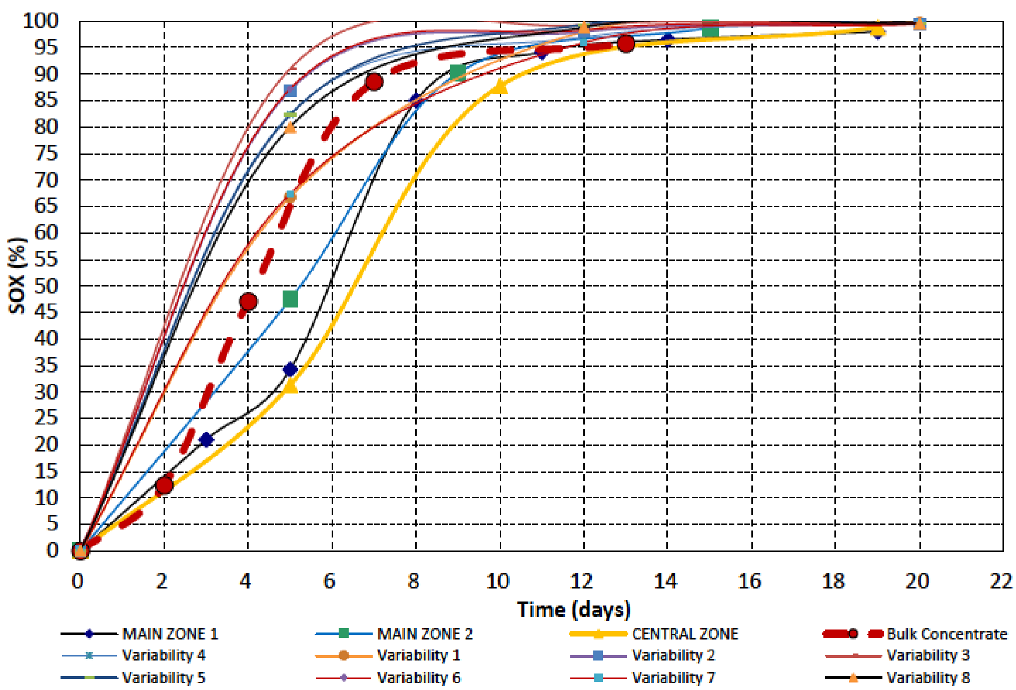
FIGURE 13-2 BENCH SCALE FLOTATION CONCENTRATE GOLD RECOVERY AS A FUNCTION OF HEAD GRADE



BioMin, (now part of Outotec) was engaged to assist with assessing the BIOX® and CIL test results. Outotec evaluated the results and broke the variability samples into two groups. The first group is variability sample Nos. 1 to 8 and the results are shown in Figure 13-3, along with the pilot plant results (Bulk Concentrate) and variability samples from 2006 (Central Zone, Main Zone 1 and 2). Outotec's observations include:

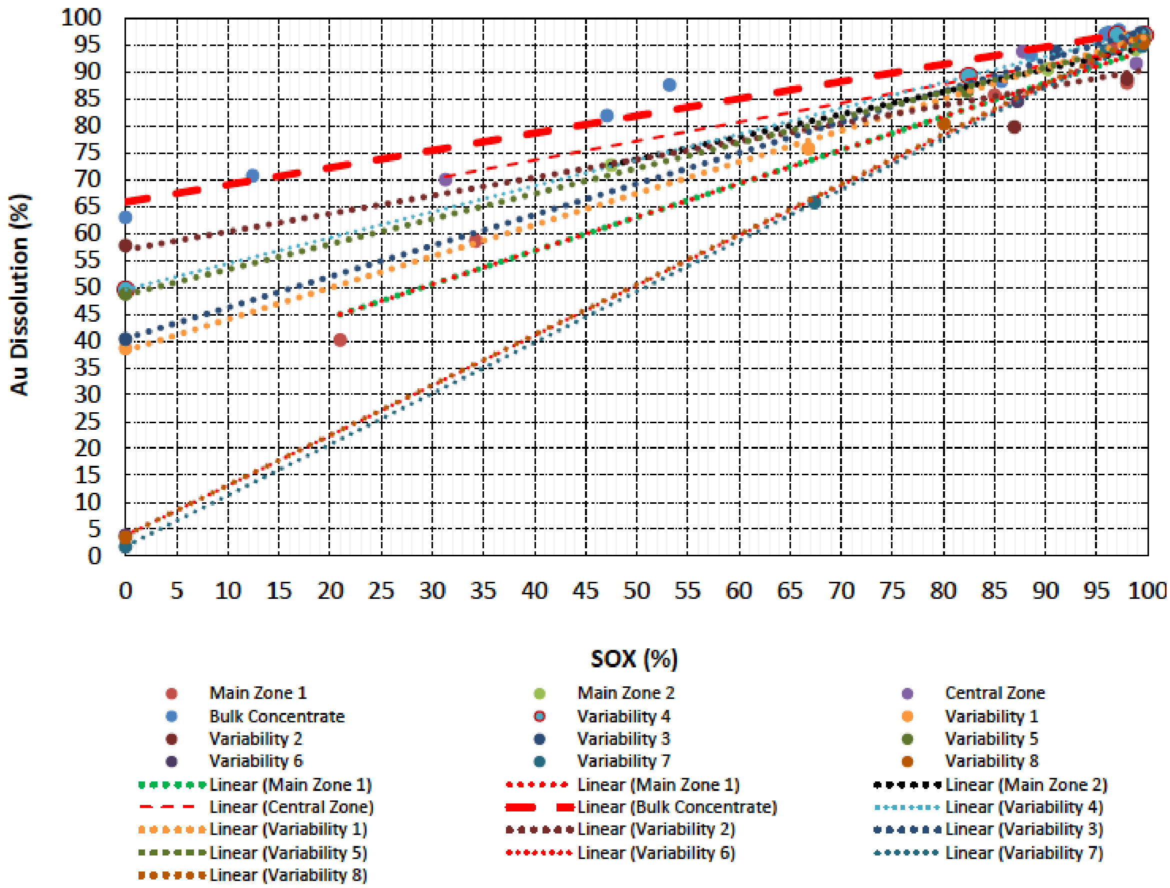
- Variability samples Nos. 1 and 7 showed slower initial sulphide oxidation rates (SOX), however, the rates increased mid-way through the oxidation period.
- All tests achieved similar results to the 2006 batch oxidation tests (96% SOX after 12 days).
- These samples should achieve similar results to the BIOX® pilot plant.

FIGURE 13-3 CONCENTRATE SULPHUR OXIDATION AS A FUNCTION OF TIME FOR VARIABILITY SAMPLES 1 TO 8



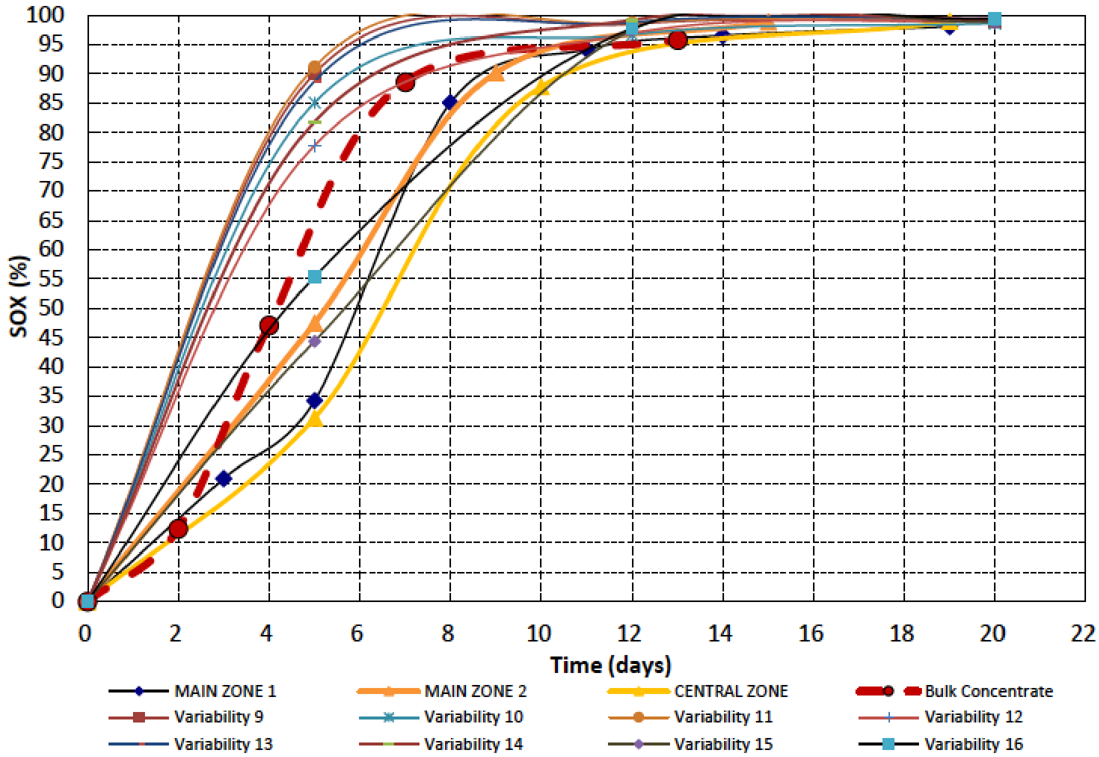
In terms of baseline gold recovery, all of the variability samples were more refractory than the 2005 Pilot Plant concentrate as shown in Figure 13-4. High SOX, 98% in 12 days oxidation, is required to achieve at least 95% gold recovery.

FIGURE 13-4 CIL GOLD RECOVERY AS A FUNCTION OF SULPHUR OXIDATION FOR VARIABILITY SAMPLES 1 TO 8



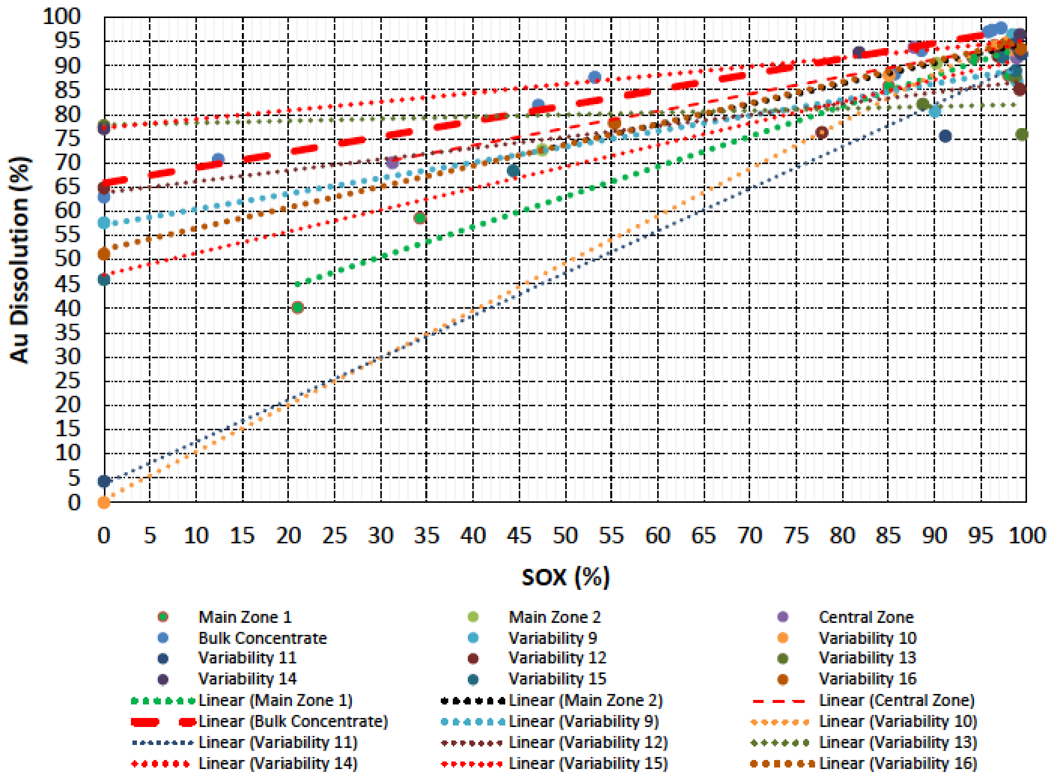
Similarly with the second group of samples, Nos. 9 to 16, all tests achieved >96% SOX after 12 days of oxidation. This compares quite well to the 2006 batch BIOX® test results. Sample Nos. 15 and 16 showed initial slower, more linear rates but ultimately achieved the same SOX. The batch SOX results and comparisons are shown in Figure 13-5. These samples achieved 98% SOX after 13 days of oxidation.

FIGURE 13-5 CONCENTRATE SULPHUR OXIDATION AS A FUNCTION OF TIME FOR VARIABILITY SAMPLES 9 TO 16



In terms of subsequent gold recovery, all samples (Nos. 9 to 16) were more refractory than the 2005 Pilot Plant composite with the exception of sample No. 14. Sample No.'s 13 and 15 achieved lower recoveries than the other samples in this group. For this sample group, gold recovery of 93% will be achieved after 98% SOX at 13 days. Figure 13-6 shows the relationship of gold recovery to SOX for this sample group.

FIGURE 13-6 CIL GOLD RECOVERY AS A FUNCTION OF SULPHUR OXIDATION FOR VARIABILITY SAMPLES 9 TO 16



In summary, the recommended CIL recoveries are 95% for variability sample No.'s 1 to 8 and 93% for sample No.'s 9 to 16.

The recommended sulphide recovery model for use in the block model, based on the 2016 variability samples, is shown in Table 13-6.

TABLE 13-6 GATSUURT SULPHIDE RECOVERY MODEL

Sample	Flotation Recovery (% Au)	Leach Recovery (% Au)	Overall Recovery (% Au)
1	90.6	95.0	86.1
2	93.7	95.0	89.0
3	89.2	95.0	84.7
4	89.2	95.0	84.7
5	86.9	95.0	82.6

Sample	Flotation Recovery (% Au)	Leach Recovery (% Au)	Overall Recovery (% Au)
6	94.7	95.0	90.0
7	94.6	95.0	89.8
8	95.9	95.0	91.1
9	93.6	93.0	87.0
10	89.2	93.0	82.9
11	92.6	93.0	86.1
12	95.8	93.0	89.1
13	97.1	93.0	90.3
14	90.0	93.0	83.7
15	86.5	93.0	80.5
16	82.0	93.0	76.2

CYANIDE DETOXIFICATION TESTING

In 2009, a cyanide destruction program was undertaken to provide design criteria to treat BIOX® CIL residue using the Boroo SO_2/air treatment equipment and to evaluate Caro's acid as an additional treatment step to remove thiocyanates so that tailings reclaim water could be recycled to Mill process water. Thiocyanates are toxic to the bacterial cultures used in BIOX® and are detrimental to the flotation process. A large composite sample of drill core from both the Central Zone and the Main Zone was assembled to generate sufficient flotation concentrate for both batch and continuous cyanide destruction testing.

The test program included:

- Flotation testing based on the 2006 SGS LRA Optimization Results
- Bulk flotation to generate concentrate for BIOX®
- Cyanidation of BIOX® product
- Batch and semi-continuous cyanide destruction of BIOX® cyanidation tailings using SO_2/air
- Polishing of SO_2/air products with Caro's acid to remove thiocyanate

The program was also an opportunity to validate the results from the BIOX® pilot plant work on an additional sample. Details of this program are found in the SGS Lakefield Research

Report "An Investigation into Gatsuurt Gold Deposit", Project 12345-001 Final Report May 2, 2011 (SGS Lakefield, 2011).

The results show the variability in head grades with the free gold content of the Gatsuurt deposit. The results of the optimized test showed a mass recovery to concentrate of 12.7% and a gold recovery of 92.2%. The high concentrate mass recovery is in agreement with the sulphur head grade of 1.5%, as shown in the relationship shown in Figure 13-1.

The BIOX® testing was carried out in three phases. The first two phases were to optimize conditions while the third phase consumed the remaining concentrate.

Initial cyanidation tests compared direct leaching of flotation concentrate to CIL of BIOX® products after 11 days and 25 days oxidation. The results are shown in Table 13-7.

TABLE 13-7 COMPARISON OF GOLD RECOVERY FROM FLOTATION CONCENTRATE AND BIOX® PRODUCTS

Feed	Calculated Head Grade (g/t Au)	Flotation Recovery (% Au)	BIOX® Duration (days)	Leach/CIL Recovery (% Au)	Overall Recovery (% Au)
Flotation Concentrate	4.60	92.2	-	35.9	33.1
BIOX® - 2	6.94	92.2	11	97.0	89.4
BIOX® - 1	4.04	92.2	25	98.0	90.2

The results confirm that the Gatsuurt sulphide ore is refractory with an overall gold recovery of 33.1%. Significant improvements in leach/CIL recovery were achieved with both 11 and 25 days sulphur oxidation. Both cyanide and lime consumptions were elevated by the sulphur oxidation.

The next phase of the test program was to optimize CIL conditions to generate bulk feed for cyanide destruction testing. The parameters evaluated include:

- Number of BIOX® product washes
- CIL cyanide concentration
- CIL retention time

The optimum conditions were determined to be:

- Four BIOX® product washes
- 1 g/L NaCN in CIL
- 48 h CIL retention time

Batch cyanide destruction tests were completed to determine conditions for the continuous tests. The requirement for discharge to the Boroo tailings management facility (TMF) is weak acid dissociable cyanide (CN_{WAD}) concentration of less than 1 mg/L. The results of the continuous runs are shown in Table 13-8.

TABLE 13-8 CYANIDE DESTRUCTION RESULTS

Test	Pulp Density	Ret. Time (min) Based on		Composition (Solution Phase)						CND Product Vol.	Cumulative Reagent Addition[2]						
		Reactor Volume	Actual Pulp Vol. [1]	pH	CN_T	CN_{WAD} CN_{Picric}	CNS	Cu	Fe		g/g CN_{WAD}			g/L Pulp		Cu	
	~%				mg/L	mg/L	mg/L	mg/L	mg/L	~L	SO_2 Equiv.	Lime	Cu	SO_2 Equiv.	Lime	mg/L Pulp	mg/L Soln
Test CIL-6 Pulp (42% Solids)				10.5	522	475	3400	4.21	8.38	--	--	--	--	--	--	--	--
Batch Test																	
CND-1	42	200	--	8.6	--	<1	--	--	--	--	7.16	1.74	0.11	2.70	0.66	40	50
Continuous Test																	
CND-2	42	138	73	8.5	45	2.1	--	3.35	15.20	3.3	5.11	0.35	0.09	1.93	0.13	32	41
CND-3	42	133	90	8.5	44	1.1	--	3.67	18.00	3.4	4.91	0.54	0.21	1.85	0.20	80	100
CND-4	42	229	169	8.5	53	1.5	--	3.36	34.80	4.4[3]	6.14	1.23	0.16	2.32	0.47	60	76

Notes:

(1) Calculated based on actual volume of pulp in reactor.
(2) Cu added as $CuSO_4$ $5H_2O$, SO_2 added as $Na_2S_2O_5$.
(3) Including pulp in reactor at end of test.

Batch test CND-1 achieved the target of <1 mg/L CN_{WAD}. Continuous test CND-3 came close to the target at 1.1 mg/L CN_{WAD}. The continuous tests were adversely affected by persistent foaming that developed in the reactor. The addition of de-foaming agents did not alleviate the problem. The additional reaction time provided in continuous test CND-4 did not result in lower CN_{WAD} concentration. The addition rate of SO_2 on the basis of g:g CN_{WAD} is in the range of typical industrial practice of 5:1, ranging from 4.9 to 6.1. There appeared to be iron stability issues with the BIOX® product, as dissolved iron (and total cyanide) appeared to increase during cyanide destruction. This presented difficulty in determining the copper addition rate to precipitate the iron and act as a catalyst. The iron instability was possibly from insufficient neutralization retention time prior to CIL.

The Boroo cyanide detox equipment provides ample retention time that will permit achieving the discharge limit of 1.0 mg/L CN_{WAD} despite the difficulties experienced in testing. The Caro's acid treatment of the cyanide destruction product was successful in reducing thiocyanate from 2700 mg/L to <2 mg/L. With the plan to store the BIOX® CIL tailings in a separate TMF, there will be no need to recycle water from this stream to the process.

It is the QP's opinion that the test samples are representative of the various types and styles of mineralization and the mineral deposit as a whole and that the test work is indicative of the predicted recoveries.

It is the QP's opinion that there are no known processing factors or deleterious elements that could have a significant effect on potential economic extraction.

14 MINERAL RESOURCE ESTIMATE

The Mineral Resource estimate was prepared by RPA and is a reasonable representation of the global Mineral Resources of the Project at the current level of sampling. The Mineral Resources conform to CIM (2014) definitions and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. Mineral Resources have been constrained within a preliminary pit shell. A summary of the Mineral Resources, exclusive of Mineral Reserves, is presented in Table 14-1.

TABLE 14-1 MINERAL RESOURCE SUMMARY, EXCLUSIVE OF MINERAL RESERVES, AS OF OCTOBER 31, 2017

Classification	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Indicated	10,988	1.9	678
Inferred	3,812	2.1	263

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability
3. The Mineral Resources have been estimated based on a gold price of US$1,450 per ounce.
4. Mineral Resources were estimated using variable cut-off grades based on processing, ore haulage, incremental mining cost, and G&A costs of $27.93/t and $41.44/t for CIP and BIOX®, respectively, estimated recoveries, and forecasted revenue of $42.50 and $42.26 per gram of gold for CIP and BIOX®, respectively.
5. Numbers may not add due to rounding.

Leapfrog Geo software (version 4.0) was used to construct the geological solids. GEOVIA GEMS software (version 7.6.3) was used to prepare assay data for geostatistical analysis, construct the block model, estimate gold and other metal grades, and tabulate Mineral Resources.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

MINERAL RESOURCE ESTIMATION METHODOLOGY

The evaluation of Mineral Resources involved the following procedures:

- Database compilation and verification;
- Definition of geostatistical resource domains and construction of wireframes;
- Data conditioning (capping and compositing) for geostatistical analysis and variography;
- Selection of estimation strategy and estimation parameters;
- Block modelling and grade interpolation;
- Validation, classification, and tabulation;
- Assessment of "reasonable prospects for eventual economic extraction" and selection of reporting assumptions;
- Preparation of the Mineral Resource Statement.

RESOURCE DATABASE

The database used to estimate resources includes 695 boreholes (94,267.6 m) completed between 1998 and 2016. Of the 695 holes, 65 holes (7,006.5 m) were drilled for metallurgical testing or other purposes, and are not appropriate for resource estimation purposes. Domains in the Main Zone are informed by 122 boreholes (18,485 m), while domains in the Central Zone were informed by 416 boreholes (57,784 m).

Exploration boreholes comprise core and reverse circulation (RC) holes, as well as a combination of the two. Table 14-2 summarizes the records in the borehole database used to model Mineral Resources.

TABLE 14-2 SUMMARY OF DATABASE RECORDS

Table	Number of Records
Collars	695
Gold Assay	48,303
Lithology	12,631
Oxidation from Logs	2,817
ICP	12,787
Composites at 2 m Intervals	17,821

Table	Number of Records
Density	927
Solid Intervals of Grade Shells	3,408
Solid Intervals of Weathering Wireframes	1,750
Bottle Roll Recovery	1,911

GEOLOGICAL INTERPRETATION AND MODELLING

Geological logs document rock type, weathering type, alteration type, structural features, and geotechnical characteristics. These data were used by Centerra geologists to complete a detailed geological data compilation to identify major geological contacts, mineralization, weathering, alteration, and structural features. This information was used to interpret the primary mineralized domains for the Main Zone and Central Zone at the Project.

WEATHERING MODEL

Using a combination of geological logs, core photos, and the percent of total sulphur (S%), a simplified weathering model was created consisting of oxide, transition, and fresh rock.

Within the Main Zone the contact surface between oxidized and fresh rock shows good correlation with low assay values of total sulphur; however, the correlation within the Central Zone is not as clearly defined. The interface between transition and fresh was typically defined by the absence of oxidation along fractures in the drill core. Centerra considers the current methodology to be acceptable to support the current Mineral Resource.

LITHOLOGICAL MODEL

Using a combination of geological logs and core photos, a simplified rock type model was created consisting of four main rock types (rhyolite, sandstone, granite, and overburden) as well as numerous smaller rhyolite dyke and diorite units. While the rhyolite and sandstone are generally modelled as single units, numerous granitic bodies have been modelled as individual sub-units which comprise the overall granite rock type.

STRUCTURE MODEL

Using geological logs and maps, a preliminary structural model was created, composed of the main Sujigtei fault as well as a small number of secondary and inferred fault structures. The main purpose of this model was to assist with geo-technical drill planning as well as guiding the interpretation of the lithological model and, in some instances, the mineralization model.

MINERALIZATION MODEL

The gold mineralization at the Project occurs primarily in fresh rock. Subordinate mineralization occurs near surface in oxidized and partly oxidized (transitional) rock. Individual grade domains at the CZ and MZ, with subsidiary zones in the South Slope (SS) and GT-60 areas, are well defined. The grade domains were reinterpreted to incorporate information from drilling completed in 2016. Grade domains were constructed to define and limit the volume of mineralized material and aid in coding of sample intervals within these volumes for accurate grade interpolation without grade smearing into areas thought to be waste. Similar to previous resource models, grade domains were constructed using cut-off grades of 0.4 g/t Au, 1.0 g/t Au, and 3.0 g/t Au, respectively. To construct these domains, mineralization displaying continuity above one of the three cut-off grades was combined into a three-dimensional solid referred to as a grade domain. The high-grade domain is contained within the medium-grade domain, which in turn is contained within the low-grade domain. No high-grade domain was constructed for the GT-60 area due to discontinuous high-grade mineralization. A low-grade domain was not constructed for the CZ due a lack of low grade intersections located outside the medium-grade domain. Some internal waste was captured within the low- and medium-grade domains to produce more coherent grade domains.

Most mineralization is oriented northeast-southwest with a steep to vertical dip. The SS mineralization, located beneath the south slope of the Gatsuurt River valley, is largely oriented sub-horizontally with variable dips and dip directions of individual domains. A small sub-vertical zone of mineralization was identified along a structure that broadly defines the contact between the SS and SSE granitoid bodies that host a large portion of the SS mineralization.

The mineralized domains are located in one of five spatial areas; each of which has its own orientation:

1. Main Zone (contains High-, Medium-, and Low-grade domain)

   The high-grade domain of this zone consists of an anastomosing vein network with a length of approximately 500 m and a down-dip extent of up to approximately 300 m. Individual veins reach thicknesses of approximately 30 m, but are typically up to approximately 5 m thick.

   The medium-grade domain comprises a main body juxtaposed to the Sujigtei fault on the southeast side and a small number of secondary bodies located immediately to the northwest and approximately 900 m to the southwest. The main body of this domain is approximately 750 m long and has a depth extent of approximately 350 m. Subsidiary veins occur parallel to the main body along the northern edge of the domain.

   The low-grade domain envelops all but the southwestern most medium grade shells in one contiguous domain. It measures approximately 1,150 m long and 370 m down dip with a thickness of up to approximately 130 m. Similar to the medium-grade domain, it is bounded to the southeast by the Sujigtei fault.

2. Central Zone, vertical orientation (contains High- and Medium-grade domain)

   The high-grade domain of this zone consists of an anastomosing vein network with a length of approximately 1,000 m and a down-dip extent of up to approximately 500 m directly adjacent to the Sujigtei fault. Individual domains reach thicknesses of approximately 25 m, but are typically up to approximately 5 m thick. There is a distinct break located roughly in the middle of this zone, resulting in two discontinuous vein systems.

   The medium-grade domain largely mirrors the extent and orientation of the high-grade domain. The dimensions are approximately 1,160 m along strike and up to 540 m in the down dip direction. This domain connects the two parts of the high-grade domain into one contiguous medium-grade shell.

3. GT-60, Central Zone, vertical orientation (contains Medium- and Low-grade domain)

GT-60 is located at the northeastern end of the Central Zone and has been interpreted to have been influenced strongly by the interaction of at least two fault directions. A high-grade zone was modelled. The medium-grade domain passes seamlessly into the Central Zone medium-grade domain but has a more northerly strike; additionally, veins strike increasingly north with increasing distance from the Sujigtei fault. Veins dip sub-vertical to vertical. A low-grade domain mirrors the general vein network of the medium-grade domain. The approximate overall dimensions of this zone are 380 m along strike, 230 m at its widest in the southwest, and 325 m deep.

4. South Slope, subhorizontal orientation (contains High-, Medium-, and Low-grade domain)

The South Slope mineralization is fully contained within two intrusive bodies called Granite SS and Granite SSE. Mineralization has been modelled not to cross between these two bodies. The orientation of mineralization in the SS area is less well defined than along the Sujigtei fault, however, the majority of mineralization in the SS area has been interpreted as largely flat-lying veins with variable dip. A zone of steeply dipping mineralization has been interpreted to follow largely the contact between the two granitoid bodies.

High-grade mineralization in the SS area comprises contiguous vein systems within the granitoid bodies as well as separate veins in Granite SSE; the latter being the deepest parts of this mineralized system.

Medium- and low-grade domains enclose higher grade domains while retaining overall shape and attitude. The thickness of individual low-grade vein envelopes can reach several tenths of metres, especially near junctions with other veins. In a number of areas, the high-, medium-, and low-grade domains share the same contact points.

5. South Slope South, subhorizontal orientation (contains High-, Medium-, and Low-grade domain).

The South Slope South area is located immediately southwest of the SS area and shares many of its characteristics, however, the orientation of mineralization steepens with increasing proximity to the Sujigtei fault, ultimately being sub-parallel to mineralization of the CZ. In this area, the division between the CZ and SSS is somewhat arbitrary.

Wireframe surfaces representing the current surface topography, base of overburden, and the redox surfaces were created from topographical surveys and borehole logs. These surfaces were used to limit the grade shells (base of overburden) and to code the mineralized blocks by oxidation state.

Wireframes in the MZ were built such that higher-grade domains created a void in the lower grade domain, whereas in the CZ wireframes were nested without creating voids in the lower grade domains. The former approach does not require recalculating block percentages contained within wireframes, whereas the latter approach does necessitate this additional step. The approaches were different for the two zones due to the relative higher complexity of the Leapfrog model of the CZ compared to that of the MZ.

Figure 14-1 shows a plan view of the 3.0 g/t Au grade shell models, drill holes, and 2013 CZ and MZ reserve pit designs.

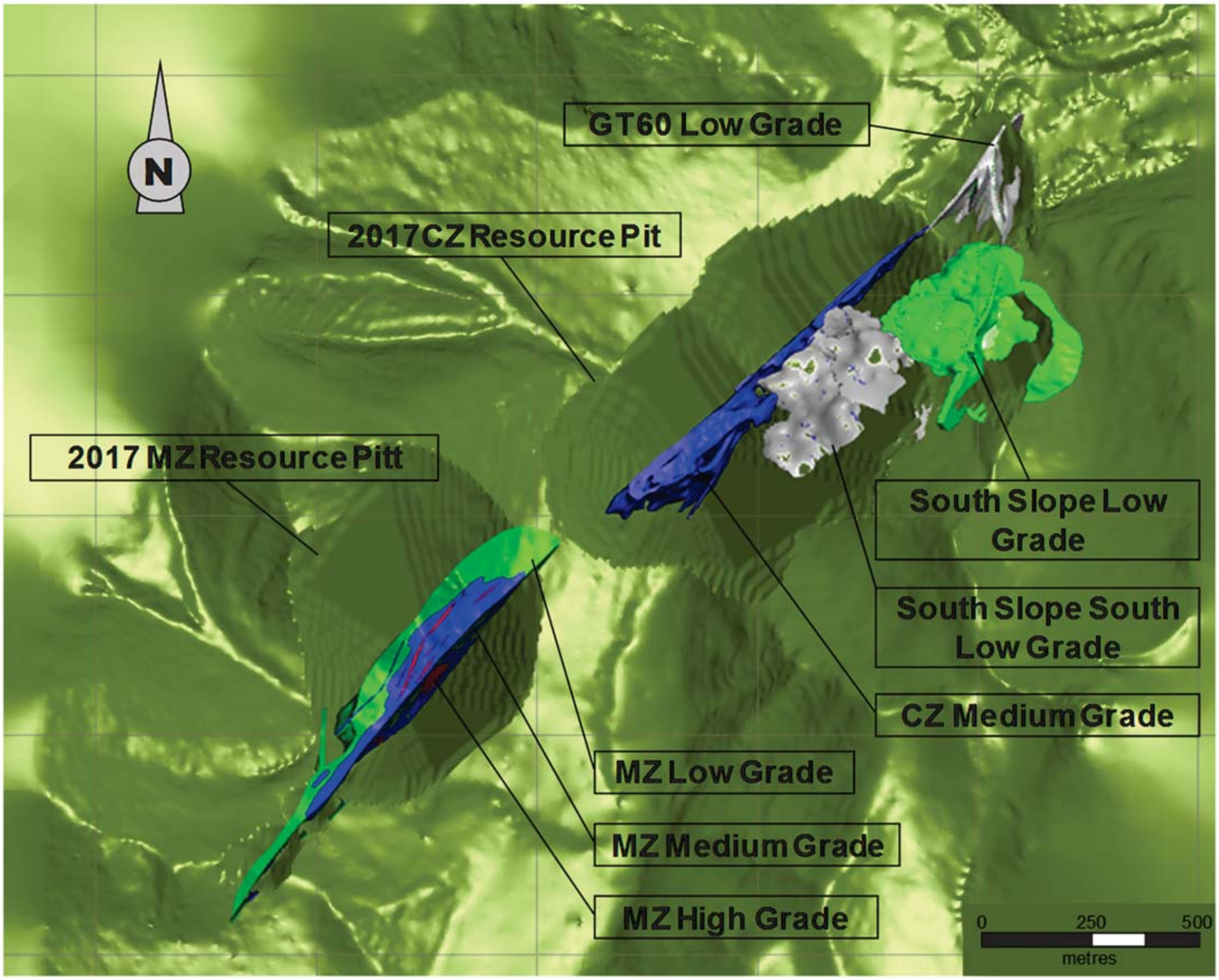


Figure 14-1

centerragold

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

centerragold mongolia

Plan View of the Grade Shell Models and Reserve Pit Design

Date: December 2017 File: Plan View.cdr

DENSITY

Density data were collected by Centerra from drill core samples taken every 10 m in mineralized material and 20 m in waste zones. Data analysis of these data suggests slight differences between mineralized and waste material as well as between oxidized, partly oxidized (transitional), and fresh material. In addition, an average bulk density of 1.90 t/m^3 was used for overburden. The average density values were assigned to the seven material classes as shown in Table 14-3.

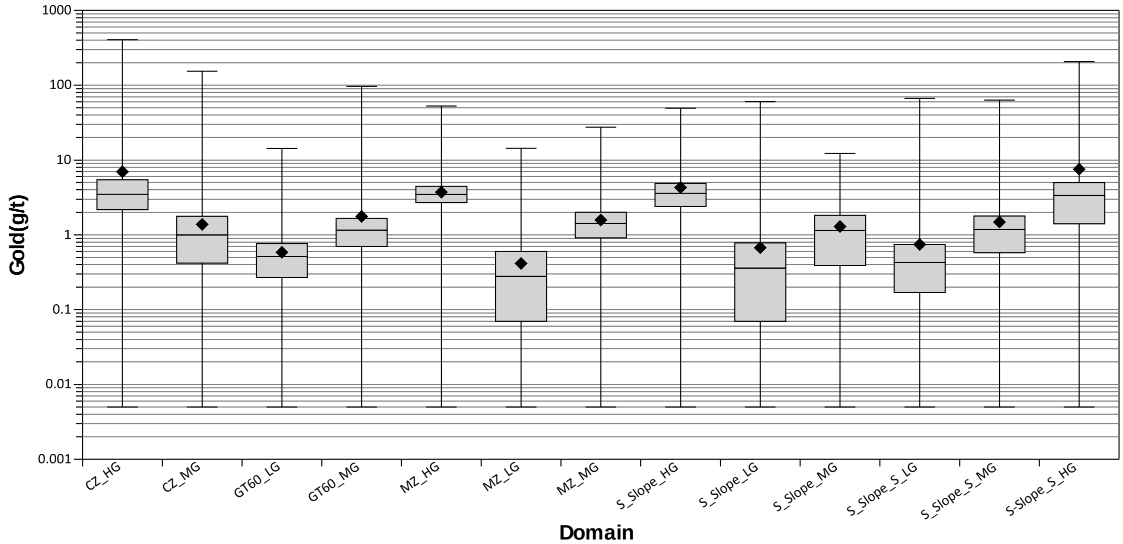
TABLE 14-3 DENSITY ASSIGNMENTS

Material	Mineralization Density (t/m^3)	Waste Density (t/m^3)
Overburden	NA	1.90
Oxidized	2.60	2.62
Transitional	2.68	2.70
Fresh	2.70	2.72

EXPLORATORY DATA ANALYSIS - ASSAYS

The first step in developing a block model estimate after completing 3D solid models is to assess the assay data contained inside the solid models and to determine whether any additional domaining is required prior to compositing. Typically, raw assay data are extracted from each domain and are then assessed using histograms and cumulative probability plots. Figures 14-2 and 14-3 show box plots and descriptive statistics for uncapped and capped gold assays, respectively within the various domains, while Figures 14-4 to 14-6 show this information for uncapped arsenic, iron, and sulphur assays. Arsenic and sulphur correlate with gold. The iron distribution displays very little spread within each zone, and its distribution appears to be independent from the gold, arsenic, and sulphur distribution. It is interesting to note that iron values are approximately one percent higher in the South Slope area compared to the South Slope South area.

FIGURE 14-2 BOX PLOTS OF UNCAPPED GOLD ASSAYS BY DOMAIN



Count	2,450	5,219	875	694	862	1,924	2702	388	1,331	755	2,231	1,316	295
Min	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
Max	406.00	154.00	14.20	96.50	52.90	14.40	27.60	49.30	60.20	12.20	66.50	63.30	207.00
Mean	6.97	1.38	0.58	1.76	3.72	0.42	1.58	4.30	0.68	1.29	0.74	1.48	7.55
Variance	366.40	11.95	0.48	21.22	5.90	0.28	1.99	21.16	3.21	1.55	6.49	6.22	355.30
StdDev	19.14	3.46	0.69	4.61	2.43	0.53	1.41	4.60	1.79	1.25	2.55	2.49	18.85
CV	2.75	2.50	1.19	2.61	0.65	1.28	0.89	1.07	2.66	0.96	3.42	1.68	2.50
Skewness	11.66	25.48	10.11	16.43	9.21	7.32	8.00	5.17	21.61	3.06	17.48	13.58	6.36
25thPerc.	0.21	0.03	0.01	0.12	0.64	0.01	0.06	0.05	0.01	0.01	0.01	0.05	0.08
Median	3.50	1.00	0.51	1.16	3.47	0.28	1.42	3.59	0.36	1.14	0.43	1.18	3.35
75thPerc.	5.44	1.78	0.76	1.67	4.46	0.60	2.01	4.85	0.78	1.83	0.74	1.78	4.99

Domain Codes

Main Zone

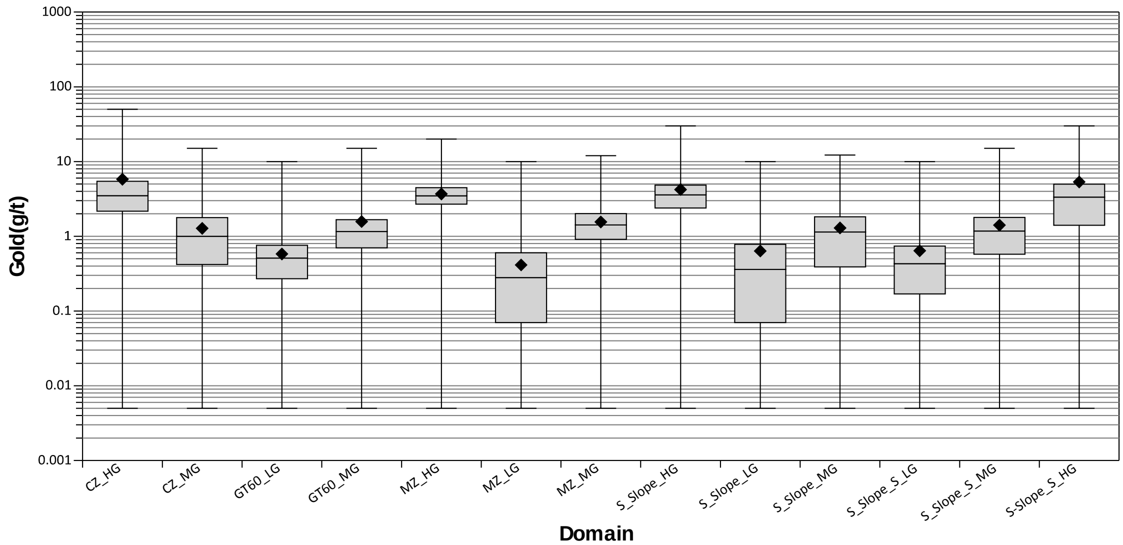
MZ_HG: Main Zone High Grade
MZ_MG: Main Zone Medium Grade
MZ_LG: Main Zone Low Grade

Central Zone

CZ_HG: Central Zone High Grade
CZ_MG: Central Zone Medium Grade
GT60_MG: GT60 Medium Grade
GT60_LG: GT60 Low Grade

S_Slope_HG: South Slope High Grade
S_Slope_MG: South Slope Medium Grade
S_Slope_LG: South Slope Low Grade
S_Slope_S_HG: South Slope South High Grade
S_Slope_S_MG: South Slope South Medium Grade
S_Slope_S_LG: South Slope South Low Grade

FIGURE 14-3 BOX PLOTS OF CAPPED GOLD ASSAYS BY DOMAIN



Count	2,450	5,219	875	694	862	1,924	2,702	388	1,331	755	2,231	1,316	295
Min	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
Max	50.00	15.00	10.00	15.00	20.00	10.00	12.00	30.00	10.00	12.20	10.00	15.00	30.00
Mean	5.80	1.28	0.58	1.58	3.68	0.41	1.56	4.22	0.64	1.29	0.64	1.41	5.33
Variance	70.74	2.07	0.41	3.88	3.52	0.25	1.32	15.92	0.98	1.55	1.03	2.33	47.79
SteDev	8.41	1.44	0.64	1.97	1.88	0.50	1.15	3.99	0.99	1.25	1.02	1.53	6.91
CV	1.45	1.13	1.11	1.25	0.51	1.22	0.74	0.95	1.56	0.96	1.58	1.08	1.30
Skewness	3.78	5.16	8.17	4.56	2.95	4.44	3.61	3.71	4.45	3.06	5.89	5.16	2.48
25thPerc.	2.17	0.42	0.27	0.70	2.69	0.07	0.91	2.39	0.07	0.39	0.17	0.58	1.40
Median	3.50	1.00	0.51	1.16	3.47	0.28	1.42	3.59	0.36	1.14	0.43	1.17	3.35
75thPerc.	5.44	1.78	0.76	1.67	4.46	0.60	2.01	4.85	0.78	1.83	0.74	1.79	4.99

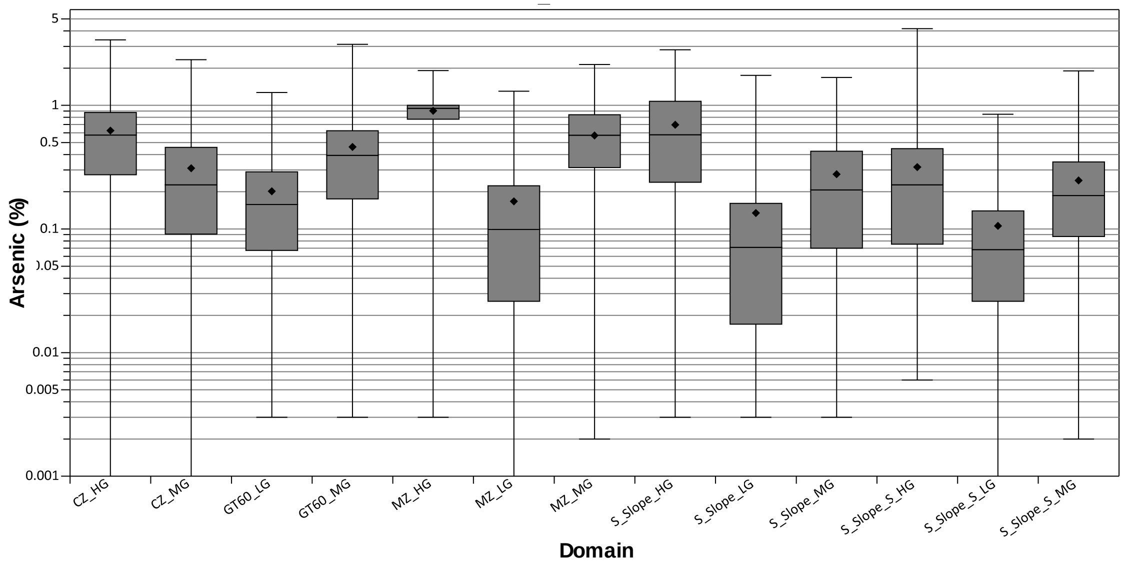
Domain Codes

Main Zone
MZ_HG: Main Zone High Grade
MZ_MG: Main Zone Medium Grade
MZ_LG: Main Zone Low Grade

Central Zone
CZ_HG: Central Zone High Grade
CZ_MG: Central Zone Medium Grade
GT60_MG: GT60 Medium Grade
GT60_LG: GT60 Low Grade

S_Slope_HG: South Slope High Grade
S_Slope_MG: South Slope Medium Grade
S_Slope_LG: South Slope Low Grade
S_Slope_S_HG: South Slope South High Grade
S_Slope_S_MG: South Slope South Medium Grade
S_Slope_S_LG: South Slope South Low Grade

FIGURE 14-4 BOX PLOTS OF UNCAPPED ARSENIC ASSAYS BY DOMAIN



Count	1,712	3,405	734	618	628	1,528	2,039	380	1,284	740	228	1,547	1,064
Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00
Max	3.39	2.34	1.27	3.12	1.91	1.30	2.14	2.82	1.75	1.68	4.17	0.85	1.90
Mean	0.63	0.31	0.20	0.46	0.90	0.17	0.57	0.70	0.13	0.28	0.32	0.11	0.25
Variance	0.19	0.08	0.03	0.14	0.06	0.04	0.10	0.29	0.04	0.07	0.15	0.01	0.05
StdDev	0.43	0.28	0.18	0.38	0.25	0.20	0.32	0.53	0.20	0.27	0.39	0.12	0.22
CV	0.69	0.91	0.89	0.82	0.28	1.20	0.56	0.77	1.45	0.97	1.24	1.10	0.87
Skewness	1.10	1.42	1.46	1.61	-0.59	1.97	0.03	0.85	3.42	1.46	5.79	2.24	1.72
25thPerc.	0.27	0.09	0.07	0.18	0.77	0.03	0.31	0.24	0.02	0.07	0.08	0.03	0.09
Median	0.58	0.23	0.16	0.39	0.95	0.10	0.57	0.58	0.07	0.21	0.23	0.07	0.19
75thPerc.	0.88	0.46	0.29	0.62	1.00	0.22	0.84	1.08	0.16	0.43	0.45	0.14	0.35

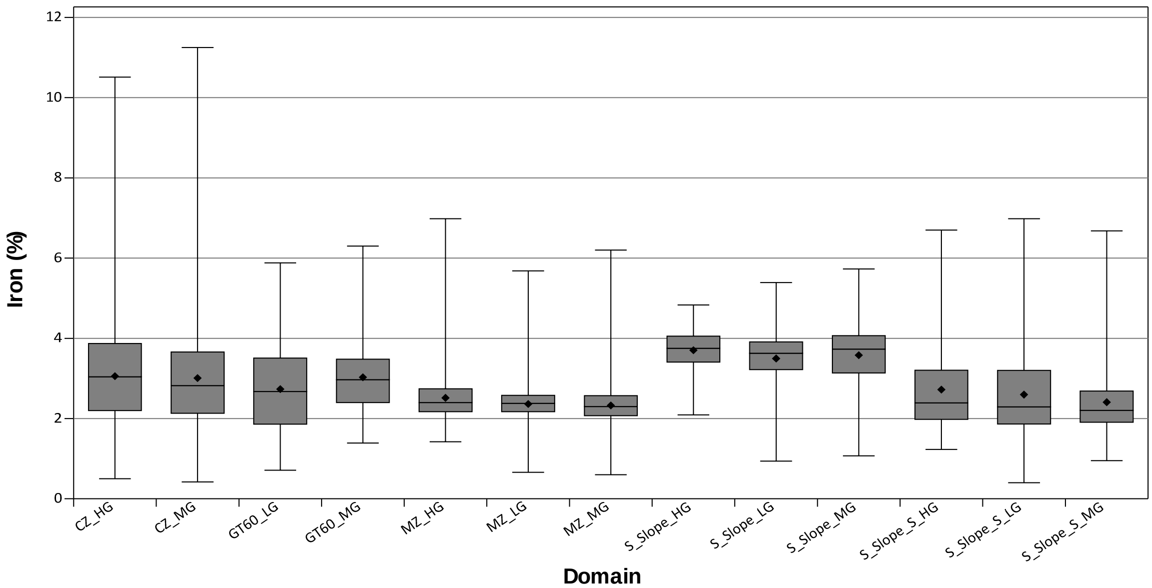
Domain Codes

Main Zone
MZ_HG: Main Zone High Grade
MZ_MG: Main Zone Medium Grade
MZ_LG: Main Zone Low Grade

Central Zone
CZ_HG: Central Zone High Grade
CZ_MG: Central Zone Medium Grade
GT60_MG: GT60 Medium Grade
GT60_LG: GT60 Low Grade

S_Slope_HG: South Slope High Grade
S_Slope_MG: South Slope Medium Grade
S_Slope_LG: South Slope Low Grade
S_Slope_S_HG: South Slope South High Grade
S_Slope_S_MG: South Slope South Medium Grade
S_Slope_S_LG: South Slope South Low Grade

FIGURE 14-5 BOX PLOTS OF UNCAPPED IRON ASSAYS BY DOMAIN

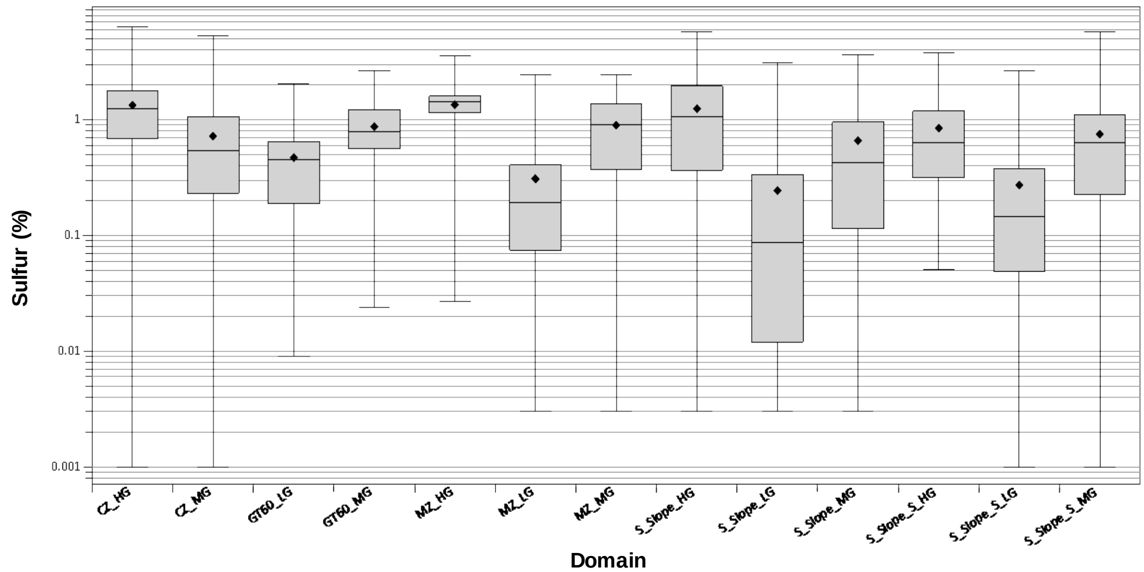


Count	1,400	2,849	99	100	491	977	1,610	194	391	319	147	905	567
Min	0.50	0.40	0.71	1.39	1.25	0.63	0.14	2.09	0.63	0.84	1.23	0.40	0.22
Max	11.11	12.30	5.88	6.31	6.98	6.03	6.71	5.96	7.27	5.73	7.60	7.49	7.21
Mean	3.05	3.01	2.73	3.03	2.52	2.36	2.33	3.70	3.50	3.58	2.73	2.59	2.41
Variance	1.45	1.48	1.26	0.78	0.41	0.24	0.35	0.43	0.55	0.79	1.29	1.20	0.74
Std Dev	1.20	1.22	1.12	0.88	0.64	0.49	0.59	0.65	0.74	0.89	1.14	1.10	0.86
CV	0.39	0.41	0.41	0.29	0.25	0.21	0.25	0.18	0.21	0.25	0.42	0.42	0.36
Skewness	0.61	1.08	0.37	0.90	2.72	0.59	1.35	-0.03	-0.76	-0.55	1.73	1.06	1.93
25thPerc.	2.15	2.10	1.79	2.41	2.16	2.16	2.06	3.31	3.21	3.10	2.04	1.82	1.85
Median	3.01	2.83	2.73	3.00	2.40	2.37	2.31	3.74	3.62	3.70	2.37	2.25	2.20
75thPerc.	3.88	3.68	3.52	3.54	2.73	2.58	2.57	4.03	3.93	4.08	3.13	3.27	2.72

Domain Codes

Main Zone		**Central Zone**			
MZ_HG:	Main Zone High Grade	CZ_HG:	Central Zone High Grade	S_Slope_HG:	South Slope High Grade
MZ_MG:	Main Zone Medium Grade	CZ_MG:	Central Zone Medium Grade	S_Slope_MG:	South Slope Medium Grade
MZ_LG:	Main Zone Low Grade	GT60_MG:	GT60 Medium Grade	S_Slope_LG:	South Slope Low Grade
		GT60_LG:	GT60 Low Grade	S_Slope_S_HG:	South Slope South High Grade
				S_Slope_S_MG:	South Slope South Medium Grade
				S_Slope_S_LG:	South Slope South Low Grade

FIGURE 14-6 BOX PLOTS OF UNCAPPED SULPHUR ASSAYS BY DOMAIN



	CZ_HG	CZ_MG	GT60_LG	GT60_MG	MZ_HG	MZ_LG	MZ_MG	S_Slope_HG	S_Slope_LG	S_Slope_MG	S_Slope_S_HG	S_Slope_S_LG	S_Slope_S_MG
Count	1,400	2,843	99	100	487	973	1,594	194	391	319	147	904	566
Min	0.00	0.00	0.01	0.02	0.03	0.00	0.00	0.00	0.00	0.00	0.05	0.00	0.00
Max	6.34	5.36	2.06	2.66	3.59	2.46	2.45	5.72	3.09	3.65	3.79	2.68	5.75
Mean	1.33	0.72	0.47	0.87	1.35	0.31	0.90	1.25	0.25	0.66	0.85	0.27	0.76
Variance	0.75	0.41	0.12	0.24	0.23	0.11	0.31	1.02	0.16	0.51	0.49	0.11	0.42
StdDev	0.86	0.64	0.35	0.49	0.48	0.34	0.55	1.01	0.40	0.71	0.70	0.33	0.65
CV	0.65	0.89	0.75	0.57	0.36	1.08	0.62	0.81	1.62	1.08	0.82	1.22	0.86
Skew ness	1.02	1.43	1.27	0.47	-0.36	1.93	0.07	0.74	3.56	1.55	1.58	2.26	1.59
25thPerc.	0.69	0.23	0.19	0.56	1.15	0.07	0.37	0.37	0.01	0.12	0.32	0.05	0.22
Median	1.24	0.55	0.45	0.79	1.42	0.19	0.90	1.06	0.09	0.43	0.63	0.15	0.63
75thPerc.	1.80	1.07	0.65	1.22	1.61	0.41	1.37	1.97	0.34	0.96	1.19	0.37	1.10

Domain Codes

Main Zone
MZ_HG: Main Zone High Grade
MZ_MG: Main Zone Medium Grade
MZ_LG: Main Zone Low Grade

Central Zone
CZ_HG: Central Zone High Grade
CZ_MG: Central Zone Medium Grade
GT60_MG: GT60 Medium Grade
GT60_LG: GT60 Low Grade

S_Slope_HG: South Slope High Grade
S_Slope_MG: South Slope Medium Grade
S_Slope_LG: South Slope Low Grade
S_Slope_S_HG: South Slope South High Grade
S_Slope_S_MG: South Slope South Medium Grade
S_Slope_S_LG: South Slope South Low Grade

Extreme high grade values, commonly called “outliers”, can lead to overestimation of grade in the block model. Histograms and probability plots were generated for each population and review determined that grade capping was required.

Table 14-4 shows the assay grade capping statistics, the selected capping values, and the theoretical metal loss. Typically, a metal loss of greater than 20% indicates that the capping level may be too severe as a significant amount of contained metal would be lost.

TABLE 14-4 ASSAY GRADE CAPPING AND METAL LOSS

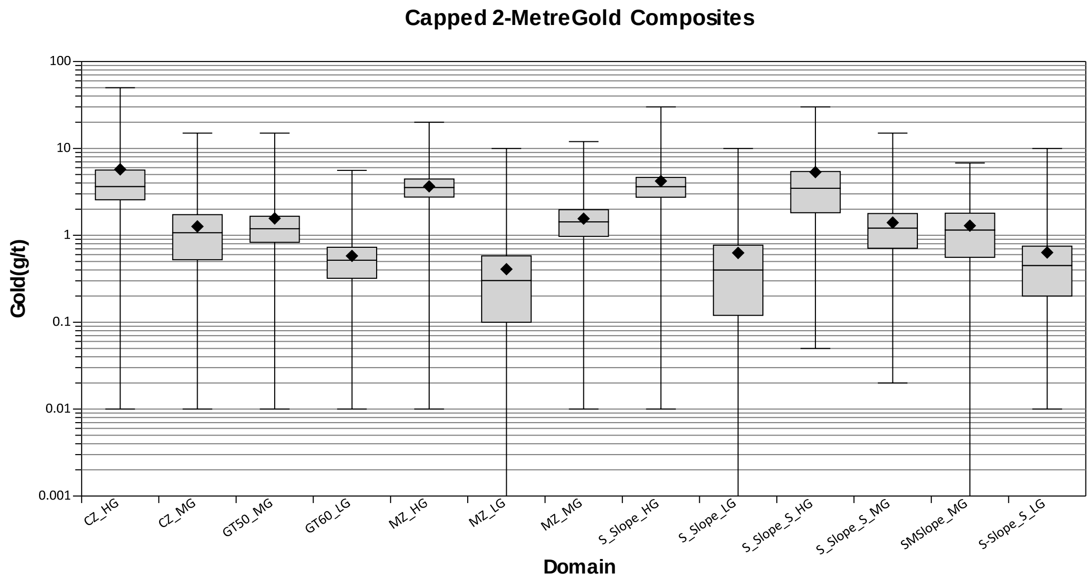
Grade Shell	Domain	Number of Assays	Raw Assay Mean	Raw Assay Cap	Capped Assay Mean	Number of Assays Capped	Metal Loss (%)
			(g/t Au)	**(g/t Au)**	**(g/t Au)**		
High Grade	MZ	862	3.72	20	3.68	4	-1.09
	CZ	2,450	6.97	50	5.8	41	-20.17
	South Slope	388	4.3	30	4.22	3	-1.90
	South Slope South	295	7.55	30	5.33	13	-41.65
	High-Grade Domains	**3,995**	**5.64**	**various**	**4.75**	**61**	**-18.74**
Medium Grade	MZ	2,702	1.58	12	1.56	7	-1.28
	CZ	5,219	1.38	15	1.28	20	-7.81
	GT60	694	1.76	15	1.58	3	-11.39
	South Slope	755	1.29	---	1.29	---	0.00
	South Slope South	1,316	1.48	15	1.41	5	-4.96
	Medium-Grade Domains	**10,686**	**1.5**	**various**	**1.42**	**35**	**-5.63**
Low Grade	MZ	1,924	0.41	10	0.41	1	0.00
	GT60	875	0.58	10	0.58	1	0.00
	South Slope	1,331	0.67	10	0.64	5	-4.69
	South Slope South	2,231	0.74	10	0.64	14	-15.63
	Low-Grade Domains	**6,361**	**0.61**	**10**	**0.58**	**21**	**-5.17**
All Grade Shells	**All Domains**	**21,042**	**2.5**	**various**	**2.19**	**117**	**-14.16**

Note: Assays are length weighted for mean grade calculations.

EXPLORATORY DATA ANALYSIS - COMPOSITES

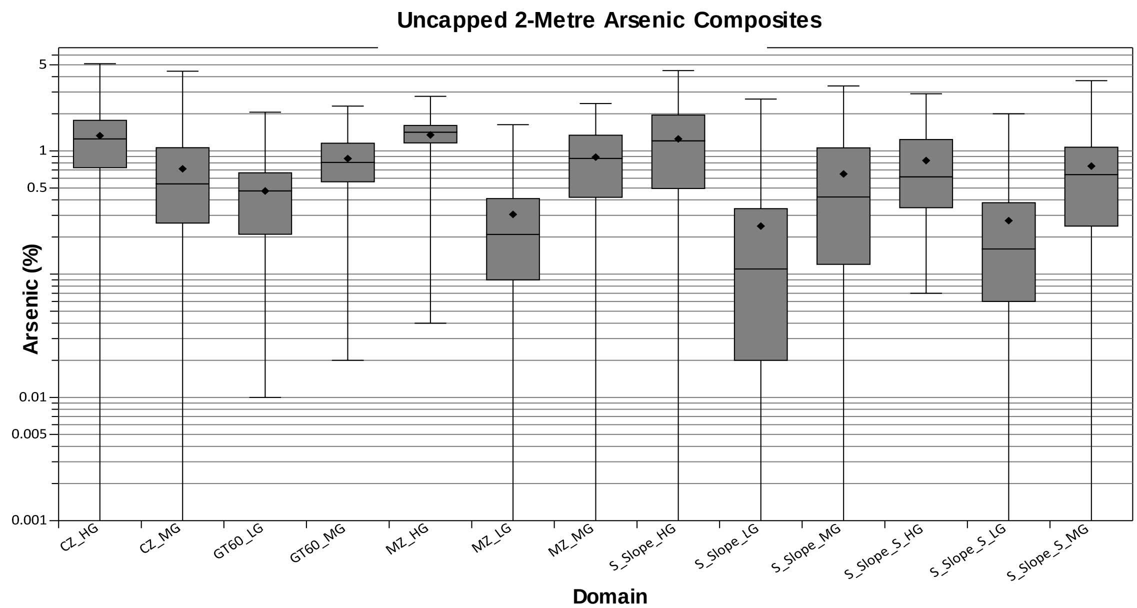
Prior to grade interpolation, the assay data within each of the individual mineralized grade shells were combined into two metre downhole composites. Figures 14-7 to 14-10 show box plots of the composite statistics for gold, arsenic, iron, and sulphur, respectively, for the various domains. Due to precision differences between the original data and data stored in GEMS (data in GEMS consider two decimals, while the original data considered three decimals), some very low-grade assays translated into zero-grade composites. This difference is considered to be not material.

FIGURE 14-7 BOX PLOTS OF 2.0 METRE CAPPED GOLD COMPOSITES BY DOMAIN



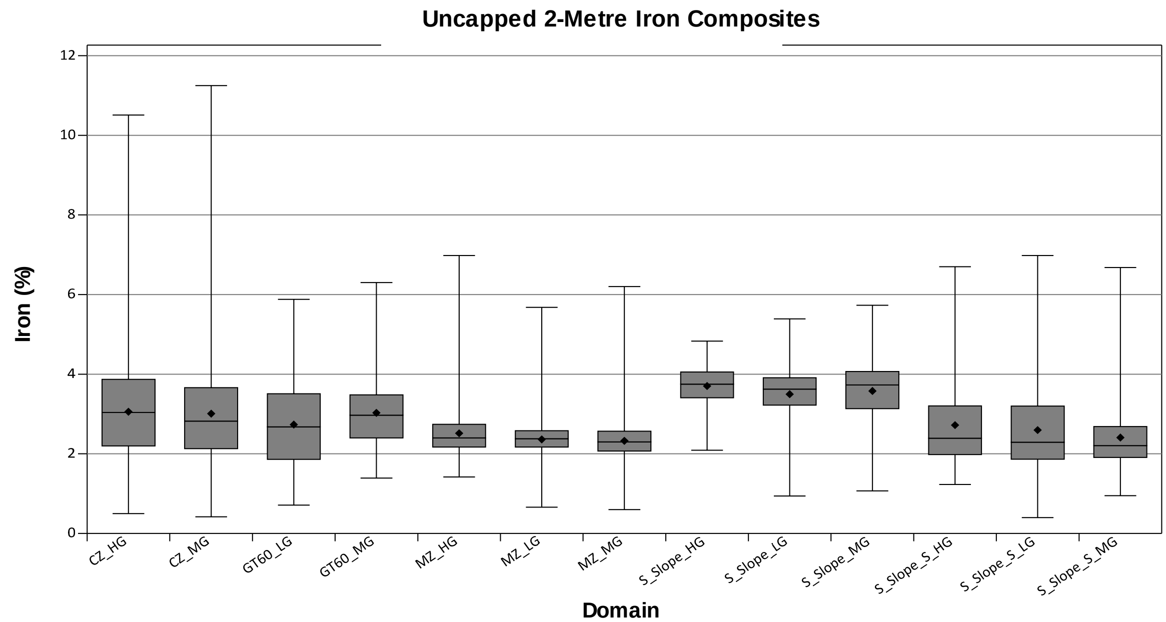
Count	1,690	3,902	543	754	667	1,904	2,355	265	1,080	216	926	564	1,783
Min	0.010	0.010	0.010	0.010	0.010	0.000	0.010	0.010	0.000	0.050	0.020	0.000	0.010
Max	50.00	15.00	15.00	5.59	20.00	10.00	11.99	30.00	10.00	30.00	15.00	6.78	10.00
Mean	5.75	1.27	1.57	0.58	3.66	0.41	1.56	4.21	0.63	5.33	1.40	1.29	0.64
Variance	50.78	1.44	2.82	0.27	2.67	0.18	1.09	11.97	0.80	38.03	1.85	1.05	0.69
StdDev	7.13	1.20	1.68	0.52	1.63	0.43	1.04	3.46	0.89	6.17	1.36	1.03	0.83
CV	1.24	0.95	1.07	0.89	0.45	1.05	0.67	0.82	1.43	1.16	0.97	0.79	1.30
Skewness	3.63	4.52	4.59	4.81	1.78	4.59	3.56	3.32	4.65	2.43	5.26	1.63	4.96
25th Perc.	2.56	0.52	0.83	0.32	2.76	0.10	0.97	2.75	0.12	1.82	0.71	0.56	0.20
Median	3.65	1.07	1.19	0.52	3.55	0.30	1.43	3.63	0.40	3.48	1.21	1.15	0.45
75th Perc.	5.64	1.73	1.66	0.73	4.44	0.58	1.97	4.65	0.77	5.43	1.78	1.80	0.75

FIGURE 14-8 BOX PLOTS OF 2.0 METRE UNCAPPED ARSENIC COMPOSITES BY DOMAIN



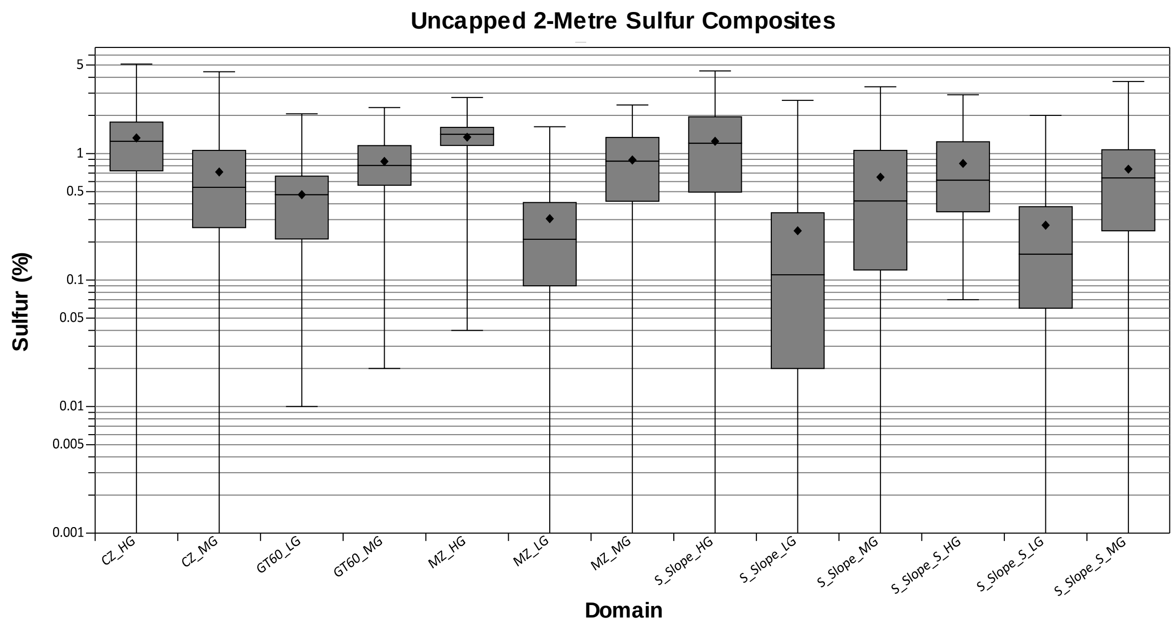
Count	1,170	2,563	638	478	456	1,515	1,734	258	1,028	551	168	1,239	750
Min	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.01	0.00	0.00	0.01	0.00	0.00
Max	3.31	1.99	1.00	1.90	1.66	1.09	1.77	2.10	1.70	1.47	3.91	0.69	1.27
Mean	0.63	0.31	0.20	0.46	0.90	0.17	0.57	0.70	0.13	0.28	0.31	0.11	0.25
Variance	0.17	0.07	0.03	0.11	0.05	0.03	0.09	0.23	0.03	0.06	0.15	0.01	0.04
StdDev	0.41	0.26	0.16	0.33	0.23	0.18	0.30	0.48	0.17	0.25	0.38	0.11	0.19
CV	0.65	0.85	0.81	0.73	0.26	1.08	0.53	0.70	1.30	0.88	1.22	1.00	0.78
Skewness	1.09	1.28	1.26	1.13	-0.65	1.82	0.05	0.60	3.41	1.34	6.07	1.96	1.37
25thPerc.	0.32	0.10	0.08	0.21	0.78	0.04	0.33	0.29	0.02	0.09	0.08	0.03	0.10
Median	0.59	0.24	0.17	0.41	0.94	0.11	0.57	0.62	0.08	0.23	0.22	0.07	0.20
75thPerc.	0.87	0.45	0.28	0.60	1.00	0.22	0.81	1.05	0.17	0.39	0.43	0.14	0.35

FIGURE 14-9 BOX PLOTS OF 2.0 METRE UNCAPPED IRON COMPOSITES BY DOMAIN



	CZ_HG	CZ_MG	GT60_LG	GT60_MG	MZ_HG	MZ_LG	MZ_MG	S_Slope_HG	S_Slope_LG	S_Slope_MG	S_Slope_S_HG	S_Slope_S_LG	S_Slope_S_MG
Count	965	2,163	103	90	376	1,100	1,449	124	314	226	104	719	396
Min	0.50	0.42	0.71	1.39	1.42	0.66	0.60	2.09	0.94	1.07	1.23	0.40	0.95
Max	10.51	11.25	5.88	6.30	6.98	5.68	6.20	4.83	5.39	5.73	6.70	6.98	6.68
Mean	3.06	3.01	2.73	3.03	2.52	2.36	2.33	3.70	3.50	3.58	2.72	2.60	2.41
Variance	1.34	1.37	1.18	0.73	0.37	0.22	0.32	0.32	0.48	0.73	1.10	1.10	0.65
StdDev	1.16	1.17	1.09	0.85	0.61	0.47	0.56	0.57	0.70	0.85	1.05	1.05	0.81
CV	0.38	0.39	0.40	0.28	0.24	0.20	0.24	0.15	0.20	0.24	0.38	0.40	0.34
Skew ness	0.56	1.07	0.38	0.93	2.76	0.30	1.14	-0.54	-1.15	-0.49	1.51	1.08	1.96
25thPerc.	2.20	2.13	1.86	2.40	2.17	2.17	2.07	3.41	3.22	3.14	1.98	1.86	1.91
Median	3.04	2.82	2.67	2.97	2.40	2.38	2.30	3.75	3.63	3.73	2.39	2.29	2.20
75thPerc.	3.87	3.66	3.51	3.48	2.74	2.58	2.57	4.05	3.91	4.07	3.21	3.20	2.69

FIGURE 14-10 BOX PLOTS OF 2.0 METRE UNCAPPED SULPHUR COMPOSITES BY DOMAIN



Count	965	2,154	103	90	372	1,092	1,427	124	314	226	104	717	395
Min	0.00	0.00	0.01	0.02	0.04	0.00	0.00	0.00	0.00	0.00	0.07	0.00	0.00
Max	5.10	4.43	2.06	2.31	2.77	1.63	2.42	4.49	2.63	3.37	2.91	2.00	3.71
Mean	1.33	0.72	0.47	0.87	1.35	0.31	0.89	1.25	0.25	0.65	0.83	0.27	0.75
Variance	0.68	0.36	0.11	0.21	0.22	0.09	0.28	0.87	0.13	0.43	0.42	0.10	0.36
StdDev	0.82	0.60	0.33	0.46	0.47	0.30	0.53	0.93	0.36	0.66	0.65	0.31	0.60
CV	0.62	0.84	0.70	0.53	0.35	0.99	0.59	0.75	1.46	1.01	0.78	1.14	0.80
Skewness	0.91	1.39	1.27	0.31	-0.65	1.71	0.10	0.53	3.03	1.39	1.27	1.91	1.15
25thPerc.	0.73	0.26	0.21	0.56	1.16	0.09	0.42	0.50	0.02	0.12	0.35	0.06	0.25
Median	1.25	0.54	0.47	0.81	1.42	0.21	0.87	1.21	0.11	0.42	0.62	0.16	0.64
75thPerc.	1.77	1.06	0.66	1.15	1.61	0.41	1.34	1.95	0.34	1.06	1.24	0.38	1.07

The coefficients of variation (CV) for the composites in all domains are all less than 1.5 indicating that the data sets behave well and should produce reasonable gold grade block estimates.

BLOCK MODEL PARAMETERS

Compared to the previous block model, the block size was reduced from 5 m x 5 m x 10 m to 5 m x 5 m x 5 m. This change to a smaller block size was motivated by a desire to conform the block geometry to the thin and local variations found primarily within the high-grade domains where anastomosing networks of thin, gold-bearing veins prevail.

A rotated block model was created using GEMS, with a rotation angle of -48° (clockwise). The block model coordinates are based on the local UTM grid (Zone 48, WGS84).
Table 14-5 summarizes the block model definition.

TABLE 14-5 GEMS BLOCK MODEL DEFINITION

	Block Size (m)	Origin (m)	Block Count
X	5	614,000	350
Y	5	5,385,585	840
Z	5	1,535	160

The grades for each of the grade domains were populated into a percent block model. The final block model includes a fully diluted grade attribute that provides a single grade per block that takes into account the percent of all grade domains within that block.

VARIOGRAPHY AND GOLD GRADE ESTIMATION

Variogram analyses were completed for each domain. The data distributions within the domains do not show good grade continuity, and neither meaningful variograms nor correlograms could be produced. Gold distribution influenced by two or more mineralizing events and/or remobilization of gold within the mineralized envelope may have led to an overall distribution that is not amenable to variogram modelling. As a result, the inverse distance estimator was used. Variogram models that mirror search distances and orientations were used to populate blocks with values used for verification purposes.

Table 14-6 shows the search parameters for the MZ and CZ.

Assignment of grades to each mineralized block was done by using the defined search ellipsoid and ranges. The grade interpolation used an inverse distance estimator to the power of three (ID^3) of the capped two-metre long composites based on a minimum of three and a maximum of eight composites in the first, and a minimum of two and a maximum of eight composites in the second estimation run. For the first estimation run, a maximum of two samples per borehole was utilized, which forces each block to use composite samples from at least two boreholes to obtain a block estimate. Estimation was performed in two passes, with the first pass having the shorter ranges and the second pass having 1.5 to 2 times the search range of pass one. Hard boundaries were used between the low, medium, and high-grade domains and between different zones except between the CZ and GT60 and the subdomains of the South Slope South area.

In conjunction with the gold grade interpolation, the average distance (in metres) of all the samples used from the block being interpolated was recorded in a separate model to be used for block resource classification.

The block model was set up as a multi-folder percent model, in which blocks covering high-, medium-, and low-grade domains only were populated in respective folders. The percent of a domain inside a block was recorded in a separate attribute. After estimation, grades and percent were combined in a fourth folder such that the individual percent amounts of the low, -medium-, and high-grade folders were added and metal content was added weighted by the corresponding percent attribute. In order to account for minimum mining units, Whittle optimization was completed based on a 50/50 percent rule by which final blocks covering less than 50% of domains were regarded as 100% waste, while blocks with 50% or more domain coverage were regarded as 100% ore. Resources were reported based on the unified percent model.

TABLE 14-6 SUMMARY OF SEARCH PARAMETERS

Search Domain	Search Ellipse Orientation		Search Range		Search Domain	Search Ellipse Orientation		Search Range	
			Pass 1	Pass 2				Pass 1	Pass 2
CZ	Prin. Az.	90	40	75	South Slope South	Prin. Az.	20	40	75
	Prin. Dip	-90	40	75		Prin. Dip	0	40	75
	Inter. Az.	44	10	20		Inter. Az.		20	35
	Min Comps		3	2		Min Comps		3	2
	Max Comps		8	8		Max Comps		8	8
	Max Comps/Hole		2			Max Comps/Hole		2	
GT60	Prin. Az.	50	40	75	South Slope South Inclined	Prin. Az.	80	40	75
	Prin. Dip	-70	40	75		Prin. Dip	-40	40	75
	Inter. Az.	20	10	35		Inter. Az.	20	10	20
	Min Comps		3	2		Min Comps		3	2
	Max Comps		8	8		Max Comps		8	8
	Max Comps/Hole		2			Max Comps/Hole		2	

Search Domain	Search Ellipse Orientation		Search Range Pass 1	Search Range Pass 2	Search Domain	Search Ellipse Orientation		Search Range Pass 1	Search Range Pass 2
South Slope SS Granite	Prin. Az.	110	40	75	South Slope South Vertical	Prin. Az.	80	40	75
	Prin. Dip	-10	40	75		Prin. Dip	-60	40	75
	Inter. Az.	20	10	25		Inter. Az.	35	10	25
	Min Comps		3	2		Min Comps		3	2
	Max Comps		8	8		Max Comps		8	8
	Max Comps/Hole		2			Max Comps/Hole		2	
South Slope SSE Granite	Prin. Az.	110	40	75	MZ	Prin. Az.	40	40	75
	Prin. Dip	15	40	75		Prin. Dip	-90	40	75
	Inter. Az.	20	15	25		Inter. Az.	30	10	20
	Min Comps		3	2		Min Comps		3	2
	Max Comps		8	8		Max Comps		8	8
	Max Comps/Hole		2			Max Comps/Hole		2	
South Slope Vertical	Prin. Az.	290	40	75	MZ Low Grade	Prin. Az.	40	50	100
	Prin. Dip	80	40	75		Prin. Dip	-90	50	100
	Inter. Az.	25	10	20		Inter. Az.	30	20	35
	Min Comps		3	2		Min Comps		3	2
	Max Comps		8	8		Max Comps		8	8
	Max Comps/Hole		2			Max Comps/Hole		2	

IRON, ARSENIC, AND SULPHUR ESTIMATION

Iron, arsenic, and sulphur were also estimated to help determine the destination of the ore for processing, either CIP or BIOX®. A total of 12,787 assay pulps from at least 180 boreholes were submitted to either Alex Stewart Lab or American Assay Lab for multi-element inductively coupled plasma (ICP) analysis. The majority of these pulps were from the main part of the CZ and MZ, with limited pulps from the GT60, SS, and SSS areas (Figure 14-11).

The BIOX® plant requires steady sulphur feed grade and throughput and therefore ore blending for sulphur grade will be necessary. Iron and arsenic grades are monitored to ensure that arsenic will be stable after BIOX® processing. Both iron and arsenic are dissolved in the front end of the BIOX® process and then precipitated in the back end. A minimum Fe:As molar ratio of 3:1 is required to ensure stability. Based on testwork, sulphide values between 0.375% S and 0.5% S and lower are acceptable for CIP processing and anything greater than 0.5% S must use the BIOX® processing.

Both sulphur and arsenic have shown a reasonable correlation with gold inside the CZ grade shells. Based on this positive correlation, both elements were estimated using parameters similar to the gold estimation.

Extensive capping analyses for arsenic, sulphur, and iron showed that no capping was required for the estimation of these elements. Grades were estimated using a three-pass approach with ID^3 interpolation. A hard boundary was used between individual wireframes for all three accessory metals.

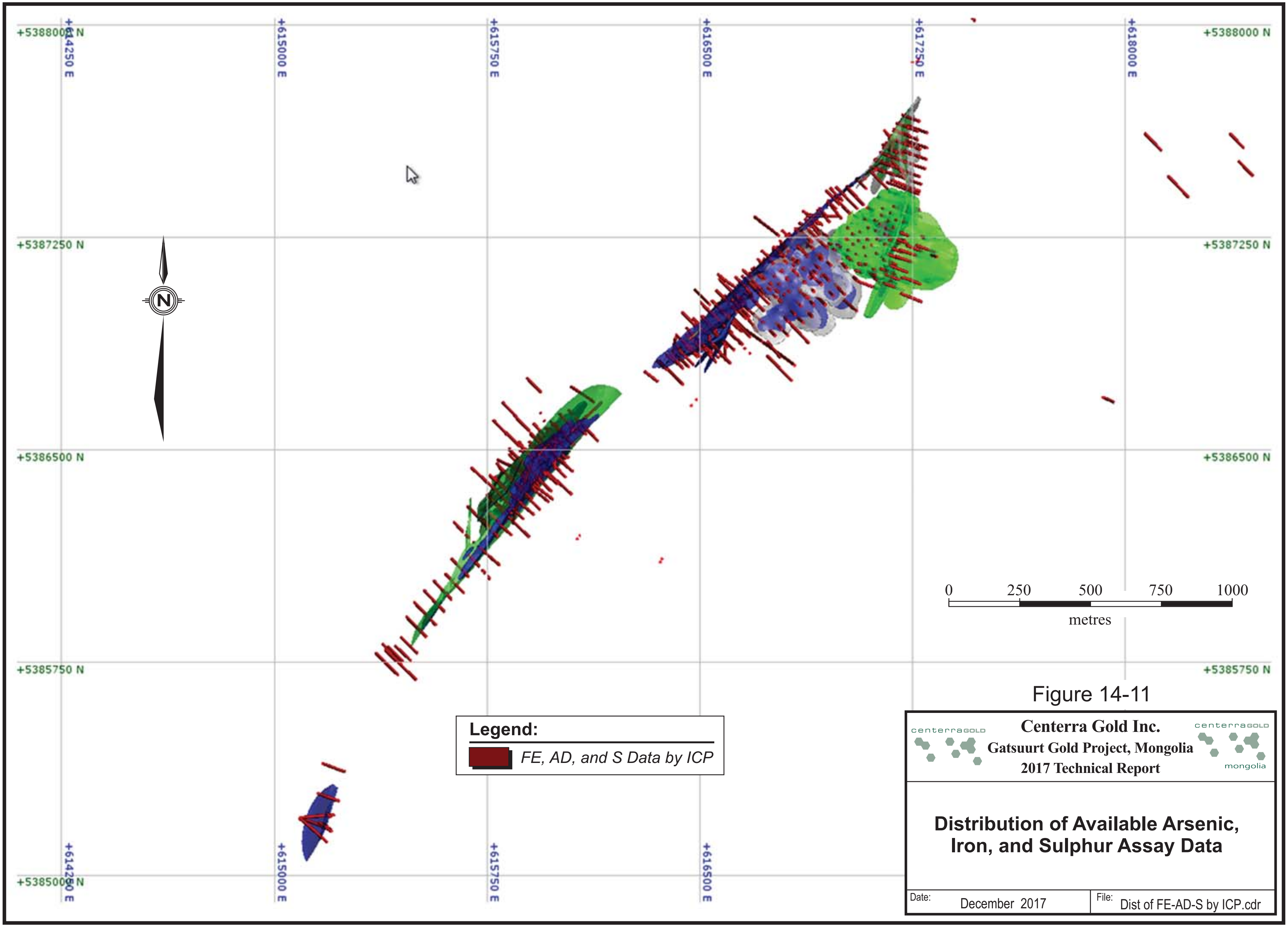


Figure 14-11

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Distribution of Available Arsenic, Iron, and Sulphur Assay Data

Date: December 2017

File: Dist of FE-AD-S by ICP.cdr

GOLD RECOVERY MODELS

Gold will be recovered using either a CIP or a BIOX® process, depending on the oxidation state of the mineralized host rock (fresh, transitional, or oxidized) and in the case of transitional material, on economic factors. The oxidation state of the rock was determined by Centerra geologists during visual core and RC chip logging.

Gold recovery in oxidized and some transitional material will use a CIP process. Testwork determined gold recoveries from these materials using a total of 1,911 bottle roll tests (Figure 14-12). Gold recoveries have been assumed using the bottle roll data in two passes using ID^3. Estimation parameters included search distances of 25 m and 100 m for the first and second pass, respectively, a minimum of one and maximum of six data, and soft boundaries between lithological rock types. Estimation parameters were selected so that no default values had to be used to populate blocks not populated during the estimation runs. Bottle roll recoveries were used to flag economical blocks in transitional and oxidized material.

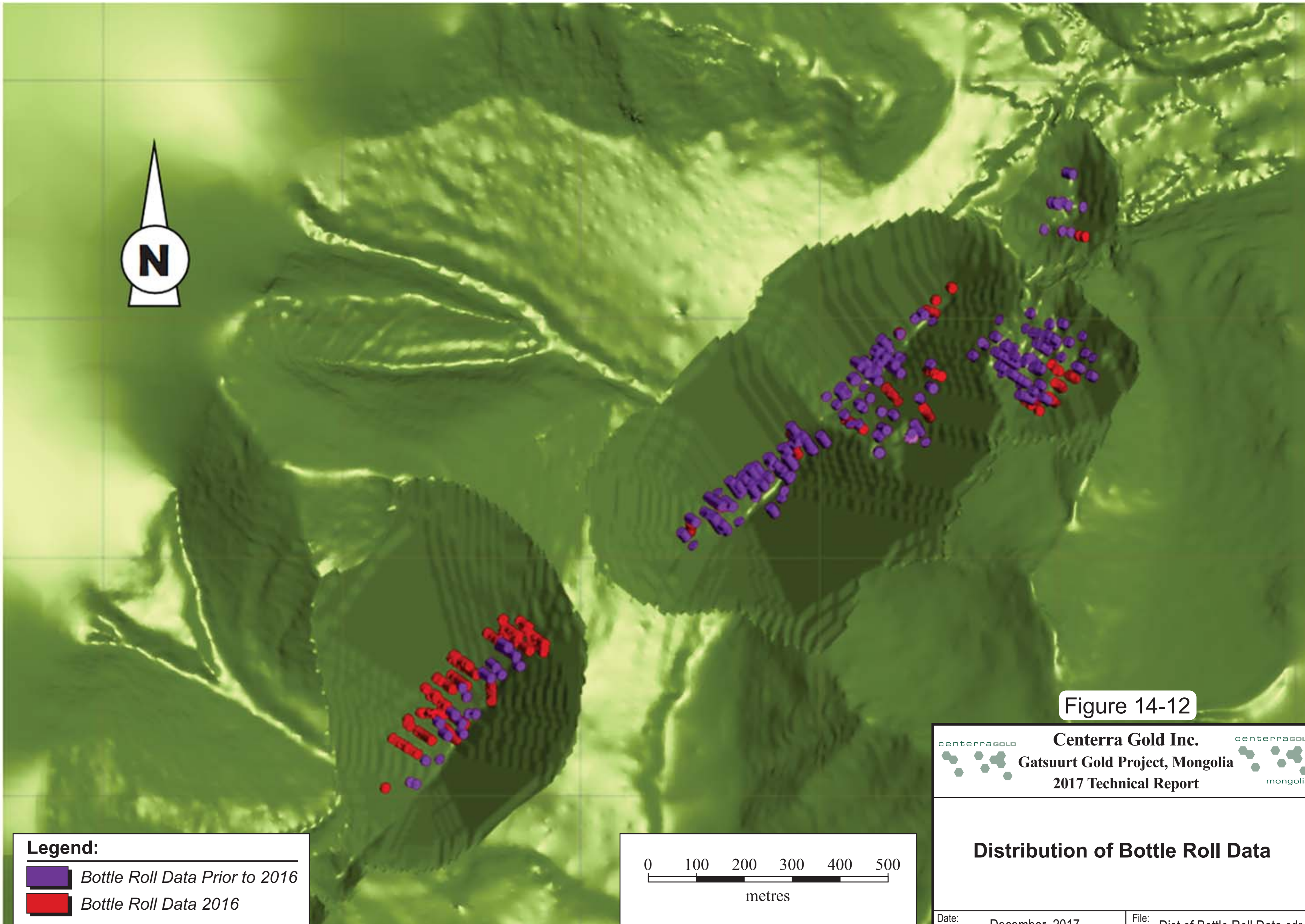

Figure 14-12

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Distribution of Bottle Roll Data

Date: December 2017 | File: Dist of Bottle Roll Data.cdr

Table 14-7 shows a summary of estimated bottle roll recoveries for the oxide and transitional zone. The two right columns consider only those blocks flagged to be included in the Mineral Resource, while the "Transition (All)" column considers all mineralized blocks in the transitional zone.

TABLE 14-7 BOTTLE ROLL RECOVERIES

Oxidation Zone	Oxide	Transitional (All)	Transitional (CIP only)	Transitional (BIOX® only)
Minimum Recovery (%)	20.7	0.3	52.8	0.8
Maximum Recovery (%)	98.5	98.8	98.8	81.8
Average Recovery (%)	80.3	45.6	75.9	36.3

A complete lithological model of host and country rocks of the Gatsuurt area was built in Leapfrog using borehole intersections. Based on testwork, gold recoveries were determined using BIOX® processing for various fresh and transitional host and country rocks (Table 14-8). The recoveries were coded into the block model in addition to the general categorical oxidation state of the lithology and were used to flag economical blocks in fresh and transitional material.

TABLE 14-8 BIOX® RECOVERIES BY ROCK TYPE

Block Model Inputs	Recovery (%)
Oxide	0.0
CZ Fresh Granite + PQSZ	86.0
CZ Trans Granite + PQSZ	84.7
CZ Fresh Sandstone	89.7
CZ Trans Sandstone	78.4
Other Fresh Untested Rock Types (Min of Fresh)	86.0
Other Trans Untested Rock Types (Min of Trans)	78.4
MZ Fresh Rhyolite	90.3
MZ Trans Rhyolite	84.5

BLOCK MODEL VALIDATION

Validation of the gold grade estimates was conducted using the following processes:

- Visual comparison of gold block grades versus the informing two metre composites on sections, and using swath plots along the three axes.
- Global and local mean grade comparison between the primary ID^3 gold grade estimates, the Ordinary Kriging (OK) model validation gold grade estimates, and the informing two metre composite gold grades.
- Local gold grade estimate check for grade bias between the primary ID^3 block estimates and validation OK block estimates using swath plots along the three axes.

A thorough visual section-by-section comparison was completed between informing data and block estimates. Sample sections for the MZ and CZ are shown in Figures 14-13 and 14-14, respectively. In addition, swath plots were used to compare the informing data with the primary estimated gold grade (Figure 14-15). Block grade estimates compared well with the informing data, indicating that the estimation parameters used in the interpolation of gold grades at Gatsuurt were appropriate for the estimation.

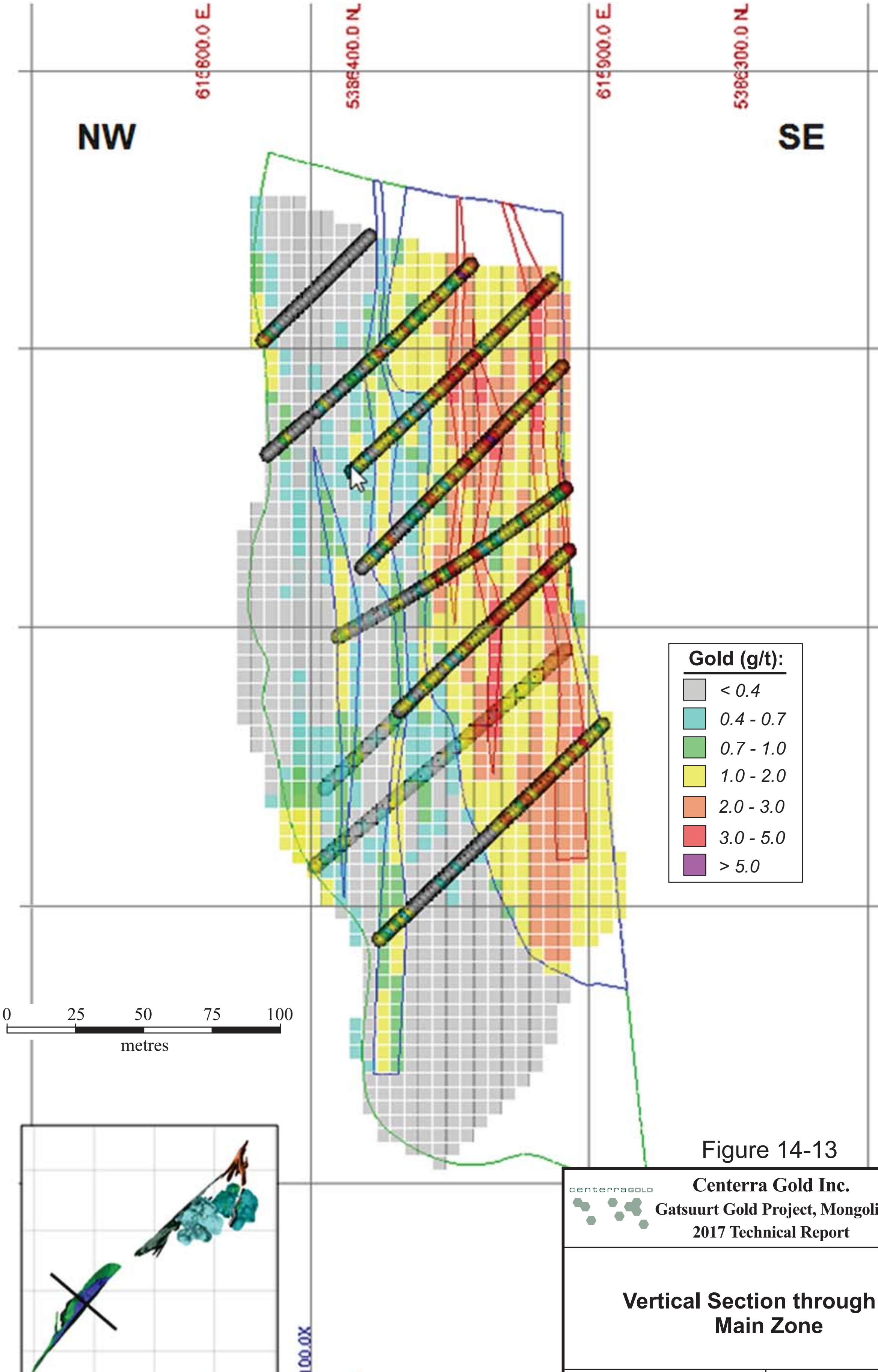

Figure 14-13

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Vertical Section through the Main Zone

Date: December 2017 | File: Vertical Section-Main.cdr

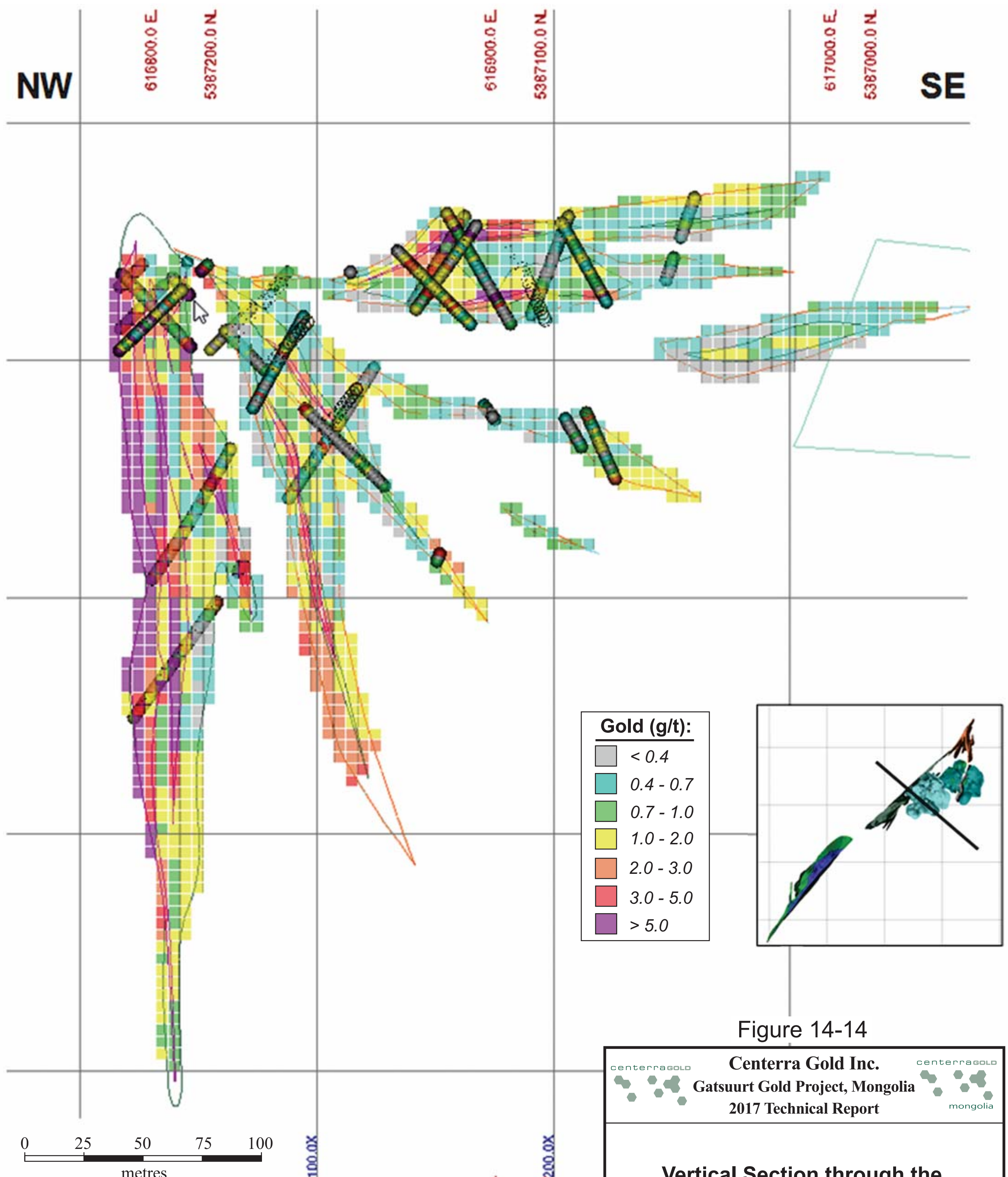

Figure 14-14

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Vertical Section through the Central Zone

Date: December 2017

File: Vertical Section-Central.cdr

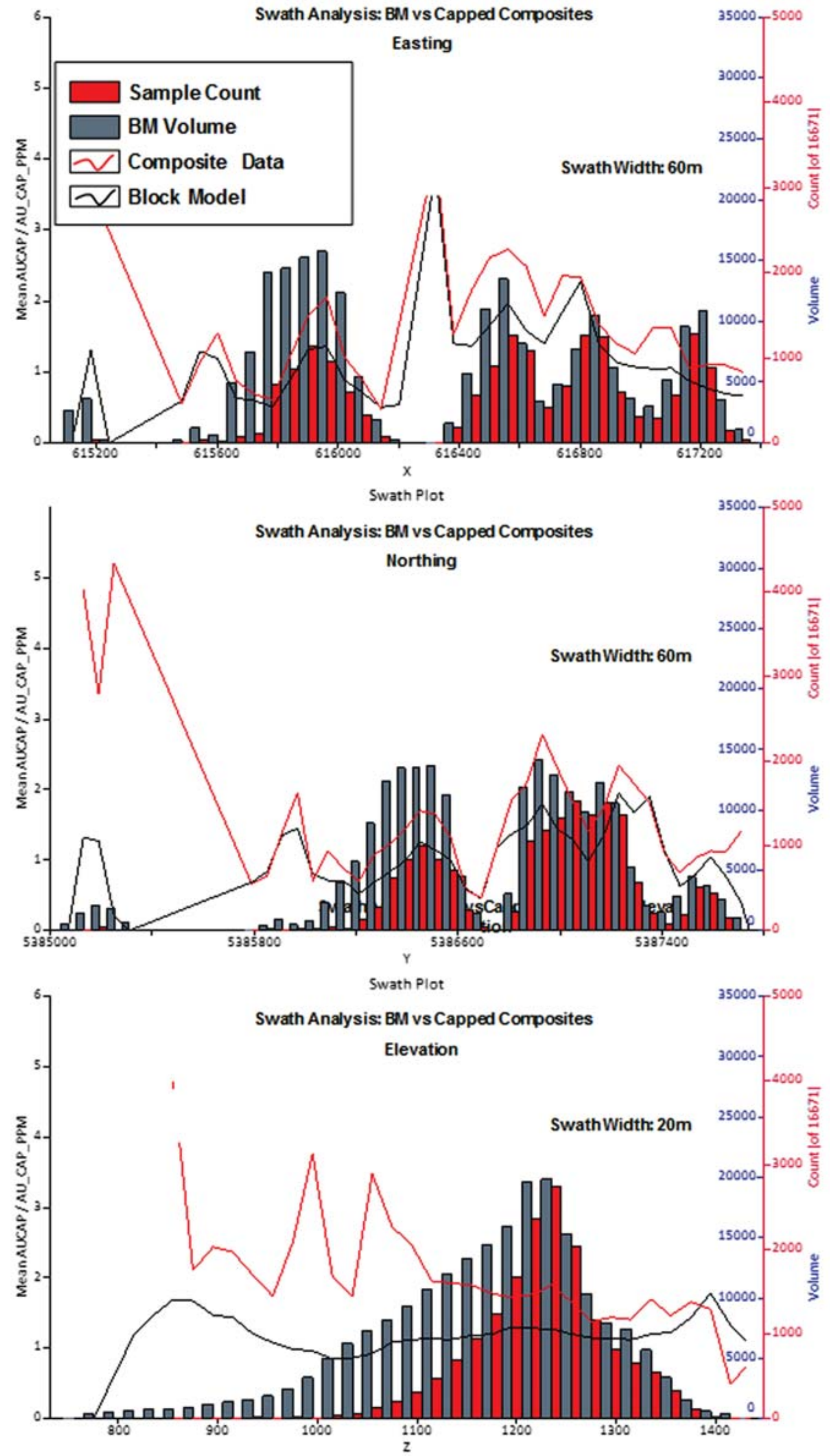

Figure 14-15

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Swath Plots of the Unified Block Grades – Block Model vs. Capped Composites

Date: December 2017	File: Swath Plots 1.cdr

Table 14-9 shows the global mean grade comparison for the individual mineralized zones between the primary ID^3 block estimates, the validation OK estimates, and the informing two metre composites. Generally, the estimation results between the primary estimator (ID^3) and those obtained from OK are in good agreement, with the highest difference in the Central Zone, where the difference is approximately 5.0%. This comparison shows that the primary estimation did not significantly over- or underestimate the mean grade of the individual zones and that grades estimated using OK are acceptable for validation purposes.

TABLE 14-9 GLOBAL MEAN GOLD GRADE COMPARISON BETWEEN ID^3, OK, AND TWO METRE COMPOSITES

Grade Shell	Domain	Number of Composites	Comp. Mean (g/t Au)	Number of Blocks*	Block ID^3 Mean (g/t Au)	Block OK Mean (g/t Au)	Percent Diff ID^3 vs. Comps	Percent Diff ID^3 vs. OK
High Grade	MZ	667	3.66	18,902	3.58	3.61	-2%	1%
	CZ	1,690	5.92	31,376	5.97	6.26	1%	5%
	SS	265	4.3	5,969	4.39	4.43	2%	1%
	SSS	216	5.53	4,719	5.28	5.51	-5%	4%
	High Grade Domains	**2,838**	**5.21**	**60,966**	**4.88**	**5.05**	**-6%**	**3%**
Medium Grade	MZ	2,355	1.56	74,489	1.58	1.61	1%	2%
	CZ + GT60	4,445	1.32	92,964	1.43	1.43	8%	0%
	SS	564	1.32	14,409	1.34	1.36	2%	2%
	SSS	926	1.41	17,219	1.38	1.40	-2%	1%
	Medium Grade Domains	**8,290**	**1.4**	**199,081**	**1.48**	**1.50**	**6%**	**1%**
Low Grade	MZ	1,901	0.42	86,165	0.42	0.42	1%	0%
	GT60	754	0.59	22,593	0.58	0.59	-2%	2%
	SS	1,078	0.64	26,481	0.58	0.58	-10%	1%
	SSS	1,783	0.64	34,587	0.65	0.64	1%	0%
	Low Grade Domains	**5,516**	**0.56**	**169,826**	**0.50**	**0.50**	**-11%**	**0%**

* Block numbers were extracted from GEMS grade folders and from the Whittle folder for the "All Domain" line item

A difference of 11% between composite mean and ID^3 block estimate in the low grade domains is attributed to clustering effects of composites compared to informed blocks, where large number of blocks are informed by a relatively small number of data.

Swath plots were used to assess the primary ID^3 and the validation OK block estimates for local grade bias. A grade bias is evident if one of the grade plots is consistently above or below the other grade plot. Typically, both grade plots should cross one another, which should result in the means being within ±10%. The swath plots can also be used to assess grade smoothing, where the primary estimate (ID^3) grade plot tends to be flat or does not have the abrupt peaks and valleys of the validation estimate (OK) grade plot.

Using the block dimensions of 5 m x 5 m x 5 m (x, y, z), block grades are averaged along each of the three axes using the block dimension as a spacing and plotted for comparing the ID^3 block estimates with the OK block estimates. Figure 14-16 shows swath plots for the deposit based on unified grades in the Whittle folder in GEMS. All swath plots appear to behave similarly between the two estimation methods and along all three of the axes.

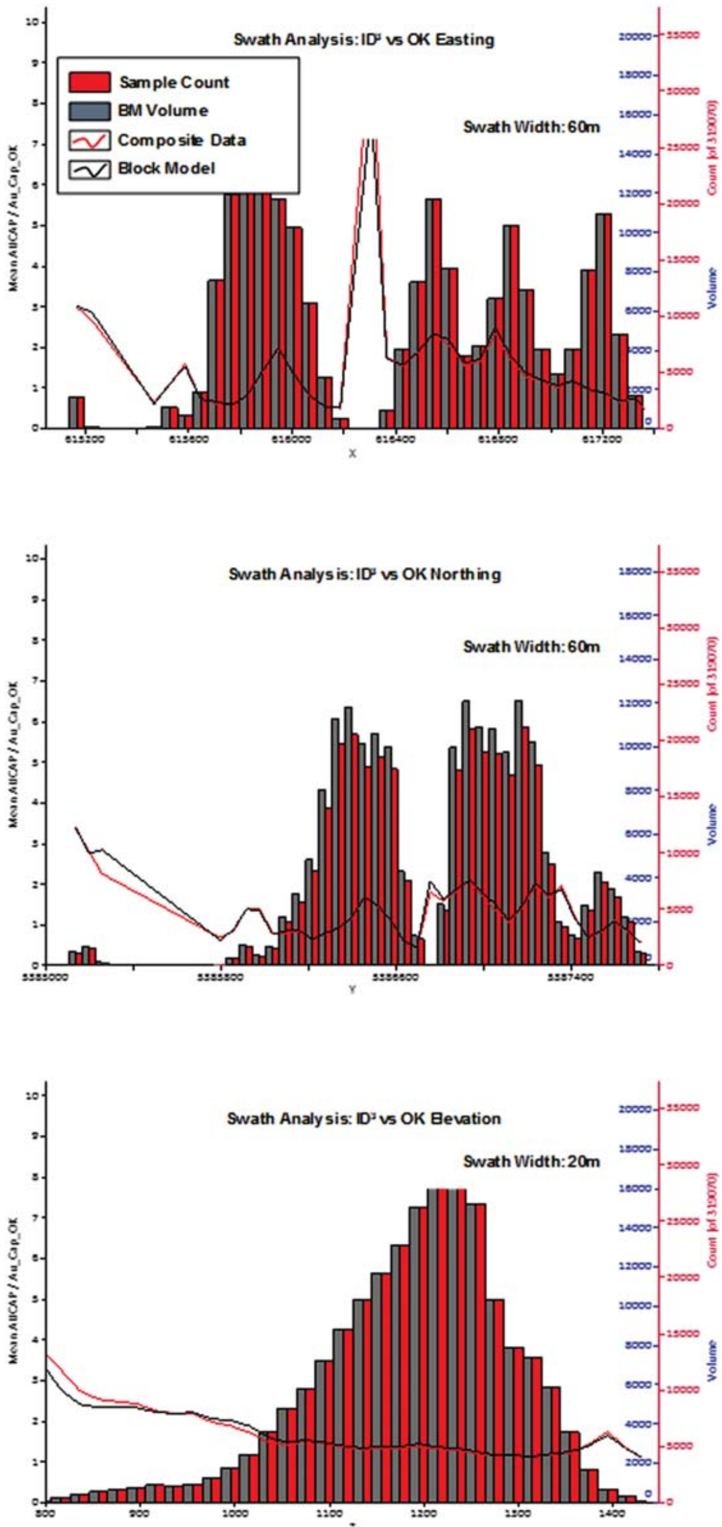

Figure 14-16

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Swath Plots of the Unified Block Grades - ID³ vs. OK

Date: December 2017	File: Swath Plots 2.cdr

The global mean grade comparison for the individual mineralized zones between the primary ID^3 block estimates and the informing two metre composites is shown in Tables 14-10, 14-11, and 14-12 for arsenic, iron, and sulphur, respectively.

TABLE 14-10 GLOBAL MEAN ARSENIC GRADE COMPARISON BETWEEN ID^3 AND TWO-METRE COMPOSITES

Grade Shell	Domain	Number of Composites	Composite Mean (% As)	Number of Blocks*	Block ID^3 Mean (% As)	Percent Diff ID^3 vs. Comps (%)
High Grade	MZ	456	0.90	20,133	0.88	-1.14
	CZ	1,170	0.63	34,577	0.64	0.44
	SS	258	0.70	**5,731**	**0.65**	-3.79
	SSS	168	0.31	**4,468**	**0.34**	4.28
	High Grade Domains	**2,052**	**0.67**	**64,909**	**0.69**	**1.42**
Medium Grade	MZ	1,734	0.57	100,976	0.49	-7.69
	CZ+GT60	3,041	0.33	**113,454**	**0.38**	6.10
	SS	551	0.28	**11,704**	**0.31**	4.94
	SSS	750	0.25	**17,191**	**0.23**	-3.58
	Medium Grade Domains	**6,076**	**0.39**	**243,325**	**0.41**	**3.01**
Low Grade	MZ	1,515	0.17	112,128	0.16	-1.71
	GT60	638	0.20	27,334	0.19	-3.00
	SS	1,028	0.13	27,140	0.12	-3.03
	SSS	1,239	0.11	36,009	0.11	-1.60
	Low Grade Domains	**4,420**	**0.15**	**202,611**	**0.15**	**1.15**
All Grade Shells	**All Domains**	**12,548**	**0.35**	**366,807**	**0.33**	**-3.20**

* Block Numbers were extracted from GEMS Grade Folders and from the Whittle Folder for the "All Domain" line item.

TABLE 14-11 GLOBAL MEAN IRON GRADE COMPARISON BETWEEN ID^3 AND TWO-METRE COMPOSITES

Grade Shell	Domain	Number of Composites	Composite Mean (% Fe)	Number of Blocks*	Block ID^3 Mean (% Fe)	Percent Diff ID^3 vs. Comps (%)
High Grade	MZ	376	2.52	20,133	2.60	1.59
	CZ	965	3.06	34,577	3.06	0.04
	SS	124	3.70	**5,731**	**3.53**	-2.37
	SSS	104	2.72	**4,468**	**2.75**	0.53
	High Grade Domains	**1,569**	**2.96**	**64,909**	**2.94**	**-0.33**
Medium Grade	MZ	1449	2.33	100,976	2.26	-1.58
	CZ + GT60	2,253	3.01	**113,454**	**3.09**	1.35
	SS	226	3.58	**11,704**	**3.54**	-0.49
	SSS	396	2.41	**17,191**	**2.46**	1.04
	Medium Grade Domains	**4,324**	**2.76**	**243,325**	**2.72**	**-0.63**
Low Grade	MZ	1,100	2.36	112,128	2.35	-0.20
	GT60	103	2.73	27,334	2.76	0.55
	SS	314	3.50	27,140	3.44	-0.85
	SSS	719	2.60	36,009	2.71	2.05
	Low Grade Domains	**2,236**	**2.61**	**202,611**	**2.62**	**0.02**
All Grade Shells	**All Domains**	**8,129**	**2.75**	**366,807**	**2.79**	**0.75**

* Block numbers were extracted from GEMS grade folders and from the Whittle folder for the “All Domain” line item.

TABLE 14-12 GLOBAL MEAN SULPHUR GRADE COMPARISON BETWEEN ID^3 AND TWO-METRE COMPOSITES

Grade Shell	Domain	Number of Composites	Composite Mean (% S)	Number of Blocks*	Block ID^3 Mean (% S)	Percent Diff ID^3 vs. Comps (%)
High Grade	MZ	372	1.35	20,133	1.39	1.54
	CZ	965	1.33	34,577	1.20	-4.95
	SS	124	1.25	5,731	1.16	-3.88
	SSS	104	0.65	4,468	0.91	4.34
	High Grade Domains	**1,565**	**0.83**	**64,909**	**1.24**	**-2.26**
Medium Grade	MZ	1,427	1.30	100,976	0.81	-4.70
	CZ + GT60	2,244	0.89	113,454	0.71	-0.68
	SS	226	0.72	11,704	0.53	-10.22
	SSS	395	0.65	17,191	0.78	1.94
	Medium Grade Domains	**4,292**	**0.75**	**243,325**	**0.75**	**-2.02**
Low Grade	MZ	1,092	0.78	112,128	0.32	1.02
	GT60	103	0.31	27,334	0.34	-16.05
	SS	314	0.47	27,140	0.24	-2.72
	SSS	717	0.25	36,009	0.30	4.97
	Low Grade Domains	**2,226**	**0.27**	**202,611**	**0.31**	**1.60**
All Grade Shells	**All Domains**	**8,083**	**0.30**	**366,807**	**0.61**	**-9.95**

* Block numbers were extracted from GEMS grade folders and from the Whittle folder for the “All Domain” line item.

CUT-OFF GRADE

Mineral Resources were estimated using variable cut-off grades based on the weathering layer, processing costs for BIOX® and CIP processing, mining costs, estimated recoveries for each process method, royalties, and a gold price assumption of $1,450/oz (Table 14-13). The gold price is based on consensus, long term forecasts from banks and financial institutions.

Oxide material potentially will be processed in a CIP plant, while fresh, unoxidized material potentially will be processed using the BIOX® system. Transition material may be processed by either system based on the estimated maximum revenue for each block.

For each process method and for each block, the estimated net value of gold was calculated based on the estimated gold grade, revenue per gram of gold, and estimated metal recovery. Within the oxide layer, blocks with a net value greater than the CIP process cost were included as part of the Mineral Resource statement. Within the fresh layer, blocks with a net value greater than the BIOX® process cost were included as part of the Mineral Resource statement. Within the transition layer, blocks with net values greater than either the CIP or BIOX® process costs were considered for the Mineral Resource. Blocks in the transitional layer were assigned to the process method giving the highest revenue.

Minimum grades considered for the reported resource are 0.68 g/t Au in oxide material, 0.67 g/t Au and 1.16 g/t Au in transitional material destined for CIP and BIOX® processing, respectively, and 1.09 g/t Au in fresh rock.

TABLE 14-13 REVENUE CALCULATION FACTORS

Material Type	Oxide	Transitional	Transitional	Fresh
Operating Cost Summary $/t	**CIP**	**CIP**	**BIOX®**	**BIOX®**
Processing Cost	$11.95	$11.95	$25.46	$25.46
Ore Haulage Cost	$7.00	$7.00	$7.00	$7.00
G&A Cost	$8.38	$8.38	$8.38	$8.38
Incremental Mining Cost	$0.60	$0.60	$0.60	$0.60
Total Operating Cost	**$27.93**	**$27.93**	**$41.44**	**$41.44**
Recoveries and Losses %				
Average Recovery (variable in block model)	80%	45%	83%	86%
Mining Recovery	100%	100%	100%	100%
Mining Loss	0%	0%	0%	0%
Revenue and Selling Costs $/oz				
Gold Price	$1,450	$1,450	$1,450	$1,450
Royalty	8.5%	8.5%	9.0%	9.0%
Selling/Marketing Costs	$5.00	$5.00	$5.00	$5.00

CLASSIFICATION

Definitions for resource categories used in this report are consistent with CIM (2014) definitions incorporated by reference into National Instrument 43-101. In the CIM classification, a Mineral Resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity, and other geological characteristics of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge, including sampling.” Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the “economically mineable part of a Measured and/or Indicated Mineral Resource” demonstrated by studies at pre-feasibility or feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

Mineral Resources at Gatsuurt were classified as Indicated or Inferred based on borehole spacing and the apparent continuity of mineralization. Resources assigned to the Indicated category have borehole spacings of 35 m or less for the MZ and 30 m or less for the CZ. Based on visual interpretation, solids were built for the volume to be classified as Indicated in both the MZ and CZ. The upper part of the MZ, which has a bottom sloping to the northwest and is parallel to the angled boreholes, and most of the principal parts of the CZ, and SS, were assigned to the Indicated category.

All splays, and areas of less certain continuity, were assigned to the Inferred Mineral Resource category.

MINERAL RESOURCE STATEMENT

The Mineral Resource estimate has an effective date of October 31, 2017. The Mineral Resources exclusive of Mineral Reserves are reported in Table 14-14 by zone and processing type.

TABLE 14-14 MINERAL RESOURCE STATEMENT EXCLUSIVE OF MINERAL RESERVES AS OF OCTOBER 31, 2017

Classification	Zone	Process Type	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Indicated	CZ	BIOX®	5,308	2.3	399
		CIP	1,551	1.2	57
		Sub-Total	**6,859**	**2.1**	**456**
	MZ	BIOX®	3,648	1.7	203
		CIP	481	1.2	19
		Sub-Total	**4,129**	**1.7**	**222**
Total Indicated (CZ+MZ)		BIOX®	8,956	2.1	602
		CIP	2,033	1.2	77
		TOTAL	**10,988**	**1.9**	**678**
Inferred	CZ	BIOX®	567	2.5	46
		CIP	90	1.6	5
		Sub-Total	**658**	**2.4**	**50**
	MZ	BIOX®	2,817	2.2	196
		CIP	337	1.6	17
		Sub-Total	**3,154**	**2.1**	**213**
Total Inferred (CZ+MZ)		BIOX®	3,384	**2.2**	241
		CIP	428	1.6	21
		TOTAL	**3,812**	**2.1**	**263**

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability
3. The Mineral Resources have been estimated based on a gold price of US$1,450 per ounce.
4. Mineral Resources were estimated using variable cut-off grades based on processing, ore haulage, incremental mining cost, and G&A costs of $27.93/t and $41.44/t for CIP and BIOX®, respectively, estimated recoveries, and forecasted revenue of $42.50 and $42.26 per gram of gold for CIP and BIOX®, respectively.
5. Numbers may not add due to rounding.

COMPARISON WITH PREVIOUS MINERAL RESOURCE ESTIMATES

The current Mineral Resource estimate supersedes a Mineral Resource estimate effective December 31, 2016. For a direct comparison between the December 31, 206 and October

31, 2017 Mineral Resource estimates, the mineral inventories both estimates were reported within a conceptual pit shell constructed in 2013 at a cut-off grade of 1.4 g/t Au.

As shown in Table 14-15, the mineral inventory in the MZ remains essentially unchanged. The mineral inventory in the CZ, including the GT60, SS, and SSS area experienced a drop of approximately 235,000 ounces, or 14.5%, of gold. This difference, primarily a loss of tonnage, is due to the remodelling of grade domains. High-grade domains were not previously modelled in the SS and SSS areas. Figure 14-17 shows side-by-side plan views of the previous and current Mineral Resource models. Lower capping values account for approximately 10% (~20,000 ounces) of the overall decrease of the mineral inventory within this reporting volume.

TABLE 14-15 COMPARISON BETWEEN 2016 AND 2017 MINERAL INVENTORIES, REPORTED WITHIN THE 2016 PRELIMINARY PIT SHELL

		2016 Estimate			2017 Estimate			Difference		
Class	**Deposit/ Class**	**Tonnes (kt)**	**Grade (Au g/t)**	**Cont. Au (koz)**	**Tonnes (kt)**	**Grade (Au g/t)**	**Cont. Au (koz)**	**Tonnes (kt)**	**Grade (Au g/t)**	**Cont. Au (koz)**
Indicated	CZ	10,554	3.4	1,156	9,710	3.1	965	-843	-0.3	-191
Inferred		226	3.0	22	59	2.5	5	-167	-0.5	-17
Indicated	MZ	5,838	2.5	463	5,654	2.5	457	-184	0.04	-6
Inferred		37	2.4	3	42	2.6	3	5	0.2	1

Note: This table is for comparison only and should not be misconstrued as a Mineral Resource Statement. The mineral inventory has been reported within a conceptual pit constructed in 2016 at a cut-off grade of 1.4 g/t Au. The Mineral Resources are inclusive of the Mineral Reserves.

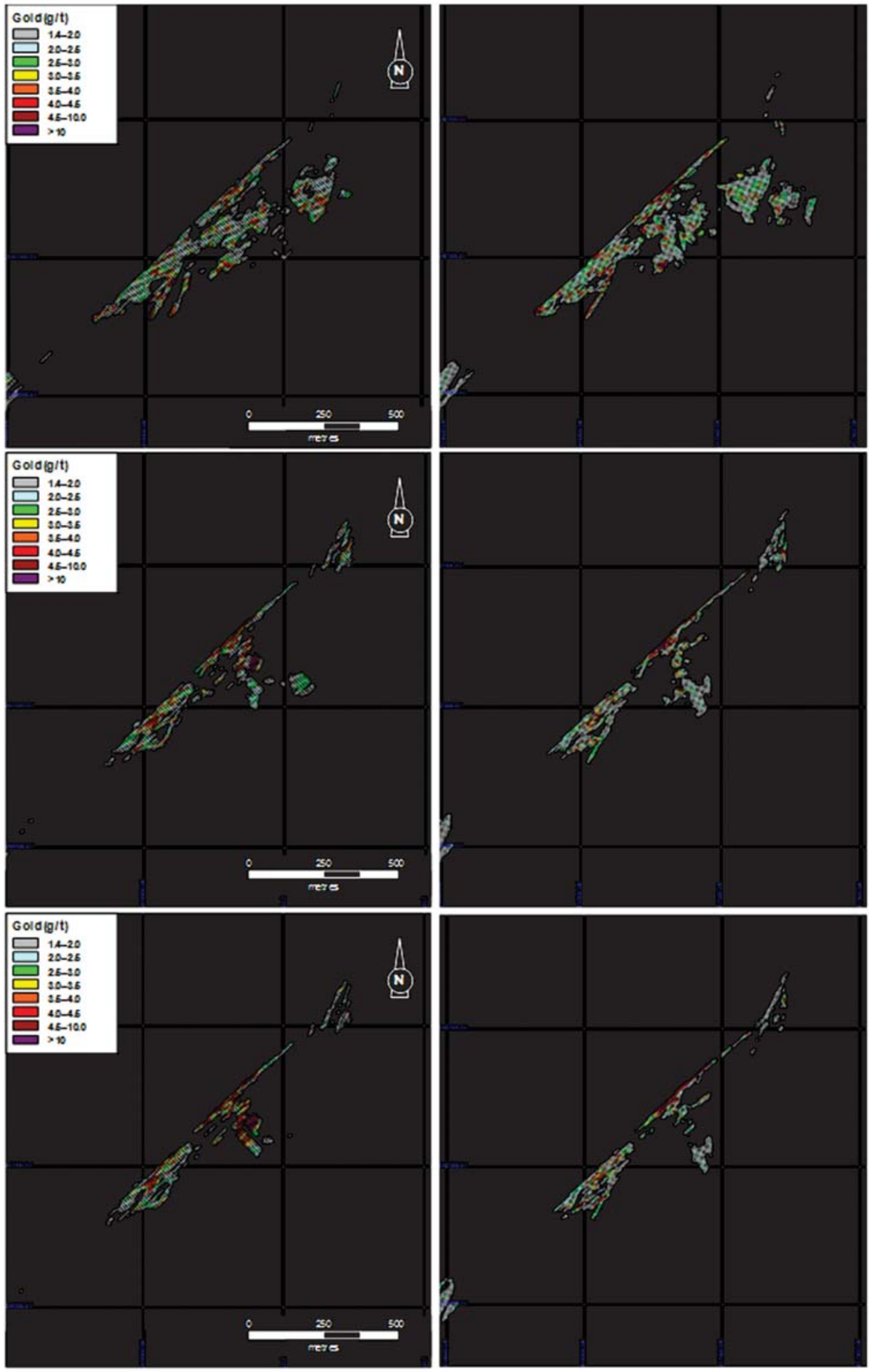


Figure 14-17

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Comparison of
2013 Block Estimates (Left)
to Current Block Estimates (Right)

Date: December 2017
File: 2013-Current Block Est.cdr

CONSOLIDATED MODEL FOR THE PURPOSE OF MINERAL RESERVES

A consolidated block model was prepared for the purpose of pit optimization, pit design, and Mineral Reserves. Results were exported to GEMS BLK format. A description of each model is provided in Table 14-16.

TABLE 14-16 DESCRIPTIONS OF CONSOLIDATED BLOCK MODELS

Attribute	File Name	Description
whittle_rock_type	Rock Type.blk2	Rock types. See Table 14-17.
whittle_density	Density.blk2	Density data transferred from low grade folder for mineralized blocks, Waste block density transferred from waste folder
whittle_percent	Percent.blk2	Percent of block inside mineralization, calculated from individual grade folders: low grade percent+medium grade percent+high grade percent
whittle_aucap	AUCAP.blk2	Interpolated gold grades using ID^3, capped composites, and transferred from grade folders and percent weighted. This is the attribute used to report final gold numbers
whittle_class	CLASS.blk2	Classification using 50/50 rule. See Table 14-17.
whittle_oxidation	Oxidation.blk2	Oxidation state classifier, transferred from low grade folder, where it was assigned based on 50/50 rule. See Table 14-17.
whittle_biox_recovery	BIOX_recovery.blk 2	Blocks assigned values as per Centerra guidelines, using 50/50 rule. See Table 14-17.
whittle_Rock Type 50pct rule	Rock Type 50pct rule.blk2	Alternative Rock Type field using the 50% rule. Min. blocks > 50% are assigned as 'ore' and min. blocks <50% are assigned as 'waste'; then all min blocks are 100% volume. To be used with "Percent 50 pct rule"
whittle_Percent 50pct rule	Percent 50pct rule.blk2	Assigned at 100% using 50% rule. To be used with "Rock Type 50pct"
whittle_Au_Diluted	Au_Diluted.blk2	Gold grades diluted to 100% at zero grade Au. To be used with "whittle_100_pct" and "Rock Type" attributes.
whittle_100_pct	100_pct.blk2	All blocks that touch mineralization diluted to 100% volume. To be used with "Au_Diluted" and "Rock Type" attributes.
whittle_as_pct	AS_PCT.blk2	Arsenic as percent, transferred from grade folders and percent weighted
whittle_fe_pct	FE_PCT.blk2	Iron percent, transferred from grade folders and percent weighted
whittle_s_pct	S_PCT.blk2	Sulphur percent, transferred from grade folders and percent weighted
whittle_pit	pit.blk2	Classifier of blocks inside the 2013 pit, used for reporting purposes and based on a 50/50 rule. See Table 14-17.
whittle_zone	Zone.blk2	Classifies blocks to belong to the Main or Central Zone; used for reporting. See Table 14-17.

Attribute	File Name	Description
whittle_buffer	Buffer.blk2	Flagged of test area along Sujigtei fault.
whittle_Zero_pct	Zero_pct.blk2	A percent volume field set to zero percent for testing purposes.

Rock types in the Rock Type block model are listed in Table 14-17.

TABLE 14-17 BLOCK MODEL ROCK CODES

Block Model	Description	Code
Rock Type	Air	0
	Waste Granites and Metasediments South of Sujigtei	99
	North Rhyolite Waste	97
	Overburden	98
	North Rhyolite Mineralization	7
	Mineralized Granites and Metasediments South of Sujigtei	9
Oxidation	Oxidized	1
	Transition	2
	Fresh	3
Zone	MZ	1
	CZ, SS, SSS, GT60	2
Class	Measured	1
	Indicated	2
	Inferred	3
Pit	In 2013 pit	1
	Not in 2013 pit	0
Buffer	Test area along fault	1
	Not within test area	0

Block Model	Description	Code
Percent 50pct rule	If volume > 50% in min. wireframe	100
	If volume < 50% in min. wireframe	0
Rock Type 50pct rule	If volume > 50% in min. wireframe	10
	If volume < 50% in min. wireframe	0
100_pct	If Rock Type = 10	100
	If Rock Type <> 10	0

15 MINERAL RESERVE ESTIMATE

The Mineral Reserve estimate relies on the Mineral Resource estimate prepared by RPA and described in Section 14 of this report. A summary of the Mineral Reserves as of October 31, 2017, is shown in Table 15-1.

TABLE 15-1 MINERAL RESERVES, AS OF OCTOBER 31, 2017

Category	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable	15,356	2.7	1,316
Total	**15,356**	**2.7**	**1,316**

Notes:

1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves are estimated using an average long-term gold price of US$1,250 per ounce and a US$/₮ exchange rate of 2,200 and have been validated by the positive project NPV that uses a 5% discount rate.
3. Numbers may not add due to rounding.

Although the QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate, the estimate should be considered within the context of the outcomes stated in this Technical Report. Of note, as outlined in Section 22, the Project's base case economic modelling results indicate a marginally positive net present value (NPV) of $39.5 million at an annual discount rate of 5%. Should any of the key revenue drivers (price, grade, recovery) incur any adverse variance to the plan, this will place the above estimate at risk.

PIT OPTIMIZATION AND PIT DESIGN

In order to define the economic mining area, pit limit optimization was completed using Geovia Whittle 4X software (Whittle 4X). This provided a series of pit shells that were used for mine design which in turn were used for estimation of mineable quantities and Mineral Reserves to meet the CIM requirement of "reasonable prospects for eventual economic extraction". The series of nested shells also helps to guide the development strategy and select a series of interim cutbacks in order to maximize value and balance ore and waste mining requirements.

APPROACH

The open pit limits were determined by considering both physical and economic constraints to mining. The economic pit limits were estimated using Whittle 4X. The terminology “pit limit optimization” refers to a process which aims to identify the highest value mining pit shape for a given series of inputs and constraints. It does not imply that mining has been “optimized” in other ways, such as mining method optimization, or development strategy, for example.

The Whittle 4X software uses the industry-standard Lerchs-Grossmann algorithm to define a three dimensional (3D) shape for the open pit which is considered the “optimal” economic shell for mining.

A key outcome of the economic pit limit optimization generation is a series of 3D surfaces or “nested pit shells” based on a range of ore selling prices. Each of these shells represents a surface that defines the break-even economic limit for specific revenue assumptions described below. The metal price sensitivity analysis is conducted by applying a “Revenue Factor” (RF) to the base case metal price. That is, a 100% RF pit shell results from multiplying the metal sales prices by 100%. A 50% RF pit shell indicates the shape of the pit and mineable quantities at 50% of the base metal price. The outcomes are very important in showing the sensitivity of the deposit to varying economic factors, including product price, and is a key consideration in the selection of the optimal pit shell for planning.

The resulting nested pit shells also indicate the likely strategic development of the deposit. The ideal pit development strategy in order to maximize cash flow involves mining successive pit shells from lowest revenue (say, 50% RF) factor to highest (say, 100% RF). That is, a pit shell based on a RF of 50% would have a higher margin than one at a RF of 80% and hence would be sequenced to be mined first.

In the case of the Gatsuurt operation, Centerra intends to process CIP ore only for the first approximately three years of feed while the BIOX® plant is under construction. The BIOX® material mined during this period will be stockpiled for processing once the BIOX® plant construction is complete. To ensure that there was sufficient CIP feed, and minimal stockpiling of BIOX® material, RPM completed a separate “CIP only” optimization which only considered revenue from the CIP process material. The purpose of this was to identify a preferable intermediate phase of pit development which would provide the required period of CIP feed while minimizing the amount of BIOX® ore and waste material to be mined.

The pit shells selected as the basis of detailed engineering design are discussed further in the section “Pit Shell Selection”.

PIT LIMIT OPTIMIZATION INPUT DATA

GEOLOGICAL MODEL

The Mineral Resource block model, developed by RPA in June 2017, was imported into the Whittle 4X software. The model contains Measured, Indicated, and Inferred Mineral Resource categories for gold mineralization.

Pit optimization for the mining study was initially run including Measured and Indicated Mineral Resource categories, but excluding Inferred Mineral Resource categories.

The optimization for the Mineral Reserves was run using a gold grade that was regularized over the whole block, that is, the gold grade was diluted over the entire block tonnes. The Mineral Reserves are reported using the partial percentage hybrid gold grade. This is discussed in the “Loss and Dilution” section.

ASSUMED LOSS AND DILUTION

A detailed Mining Accuracy Sensitivity Analysis was conducted to calculate the inherent model loss and dilution, simulating varying mining methods, from highly selective with small bench heights and equipment to bulk mining methods. The results of this work were used to estimate the Mineral Reserve tonnes and grade. This is described further in the section “Loss and Dilution”.

For the purpose of the pit optimization, however, a regularized fully diluted block model size of 5 m x 5 m x 5 m (X size x Y size x Z size) was used to account for loss and dilution. The difference between the regularized block model and the partial percentage hybrid model was found to be immaterial for the purposes of pit limit optimization.

GEOTECHNICAL SLOPE ZONE PARAMETERS

The design slope parameters used for the optimization were derived from the geotechnical studies completed for the Project. These studies are further detailed under “Geotechnical Slope Design” in Section 16 Mining Methods.

Table 15-2 outlines the parameters used for the purpose of the optimization with the slope zone regions defined in the block model.

TABLE 15-2 WHITTLE SLOPE ZONES

Material Description	BM Slope Zone Code	Interramp Design Angle (°)	Overall Whittle Design Angle (°)
Oxidized North Rhyolite Mineralization	17	44	38
Oxidized Mineralized Granites and Metasediments South of Sujigtei	19	41	35
Transition North Rhyolite Mineralization	27	48	42
Transition Mineralized Granites and Metasediments South of Sujigtei	29	48	42
Fresh North Rhyolite Mineralization	37	51	45
Fresh Mineralized Granites and Metasediments South of Sujigtei	39	48	42
All Overburden	98	29	27

PHYSICAL CONSTRAINTS

No physical constraints were applied in the optimization. The resulting optimizations were compared to the mining licence boundary against which all pits were found to be well within this boundary.

METAL PRICE FORECAST

The optimizations were completed using a long term average gold price of US$1,250/oz. Based on long term consensus price forecasts of gold, the QP is of the opinion that, for the purposes of the optimization, the value used is reasonable.

Only Measured and Indicated Mineral Resources were considered to produce revenue in the optimization, as per the CIM guidelines.

OPERATING COSTS

The operating costs for use in the optimization are summarized in Table 15-3.

TABLE 15-3 WHITTLE COST INPUT SUMMARY

	Unit	Value
Direct Mining Costs		
Rock Mining Cost	US$/t	1.80
Gatsuurt to Boroo Haulage (~55km)	US$/t Ore	7.00
Ore Mining Cost	US$/t Ore	2.40
Vertical Lift Haul Cost	US$/t per m vert	0.0031
Mining Support Costs (G&A)		
G&A	US$/t Ore	8.38
Processing Cost		
Oxide/CIP ore	US$/t Ore	11.95
Sulphide/BIOX® ore	US$/t Ore	25.46
Sales Cost and Price		
Gold Sales cost and duties	US$/oz	5.00
Royalties		
Government Royalty at US$1,250/oz	%	12
Gatsuurt LLC [1]	%	3
BIOX® material [2]	%	0.5

Notes:

1. Calculated as 3% of gross revenue less government royalties
2. Only applied to BIOX® material

METALLURGICAL RECOVERY FACTORS AND COSTS

Gold doré is the only product produced by the Project. This is achieved using two processing methods, CIP and BIOX®.

The unit costs for the processing methods are summarized in Table 15-3.

The process recovery is determined on a block by block basis and is determined by rock type. All oxide material reports to the CIP process whereas all sulphide material reports to the BIOX process. The transitional material can report to either the CIP or BIOX processes depending on the revenue generated for the block. For the purposes of the Reserve estimate the transition material reports to the process that produced the highest revenue. On this basis, a block in the transition material is assigned a recovery according to the process decision.

Both process streams were included in the optimization and a cash flow method was used to determine the relevant process stream for each block. This methodology is discussed in further detail in the section “Cut-Off Grades”.

PIT SHELL SELECTION

The optimization results showed that there were two distinct and separate pit areas, known as the Central and Main Zones (Figure 15-1). These two zones were further analyzed individually to investigate their individual characteristics.

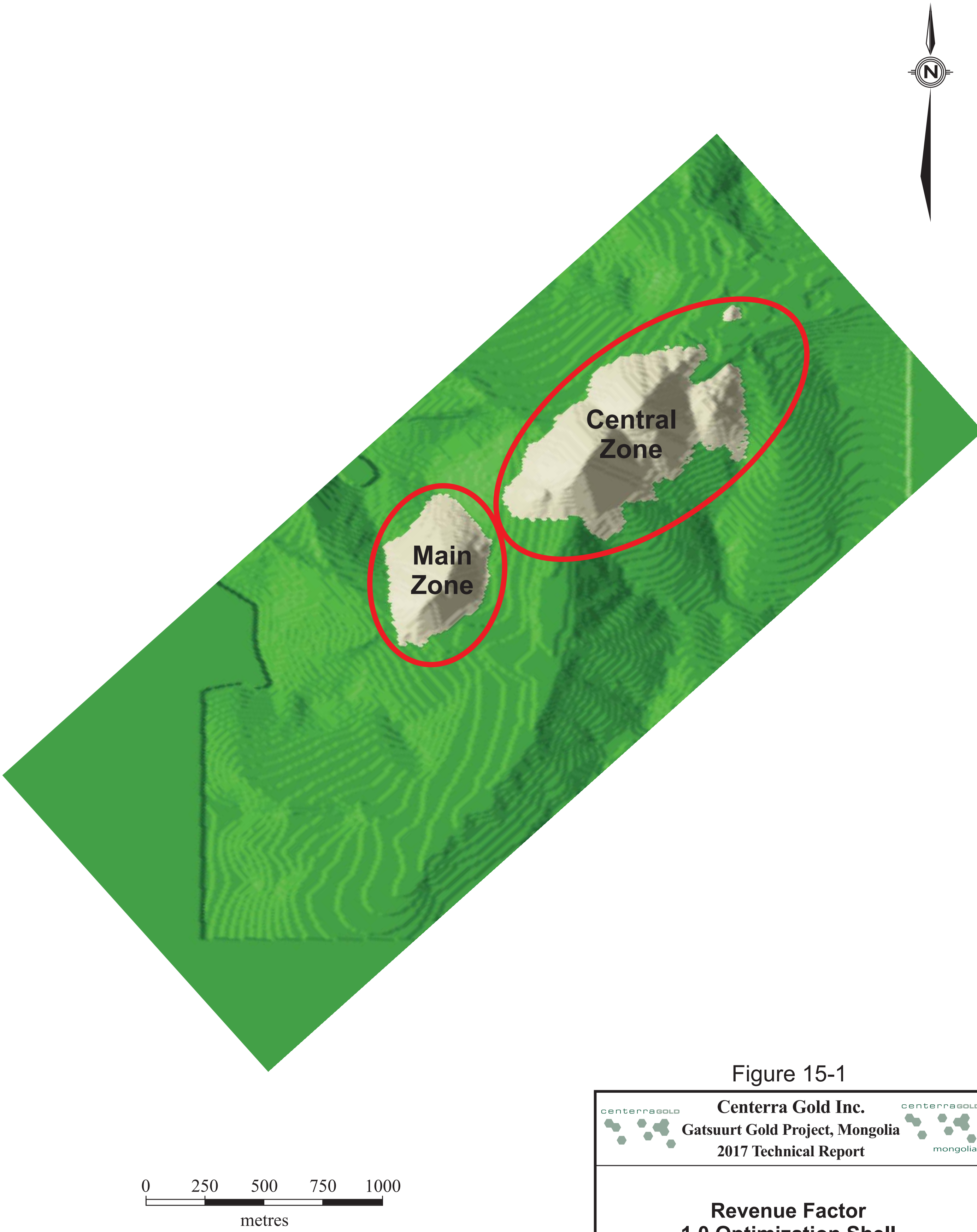


Figure 15-1

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

centerragold
centerragold mongolia

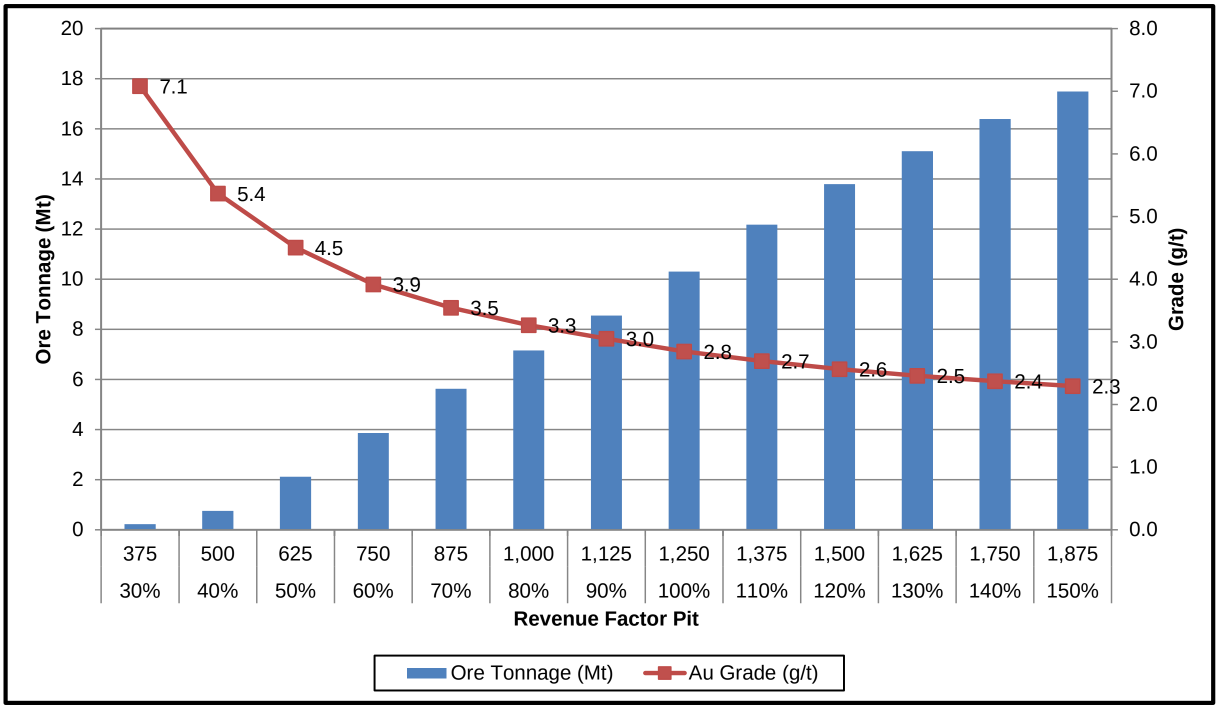
Revenue Factor
1.0 Optimization Shell

Date: December 2017	File: Revenue Factor.cdr

CENTRAL PIT ULTIMATE SHELL SELECTION

The Central Zone optimization shell tonnage and grade results are summarized in Figure 15-2. With an increasing RF, there is a consistent increase in ore tonnage and a declining gold grade.

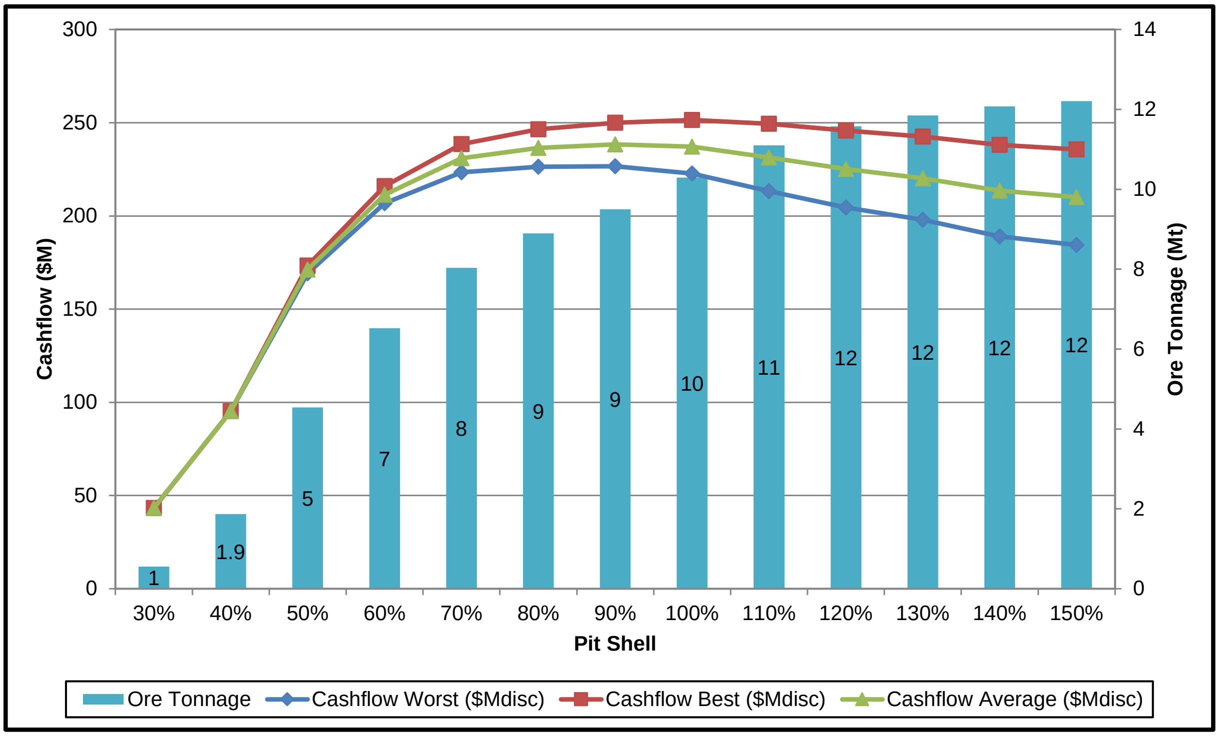
FIGURE 15-2   CENTRAL PIT LIMIT OPTIMIZATION RESULTS



A cash flow analysis was performed for the Central Zone using a 10% discount rate and a 1.7 Mt/a processing throughput rate, with the results shown in Figure 15-3.

The best-case scenario is one where the incremental optimized pits are mined successively up to the given pit shell, and the worst-case scenario is where the given pit is mined on a bench by bench basis from top to bottom. An “average cash flow” was also calculated which is the arithmetic mean of the best-case and worst-case scenarios.

FIGURE 15-3 CENTRAL PIT LIMIT OPTIMIZATION CASH FLOW



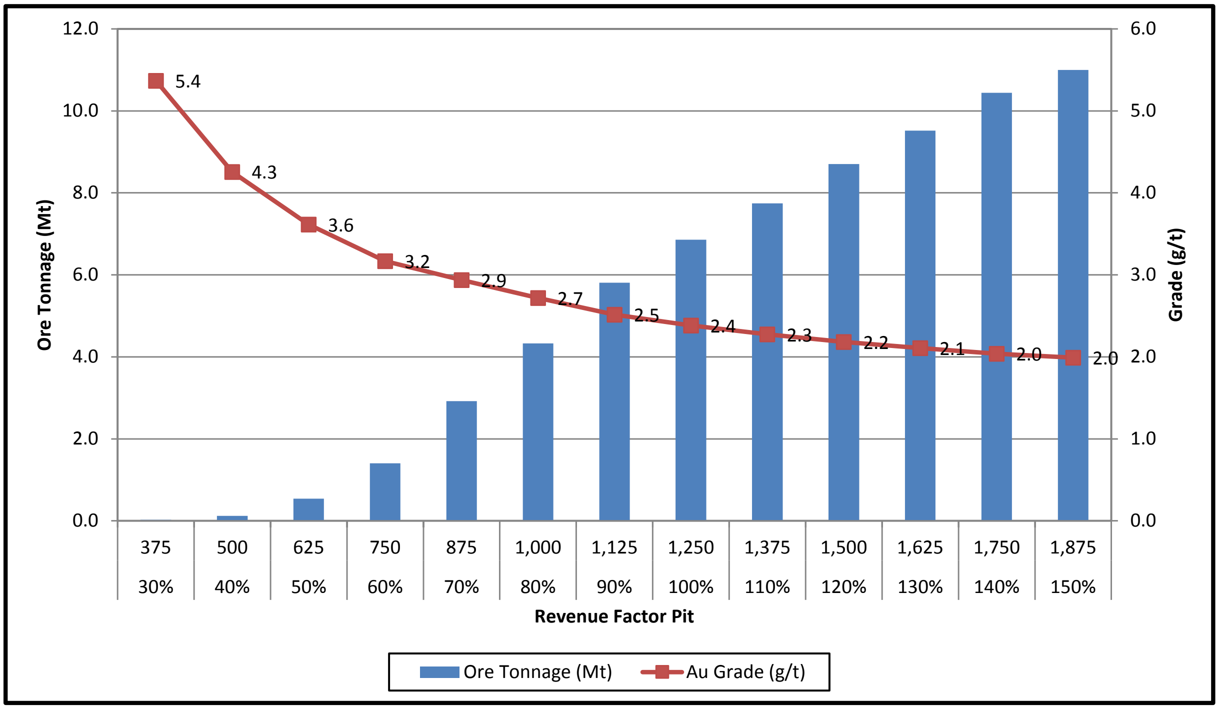
Due to the size of the optimized pit shells and the working room requirements of the expected mining equipment, it was likely that there would only be two or three stages used in the development of the ultimate pit shell. As a result, it was expected that the likely cash flow of the final design and schedule would be more closely represented by the worst-case scenario than the best-case scenario, and as such it was decided that the best suited pit shell to base the final pit design on would be the RF 90% pit shell. This reflected an optimization gold price of US$1,125/.oz, which is a realistic expectation of future prices that provides a level of insulation against deteriorating economic conditions. It also provides a reasonable opportunity for further expansion of the pit should more favourable economic conditions develop towards the end of the mine life.

MAIN PIT ULTIMATE SHELL SELECTION

The Main Zone optimization tonnage and grade results for each shell are summarized in Figure 15-4.

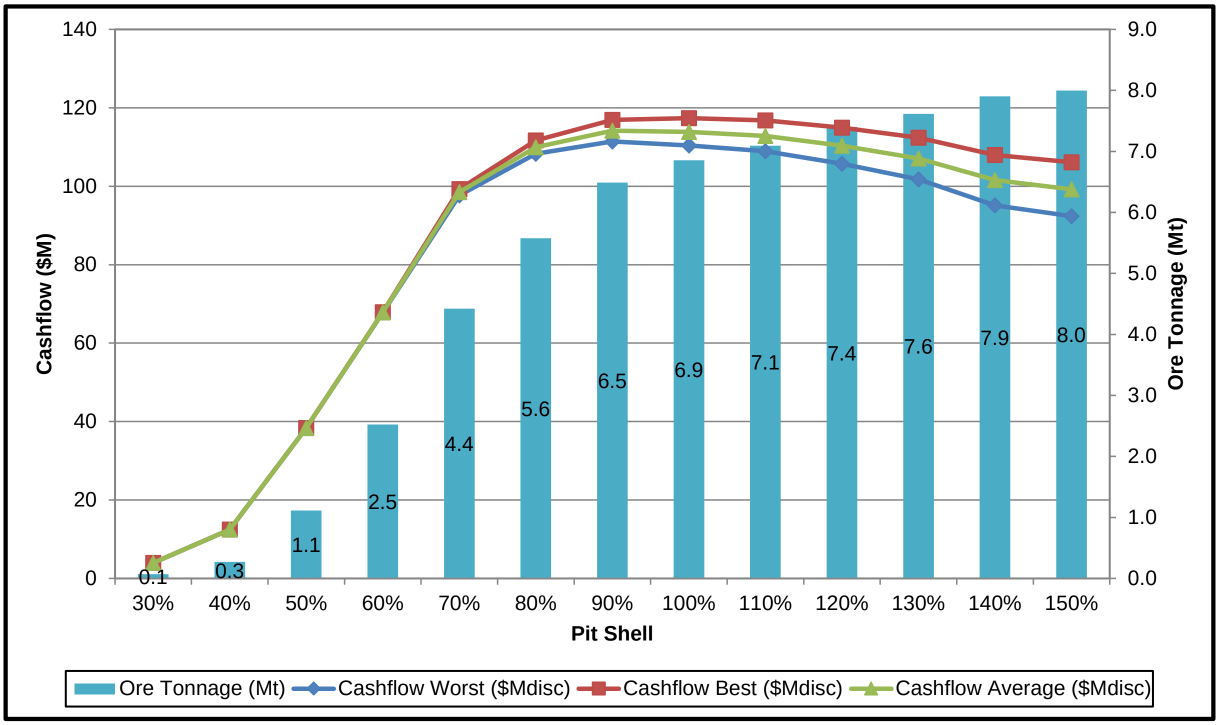
Similar to the Central Zone, the Main Zone optimization shows that, with an increasing RF, there is a consistent increase in ore tonnage with a declining gold grade.

FIGURE 15-4 MAIN PIT LIMIT OPTIMIZATION RESULTS



The cash flow analysis for the Main Zone used a 10% discount rate and a 1.7 Mt/a processing throughput rate shown in Figure 15-5.

FIGURE 15-5 MAIN PIT LIMIT OPTIMIZATION CASH FLOW



Due to the size of the optimized pit shells and the working room requirements of the expected mining equipment, it was likely that there would only be two stages used in the development of the ultimate pit shell for the Main Zone. As such, it was expected that the likely cash flow result of the final design and schedule would be more closely represented by the worst-case scenario than the best-case scenario. Accordingly it was decided that the best suited pit shell for the final pit design for the Main Zone would also be the RF 90% pit shell. This reflected an optimization gold price of US$1,125/oz, which increases the pit margin and reduces the risk against deteriorating economic conditions. It also provides for a reasonable opportunity further expansion of the pit should more favourable economic conditions develop.

PIT PHASE SHELL SELECTION

The key factors in consideration of the pit shells for phase selection were:

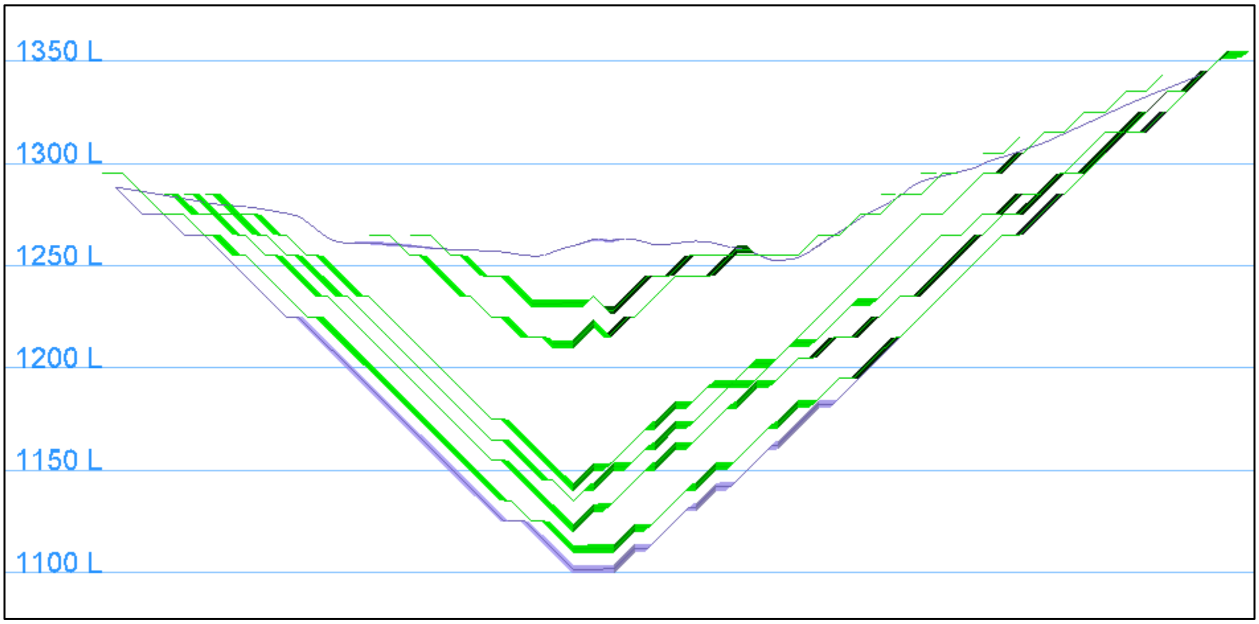
- Working room between phases.
- Prioritizing and providing sufficient CIP feed for the first two years of processing.
- Minimizing stockpiled BIOX® material during the CIP feed period.
- Water diversion bench location.

Numerous options were evaluated to determine a practical phasing approach for the development of the operation. Each zone was considered in isolation with respect to working room and location of the diversion bench, but both pits were considered when calculating the CIP feed capacity.

Both pits used a single intermediate phase and an ultimate final stage design. This was predominately due to the working room limitations between the incremental shells being such that a three staged development would result in small and prohibitively unproductive working areas.

The Central Pit showed a nearly common highwall to the southeast between the optimized incremental pits, shown in Figure 15-6. This wall was determined to be the best option for the placement of the diversion bench for the Gatsuurt River; it could be established in the first phase and would not need to be relocated and re-established in the subsequent phase. Initial phase designs had the diversion bench being placed in the northwest wall of the pit phases in an attempt to reduce the extra stripping required for the bench; however, this resulted in difficulties maintaining the continuity of the diverted water flow between the phase developments. The benefits of alignment along the southeast wall were determined to outweigh the minor penalty of the increased waste stripping.

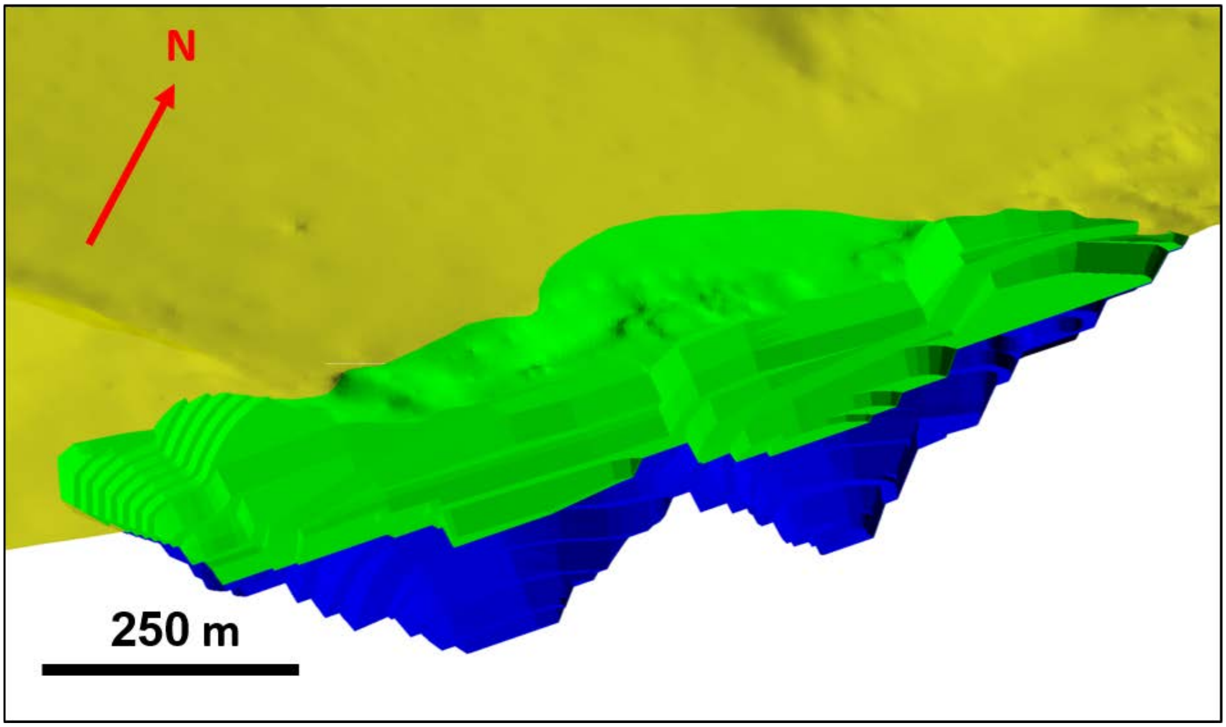
FIGURE 15-6 CENTRAL PIT OPTIMIZATION SHELLS CROSS SECTION



The intermediate stage of the Central Pit was based on a combination of the RF 70% shell produced by the CIP only optimization and the RF 45% shell of the optimization considering all processes. This shell combination provided a suitable tonnage of CIP ore at a reasonably low stripping ratio and resulted in an appropriate cut-back width at depth.

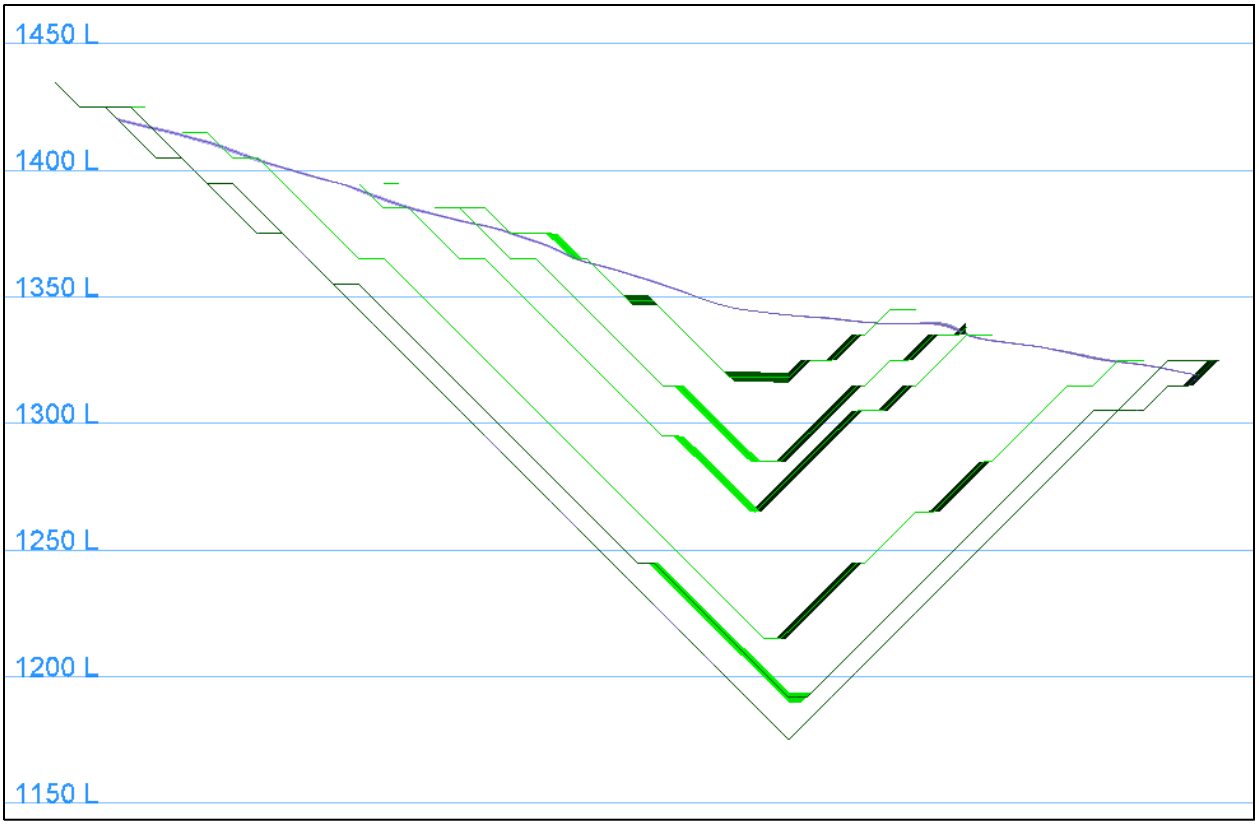
Figure 15-7 shows a long section of the Phase 1 (green) and Phase 2 (ultimate pit; blue) designs of the Central Pit.

FIGURE 15-7 CENTRAL PIT LONG SECTION



The Main Pit phase selection was completed in a similar manner to the Central Pit. The optimized incremental pit shells progressed outwards in all directions (Figure 15-8). The first stage of the Main Pit was based on the RF 60% shell which captured the vast majority of the CIP ore in the Main Zone and offered a reasonable cut-back width to the ultimate pit shell.

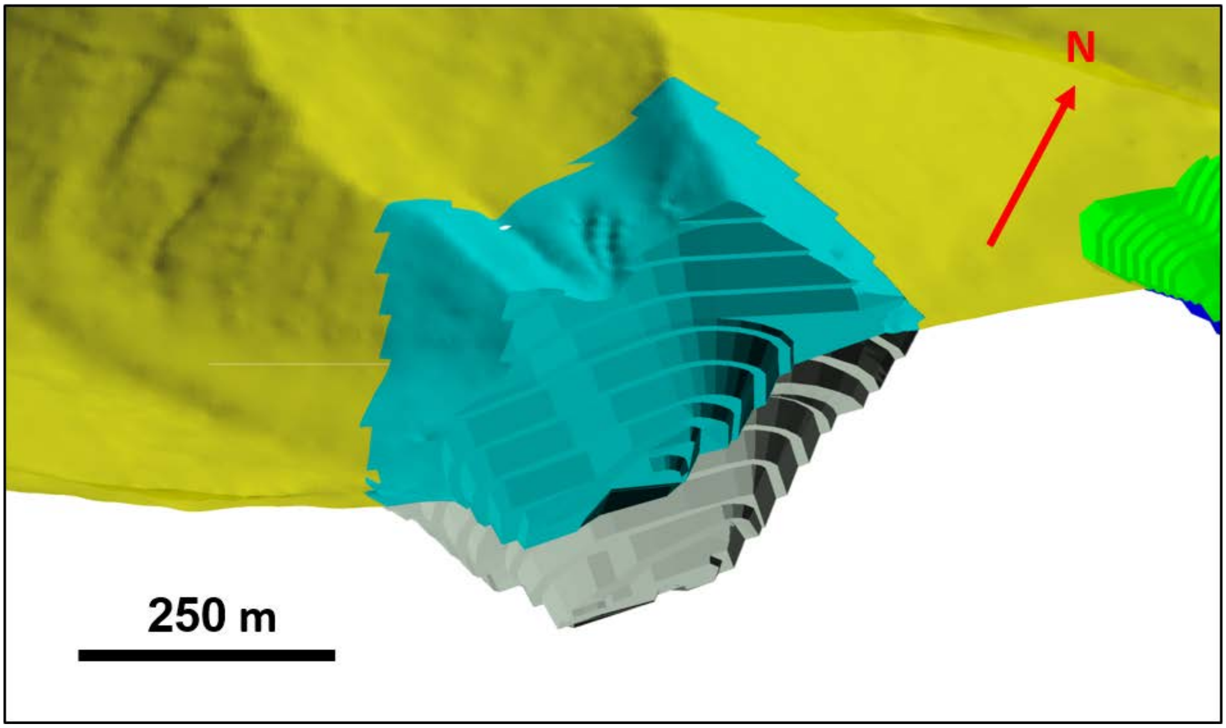
FIGURE 15-8 CROSS SECTION OF MAIN PIT ECONOMIC PIT SHELLS



The combined shells for the Central and Main Zone intermediate stages provided approximately 3.9 million tonnes of CIP ore, 3.4 million tonnes of BIOX® ore, and 18.1 million tonnes of waste for a total of 25.4 million tonnes. It was noted that the scheduling will focus on the CIP ore for the first two years and that the majority of BIOX® ore in the intermediate shell is spatially located at depth and will be encountered towards the end of the phase progression.

Figure 15-9 shows a cross section of the Phase 1 (blue) and Phase 2 (ultimate pit; grey) designs of the Main Pit.

FIGURE 15-9 MAIN PIT CROSS SECTION



DETAILED PIT DESIGN PARAMETERS

The geotechnical design recommendations made by AdiuvareGE are summarized in Table 15-4.

TABLE 15-4 OPTIMIZATION AND GEOTECHNICAL SLOPE CRITERIA

Slope Angles (degrees)	Bearing (Degrees)	Overburden	Rhyolite Competent Oxide	Rhyolite Transition	Rhyolite Fresh	Granite Competent Oxide	Granite Transition/ Fresh
Inter Ramp Slope Angles	45	29	44	48	51	41	48
	180	29	44	48	51	41	48
	225	29	44	48	44	44	48
	360	29	44	48	44	44	48
Pit Design	**Units**	**Overburden**	**Rhyolite Competent Oxide**	**Rhyolite Transition**	**Rhyolite Fresh**	**Granite Competent Oxide**	**Granite Transition/ Fresh**
Batter Angle	Degrees	50.0	60.0	65.0	60-70	55-60	65.0
Berm Width	m	5.0	9.0	9.0	9.0	9.0	9.0
Bench Height	m	5.0	20.0	20.0	20.0	20.0	20.0
Ramp Width (two lane)	m	22.0	22.0	22.0	22.0	22.0	22.0
Ramp Width (single lane)	m	12.0	12.0	12.0	12.0	12.0	12.0
Double Lane Ramp Gradient	%	10.0	10.0	10.0	10.0	10.0	10.0
Single Lane Ramp Gradient	%	10.0	10.0	10.0	10.0	10.0	10.0
Dump Design	**Units**	**All**					
Batter Angle	ratio	1:1.5					
Berm Width	m	20.0					
Bench Height	m	20.0					
Ramp Width (two lane)	m	22.0					
Double Lane Ramp Gradient	%	10.0					

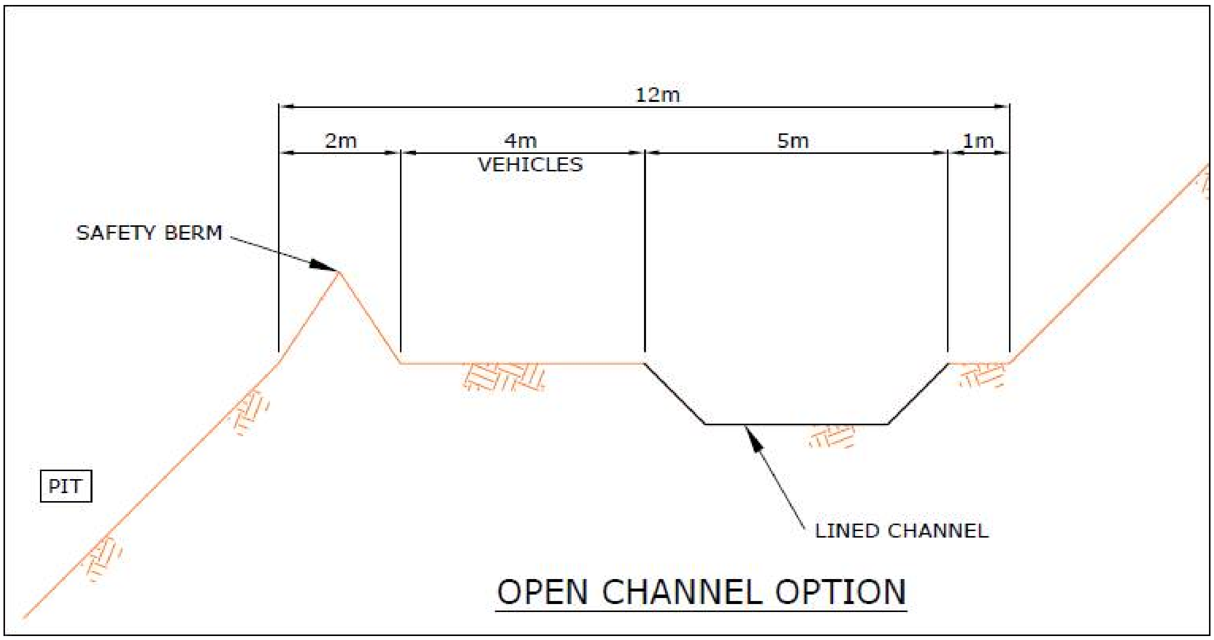
Source: Table 9-1 Recommendations for optimising bench face angles for certain zones, Page 34, Report on Gatsuurt Project Pit Slope Feasibility Study Update, AdiuvareGE, February 10, 2017.

Sustainability East Asia LLC (SEA) provided recommendations for the design and implementation of a water diversion bench to be incorporated into the pit designs, in the form of a memorandum with the subject “Options to Divert River Flows at Central Zone Pit” dated April 10, 2017 (Sustainability East Asia, 2017b). This bench is intended to provide a path for water travelling along the Gatsuurt River and its associated tributaries intercepting the upstream edge of the pit, to be channelled along a specially designed bench of the pit, so that it exits downstream and prevents flow into the pit void.

In summary, the slope of the diversion bench is to be a continuous fall of approximately 1.7% defined by the river bed level at the upstream end and the non-impacted water diversion channel continuing downstream of the pit. The memorandum outlines potential cross-sectional design criteria and options with recommendations.

The SEA recommended diversion bench cross-section is presented in Figure 15-10, which shows a total design width of 12 m, incorporating a 5 m wide open channel, lined where necessary; a 4 m wide vehicle running surface; and a 2 m wide safety berm.

FIGURE 15-10 SEA RECOMMENDED DIVERSION BENCH DESIGN CROSS SECTION



Following a review of the SEA diversion bench design criteria some modifications to the recommendations were made. The modifications were specifically related to the width of the vehicular berm of the diversion bench, necessitated by changes in some of the mining assumptions made after the SEA memorandum was published.

In an effort to reduce mining costs, it was decided to modify the ore haulage by placing ore on ROM stockpiles located near the pit exits rather than the initially planned single ROM located to the east of the Central Pit. This change presented certain requirements of the vehicular running surface, those being:

- The need to haul ore from the Main ROM location, along this diversion bench, to the Boroo plant;
- The need for sufficient width to allow safe two way travel with the ore haulage fleet;
- The intermittent need for travel of the larger mining haulage and support equipment along this same bench.

The last requirement is to allow for the efficient relocation of mining equipment between pits and also for travel between the pit and office/workshop locations. It is assumed that the travel of the larger production haulage trucks and support equipment along this bench would be only one way at a time and demand a level of operational control so that no other traffic is permitted to travel in the opposite direction while the larger equipment is traversing along the bench. To accommodate the larger equipment and dual direction of travel of the ore haulage fleet, the width of the vehicular running section of the berm needed to be increased. In order to have sufficient width for a 100 t class rear dump truck factor of 1.5 times the working width of a CAT 777 was used, which equated to approximately 9 m, allowing sufficient room for dual traffic flow of the ore haulage truck fleet, and single way travel of the pit production truck fleet. RPM also increased the width of the safety berm to allow for the larger production trucks to a width of 3 m. All other dimensions remained the same which resulted in the total design width of the revised diversion bench to be 18 m.

The pit phases and ultimate pit shells were designed based on the above geotechnical and diversion bench parameters with concern also given to numerous other factors including:

- Diversion bench location.
- Bench access considerations.
- Minimum working constraints.
- Waste minimization.
- Maximizing CIP ore contained within the ultimate shell to be included in the first phase of each pit.
- Minimizing the BIOX® ore within the first phase.
- Practicalities of mining the phases and ultimate designs.

MINERAL RESERVE CLASSIFICATION

As per the CIM guidelines, a Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. The Gatsuurt Mineral Resources do not contain Measured Mineral Resources, and as such the Indicated Mineral Resources have been converted to Probable Mineral Reserves.

Mineral Reserves were estimated using a gold price of US$1,250/oz within the designed ultimate pit shell, along with the relevant metallurgical recoveries, and costs outlined in Table 15-3.

CUT-OFF GRADES

A single cut-off grade was not used in the Project, as project costs and processing methodology (particularly variable processing recoveries) varied with ore type and grade. Instead, the blocks within the model were encoded using the same cash flow methodology used by Whittle 4X to determine the profitability of a block.

The Whittle 4X cash flow method assesses the value of processing each parcel of material through each process stream and mining the material as waste. The block value considers mining and processing costs outlined in Table 15-3, revenue using a gold price of US$1,250/oz, and individual block process recoveries. It then selects the stream that produces the least cost or most profit.

Oxide flagged blocks may possess a recovery factor greater than 0% for the CIP process with all blocks having a 0% recovery for BIOX® processing. Conversely, fresh or sulphide blocks may possess a recovery factor greater than 0% for BIOX® but will have a 0% recovery factor for CIP processing. In both of these material types, Whittle and the block coding logic will evaluate the value of each block by subtracting the cost of processing from its expected revenue. If the block produces a positive value for processing, it is flagged as being processed through the respective process stream (CIP for oxide material and BIOX® for sulphide/fresh material). If the block produces a negative value, it is flagged as waste material.

Transitional flagged material has the possibility of being processed through both the CIP and BIOX® streams and consequently may possess recovery factors for both CIP and BIOX® processing. For this material, both the Whittle cash flow method and the block coding logic will evaluate the expected revenue from both processes and determine the stream that has the higher value, flagging the block for that particular process. Depending on the individual blocks' properties and resulting economics, some transitional blocks are processed through the CIP stream, some processed through the BIOX® stream, and the remaining being categorized as waste.

SUB-GRADE MATERIAL

The Project has a further classification of material that is below the economic value to be classified as ore, but would be economic at the gold price used to define the Mineral Resources. This is referred to as "Sub-Grade Material".

This material uses the same logic described in the "Cut-Off Grades" above using the optimization costs outlined in Table 15-3 but at a gold price of US$1,450/oz.

The Sub-Grade Material at Gatsuurt includes only Indicated Mineral Resources, is not included in the declared Mineral Reserves, and is considered a sub-classification of waste material.

LOSS AND DILUTION

A Mining Accuracy Sensitivity Analysis was performed to model ore loss and dilution for a range of mining methods, from highly selective methods with small equipment to bulk mining methods. The analysis varied the mining accuracy along a simulated hard defined ore-waste contact, such as the fault zones present in the Gatsuurt deposit.

The mining accuracy study indicated that the highest project value was delivered through a selective mining method which minimized the ore loss and dilution. It indicated that the target smallest mining unit (SMU) based on the current geological model should be 5 m (x-direction) by 5 m (y-direction) by 5 m (height). In the QP's opinion this is a reasonable SMU for selective mining of the deposit.

For the final estimation of ore loss and dilution, where the geological model block size equalled the SMU, no ore loss and dilution was assumed as given the accuracy of the selective mining approach. Where the block model had ore parcels less than the SMU, a mining accuracy along the ore : waste boundary of 2 m was assumed. That is, 2 m of dilution would occur along the contact. A similar approach was also taken along the fault contact with a mining accuracy of 2 m.

The modelling approach of the mineralization used by RPA set the majority of the ore blocks at the SMU size. As such, the resulting modelled ore loss assuming selective mining is 4.7% and waste rock dilution is 1.0%. The QP recommends further analysis of ore loss and dilution once operational data becomes available.

MINERAL RESERVE ESTIMATE

The Mineral Reserve estimate for the Project is summarized in Table 15-5.

TABLE 15-5 MINERAL RESERVES BY ZONE AS OF OCTOBER 31, 2017

Classification	Zone	Process Type	Tonnes (kt)	Grade (Au g/t)	Contained Au (koz)
Probable	CZ	CIP	3,504	2.3	256
		BIOX®	5,536	3.2	573
		Sub-Total	**9,040**	**2.9**	**829**
Probable	MZ	CIP	1,342	1.9	83
		BIOX®	4,974	2.5	404
		Sub-Total	**6,316**	**2.4**	**487**
TOTAL Probable (CZ+MZ)			**15,356**	**2.7**	**1,316**

Notes:
1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves are estimated using an average long-term gold price of US$1,250 per ounce and a US$/₮ exchange rate of 2,200 and have been validated by the positive project NPV that uses a 5% discount rate.
3. Numbers may not add due to rounding.

The above Mineral Reserve estimate should be considered within the context of the outcomes stated in this report. Of note, as outlined in Section 22, the Project's base case economic modelling results indicate a marginally positive NPV of $39.5 million at an annual discount rate of 5%. Should any of the key revenue drivers (price, grade, recovery) incur any adverse variance to the plan, this will place the above estimate at risk.

The results of the economic analysis to support Mineral Reserves represent forward looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Uncertainty that may materially impact Mineral Reserve estimation include realized prices, market conditions, capital and operating cost estimates, foreign exchange rates, resource model performance, recoveries, and the timely and successful implementation of recommended actions.

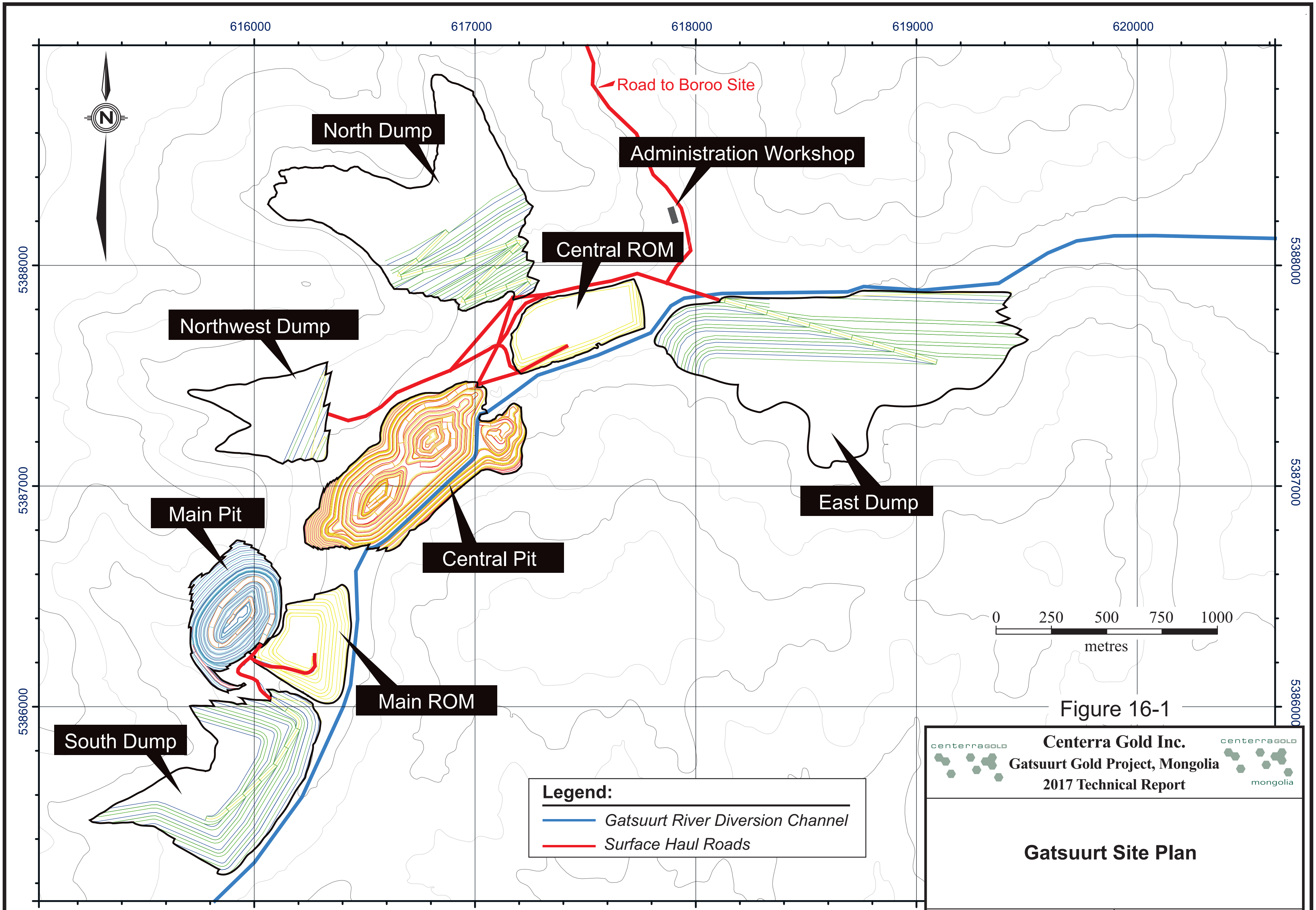
16 MINING METHODS

INTRODUCTION

The Project will be operated using a traditional Contractor-operated truck and shovel/excavator mining method. This is considered to be the most appropriate method as it is robust, commonly understood, and provides the required flexibility for the Project with regard to capacity increases and decreases, and shifts of mining location.

The general site layout is shown in Figure 16-1. The Project is comprised of the Central and Main Pits which will have a ROM ore storage area near each of the two pit exits. As well, there will be four dumps for ex-pit storage of waste material. A diversion bench through the Central Pit will be constructed to prevent interruption of the Gatsuurt River water flow through the Project. Surface haul roads will connect the pits to the dumps and ROM areas. An existing haulage road connects the Project to the Boroo processing facility, 52 km away.

The expected life of the Project is approximately nine years of mining (106 months duration) with ore feed, post-mining, extending the operation out to 10 years (119 months). Total ore mined and fed to process during this period is 15.4 million tonnes at an average grade of 2.7 g/t Au, requiring the removal of 72.6 million tonnes of waste and sub-grade material for a total material movement of 87.9 million tonnes.



Source: RPM Global, 2017.

GEOTECHNICAL SLOPE DESIGN

INTRODUCTION

The geotechnical slope designs contained in this section build upon pit slope design recommendations provided for the Project by SNC-Lavalin in 2005 (SNC-Lavalin, 2005). SNC-Lavalin completed the designs to a feasibility level of study and this background information has been referenced and relied upon, where appropriate, as part of the slope design update. This update was completed by AdiuvareGE in 2017 and used not only the SNC-Lavalin report, but also the results from additional data acquisition and analysis completed as part of the Company's 2016 field program.

2016 FIELD PROGRAM

In 2016, an additional geotechnical field program was carried out at Gatsuurt to build upon and confirm the existing geotechnical open pit slope designs for the Project. The field program was an update to SNC-Lavalin's field investigation and pit slope designs that were carried out in 2005.

The 2016 geotechnical field program included a total of twelve drill holes with four holes drilled in the Main Zone for a total of 1,657 m and eight holes drilled in the Central Zone for a total of 1,657 m. The drill hole locations for the Main and Central Pits are presented in Tables 16-1 and 16-2, respectively.

TABLE 16-1 SUMMARY OF GEOTECHNICAL DRILL HOLES - MAIN ZONE PIT

Hole-ID	Easting	Northing	Elev.	Inc.	Azimuth	Depth (m)	Instrumentation
GT-536GT	615,859	5,386,329	1,328	-50	192	145	None
GT-539GT	615,989	5,386,432	1,349	-51	359	175	None
GT-550GT	615,886	5,386,421	1,349	-55	259	185	Piezometer
GT-553GT	615,887	5,386,399	1,341	-50	127	185	Piezometer

Note: Oriented Core logging, point load testing, and packer testing completed on all holes

TABLE 16-2 SUMMARY OF GEOTECHNICAL DRILL HOLES – CENTRAL ZONE PIT

Hole-ID	Easting	Northing	Elev.	Inc.	Azimuth	Depth (m)	Instrumentation
GT-541GT	616,875	5,387,327	1,247	-55	090	200	None
GT-544GT	616,613	5,387,034	1,259	-50	140	225	None
GT-557GT	616,777	5387197	1,258	-52	128	280	Piezometer
GT-565GT	616,780	5,387,257	1,253	-61	308	216	Piezometer
GT-571GT	616,537	5,387,012	1,270	-70	315	230	Nested Piezometers
GT-578GT	616,418	5,386,908	1,287	-61	308	216	Piezometer
GT-583GT	616,887	5,387,112	1280	-65	128	150	Piezometer
GT-603GT	616,738	5,386,884	1308	-55	140	140	None

Note: Oriented Core logging, point load testing, and packer testing completed on all holes

Oriented core logging and packer testing were completed on each hole. Point load testing was undertaken on representative rock specimens with a target to complete two point load tests every five metres (one axial and one diametral). Rock specimens were also collected for laboratory testing purposes. Vibrating wire piezometers were installed in select holes for long term monitoring of ground water levels.

Laboratory testing was completed by Soil Trade LLC in Mongolia. All drill holes were logged by Soil Trade LLC personnel under Centerra's supervision. Laboratory testing was conducted on selected core specimens to determine intact geotechnical properties.

The Central Pit is the larger of the two pits and is elliptical in shape with a northeast-southwest axis and is approximately 800 m long (Figure 16-2). The Main Pit is more oval in shape, again with a northeast-southwest axis, and is approximately 475 m long. Both pits have a maximum elevation change from pit floor to crest of approximately 250 m, however, in general, pit depths are in the region of 150 m to 160 m.

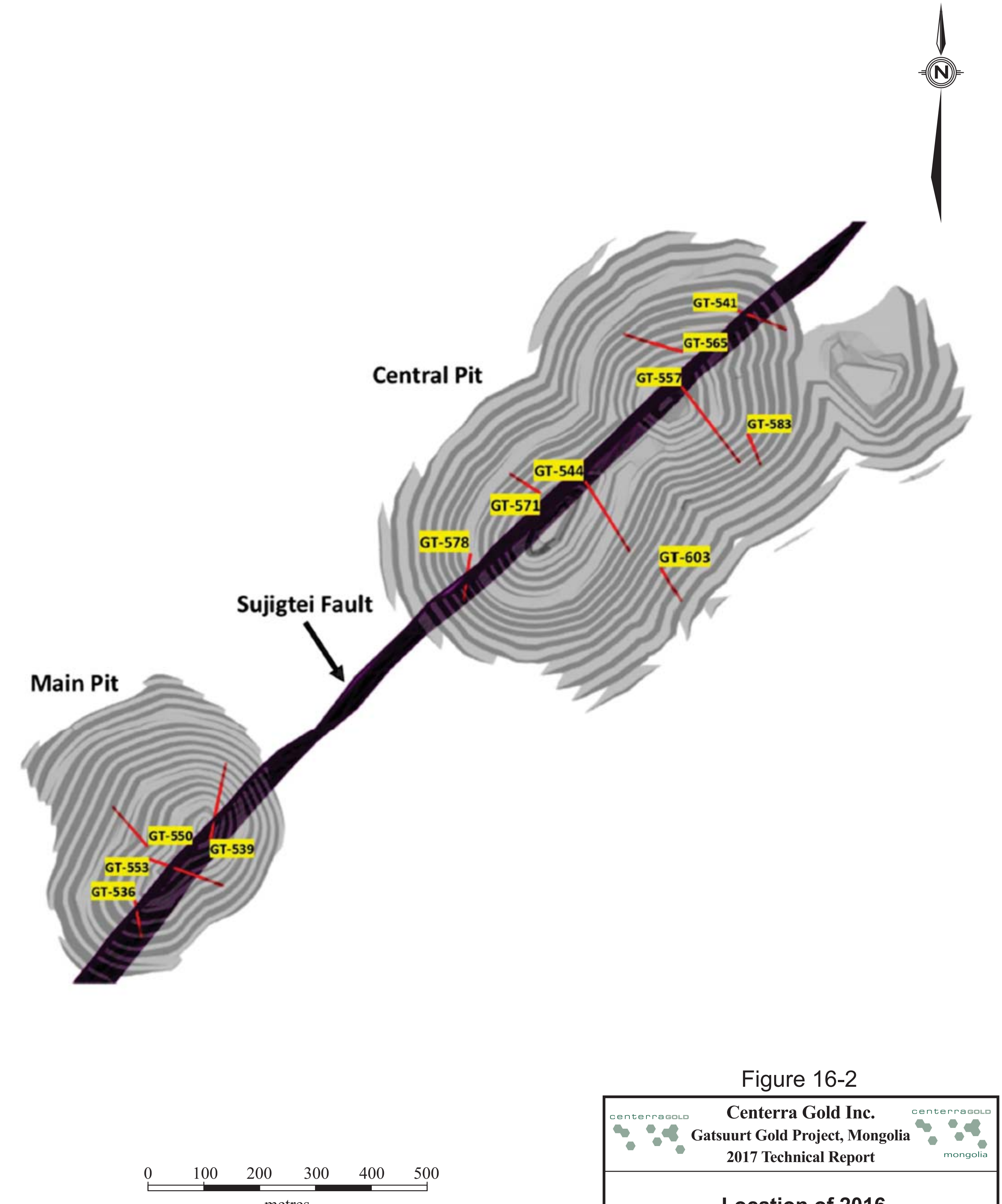


Figure 16-2

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Location of 2016 Geotechnical Drill Holes (Main and Central Zones)

Date: December 2017 | File: 2016 Geotech DDH.cdr

The parameters that were routinely collected as part of the field investigation include:

- Total Core Recovery % (TCR)
- Rock Quality Designation % (RQD)
- Fracture Frequency (FF)
- Strength Index (SI)
- Point Load Strength Index IS(50)
- Weathering Index (WI)
- Micro Defects Intensity (MDI)
- Micro Defects Strength (MDS)
- Permeability K (m/sec)
- Rock Mass Rating (RMR 1976)

GEOTECHNICAL CHARACTERIZATION

A geotechnical model was established for the deposit using the available data that included the following components:

- Lithological information and pit geology
- Geological structure
- Material properties determined from the assessment of core as well as laboratory testing
- Rock mass classification
- Rock mass properties
- Hydrogeological information

Geotechnical domains have been selected based on lithology and the degree of oxidation, namely the oxidized zone, transition zone, and the fresh zone. As mentioned in the previous section, the rhyolites were treated as a single domain and the granites and metasediments were combined into a single domain. The combining of granites with the metasediments was further substantiated by the results of the laboratory data and rock mass classification which indicated that these lithologies had similar geotechnical strength characteristics. These units have been classified as being poor to good.

For illustrative purposes, Table 16-3 provides the average values determined for the various geotechnical parameters logged by domain.

TABLE 16-3 SUMMARY OF AVERAGE GEOTECHNICAL PARAMETERS LOGGED BY DOMAIN

Lithology	Oxidation State	Number Intervals	TCR (%)	RQD (%)	FF	SI	WI	MDI	MDS	RMR(76)
Rhyolite	Oxide	90	94	34	14	2	3	2	0	35
Granite	Oxide	40	95	51	13	3	3	1	0	38
Rhyolite	Transition	327	99	68	11	4	2	1	1	53
Metasediments	Transition	190	100	75	9	4	2	1	1	58
Granite	Transition	80	99	80	7	4	2	1	1	63
Rhyolite	Fresh	107	100	76	8	5	1	1	1	63
Metasediments	Fresh	312	100	78	8	4	1	1	1	62
Granite	Fresh	54	100	79	8	4	1	1	1	60

Notes:
1. TCR - Total Core Recovery %
2. RQD - Rock Quality Designation %
3. FF - Fracture Frequency
4. SI - Strength Index
5. WI - Weathering Index
6. MDI - Micro Defects Intensity
7. MDS - Micro Defects Strength
8. RMR - Rock Mass Rating

The geology of the underlying area is predominantly Permian rhyolite and Boroo complex granodiorite, granites and diorites and Kharaa Series metasedimentary rocks. The pit area is characterized by the presence of two main lithostructural units separated by a sub-vertical active fault known as the Sujigtei fault. Volcanics (Rhyolite) are found on the northwest side of the fault, whereas Boroo granite occurs on the southeast side. These rocks display variable rock weathering profiles varying with depth from highly weathered at the top to fresh competent rock. Where unexposed, the bedrock is overlain by colluvial and/or alluvial deposits consisting of sand and gravel with traces of clay and silt.

The main structure in the area is the Sujigtei fault as mentioned above. Other structures have been identified and measured during core logging, and generally include faults, shears, veins, joints, bedding, and foliation. The interpretation of faulting within the rhyolites is not well understood. This is an area that the QP recommends further study be completed during start up by a qualified structural geologist.

Structural sets were determined for the slope stability analysis using the results from oriented core data. Results from the structural assessment indicate that the dominant structures tend to strike parallel with the steeply dipping Sujigtei fault, but vary from moderate to steeply dipping. Other major structural sets were also identified. Structural results, as well as visual assessment of the core photographs, suggests that the rock mass will be blocky and kinematic stability will be a driving factor for the pit slope design.

It should be noted that the weathering profile (in particular, weathering along joints), as well as the existence of micro-fractures noted in the core, may influence rock mass stability at a bench and inter-ramp scale. The QP has recommended that additional discontinuum numerical modelling would be prudent once rock exposures have been mapped during early mining.

Laboratory testing was conducted on selected core samples to determine intact geotechnical properties. All geotechnical testing was conducted by Soil Trade LLC in Mongolia. Testing that was carried out included: Brazilian tensile, uniaxial compressive strength, tri-axial strength and shear strength testing on selected joint surfaces. In addition to the laboratory testing, point load tests were carried out by Soil Trade LLC, to provide a basis for determining intact rock mass strength at frequent core intervals (testing targeted every 5 m interval). All laboratory testing and results were compiled and statistical analyses was performed in spreadsheets.

A summary of the geotechnical parameters determined for the different geotechnical domains is shown in Table 16-4.

TABLE 16-4 SUMMARY OF MATERIAL PROPERTIES BY GEOTECHNICAL DOMAIN

Domain	Mean UCS (MPa)	GSI	Mohr Coulomb (Φ)	Mohr Coulomb (C)	Hoek and Brown (mb)	Hoek and Brown (s)	Hoek and Brown (a)
Granite Oxide	33	38	40	0.33	0.749	0.001	0.513
Granite Transition	59	63	48	1.40	2.425	0.016	0.502
Granite Fresh	49	60	47	1.13	2.420	0.012	0.503
Metasediment Oxide	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Metasediment Transition	58	58	43	0.78	1.248	0.002	0.509
Metasediment Fresh	58	62	48	1.35	2.435	0.015	0.502
Rhyolite Oxide	31	35	39	0.30	0.70	0.0007	0.516
Rhyolite Transition	67	53	49	1.10	2.113	0.005	0.505
Rhyolite Fresh	75	60	51	1.66	2.91	0.016	0.502

This data was subsequently used for rock mass classification and geotechnical assessment. Data reduction was carried out and compiled and formed the basis of the slope design recommendations that were provided by AdiuvareGE in their 2017 slope design recommendation report (Adiuvare, 2017).

The hydrogeological data captured during the field investigation included packer testing and the placement of piezometers. These data have been incorporated in the stability analyses performed as part of the stability evaluation work, where appropriate. The piezometer data was used to set the phreatic surface during stability analyses. The packer testing indicated that in general rock permeability was low with higher permeability associated with geological structures.

A summary of hydraulic conductivities measured during the packer testing program is shown in Table 16-5. The values were used for limit equilibrium modelling, where appropriate.

TABLE 16-5 SUMMARY OF HYDRAULIC CONDUCTIVITIES FROM PACKER TESTING

Pit	Lithology	Oxidation State	Number of Intervals	Ave k (m/s)	Min. (m/s)	Max. (m/s)	St. Dev. (m/s)
Main	Rhyolite	Oxide	3	3.93E-04	0	1.00E-03	5.34E-04
Main	Rhyolite	Transition	32	2.56E-05	1.28E-06	1.27E-04	3.34E-05
Main	Metasediment	Transition	4	1.77E-05	7.41E-06	4.75E-05	1.98E-05
Main	Rhyolite	Fresh	1	1.31E-05	1.31E-05	1.31E-05	-
Main	Metasediment	Fresh	13	3.23E-05	1.38E-06	6.60E-05	2.26E-05
Main	Granite	Fresh	3	1.18E-05	1.91E-06	1.95E-05	9.00E-06
Central	Rhyolite	Oxide	4	6.90E-05	0	2.44E-04	1.18E-04
Central	Granite	Oxide	1	2.93E-05	2.93E-05	2.93E-05	-
Central	Rhyolite	Transition	29	8.12E-05	0	2.73E-04	6.20E-05
Central	Metasediment	Transition	33	3.80E-05	0	1.40E-04	3.92E-05
Central	Granite	Transition	21	5.22E-05	0	1.89E-04	5.37E-05
Central	Rhyolite	Fresh	20	7.52E-05	6.71E-06	2.05E-04	7.05E-05
Central	Metasediment	Fresh	49	6.52E-05	1.30E-05	2.76E-04	4.44E-05
Central	Granite	Fresh	8	7.25E-05	2.50E-05	1.58E-04	4.41E-05

KINEMATIC ASSESSMENT

The susceptibility of kinematic failure mechanisms due to planar, wedge, and toppling failure was assessed using the Rocscience Dips Software (Dips). This program requires structural data in the form of dip and dip directions of discontinuities within the rock mass, as well as expected friction angles for these discontinuities. Friction angles were developed as input parameters from the available data which included comparison of joint parameters Jr and Ja logged from the core, as well as friction angles provided by direct shear tests on selected joint surfaces.

Table 16-6 provides a summary table of the friction angles determined from the logged Jr and Ja parameters, where the friction angle can be approximated using the relationship:

$$\text{Friction angle } (\Phi) = \tan^{-1}(Jr/Ja)$$

Also shown in Table 16-6 is a summary of the friction angles determined from direct shear tests on selected joint surfaces in the laboratory.

TABLE 16-6 FRICTION ANGLES SELECTED AS INPUT PARAMETERS TO THE KINEMATIC ASSESSMENT

Domain	**UCS (MPa)**	**RMR(76) GSI**	**Jr**	**Ja**	**Φ (Jr/Ja)**	**Φ** (Lab)	**Φ (Model)**
Granite Oxide	33	38	2.1	3.6	30	29	28
Granite Transition	59	63	2.2	2.9	37	34	30
Granite Fresh	49	60	2.1	3.5	31	31	35
Metasediment Oxide	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Metasediment Transition	58	58	1.9	3.6	28	30	30
Metasediment Fresh	58	62	2.1	2.9	37	34	35
Rhyolite Oxide	31	35	2.1	4.3	26	30	26
Rhyolite Transition	67	53	2.1	3.4	31	36	30
Rhyolite Fresh	75	63	2	3.9	27	30	30

Notes:

1. Granite transition was downgraded to more closely match metasediment transition (have similar properties with more data on metasediment).
2. Granite fresh upgraded to more closely match metasediment fresh (have similar properties with more data on metasediment and occur on same wall).
3. Rhyolite oxide based on lab results.
4. Rhyolite transition and Rhyolite fresh have same friction angle based on Jr/Ja logging results and a small number of lab tests on the fresh material.

To provide a means of assessing potential kinematic failure for different slope azimuths and different failure mechanisms, design charts were assembled which allowed a quick assessment of potential failures.

The kinematic design process entailed filtering the structural data for faults, shears, foliation, and joints to provide a structural data set for subsequent analyses. These data were further sorted for the selected lithologies and geotechnical domains; namely the rhyolite (competent oxide, transition, and fresh) and granite and metasediments (competent oxide, transition, and fresh).

Dips was then used to assess the potential for planar, wedge, and toppling failure for different bench face angles and for different slope azimuths. Bench face angles of 55° to 90° were assessed using increments of 5°. Slope azimuths were assessed from 0° to 360° in 10° increments.

The output from Dips included the number of critical poles for the particular failure mechanism being assessed. In order to plot the results in a meaningful way the critical poles were normalized to provide a percentage of the maximum pole value provided from the assessment. For example, for the Rhyolite oxide domain and for planar failure it was found that a critical pole maximum of 15% was obtained. The results of the critical poles determined for each bench face dip and slope dip direction were then normalized to this maximum percentage, i.e., the 15% maximum value equated to 100% for plotting purposes.

A design circle of 50% was determined to be appropriate to estimate the bench face angles. This suggests that 50% of the critical poles will likely contribute to planar failure at a bench scale. Until site specific data can be used to calibrate these design charts the value of 50% has been used to provide design recommendations at a feasibility study level.

STABILITY EVALUATION

Overall slope stability was then assessed using limit equilibrium techniques utilizing commercially available software. From this information, bench face and inter-ramp angles, as well as overall slope angles, could be assessed and recommendations made.

Evaluation of the rock mass and results of the stability analyses indicate that the rock quality and structural conditions will support moderate to steep inter-ramp angles. A summary of key aspects includes:

- Rock mass strength varies and slope angles will reflect this with shallower slopes to be developed in the weaker material.
- Groundwater is likely to influence slope stability and has been included in the stability analyses. Based on visual assessment of the core it is likely that, due to the geological structure, the rock mass is likely to drain naturally due to dewatering for operating purposes. However, the development of pore pressures in the slope cannot be ruled out and provision should be made for drainage measures to be incorporated in some slopes.

- Seismicity has also been included in the stability analyses. Ground acceleration values have been based on an Operating Basis Earthquake (OBE) selected to correspond to an earthquake of 1 in 475 year return period (10% probability within a given 50 year period) with an estimated peak (maximum) ground acceleration of approximately 0.12g.

- Kinematic stability analyses indicate potential for significant planar and wedge type failures. There is also potential for toppling type failures in some areas of the pits.

SLOPE DESIGN RECOMMENDATIONS

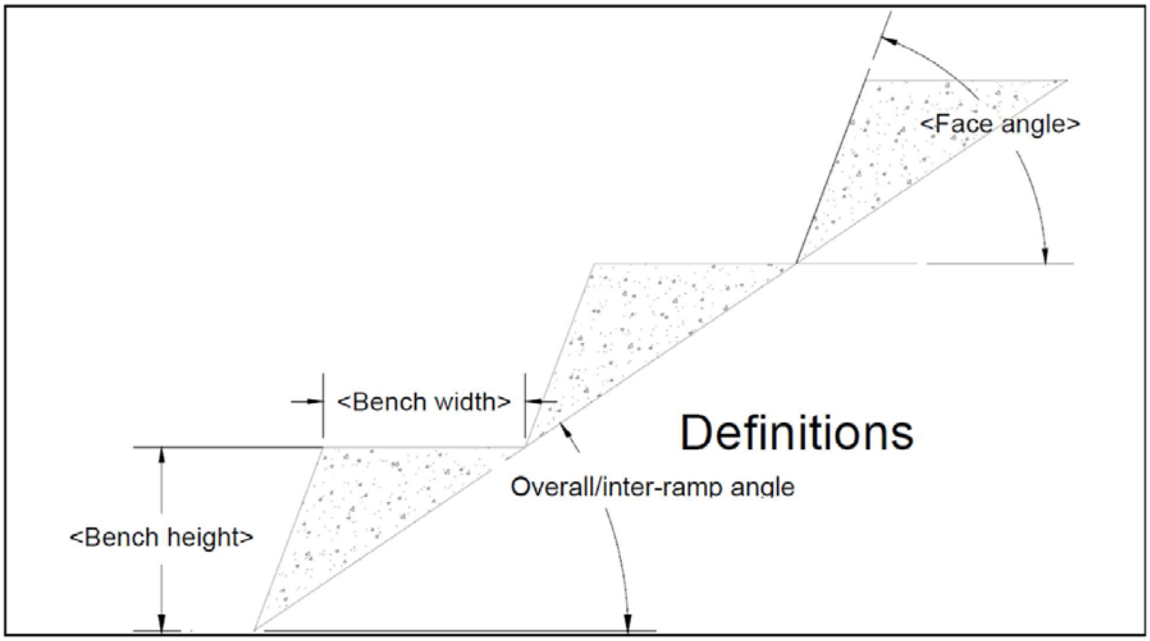
The overburden material was not assessed in this study as additional data was not collected during the 2016 drilling campaign. Therefore, the SNC-Lavalin recommendations for bench face angles have been repeated in Table 16-7. The recommendations for bench widths, however, differ in this report, as it is the QP's opinion that larger bench widths will be required in this material to facilitate catchment of sloughed material and the possible placement of drains for the catchment of surface water especially at the interface between highly weathered material and competent oxide. The SNC-Lavalin field program included extensive overburden drilling and laboratory testing to determine the friction angles and cohesion values used in their analyses. Additional testing was not completed as it would have been redundant at this time.

In addition, it should be noted that in this study a distinction has been made between oxide and competent oxide. Based on assessment of the core and the interface wireframes, it is evident that, in some cases, the base of overburden is not necessarily the base of the highly weathered material; both will behave in a similar way from a rock slope performance perspective.

The QP recommended that the base of the highly-weathered material is determined from all available drill hole data and a wireframe developed for this surface. This will provide a useful tool for further slope design adjustments.

The slope design definitions adapted for this study are given in Figure 16-3.

FIGURE 16-3 SLOPE DESIGN DEFINITIONS



The recommended slope design is shown in Tables 16-7 and 16-8.

TABLE 16-7 SLOPE DESIGN RECOMMENDATIONS FOR OVERBURDEN AND HIGHLY WEATHERED MATERIAL

Pit	Zone	Domain	Slope Azimuth (deg.)	BFA (deg.)	Bench Width (m)	Bench Height (m)	IRA (deg.)
Main & Central	Overburden & all highly weathered material	All domains	All	50	5	5	29

TABLE 16-8 SLOPE DESIGN RECOMMENDATIONS FOR BEDROCK

Pit	Zone	Domain	Slope Azimuth (deg.)	BFA (deg.)	Bench Width (m)	Bench Height (m)	IRA (deg.)
Main & Central	I	Rhyolite Competent Oxide	045-225	60	9	20	44
Main & Central	II	Rhyolite Transition	045-225	65	9	20	48
Main & Central	III	Rhyolite Fresh	045-180	70	9	20	51
Main & Central	IV	Rhyolite	180-225	60	9	20	44

Pit	Zone	Domain	Slope Azimuth (deg.)	BFA (deg.)	Bench Width (m)	Bench Height (m)	IRA (deg.)
		Fresh					
Main & Central	V	Granite Competent Oxide	225-360	60	9	20	44
Main & Central	VI	Granite Competent Oxide	360-045	55	9	20	41
Main & Central	VII	Granite Transition	225-045	65	9	20	48
Main & Central	VIII	Granite Fresh	225-045	65	9	20	48

Note: a geotechnical berm should be incorporated in the steeper slopes to decouple the slopes for operational safety. This will impact the overall slope angles. It is recommended that a geotechnical berm or ramp be placed every 100 m.

WASTE DUMPS

The waste rock material from the Main and Central pits will be directed to four waste dumps denoted the East, North, North-West, and South facilities (Figure 16-1). The waste dump locations are considered to be the best available options from an operational standpoint taking into account the valley's topography, hydrological conditions, and haulage distances from the pit's rims. The final height of the waste dumps varies from 55 m (South dump) to 110 m (North dump). The only dump that is currently permitted is the East dump. Of note, the permitted East dump has sufficient capacity to accommodate all of the waste materials produced over the life of mine (LOM).

A variety of engineering studies and drilling programs (geotechnical, hydrological, and environmental) were carried out on the Property and contributed to a better understanding of the expected East waste dump behaviour pattern over the LOM period. This understanding has been applied to the other three waste dumps under the assumption that subsurface conditions are the same and do not vary significantly from one location to the next. Thus, the cross sectional geometry of the overburden and bedrock domains, as well as the strength properties of the major materials were estimated based on geotechnical drilling programs carried out in 2005, and most recently in 2016 by Soil Trade LLC (Mongolia). Material strength parameters used for the stability assessment are illustrated in Table 16-9.

TABLE 16-9 STRENGTH PROPERTIES OF THE MATERIALS

Material	Unit weight (kN/m^3)	Cohesion (kPa)	Friction Angle (°)	Source
Waste rock	22.0	a=1.7456	b=0.8977	Leps, 1970-average
Overburden	21.0	0.0	32	Soil Trade LLC, 2005, 2016
Bedrock	27.0	0.0	40	Centerra, 2017

A better understanding of the subsurface lithology and hydrogeological conditions is required during the detailed engineering stage to confirm the design criteria. Stability calculations determined that the proposed waste dump configuration results in a stable facility for the duration of the mine operation even though conservative assumptions, with regard to expected water table and seismic load, were used in the course of the analyses (Table 16-10 and Figure 16-4). The analyses were performed using a computer software program, “SLIDE”, Version 6, developed by RocScience Inc. of Toronto, Ontario, using the Morgenstern-Price analysis method. The highest sections of the rock dumps were selected to represent the most critical waste rock slopes.

TABLE 16-10 RESULTS OF STABILITY CALCULATIONS

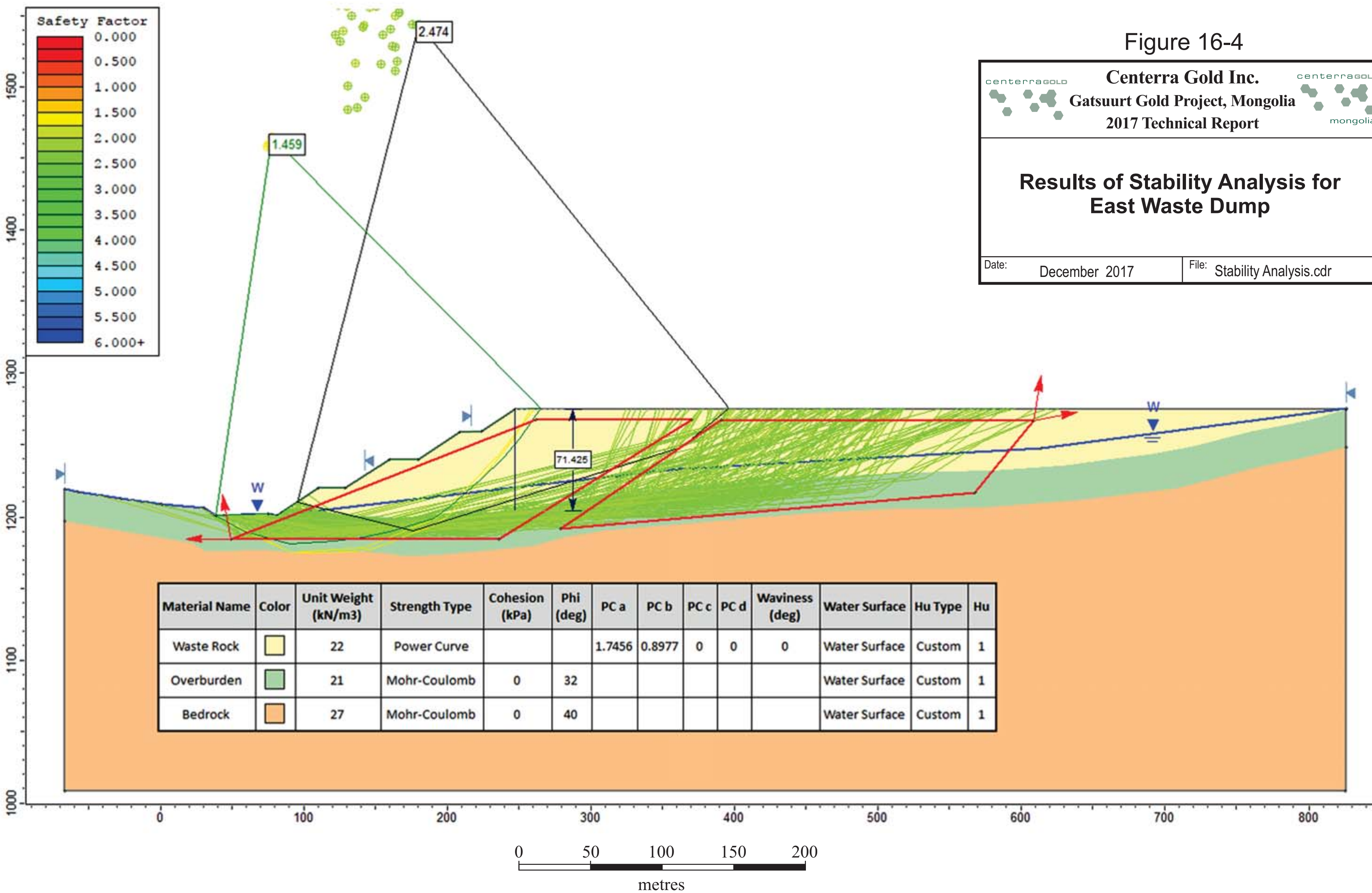
Waste Dump	Final height, m	FOS if fails at the face	FOS if fails through contact
East	70-75	1.46	2.07
North-West	60-65	1.36	1.99
North	105-110	1.31	1.96
South	55-60	1.21	1.47

Figure 16-4

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Results of Stability Analysis for East Waste Dump

Date: December 2017 | File: Stability Analysis.cdr



Material Name	Color	Unit Weight (kN/m3)	Strength Type	Cohesion (kPa)	Phi (deg)	PC a	PC b	PC c	PC d	Waviness (deg)	Water Surface	Hu Type	Hu
Waste Rock		22	Power Curve			1.7456	0.8977	0	0	0	Water Surface	Custom	1
Overburden		21	Mohr-Coulomb	0	32						Water Surface	Custom	1
Bedrock		27	Mohr-Coulomb	0	40						Water Surface	Custom	1

The minimum required factor of safety (FOS) for the waste rock dump slope was selected to be 1.3 under static conditions and 1.1 under seismic loading conditions, based on commonly accepted engineering practice for non-water retaining earth embankments.

Short-term deformations may still happen, but to a degree that is considered acceptable. Although further investigation is required, the minimum FOS for the ultimate waste dumps are considered acceptable. Additional field data should be collected, tested, and incorporated into the waste dump models, and the dump designs should be adjusted as required.

During the operation stage, appropriate monitoring will be carried out using prisms to identify any accelerations of the dump. If any acceleration is identified in a particular section of the waste dump, it will be closed off and dumping will not be permitted until an acceptable decrease in movement rate is observed.

GRADE CONTROL

Grade control activities were not discretely scheduled in the Mine Plan, however, it is anticipated that these activities will be completed using blast hole sampling. For the purposes of scheduling, it is also expected that these activities will be completed on a just-in-time basis and that there will be sufficient capacity available to ensure that the mine production activities are not impeded.

MINING METHOD

The Project will be operated using a traditional truck and shovel/excavator mining method. This is considered to be the most appropriate method as it is robust, commonly understood, and provides the required flexibility for the Project with regards to capacity increases and decreases, and shifts of mining location.

The primary production equipment fleet will excavate waste material and haul to waste storage dumps, and transport ore to ROM pads near the pit exits. A separate fleet of on highway equipment will be utilized for ore re-handle and haulage from the ROM locations to the Boroo processing facility. Although the processing facility is 52 km from the Project, the calculated haulage distance from pit stockpiles to the processing stockpiles to be 55 km.

Pit mining will be conducted by blasting 10 m benches that are mined using several flitches (production bench height) of appropriate height for the equipment selected. The main production machines are a number of excavators matched with 50 t class rear dump trucks. These will be supported by ancillary equipment such as dozers, graders, front end loaders (FELs), and water trucks.

Ore re-handle from the ROM stockpile locations to the Boroo processing facility will be completed by a separate fleet that consist of a series of FELs and 40 t on-highway haulage trucks.

The proposed fleet compliment is further described in the section “Mine Equipment”.

PIT PHASE DESIGN

The phase design was completed with consideration for the following factors:

Recommended geotechnical design parameters.

- Diversion bench location.
- Access considerations.
- Minimum working constraints.
- Waste minimization.
- Maximizing CIP ore contained within the ultimate shell to be included in the first phase of each pit.
- Minimizing the BIOX® ore within the first phase.
- Practicalities of mining the phases and ultimate designs.

WASTE DUMP DESIGN

The waste dumps were designed according to the geotechnical recommendations outlined in Table 15-4.

Further details concerning the waste dumps are included in the sections “Waste Rock Storage” and “Disposal of Potential Acid Generating (PAG) Material”.

MATERIAL CLASSIFICATION

As discussed in the section “Cut-Off Grades”, each block within the block model was encoded using the same cashflow methodology utilized by Whittle 4X.

The cash flow method assesses the value of processing each parcel of material through each process stream as well as mining the material as waste. It then selects the stream that produces the least cost or most profit (i.e., the highest cashflow). Only the Indicated Resource blocks that produced a positive cash flow result were flagged and processed as either CIP ore, or BIOX® ore, depending on the best result; of the remaining blocks those that produced a positive cashflow for processing using a gold price of US$1,450/oz were flagged as either sub-grade CIP (SGCIP) or sub-grade BIOX® (SGBIOX). The blocks remaining after this process were deemed to be waste as they were a combination of either sub-economic grade, had insufficient recovery (or no recovery attributed), or were classified as Inferred Resources.

The material classification used the optimization assumptions for the cost inputs, outlined in Table 15-3, and the block recoveries and a gold value of US$1,250/oz for the revenue calculations. It is recommended by the QP that the material flagging process be reviewed and updated once more accurate cost information becomes available and at any time there is a notable change in either costs or the gold price. This should be done with consideration of the potential effect on reported Mineral Reserves.

Preliminary investigative geochemical studies suggested that acid generation of waste, or acid rock drainage (ARD), was a concern for the Project. As such, the waste material required further categorization into either PAG or non-acid generating (NAG) material. This classification was necessary to ensure adequate engineering design and operational treatment of the waste to mitigate acid generation and prevent potential environmental damage as a result.

Initial coding guidance used the “rocktype” numerical field in the supplied block model to identify the acid generation potential of the host rock. It was assumed that all rock was acid generating with the exception of only the meta-sediments which had a “rocktype” code of “6” or “96”.

The proportion of NAG material is estimated at 28% and the PAG at 72%. The classification scheme of the PAG material is considered conservative due to the level of uncertainty. Further study and sample testing is recommended to confirm and increase confidence in the waste rock categorization.

PRE-PRODUCTION SCHEDULE

The CIP ore is contained predominately within the oxidized zone, with minor tonnes occurring in the transitional zones. This fact, combined with the targeting of CIP ore for the first two years of plant feed, resulted in ore mining beginning in the first few months of production and no pre-production was necessary.

However, to enable maximization of the continuous feed duration of the CIP material at the beginning of the mine life, the commencement of feed was delayed until nine months after mining. The ore material mined during this period will be stockpiled on the pits respective ROM stockpiles where it will later be reclaimed and hauled to the Boroo plant.

One benefit of this approach will be in the ability for blending. This material can be reclaimed to achieve a desired result of either bringing grade and ounces forward or maintaining a smoothed feed grade.

LIFE OF MINE PLAN

INTRODUCTION

The LOM planning was completed using industry standard software, XPAC Open Pit Metals Solutions (OPMS).

The scheduling approach is based on the results of the development strategy and mine designs for the pit areas. The schedule considers the targets for crusher feed as well as equipment limitations. A driver of the schedule was the focus on maximising the continuous flow of CIP material during the initial years of mining to keep the processing plant operating at capacity with the aim of delaying transition to the BIOX® process. This approach was taken to allow time to convert the plant and minimize the amount of waste material movement and BIOX® ore that was stockpiled. Other factors were considered and described in more detail in the sections below.

SCHEDULE INPUTS

The inputs to the schedule were:

- physical scheduling constraints (such as lease boundary);
- geological Block Model from Mineral Resource estimate;
- mineable quantities material characterization/flagging;
- topography;
- designed pit solids (consisting of pit design stages and topography); and
- process plant feed requirements and construction/conversion ramp-up schedule.

ASSUMPTIONS, TARGETS, AND CONSTRAINTS

OPMS utilizes a series of assumptions, rules, and constraints to guide and control the mining schedule. A summary of key assumptions and constraints is provided below:

- The loading units assumed in the schedule were 100 t class excavators with assumptions derived from earlier studies of the following:
    - Availability of 85%;
    - Utilization of 80%;
    - Efficiency of 70%;
    - Productivity of 1,020 t/hr;
    - Resulting production capacity of approximately 350kt per month per machine; and
    - A reduced production rate was applied when the mining area became constrained to single lane access.

- The trucks selected were 50 t class models matched to the excavators.

- Capacity production feed rate to the process plant was 4,800 t/hr.

- A six month gradual ramp up of feed rate after conversion of the plant to BIOX® feed was applied. Table 16-11 shows the ramp-up applied to the BIOX® feed post-cessation of initial CIP feed.

TABLE 16-11 BIOX® CONVERSION FEED RATE RAMP-UP

Month after BIOX® conversion	**Month 1**	**Month 2**	**Month 3**	**Month 4**	**Month 5**	**Month 6**	**Month 7**
Percent of full production	15%	30%	45%	60%	75%	95%	100%
Daily throughput (t)	720	1,440	2,160	2,880	3,600	4,560	4,800
Monthly scheduled feed (t)	22,000	44,000	66,000	88,000	110,000	139,000	148,800

- The CIP ore mined and stockpiled during the BIOX® phase of processing will be processed after all BIOX® ore has been mined and fed. There will be a one month delay with no feed to account for the conversion of the plant back to CIP.
- A maximum sulphur grade of 1.30% has been applied to the BIOX® feed ore.

SCHEDULE OBJECTIVES

The schedule was constructed with several key goals in mind. The most important of these goals were to:

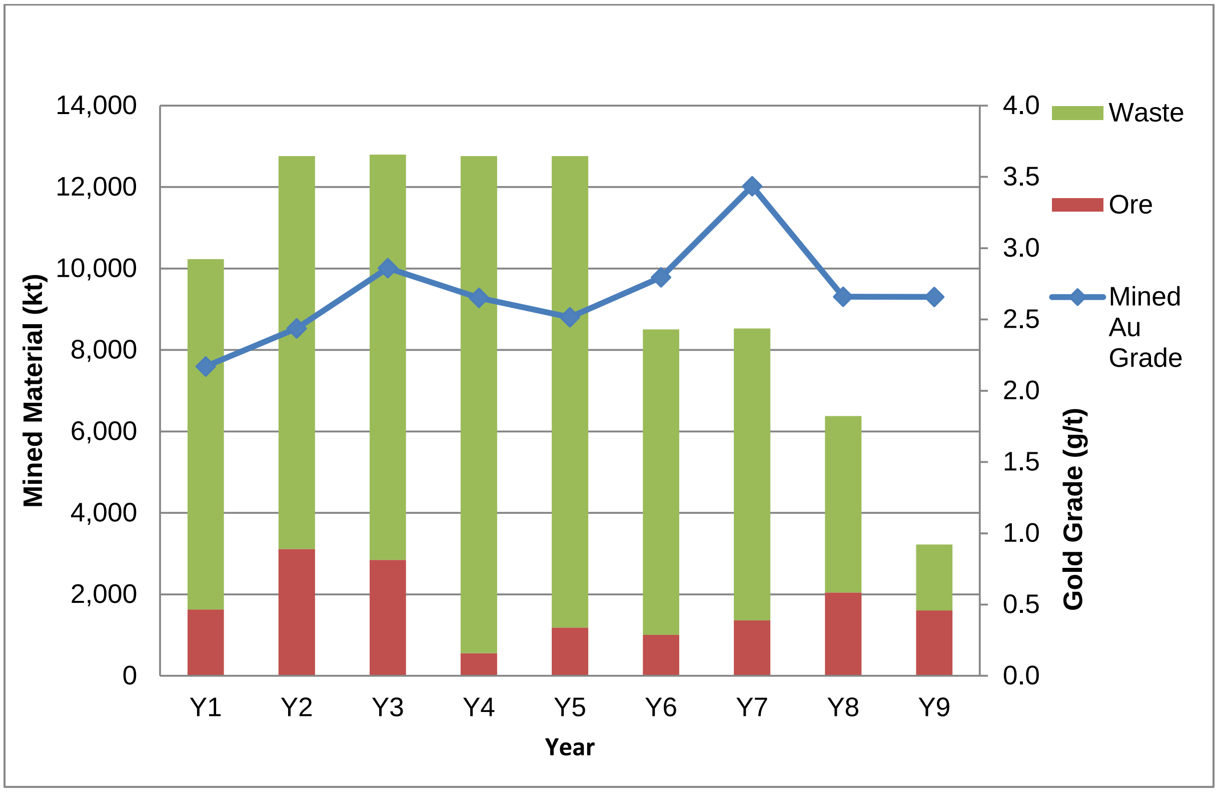
- maximize the amount of CIP ore feed in the initial years of the schedule;
- maximize the duration of CIP ore feed at the target capacity rate in the initial CIP feed period;
- minimize the amount of BIOX® ore stockpiled during the initial CIP feed period;
- maximize feed gold grade during all periods;
- minimize the waste haulage distance;
- prioritize the establishment of the diversion benches/berms; and
- ensure practical mining access is maintained throughout the schedule.

SCHEDULE RESULTS

TOTAL MATERIAL MOVEMENT

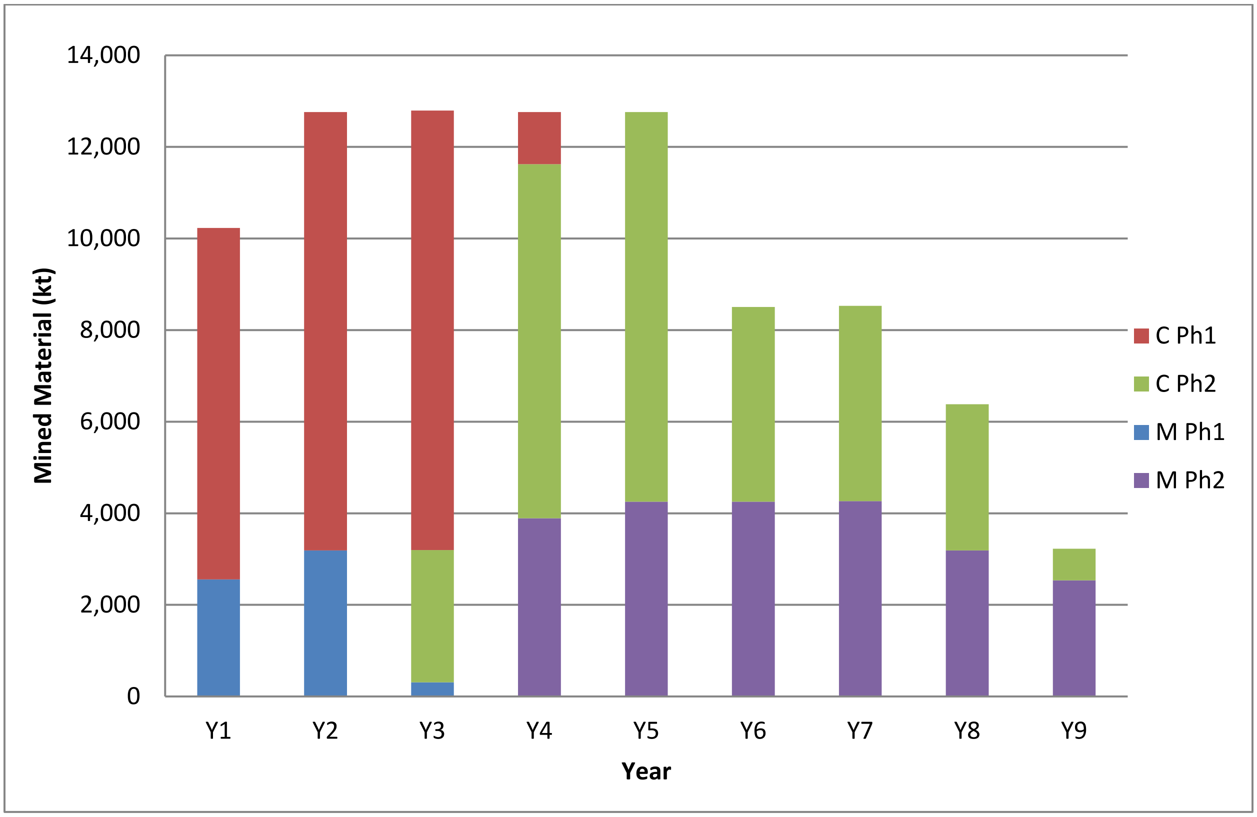
The total material movement is shown in Figure 16-5. During the approximately nine years of mining (total duration of 106 months), a total of 87.9 million tonnes of material is excavated from both pits, consisting of 15.4 million tonnes of ore and 72.6 million tonnes of waste. The mined gold grade is variable over this period and averages 2.7 g/t.

FIGURE 16-5 TOTAL MATERIAL MINED



The material mined by pit and phase is shown in Figure 16-6. Year 3 sees the completion of mining of Main Pit Phase 1 (M Ph1) and the relocation of the fleet into Central Pit Phase 2 (C Ph2); Year 4 sees completion of mining in Central Pit Phase 1 (C Ph1) with C Ph2 continuing and Main Pit Phase 2 (M Ph2) beginning.

FIGURE 16-6 TOTAL MATERIAL MINED BY PIT AND PHASE



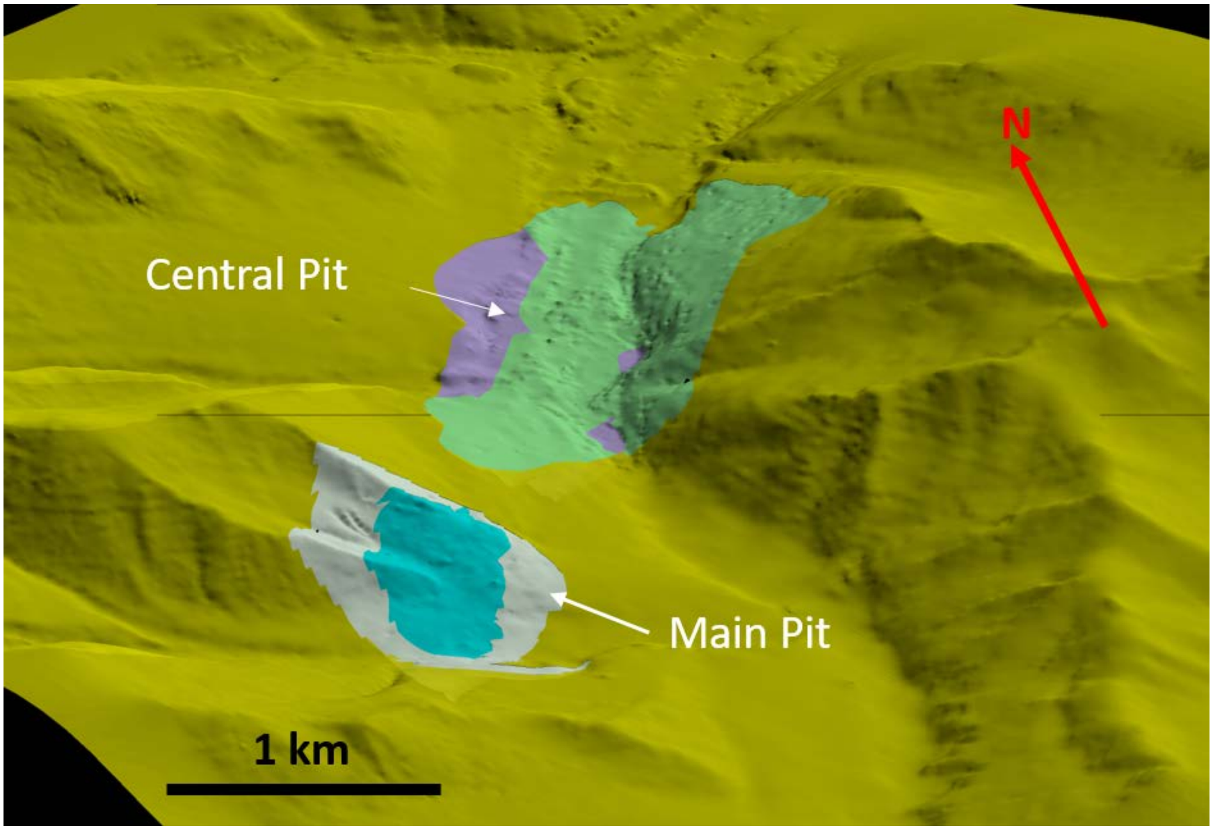
The graph shows that, mostly up to and including Year 5, the annual production from the two pits is split by roughly one third from the Main Pit and two thirds from the Central Pit. This split is approximately the same as the total tonnage proportions of the pits; 32% from Main Pit and 68% from Central Pit. It is also a reflection of the number and scheduled locations of the excavator fleets. During this period, the schedule is configured generally so that one excavator unit and truck fleet is situated in the Main Pit and two excavator units and truck fleets are in the Central Pit. The notable exception is in Year 3 where it was necessary to place all three excavation fleets in the Central Pit for the majority of that period. This was done to accelerate pre-stripping of the waste material in the second phase of the Central Pit, to allow the drawdown of ore stockpiles, and to maximize mined ore gold grade due to the grade of the Central Pit being slightly higher, discussed further in the following chapter.

These graphs also illustrate the variation in total scheduled capacity of the mining fleets. In the first year, productivity of the fleet was de-rated to 60% of capacity for the first six months of the Project. This was done to account for the delays normally associated with the beginning of a project. It was also an attempt to account for the expected lower productivity that would be encountered establishing productive working benches on the steep terrain at the Project, commonly referred to as “pioneering”. The features of the terrain are evident in Figure 16-7.

In the Central Pit from Year 6 onwards, the combination of working room being more restrictive and the stripping ratio improving reduced the amount of waste material required to be mined per tonne of ore. This resulted in only one excavator in the pit during this period. The productivities and capacity of each excavator unit in the Central Pit were unaltered.

From Year 8 onwards, the working room in both pits becomes even more restricted. To account for the expected penalty in productivity of the reduction in working room, the production capacity was reduced, and consequently the productivity, for the remaining periods where the working room was deemed to be potentially limiting.

FIGURE 16-7 ISOMETRIC VIEW OF PIT TERRAIN



WASTE HAULAGE

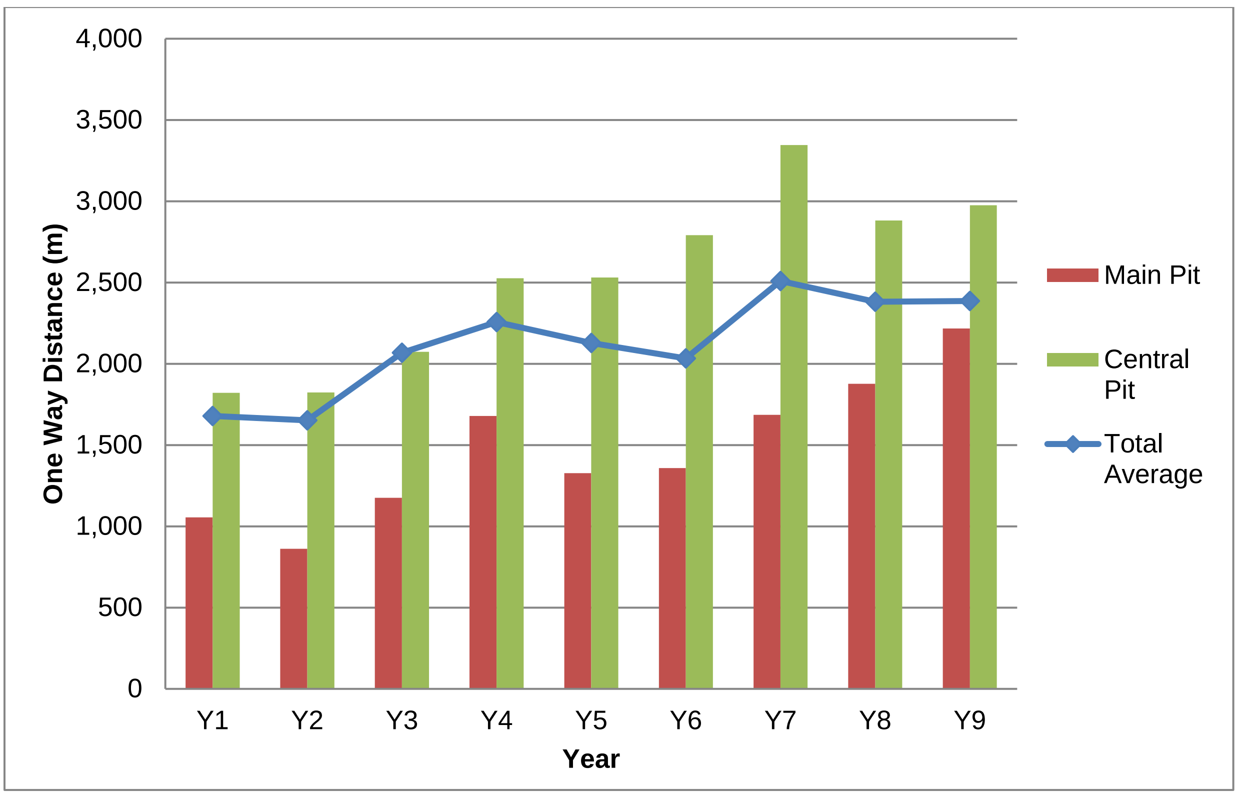
Waste haulage is often a major cost centre of a project. A key feature of the OPMS software is its ability to accurately model haulage using detailed road networks specific to the project. The schedule developed for Gatsuurt used such a road network to accurately measure the haulage requirements for input into the detailed costing exercise. The various pits and dumps are shown on the mine plan in Figure 16-1.

The average one way waste haulage distances for each pit, and the combined weighted average, is shown in Figure 16-8. The key points of the graph are that; the average annual haulage distance is generally increasing with time, due to a generally increasing depth of mining; and the Central Pit haulage is notably longer than the Main Pit as a result of several factors including the location of the dump/s in relation to the pit exits, the distance between pit centroid and dump centroids being further apart given the larger pit size and dump size requirement; and depth of mining/vertical haulage differential is greater in the Central Pit.

Haulage for the Main Pit sees a marked reduction in Year 5 which is attributable to the commencement of mining Phase 2 where the material being moved is from elevations near the surface. This trend is not as evident in the Central Pit owing to the dump locations; in particular, the Central Pit Phase 1 material is mostly taken to the Northwest Dump, however, this dump is filled to capacity with the completion of Phase 1 and material from Phase 2 is consequently taken to the North and East Dumps, a greater haulage distance from the pit source, see Figure 16-1.

The waste rock storage and handling is described in greater detail in the sections "Waste Rock Storage" and "Disposal of Potential Acid Generating (PAG) Material".

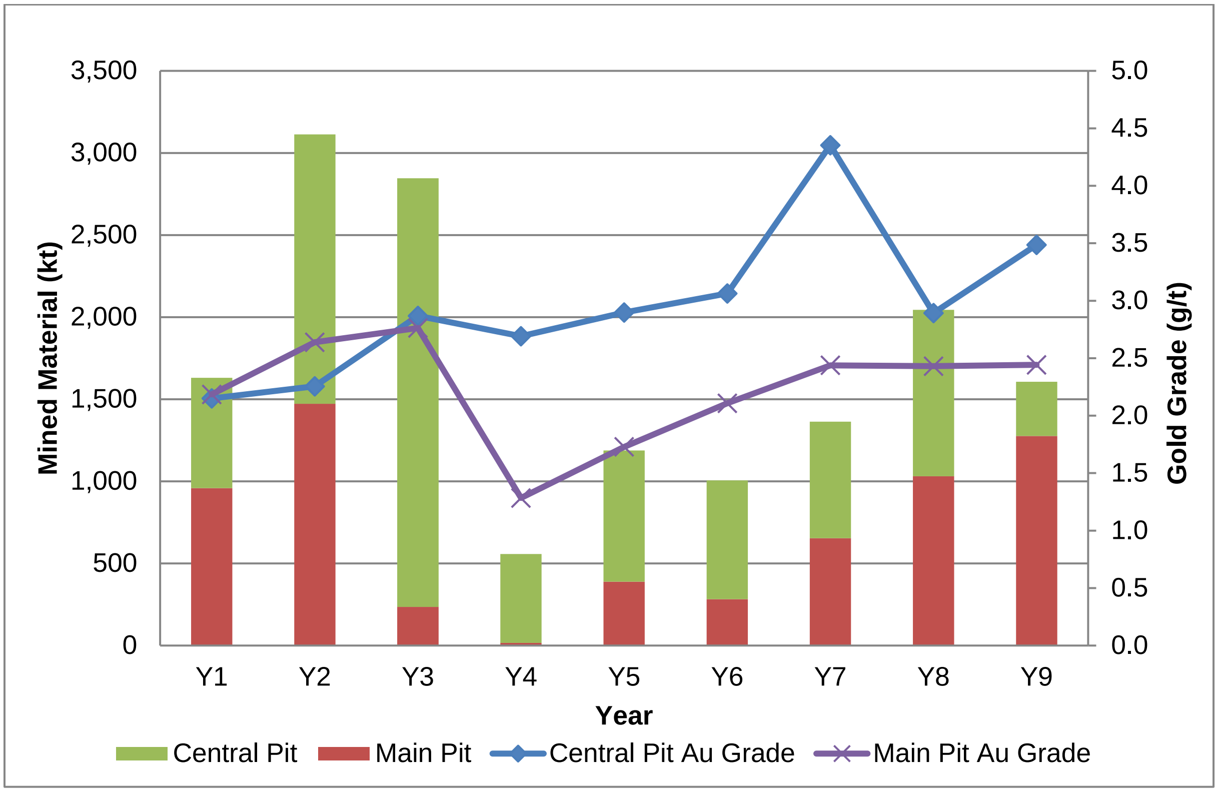
FIGURE 16-8 ONE WAY WASTE HAULAGE DISTANCES



ORE MINING

The ore tonnage and grade scheduled from each pit are shown in Figure 16-9. The data presented in this graph shows that in Year 3, the amount of ore mined from the Main Pit is minimal. It also shows that the amount of ore mined in Year 4 is minimal which is reflective of pre-stripping occurring in the second phases of both pits during this year. The following years see a generally increasing amount of ore tonnes being mined annually; this corresponds with the decreasing stripping ratio associated with the depth of the pits.

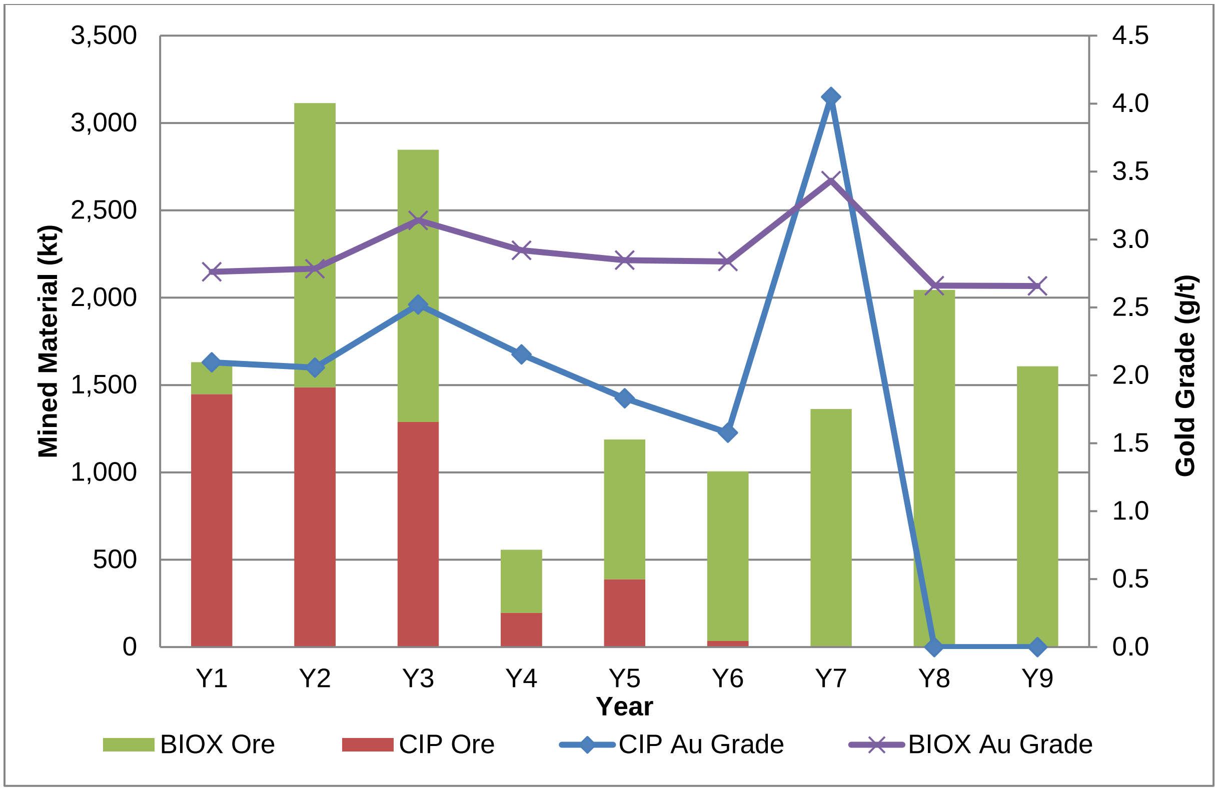
FIGURE 16-9 ORE TONNAGE AND GRADE BY PIT



The ore mining by process type is shown in Figure 16-10. During the early years the mining effort focusses on the excavation of CIP ore. The first three years of mining produce approximately 4.2 million tonnes of CIP ore, 87% of the total CIP ore mined over the life of the Project; coinciding with the completion of Main Pit Phase 1 and majority completion of Central Pit Phase 1. The remainder of the CIP ore is contained in the upper oxidized and transitional levels of the Phase 2 stages of each pit; because mining is completed in an essentially top down sequence, this residual 624,000 tonnes of CIP ore is mined in the few years following the completion of the first phases.

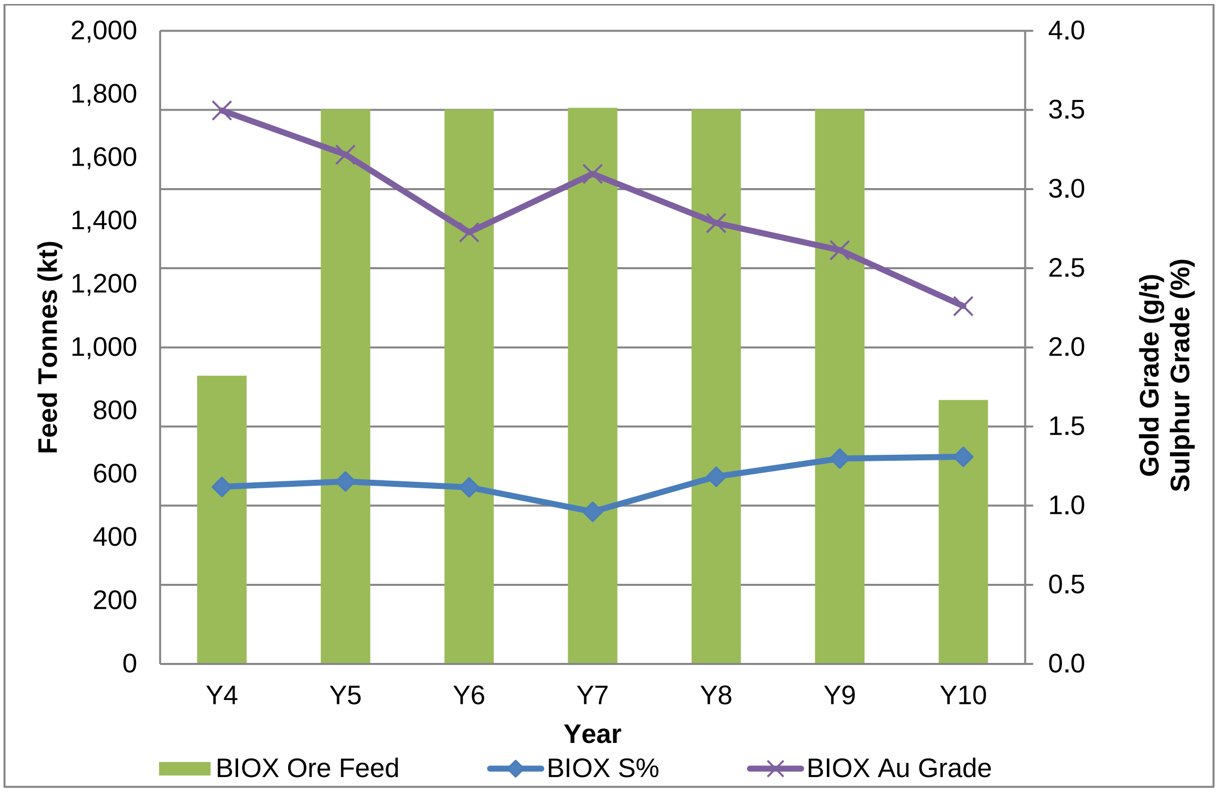
It also can be seen that the gold grade of the CIP ore is generally higher over the first three years. Both of these above points were the principal objectives in the shell selection for staging of the pit design.

FIGURE 16-10 TOTAL ORE MINED BY PROCESS TYPE



The schedule also attempted to bring gold forward to help improve the NPV result of the Project. Despite the average gold grade being slightly higher in Central Pit, because of the constraints on the sulphur feed grade, it was necessary to bring the mining of the slightly lower grade Main Pit material forward in the schedule. Delaying the mining of Main Pit resulted in a sulphur grade that exceeds the 1.3% maximum in the latter part of the plant feed schedule, which Centerra stressed would be problematic for BIOX processing. Bringing the Main Pit material forward in the schedule allowed the blending of the two different pits ore, maintaining a sulphur grade at an acceptable limit; see Figure 16-11. The QP believes that further refinement of the sulphur feed grade can be achieved at an operational level with appropriate grade control and stockpile management.

FIGURE 16-11 BIOX® FEED TONNES AND GRADE



STOCKPILE STRATEGY AND STOCKPILE BALANCE

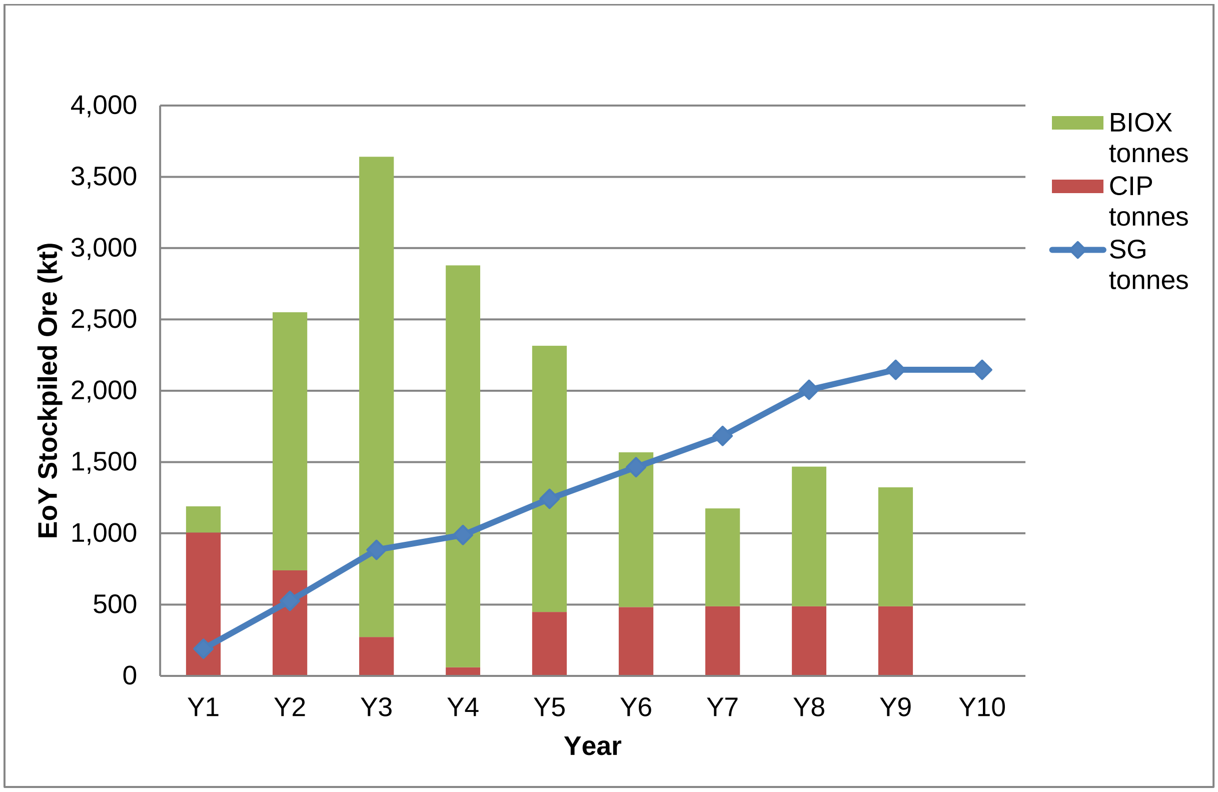
The process plant at Boroo is located approximately 52 km from Gatsuurt. Transportation of the ore material to the plant would be cost prohibitive for the primary production mining fleet. As such, there is a separate haulage fleet specifically for this long haul; with ore mined from the pit dumped at ROM stockpiles near the pit exit, which is then re-handled into the ore haulage fleet for transportation to the processing facility.

Stockpiling was necessary to ensure a consistent feed quantity to the process plant and provides an opportunity to selectively reclaim ore to either maximize the feed grade or provide a more consistent grade. This ability is further enhanced by the use of graded stockpiles, or “bins” where material within a certain range of properties is separately stored from material with a different range of properties.

The schedule utilized four stockpiles for each pit in an attempt to produce a feed of the highest gold grade possible. The stockpiles nominated and used were high grade and low grade stockpiles for CIP material and high grade and low grade stockpiles for BIOX® material. The low grade CIP material was defined as CIP flagged ore with an upper grade cap of 1.7 g/t Au; CIP material with higher grade is placed onto the CIP High Grade Stockpile. The low grade BIOX® material was defined as BIOX® flagged ore with an upper grade cap of 2.5 g/t Au; material with higher grade placed onto the BIOX® High Grade Stockpile. Plant feed was prioritized on feeding higher gold grade with limits on the sulphur grade as discussed in the following section.

The initial CIP processing began in month 10 and continued until month 39, with BIOX® processing beginning in month 40. Ore mining occurred in every month of the schedule. Since the process plant is only able to process one material type at a time, the CIP ore mined in the first 9 months and the BIOX® ore mined from month 1 up to month 40, was stockpiled. The result was a constantly changing stockpile inventory that grew in size until around the time BIOX® processing commenced. Furthermore, the sub-grade material was also stored on stockpile for potential future processing. The stockpile end of year (EoY) totals are shown in Figure 16-12.

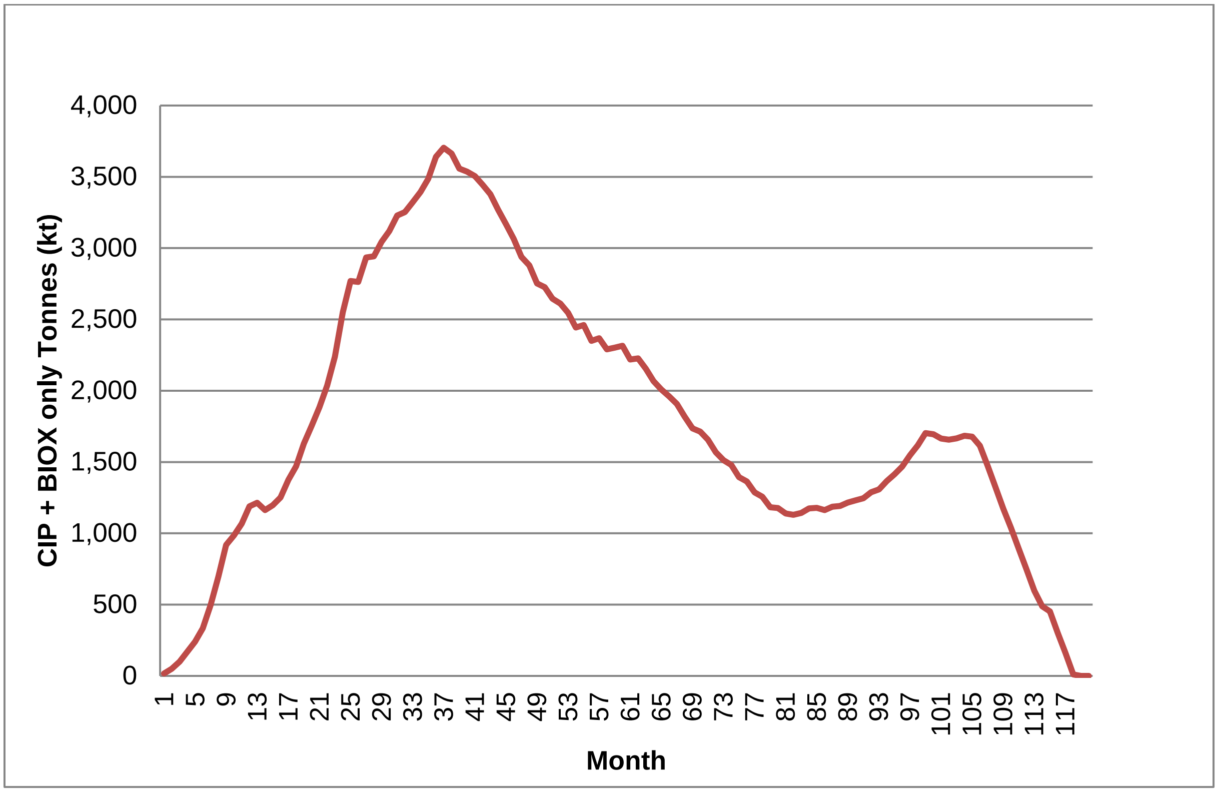
FIGURE 16-12 END OF YEAR STOCKPILE TOTALS



The stockpile ore inventories on a monthly basis, shown in Figure 16-13, yield a maximum value of 3,704 kt of CIP plus BIOX® ore at the end of month 37. The total inventory generally decreases after this period until approximately months 82 through 100 (approximately Year 7 and Year 8) where there is a slight increase in the stockpiled ore inventories. This is a function of the productivity as Year 7 still has full productivity of both excavators, and the stripping ratio of the mined areas as Year 8 has a slightly reduced productivity but is in the lower parts of the pits that possess favourable stripping ratios.

Stockpile inventory, including the addition of the sub-grade material, never exceeds the design ROM capacity for either pits.

FIGURE 16-13 END OF MONTH CIP AND BIOX® STOCKPILE INVENTORY

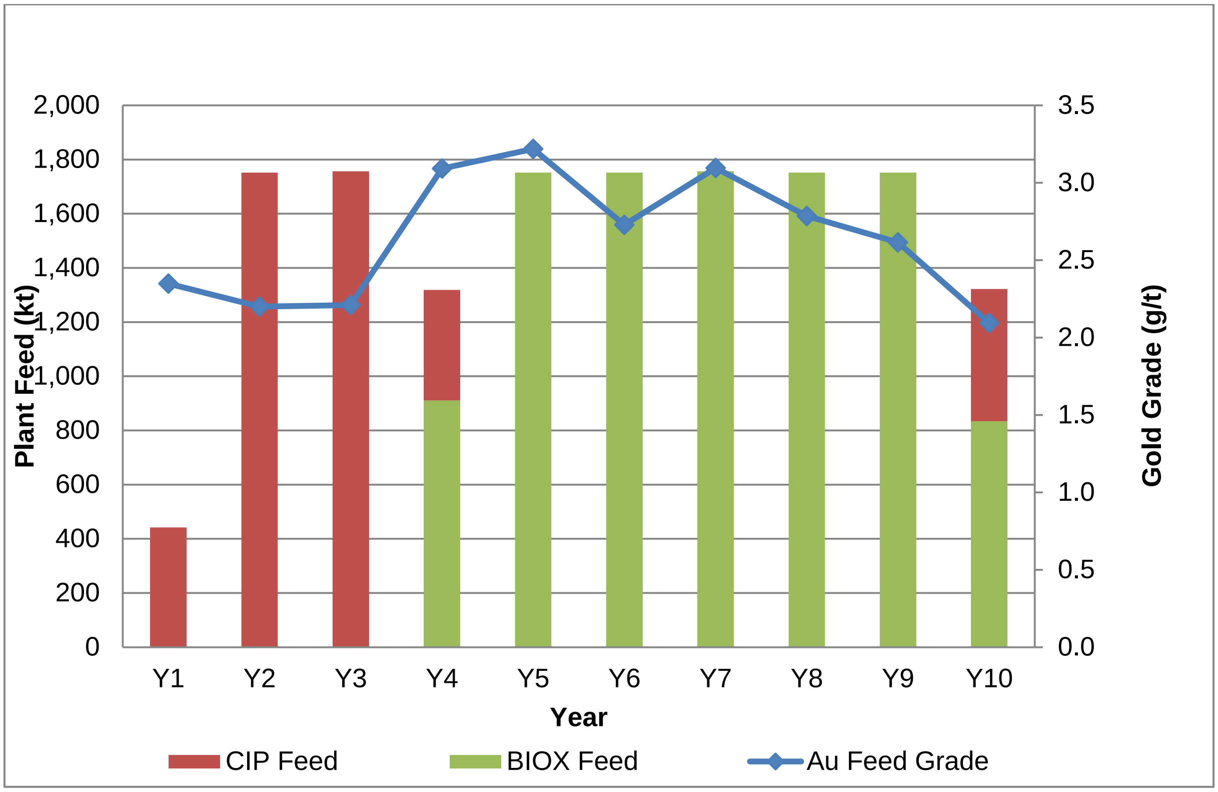


PROCESS PLANT FEED

One objective of the feed schedule was to start processing of the CIP ore as early as possible, with another being to maximize the duration of continuous CIP feed. Numerous options were run for the CIP feed start date to obtain an outcome that balanced the processing commencement with the feed duration. The result was that process plant start-up was delayed for nine months from mining commencement with CIP ore feed beginning in month 10. This enabled full plant feed rates from month 10 until month 39 after which the plant would be converted to a BIOX® plant. The conversion of the plant from CIP to BIOX® is scheduled to take place in the beginning of month 40 and includes a six month ramp-up period; full production beginning in month 46. The ramp up period and its productivities was shown previously in Table 16-11.

Figure 16-14 shows the tonnes fed to the process plant and the delivered gold grade. The delayed start to processing the CIP ore is evident in Year 1 total plant feed. Similarly the ramp up period of BIOX® processing after initial completion of the CIP is evident by the reduced total tonnes fed in Year 4.

FIGURE 16-14 PROCESS FEED TONNES



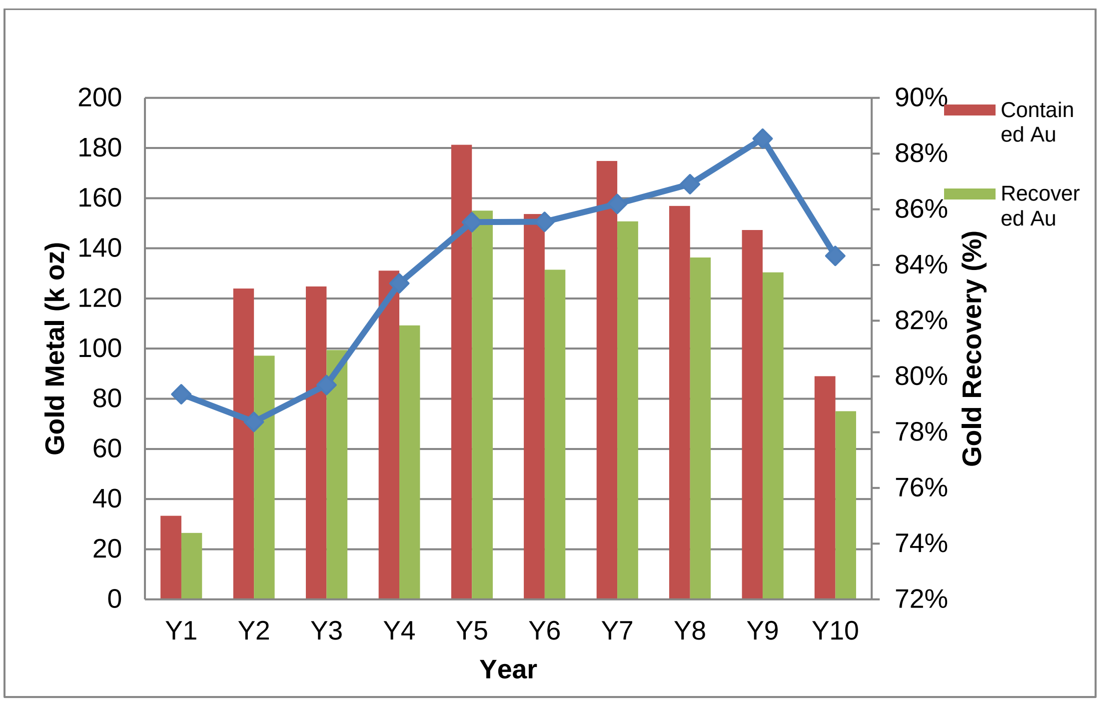
Following completion of processing sulphide material, the plant reverts back to CIP processing after one month closure to perform the conversion. The CIP material scheduled to be mined after the cessation of the initial CIP feed program, is stockpiled for this purpose. The final year of processing, Year 10, shows the remaining tonnes of CIP material. It should be noted that this year sees completion of feed of all material stockpiled, with approximately only 10 months of full production feed occurring over the year.

METAL PRODUCTION

The Project ores possessed variable recoveries for both process streams. As previously outlined, the process stream that was determined to be the most profitable was nominated on a block by block basis with due consideration to factors such as the gold grade and the recovery for each process stream. The recovery value was carried through the modelling and scheduling process to calculate the recovered metal for reporting. Based on the schedule, the contained and recovered metal, along with the weighted average recovery, delivered to the processing streams is shown in Figure 16-15.

The contained metal and recovery show a generally increasing trend, which coincides with the mining and feed characteristics of the schedule. These characteristics are; an initial CIP feed, which has a generally lower average recovery and grade; from Year 4 onwards, a transition to higher grade and higher recovery BIOX® feed; with mining depth, material with an increased recovery becomes more prevalent; in the final few years, the residual ore on stockpiles is fed, and owing to the prioritization of higher grade material in the earlier part of the schedule, this material has a notably lower gold grade, evident by the reduction of contained and delivered ounces; and the final year sees approximately 10 months of production and has the remnants of the CIP ore fed which as stated previously, has a generally lower grade and recovery.

FIGURE 16-15 PROCESS FEED GOLD METAL AND RECOVERY



SCHEDULE TABLES

Table 16-12 below provides a summary of key metrics of the produced schedule.

TABLE 16-12 SCHEDULE SUMMARY

DESCRIPTOR	MATERIAL	UNITS	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Totals / Ave.
Excavated													
Total Material	**All**	**kt**	10,229	12,759	12,794	12,759	12,759	8,506	8,530	6,380	3,227	0	87,944
Waste	Waste	kt	8,597	9,646	9,948	12,202	11,572	7,501	7,167	4,335	1,620	0	72,588
Ore	Ore	kt	1,631	3,113	2,846	558	1,188	1,005	1,363	2,044	1,607	0	15,356
Central Pit	**All**	**kt**	7,671	9,570	12,485	8,868	8,506	4,253	4,265	3,190	692	0	59,500
Central Pit Total Waste	Waste	kt	6,999	7,929	9,874	8,326	7,708	3,530	3,556	2,177	361	0	50,459
Central Pit	Ore	kt	672	1,641	2,611	542	799	723	709	1,013	331	0	9,040
Central Pit Au Grade	Ore	g/t	2.2	2.3	2.9	2.7	2.9	3.1	4.4	2.9	3.5	0.0	2.9
Main Pit	**All**	**kt**	2,557	3,190	309	3,892	4,253	4,253	4,265	3,190	2,535	0	28,444
Main Pit Total Waste	Waste	kt	1,598	1,717	74	3,876	3,864	3,971	3,611	2,159	1,259	0	22,129
Main Pit	Ore	kt	959	1,473	235	16	389	283	654	1,031	1,276	0	6,316
Main Pit Au Grade	Ore	g/t	2.2	2.6	2.8	1.3	1.7	2.1	2.4	2.4	2.4	0.0	2.4
Ore Mined													
Central Pit CIP	**CIP**	**kt**	**647**	**1,270**	**1,289**	**180**	**106**	**8**	**5**	**0**	**0**	**0**	**3,505**
Central Pit CIP Au Grade	CIP	g/t	2.1	2.1	2.5	2.2	2.5	2.1	4.0	0.0	0.0	0.0	2.3
Main Pit CIP	**CIP**	**kt**	**800**	**217**	**0**	**16**	**282**	**26**	**0**	**0**	**0**	**0**	**1,342**
Main Pit CIP Au Grade	CIP	g/t	2.1	2.0	0.0	1.3	1.6	1.4	0.0	0.0	0.0	0.0	1.9
Central Pit BIOX®	**BIOX**	**kt**	**25**	**371**	**1,322**	**362**	**692**	**715**	**704**	**1,013**	**331**	**0**	**5,536**
Central Pit BIOX®Au Grade	BIOX	g/t	2.8	2.9	3.2	2.9	3.0	3.1	4.4	2.9	3.5	0.0	3.2
Main Pit BIOX®	**BIOX**	**kt**	**158**	**1,255**	**235**	**0**	**107**	**256**	**654**	**1,031**	**1,276**	**0**	**4,974**
Main Pit BIOX®Au Grade	BIOX	g/t	2.8	2.8	2.8	0.0	2.2	2.2	2.4	2.4	2.4	0.0	2.5
Ore Feed to Plant													

DESCRIPTOR	MATERIAL	UNITS	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Totals / Ave.
CIP Feed	CIP	kt	442	1,752	1,757	408	0	0	0	0	0	488	4,847
CIP Feed Au Grade	CIP	g/t	2.3	2.2	2.2	2.2	0.0	0.0	0.0	0.0	0.0	1.8	2.2
BIOX® Feed	BIOX	kt	0	0	0	911	1,752	1,752	1,757	1,752	1,752	834	10,509
BIOX® Feed Au Grade	BIOX	g/t	0.0	0.0	0.0	3.5	3.2	2.7	3.1	2.8	2.6	2.3	2.9
Total Ore Feed to Plant	**Ore**	**kt**	**442**	**1,752**	**1,757**	**1,319**	**1,752**	**1,752**	**1,757**	**1,752**	**1,752**	**1,322**	**15,356**
Total Au Feed Grade	**Ore**	**g/t**	**2.3**	**2.2**	**2.2**	**3.1**	**3.2**	**2.7**	**3.1**	**2.8**	**2.6**	**2.1**	**2.7**
Contained Au	**Ore**	**koz**	**33**	**124**	**125**	**131**	**181**	**154**	**175**	**157**	**147**	**89**	**1,316**
Contained CIP Au Metal	Ore	koz	33	124	125	29	0	0	0	0	0	28	339
Contained BIOX® Au Metal	Ore	koz	0	0	0	102	181	154	175	157	147	61	977
Recovered Au	**Ore**	**koz**	**26**	**97**	**99**	**109**	**155**	**131**	**151**	**136**	**130**	**75**	**1,111**
Recovered CIP Au Metal	Ore	koz	26	97	99	22	0	0	0	0	0	21	267
Recovered BIOX® Au Metal	Ore	koz	0	0	0	87	155	131	151	136	130	54	845

Note: Numbers may not add due to rounding.

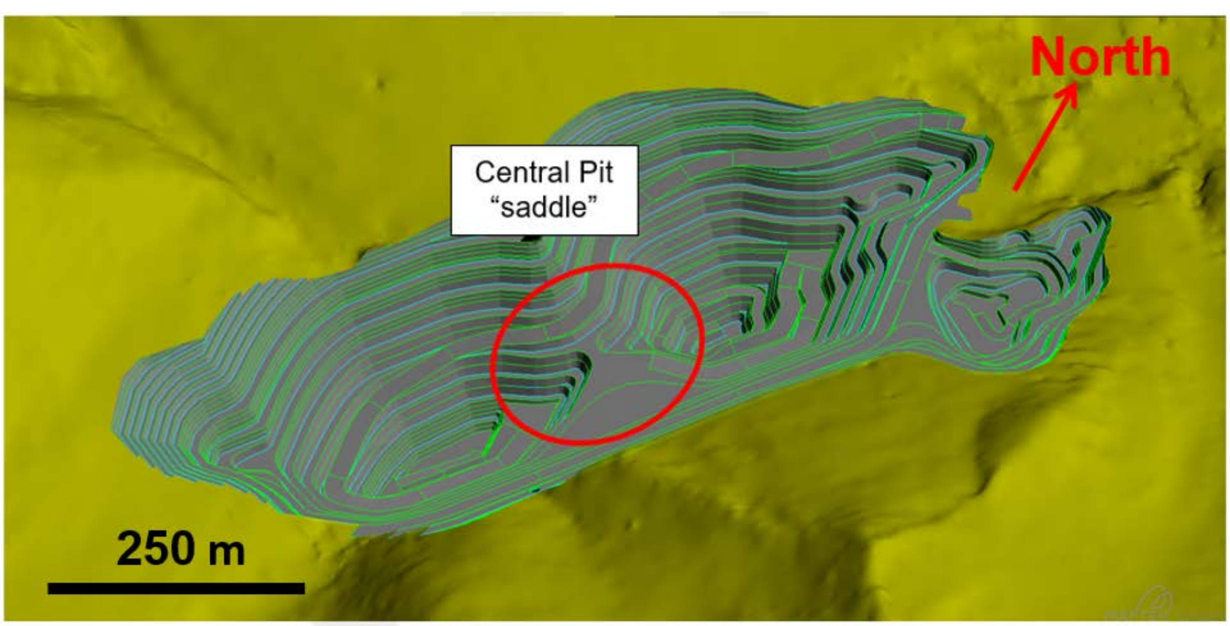
SCHEDULE CONCLUSIONS

The schedule produced is a practical and achievable schedule that satisfies the key goals defined. The advancement and progression of the pits and dump are constrained by practical rules within the OPMS software. The feed schedule, constraints, and limitations are similarly set by rules within OPMS.

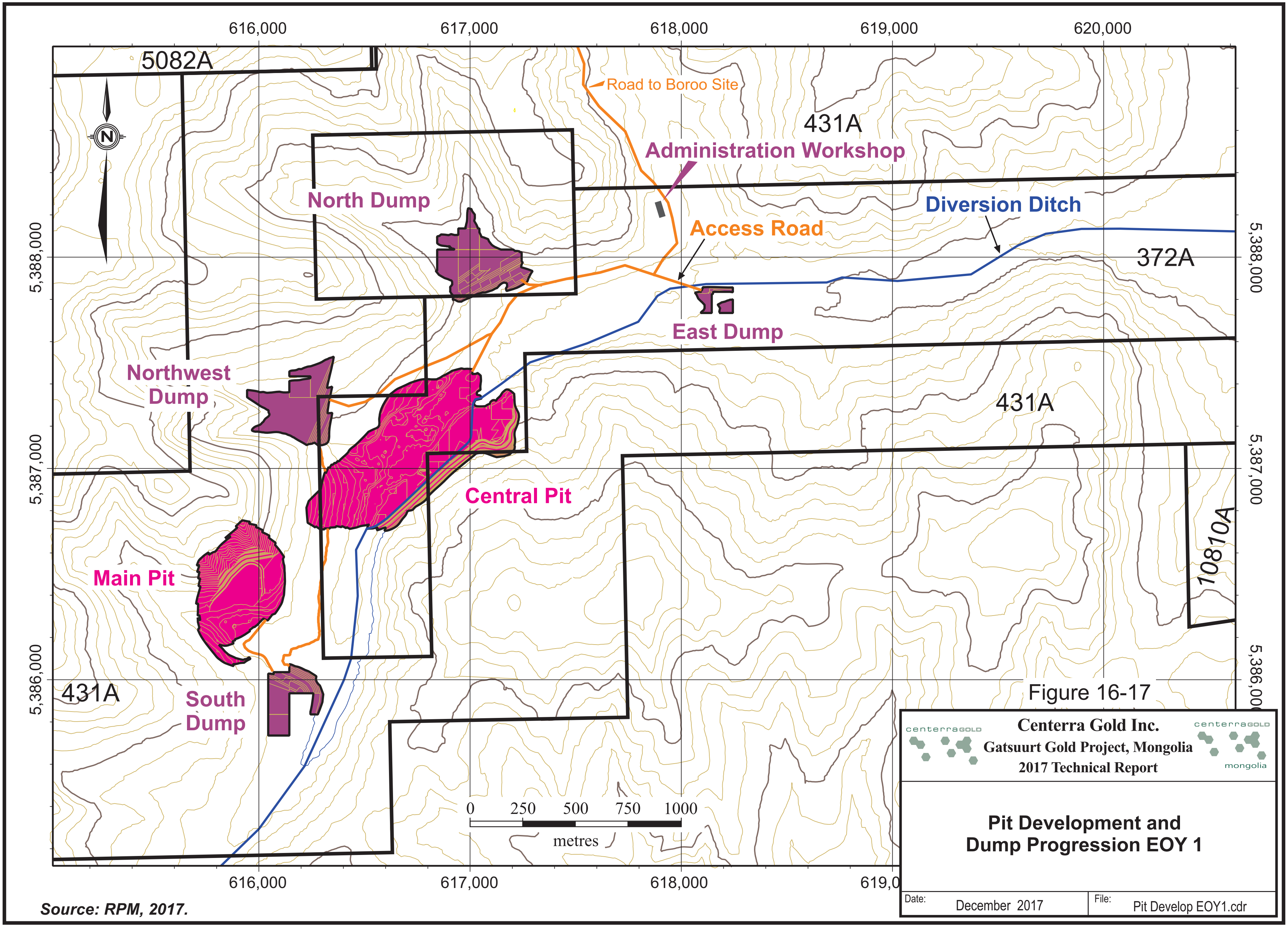
MINE PLAN DESIGNS

The development and progression of the pits and dumps is controlled by OPMS. This control is established and defined by setting rules for both the pits and dumps. Generally speaking, the pit progression is a top down approach, with new benches being established from the ramp and progressing outwards. The Central Pit possessed a “saddle” in the middle of the pit, as seen in Figure 16-16, which necessitated the north side of the saddle to be mined before the south to ensure that access was not compromised. This type of control was necessary and considered for all areas of the pit ensuring an appropriate mine plan was developed.

FIGURE 16-16   CENTRAL PIT DESIGN SADDLE



The annual pit development and dump progression is shown in Figure 16-17 to Figure 16-25.


616,000
617,000
618,000
619,000
620,000
5082A
Road to Boroo Site
431A
Administration Workshop
North Dump
Diversion Ditch
Access Road
372A
5,388,000
East Dump
Northwest Dump
431A
5,387,000
Central Pit
10810A
Main Pit
5,386,000
431A
South Dump
Figure 16-17
centerra GOLD
Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report
centerra GOLD mongolia
0 250 500 750 1000
metres
Pit Development and Dump Progression EOY 1
Date: December 2017
File: Pit Develop EOY1.cdr
Source: RPM, 2017.

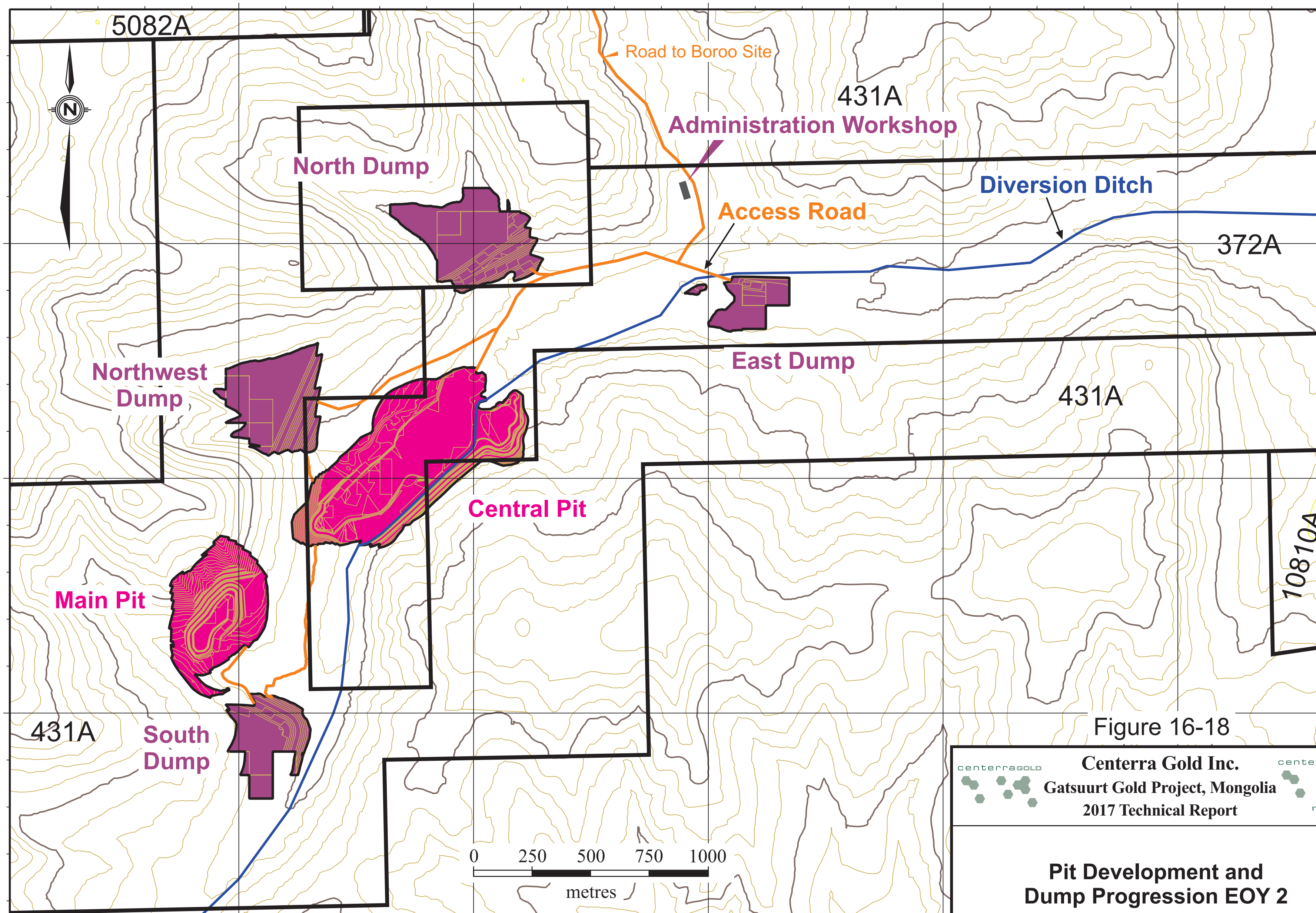


Figure 16-18

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Pit Development and Dump Progression EOY 2

Date: December 2017 | File: Pit Develop EOY2.cdr

Source: RPM, 2017.

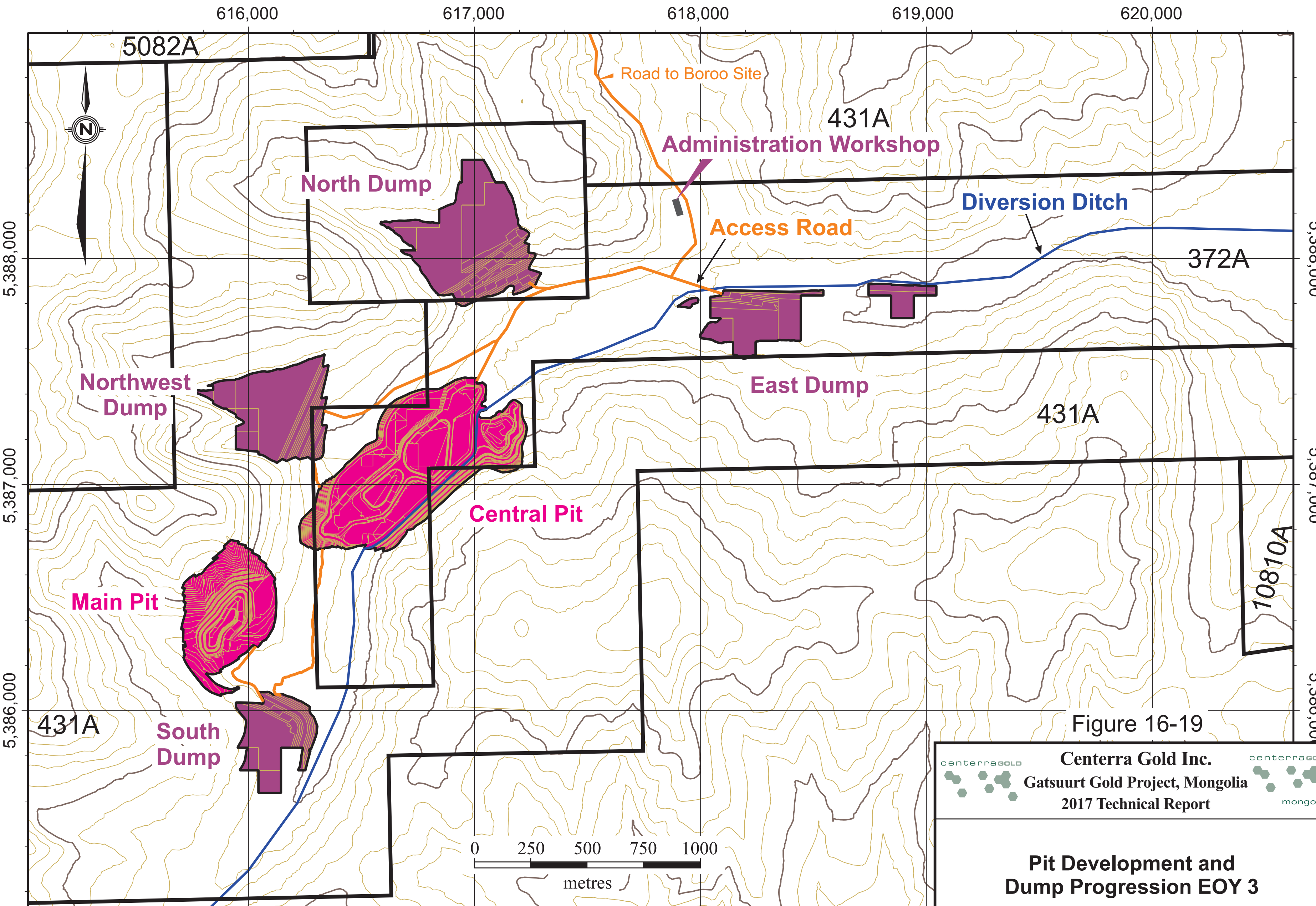


Figure 16-19

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Pit Development and Dump Progression EOY 3

Date: December 2017	File: Pit Develop EOY3.cdr

Source: RPM, 2017.

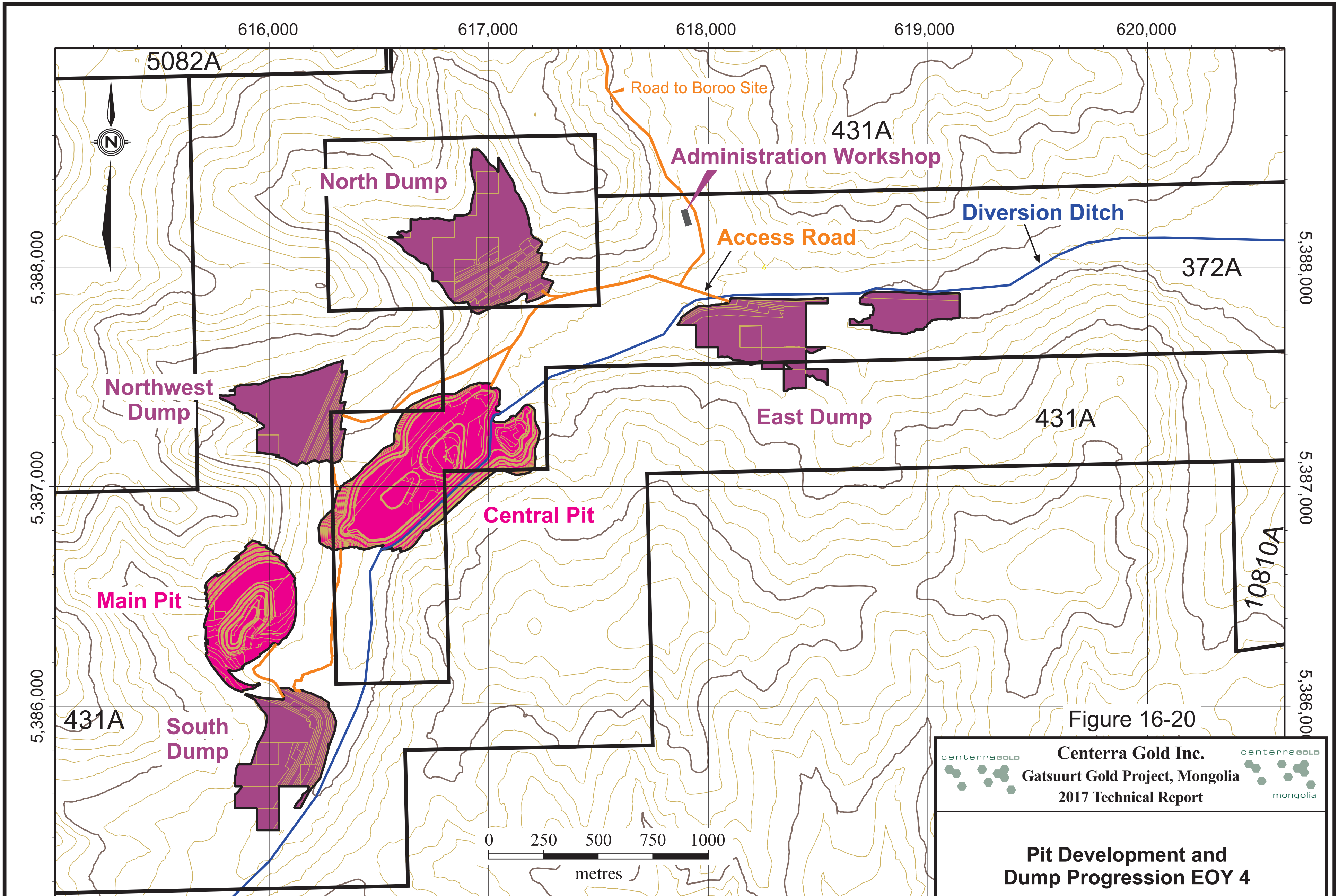


Figure 16-20

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Pit Development and
Dump Progression EOY 4

Date: December 2017 | File: Pit Develop EOY4.cdr

Source: RPM, 2017.

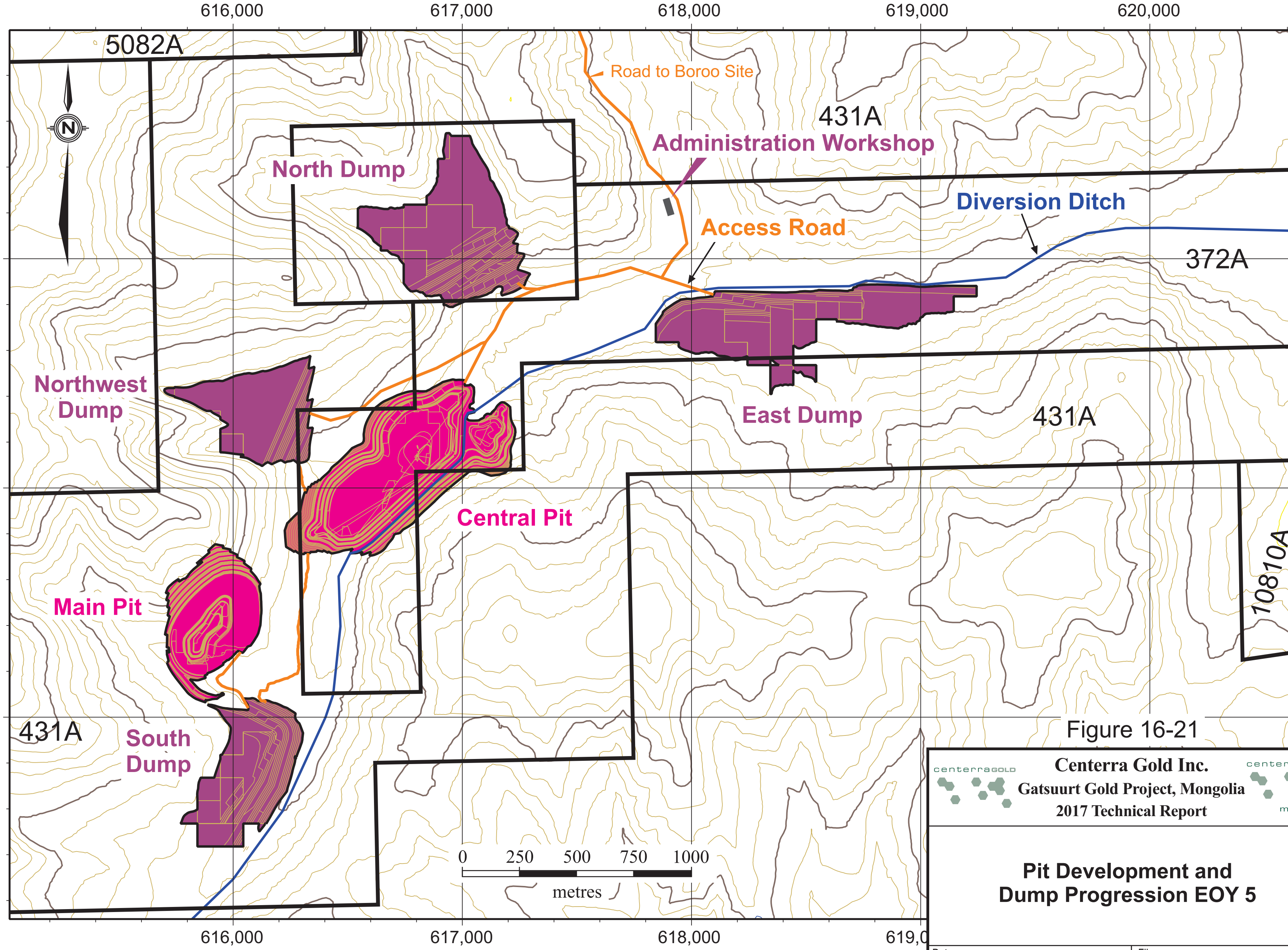


Figure 16-21

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Pit Development and Dump Progression EOY 5

Date: December 2017 | File: Pit Develop EOY5.cdr

Source: RPM, 2017.

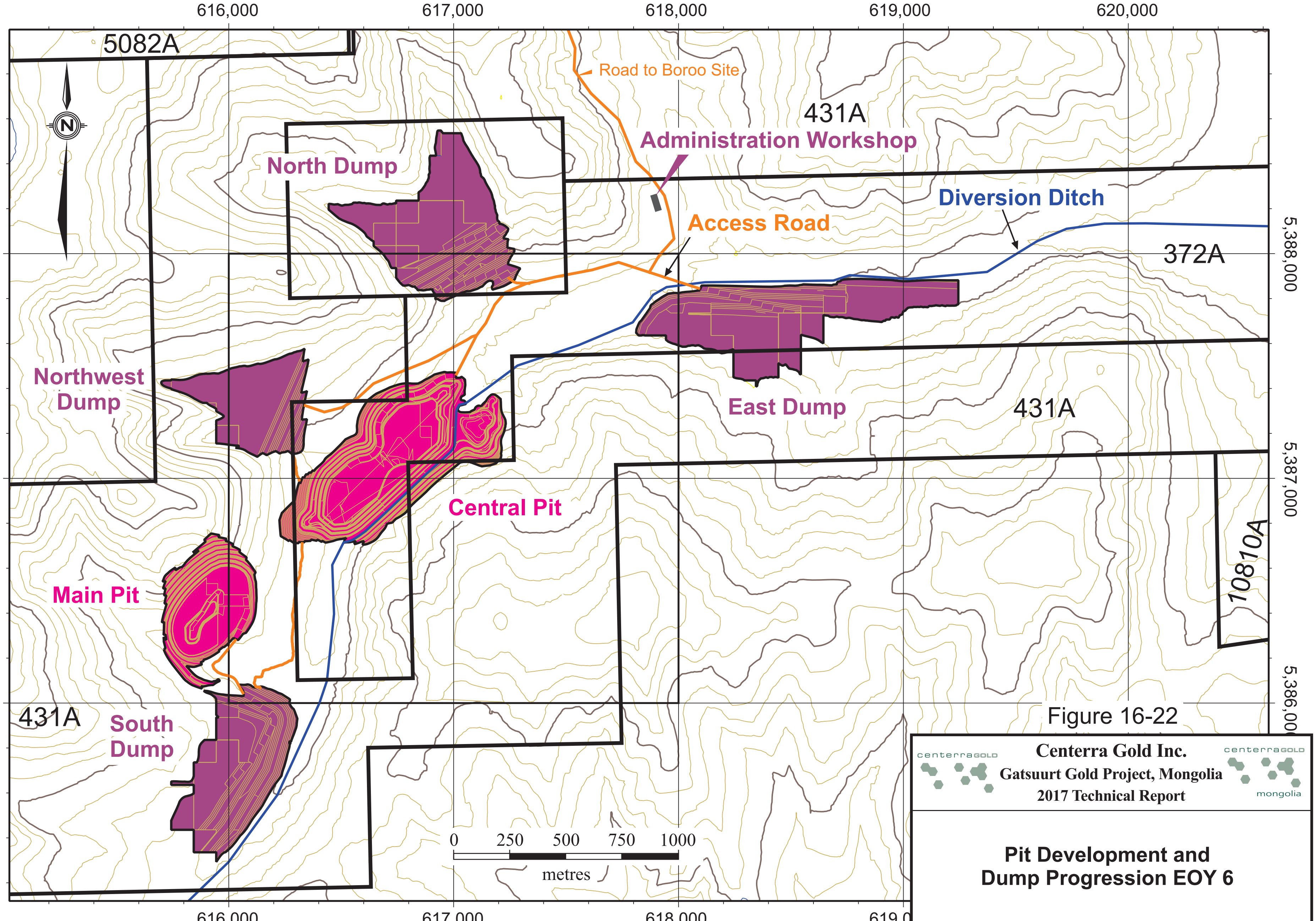


Figure 16-22

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Pit Development and Dump Progression EOY 6

Source: RPM, 2017.

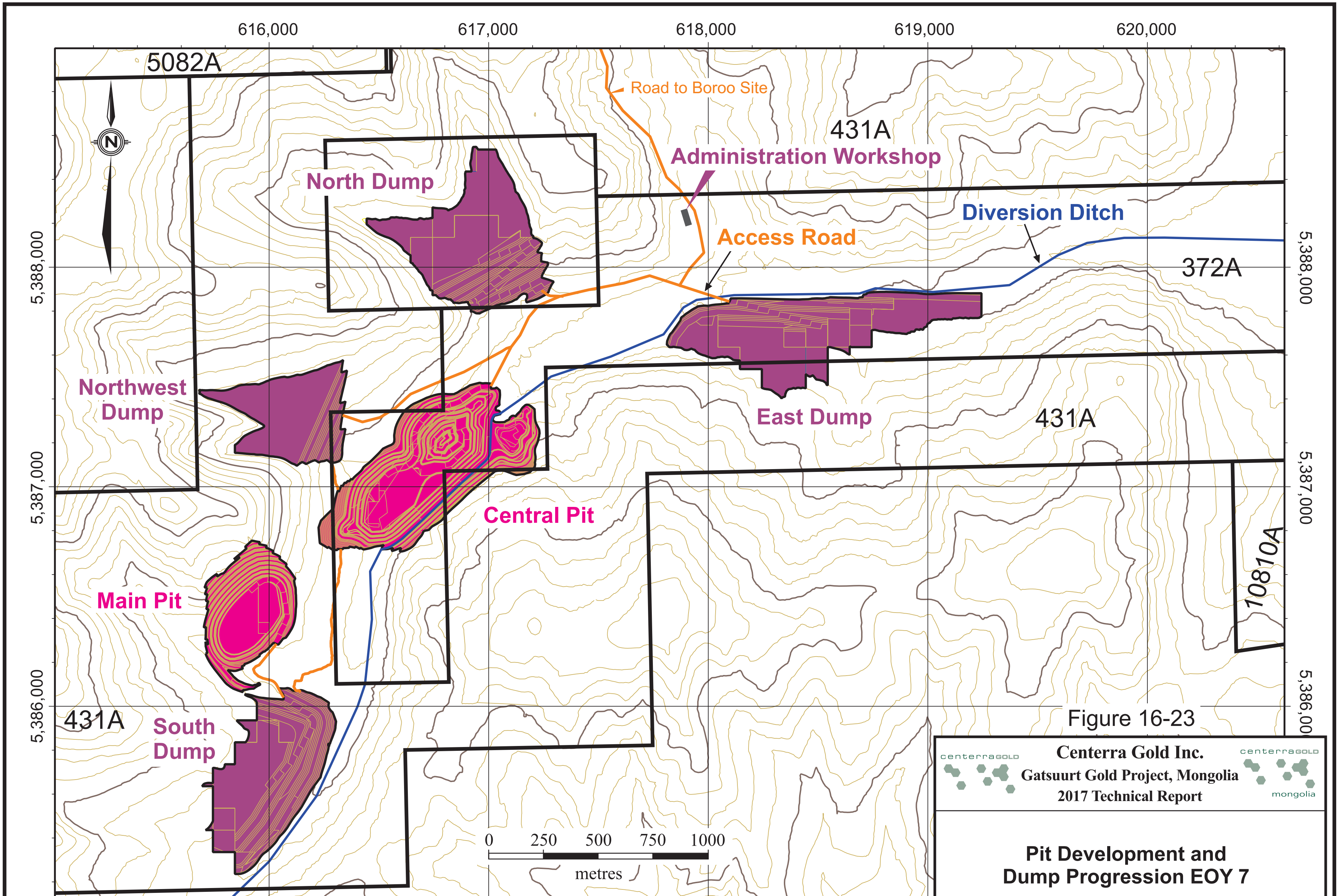


Figure 16-23

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Pit Development and Dump Progression EOY 7

Date: December 2017	File: Pit Develop EOY7.cdr

Source: RPM, 2017.

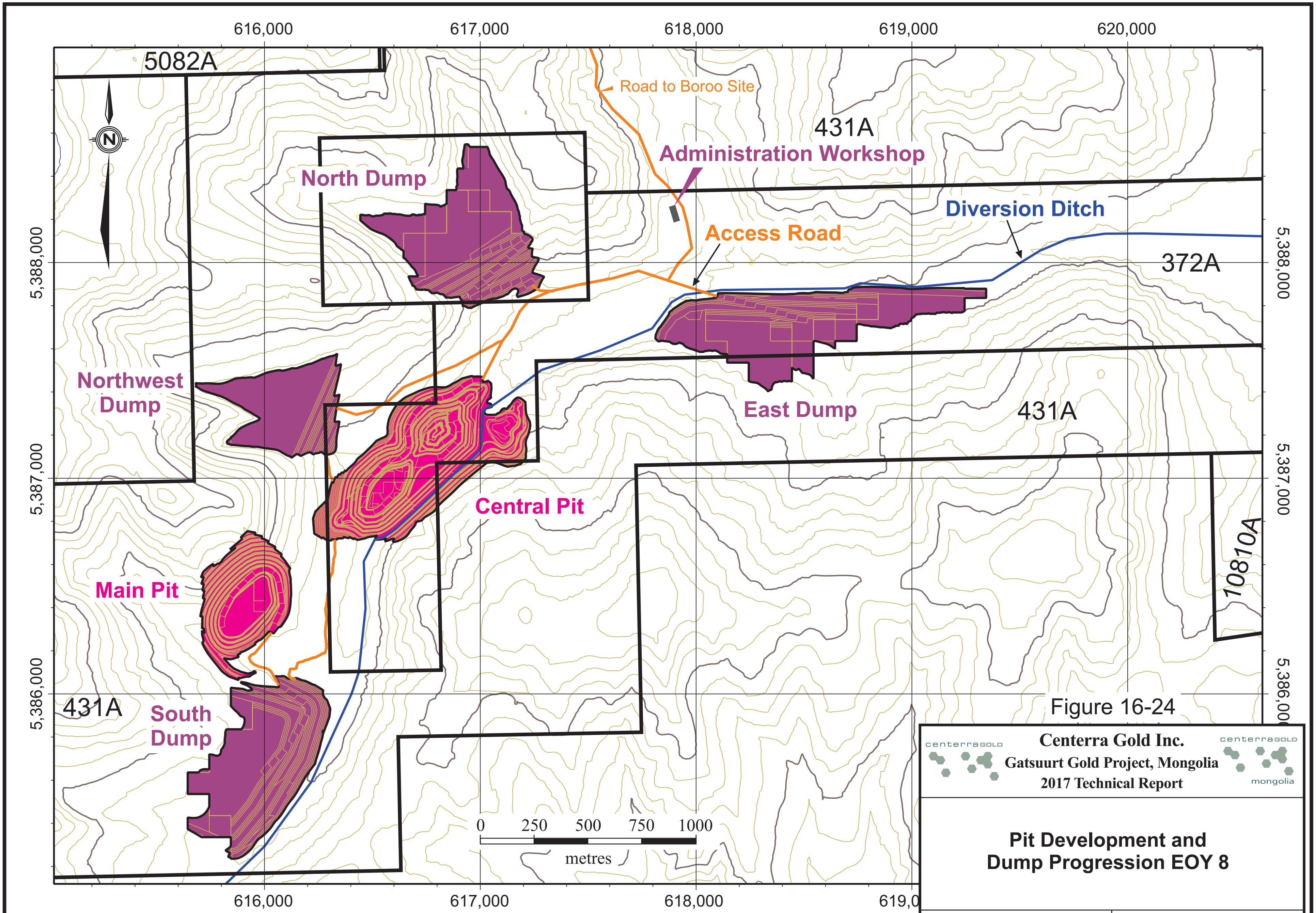


Figure 16-24

Pit Development and Dump Progression EOY 8

Source: RPM, 2017.

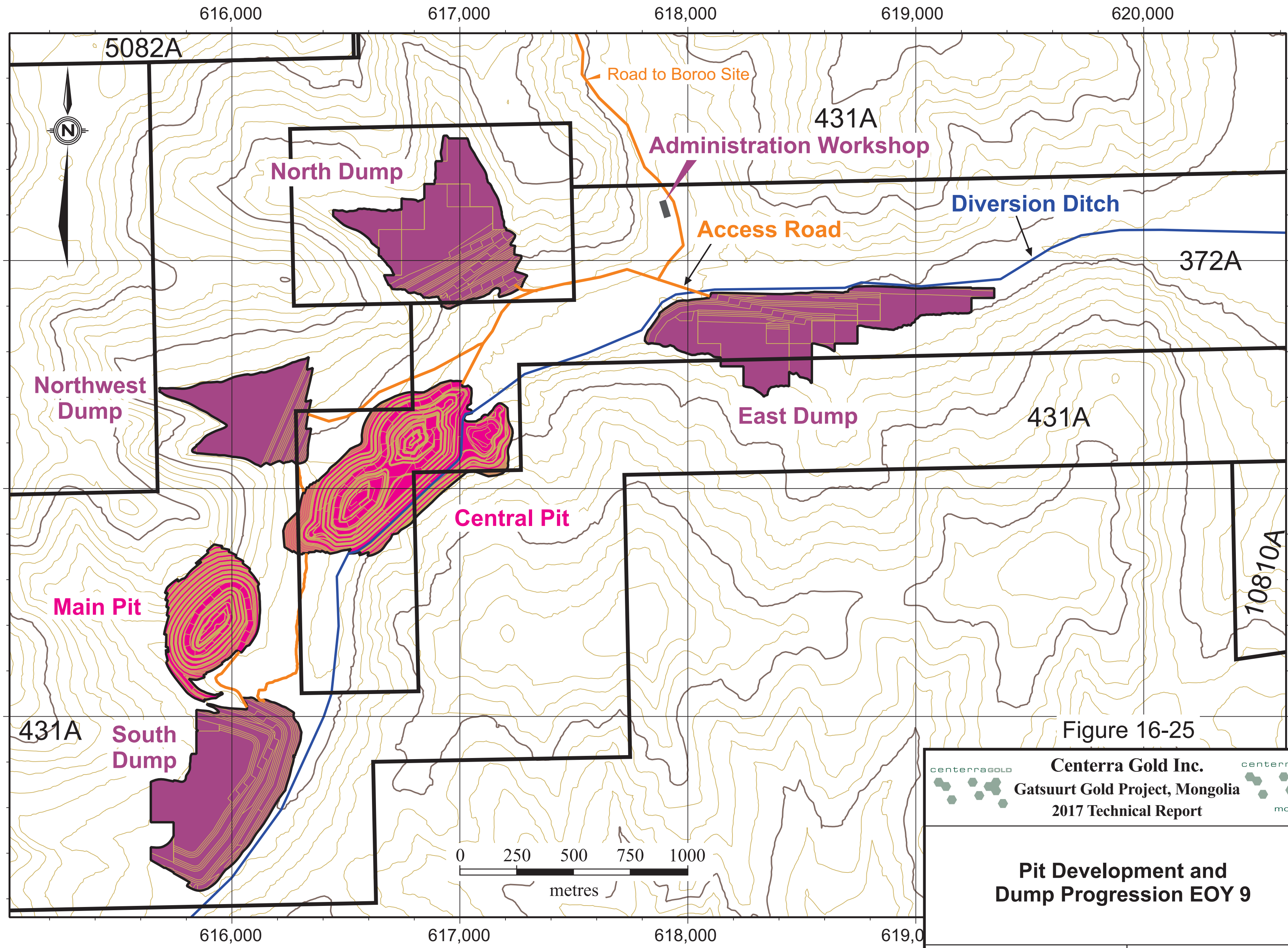


Figure 16-25

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report

Pit Development and Dump Progression EOY 9

Date: December 2017 File: Pit Develop EOY9.cdr

Source: RPM, 2017.

WASTE ROCK STORAGE

The location of the waste dumps were situated with consideration given to several key factors, primarily to be within the licence boundary and to provide the shortest haulage distance. There were four dump areas identified. The dumps and their capacities are summarized in Table 16-13 while Figure 16-1 shows the location of the dumps.

TABLE 16-13 DESIGNED WASTE DUMP CAPACITIES

Dump	LCM (x1000)	BCM @ 1.25 SF (x1000)
East Dump	25,069	20,055
North Dump	25,523	20,418
North-West Dump	3,908	3,126
South Dump	10,936	8,749
Total Capacity	**65,436**	**52,349**

Placement of waste material was assumed to have a 25% swell factor. This factor was based on measured swell factors from the Boroo operation.

A description of the design considerations particular to each dump is given below:

- East Dump – this dump is the only currently permitted dump. The haulage distance for this dump from both pits is considerable and this fact was key reason for investigating alternative locations for waste dumps. The capacity of this dump is substantial and is approximately the capacity required for all the Central Pit waste material. This capacity was a design consideration to mitigate the risk that any of the other dump designs are denied governmental approvals.

- North Dump – this dump is placed and designed to provide a substitute for the East Dump. The horizontal distance of and entrance to this dump from the Central Pit exit is considerably shorter than the East dump. Scheduling of this dump takes advantage of this feature and OPMS will preferentially dump at locations with the shortest haulage. The status of the land on which the North Dump is to be situated needs to be clarified.

- North-West Dump – this is a small dump that was designed to be utilized for Phase 1 of the Central Pit. The height of this dump is restricted by the lease boundary to the west.

- South Dump – this dump is designed for the sole use of the Main Pit. Its capacity, with a minor amount of excess, is matched to the waste volume of Main Pit. Its southwestern edge is constrained by the valley, to permit the natural flow of water past the dump, and also to the south by the Gatsuurt River.

All dumps are designed to be constructed using 10 m lifts with ROM material. No selective handling, sizing, or compaction other than that normally encountered with its construction will be necessary.

It is noted that the schedule does not completely fill all the dumps to design capacity. It is recommended that once approvals for the dump locations are finalized, the dump designs are modified such that their design capacity matches the scheduled movements to those dumps. This will also ensure that the environmental NAG enclosure requirements are satisfied.

RUN OF MINE ORE AND SUB-GRADE MATERIAL STORAGE

The Project utilizes two distinct ROM areas; one for each pit located near each pits exit, see Figure 16-1. The design capacity of each ROM area was developed to be as large as reasonably possible given the space constraints, ensuring there is sufficient capacity for the expected amount of material that will be stockpiled.

The approach of using stockpiles near each pit exit was used to minimize the haul truck cycle times for ore from the pits to the stockpile locations. It is expected that the savings derived from the reduction in haulage distance by the mining equipment will be significantly more than the increase in costs incurred by the extra haulage distance for the ore haulage contractor.

It is also assumed that the sub-grade material will be stockpiled on the ROM pads for possible reclamation after cessation of mining and the depletion of ore stocks.

Table 16-14 shows the capacity of each ROM area, noting that if needed, it is possible for the capacity of the Central ROM to be increased notably with an increase in height, achieved by adding lifts to the current design.

TABLE 16-14 ROM AREA CAPACITIES

ROM	LCM (x1000)	BCM @ 1.25 SF (x1000)	Approximate capacity (kt)
Central ROM	3,051	2,441	6,347
Main ROM	3,698	2,959	7,692
Total ROM	**6,749**	**5,400**	**14,039**

The maximum amount of stockpiled ore material in the current schedule is in the order of 2.1 million tonnes for the Central Pit ROM and 1.8 million tonnes for the Main Pit ROM; with a further maximum storage requirement of 1.3 million tonnes of Central Pit sub-grade ore and 0.9 million tonnes of Main Pit sub-grade ore. The current design ROM capacity far exceeds this requirement indicating that there is ample physical area available for utilizing numerous stockpiles based on grade and material type properties supporting blending of ore feed. The maximum amount of stockpiled ore material in the current schedule is in the order of 2.1 million tonnes for the Central Pit ROM and 1.8 million tonnes for the Main Pit ROM; with a further maximum storage requirement of 1.3 million tonnes of Central Pit sub-grade ore and 0.9 million tonnes of Main Pit sub-grade ore. The current design ROM capacity far exceeds this requirement indicating that there is ample physical area available for utilizing numerous stockpiles based on grade and material type properties supporting blending of ore feed.

DISPOSAL OF POTENTIAL ACID GENERATING MATERIAL

The potential for acid generation from the waste material mined has been identified. A waste classification scheme was developed to identify the PAG by lithological rocktype, as described in the section “Material Classification”.

As part of the waste management recommendations, it was indicated that the PAG waste be co-mingled with NAG waste in an “internal cell” of each waste dump. The placement of the waste would not require selective handling or mixing, and the waste dumps would then have a 5-10 m NAG cover placed at the base of pile, on top, and around all sides.

The dump designs incorporated a NAG cover over the top of the waste dumps, and the OPMS schedule was configured such that this cover was only allowed NAG material. Issues were experienced with the schedule where it was identified that there was insufficient NAG material as well as the timing of its excavation causing stoppages in the schedule. Due to the amount of NAG waste and its release in the schedule the dumps were designed to only have a NAG cover, with no NAG base. The current proportions allow a maximum depth of cover of each dump as outlined in Table 16-15.

TABLE 16-15 MAXIMUM DUMP NAG COVER

Dump	**Max** approximate cover (m)
Main Pit (South) Dump	6.5 - 7
Northwest Dump	6.5 - 7
North Dump	13
East Dump	6.5 - 7

Calculation of these maximum covers, assume the following:

- Cover is vertical depth of NAG material above PAG cell.
- PAG cell is placed directly on natural surface level (topsoil would be removed beforehand)
    - o The schedule is configured this way but uses a much thinner NAG cover with NAG and PAG co-mingled in the PAG cell.
    - o If a NAG base is required cover and base depths would be approximately half of those outlined above.
- Whole dump volumes have been used in the calculation. i.e. assumption that the whole dump will be filled.
    - o The schedule does not fill the whole of East Dump (<50% filled) or North Dump (~90% filled).

It is recommended that the NAG and PAG waste classification scheme be further investigated and refined as part of any detailed engineering phase, after which the dump designs and schedule would need to be revised to ensure appropriate dump sizing, NAG cover, and timing issues are resolved.

TOPSOIL REMOVAL AND RECLAMATION

Topsoil removal or reclamation have not been scheduled as discrete activities. It is assumed that this will be completed as part of the site works in establishing the Project.

The amount of topsoil that will need to be removed was estimated from the foot prints of the pits, dumps, and ROM areas. These totals are shown in Table 16-16. The volumetric calculations use the assumption that the topsoil depth averages 0.3 m. It does not include the area or topsoil from the footprint of the surface haul roads.

TABLE 16-16 MINING AREA FOOTPRINT AND TOPSOIL VOLUMES

Area	Footprint (m^2)	Volume (bcm)
Central Pit	411,298	123,389
Main Pit	185,801	55,740
East Dump	768,759	230,628
North Dump	595,083	178,525
Northwest Dump	173,198	51,959
South Dump	406,122	121,837
Central Pit ROM	156,042	46,813
Main Pit ROM	151,028	45,308
Total	**2,847,331**	**854,199**

MINING INFRASTRUCTURE (RAMPS, MATERIAL HANDLING, DEWATERING, MAINTENANCE, POWER, COMMUNICATIONS)

Site infrastructure is covered in detail in Section 18, Project Infrastructure.

Items particular to the mining operation include, site haul roads, establishing the pit dewatering bench, and in-pit dewatering activities.

The mining contractor will be responsible for the in-pit dewatering activities. This will include the purchase, maintenance, and operation of dewatering pumps and piping infrastructure. It is also anticipated that the contractor will construct and maintain any required staging or water storage ponds for water pumped from the pit areas. These ponds would also be used as fill points for watercarts for dust control purposes.

Water run-off from dumps and road will also need to be managed by the contractor.

The centerlines of the surface haul roads have been identified and designed to be established and used for the Project. This was done as part of the haulage design for the OPMS schedule. The site plan showing these roads was shown in Figure 16-1. The total length of these roads is approximately 3.1 km. There is also a haulage road required for the haulage of ore from the Main Pit ROM, to entrance of the dewatering diversion bench of Central Pit, an additional 1.6 km in length. Further roads are expected to be required for access to workshop and office areas, although these are not expected to accommodate large volumes of heavy vehicle traffic.

Communication infrastructure is expected to be minimal with the use of surface radio equipment. Mobile repeater towers are commonly used, easily sourced, and would be sufficient for an operation the size of the Project.

The in-pit diversion bench is an open drain which is lined designed to divert the water flow from the Gatsuurt River, capturing it upstream of the pit, directing it along the designed bench exiting downstream of the pit design. It is designed to accommodate the maximum expected flow experienced during the summer months of approximately 40 liters per second. The schedule is configured to concentrate initial mining in the south eastern wall of Central Pit Phase 1 so that the diversion bench is established as a priority. Until this diversion bench is completely established, there will be a need for localized and interim dewatering efforts so that mining operations are not detrimentally affected.

The Gatsuurt River is not intersected by the Main Pit, however, there is a tributary that will need to be diverted. This diversion is incorporated into the Main Pit Phase 2 design, using the initial 12 m total width design criteria as recommended by SEA. This tributary is not expected to experience the same water flows as the Gatsuurt River, presenting an opportunity to review and revise the diversion design for this area to a dimension more appropriate for the expected flow.

MINE EQUIPMENT

APPROACH

A mining contractor will be used to complete all mining related activities. As such, the equipment used will be dictated by the agreement between Centerra and their chosen contracting company.

For the purposes of the schedule and costing exercise, an equipment sizing and fleet estimation was conducted for the Project. The results of this formed the basis of the equipment parameters input into the OPMS schedule.

The approach followed in the selection of suitable open pit equipment sizing involved analyzing a number of factors and considerations such as:

- Production schedule – annual equipment capacity (based on operating hours and productivity) for ore and waste in relation to achieving the production targets;
- Existing equipment – the existing equipment already purchased by Centerra and equipment already owned by potential contractors;
- Working room analysis - the number of excavators that can fit in the available working room at key stages throughout the life of the operation;
- Selectivity requirements – consideration of loss and dilution from different size loading units to balance unit mining cost (higher selectivity results in higher costs) vs loss and dilution incurred;
- System redundancy– ensuring the system can cope with downtime in one excavator for short periods, given the remote nature of the site; and
- Flexibility – ability to mine multiple locations concurrently; due to the multi-pit and stage nature of the Project, the ability to mine in multiple locations at once is advantageous.

KEY INPUTS AND ASSUMPTIONS

The sources for the key inputs and assumptions to the equipment sizing and selection process were:

- Fleet available from contractors ; and
- An in-house Equipment Database and prior experience.

BREAKDOWN OF TIME

The key assumptions made to estimate equipment operating hours include:

- Continuous mining project operating two 12 hour shifts per day, 365 days per year;
- Seven days per year lost due to wet and other weather related issues;
- Mechanical availability of 85% for new major equipment[1];
- Major plant approximately:
  - 5,800 to 6,000 operating hours per year;
  - 4,800 to 5,000 effective hours per year (excluding truck presentation factor);
- Dozers operating on reduced roster to reflect intermittent nature of work;
- Watercart scheduled to be utilized seasonally, with lower requirements in winter.

Table 16-17 outlines the annual breakdown of time by equipment group.

TABLE 16-17 ANNUAL BREAKDOWN OF TIME BY EQUIPMENT GROUP

	Unit	Excavators	Trucks	FEL	Dozers	Graders	Drills	Watercart	Fuel Truck
Scheduled	hour	8,760	8,760	8,760	6,576	6,576	8,760	4,440	6,576
Available	hour	7,458	7,458	7,458	5,588	5,742	7,458	3,852	5,742
Operating	hour	5,941	5,853	5,853	4,291	4,416	5,799	2,977	4,416
Effective	hour	4,955	4,984	5,039	3,611	3,633	4,880	2,452	3,633
Availability	%	85%	85%	85%	85%	87%	85%	87%	87%
Utilization	%	80%	78%	78%	77%	77%	78%	77%	77%
Efficiency	%	83%	85%	86%	84%	82%	84%	82%	82%

EXCAVATOR PRODUCTIVITY

Key assumptions made in relation to equipment productivity for input into the OPMS schedule include:

- An average material density of 2.70 t/m3 assumed for ore and 2.68 t/m3 assumed for waste;
- An average moisture of 1% assumed for ore and waste;

[1] Chinese produced machines are assumed to be used which generally have a lower average availability when compared to Western manufactured machines.

- 100% of material blasted with a swell factor to the bucket of 1.40 assumed for all blasted material;

- Conservative bucket fill factors of 0.88 – 0.95 assumed to reflect flitch height and bucket size;

- 100 t excavators (6 m^3 bucket SAE 1:1) loading 50 t rear dump trucks:
    - 1,020 tonnes per operating hour (t/Op.h) in waste (excluding truck presentation);
    - 1,020 t/Op.h in ore material (excluding truck presentation);
    - 5 pass loading.

- Truck payload within +/-5% of manufacturer recommended payload.

DRILL AND BLAST

Drill and blast activities will be carried out by the selected contractor. It is also expected that the contractor will be responsible for the purchase and supply, storage, and associated permitting for the explosives used at the Project.

MAJOR EQUIPMENT

Based on the requirements of the production schedule and the likely machines available from potential contractors, a total of three 100 t class excavators were scheduled loading a truck fleet consisting of 50 t rear dump trucks.

SUPPORT EQUIPMENT SELECTION AND SIZING

Based on the major equipment selection, consideration of the site layout, and various activities, the following are the QP's recommendations for support equipment selection:

- 6 m^3 FEL on the ROM capable of loading 1.75 Mt/a to a fleet of 40 t rear dump trucks delivering ore from Gatsuurt to the processing plant at Boroo (a distance of 52 km);

- FEL also to provide backup for ROM loading, as a stemming loader, and various other support tasks;

- A significant length of permanent and temporary haul roads need to be established and maintained in a hard rock environment:
    - Cat 14H graders (for developing permanent roads and maintenance),
    - 30 t water carts, for dust suppression; and
    - 230 to 300 kW tracked dozers in pit supporting mining operations and temporary road development and ex-pit on dumps, rehabilitation, and ex-pit road development.

FLEET ESTIMATION

A Fleet Estimation exercise was completed to compare between a first principal estimate and the various contractor estimates of the number of machines required to complete the schedule.

The approach adopted for Fleet Estimation included:

- Truck cycle times estimated in OPMS software using an extensive array of three dimensional source to destination haul profiles over the mine life;
- An iterative modelling process was utilized to calculate an optimal truck fleet size and correct excavator productivities for varying truck presentations on a quarterly basis;
- Excavator and truck fleet operating hours and truck fleet numbers modelled on an annual basis;
- Support fleet operating hours and numbers determined from truck and loader fleet hours, numbers and sizes, the number of mining locations, active road lengths, seasonal weather events and other support task requirements.

WASTE AND ORE HAULAGE FROM PITS

The OPMS schedule calculated detailed waste and ore haulage from the pits to the waste dumps and ROM pads. The average one direction waste haulage was outlined in Figure 16-9. OPMS calculated the haulage travel time for each step (parcel of material) of the schedule, calculating the total travel hours required for the equipment to move material from its source to it allocated destination. These totals were imported into the Fleet Estimation software, and combined with estimated average loading times, dumping times, and truck queue times, the total number of truck hours to move the schedule tonnes was determined.

ORE HAULAGE FROM GATSUURT TO BOROO

The travel distance from Gatsuurt Mine site to the processing plant in Boroo has a one way distance approximately 52 km. HaulSIM software was used to simulate this haulage system. The haul road route used in the model was provided by Allnorth.

The cycle time (including loading and dumping time) for ore delivery from Gatsuurt to Boroo and return, was estimated to be approximately 2.4 hrs per trip.

FLEET NUMBERS

The Fleet estimation software takes the above haulage times and material movements and using the assumptions for productivity, annual hours outlined in this section, estimates the total number of machines required for the Project on an annual basis.

Table 16-18 summarizes the major and support equipment requirements for the Open Pit. This summary table is provided only as reference for the indication of likely equipment classes and numbers that would be required to execute the schedule.

TABLE 16-18 ESTIMATED FLEET REQUIREMENTS

Plant Group	Size	Unit	Max	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9[2]	Y10
Excavator	100 t	No.	3	3	3	3	3	3	2	2	2	2	0
Ore Haul Truck	36 t	No.	22	22[1]	22	22	22	22	22	22	22	22	22[3]
Rear Dump Truck	50 t	No.	22	15	17	20	22	22	14	16	12	12	0
Drill	3-5 in	No.	7	5	7	7	7	7	5	5	4	4	0
Tracked Dozer	300kW	No.	1	1	1	1	1	1	1	1	1	1	0
Tracked Dozer	231kW	No.	5	5	5	5	5	5	4	4	3	3	0
Front End Loader	6.4 m^3	No.	1	1	1	1	1	1	1	1	1	1	1
Grader	160kW	No.	4	3	4	4	4	4	4	4	4	3	2
Wheel Excavator	1 m^3	No.	1	1	1	1	1	1	1	1	1	1	0
Fuel Truck	20 t	No.	1	1	1	1	1	1	1	1	1	1	0
Watercart	30 t	No.	4	3	4	4	4	4	4	4	4	3	2

Notes:

1. Only three months of ore haulage required during the first year. Number of units required is 22, but only required for the three months.
2. In Year 9, there are only 10 months of production from the pits. Ore haulage will be required for full year.
3. The final year of feed has a 10 month duration. 22 units will be required for these 10 months.

17 RECOVERY METHODS

BACKGROUND

Several studies have been completed on the Project since 2005. These include:

- Gatsuurt Gold Project, Mongolia, Process Options Trade-off Study, SNC-Lavalin, July 2005). The study advanced biological oxidation of flotation concentrates (bio-oxidation) over pressure oxidation and whole ore direct cyanidation for use in the 2005 feasibility study

- The 2005 feasibiity study was based on a split-plant option, which includes crushing, grinding, flotation and flotation concentrate oxidation facilities at the Gatsuurt site, with shipping of the oxidized concentrate and gravity concentrate to Centerra's existing Boroo facility for cyanidation and gold recovery. The study showed this option was not economic due to high operating costs

- The March 2006 Centerra in house feasibility study was based on ore being hauled for processing at the Boroo site. Initial production of the direct leaching (oxide/upper transition) Gatsuurt ore will be processed through the existing Boroo processing facilities. Flotation and bio-oxidation facilities will be added during this phase to process Gatsuurt sulphide (including lower transition) ore. The study concluded the economics were positive.

- In 2009 Project development was reinitiated. The Project was divided into three phases as follows:
  - Phase 1; Gatsuurt to Boroo road and Gatsuurt site infrastructure.
  - Phase 2A; basic engineering for the BIOX® and related facilities addition to the Boroo site.
  - Phase 2B; detailed engineering and procurement for the BIOX® and related facilities addition to the Boroo site.
  - Phase 3; construction and all related in country activities for the BIOX® and related facilities addition to the Boroo site.

Basic Engineering was commenced in September 2009. Basic Engineering was completed by March 2010 and the initial stages of Detail Engineering were commenced. Project development was suspended in May 2010 due to legislative changes.

The current Technical Report is based on the 2009 Basic Engineering updated, as required, to provide the level of accuracy required.

The Boroo Mill facilities are described in the "Technical Report On The Boroo Gold Mine Mongolia, Centerra Gold Inc.", dated December 17, 2009 (Raponi and Redman, 2009).

PROCESS DESIGN CRITERIA

Process design criteria developed during 2009 Basic Engineering was updated by Outotec and Bantrel (Bantrel, 2009). Process design criteria is shown in Table 17-1.

TABLE 17-1 KEY PROCESS DESIGN CRITERIA

Criterion	Units	Value	Source
Processing rate	t/a	1 752 000	Centerra
Process plant operating utilization	%	92	Existing
Processing rate - annual average	t/d	4 800	Estimated from other Criteria
Primary crusher product - 80% passing	mm	129	Existing
Grinding circuit product - 80% passing	µm	75	Metallurgical Test Data
Gravity concentrate gold and silver recovery	% of mill feed	15	Typical
Flotation concentrate mass recovery	% of mill feed	6.8	Metallurgical Test Data
Flotation gold and silver recovery	% of mill feed	79.7	Metallurgical Test Data
Flotation pyrite and arsenopyrite recovery	% of mill feed	98.4	Metallurgical Test Data
Flotation rougher residence time	min	28.8	Metallurgical Test Data
Flotation concentrate gold grade	g/t	24	Metallurgical Test Data
Float concentrate silver grade	g/t	6	Metallurgical Test Data
Flotation concentrate sulphide sulphur grade	% S=	11.9	Metallurgical Test Data
Flotation concentrate arsenic grade	%As	5.6	Metallurgical Test Data
Flotation concentrate carbonate grade	% CO_3	3.0	Metallurgical Test Data
BIOX® plant availability	%	95	Outotec
BIOX® plant capacity (maximum)	t/h	19.4	Outotec
BIOX® sulphur feed rate (maximum)	t/h	1.73	Outotec
BIOX® retention time – total	Days	5.20	Outotec
Number of primary reactors (in parallel)		4	Outotec
Number of secondary reactors (in series)		3	Outotec
Total sulphide oxidation	%	95	Outotec
Total arsenic dissolution	%	100	Outotec
BIOX® limestone consumption	kg/t conc.	228	Outotec
Number of counter current decantation (CCD) washing stages		3	Outotec
Wash water ratio (wash water: feed solids)	w:w	5.39	Outotec
Ferric concentration in feed to CCD	g/L	33	Outotec

Criterion	Units	Value	Source
Ferric concentration in feed to CIL	g/L	<1	Outotec
Solution neutralization limestone addition	kg $CaCO_3$/t conc.	710	Outotec
Solution neutralization lime addition	kg CaO/t conc.	0	Outotec
Flotation tails added to solution neutralization	%	Up to 100	Outotec
Slurry neutralization residence time	h	4	Outotec
Neutralized slurry product pH	pH	11	Outotec
CIL leach time	hours	24	Outotec
CIL carbon concentration	g/L	20	Bantrel
NaCN addition	kg/t CIL feed	20	Outotec
Lime addition	kg/t CIL feed	15	Outotec
Number of CIL stages		6	Bantrel

PROCESS DESCRIPTION

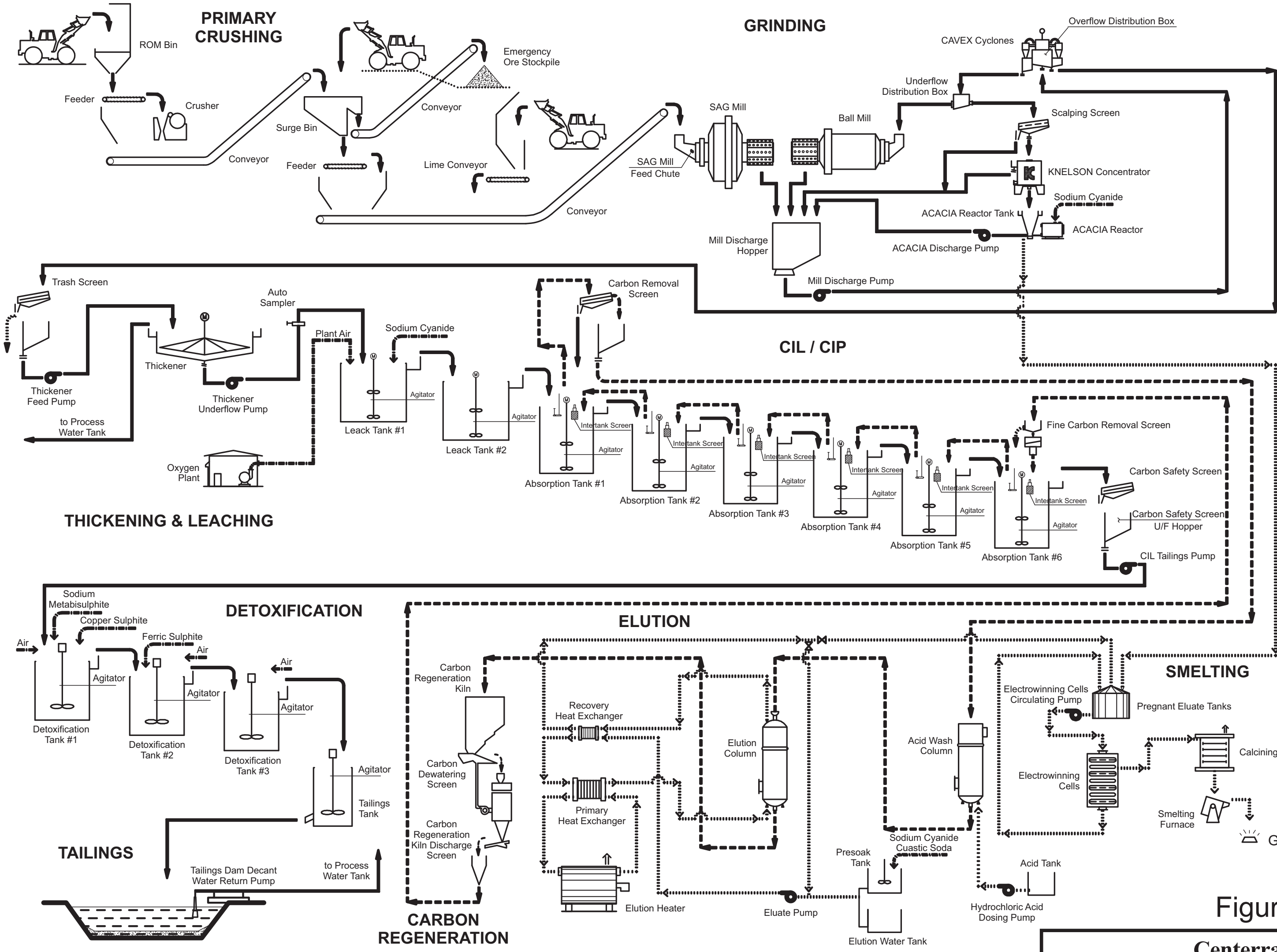
Gatsuurt ore will be treated at Centerra's Boroo Mine. Processing of the oxide ore will occur from Year 1 to Year 4 through the existing CIP facility at Boroo and the sulphide ore will be treated through a BIOX® process followed by a CIL facility which will be operational by Year 4.

OXIDE ORE PROCESSING

A simplified process flowsheet of the current Boroo process plant that will be used to process oxide ore is provided in Figure 17-1.

ROM ore is trucked to the existing Boroo crushing plant and either dumped directly into the crusher feed hopper or placed in a ROM stockpile for subsequent reclaim using a front end loader.

The flowsheet is a standard layout that consists of crushing, grinding, gravity concentration, cyanide leaching, and gold recovery in a CIP circuit.



Flow of ore/slurry
Flow of carbon
Flow of solution
Other

Figure 17-1

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
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Gatsuurt Oxide Process Flow Sheet

Date: December 2017 | File: Oxide Process Flow Sheet.cdr

A jaw crusher reduces the ore to 100% minus 200 mm. The crushed ore is fed directly to a semi-autogenous (SAG) mill or to a temporary coarse ore stockpile from which it can be reclaimed during crusher maintenance.

Lime for pH control is added to the SAG mill feed which, together with a ball mill, reduces the particle size to 80% passing 75 µm. Discharge from both mills is classified with cyclones. A portion of the total cyclone underflow reports to the gravity circuit, which consists of two 750 mm Knelson concentrators followed by an Acacia reactor where the gravity-recovered gold is leached in high cyanide solution. The remainder of the cyclone underflow plus the gravity circuit tailing reports to the ball mill.

The cyclone overflow is thickened prior to the leaching circuit that consists of two pre-leach tanks where air is injected, followed by six CIP tanks. Gold in solution from the leaching circuit is recovered on the carbon in the CIP circuit and subsequently stripped from the carbon and again put in solution to be recovered by electrowinning, followed by smelting and the production of doré bullion.

CIP tailings are detoxified to meet a target cyanide level of one milligram per litre (mg/L) using a modified sulphur dioxide-air process. Heavy metals are removed by treatment with ferric sulphate. The tailings are discharged by gravity to the permanent tailings facility 5 km from the process plant. The tailings storage is designed for zero discharge, with all of the water being reclaimed for re-use in the mill.

The Boroo mill was initially designed with a capacity to process 1.8 Mt/a of ore per year (5,000 t/d) but typically operated at 2.2 Mt/a or 7,000 t/d. Throughput for processing Gatsuurt ore was selected to provide the required fineness of grind for flotation feed of 80% passing 75 µm. Based on Gatsuurt ore hardness and the installed Boroo grinding equipment which at 92% operating time, grinding 217 t/h from 80% passing 129 mm to 80% passing 75 µm. The grinding circuit operation with Gatsuurt ore hardness has been modeled with JKSimMet to confirm that the required throughput can be achieved.

SULPHIDE ORE PROCESSING

Sulphide ore is milled through the existing Boroo crushing, grinding, and gravity circuit (Figure 17-1). During Boroo ore processing, the gravity circuit would typically recovery between 25% and 55% of gold production. For this report, gravity gold recovery estimated at 15% gold.

Leached gravity concentrates from the Acacia reactor are returned to the cyclone feed sump. The solids will contain cyanide. The Acacia reactor incorporates solids washing prior to transfer to ensure the cyanide concentration is below concentrations that are detrimental to flotation and toxic to the BIOX® process.

Grinding circuit cyclone overflow is fed to a new bank of six 40-m^3 capacity mechanically agitated flotation cells to recover approximately 6% to 8% by weight of flotation feed as a rougher concentrate. The flotation cells are arranged in one group of four cells, followed by a groups of two cells, which allow concentrate from the last two cells to be recirculated to flotation feed if required to reduce non-sulphide gangue recovery to rougher concentrate.

Flotation recoveries, residence times, concentrate grades, and flotation reagent consumptions selected by SNC-Lavalin have been based on the testwork completed up to 2006.

Flotation recoveries, residence times, concentrate grades, and flotation reagent consumptions selected by SNC-Lavalin have been based on the data described in Section 13.

The design for the BIOX® circuit was supplied by Outotec in an engineering package based on pilot scale testwork conducted by Biomin in South Africa during 2005. Reagent consumptions, process operating conditions, reactor size, and installed agitator power were also provided by Outotec.

Thickened slurry is pumped from the concentrate thickener to the BIOX® surge tank, which provides 24 hours storage. Slurry is pumped from the surge tank to a splitter, which sequentially feeds the four primary BIOX® reactors. Water is added to maintain slurry density at 20% solids in the feed to the primary reactors. Slurry flows by gravity from the BIOX® primary reactors through a launder to the first BIOX® secondary reactor. Slurry flow between the secondary BIOX® reactors is by gravity. All BIOX® reactors have a capacity of 1,320 m^3. The primary reactors and first stage secondary reactor have 200 kW agitators. The remaining second stage secondary reactors have 132 kW agitators. All of the BIOX® reactors and agitator wet ends are constructed of 304L type stainless steel.

The oxidation of the sulphide minerals occurs due to the action of the bacteria *thiobacillus ferrooxidans* on the ore particles. Essential nitrogen, phosphorus, and potassium required for bacterial growth are added as a solution. Sufficient carbon dioxide is assumed to be available from decomposition of the carbonate minerals in the ore and from the finely ground limestone slurry added to the reaction tanks to maintain solution pH in the optimum range for bacterial growth, typically between 1.2 and 1.5.

Oxygen required for the oxidation reaction is supplied in the form of compressed air introduced into the tank by a sparging system and agitator combination specifically designed for oxygen transfer. Due to the low oxygen transfer efficiency of agitated tanks, a large amount of air is necessary.

An inoculum preparation system is provided to generate bacterial culture for start-up, and to replace bacteria in the event that changes in process conditions have caused bacterial mortality. Three agitated tanks of varying capacity are provided to sequentially create larger volumes of inoculum.

The reaction tanks and the inoculum tanks are fitted with submerged stainless steel coils which heat the slurry with steam or cool the slurry with water as required to maintain process slurry temperature in the range of 40°C to 45°C. The overall peak cooling load at the maximum sulphide sulphur level of 1.4% is estimated to be approximately 12,000 kW. The peak heating load to maintain slurry temperature during winter when the process plant is shut down is estimated to be approximately 4,500 kW. Heating is not required during normal operation.

The product from third secondary BIOX® reactor is washed in a counter current decantation (CCD) circuit consisting of three thickeners in series. Each thickener is 13,500 mm in diameter constructed of 304 type stainless steel. The underflow from the third CCD thickener is neutralized with hydrated lime in two tanks in series. Each tank has a capacity of 50 m^3 and are constructed in 304 type stainless steel. The pH is raised to 10.5 in preparation for subsequent cyanidation.

The BIOX® solution neutralization circuit consists of six 700 m^3 agitated tanks in series. Overflow from the first CCD wash thickener is neutralized, first by contact with flotation tailings and finely ground limestone at pH 10.5. The addition of flotation tailings to neutralization provides a small contribution to the carbonate requirements but also provides seeding for the precipitated gypsum and may act to prevent scale build up on the tank walls.

Neutralized solution and solids are discharged to the 40,000 mm diameter water recovery thickener. The balance of flotation tailings not used in neutralization are added to the recovery thickener to enhance the settling characteristics of the neutralized slurry. Thickener overflow water is recycled as BIOX® feed dilution water and CCD wash water. Thickened underflow is pumped to the tailings pumpbox, where it is pumped to a new TMF. This TMF will be cyanide free which will permit the recycling of supernatant water to the process for use in grinding and flotation.

Water from the solution neutralization circuit is kept in closed-circuit within the BIOX® circuit to prevent contamination and subsequent risk of bacterial mortality. In addition, BIOX® solutions will likely contain significant concentrations of sulphates and gypsum, which could be detrimental to the rest of the plant.

No.3 CCD thickener underflow is pumped to slurry neutralization step. This consists of two tanks and each stage provides two hours residence time. Hydrated lime is used for both stages of slurry neutralization or to raise the pH to 11.5.

A cooling tower is used to remove heat from water circulated through cooling coils in the biological oxidation tanks. A separate cooling tower is used to remove heat from process air blower aftercoolers. The designed slurry temperature required is 42°C.

The CIL circuit receives pH adjusted slurry from the BIOX® process. The CIL circuit was designed to accommodate the existing three tonne elution circuit while minimizing capital. The leaching kinetics exhibited in metallurgical testwork showed that the BIOX® slurry is fast leaching to 90% within the first four hours, followed by a relatively flat profile of a slowly leaching tail. The CIL circuit is capable of achieving approximately 95.9% gold recovery, based on BIOX®, providing an overall recovery of 87%.

The CIL feed will be pumped to the leach feed distribution box where it will combine with various internal recycle streams and be directed to the first available CIL tank (CIL Tank 1 or CIL Tank 2). Transfer of slurry between the tanks will be by gravity. A total of six CIL tanks will be installed for the sulphide ore processing phase of the Project. The circuit will provide 52 hours of leach time at design throughput. Cyanide will be added to any of the first three CIL tanks to leach the gold. Cyanide analysis will be undertaken by the automatic analyser to allow for automatic control of cyanide addition.

Each CIL tank will have 312 m^3 live volume, will be air sparged, mechanically agitated, and contain a carbon screen. The leaching slurry will proceed by gravity through the series of tanks with the possibility of bypassing any tank in the series for maintenance. The mechanically swept wedge wire inter-tank carbon screens will prevent carbon from flowing with the slurry and retain the carbon in each vessel. Granular activated carbon will be added to the final tank in the series, be pumped counter currently to the slurry flow via recessed impeller pumps, and be extracted from the first tank in the series. Leached gold will be progressively adsorbed onto the activated carbon in each tank in the series.

The average carbon concentration will be 20 g/L, the carbon advance rate will be three tonnes per day and the carbon inventory in the CIL circuit will be 36 t.

Loaded carbon and slurry extracted from the first CIL tank will report to the loaded carbon screen. Water sprays will clean the loaded carbon of pulp, and the slurry will report to screen underflow and return to the CIL circuit. The cleaned loaded carbon will be collected in the carbon transfer hopper and be pumped to the existing acid wash column. CIL tailings will be treated in the existing Boroo cyanide destruction facilities, using the SO_2/air process. Treated tailings will be discharged to a new Boroo tailings facility. This small new cell will hold the CIP tailings over the life of the Gatsuurt processing. No water will be recycled from this cell to the process.

Major consumables at the Boroo site include grinding media, flocculant, hydrated lime, flotation collector, flotation frother, limestone, sodium cyanide, granular activated carbon, copper sulphate, sodium metabisulphite, caustic soda, ferric sulphate, fuel oil, nutrient, small amounts of cooling tower reagents, and scale removal reagents.

The existing Boroo plant air, instrument air, and electrical power distribution equipment will not require modifications.

POWER AND WATER REQUIREMENTS

The addition of the biological oxidation circuit to the existing Boroo Gold Mine processing plant will require an additional 12 MW of electrical power. The existing power at Boroo is supplied from a regional electrical power grid. When the demand and utilization factors are applied, the estimated peak demand is expected to be approximately 8.9 MW. This will require an additional transformer of 15 MVA, 110-33 kV capacity to support Project requirements.

Fresh water required at the Boroo process plant will be taken from the existing bore field located in Boroo river valley to the east of the mine and pumped to a water storage tank at the process plant site. The fresh water supply system is designed to supply an average flow of 300 m^3 per hour.

18 PROJECT INFRASTRUCTURE

INTRODUCTION

The Gatsuurt Gold Project requires the support of a well-established and maintained infrastructure especially considering that all ore from the Gatsuurt pit is transported via an existing (public) haul road to the Boroo Mine site. This is especially challenging considering the climatic changes.

Site infrastructure will be comprised of a number of newly constructed facilities at the Gatsuurt site, as well as an expansion of existing facilities at Centerra's Boroo Gold Mine and Mill.

The facilities at Gatsuurt include, as of December 2016:

- Fifty-two kilometre gravel access haul road from Gatsuurt to Boroo
- Site areas cleared for waste and sub-grade rock dumps
- Electrical infrastructure, which is mostly in place (all buildings, shops, and offices) with the exception of the pit area for dewatering and a pit waste water treatment facility, if required
- Fresh/Potable/Process water supply and intra-site distribution system
- Waste water/sewage treatment and handling facility
- Mine services buildings, including on-site eating facility and first aid stations
- Satellite telecommunication

The existing facilities at the Boroo Gold Mine will be used to process ore from the open pit at the Gatsuurt Gold Project. The Boroo facilities include:

- Process plant, including facilities for unloading and feeding of ore, as well as grinding and flotation of materials
- Tailings management and heap leach pads, including impervious multi-layered basal linings of waste dumps
- Recycling treatment circuits to remove arsenic and cyanide from tailings
- Storage facilities for chemicals and reagents
- Dewatering facilities and return water lines
- Elevated tailings dam to accommodate additional tailings
- Warehousing

- Administration offices

Water supplies for villages in the Gatsuurt area are obtained directly from water wells and local rivers. In addition, former placer operation dams have ponded river water at several sites in the Gatsuurt River Valley. A water treatment plant has been considered as part of the Gatsuurt site infrastructure due to ARD issues mentioned in Section 16.

The existing Boroo site is shown in Figure 18-1. The proposed Gatsuurt Mine site is shown in Section 16 on Figure 16-1.

Waste dump details are found in Section 16 (Waste Dump Design). Tailings requirements are discussed in Section 17 and also further in this Section.

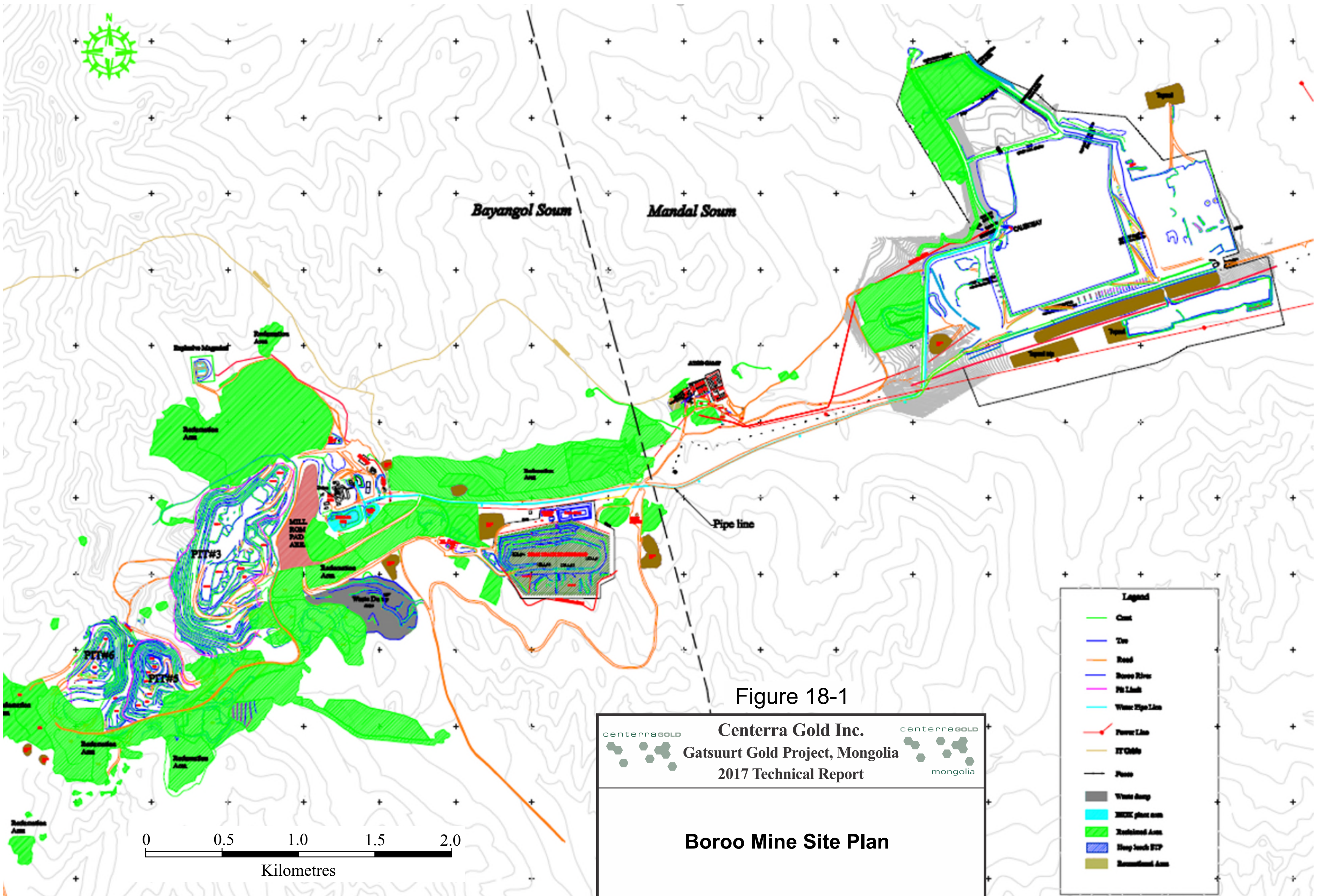

Figure 18-1

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Gatsuurt Gold Project, Mongolia
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Boroo Mine Site Plan

Date: December 2017	File: Boroo Mine Site Plan.cdr

ACCESS ROADS

The Boroo Gold Mine is located approximately 90 km north of Ulaanbaatar in the Mandal Soum of Selenge Aimag in central northern Mongolia. The new Gatsuurt process plant facilities will be constructed adjacent to the existing Boroo process plant. The access road connecting the Gatsuurt site with the Boroo Mine is 52 km in length measured from the ore stockpile pad at the Gatsuurt site to the ROM pad at the Boroo crusher located in the Boroo site. The road is of all-weather, earth and stone construction designed to accommodate 40-t tractor trailers. Refer to Figure 18-1 and 18-2 for more details.

Access to the Gatsuurt Gold Project is, in part, via the highway that connects Ulaanbaatar to Sükhbaatar (Suhbaatar) near the north border with Russia which roughly takes one and a half hours. From the highway, there is a two lane all-weather road through the general countryside which leads to the site, about 10 km from the highway. The access road between the Boroo and Gatsuurt sites is by public road, shown as the brown line in Figure 18-2.

Between the two sites, this public road will serve as the haul road to transport ore from the Gatsuurt site to the Boroo site where it will be processed. The road includes one two lane bridge over the Boroo River.

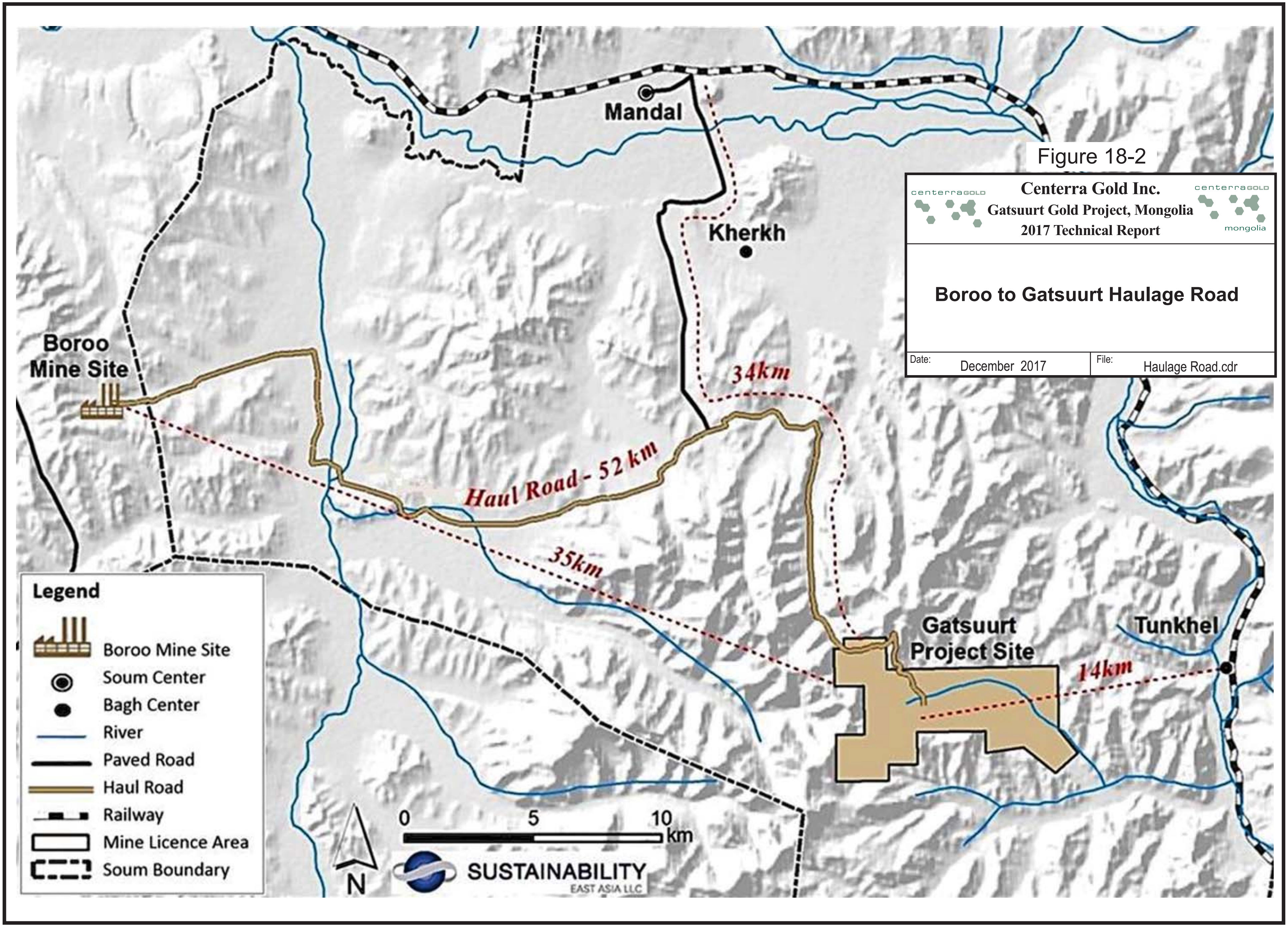

Figure 18-2
centerraGOLD
Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report
centerraGOLD mongolia
Boroo to Gatsuurt Haulage Road
Date: December 2017
File: Haulage Road.cdr
Mandal
Kherkh
Boroo Mine Site
Haul Road - 52 km
34km
35km
14km
Gatsuurt Project Site
Tunkhel
Legend
Boroo Mine Site
Soum Center
Bagh Center
River
Paved Road
Haul Road
Railway
Mine Licence Area
Soum Boundary
0 5 10 km
N
SUSTAINABILITY
EAST ASIA LLC

POWER SUPPLY

The power demand of the existing Boroo facility is 10 MW with an expected increase of 8 MW to support the new process facilities. Power is currently supplied to the existing Boroo facility via a 110 kilovolt (kV) electric power line that crosses the Sujigtei River Valley within 25 km of the Gatsuurt Gold Project. A 35 kV power line carries electricity from the Zuunkharaa distribution centre to Tunkhel and from there, via public sources, a 10 kV line supplies power to the Gatsuurt site. Both of the powerlines feeding the mine sites are in need of rehabilitation due to aged infrastructure and continued outage issues, and both existing lines would need to be demolished.

A new single circuit 110 kV overhead line (OHL), 90 km long, would need to be constructed from the Darkhan substation to the Boroo substation with an ACSR-240/32 conductor, in addition to expanding the existing 110/33 kV Boroo substation to 30 MVA. The general routing of the power line is shown in Figure 18-3.

Power to the Gatsuurt site would be supplied via a new, 40 km long, single circuit 110 kV OHL with an ACSR-120/19 conductor from the Zuunkhaara substation to Gatsuurt with the construction of a new 110/10 kV, 1x10 MVA substation.

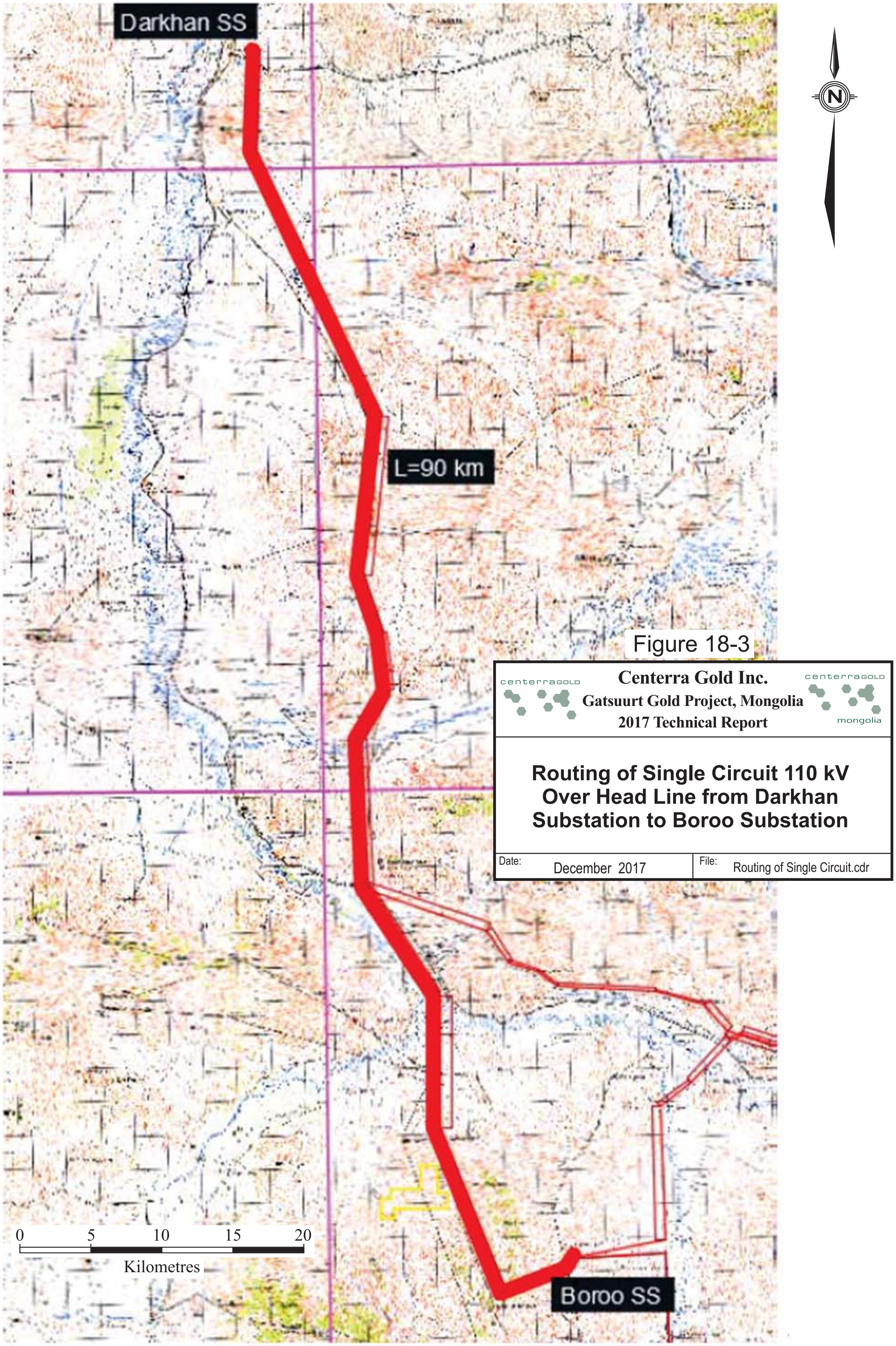

Figure 18-3

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
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centerraGOLD

centerraGOLD mongolia

Routing of Single Circuit 110 kV Over Head Line from Darkhan Substation to Boroo Substation

Date: December 2017	File: Routing of Single Circuit.cdr

PROCESS PLANT FACILITIES

The existing Boroo Gold plant (currently on Care and Maintenance) employs a Leach/CIP and gravity concentration for gold recovery. The plant comprises crushing, grinding, gravity concentration, thickening, Leach and Adsorption, and cyanide detoxification steps as shown in Figure 18-4. Detoxified tailings are deposited into a zero discharge tailings storage facility.

A rehabilitation of the existing Boroo facility is required prior to start-up and processing of oxide ore. This rehabilitation affects mechanical equipment, structural steel, electrical equipment, I.T., and instrumentation, etc.

For processing of sulphide ore, new process facilities will be built adjacent to the existing process plant during the first three years of operation. These are:

- Flotation, including concentrate thickening, reagent mixing and distribution, flotation air;
- BIOX® circuit for sulphide oxidation;
- BIOX® product solids neutralization;
- BIOX® solution neutralization and water recovery thickener; and,
- CIL circuit for gold recovery.

The proposed new layout of the Boroo process plant is shown in Figure 18-5.

Figure 18-4

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
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Existing Boroo Overall Site Plan

Date: December 2017	File: Boroo Site Plan.cdr

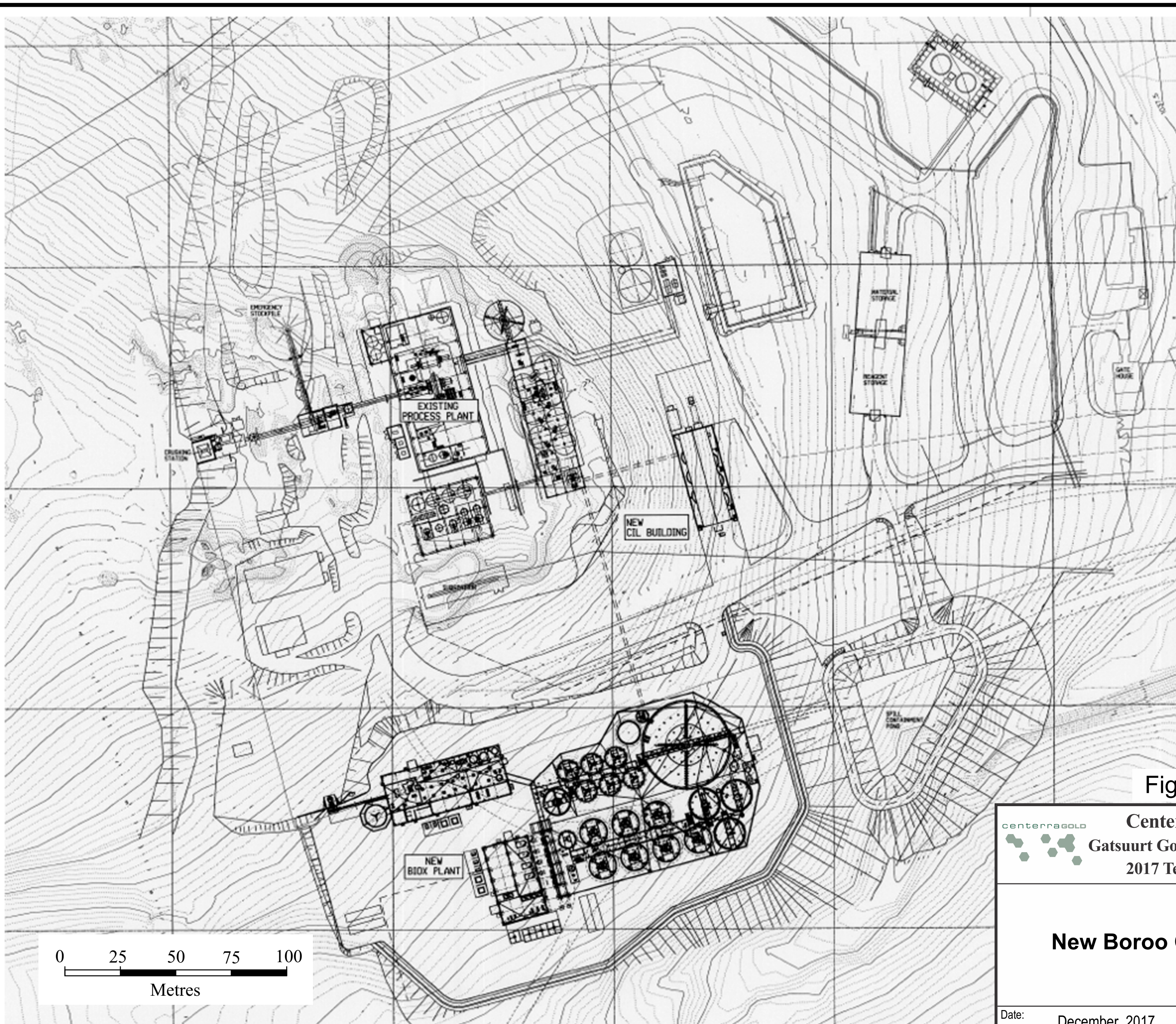

Figure 18-5

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
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New Boroo Overall Site Plan

Date: December 2017	File: New Boroo Site Plan.cdr

SITE SERVICES

An existing potable water treatment plant will treat the fresh water prior to storage in the potable water storage tank. Sewage will be collected and chlorinated before disposal. Effluent from the sewage treatment plant will be discharged into the tailings facility at Boroo and a septic field at the Gatsuurt site. The capacity of the sewage system will be increased to accommodate the enlarged permanent camp and the addition of temporary facilities.

At the Gatsuurt site, diesel and gasoline storage facilities will be provided at the mine services area. One diesel fuel storage tank and one gasoline storage tank will be installed above-ground. A mine maintenance/operations administration building and a security gatehouse facility will be built at the mine site.

ACCOMMODATION

During the operating phase of the Project, Gatsuurt operating personnel will be housed at the existing facilities at the Boroo Mine and transported to and from Gatsuurt on a shift to shift basis.

A 400-person construction camp will be located to the east of the existing Boroo accommodation facilities. This camp will include 30-person management accommodation units built next to the Boroo facility, and 80 4-person gers, kitchen/dining facility, and recreation facility.

There is no planned Centerra worker accommodation facility to be located at Gatsuurt. The mining contractor will construct a small camp to house mining personnel.

TAILINGS MANAGEMENT FACILITY

The Boroo site currently contains a TMF located approximately 5 km south of the process plant and 2 km south of the camp. This existing TMF contains a Main Cell at elevation 934.5 MASL which will be expanded to an elevation of 937.7 MASL for the additional 3.6 million m^3 volume requirement for the oxide ore with an added capacity of 0.44 million m^3 for the BIOX® CIP tailings. The BIOX® CIL tailings will be stored in a new East Cell which will occupy 9.8

million m^3 of tailings and constructed from an elevation of 893 MASL to an elevation of 924 MASL. The TMF is shown in Figure 18-6.

WATER TREATMENT PLANT

A new Waste Water Treatment Plant is required at the Boroo site for the removal of metals (arsenic) from the water currently stored in different pits. The water treatment plant will be erected at the Boroo site and operated there during the oxide campaign period. It will then require disassembly, transportation and re-erection at the Gatsuurt site for operation during the subsequent sulphide campaign. The new Plant is based on a proposal provided by Veolia Water Technologies Korea Co. Ltd (VWT Group) using their MetClean™ process.

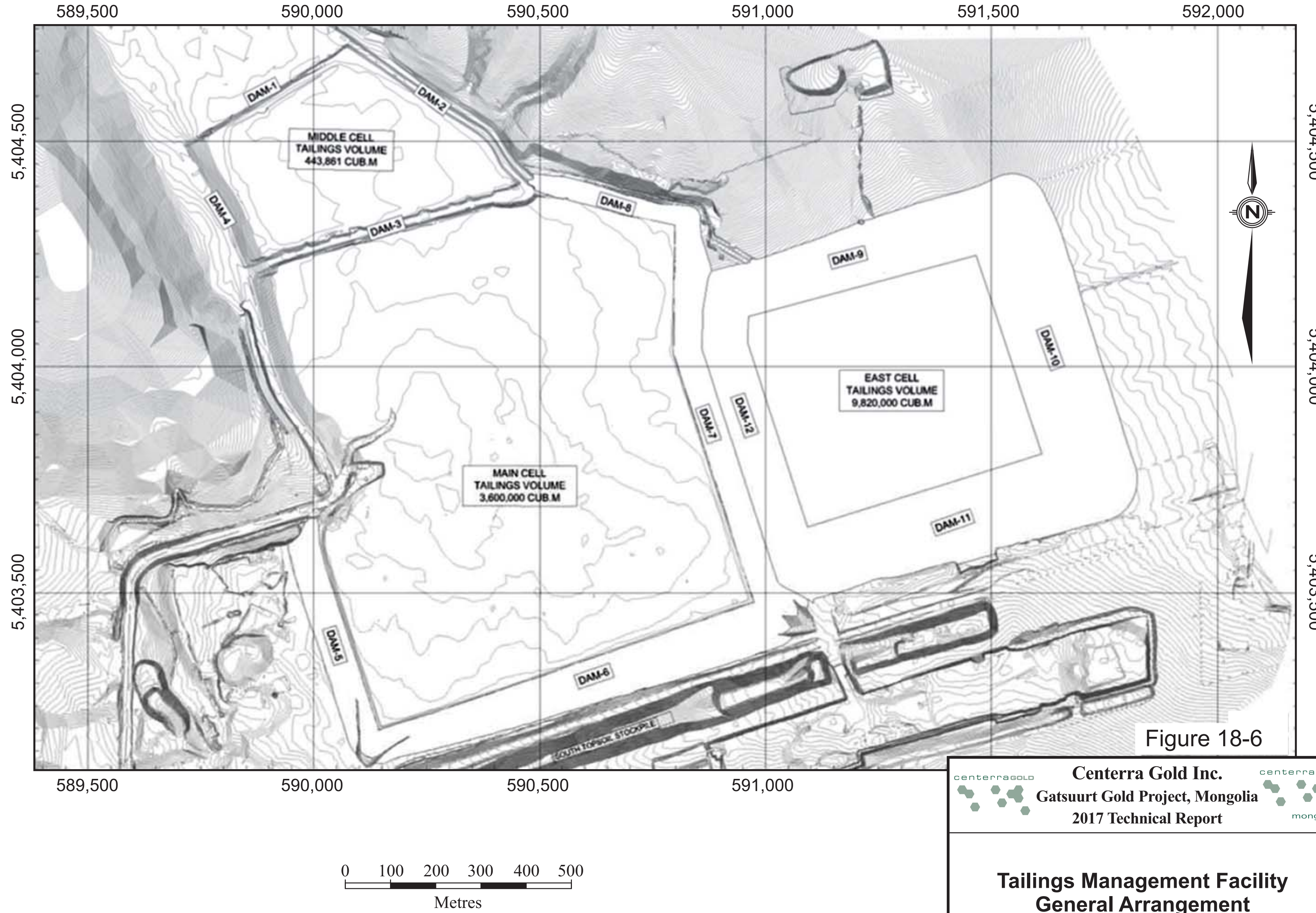

Figure 18-6

Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
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centerraGOLD

centerraGOLD mongolia

Tailings Management Facility General Arrangement

Date: December 2017

File: Tails Facility.cdr

Source: Usny Erchim, 2017.

19 MARKET STUDIES AND CONTRACTS

MARKETS

Gold is the principal commodity which will be produced at Gatsuurt and is freely traded at prices that are widely known, so that prospects for sale of any production are virtually assured. All gold produced by Gatsuurt will be in the form of doré bars, which will then be shipped to a refiner who will refine the doré into bullion. The bullion will then be sold directly on the open market to gold trading institutions at prevailing market prices.

CONTRACTS

As noted above, prior to commencing Project development, CGM expects to enter into, among others, an Investment Agreement and Deposit Development Agreement with the Government of Mongolia as well as one or more Community Development Agreements.

The LOM production schedule contained in this Technical Report is based on utilizing a mining contractor. No contracts are in place at this time for such a contractor.

No gold sales contracts have been entered into in connection with the Project.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

ENVIRONMENTAL AND SOCIAL SETTING

The Gatsuurt gold deposit is located approximately 35 km to the south of Centerra's existing Boroo Gold Mine and processing facility. Tunkhel village is located 14 km to the east and Zuunkharaa town is 34 km to the northwest of the Gatsuurt Project. The existing 52 km access road between the Boroo and Gatsuurt sites was permitted, engineered, and constructed between 2005 and 2010 as a mine haul road which was converted by order of road administrative authorities into a combination of mine and public road. The road is maintained solely by Centerra.

Artisanal placer gold mining of the river alluvium has occurred in the past within the Project area along the Gatsuurt River valley and has significantly altered the appearance, drainage and water quality within the valley and permanently damaged the top 5 to 15 m of the land surface. Most of the soils, vegetation, aquatic and subaquatic ecosystems were disrupted and/or permanently lost as a result of this alluvial mining. These losses to the ecosystem may restrict the range of reclamation that can be performed on the property. However, the mine closure and reclamation planning proposes to use available soils and materials to restore the site at the cessation of mining to achieve a safe and stable landscape.

The Gatsuurt area is characterized by numerous shallow excavations, piles of alluvial sands and trenches of various dimensions that are located within the proposed mine area as a result of these past mining activities.

Water quality within the Gatsuurt River valley has been impacted by the previous mining activities by other companies, with increased sedimentation of surface water and higher than normal concentrations of some heavy metals and other elements, including arsenic, following the exposure to air of previously buried and saturated soil and rock layers. The exposure of these rocks during past mining activities causes chemical reactions that release naturally occurring metals into the environment through a process called acid rock drainage. In addition to the impacts of past mining activities, the forest surrounding the Gatsuurt valley has been and continues to be subjected to harvesting for timber. Evidence suggests that much of the timber harvesting in the Gatsuurt areas is unregulated.

Mandal Soum has the largest population of any soum in Mongolia at just over 25,000 people. The soum is a prime agricultural and cropping area in Mongolia. Furthermore, 70% of the Soum is covered by forested land, which supports forestry. Similar to Mandal Soum, Bayangol is a prime agricultural area, and is connected to Ulaanbaatar, Darkhan and Selenge by a paved road, and by railway, on road and rail routes connecting Mongolia with Russia. Some of the key social issues in the area include the lack of vocational education opportunities, unemployment and underemployment, extensive land degradation due to overgrazing and poor agricultural practices, and inadequate availability of health care personnel.

The examination of extensive drilled holes at Gatsuurt has not found any permafrost that could be affected by the mining and associated development activities. It is not expected that any permafrost will be disturbed by the Gatsuurt mining development; however, should permafrost be identified in the Project Area, Centerra will consult with the relevant authorities and the Mongolian Academy of Science to ensure that any adverse impacts to environmental, social or cultural values are mitigated accordingly.

ENVIRONMENTAL STUDIES

A comprehensive environmental baseline study was prepared for Gatsuurt in 2005 and 2006. The environmental monitoring program that was established with the 2005 baseline program has expanded since that date with the addition of more types of monitoring and the expansion of existing monitoring programs. This expanded monitoring program provides international level environmental information that will facilitate future operations. This monitoring data will continue to be collected for surface water, groundwater, air, soil, and meteorological data.

A detailed EIA for the Project was approved in December 2009, and later updated in 2014 and 2016. In April 2016, conclusion and recommendations on the updated EIA were issued by Ministry of Environment and Tourism (previously Ministry of Environment, Green

Development and Tourism). In December 2016, a detailed EIA was revised and submitted by CGM to the Ministry.

Public consultation meetings of the Gatsuurt detailed EIA were organized in July 2016. Findings from the detailed EIA were introduced by the consultant company, Eco Trade. Local community representatives actively participated in the meetings and Centerra believes that their perception of the Project was positive. Most of the questions and comments raised during the meeting were related to employment opportunities and socio-economic benefits to the local community as well as environmental issues.

The Project is also subject to financing from the European Bank for Reconstruction and Development (EBRD) as part of the Corporate Facility entered into with Centerra. In 2016, an EBRD compliant Environmental & Social Impact Assessment (ESIA) was completed that included, specific sections of cultural heritage and also a detailed social baseline and impact assessment. The ESIA was disclosed to the public and all interested parties at public locations such as local administration offices, library, and information centres. The documents included:

- Non-Technical Summary
- Social Impact Assessment
- Environmental and Social Action Plan
- Stakeholder Engagement Plan
- Social Management Plans
- Environmental Management Plans
- Intangible Cultural Heritage Study Report
- Project Description and Environmental Impact Assessment component of the Gatsuurt Project Feasibility Study (2014)
- Detailed Environmental Impact Assessments for the BIOX Plant Project, Ore Transportation Road Project, and for Boroo Gold Heap Leach Project;

Environmental studies completed in 2016 and 2017 included:

- Permafrost Study in the Gatsuurt deposit area by the Institute of Geography and Geo-Ecology, Science Academy of Mongolia

- Gatsuurt Archeological and Paleontological Studies (Archeological rescue & salvage excavation) by Institute of History and Archeology, Science Academy of Mongolia

- Rare plants study by Botanical Institute of Mongolia
- Study on Content and Spatial Distribution of Arsenic in the Soil of Kharaa River Basin by Institute of Geography and Geo-Ecology, Science Academy of Mongolia
- Study on Concentration and Spatial Distribution of Arsenic in Water of Kharaa River Basin by Institute of Chemistry and Chemical Technology, Science Academy of Mongolia
- Gatsuurt 2016 - Hydrogeological Investigation: Mine Water Management Assessment Report by 'Groundwater Solution'
- Centerra Gatsuurt – Phase 2 Geochemistry Report by Schafer Limited
- Gatsuurt Water Management Plan for Project Feasibility Study by Sustainability East Asia

PROJECT PERMITTING

CGM has secured many of the operating permits required to operate the Project in accordance with Mongolian regulatory requirements. Approved permits and licences include Land Use Permit for the Gatsuurt Mine, Gatsuurt Mineral Reserve Report Approval (2013), Gatsuurt Mining Licence, Gatsuurt Feasibility Study (2014), BIOX® Plant Environmental Impact Assessment (EIA) (2010), Gatsuurt – Boroo Haul Road EIA (2006), Water Reserve Approval (2010), and Gatsuurt Landfill Permit (2007).

Outstanding permits and licences include Hazardous Chemicals Permit, Gatsuurt Gold Mine Detailed EIA (submitted), Water Use Permit, and Mine Closure Plan. The status of current permits and approvals is summarized in Table 20-1.

TABLE 20-1 SUMMARY OF MAJOR PERMITS AND APPROVALS

Approvals and Permits	Project Activity	Statutory Basis	Approval / Permit Status (June 2017)
Hazardous Chemicals Permit	Boroo Plant Expansion – Use of hazardous materials in the Boroo process plant	Law on Toxic and Hazardous Chemicals	This permit will be required prior to the Boroo process plant operation starts.
Land Use Permit – Gatsuurt Mine	Land use permit and associated land use agreement is required to use the Gatsuurt mine land area.	Law on Land	Approved.
Gatsuurt Reserve Approval	CGM is required to prepare a Mineral Reserve report (RR) in compliance with Mongolian regulations.	Minerals Law Law on Sub Soil	Reserve approved on December 27, 2013. An updated Mineral Reserve report has been submitted and is under review.
Gatsuurt Mining Licence	Conduct operational mining of the Gatsuurt deposit	Law on Licensing	Licences in place for Gatsuurt tenements
Gatsuurt Feasibility Study	Mongolian Feasibility Study for Gatsuurt Gold Mine	Minerals Law	Approved in 2014. Following approval of the 2017 Mineral reserve report an updated Mongolian feasibility study will be submitted for approval.
BIOX® Plant EIA	Approval for the construction and operation of the BIOX® Plant	Law on Environmental Impact Assessment	Approved in 2010.
Gatsuurt- Boroo Haul Road EIA	Approval for the construction and operation of the haul road between Gatsuurt and Boroo sites.	Law on Environmental Impact Assessment	Approved in 2006.
Gatsuurt Gold Mine Detailed EIA	Approval to mine the Gatsuurt deposit, including associated mineral production activities.	Law on Environmental Impact Assessment	Submitted to the Ministry of Environment and Tourism.
Water Use Permit	Use of water for processing activities and BIOX® Plant	Law on Water	Not yet required. Water use permit will be sought after the mine feasibility study and Environmental Impact Assessments are approved.
Water Reserve Approval	Extraction of water from Project borefields	Law on Water	Approved in 2010.
Historical and cultural heritage exploration and survey	The conduct of archaeological and cultural heritage surveys to ensure mining activities do not disturb sites that may be in the Project area.	Law on Protecting Cultural Heritage	Surveys undertaken in accordance with legislation. No sites found within Project operational areas.
Gatsuurt Landfill Permit	Construction and operation of a landfill	Law on Waste	Permit obtained in 2007.
Mine Closure Plan	Mine closure	Law on Environmental Impact Assessment	Not yet required, CGM is required to develop mine closure plan between start of the mine operation and three years prior to the mine closure.

SOCIAL OR COMMUNITY REQUIREMENTS

DEMOGRAPHY

There are unique demographic challenges confronting Mandal and Bayangol soums. Due to its large population of youth between the ages of 20 and 24, the key challenge for Mandal Soum is driving vocational educational, economic opportunities and work spaces for youth. In Bayangol soum, the key challenge, based on its large population of young children, is providing sufficient educational facilities. Statistically, migration is declining in the Project area, although there is a perception among communities in the Project area that there is an influx of migrant herders who are viewed as a key source of pastureland degradation and land use conflict. An emerging risk is the growing number of female headed households in the Project area and the accompanying increase in vulnerable households. The highest percentage of the population with no formal education at all is seen in Tunkhel bagh (11%) and so Tunkhel bagh citizens may be less equipped to take advantage of any economic and employment opportunities presented by the Project due to lagging education indicators.

SOCIAL STRUCTURES

The levels of trust in public institutions in the Project area are generally low due to community dissatisfaction with the activities of public organizations, especially political parties. Community perceptions on corruption are also a key source of discontent with the political situation in both Mandal and Bayangol. In both soums, the soum Citizens Representative Khural (CRK) and soum Government have the greatest overall levels of trust among public organizations. Baghs face unique challenges in the Project area, including the perception among bagh citizens that their voice does not count or is not heard in decision-making. Welfare and community upliftment projects have generally low levels of awareness in the Project area. The program with the highest level of awareness is the “restocking livestock program” run by the Government.

ECONOMY

Steep price inflation of goods and services is the major concern for households in the Project area, as are the poor quality of goods. Tunkhel bagh households experience these issues most acutely in the Project area due the bagh’s more remote location. A related issue is that households in the Project area appear to be increasingly living on credit and are struggling to pay back their loans. Households in the Project area are also increasingly frustrated and limited by their lack of access to professional assistance, knowledge, and technology on effective crop and vegetable farming methods that may assist them in being better farmers and in obtaining higher yields. Moreover, livestock insurance coverage is low in the Project area, driven by a general lack of awareness of insurance (in the case of dzud, fire, drought, or natural disaster). There is also a general lack of awareness about the formal mining sector in the Project area, the processes and technicalities thereof, and the potential benefits and risks associated with large scale mining.

SOCIAL INFRASTRUCTURE

Kindergarten facilities are stretched to over-capacity in both Mandal and Bayangol, with children being turned away. There are limited opportunities in the Project area to access tertiary and vocational education and the facilities that do exist in Mandal Soum provide vocational training for a predominantly male student population in traditionally male professions. Households in the Project area primarily receive information, news, and entertainment from television rather than radio. Vehicles and the railway are the most common forms of public transport, with rail transport the most popular. Baseline studies suggest that construction of the Project haul road has enhanced opportunities for local small businesses.

The main sources of drinking water in the Project area are wells, rivers and springs, and mobile distribution points. Bayangol sources more water from unprotected wells (at a higher risk of contamination than protected wells) than Mandal Soum. Most households residing in soum centres consume energy from the central energy system, while 21.5% of surveyed households in Mandal Soum and 39.9% of surveyed households in Bayangol Soum use renewable energy. Solar systems are prevalent across all surveyed areas, in particular, for herders and those in Bayangol. Tunkhel Bagh was the exception at the bagh level with a greater reliance on the local diesel system compared to other baghs. 19.5% of surveyed households in Mandal Soum also reported that their household energy resource is based on the local diesel system, which only works in the winter months between October and April. Heat supply is an essential basic service in Mongolia, but it is not accessible to everyone, in particular, people in rural areas. The current heating system in soums mainly consists of (i) small stoves; and (ii) centralized and decentralized coal fired boilers. Most household heating is independent of the centralized grid.

LAND USE AND NATURAL RESOURCES

Overgrazing and the accompanying pastureland degradation is a central issue in the Project area, due to existing herders and livestock owners having increased the number of animals in recent years. Overgrazing problems have resulted in land use disputes among herders in some areas, and between herders and crop farmers. Most herders have a winter shelter and spring grazing area, even if these are not always formally registered, and so, many do not pay land use fees. This means that there is less funding available to be reinvested into environmental improvements (regulation requires 15% of land use fees are spent on environmental improvements). A related theme is that more than half of all surveyed households do not hold possession certificates for use of pastureland. With respect to water resources, household water availability is not currently a major concern, although the potential impact of mining on water resource availability and quality are key issues. This indicates that water pollution is increasingly likely to become a key community concern particularly in Tunkhel Bagh, regardless of any actual impacts.

The social baseline survey found that 60% of Mandal Soum households were involved in Artisanal and Small-scale Mining (ASM) – to supplement their incomes/livelihoods, as a hobby, or as a poverty reduction strategy. Around 228 soum residents were members of the 30 officially registered ASM cooperatives (three large NGOs, with around 30 to 100 members and 27 small cooperatives with 3 - 10 members). The ASM context changed from when the social baseline study for the Project was conducted in March – April 2015 compared to September 2015 when ASM manifested in a different form. In September over 5,000 miners were recorded conducting trespass and illegal mining (TIM) at the Project site. These miners gained access to the site in large vehicle convoys and overcoming safety and environmental measures in place at the site. Since September 2015, the illegal activity at Gatsuurt had escalated and BGC's security in cooperation with the Mongolian Police have undertaken some 28 operations during which over 5,000 intruders and more than 200 vehicles have been expelled from the area.

COMMUNITY HEALTH, SAFETY AND SECURITY

There is a lack of health care personnel (doctors and nurses) in the Project area of influence. Specifically, Mandal Soum has a significant shortage of doctors, but a relatively better supply of nurses, while Bayangol Soum lags behind both World Health Organization (WHO) indicators for doctors and the national average for nurses. Key problems households are experiencing with health services include the poor quality of health care; no time to go to the hospital; and, that hospitals are too far away. Infectious diseases have increased in the Project area since 2014, the most common of which are Sexually Transmitted Infections.

Indicative traffic surveying shows most vehicles travelling along the public Gatsuurt haul road are light vehicles. The main causes of traffic accidents in the Project area are related to risky driving practices. Moreover, police statistics indicate that the major crimes in Selenge Aimag are related to traffic safety and the use of motor vehicles. Just under half of crimes registered in Mandal Soum are crimes related to human life and health, while a third of all crime in Bayangol Soum is related to traffic safety and the use of motor vehicles, which may be explained by the location of the soum along the international, paved road between Ulaanbaatar and the Russian border. Most households surveyed at the soum, bagh and household levels rate local crime-fighting capacity as “average and insufficient”. Child labour was also reported by over 30% of respondents as a key issue in Bayangol Soum, compared to slightly over 20% in Mandal Soum. Child labour most often takes the forms of herding, artisanal mining or working for forestry companies.

CULTURAL HERITAGE

Worship of Noyon Uul (mountain), located immediately north of the Project area is a traditional heritage practice near the Project area. According to local households, worship of the mountain occurred before the socialist era, but was then forbidden during communist rule (1920s to 1990s). The idea of worshipping the mountain once again was initiated during the transition period in the 1990s. The Ethnographic Report of the Intangible Cultural Heritage in the Region of Gatsuurt Mining Area (conducted in 2015 by The Institute of History and Archaeology, Mongolian Academy of Sciences as part of the Socio-economic Baseline Studies) found that worship rituals performed at Noyon Uul are flexible, and have been performed at various places near and on the mountain. Shamans also use the area to worship several times a year.

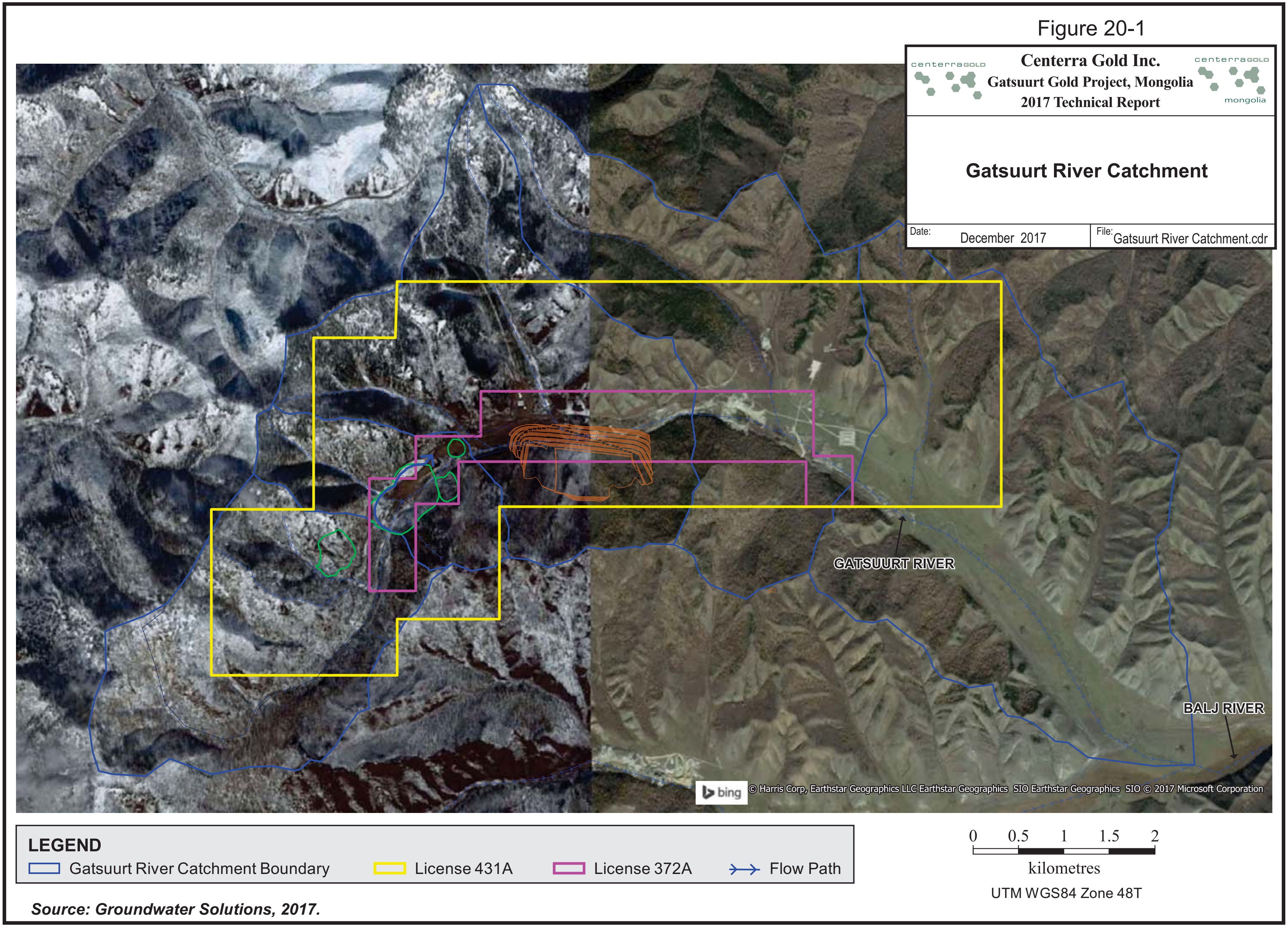
The nearest artefact and heritage sites to the Gatsuurt Mine Licence Area are mostly centred in two Locally Protected Areas: Zuun Modnii Gol and Noyon Uul sites which contain the Sujikhtai Hunnu Graves site. A site of national importance, the Sujikhtai Hunnu Graves site is located seven kilometres from the Gatsuurt project area and outside any potential influence from blasting or vibration impacts. The Gatsuurt road, which existed as a public road prior to the Project being developed, passes through two Locally Protected Areas – Zuun Modnii Gol and Noyon Uul (Sustainability East Asia, 2016).

Five tangible heritage sites were recorded in a recent (2016a) archaeological study conducted by the Institute of History and Archaeology in the Gatsuurt Mine Licence Area. A further salvage archeological excavation was conducted by the Institute (2016b) with the purposes to investigate, excavate, record and salvage archaeological sites in and around the Gatsuurt Mine area. A total of 18 sites were investigated, five of these inside the mine licence area (Figure 20-1). Although many of the sites showed signs of disturbance due to looting, many artefacts were recovered, including pottery, wood, fabrics, metal and stone tools, and human remains. The age of the sites ranges from the Bronze Age to the 18th Century AD.

Heritage studies were commisioned by CGM to ensure compliance with the applicable Mongolian regulations, including the Constitution of Mongolia and the Procedure for the Research on Cultural Heritages of Mongolia.

In the words of the Studies' authors it was concluded that "…the salvage archaeological excavation has been successfully accomplished in accordance with the stipulated plan in the contract made upon the Law on the protection of cultural heritages and other associated laws and procedures. We hereby inform that henceforth, the company is free to start its mining exploitation at Gatsuurt deposit. However, the company should always be aware of buried chance finds and operate cautiously, and we believe that the company agents will report directly to the professional organization if any chance finds have been found during operation" (Institute of History and Archeology, 2016b).

Figure 20-1



Source: Groundwater Solutions, 2017.

ENVIRONMENTAL AND SOCIAL MANAGEMENT PLANS

Management of significant Project environmental and social aspects and impacts is through a suite of Management Plans that will be implemented prior to construction activities. The following Environmental and Social Management Plans have been developed:

ENVIRONMENTAL

- Biodiversity Management Plan
- Water Management Plan
- Acid Rock Drainage Management Plan
- Hazardous Materials Management Plan
- Traffic Management Plan

SOCIAL

- Stakeholder Engagement and Grievance Management
- Social Management Plan

MINE CLOSURE

MINE CLOSURE REQUIREMENTS

Because some of the data required to prepare a detailed closure plan will be collected during operations, the current closure approach remains conceptual. While not detailed, the closure concepts contained herein are based on available site data and good practice in mine closure. The assumptions used to develop the current closure concepts will require further investigation during operations.

The Project is assumed to achieve the closure objective in approximately eight years. This will include three years of active closure construction and reclamation activities, followed by a minimum of five years of water treatment for contact water from the dumps, and post-closure monitoring.

The closure objectives include the following best practices:

- Develop a post-mining landform that is stable, where vegetation approximates the existing surroundings over the intermediate and long-term and soil loss is comparable;

* Route the non-contact surface water and river flows around the planned pits and dumps;
* Reestablish a natural river channel below the project;
* Reduce the flow of contact water from the waste rock dumps;
* Reduce the potential for acid generation in the waste rock dumps; and

Potential impacts to water resources represent the most significant risk at closure. At closure, sources of potentially contaminated water at the Gatsuurt site include the open pits, waste dumps, and low grade stockpile (Sustainability, 2017). Detailed closure activities by major facility are detailed below.

PRE-MINING ACTIVITIES

Prior to development, growth media (GM) will be stripped and stockpiled for use in future use. GM stockpile locations will be strategically placed to avoid waste rock dumps, pits, diversion channels and ancillary facilities supporting future actives. The Gatsuurt River will be diverted into a temporary channel to control and maintain unimpacted water during operations and allow for construction of a contact water and sediment basin reservoir. Road access will be developed, as needed, to support mining operations. GM will be stripped from road footprints and stockpiled.

Prior to the proposed development, alluvial gravels in the river channel were mined and left unreclaimed. These areas fall within and outside of the proposed mining footprint, but will be reclaimed during closure. Restoration of the river channel is discussed in more detail below.

CLOSURE SCHEDULE

Reclamation activities are assumed to last eight years, including three years of active reclamation and five years of post-closure water management and monitoring (Table 20-2). Water treatment could potentially extend the timeline beyond five years, but treatment requirements are unknown at this time. Pits will be reclaimed first, followed by waste rock dumps, water management structures, and finally roads, camps, and yard areas. In addition to the site reclamation, a tailings facility at the Boroo Mine containing processed ore originating from Gatsuurt will be closed. Closure of the tailing facility includes construction of an evapotranspiration pond and treating supernatant water. Water treatment at Boroo is estimated to take ten years.

TABLE 20-2 CLOSURE SCHEDULE

Schedule of Operations	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Install safety berms and fencing											
Regrade dumps											
Cover and revegetate dumps											
River restoration											
Demo buildings and reclaim yard areas											
Reclaim non-critical roadways											
Decommission river diversion (regrade/reveg.)											
Decommission pond (regrade/reveg.)											
Decommission seepage collection system											
Water treatment and monitoring *											
Boroo - evapotranspiration pond construction											
Boroo - evapotranspiration pond reclamation											
Boroo - tailings water management											

CLOSURE PLAN

Figure 20-2 represents the post-mining topography of the Gatsuurt site and outlines areas of the river channel disturbed prior to mining activity, probable water management features, and the major mining facilities, as discussed below.

PITS

Reclamation of the pits will be limited to perimeter protection and flooding at closure. Pumps will be removed from the pit sumps allowing the pits to fill over time. An alternate method suggested in the Water Management Plan (Sustainability 2017), would be to temporarily divert the Gatsuurt River into the pits to reduce flooding time. The flooding method is dependent upon future geotechnical and geochemical assessment. Access roads into the pits will be blocked and a berm, with fencing, will be constructed around the pit perimeters to discourage access and warn of danger. No other reclamation activities are planned for pit closure.

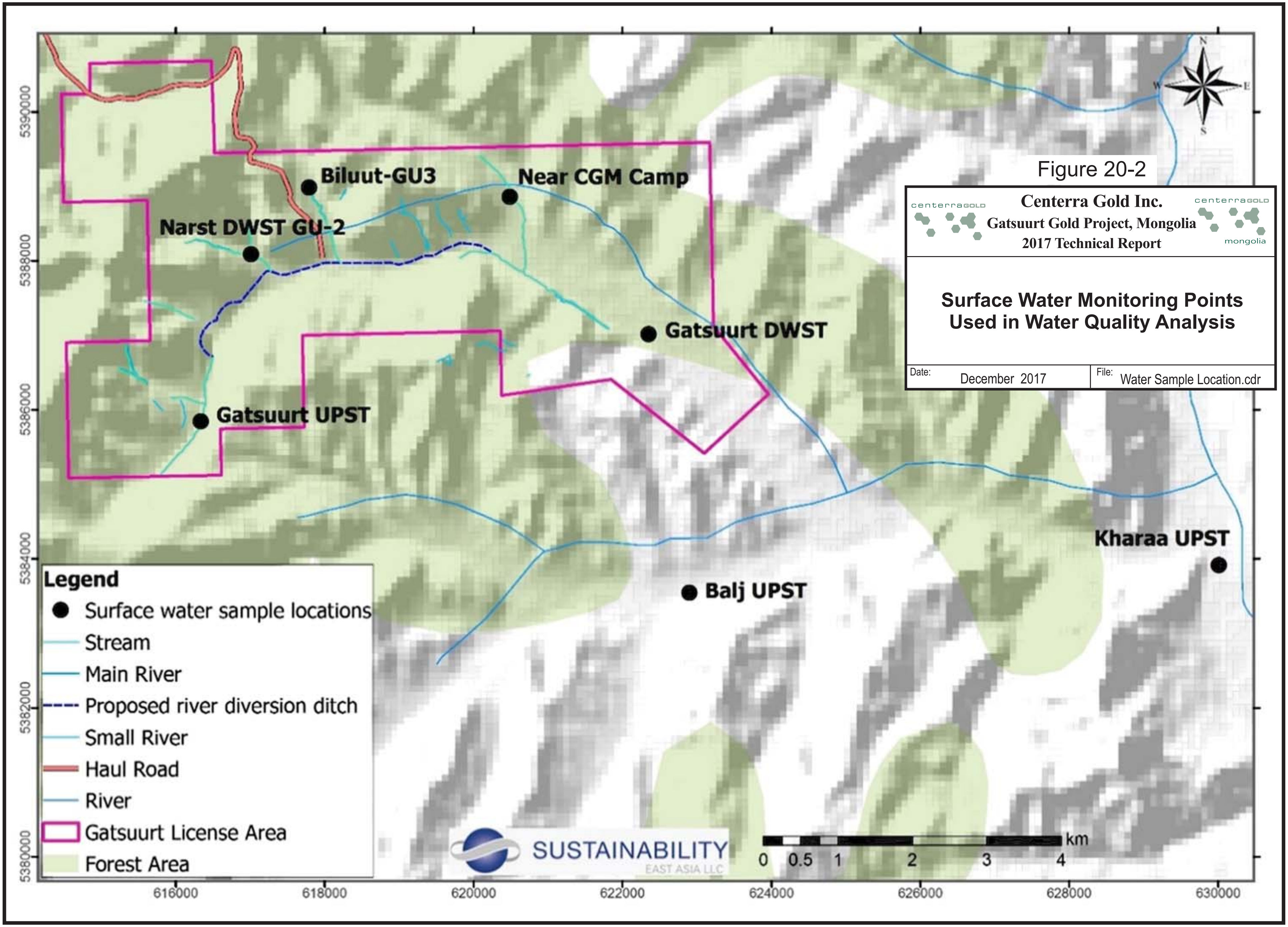


Figure 20-2
Centerra Gold Inc.
Gatsuurt Gold Project, Mongolia
2017 Technical Report
centerragold
mongolia
Surface Water Monitoring Points Used in Water Quality Analysis
Date: December 2017
File: Water Sample Location.cdr
Biluut-GU3
Near CGM Camp
Narst DWST GU-2
Gatsuurt DWST
Gatsuurt UPST
Kharaa UPST
Balj UPST
Legend
Surface water sample locations
Stream
Main River
Proposed river diversion ditch
Small River
Haul Road
River
Gatsuurt License Area
Forest Area
SUSTAINABILITY
EAST ASIA LLC
0 0.5 1 2 3 4 km
616000
618000
620000
622000
624000
626000
628000
630000
5390000
5388000
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E

WASTE ROCK DUMPS

Waste rock will be deposited in one of four dumps located in close proximity to the pits including two large facilities (North and East Dumps) and two smaller facilities (South and North-West Dumps). The waste rock will be deposited in valleys adjacent to the Gatsuurt River. Waste rock within the dumps will be a mixture of both PAG and NAG material.

The PAG dumps represent a primary source for potential contaminant transport from the site. The presence of oxygen and precipitation in contact with PAG waste will produce acidic conditions that will mobilize metals and metalloids. The Geochemistry Report (Schafer 2017) has identified a strategy to reduce the potential for acid formation within the closed dump. Schafer (2017) recommends a 5 m shell to encapsulate the comingled PAG/NAG, and capped with 0.5 m layer of soil on all dump surfaces. RPM has estimated that sufficient NAG material is available to provide between 7 m to 13 m layer of NAG material on the slopes and top surfaces of all dumps. This cover will have the effect of limiting the water infiltration into the dump. Industry experience indicates cover effectiveness can vary, with reduction in infiltration generally on the order of 80 percent of mean annual precipitation, dependent on climate. Development and refinement of the cover design during operations will be critical to limiting outflow from the PAG facilities. The closure cost estimate assumes the conservative approach of placement of NAG cover on all dump faces.

The current dump configuration will allow for all dump slopes to be regraded to a final slope of approximately 2.5:1 (horizontal to vertical). The top surfaces should be graded to route stormwater off the dumps as quickly as possible in order to limit infiltration into the dumps (Schafer, 2017).

The ore on the ROM stockpiles represents an asset and will be removed prior to closure for processing. The remaining ROM stockpile foundations do not represent a significant closure liability and will be regraded and closed in place.

All waste rock dumps will be capped with 200-500 mm of GM and revegetated. Yard areas and ROM stockpiles will be regraded to promote drainage, scarified, and capped with at least 150 mm of GM, and revegetated.

BOROO SUPPORT FACILITIES

Two personnel camps, an administration and workshop building, and supporting ancillary structures will support operations. Prior closure plans prepared by Centerra suggested that the local government may desire ownership of some or all of the structures on site. The closure estimate assumes that 50% of the camp structures will be removed, and 50% will be left in place or salvaged. All yard areas and connecting access roads will be reclaimed. Reclamation of the facilities does not include any salvage and assumes all structures have concrete foundations. The construction of the camp buildings is unknown at this time but may consist of lightly constructed mobile buildings which would likely be salvaged at closure.

WATER MANAGEMENT

Water management during operations will consist of maintaining diversion around dumps and pits, routing the Gatsuurt River in a diversion channel to avoid mining operations, including through the Central Pit, and collection of impacted water in a pond located downstream of the East Dump.

The existing Gatsuurt River channel was disturbed by previous alluvial (placer) mining activity. Provisions have been made to remove and regrade portions of these prior disturbances. By removing these significant features and developing an initial river channel that minimizes initial stream energy, the river will be allowed to meander and reform banks and a sinuous stream bed. This approach is discussed in further detail in the Site Wide Water Management Plan (Sustainability, 2017).

During operations, diversion channels will be constructed to route non-contact water past the various contact sources associated with the mine. This will include a pit-bench channel and a number of channels routing water primarily around pits and dumps. At closure, this routing system will be modified to provide a stable and long-term diversion around the closed facilities. The dumps will be designed and regraded to facilitate long-term stability of stormwater routes and to prevent the streams or river from destabilizing closed facilities. At closure, the river may be temporarily diverted into pits to speed inundation.

Gatsuurt water quality predictions and water treatment requirements at closure are uncertain at this time. Geochemistry of the waste rock material indicates that the majority is NAG resulting in leachate runoff being potentially alkaline in nature (Schafer 2017). During operations impacted water will be collected from waste rock dumps, run of mine ore piles, and pits. The impacted water during operations will be treated using MetClean® water treatment technology.

Schafer (2017) recommends placing 5 m of inert waste rock cover on all waste rock dump surfaces, followed by 500 mm of GM and revegetation. Implementation of this design should result in a low risk impact to surface and groundwater quality.

Water quality prediction of pit water at closure is uncertain, but may pose a high risk of adversely affecting surface and groundwater quality (Schafer 2017). Closure methods for the pits include flooding, either by natural seepage, or diverting the Gatsuurt River into the pits. The resulting water quality prediction is unknown, but arsenic species may be present (Sustainability 2017).

Seepage from waste rock dumps will likely continue for some time after regrading and revegetation, until the effectiveness of the cover design is realized. Collection and treatment is assumed for five years after reclamation activities are complete.

The closure estimate assumes water treatment for the full eight years of closure and monitoring. Costs include maintenance and operation of the water treatment plant and consumables. These costs include treatment of seepage from waste rock dumps and potential contaminated water in-pit. The actual seepage volume at closure is unknown, but is expected to be less than operational flows. Assuming full treatment costs during the eight year closure period is likely conservative, however, treatment costs beyond eight years is unknown at this time.

BOROO TAILINGS FACILITY

Ore from the Gatsuurt Project will be trucked to the Boroo mill facility, approximately 52 km away. The ore will be processed and placed in both the existing tailings facility and a new tailings cell developed for the Gatsuurt ore.

The closure plan at the Boroo site includes managing and recirculating supernatant and drainage water, and disposal of water via evaporation over a 10-year period. The plan does not consider closure of the tailings facility itself because this cost is included as part of the Boroo Mine closure plan.

The closure of the Boroo TSF cell will require construction of an evapotranspiration (ET) cell to collect and manage drainage from the tailings facility. This may be achieved by enhanced evaporation. Alternately, the water may be treated and discharged.

Once the TSF supernatant and initial drainage are removed, a two-metre rock cover will be placed on the surface. This will allow mobile equipment onto the facility and provide a foundation for placement of a soil cover between 0.5 and 1.0 m in thickness. This cover will minimize long-term infiltration into the facility.

Long-term drainage will be managed through both active and passive evaporation. Introduction of the remaining fluids into the downstream surface, or groundwater, may be possible, based on the chemistry and quantity of flow.

Water quality predictions of supernatant water from the tailings is unknown at this time (Schafer 2017), but is expected to be a low risk. A detailed estimate for management of the supernatant water has not been prepared at this time.

ESTIMATED CLOSURE COSTS

Closure costs have been estimated at $15.4 million for the Project and will be incurred after mining is complete.

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Final closure costs are expected to change once further information becomes available and mine plans are refined.

21 CAPITAL AND OPERATING COSTS

CAPITAL COSTS

ASSUMPTIONS

The following material assumptions have been used in the LOM plans, estimates of operating and capital costs, and Mineral Reserve and Mineral Resource estimates:

- A gold price of US$1,250 per ounce
- An exchange rate of US$1: MNT ₮2,200
- Electricity cost of US$0.10 per kilowatt-hour
- Diesel fuel cost of US$0.83 per litre

The following is excluded from the capital cost estimate:

- Project financing and interest charges
- Escalation during construction
- Permits, fees, and process royalties
- Environmental impact studies
- Additional civil, concrete work due to any adverse soil conditions and location
- Cost of geotechnical investigation for new facilities
- Working capital
- Sunk costs
- Pilot Plant and other testwork
- Exploration drilling
- Costs of fluctuations in currency exchanges
- Project application and approval expenses
- Future expansion
- Relocation of any facilities, if required

PRE-PRODUCTION CAPITAL

The Project consists of two phases: refurbishment of the existing mill at the depleted Boroo Mine to process leachable ore from the Gatsuurt pit, and construction of a BIOX® plant to process the sulphide ore. Pre-production costs for the first phase have been assumed to occur in Year 0 and Year 1, and pre-production costs for the second phase have been assumed to occur in Years 2 to 4, with mining for the first phase commencing in Year 1 and for the second phase commencing in Year 4. Total initial direct and indirect capital costs and pre-production operating costs are estimated to be $244.9 million, excluding the closure deposit. This total includes all applicable Owner's costs, Boroo refurbishment, new sulphide process circuit (Flotation, BIOX®, and CIP plant), infrastructure, engineering, mining, and project management costs, as well as taxes, with a contingency of 10% upon commencement of construction. The estimates are found in Table 21-1.

TABLE 21-1 PRE-PRODUCTION EXPENDITURES SUMMARY

Item	Total ($ x 1000)
Mining and Mine Construction	2,815
Refurbishment of Boroo Mill	7,728
Upgrade of Haul Road	14,000
Utilities (Powerlines and Substations)	11,501
Tailings Management Facility	21,342
Water Diversion and Treatment	9,981
Sulphide Circuit (Flotation, BIOX® and CIP)	95,397
Owner's Costs	27,464
Contingency (10%)	26,989
Taxes	**27,639**
TOTAL	**244,856**

Table 21-2 shows pre-production expenditures separated into direct and indirect costs.

TABLE 21-2 PRE-PRODUCTION EXPENDITURES

Item	Total ($ x 1000)
Direct Costs	
Mining and Mine Construction	2,815
Utilities	11,305
Infrastructure	22,801
Processing including BIOX® and CIL	67,130
Tailings Management Facility	21,342

Item	Total ($ x 1000)
Total Direct Costs	**125,394**
Indirect Costs	
Owner's Costs	26,732
Construction/Project Management	38,103
Contingency (10%)	26,989
Total Indirect Costs	**91,826**
Taxes	**27,639**
TOTAL	**244,856**

DIRECT COSTS

MINING AND MINE CONSTRUCTION COSTS

The capital costs associated with mining and mine construction include mining costs through to production. These costs are further detailed in Table 21-3.

TABLE 21-3 MINING AND MINE CONSTRUCTION CAPITAL COST

Item	Total ($ x 1000)
Mining Haul Roads	2,710
Pit De-watering	105
TOTAL	**2,815**

UTILITIES AND INFRASTRUCTURE CAPITAL COSTS

The utilities portion of the capital cost includes all of the capital required to deliver electrical power to the site, including upgrading the utility substations at Darkhan and Zuunkharaa, the main substations at Boroo and Gatsuurt, as well as 90 km of 110 kV powerline from Darkhan to the Boroo substation, and 40 km of powerline from Zuunkharaa to the Gatsuurt substation, and modifications to the existing overhead line at Boroo.

The infrastructure portion of the capital cost includes upgrades to the existing 52 km haul road between Boroo and Gatsuurt to suit the required volume of haul truck traffic, and to pave the section with the highest amount of public traffic. Additionally, water diversion structures are required to collect contact water and divert this to a settling pond and associated water treatment plant.

These costs are further detailed in Table 21-4.

TABLE 21-4 UTILITIES AND INFRASTRUCTURE CAPITAL COST

Item	**Total ($ x 1000)**
Utilities	
Site Substations	3,021
Site Distribution	399
Darkhan Tie and Power Line	5,930
Zuunkharaa Tie and Power Line	1,955
Subtotal Utilities	**11,305**
Infrastructure	
Haul Road Upgrade	12,935
Water Treatment Plant	3,819
Diversion Channels	3,881
Settling Pond	2,166
Subtotal Infrastructure	**22,801**
TOTAL	**34,106**

PROCESSING AND TAILINGS STORAGE

The processing portion of the capital cost includes refurbishment of the existing Boroo mill, as well as the new sulphide circuit. The processing cost summary is shown in Table 21-5.

TABLE 21-5 PROCESSING CAPITAL COST SUMMARY

Item	**Total ($** x 1000)
Boroo Refurbishment	5,928
Sulphide Circuit	61,202
TOTAL	**67,130**

The tailings storage portion of the capital cost includes a lift to the existing Main cell and a new tailings impoundment for the PAG tailings produced by the BIOX® plant. The tailings storage cost summary is shown in Table 21-6.

TABLE 21-6 TAILINGS STORAGE FACILITY COST SUMMARY

Item	Total ($ x 1000)
NAG Tailings Impoundment	9,461
PAG Tailings Impoundment	11,881
TOTAL	**21,342**

INDIRECT COSTS

OWNER'S COSTS

The Owner's cost portion of the capital cost primarily includes the operation of the Ulaanbaatar and Gatsuurt offices for the duration of the construction phase of the Project.

There is an allowance in this section for construction and pre-commissioning insurance, as well as for light vehicles, including the fire truck and emergency rescue vehicle. The Owner's capital cost summary is shown in Table 21-7.

Permitting fees are ongoing and will be accounted for the in the Operating Expenses for the remainder of the mine life.

The capital set aside for land acquisition is an estimate based on the amount of private land that may need to be acquired, and is based on current market rates.

TABLE 21-7 OWNER'S CAPITAL COST SUMMARY

Item	Total ($ x 1000)
Ulaanbaatar Office G&A	12,540
Gatsuurt Project Office	4,579
Accommodation	3,028
Vehicles and Mobile Equipment	2,712
Permitting/Land Acquisition	1,037
Other Costs	2,836
TOTAL	**26,732**

CONSTRUCTION AND PROJECT MANAGEMENT

The construction and project management portion of the capital cost includes all of the capital costs associated with engineering and managing the construction and commissioning of the Project. The engineering component is based on proposals from engineering, procurement, and construction (EPC) firms and engineering, procurement, construction, and management (EPCM) firms for detailed engineering. The construction and project management capital cost summary is shown in Table 21-8.

TABLE 21-8 CONSTRUCTION MANAGEMENT CAPITAL COST SUMMARY

Item	Total ($ x 1000)
Detailed Engineering	17,473
Construction Management/Commissioning	10,539
Vendors' Representation	488
Temporary Facilities	2,609
Freight	4,278
Construction/Critical Spares	1,833
First Fills	883
TOTAL	**38,103**

CONTINGENCY

Contingency is 10% per RPMGlobal's recommendations for feasibility studies (RPMGlobal Perspectives, Issue No.128 – June 2015, Minimum Engineering Study Requirements Update). Centerra has conducted an internal risk review and determined 10% contingency to be suitable.

TAXES

Taxes considered in the capital cost include VAT, withholding tax on services such as engineering and in-Mongolia training of personnel performed by non-Mongolian entities, custom tax on imported equipment and materials, as well as the fee payable to the Fund for Development of Construction Codes and Regulations.

SUSTAINING CAPITAL

Sustaining capital requirements for the Project are minimal, primarily due to the short life of mine, as well as contracting out of the mining tasks, obviating the need for allocating sustaining capital for mobile mining equipment. The sustaining capital estimate is found in Table 21-9.

TABLE 21-9 PROJECT SUSTAINING CAPITAL COST SUMMARY

Item	Total ($ x 1000)
Tailings Storage Facility	32,831
Haul Road	3,810
TOTAL	**36,641**

WORKING CAPITAL

Working capital is excluded from the capital cost.

CLOSURE COST

Closure costs have been estimated at $15.4 million for the Project. This value has been based on mine closure industry experience in Mongolia. Prior to construction, a more detailed Conceptual Closure Plan (CCP) will be developed that will further expand on the aforementioned closure concepts, and use a systematic approach for more accurately estimating the closure costs such as Standardized Reclamation Cost Estimator (SRCE).

OPERATING COSTS

The LOM operating cost estimates on a per tonne processed basis are: mining $10.97, ore haulage $5.37, oxide processing $11.90, sulphide processing $26.42, and general and administration $8.92 for a total operating cost (excluding royalties) of $47.09 per tonne processed (Table 21-10).

TABLE 21-10 LIFE OF MINE OPERATING COST SUMMARY

Cost Centre	Unit Value $/t Total Material	Unit Value $/t Ore	Value $M
Mining Costs			
Contract Mining	1.62	9.26	142
Overheads	0.22	1.25	19
Water Management	0.08	0.46	7
Total Mining	**1.92**	**10.97**	**168**
Ore Haulage	**0.94**	**5.37**	**82**
Total Mining and Haulage	**2.85**	**16.34**	**251**
Processing Costs			
Oxide Processing	0.66	11.90	58
Sulphide Processing	3.15	26.42	277
Total Processing	**3.81**	**21.83**	**335**
General and Administration	**1.56**	**8.92**	**137**
Total Operating Cost	**8.22**	**47.09**	**723**

MINING COSTS

An annual mining cost model for the Project has been prepared using an economic modelling software package and on a constant 2017 US dollar (US$) basis. Additional items in the model include provision for items such as exchange rate, labour costs, work rosters, and unit prices for certain consumable items, such as diesel and explosives. No allowance has been included for contingency.

Mine operating costs are based on a contract mining scenario. Several local contractors submitted tenders using the mine plan presented herein. These tenders were subsequently adjusted to reflect the Company's view with respect to diesel pricing and exchange rate. The level of detail of the operating costs are commensurate with a feasibility study (+/-15%) as the key mining cost items are supported by third party tender submissions for contract mining and a first principles model to account for sufficient technical staff supervision and management of the mining contractor. While the mining cost estimate is considered by Centerra to be within a +/-15% level of accuracy, the QP recommends that a contingency of 10% be added to the contract mining cost to account for uncertainties associated with the tenders received, although the inclusion of a 10% contingency does not impact the Mineral Reserve estimate.

The mine operating cost also includes Owner's cost for overhead and water management.

The operating cost for ore haulage from Gatsuurt to Boroo is based on a first principles model that considered the haulage route and the proposed road upgrade.

ECONOMIC ASSUMPTIONS

The following outlines the economic assumptions that form the basis of the operating cost estimates:

- All costs in US dollars (US$);
- Exchange rate of 2,200 MNT/USD
- Fuel Price assumed to be $0.83/l
- The time periods in the model are calendar years commencing January 1, Year 1.
- The mining contractor tenders were standardized to common assumptions for exchange rate and fuel price.

Figure 21-1 shows the proportion of the LOM mine cost components, as per the Contractor Model, which are also detailed in Table 21-11. The contract mining rate is the largest cost coming in at 57% of the total cost. This is followed by the ore transport cost which makes up 33%. To account for additional uncertainties that can arise under a contract mining scenario it is recommended that in future studies an additional contingency of 10% be applied to the contact mining cost.

FIGURE 21-1 CONTRACTOR MODEL LOM CASH OPERATING COST PROPORTION

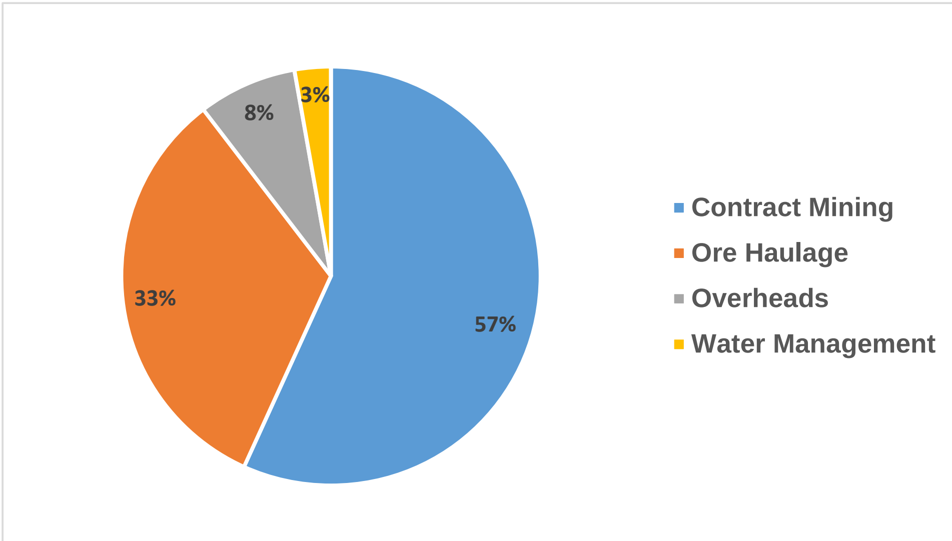
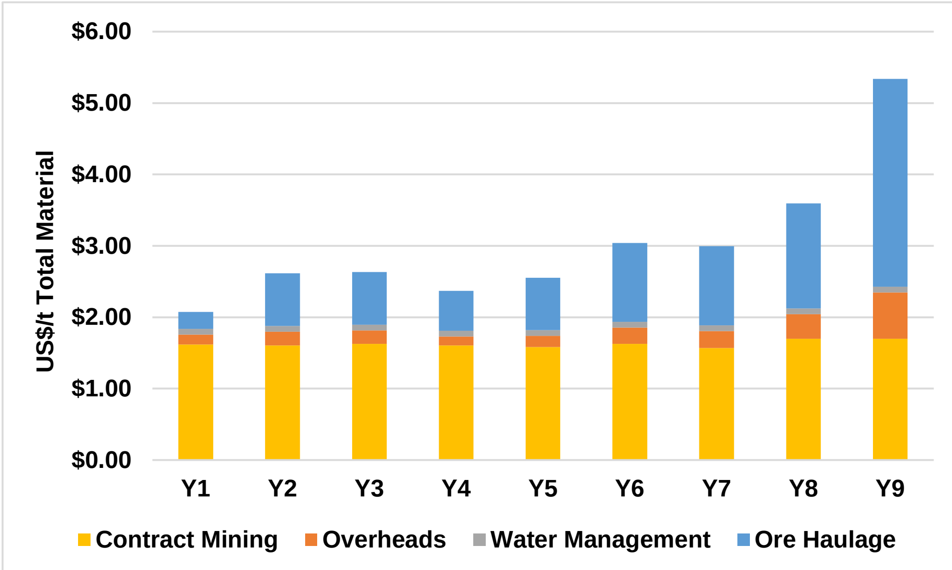


TABLE 21-11 TOTAL MINE OPERATING COST SUMMARY

Cost Centre	Unit Value $/t Total Material	Unit Value $/t Ore	Value $M
Contract Mining	1.62	9.26	142
Ore Haulage	0.94	5.37	82
Overheads	0.22	1.25	19
Water Management	0.08	0.46	7
Total (excl. Haulage)	1.92	10.97	168
Total Mine Operating Cost	**2.85**	**16.34**	**251**

Figure 21-2 shows the cash operating costs per year for the Contractor Model. The contract mining costs are relatively constant throughout the mine life. The ore haulage cost to Boroo increases in the later years when expressed on a $/t total material.

FIGURE 21-2 CONTRACTOR MODEL CASH OPERATING COST



CONTRACTOR MINE OPERATING COST

Tender submissions to complete mining activities at the Gatsuurt Mine were received in September 2017 from two independent mining contractors based in Mongolia. The tender submissions were based on the mine plan provided herein including the mine quantity schedule and other relevant details of the Project.

The tender submissions were standardized to ensure that the exchange rate and fuel prices across the tenders were consistent with the operating cost assumptions. Table 21-12 summarizes the received contractor mining rates.

TABLE 21-12 AVERAGE CONTRACTOR MINING RATES SUMMARY

Submission	Mining Rate ($/t Total Material)
Contractor 1	1.19
Contractor 2	1.46
Adjusted for Model	1.54

There were insufficient details in both of the tender submissions to allow detailed analysis and confirmation of the rates. A review of the mining equipment fleet provided by Contractor

1 suggests there is insufficient equipment in their submission to achieve the required production.

The selected rates for this report are based on the tender submission of Contractor 2. This submission did not break the costs down into cost centres or cost elements. The nominated percentages in the rise and fall formula in the mining contractor quotation were used to break the annual rates down and adjust the fuel price components.

The scope of the tender was for all mine activities including drilling and blasting, waste removal, and delivery of ore to the designated stockpiles. A separate contract is to be awarded for transportation of ore from Gatsuurt to Boroo. According to the tender form the rates include:

- All equipment mobilization and demobilization charges.
- Loading and hauling of all ore and waste according to the LOM Schedule.
- Drill and blast of ore and waste, as required, based upon the blast design parameters provided, including pre splitting of final wall limits at spacing provided.
- All support required to safely carry out the works relating to maintenance of mine haul roads, waste dumps, and stockpile areas.
- All site infrastructure, mobilization, and establishment costs including camp facilities, mine offices, and workshop equipment at Gatsuurt.
- Pit dewatering, as required, to maintain safe and efficient mine operations and grade control.

As pit dewatering is covered under the Owner's cost, the equivalent rate ($0.08/t) was deducted from the contractor rates to avoid double counting. The resultant schedule of adjusted rates is shown in Table 21-13 which includes the pit dewatering deduction and the standardization of the fuel cost to the Contractor 2 base rates.

TABLE 21-13 SELECTED CONTRACTOR UNIT RATE SUMMARY

Year	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Unit Cost ($/t)	1.54	1.53	1.55	1.53	1.51	1.55	1.50	1.62	1.62

The adjusted tender submission unit rates form the basis of the mine operating cost estimate and exclude costs related to Company supervision and management. The contract rate is applied annually to the mine quantity schedule to estimate the annual mining costs. In addition to the contractor rates there are a number of mine owner costs including the ore haulage costs.

The mining contractors have their own equipment database based on preferred equipment suppliers. The contractor equipment costs include an ownership cost which is the mechanism by which the contractor recovers the mining equipment capital cost.

Drill and blast consumable costs are included on a $/t unit rate basis in the mining contractor estimates.

OWNER'S OVERHEAD COSTS

The Owner's overhead costs include labour costs based on actual planned salaries. Rates for Centerra staff, entered into the model, include allowances for:

- Employee Social Insurance,
- Employer Social Insurance, and
- Personal Income Tax.

Staff salaries are variable depending on the role and level of experience of the employee. Operating labour and maintenance positions had a fixed annual salary applied.

The mining cost assumes a Contractor Model and as such, the operating and maintenance labour costs are included in the contractor rates provided. The total mine staff required is 33 in Year 1 under this scenario (Table 21-14).

TABLE 21-14 MINE STAFFING LIST

Position	Number of Personnel
Mine Administrator	1
Mine Superintendent	1
Mine Foreman	1
Mine Engineer	1
Mine Surveyor	1
Mine Surveyor Helper	1
Senior Geologist	1

Position	Number of Personnel
Geologist	2
Geology Assistant	1
Environmental Engineer	1
Environmental Officer	1
Environmental Coordinator	1
Field Trainer Operator	1
Safety Superintendent	1
Safety Officer	1
Trainer	1
Maintenance Superintendent	1
Maintenance Administrator	1
Electrical Engineer	1
Electrician	1
Tradesperson	1
Mine Maintenance Foreman	1
Scheduler	1
FF Attendant	1
Light Vehicle Mechanic	1
Dispatcher	1
Warehouse Superintendent	1
Warehouse Person	1
Warehouse Data Entry Clerk	1
Crane Operator	1
Dewatering Operator	1
Driver	1
Total	**33**

The following operating costs have been applied in the model on a unit rate basis. The costs are based on a cost database and experience with similar projects in the region:

- Grade Control - $0.25/t Ore. Grade control will be carried out by Centerra during operations.

- Water Management - $0.08/t Total Material. Water management will be completed by Centerra. The cost is based on benchmark costs from similar projects.

- G&A - $0.50/t Ore. The mining G&A covers cost items including IT, software licencing, office consumables, communications, consultants, and technical advisors. The Centerra staff are costed elsewhere in the model.

ORE TRANSPORT

Ore transportation costs were developed based on a first principles approach that included fleet estimation and costing. The ore transport fleet includes haul trucks and support equipment including graders, water trucks, and small front end loaders. The average LOM haulage cost has been estimated at $5.36/t ore hauled.

PROCESSING COSTS

BACKGROUND

Processing costs were developed based on the Bantrel and Outotec design criteria (Bantrel, 2009) and historical operating costs from the Boroo operation. Consumable and reagent costs for flotation, BIOX®, neutralization and BIOX® product leaching were based on the design criteria and current price quotations. Additional operations and maintenance personnel have been incorporated for the flotation, BIOX®, and neutralization areas. Gold recovery includes leach, CIP, carbon elution and regeneration, and gold room costs.

PROCESS LABOUR

The process staffing roster was developed from the previous Boroo operation. The staffing roster for processing Gatsuurt oxide and sulphide ores is shown in Table 21-15. Processing of sulphide ore with the BIOX® process will require additional staffing. This includes several expatriate positions as well as additional operation and maintenance personnel.

TABLE 21-15 PROCESS STAFFING LIST

Position	Status	Oxide	Sulphide
Supervision			
Director of Processing	National	1	1
Administrative Assistant	National	2	2
Technical Services			
Chief Metallurgist	Expatriate	0	1
Senior Metallurgist	National	1	1
Metallurgist	National	2	2
Metallurgical Technician	National	2	2
Chemist	National	1	1
Mill Operations			
Processing General Foreman	Expatriate	0	1
Mill Clerk	National	0	2
Senior Foreman	National	2	2
Shift Foreman	National	4	4
Senior Operator	National	4	4
Crusher / Reclaim Operator	National	4	4
Grinding Operator	National	4	4
Flotation Operator	National	0	4
BIOX® Operator	National	0	4
Leach/CIP Operator	National	4	4
Carbon Stripping Operator	National	2	2
Detox Operator	National	4	4
Gold Room Operator	National	2	2
Reagent Mixer	National	4	6
Mobile Equipment Operator	National	2	2
Day Crew Labourer	National	0	10
Mill Maintenance			
Maintenance General Foreman	Expatriate	0	1
Maintenance Clerk	National	2	2
Planning Engineer	National	2	2
Electrical Foreman	Expatriate	0	1
Electrical Foreman	National	1	1

Position	Status	Oxide	Sulphide
Electrical Engineer	National	2	2
Electrician/Instrument Technician	National	12	12
Mechanical Foreman	National	1	1
Mechanical Engineer	National	2	2
Millwright	National	10	12
Building Maintenance	National	2	2
Welder	National	8	8
Pipefitter	National	2	2
Driver / Hoist Operator	National	2	2
Lubeman	National	2	2
Toolcrib Attendant	National	2	2
Senior Engineer	National	1	1
Engineer Trainer	National	2	2
Total Personnel		**98**	**126**

PROCESS COSTS DETAILS

Costs for processing oxide ore are projected at $11.90/t milled and for processing sulphide ore are $26.42/t milled. Costs include VAT and applicable import duties. The annual and unit operating cost detail for the processing of oxide and sulphide ore is summarized in Table 21-16.

TABLE 21-16 PROCESS OPERATING COST SUMMARY

Operating Cost Area	Oxide		Sulphide	
	Annual $ (000)	$/t	Annual $ (000	$/t
Personnel				
Supervision/Technical	137		137	
Mill Operations	801		1,660	
Mill Maintenance	1,126		2,131	
Technical Services	174		690	
Total Personnel	**2,238**	**1.28**	**4,618**	**2.64**
Electrical Power				
Crushing and Grinding	3,714		3,714	
Flotation	-		390	

Operating Cost Area	Oxide		Sulphide	
	Annual $ (000)	$/t	Annual $ (000	$/t
BIOX® and Neutralization	-		1,296	
Leach, CIP, Carbon Elution, Gold Room	485		321	
Reagents	14		361	
General Utilities	562		2,152	
Building Services	74		691	
Total Electrical Power	**4,849**	**2.77**	**8,925**	**5.09**
Water Use Fees	**395**	**0.23**	**589**	**0.34**
Operating Consumables				
Crushing and Grinding	3,963		3,963	
Flotation	-		534	
BIOX® and Neutralization	-		8,952	
Leach/CIP or BIOX® CIL	1,319		5,793	
Elution	138		93	
Cyanide Destruction	1,524		1,598	
Total Operating Consumables	**6,944**	**3.96**	**20,933**	**11.95**
Supplies and Services				
Assay services	1,500		1,875	
BIOX® cooling water treatment			100	
Mechanical/Electrical Existing Boroo	3,990		3,990	
BIOX® building maintenance	-		22	
BIOX® mechanical equipment maintenance	-		2,805	
BIOX® electrical/instrumentation maintenance	-		181	
Miscellaneous operating supplies	932		2,243	
Total Supplies and Services	**6,422**	**3.67**	**11,216**	**6.40**
Total Mill Operating Costs	**20,848**	**11.90**	**46,281**	**26.42**

GENERAL AND ADMINISTRATION COSTS

BACKGROUND

General administration and operations support costs were developed based on Centerra's historic and current practice of managing properties and complying with respective standards, laws, and regulations. A total of $137 million ($13.8 million per year) over the

LOM will be spent for general administration and operations support costs. Costs include VAT and other applicable taxes and fees.

General administration expenses over the LOM are projected at $65 million and include executive and administration expenses for financing and accounting, managing Ulaanbaatar head-office, providing information technology and communication services, as well as maintaining relations with the government and communities.

Operations support costs over the LOM are projected at $72 million and include support for the main operations at the mine sites such as maintaining buildings, camps and roads, providing supply chain and security services, developing human resources, and most importantly, to promoting operating according to environmental standards and safety culture. Table 21-17 presents the administration and operations support cost summary.

TABLE 21-17 GENERAL ADMINISTRATION AND OPERATIONS SUPPORT COST SUMMARY

Area	Annual $ (000)	$/t
Personnel		
General Administration	2,168	
Support Operations	1,966	
Project	92	
Subtotal	**4,226**	**0.28**
Taxes and Fees		
Immovable Property Tax	1,500	
Licence Fee	370	
Land Use Fees	191	
Other Taxes and fees	213	
Subtotal	**2,274**	**0.15**
Other		
Community Relations and Supports (including CDA cost)	1,200	
Camp Catering Contractor	950	
Security Services Contractor	735	
Property & Liability Insurance	800	

Area	Annual $ (000)	$/t
Contractors & Consultants	731	
Office/Space Rental	454	
Audit and Legal Fees	390	
Other expenses	2,062	
Subtotal	**7,322**	**0.47**
Total General Administration and Support Operations Costs	**13,822**	**0.90**
General Administration	8,031	0.52
Support Operations	5,791	0.38

22 ECONOMIC ANALYSIS

The objective of the study was to determine the potential viability of the proposed Gatsuurt open pit gold mine, including modifications to an off-site processing plant to enable the effective treatment of refractory ores hauled from Gatsuurt.

The viability of the Project has been assessed on the basis of a discounted cash flow model, from which net present value (NPV), internal rate of return (IRR), payback, and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

All amounts are in United States dollars ($) unless otherwise stated.

ECONOMIC CRITERIA

REVENUE DRIVERS

- 4,800 tonnes ore per day mining and milling (1.75 million tonnes per year).
- Mill recovery by zone, as indicated by testwork, averaging 78.6% for CIP and 86.5% for BIOX® for an overall gold recovery of 84.4% into doré.
- Metal price: US$1,250 per ounce gold.
- Net revenue includes doré refining, transport, and insurance costs totalling $4.28 per ounce gold.
- Revenue is recognized at the time of production.

COSTS

- Pre-production period: 10 months.
- Mine life capital totals $296.9 million, comprising Construction ($244.9 million, including a contingency of $27 million), Sustaining capital ($36.6 million, excluding changes in working capital) and Mine Closure and Reclamation ($15.4 million in total, or $13.9 million when discounted to the year of mine closure).
- Mine life: 10 years.

- Life of mine production plan as summarized in Table 22-1.
- Exchange rate US dollar to Mongolian tugrik: $1.00 = MNT2,200.

TAXATION AND ROYALTIES

The Project cash flow forecast includes provision for royalties payable as follows:

- Government of Mongolia: 12% of sales revenue (at base case gold price).
- Gatsuurt LLC: 3% of sales revenue (after payment of Government royalty).
- BIOX® royalty of 0.5% of gold sales resulting from use of that technology.

In addition, provision is also made for

- Corporate income taxes payable to the Government of Mongolia at the statutory rates of 10% for taxable income up to 3.0 billion tugriks and 25% for taxable income above this amount.

BASE CASE CASH FLOW

The LOM production plan is presented in Table 22-1 and the LOM cash flow forecast is summarized in Table 22-2.

TABLE 22-1 LIFE OF MINE PRODUCTION PLAN SUMMARY

		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Ore mined	kt	**15,356**	1,631	3,113	2,846	558	1,188	1,005	1,363	2,044	1,607	
Gold grade	Au g/t	**2.7**	2.2	2.4	2.9	2.7	2.5	2.8	3.4	2.7	2.7	
Waste mined	t x 1000	**72,588**	8,597	9,646	9,948	12,202	11,572	7,501	7,167	4,335	1,620	
Total Tonnes mined	t x 1000	**87,944**	10,229	12,759	12,794	12,759	12,759	8,506	8,530	6,380	3,227	
Average Ore mined	kt/d		4.0	8.5	7.8	1.5	3.3	2.7	3.7	5.6	4.4	
Average mined	kt/d		28.0	34.9	35.1	35.0	35.0	23.2	23.4	17.5	8.8	
Stripping Ratio	w/o	**4.7**	5.3	3.1	3.5	21.9	9.7	7.5	5.3	2.1	1.0	
Tonnes Milled	**kt**	**15,356**	**442**	**1,752**	**1,757**	**1,319**	**1,752**	**1,752**	**1,757**	**1,752**	**1,752**	**1,322**
Head grade gold	Au g/t	**2.7**	2.4	2.2	2.2	3.1	3.2	2.7	3.1	2.8	2.6	2.1
Tonnes Milled per Day	kt/d		1.21	4.79	4.81	3.61	4.80	4.79	4.81	4.80	4.80	3.61
Recovery	%	**84.4%**	79.4%	78.4%	79.7%	83.3%	85.5%	85.6%	86.2%	86.9%	88.5%	84.3%
Gold Ounces Produced	**oz x 1000**	**1,111**	**26**	**97**	**99**	**109**	**155**	**131**	**151**	**136**	**130**	**75**

TABLE 22-2 LIFE OF MINE CASH FLOW FORECAST

		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11+
Sales and Revenue													
Gold Price	US$/oz	**1,250**	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Gold Produced (Sold)	oz x 1000	**1,111**	26	97	99	109	155	131	151	136	130	75	-
Gold Sales	**$ million**	**1,386.1**	**33.0**	**121.2**	**124.0**	**136.2**	**193.4**	**164.0**	**188.0**	**170.1**	**162.6**	**93.6**	**0.0**
Operating Costs													
Mining and Haulage(1)	$ million	**250.9**	21.2	33.4	33.7	30.2	32.6	25.9	25.5	22.9	17.2	8.0	0.0
Processing(1)	$ million	**335.3**	5.3	20.8	20.9	28.9	46.3	46.3	46.4	46.3	46.3	27.8	0.0
General and Administration(1)	$ million	**137.0**	8.6	13.8	13.8	13.8	13.8	13.8	13.8	13.8	13.8	11.5	0.0
Royalties(1)	$ million	**208.9**	4.8	17.8	18.2	20.6	29.4	24.9	28.6	25.8	24.7	14.1	0.0
Cash Operating Costs	**$ million**	**932.0**	**39.9**	**85.8**	**86.6**	**93.5**	**122.1**	**110.9**	**114.3**	**108.9**	**102.0**	**61.5**	**0.0**
Capital and Other Costs													
Construction Capital	$ million	**217.9**	17.3	18.6	30.1	91.2	0.6	0.3	0.4	0.3	0.0	0.0	0.0
Contingency	$ million	**27.0**	1.9	2.2	4.3	9.7	0.4	0.3	0.5	0.4	0.0	0.0	1.4
Sustaining Capital(2)	$ million	**36.6**	0.1	3.1	12.4	5.8	3.5	3.1	4.2	3.6	0.5	0.5	0.0
Reclamation & Closure	$ million	**15.4**	1.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	13.8
Capital Expenditure	**$ million**	**296.9**	**20.8**	**23.8**	**46.8**	**106.7**	**4.4**	**3.7**	**5.1**	**4.3**	**0.5**	**0.5**	**15.2**
Working Capital	$ million	**0.2**	2.9	9.3	0.2	2.5	7.6	(2.2)	2.0	(1.4)	(0.4)	(20.2)	0.0
Discount Post Closure Cost	$ million	**(1.6)**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	(1.6)
Constr.+Contingency	*$ million*	***244.9***	*19.2*	*20.7*	*34.3*	*100.9*	*1.0*	*0.6*	*0.8*	*0.7*	*0.0*	*0.0*	*1.4*
Cashflow													
Pre-tax cashflow	$ million	**158.6**	(30.6)	2.2	(9.6)	(66.4)	59.4	51.6	66.6	58.3	60.5	51.9	(13.6)
Income Tax Payable	$ million	**43.5**	0.0	0.7	5.0	5.0	8.5	4.4	8.6	5.6	5.5	0.2	0.0
Free Cash Flow (FCF)(2)	**$ million**	**115.1**	**(30.6)**	**1.5**	**(14.6)**	**(71.5)**	**50.9**	**47.3**	**58.0**	**52.7**	**55.0**	**51.6**	**(13.6)**
Cumulative FCF	$ million		(102.1)	(100.7)	(115.3)	(186.8)	(135.9)	(88.6)	(30.7)	22.0	77.1	128.7	115.1
All In Sustaining Cost(2)	US$/oz	**870**	1,512	915	996	909	810	867	786	825	786	825	
All In Cost before tax(2)	US$/oz	**1,103**	2,295	1,129	1,341	1,833	816	871	792	830	786	825	
All In Cost after tax(2)	US$/oz	**1,143**	2,295	1,129	1,341	1,833	816	871	792	830	786	825	

Notes:

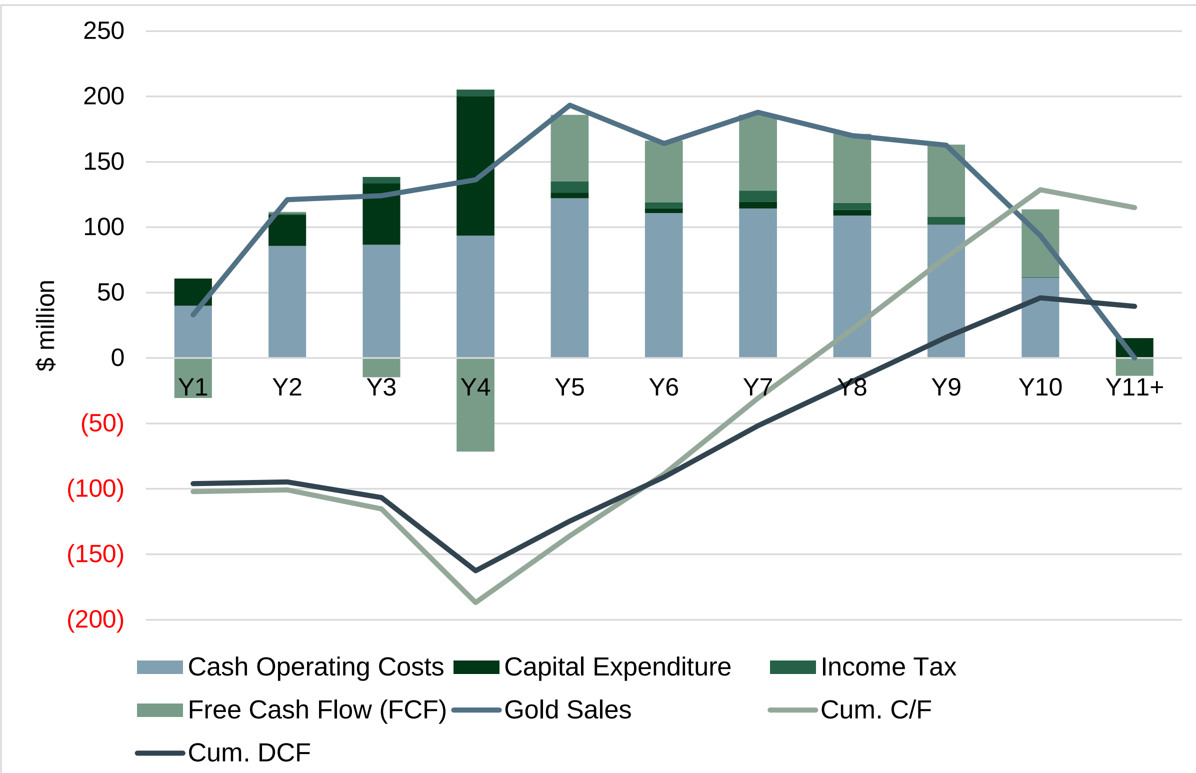
(1) Values do not include depreciation.

(2) Non-GAAP measure, see discussion under "Non-GAAP Measures" in Centerra's Management's Discussion and Analysis for the year ended December 31, 2016, and for the quarter ended September 30, 2017.

CASH FLOW ANALYSIS

Considering the Project on a stand-alone basis, the undiscounted after-tax cash flow totals $115.1 million over the mine life and simple payback occurs during Year 8 following the start of production. Figure 22-1 shows the key elements of the discounted cash flow analysis.

FIGURE 22-1 ANNUAL DISCOUNTED CASH FLOWS



The on-site cash operating cost averages $651 per ounce of gold, excluding royalties. Including royalties, the cash operating cost rises to $839 per ounce of gold. The All-in Sustaining Cost (AISC), including sustaining capital cost, is $870 per ounce of gold, and when construction and closure costs are included, the All-In-Cost equals $1,103 per ounce of gold before taxes and $1,143 per ounce of gold after tax. The average annual gold production during operation is approximately 111,000 ounces per year.

The NPV, at an annual discount rate of 5%, is $39.5 million, and the Project's IRR is 9.3%.

TABLE 22-3 NPV VERSUS ANNUAL DISCOUNT RATE

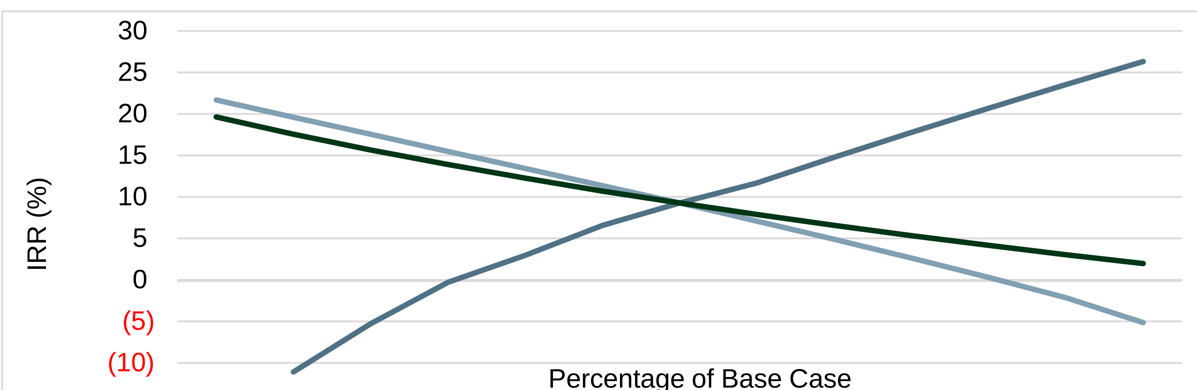
Parameter Variables	Units	Base Case							
Discount Rate	%	5%	6%	7%	8%	9%	10%	11%	12%
Net Present Value	$M	39.5	28.5	18.7	9.9	1.9	(5.3)	(11.7)	(17.5)

SENSITIVITY ANALYSES

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities to test Project IRR and NPV against changes in revenue drivers (e.g., gold price, head grade and process recovery), operating costs, and capital expenditures.

IRR sensitivity over the base case was calculated for -30% to +30% variations in each of the three parameters listed above. The sensitivities results, on an after-tax basis, are shown in Figure 22-2.

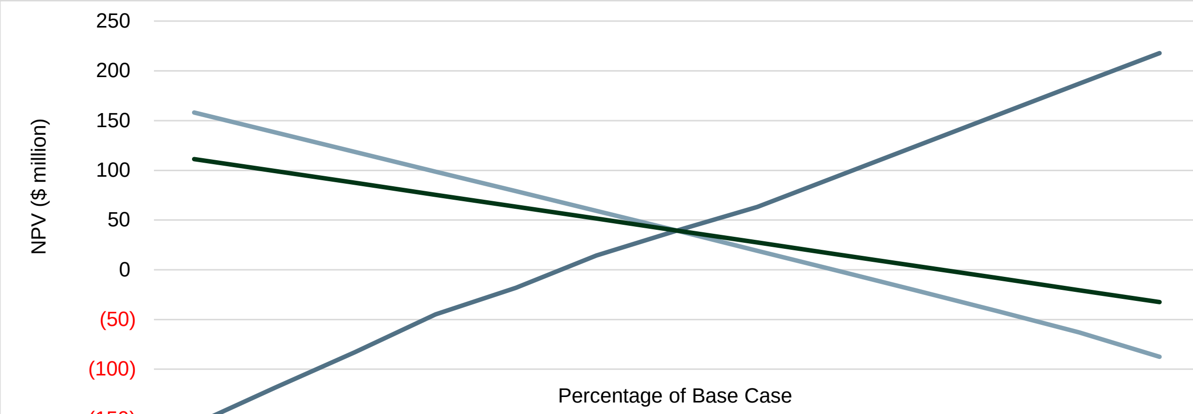
FIGURE 22-2 SENSITIVITY OF AFTER-TAX PROJECT IRR



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Gold		(11.1	(5.3)	(0.3)	2.9	6.6	9.3	11.7	14.8	17.8	20.7	23.5	26.3
Opex	21.7	19.6	17.6	15.5	13.4	11.4	9.3	7.1	4.9	2.6	0.3	(2.1)	(5.1)
Capex	19.6	17.6	15.7	13.9	12.3	10.7	9.3	7.9	6.6	5.3	4.2	3.0	2.0

Sensitivity of NPV of the after-tax cash flow at the base case annual discount rate of 5% was also tested for -30% to +30% variances in the same parameters. The results are shown in Figure 22-3.

FIGURE 22-3 SENSITIVITY OF AFTER-TAX PROJECT NPV



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Gold	(155.7)	(118.8)	(82.9)	(44.9)	(18.0)	14.3	39.5	63.2	94.2	125.1	156.1	187.0	218.0
Opex	158.0	138.3	118.5	98.7	79.0	59.2	39.5	19.2	(1.0)	(21.4)	(42.0)	(62.9)	(87.4)
Capex	111.3	99.4	87.4	75.4	63.4	51.4	39.5	27.5	15.5	3.5	(8.5)	(20.5)	(32.5)

The sensitivity of the Project NPV to discount rate at various gold prices was determined, as shown in Table 22-4.

TABLE 22-4 NPV VERSUS GOLD PRICE AND DISCOUNT RATE

Parameter Variables	**Discount Rate (%)**			
Gold Price ($/oz)	0%	**5%**	8%	10%
1,000	(55.9)	(82.9)	(91.7)	(95.5)
1,100	16.0	(31.2)	(48.7)	(57.3)
1,200	79.1	13.8	(11.4)	(24.2)
1,250	**115.1**	**39.5**	**9.9**	**(5.3)**
1,300	139.7	57.0	24.4	7.7
1,400	209.0	106.6	65.5	44.1
1,500	278.3	156.1	106.6	80.5

ALTERNATIVE SCENARIOS

The base case described above considers mining and processing 4,800 t/d of oxide and sulphide ores by means of CIP and BIOX® processes, respectively. Two alternative scenarios were also considered:

- "Higher CIP Throughput" mine plan producing and treating 6,000 t/d by CIP, then 4,800 t/d by BIOX®.
- "CIP Only" case, mining and treating 6,000 t/d by CIP over a three year period.

TABLE 22-5 COMPARISON TO ALTERNATIVE SCENARIOS

Item	Units	Base Case	Higher CIP Throughput Case	CIP Only Case
Ore Mined	kt	15,356	15,736	8,270
Gold Grade	g/t	2.67	2.63	2.45
Waste Mined	kt	72,588	72,206	36,061
Total Mined	kt	87,944	87,942	44,331
Stripping Ratio	(W:O)	4.73	4.59	4.36
Production Rate	t/d	4,800	6,000	6,000
Milled + CIL	kt	4,847	7,580	6,576
Milled + BIOX®	kt	10,509	8,156	0
Total Milled	kt	15,356	15,736	6,576
Head Grade Gold	g/t	2.67	2.63	2.45
Recovery	%	84.4	79.3	70.8
Gold Produced	koz	1,111	1,055	367
Revenue from Metal Sales	$M	1,390.9	1,320.6	459.9
Cost of Metal Sales	$M	(4.8)	(4.5)	(1.6)
Direct Operating Costs (1)	$M	(723.1)	(672.9)	(236.4)
Royalties (1)	$M	(208.9)	(197.3)	(67.3)
EBITDA	$M	454.1	445.8	154.6
Income Tax	$M	(43.5)	(44.3)	(5.5)
Capital Expenditure (2)	$M	(295.5)	(292.1)	(97.8)
Free Cash Flow (3)	**$M**	**115.1**	**92.4**	**51.2**
Cash Operating Cost (3)	$/oz	839	823	831
All-in Sustaining Cost (3)	$/oz	870	826	832
NPV @ 5%	**$M**	**39.5**	**35.3**	**29.1**
NPV @ 8%	$M	9.9	(5.7)	18.3
NPV @ 10%	$M	(5.3)	(9.7)	12.0
Project IRR	%	9.3%	8.7%	14.5%

Notes:
(1) Values do not include depreciation.
(2) Post-closure capital expenditures are discounted to year of closure.
(3) Non-GAAP measure, see discussion under "Non-GAAP Measures" in Centerra's Management's Discussion and Analysis for the year ended December 31, 2016, and for the quarter ended September 30, 2017.

23 ADJACENT PROPERTIES

There are no adjacent properties to be discussed in this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

LEGAL PROCEEDINGS AGAINST LICENCES

In the first quarter of 2016, a non-governmental organization (NGO) called "Movement to Save Mt. Noyon" filed a claim in Mongolian court against the Mongolian Mineral Resources Authority (now MRPAM) requesting that MRPAM annul two administrative decisions related to the mining licences underlying the Project. CGM is involved in the claim as a third party. One administrative decision related to a routine approval of a change of name of the Gatsuurt licence holder. That administrative decision does not affect the validity of the Gatsuurt licences. The second decision related to a non-material licence. The claimant's request has previously been granted twice (in May 2016 and May 2017) by the lower court and overturned both times on appeal. On July 26, 2017, the Mongolian lower court granted the claimant's request to suspend the two administrative acts and that decision has subsequently been upheld by an appellate court. While Centerra believes that this claim is without merit, there are no assurances that the claim will be resolved in favour of CGM. Subsequent adverse rulings of the Mongolian courts which may otherwise relate to the Gatsuurt licences or delays in the court process may have a material adverse impact on the Company's future cash flows, earnings, results of operations, or financial condition.

25 INTERPRETATION AND CONCLUSIONS

Based on the information contained herein, the QPs, as authors of this Technical Report, offer the following interpretations and conclusions:

GEOLOGY

- The Gatsuurt deposit is located in the northeast-trending Mongol-Okhotsk belt in north-central Mongolia along the regional Sujigtei fault. Mineralization is hosted in early Paleozoic meta-sediments and early Paleozoic granodiorites, granites, and diorite that have intruded the meta-sedimentary rocks.

- Mineralization is structurally controlled along first, second, and possibly third-order faults that are generally steeply dipping and parallel to sub-parallel to the Sujigtei fault.

- Mineralization is concentrated in two distinct areas that are sinistrally separated by approximately 800 m along the Sujigtei fault.

- Mineralization occurs in: (a) quartz-sericite-pyrite-arsenopyrite-iron carbonate (siderite) veinlets; (b) quartz-native gold veinlets; and (c) black silica or pervasive quartz silica rock, in which disseminated sulphides and native gold make up one of the highest-grade components of the deposit.

- The geology of the Project area has shown to contain considerable amounts of gold mineralization in two laterally distinct zones; the geology and geological controls on mineralization are sufficiently well understood for resource modelling and estimation.

MINERAL RESOURCES

- The updated Mineral Resource Model described in this report considers 630 core boreholes (87,261 m) completed between 1998 and 2016 and is based on a new interpretation of geology and grade distribution and considers three nested grade domains with increasing cut-off grades of 0.4 g/t Au, 1.0 g/t Au, and 3.0 g/t Au.

- Estimation parameters largely follow those employed in the previous resource model; however, block sizes were reduced to 5 m x 5 m x 5 m to conform the block geometry to the thin and local variations found primarily within the high-grade domains.

- The current model considers high grade domains in the Central Zone in areas where none were considered previously. This change has led to a slight overall decrease in reported Mineral Resources. However, the QP considers the current model to be more robust than previous models.

MINING AND MINERAL RESERVES

- Although the QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate, the estimate should be considered within the context of the outcomes stated in this Report. Of note, as outlined in Section 22, the Project's base case economic modelling results indicate a marginally positive NPV@5% of $39.5M. Should any of the key revenue drivers (price, grade, recovery) incur any adverse variance to the plan; this in turn places the above estimate at risk.

MINERAL PROCESSING

- Gold mineralization is categorized as oxide, transition, or sulphide.

- The metallurgical properties of this mineralization bear important implications for the design of a suitable processing facility to recover the gold and the economic significance of the Project.

- Gatsuurt ore will be treated at Centerra's Boroo Mine. Processing of the oxide ore will occur through the existing CIP facility at Boroo and the sulphide ore will be treated through a BIOX® process followed by a CIL facility which will be operational by Year 4.

ECONOMIC ANALYSIS

- The base case results indicate a marginally positive NPV at an annual discount rate of 5%. However, there is considerable down-side risk should any of the key revenue drivers (price, grade, recovery) present any adverse variance to plan.

- Analysis of the alternative scenarios suggests that the increase in production rate is unlikely to result in any significant reduction in unit costs. Nevertheless, the CIP Only case (mining and treating 6,000 t/d by CIP over a three year period) generates a higher IRR (14.5%) than the base case (9.3%) and slightly greater NPV at a discount rate of 8% or more, suggesting that this option may be worthy of further investigation.

- It is apparent that under base case conditions, the treatment of oxide ores in the existing process plant is profitable. However, it does not, by itself, generate sufficient cash flow to fund the development of the BIOX® circuit required to treat refractory ore from Gatsuurt, which therefore requires further investment during the first four years of the operating period.

- Once the BIOX® plant is in place, the Project begins to generate better returns (and on a simple, undiscounted basis pays back that investment within four years). However, the relatively short mine life then curtails operations before significant value is created.

RISK MANAGEMENT

Centerra defines risk as an uncertainty that, if it occurs, may have a positive or negative effect on the outcome of the Company's objectives. Risk treatment workshops for this Technical Report were undertaken in order to determine the risk profile for the Project. Technical and non-technical risks were considered and four “high severity” risks were identified and described below. The following risk factors should be read in conjunction with Centerra’s Annual Information Form for the year ended December 31, 2016 and, in particular, the section entitled “risks that can affect our business” therein.

EXTREME AND HIGH SEVERITY THREATS

HIGH SEVERITY RISK EVENTS

Risk Event: Accident on the Haul Road Causing Death or Serious Injury;

The analysis of this risk prior to mitigation determined the impact to be major (4) and the likelihood to be likely (4). The causes for this risk include a high volume of mine related traffic interacting with public, herders/animals on the haul road, drunk drivers from the public, speeding, poor visibility/poor line of sight, and seasonal poor driving conditions. The possible impacts include loss of life, prolonged stoppages which affect the hauling of ore and damage to property.

Risk Event: Water Treatment Required for all Contact Water Higher than Expected Costs

The analysis of this risk prior to mitigation determined the impact to be major (4) and the likelihood to be likely (4). The causes for this risk are: more ARD material than in the current model, higher inflow rates into the pit, passive water treatment unable to meet local water regulations, and mixing zones have not been seen to date in Mongolia.

Possible impacts include higher than expected costs and reputational damage.

Risk Event: Expectations of the Local Community Exceed the Capacity of the Project to Deliver

The analysis of this risk prior to mitigation determined the impact to be major (4) and the likelihood to be possible (3). The causes for this risk are misperception of the community due to rumors, trade union involved in creating instability, and lack of skill set in local community.

The impacts of this risk are the loss of social licence and the inability to negotiate a Community Development Agreement.

Risk Event: Possibility of an organized artisanal miners or trespassers incursion

The analysis of this risk prior to mitigation determined the impact to be high (4) and the likelihood to be possible (3). The causes for this risk are lack of alternative income generating opportunities in the area, Gatsuurt, having an artisanal mining area history, and lack of enforcement regulations in respect of artisanal miners in Mongolia. This risk could impact production, result in injury or fatality, loss of social licence, and an incursion of artisanal miners could destroy reclamation work.

Risk Event: Possibility of losing Local ‘Social Licence to Operate’ due to failure to negotiate and conclude a Community Development Agreement

The analysis of this risk prior to mitigation determined the impact to be high (4) and the likelihood to be possible (3). The causes for this risk are (i) a failure to reach agreement on the terms of the Community Development Agreement (CDA) and not establish the CDA with local governments because the local governments may expect more than what the project can offer; and (ii) the Aimag and Soum governments fail to work collaboratively work and conclude the CDA with CGM. This risk could potentially result in (i) losing CGM’s local ‘social licence to operate’, (ii) a poor relationship with Selenge Aimag, Bayangol and Mandal soums, (iii) delays in getting permits and approvals from the local government, and (iv) resistance and protest from the community members and civil society on the Project.

Several other risks identified through the report include:

1) In Section 15, the results of the economic analysis to support Mineral Reserves represent forward looking information that is subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented in this report. Uncertainty that may materially impact Mineral Reserve estimation include realized prices, market conditions, capital and operating cost estimates, foreign exchange rates, resource model performance, recoveries, and the timely and successful implementation of recommended actions.

2) In Section 20, three main risks were identified:

    a. The main sources of drinking water in the Project area are wells, rivers and springs, and mobile distribution points. Bayangol sources more water from unprotected wells (at a higher risk of contamination than protected wells) than Mandal Soum.

b. The main causes of traffic accidents in the Project area are related to risky driving practices. Moreover, police statistics indicate that the major crimes in Selenge Aimag are related to traffic safety and the use of motor vehicles.
c. Potential impacts to water resources represent the most significant risk at closure with sources of potentially contaminated water from the open pits, waste dumps, and low grade stockpile. Water quality prediction of pit water at closure is uncertain, but may pose a high risk of adversely affecting surface and groundwater quality. The resulting water quality prediction is unknown, but arsenic species may be present.

3) Claim to Revoke Gatsuurt Mining Licences:

a. As noted above an NGO has made a claim to revoke the Project's licences. If that claim is successful the validity of the Project licences may become uncertain. While Centerra does not believe the NGO's claim has any merit, there are no assurances that a Mongolian court or governmental agency will not take action that is adverse to the Project's licences.

26 RECOMMENDATIONS

Based on the information contained herein, the QPs, as authors of this Technical Report, offer the following recommendations.

GEOLOGY AND MINERAL RESOURCES

- The Project merits further exploration expenditures to improve overall Project economics.

- While the overall geology is well understood, additional investigations through oriented core drilling is warranted especially in the South Slope area of the Central Zone to better understand the structural controls on mineralization and overall geometry of the mineralized zone.

- Additional core drilling is warranted to test for a down-dip extension of mineralization in the Central Zone.

- Borehole spacing should be no less than currently available throughout the deposit area.

- Additional drilling should be considered in mineralized areas along strike that have not been included in the current Mineral Resource Model in order to include mineralization in those areas in future resource updates.

GEOTECHNICAL ASPECTS

- The interpretation of faulting with the rhyolites is not well understood and will require further investigation by a qualified structural geologist.

- The weathering profile (in particular, weathering along joints), as well as existence of microfractures noted in the core, may influence rock mass stability at a bench and inter-ramp scale. Discontinuum numerical modelling should be considered once rock exposures have been mapped during early mining.

- The base of the high-weathered material should be determined from all available drill hole data and a wireframe developed for this surface during the detail design phase of this project.

- Structurally controlled failure could be sensitive to the presence of groundwater pressures. The most recent hydrogeological data should be assessed to see if it impacts any recommendations made in this report and a hydrogeological program should be initiated to include monitoring of water pressures near the pit slopes during early pit development. This program can then be used as a basis for assessing potential impacts of groundwater pressures on pit slopes and any need to provide dewatering measures for geotechnical purposes.

- Additional drilling should be completed on the waste dump footprints to confirm subsurface conditions assumed as part of this study.

MINING AND MINERAL RESERVES

- The material flagging process should be reviewed and updated once more accurate cost information becomes available and at any time there is a notable change in either costs or metal prices. This should be done with consideration of the potential effect on reported Mineral Reserves.

- Further refinement of the sulphur feed grade can be achieved at an operational level with appropriate grade control and stockpile management. Investigations into the practicality and effectiveness of further classification of the ore into sulphur graded stockpiles should be carried out as an optimization exercise.

- The NAG and PAG waste classification scheme should be further investigated and refined as part of any detailed engineering phase, after which the dump designs and schedule would need to be revised to ensure appropriate dump sizing, NAG cover, and timing issues are addressed.

- Once approvals for the dump locations are finalized, the dump designs should be modified such that their design capacity matches the scheduled movements to those dumps.

- Clarification should be sought on the status of the land on which the North Dump is to be situated.

- Further analysis of ore loss and dilution should be carried out once operational data becomes available.

MINERAL PROCESSING

- Conduct a thorough review prior to commencement of detailed engineering phase to assess optimum building and equipment layout for Flotation and BIOX® areas to identify cost reduction potential, some of which include:
    - Evaluate opportunity for reduction in size of BIOX® and Neutralization containment area walls to potentially reduce size and concrete requirements.
    - Review current BIOX® Utility Building size and equipment layout for optimal layout, maintenance, and potential reduction of overall building size.
    - Evaluate potential for Flotation Building height reduction when equipment details are available for the Limestone Mill, cyclones, and flotation cells.

INFRASTRUCTURE

- Due to the geographic proximity and potential for combining of shared utilities and services, it is recommended that the CIL and Cyanide Detox circuits be integrated into the BIOX® circuit and detailed engineering and construction be executed simultaneously.

- This would provide a more complete cost estimate and take advantage of synergies such as improved layout, reduced materials, and vendor packaging to lower cost, construction flexibility, and maximization of schedule. The execution schedule and construction strategy was prepared on this basis.

- Evaluate the option to use pre-engineered building structures for buildings. The layout is suitable to pre-engineered building. There may be an opportunity for cost reduction, improved construction efficiency and reduction in overall schedule.

OPERATING COST

- A contingency of 10% should be applied to the contract mining cost in any future economic modelling, although the inclusion of a 10% contingency does not impact the Mineral Reserve estimate.

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28 DATE AND SIGNATURE PAGE

This report titled "Technical Report on the Gatsuurt Gold Project, Mongolia" and dated December 22, 2017 (with an effective date of October 31, 2017) was prepared and signed by the following authors:

(Signed) "*Gordon D. Reid*"

Dated at Toronto, ON
December 22, 2017

Gordon D. Reid, P.Eng.
Vice President, and Chief Operating Officer
Centerra Gold Inc.

(Signed) "*Boris Kotlyar*"

Dated at Toronto, ON
December 22, 2017

Boris Kotlyar, M.Sc., P.Geo., AIPG
Chief Geologist, Global Exploration
Centerra Gold Inc.

(Signed) "Tommaso Roberto Raponi"

Dated at Toronto, ON
December 22, 2017

Tommaso Roberto Raponi, P.Eng.
President
T.R. Raponi Consulting Limited

(Signed and sealed) "Lars Weiershäuser"

Dated at Toronto, ON
December 22, 2017

Lars Weiershäuser, Ph.D., P.Geo.
Associate Senior Geologist
Roscoe Postle Associates Inc.

(Signed) “Igor Bojanic”

Dated at St Leonards, Australia
December 22, 2017

Igor Bojanic, FAusIMM, MIQA
Head of Metals - ARC
RPM Advisory Services Pty Ltd

(Signed and sealed) “Chris Sharpe”

Dated at Toronto, ON
December 22, 2017

Chris Sharpe, P.Eng
Director, Technical Services
Centerra Gold Inc.

(Signed) “William Pitman”

Dated at Oakville, ON
December 22, 2017

William Pitman, M.Sc., P.Eng., ACSM
Director and Principal Geotechnical Engineer
Adiuvare Geology and Engineering Ltd.

(Signed) “Kevin P.C.J. D'Souza”

Dated at Toronto, ON
December 22, 2017

Kevin P.C.J. D'Souza, MEng, CEng, ARSM, FIMMM, FRGS
Vice-President, Security, Sustainability & Environment
Centerra Gold Inc.

(Signed and sealed) “Christopher Jacobs”

Dated at Toronto, ON
December 22, 2017

Christopher Jacobs, CEng, MIMMM, MBA
Vice President & Mining Economist
Micon-International

29 CERTIFICATE OF QUALIFIED PERSON

GORDON D. REID

I, Gordon D. Reid, P.Eng., as an author of this report entitled "Technical Report on the Gatsuurt Gold Project, Mongolia" prepared for Centerra Gold Inc. with an effective date of October 31, 2017, do hereby certify that:

1. I am Vice President, and Chief Operating Officer of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of the University of Manitoba in 1994 with a Master's degree in Business Administration, and a graduate of Michigan Technological University in 1981 with a Bachelor of Science degree in Mining Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #38536504). I have worked as a mining engineer for a total of 36 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Since 2004, employed by Centerra Gold Inc. in various positions including VP & COO, Corporate VP Operations, President Kumtor Operating Company, and VP Business Development, involved in preparation of and adherence to annual mine plans and budgets, review and approval of capital programs, mine optimization studies, and other engineering and management related tasks.
   - From 1998 to 2002 was the Director – Technical Services responsible for the development of a feasibility study and Environmental Impact Assessment for a mine development project in Wisconsin.
   - From 1986 to 1992 was Chief Mine Engineer and Mine Superintendent at an operating mine in Ontario responsible for planning and implementing long term and short term mine development and production strategies to achieve budgeted production and cost targets.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Gatsuurt Property numerous times and last in March 2017. Prior to this, I have completed numerous inspections of the Gatsuurt Property since 2006.

6. I have overall responsibility for this Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed) "*Gordon D. Reid*"

Gordon D. Reid, P.Eng.

BORIS KOTLYAR

I, Boris Kotlyar, M,Sc., P.Geo., AIPG, as an author of this report entitled "Technical Report on the Gatsuurt Gold Project, Mongolia" prepared for Centerra Gold Inc. with an effective date of October 31, 2017, do hereby certify that:

1. I am Chief Geologist, Global Exploration with Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Canada, M5J 2P1.

2. I am a graduate of Moscow State University in 1975 with a M.Sc. Degree in Geology and Exploration.

3. I am registered, since 2009, with the American Institute of Professional Geologists (AIPG) as a Certified Professional Geologist in the Province of Ontario (AIPG # 27278), as well as Fellow of SEG (#439501) since 1996. I have worked as a geologist for a total of 42 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Geology, exploration
   - Sample preparation and analysis
   - Data verification

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have visited the Gatsuurt Project numerous times over the course the last 21 years.

6. I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, and 21 of the Technical Report.

7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101 as a result of my employment with Centerra Gold Inc.

8. I have had extensive prior involvement with the Gatsuurt Property that is the subject of the Technical Report, from the very outset of its exploration in 1998.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report/Section Nos. 4, 5, 6, 7, 8, 9, 10, 11, 12, and 21 in the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed) "*Boris Kotlyar*"

Boris Kotlyar, M.Sc., P. Geo., AIPG

TOMMASO ROBERTO RAPONI

I, Tommaso Roberto Raponi, P.Eng., as an author of this report entitled "Technical Report on the Gatsuurt Gold Project, Mongolia" prepared for Centerra Gold Inc. with an effective date of October 31, 2017, do hereby certify that:

1. I am currently employed as President of TR Raponi Consulting Ltd. (Ontario Certificate of Authorization No. 100505108) with an office at 15-223 Rebecca Street, Oakville, Ontario, L6K 3Y2.

2. I am a graduate of the University of Toronto with a BASc. in Geological Engineering, 1984. I have practiced my profession continuously since 1984.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg. No. 90225970) and the Association of Professional Engineers and Geoscientists of BC (Reg. No. 23536). I have worked as an independent consultant since 2016. I have worked as a mining engineer/geologist for a total of 33 years since my graduation. My relevant experience for the purpose of the Technical Report is in the development, design, commissioning and operation of mineral processing plants in Canada, United States, Mexico, Brazil, Venezuela, Surinam, Chile, Kyrgyzstan, Mongolia, Turkey, and Saudi Arabia.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Gatsuurt Project on August 17 to 19, 2015.

6. I am responsible for Sections 13, 17, and parts of 21 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had prior involvement with the property that is the subject of the Technical Report as a result of my previous employment with Centerra Gold Inc.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report Section Nos. 13, 17, and parts of 21 in the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed) ***"Tommaso Roberto Raponi"***

Tommaso Roberto Raponi, P.Eng.

LARS WEIERSHÄUSER

I, Lars Weiershäuser, Ph.D., P.Geo., as an author of this report entitled "Technical Report on the Gatsuurt Gold Project, Mongolia" prepared for Centerra Gold Inc. with an effective date of October 31, 2017, do hereby certify that:

1. I am an Associate Senior Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

2. I am a graduate of South Dakota School of Mines and Technology in Rapid City, SD, USA in 2000 with a Masters of Science in Geology and Geological Engineering and the University of Toronto, Toronto, Canada in 2005 with a Doctorate in Geology.

3. I am registered as a Professional Geologist in the Province of Ontario (APGO# 1504). I have worked as a geologist for a total of 13 years since my graduation from the University of Toronto. My relevant experiences for the purpose of the Technical Report are:
   - The preparation of various Mineral Resource Models of gold and base metal deposits.
   - The preparation of numerous technical reports for gold and base metal projects.
   - The review and audit of project data of gold and base metal deposits, including the analysis of analytical quality control data.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Gatsuurt Project on November 15 and 16, 2017.

6. I am responsible for Section 14 and parts of Sections 10, 11, and 12 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Section Nos. 10, 11, 12, and 14 in the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed and sealed) ***"Lars Weiershäuser"***

Lars Weiershäuser, Ph.D., P.Geo.

IGOR BOJANIC

I, Igor Bojanic, FAusIMM, MIQA, as an author of this report entitled “Technical Report on the Gatsuurt Gold Project, Mongolia” prepared for Centerra Gold Inc. with an effective date of October 31, 2017, do hereby certify that:

1. I am Head of Metals within the Advisory division for the Australiasia and Russia region with RPM Advisory Services Pty Ltd, of Level 4, 55 Chandos Street, St Leonards, Australia. RPM Advisory Services Pty Ltd is a wholly owned subsidiary of RPMGlobal Holdings Limited.

2. I am a graduate of the University of New South Wales (Sydney, Australia) in 1984 with a Bachelor of Engineering (Mining, Hons). I also have tertiary qualifications in Business and Environmental Management.

3. I am a Fellow of the Australian Institute of Mining and Metallurgy. I have worked as a mining engineer for a total of 32 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Browns Creek Gold Mine, Australia (1987 to 1990), Mining Engineer
   - Telfer Gold Mine, Australia (1990 to 1992), Senior Mining Engineer
   - Extended consulting positons at Tuckabianna Gold Mine (Australia), Golden Cross Gold Mine (New Zealand) and Macres Gold Mine (New Zealand) (1994 to 1995)
   - Currently the Manager of Advisory Consulting for the Australasia region including Mongolia.
   - Have completed numerous technical studies over the last decade on mining projects in Mongolia, including participation in NI 43-101 reports.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I did not conduct a personal inspection of the relevant property.

6. I am responsible for Section 15 and parts of Sections 16 and 21 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, that part of the Technical Report that I am responsible for as a Qualified Person (namely Section 15 and parts of Sections 16 and 21 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed) ***"Igor Bojanic"***

Igor Bojanic, FAusIMM, MIQA

CHRIS SHARPE

I, Chris Sharpe, P.Eng., as an author of this report entitled "Technical Report on the Gatsuurt Gold Project, Mongolia" prepared for Centerra Gold Inc. with an effective date of October 31, 2017, do hereby certify that:

1. I am Director Technical Services with Centerra Gold Inc a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of Dalhousie University, Halifax in 2001 with a bachelor of Mining Engineering degree.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #100053512). I have worked as a mining engineer for a total of 16 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Geotechnical experience gained working for Golder Associates
   - Geotechnical experience gained working for Wardrop

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Gatsuurt Project on May 2nd to 6th, 2016.

6. I am responsible for parts of Section 16 of the Technical Report.

7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101 as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, that part of the Technical Report that I am responsible for as a Qualified Person (namely parts of Section 16 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed and Sealed) ***"Chris Sharpe"***

Chris Sharpe, P.Eng.

WILLIAM PITMAN

I, William Pitman, M.Sc., P.Eng., ACSM, as an author of this report entitled "Technical Report on the Gatsuurt Gold Project, Mongolia" prepared for Centerra Gold Inc. with an effective date of October 31, 2017, do hereby certify that:

1. I am a Director and Principal Geotechnical Engineer with Adiuvare Geology and Engineering Ltd. of 151 Elmwood Road, Oakville, Ontario, L6K 2A6.

2. I am a graduate of the Camborne School of Mines, University of Exeter, UK with a Bachelors Degree in Mining Engineering (1984) and a Masters Degree in Mining Engineering (1993).

3. I am registered as a Professional Engineer in the Province of Ontario, licence No. 100029014 and as a Professional Engineer in the Province of Newfoundland and Labrador, registration No.04637. I have practiced my profession continuously since 1984. I have over 33 years' experience in geotechnical engineering for open pit and underground mines. Prior to joining Adiuvare Geology and Engineering Ltd., I worked for Golder Associates for 23 years and for Australian Mining Consultants (Canada) Ltd. for five years. Prior to that I worked at several mining operations.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Gatsuurt Project.

6. I am responsible for parts of Section 16 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and Form 43-101F1 and have prepared parts of Section 16 of the Report in compliance with NI 43-101 and Form 43-101F1

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, that part of the Technical Report that I am responsible for as a Qualified Person (namely parts of Section 16 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed) ***"William Pitman"***

William Pitman, M.Sc., P.Eng., ACSM

KEVIN P.C.J. D'SOUZA

I, Kevin P.C.J. D'Souza, MEng, CEng, ARSM, FIMMM, FRGS, as a co-author of this report entitled "Technical Report on the Gatsuurt Project, Mongolia", prepared for Centerra Gold Inc., with an effective date of October 31, 2017 do hereby certify that:

1. I am Vice President Security, Sustainability & Environment of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of Royal School of Mines, Imperial College of Science Technology, and Medicine University of London in 1993 with a 1st Class Master's (MEng) degree in Mining Engineering.

3. I am registered as a Charted Engineer (1997, Reg.500601) through the UK's Institute of Materials, Minerals and Mining (IMMM). I have worked as a sustainability and environment professional in the mining industry for a total of 24 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Worked in a wide range of operational, managerial and leadership roles that range from remote exploration camps, mine construction sites, operational mines and closure sites, to Corporate and Ministerial offices, with a focus on environmental management, sustainability and mine security.

   - Managed sustainable mining programs for major NGOs and worked for International Funding Institutions (IFIs) supporting emerging market Governments on macro-level mining and environmental policy reform and investment programs (including the World Bank, IFC, EBRD, ILO, USAID and the UK's DFID).

   - Worked for, or as a consultant, with numerous junior exploration and mining companies and also many of the industry's majors including Barrick Gold, AngloGold Ashanti, Gold Fields, Rio Tinto, BHP Billiton, Kinross, and De Beers.

   - Direct mining experience in around fifty countries worldwide largely in Asia, Latin America and sub-Saharan Africa.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Gatsuurt Property numerous times and last in March 2017. Prior to this, I have completed numerous inspections of the Gatsuurt Property since 2013.

6. I am responsible for Section 20 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101 F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report/Section 20 in the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed) ***"Kevin P.C.J. D'Souza***

Kevin D'Souza, MEng, CEng, ARSM, FIMMM, FRGS

CHRISTOPHER JACOBS

I, Christopher Jacobs, CEng, MIMMM, MBA, as an author of this report entitled "Technical Report on the Gatsuurt Gold Project, Mongolia" prepared for Centerra Gold Inc. with an effective date of October 31, 2017 do hereby certify that:

1. I am employed by, and carried out this assignment for, Micon-International Limited, Suite 900 - 390 Bay Street, Toronto, Ontario M5H 2Y2.

2. I hold the following academic qualifications:
   - B.Sc. (Hons) Geochemistry, University of Reading, 1980;
   - M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.

3. I am a Chartered Engineer registered with the Engineering Council of the U.K. (registration number 369178). Also, I am a professional member in good standing of The Institute of Materials, Minerals and Mining; and The Canadian Institute of Mining, Metallurgy and Petroleum (Member). I have worked in the minerals industry for more than 35 years; my work experience includes 10 years as an exploration and mining geologist on gold, platinum, copper/nickel, and chromite deposits; 10 years as a technical/operations manager in both open-pit and underground mines; three years as strategic (mine) planning manager and the remainder as an independent consultant when I have worked on gold deposits, inter alia.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Gatsuurt Property.

6. I am responsible for Section 22 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the portion of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Section 22 in the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December, 2017

(Signed and Sealed) *"Christopher Jacobs"*

Christopher Jacobs, CEng, MIMMM, MBA