Chaarat Gold Holdings Ltd.

Tulkubash Gold Project

Bankable Feasibility Study Update Report

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REVISION STATUS

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INTRODUCTION TO AUTHOR

LogiProc (Pty) Ltd. Established in 1987 and trading in its current form since 1999, the LogiProc group operates from an engineering head office situated in Lonehill, north of Sandton, Johannesburg, South Africa. LogiProc has regional offices in Southern Africa and offices in other countries have been established as dictated by individual project needs.

LogiProc has a core staff of qualified engineers includes Chemical, Metallurgical, Mechanical, Electrical, Instrumentation, Civil and Structural engineers, all with a strong project engineering background. LogiProc also maintains a strong network of competent mining consultants and specialists, including Sound Mining introduced below, with whom LogiProc teams up on particular projects.

LogiProc, in the form of *ULS Mineral Resource Projects,* was involved with the feasibility study generation on the Steenkampskraal Rare Earth Element Project in the Western Cape province of South Africa Canadian National according to Instrument 43-101 (NI 43-101) code. The feasibility study was to an accuracy (+-15%) and included the mine, process facility and all the associated infrastructure to produce per year 5000 tons of separated Total Rare Earth. LogiProc was responsible for the plant and surface infrastructure portion of this mine.

Also, LogiProc was a junior partner to Ausenco Services Pty Ltd in the update of the Eurasian Resources Group S.a.r.l. Roan Tailings Reclamation Project Phase 3 Expansion Feasibility Study Report to an estimating accuracy in line with AACE Class 2. LogiProc was assisted and advised by Sound Mining with regards to compilation aspects of the 2021 BFS.

Sound Mining (Pty) Ltd. is a consultancy specializing in the Mining Sector. Its consultants have extensive experience in preparing mine designs and schedules, resource and reserve statements, compliant competent persons' reports, technical advisors' and valuation reports for mining and exploration companies. Sound Mining staff are members of the various regulatory bodies in South Africa and Australia that enable them to report to: -

- the SAMREC Code 2016 and the SAMVAL Code 2016;
- the Canadian National Instrument 43-101 Standards for Disclosure for Mineral Projects (NI 43-101 2011) and the 'Canadian Institute of Mining, Metallurgy and Petroleum Standards and Guidelines for Valuation of Mineral Properties' (CIMVAL 2003); and
- the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (JORC 2012) and the 'Australian Minerals Institute Guidelines for Technical Economic Evaluation of Minerals Industry Projects' (VALMIN).

Sound Mining's due diligence studies are founded on the professional best practice principles established by the South African Institute of Mining and Metallurgy (SAIMM) to which the key personnel dedicated to this Project are registered as either Fellows or Members.

ABBREVIATIONS

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1. SUMMARY

1.1. INTRODUCTION

Chaarat Zaav Closed Joint Stock Company (CZ), a wholly-owned subsidiary of Chaarat Gold Holdings Limited (Chaarat), holds two licences for gold on a property located in the Kyrgyz Republic. Two zones currently make up the Property: the Tulkubash zone and the Kyzlytash zone.

Gold mineralisation within the Project area is divided into two types: the Tulkubash mineralisation (the Tulkubash zone), which is oxidized material, and the Kyzyltash mineralisation (the Main and Contact zones), which is sulphide-rich, unoxidized refractory material.

In 2019, Chaarat retained LogiProc to update an existing BFS prepared by Tetra Tech (Tt) in April 2018, that detailed the scope, design features and economic viability of the Tulkubash Gold Project (the Project).

During 2019/2020, further work was undertaken by Chaarat to:

- complete additional recovery test work in the Mid and Satellite/East zones;
- better define the resource; and
- update the project costs, to capture changes in development, construction, operating and in-country costs.

LogiProc (Pty) Ltd was retained to update the 2019 BFS with the above information.

The Project is located close to the border with Uzbekistan in the Sandalash Range of the Alatau Mountains, in Kyrgyzstan. The Project area is about 300 km southwest of the capital, Bishkek, as shown in Figure 1-1.

FIGURE 1-1 LOCATION OF THE PROJECT

LogiProc led a group of qualified consulting companies, commissioned both by Chaarat and LogiProc, to assist with the completion of this updated Feasibility Study. Table 1-1 outlines the responsibilities of each company.

TABLE 1-1 QUALIFIED CONSULTANT RESPONSIBILITIES

Company	Responsibility
LogiProc	Overall project management; mineral processing and metallurgical testing; recovery methods; project infrastructure; capital cost estimate, economic analysis, operating cost estimate, project execution plan.
Viktor Usenko Evgeny Fomichev	Geological block model and associated data integrity.
Peter Carter	Mining method review; and ore reserve statement. Competent person for ore reserves and Mining Engineering.
WAI	Environmental studies, permitting, and social or community impact; geochemistry; hydrology; hydrogeology.
Ausenco	Heap leach facility design.

The work was led by Process Manager Richard Bewsey (Process Director at LogiProc).

The effective date of this updated Bankable Feasibility Study (BFS) is 24 May 2021 and the effective date of the Mineral Resource Estimate is 07 November 2020.

All currency is reported in United States dollars (USD), and all measurements are reported using the International System of Units (SI), unless otherwise noted.

1.2. GEOLOGY

The Property is located within the Middle Tien Shan province, which is made up of fragments of Late Devonian-Carboniferous rocks deposited in a forearc accretionary complex that was subsequently subjected to intense folding and thrusting during the upper Palaeozoic era.

The gold deposits are hosted within a northeast-trending sequence of Cambro-Ordovician siliciclastic rocks (the Chaarat Formation), which have been overthrust by Devonian-age quartzites (the Tulkubash Formation). The sedimentary rocks hosting mineralisation strike north-easterly and dip 40° to 75° northwest. Permo-Triassic-age granodiorite and diorite intrusive rocks are closely associated with the gold mineralisation and, in some areas, are mineralised.

The mineralisation is controlled by a series of subparallel brittle shear zones that are the result of a predominantly sinistral strike slip motion of the Sandalash Fault Zone. It occurs in clusters along various extensional structures related to releasing bends (Kramer 2009; Jakubiak 2017). Gold mineralisation is divided into two types of mineralisation; the Kyzyltash mineralisation (the Main and Contact zones) is sulphide-rich and refractory while the Tulkubash mineralisation is oxidized and can be processed using conventional heap leach methods.

Both the Tulkubash and the Kyzyltash mineralisation are classified as orogenic gold deposits. The Tulkubash mineralisation exhibits characteristics of shallow epithermal mineralisation, and is further classified an epizonal orogenic deposit. The Kyzyltash mineralisation formed in a much deeper environment and is classified as mesothermal orogenic gold deposits.

Only the Tulkubash mineralisation has been considered in the 2021 BFS Update. Individual gold-bearing lodes range from 5 m to 45 m in true thickness. Where multiple lodes are present, the Tulkubash zone can range up to 250 m in width with the individual lodes separated by barren rock. Development drilling of the Tulkubash deposit has revealed that the zone is remarkably continuous, but blossoms and thins along its defined length.

1.3. Exploration and Drilling

In 2004, a soil sampling geochemical survey identified numerous gold anomalies of greater than 1 g/t gold over a 4 km strike length, with the maximum value of 73 g/t in one sample. These anomalies range from 100 m to 800 m in length (along strike) and 50 m to 150 m in width.

Follow-up trenching and rock chip sampling confirmed the Tulkubash deposit, and then continued to return positive results along extensions of the trend over a 10 km strike length.

The Tulkubash database has been generated from exploration drilling (Table 1-2) and now contains data from 710 diamond drillholes, and in addition, some samples cut from trenches, totalling 100,353.7 m of sampling.

Table 1-2 Exploration Drilling at Tulkubash

Year	Tulkubash Zone	
	No. of Holes	Total Metres
2000	-	-
2004	-	-
2005	1	150
2006	7	1,393
2007	12	2,374
2008	-	-
2009	5	802

Year	Tulkubash Zone	
	No. of Holes	Total Metres
2010	37	4,271
2011	128	15,984
2012	39	6,842
2013	14	1,781
2014	48	5,813
2015	-	-
2016	12	1,185
2017	135	17,420
2018	121	19,822
2019	130	20,077
2020	21	2,434
Total	**710**	**100,348**

Drilling was conducted on a 40 m by 40 m grid spacing, for which drilling lines were angled with a 42° east rotation to correspond with the orientation of the strike of the deposit. The majority of the drillholes were drilled as inclined holes in order to cut the mineralised structures as close to right angles as possible. Underground drilling and some earlier parallel-to-strike drilling are exceptions.

The 2021 Exploration Plan includes infill and extensional drilling in Tulkubash Mid and East Zones with the aim of adding resources to Tulkubash. Initial drill testing of the Mid Karator and Isakuldy exploration targets is also planned for 2021.

1.4. Mineral Processing and Metallurgical Testing

Numerous mineralogical and metallurgical testwork programmes have been completed on the Tulkubash ore samples. A sample suitable for heap leach testwork was defined as any material within the selected pit shell that had a total sulphur (S_{TOTAL}) content of 0.5 % or less (S_{TOTAL} ≤0.5 %) and a nominal cut-off grade of 0.2 g/t gold.

Three commercial laboratories—Wardell Armstrong International (WAI) (2017), McClelland Laboratories Inc. (MLI) (2018), and ALS-Stewart Kara Balta Laboratory (ALS-Stewart) (2019), were used to complete the metallurgical testwork.

WAI tested 23 variability composite samples collected from dedicated metallurgical drillholes within the zone of mineralisation, but these were not restricted to the proposed open pit. WAI also tested two master composites; the first master composite consisted of sub-samples from all variability samples, and the second master composite consisted of selected variability samples representing the heap leach ore within the designed open pit, mainly. WAI completed the testwork between October 2016 and March 2017.

MLI completed a separate testwork programme in 2018, which included a variability test programme consisting of 48 coarse ore bottle roll tests, followed by 11 column leach tests simulating heap leach conditions. MCL completed the testwork between December 2017 and July 2018.

ALS-Stewart completed tests on 22 composites collected form metallurgical drillholes around and within the east and mid pits. Head assays and bottle roll tests were completed on each composite. ALS-Stewart completed the testwork in 2019.

LogiProc analysed all the metallurgical testwork results with the objective of identifying optimal heap leach conditions. The WAI, MLI, and ALS-Stewart metallurgical studies indicate that the oxide ore is amenable to cyanide heap leaching and can be efficiently processed using a heap leach-based flowsheet.

Based on the metallurgical testwork results, the expected LoM recovery for gold and silver is calculated to be 73.6% and 63.4%, respectively.

1.5. MINERAL RESOURCE ESTIMATE

The economic parameters considered for the Mineral Resource declaration were obtained from the Client and include:

- Gold price of USD1,800/tr oz;
- Gold recovery of 72.6%;
- Mining cost of USD1.89/t;
- Operating cost of USD7.24/t; and

The updated Mineral Resource for Tulkubash is summarised in Table 1-3. The definitions of Mineral Resources as outlined within the JORC code (2012) for Mineral Resources were adopted in order to classify the Resources.

The effective date of the updated Mineral Resource is 7th November 2020.

TABLE 1-3 TULKUBASH MINERAL RESOURCE STATEMENT (EFFECTIVE 7 NOVEMBER 2020)

Classification	Quantity (Kt)	Grade Au (g/t)	Contained Metal Au (koz)
Measured	-	-	-
Indicated	28,505	0.86	789
Inferred	21,412	0.56	388

Notes:

1. *Numbers are rounded in accordance with disclosure guidelines and may not sum accurately;*
2. *The Mineral Resource has been estimated using 5.0 m x 5.0 m x 5.0 m (x, y, z) blocks;*
3. *The estimate was constrained to the mineralised zone using wireframe solid models;*
4. *The wireframes were sub-domained to isolate the strongly mineralised main zone from the gold mineralisation in the main structural corridor;*
5. *Grade estimates were based on 1.5 m composited assay data; and*
6. *The Mineral Resource estimate has been reported to 0.21 g/t cut-off grade.*

1.6. RESERVE ESTIMATES

The Ore Reserves for the Tulkubash Gold Project have been updated according to the code prescribed by the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ('the JORC Code'), 2012. The Ore Reserves have been estimated by considering only the Measured and Indicated Mineral Resources that can be exploited economically. The Ore Reserve estimate has been based on the latest geological block model, which included processing recovery data in each of the 5 m by 5 m by 5 m blocks that informed the pit optimisation and subsequent final open pit design.

The 2020 EOY Ore Reserve is based on a revised Resource model which incorporates the results of exploration drilling up to the end of 2020, a new geological interpretation, and technical and economic parameters established in the 2019 BFS or modifications based on subsequent work.

The 2020 EOY Ore Reserve estimate is stated in Table 1-4, which reports a contained gold content of 571 koz, all of which have been categorised as Probable.

Table 1-4 Tulkubash Ore Reserves as at 2020 EOY

Category	Quantity (Mt)	Grade (g/t)	Content (kg)	Content (koz)
Proven	-	-	-	-
Probable	20.9	0.85	17,760	571
Total	**20.9**	**0.85**	**17,760**	**571**

Source: Chaarat, 2021

Notes:

1. *This statement of Ore Reserves has been prepared by Mr Peter C Carter, an independent consulting mining engineer, based on a review of work performed by Chaarat Gold technical staff;*
2. *Mr Carter is a member of the Association of Professional Engineers and Geoscientists of British Columbia and is qualified as a Competent Person under the JORC Code, 2012;*
3. *The Ore Reserve has been reported in accordance with the classification criteria of the JORC Code, 2012 and is 100% attributable to Chaarat;*
4. *Any apparent computational errors are due to rounding and are not considered significant;*
5. *Ore Reserves are reported with appropriate modifying factors of mining dilution (8%) and mining recovery (97.5%);*
6. *Ore Reserves are reported at the head grade delivered to the leach pad;*
7. *The Ore Reserves are stated at a price of USD1,450/tr oz as at 2020 EOY;*
8. *Although stated separately, the Mineral Resources are inclusive of Ore Reserves;*
9. *No Inferred Mineral Resources have been included in the Ore Reserve estimate;*
10. *Quantities are reported in metric tonnes; grades are reported in grams per metric tonne = ppm (parts per million);*
11. *The input studies are to the prescribed level of accuracy; and*
12. *The Ore Reserve estimates contained herein may be subject to legal, political, environmental or other risks that could materially affect the potential development of such Ore Reserves.*

Table 1-5 provides a comparison of the 2020 EOY Ore Reserve to the previously reported 2018 EOY Ore Reserve. This Shows that the 2020 EOY Ore Reserves represent a 6% decrease in ore tonnage and an 8% decrease in grade compared to the 2018 EOY Ore Reserves. Overall, these changes result in a 13% decrease in contained ounces of gold.

The Inferred Resources within the pit limits, which are currently treated as waste, offer the potential to increase ore tonnage and contained ounces, along with decreasing the Strip Ratio (t:t) in the order of 5% to 10%.

Table 1-5 Comparision of Tulkubash Ore Reserves as at 2018 EOY and 2020 EOY.

Parameter	Units	2018 EOY	2020 EOY	Variance
Ore	Mt	22.2	20.9	-6%

Parameter	Units	2018 EOY	2020 EOY	Variance
Grade (Au)	g/t	0.92	0.85	-7%
Metal (Au)	koz	658	571	-13%
Waste	Mt	58.6	54.1	-8%
Total	Mt	80.8	74.9	-7%
Strip Ratio	t:t	2.64	2.59	-2%
Recovery	%	68.9	73.6	7%
Recovered Au	koz	453	419	-7%

Source: Chaarat, 2021

1.7. Mining Methods

The Tulkubash open pit forms part of a near-vertical mineralised lode system located in mountainous terrain and the Tulkubash 2020 EOY open pit design is composed of three separate pits arranged along the strike of the orebody over 2 km. The pits are situated in steep, mountainous terrain at elevations of 2,300 masl to 2,800 masl. The deposit is divided up into two zones: the Main Zone and the Mid Zone.

The hydrogeology for the open pit designs has been informed by field investigations conducted by SRK Consulting and Tetra-Tech Engineering in 2010 and 2014 respectively. Based on this earlier work, a finite-element groundwater model was developed by Wardell Armstrong International (WAI) in 2017. This has forecast discharge rates of between 4 m^3/hr and 6 m^3/hr (or 1.0 ℓ/s and 1.5 ℓ/s) at depths correlating to approximately 2,500 masl.

Kyrgyzstan is a seismically active region and studies have been conducted to establish the technical parameters which appropriately reflect seismic conditions at the site. The primary criteria is Peak Ground Acceleration (PGA) which was determined to be 0.157 G based on a 10% probability in a 50-year return period. The pit design approach has been strongly informed by the multiple interacting joint sets which form a highly blocky rock mass. As a result, structural failure risks including planar, wedge and toppling mechanisms can be expected in planned open pit. In order to mitigate this, it is important that careful blating practises are deployed in all phases of the mine planning process, particularly in the final bench configurations. Slope stability analysis has targeted a minimum factor of safety (FoS) of 1.2 for the inter-ramp angles (IRA), and 1.3 for the overall pit slopes. This work demonstrated that in all instances, the FoS remains above 1.3. As expected, the FoS was reduced by hydro-geological influences. Mitigation of this risk can if necessary be addressed through water management practices including horizontal drainage and pumping.

Geotechnical design criteria has considered bench face angles in the final designs varying between 60° and 75°, with 8 m berm widths to comply with local regulations and to allow mechanised cleaning. Inter-ramp angles (IRA) of around 51° and 58° were applied to the different design sectors. The IRA for the fault zone area was reduced to 45°.

Mine planning of the open pit was based on the 07 November 2020 Mineral Resource model which was re-blocked to the parent block size of 5 m by 5 m by 5 m for the pit optimisation and subsequent final open pit design, sequencing and scheduling. These dimensions appropriately simulated the planned 5 m excavation lifts.

The mine design was guided by the results of a pit optimisation exercise using suitable software which deploys the Lerchs-Grossman algorithm to generate a series of nested pit shells. From these results it was possible to identify optimal pit locations and geometries which were used with the pit design criteria to complete the pit design and enable the declaration of an ore reserve.

The Main Zone Pit is approximately 1.3 km in length and situated at the southwestern end of the mining area. It is the single largest pit accounting for over 90% of the reserve by both tonnage and contained gold. The Main Zone Pit hosts a reserve of 19.4 Mt ore grading 0.86 g/t Au, containing 538 koz Au. Associated with the ore is 50.4 Mt of waste resulting in a strip ratio of 2.6:1 (t:t). Overall, the final highwall ranges between 250 m to 300 m in height.

The Mid Zone Pit design is composed of two separate small open pits. These pits are arranged along strike length about 150 m northeast of the Main Zone Pit. The Mid Zone accounts for approximately 7% of the reserve by tonnage and 6% of the contained gold. The Mid Zone Pits host a reserve of 1.4 Mt ore grading 0.72 g/t Au, containing 33 koz Au. Associated with the ore is 3.7 Mt of waste resulting in a strip ratio of 2.6:1 (t:t).

Haul roads connecting the open pit area to the Sandalash River Bridge and the waste dump will be constructed during pre-production. The deposit will be developed and mined using conventional hard rock open pit mining techniques. All topsoil, vegetation and organic material will be cleared and deposited in designated stockpiles (SP) to be used in the future for rehabilitation and mine closure. Where possible, existing roads will be used to move equipment into the mining areas. Steady state production benches will be at least 25 m wide with 5m drilling and blasting of 5 m production benches. Once steady state mining conditions have been established in the initial pit areas, a continuous sequence of access development and bench development will follow with lateral development along the orebody strike.

The mining plan calls for 4.6 years of production mining preceded by 13 months of pre-production stripping, a total of 68 months. Total mined tonnage over the LoM, including pre-stripping, is 74.9 Mt with an average mining rate of 13.0 Mtpa or about 37,000 tpd. The mining rate peaks in 2025 at 18.5 Mtpa or about 53,000 tpd. At steady-state, annual ore production is 13.5 ktd or 4.92 Mtpa. The LoM schedule provides for contained metal of 571.1 koz of gold and 845.7 koz of silver respectively. The ore process rate at full production is 4.9 Mtpa and stockpiling is practised where mining rates exceed this figure.

An average Strip Ratio of 2.59 (t/t) has been planned over the LoM.

Waste rock will be stored on a Waste Rock Dump (WRD) in the adjacent Irisay Valley west-southwest of the mine area and used to backfill a portion of the mined-out pits.

The open pit mining operation will operate continuously for 350 days per year, with ten days lost due to bad weather or supply-related issues. The mining crews will work 12-hour shifts on a 15-day rotation (15 days on, and 15 days off), rotating cycle to facilitate a continuous operation.

All of the material to be mined from the open pits will require blasting prior to loading. Surface crawler-type drill rigs will drill 5 m benches with controlled final wall perimeter blasting. Blasting will be accomplished with shock-tubes (i.e., non-electric detonation) and ANFO. A maximum of five drill rigs will be required to achieve these production targets.

Digging and loading will occur on 5 m lifts to match the height of the working face to the size of the equipment and to facilitate digging selectivity when separating ore and waste. Smaller excavators with hydraulic rock breakers will be used to clean walls and break oversize rock at the face to maximise excavator loading productivity. FELs will be used to support the primary excavators. Simulation studies have confirmed that four to five excavator units will be sufficient to achieve the planned mining schedule, if supported by a single FEL during the mining plan for peak production during 2024 to 2026.

All operational hauling for RoM and Waste has been planned using Mercedes Actros 3,340 dump trucks with a capacity of 34.5 t. Haulage fleet simulation has forecast a maximum truck haulage fleet of 72 trucks will be necessary in order to support the production peak forecast during 2024. The haul roads will be 15 m wide inclusive of berm, ditches, and carriageway. This will permit dual-lane traffic. Truck haulage routes will negotiate an average 4.3 km route before crossing the Sandalash River bridge from where they will continue to haul a further 5.6 km to the RoM Pad.

Effective grade control management remains a crucial part of the mining strategy and will involve the sampling of blasthole cuttings after drilling. These will be assayed for gold, silver, carbon, sulphur, and cyanide solubility.

General groundwater inflows will be managed through pit sumps for onward pumping via pipelines, to a holding pond from where the water can either be used for dust suppression or discharged.

Mine Operations will conduct maintenance on the mining equipment fleet so that sufficient equipment hours are available to meet safety standards and production requirements on an ongoing basis. Average equipment availability over the LoM is planned to be 85%.

A Mining Contractor will be employed to hire the workforce, train operators, provide mining equipment, and conduct all of the activities necessary to meet the planned production targets. The contract will also cater for the housing and feeding of all mining personnel. It is estimated that the Mining Contractor will employ a maximum of 524 persons with an average of 365.

The Owner's team will consist of 22 positions, with about half of these being associated with grade control activities.

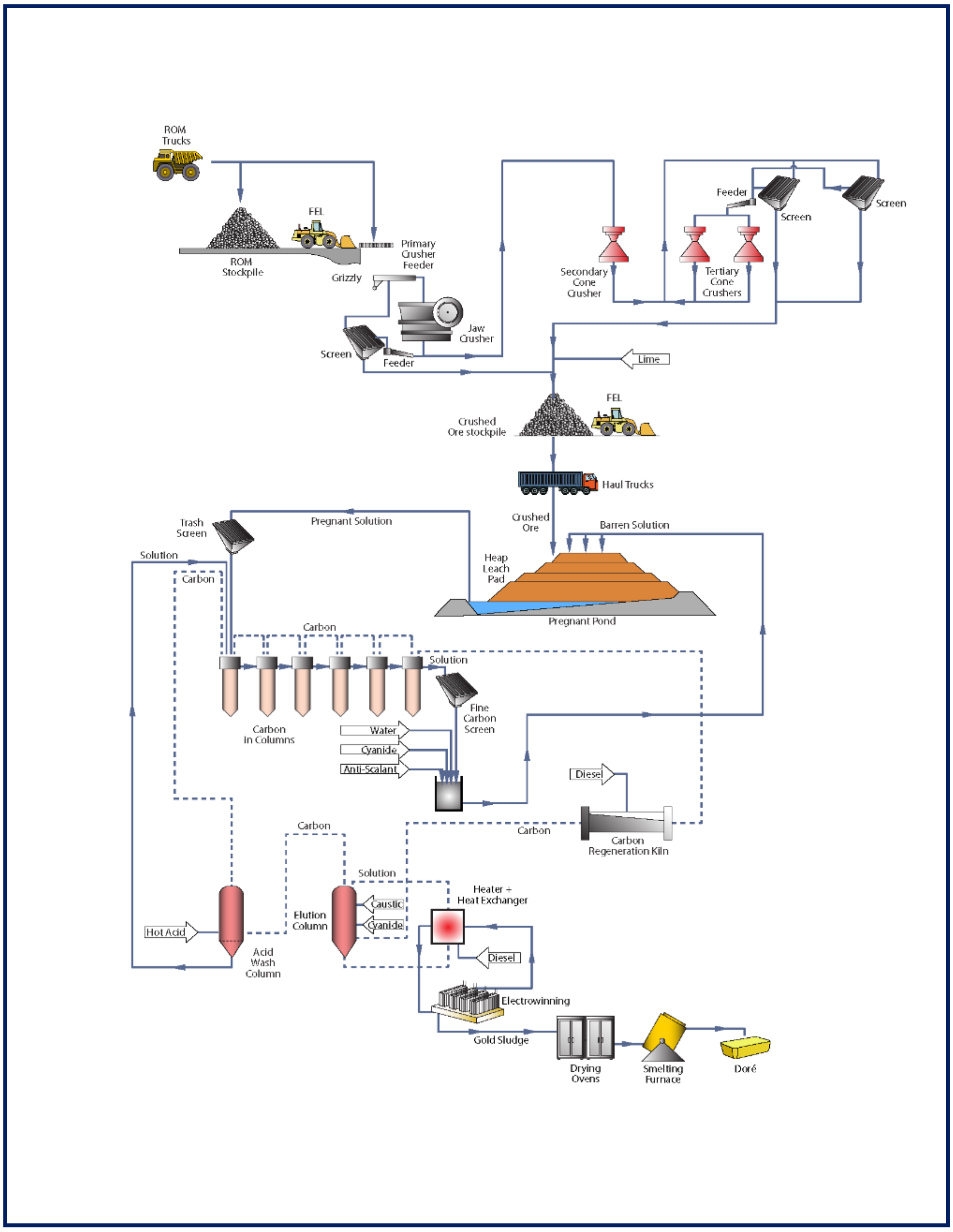
1.8. RECOVERY METHODS

The Project process design is based on the testwork presented in Section 13. A successful process design is one that results in a flowsheet that is as simple as possible to supply, operate, and maintain, whilst at the same time maximising gold and silver recoveries and minimising power requirements.

Ore that is suitable for heap leach processing is defined as any material defined within the selected pit shell and which has a total sulphur content of 0.5% or less (S_{TOTAL} ≤0.5%), and is above the cut-off grade of 0.2 g/t.

Figure 1-2 shows a conceptual block flow diagram of the of proposed process facility.

FIGURE 1-2 CONCEPTUAL BLOCK FLOW DIAGRAM



A conventional three-stage crushing circuit will crush the run-of-mine (ROM) ore to a P_{80} of 12.5 mm at a rate of 13,500 t/d. Trucks will haul the crushed ore to the heap leach pad where it will be stacked in a permanent multi-lift heap leach, with a 7 m stack height per lift.

The lifts will be irrigated with a dilute cyanide solution at a rate of 10 ℓ/m^2/hr to dissolve the gold and silver from the ore into the solution. Once the solution reaches the base of the heap, it will flow to the pregnant solution pond. From there it will be gravity fed to the ADR plant for gold and silver recovery. The precious metals from this pregnant solution will adsorb onto granular activated carbon in the carbon columns of the ADR plant. The barren solution discharged from the carbon columns will be recirculated to the heap leach pad, after dosing with the required amount of cyanide to make up for depletion.

The loaded carbon will be pressure stripped with a hot caustic solution to re-dissolve the precious metals into an eluate solution. The eluate will be treated using conventional electrowinning to produce gold-rich sludge suitable for direct smelting on site into gold Doré. Gold Doré bars will be transported off-site to a suitable refinery.

At the end of its production life, the heap leach pad will be rinsed with water to ensure environmental compliance.

The LoM gold and silver recoveries are calculated to be 73.6% and 63.4%, respectively.

1.9. Project Infrastructure

The Project will require the development of several new infrastructure items, in addition to those already existing.

The locations of the project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and to ensure efficient and convenient operation of the mine haul fleet.

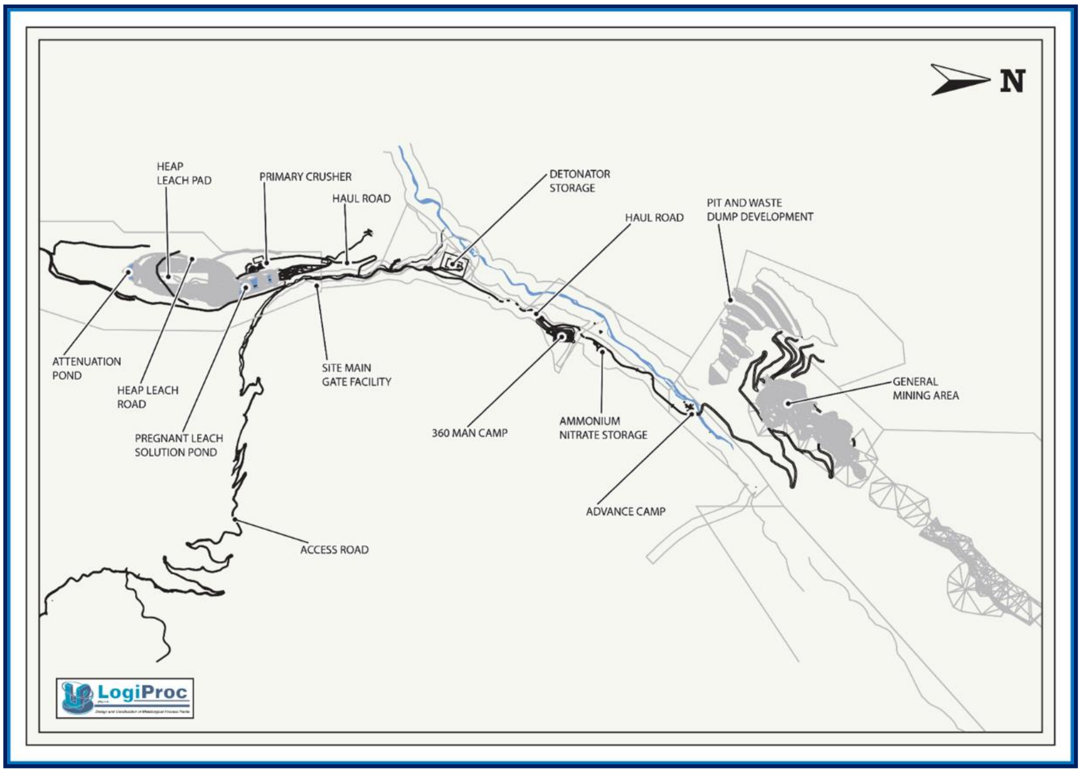
The following Project infrastructure and facilities will be included on site:

- Some off-site facilities already exist (e.g. Chatkal Station and Kumbel Pass Checkpoint). However, the Kumbel Pass to Site Gatehouse road is being upgraded.
- Mining and related facilities constructed by the Mining Contractor, including Main Ore Haul Road, Pit Roads, Ancillary Roads and Platforms, Detonator storage, Ammonium Nitrate Storage and temporary Mine Maintenance Workshop.
- Camp facilities - a 360 modular Man Camp, local diesel power generation and distribution, Water storage and reticulation and sewage treatment plant. A portion of this facility has been constructed, delivered and installed.
- General facilities in the Processing (Dry Valley) area including – Gatehouse with weighbridge, Emergency Response Team (ERT) room, and Process area roads.
- Crushing Area including – ROM pad, Primary Crusher, Bypass screen circuit, Secondary and Tertiary crushers, Screenhouse, Conveyors, Lime addition, Fine Ore Stockpile and Truck Load-Out.
- Heap Leach Facility comprising - a phased lined heap leach pad with underdrain system and collection pipes, Pregnant Leach Solution Pond (PLS), PLS Overflow Pond, Emergency Stormwater Pond, Attenuation Stormwater Pond, Sedimentation Pond, Perimeter Access Roads and Stormwater Diversion Channels.
- Gold process area including - secure ADR plant, goldroom, reagent mixing facility, and reagent storage facility, plus ancillary infrastructure including - administration offices, clinic and laboratory.
- Power Station including - power generation, substations, Fuel Farm, internal utilities, MV site wide distribution and area E-Houses.

- General Processing infrastructure including – water supply (bore), raw and fire water distribution, workshop/warehouse, ADR gatehouse, area offices, area process control systems.
- Temporary and permanent facilities such as mobile crusher, Batch plant, laydowns, borrow pits, temporary and permanent stockpiles.

An overall site layout for the Tulkubash Gold Project is shown in Figure 1-3.

FIGURE 1-3 TULKUBASH GOLD PROJECT SITE LAYOUT



1.10. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project is in compliance with Kyrgyz legislation for environmental and social aspects but when mining operations start, further work will be required to ensure that the Project continues to comply with state legislation and international best practice.

An Environmental and Social Impact Assessment (ESIA) was completed in 2020. This will inform the development and implementation of a comprehensive Environmental and Social Management System (ESMS) to govern the management and monitoring of measures identified by the ESIA as necessary to mitigate project impacts. The closure plans were developed to international standards as part of the ESIA process to support the implementation of commitments made in the LoM and after mine closure.

Chaarat will develop final decommissioning, reclamation, and closure plans as the Project progresses to steady state production. A framework mine closure and rehabilitation plan has been developed to international standards as part of the ESIA process and should be continuously updated as the Project progresses.

Chaarat has developed an Emergency Preparedness and Response Plan (EPRP) for the Project. The HSE Manager will manage environmental health and safety units. HSE department will consist of safety engineers, environmental engineer and specialists, avalanche and rescue team and medical team.

The ESMS outlines the environmental and social monitoring that Chaarat will implement. Noise impacts on local communities associated with peak operations of the Project are predicted to be significantly below noise levels recommended by the International Finance Corporation (IFC), the World Health Organization (WHO), and Kyrgyz Republic guidelines. Main air emissions are expected to be dust from haulage vehicles operating on unpaved roads and combustion emissions from site operations and haulage vehicles. The ESMS includes several mitigating measures that will be implemented by Chaarat.

1.10.1. Socio-economic

The Project's area of socio-economic influence stretches across Jalal-Abad and Talas regions. This influence is strongest at the five settlements within Chatkal District, including the village districts of Kanysh-Kiya and Chatkal.

Chaarat has participated in both stakeholder dialogue and community development programmes. At their main office in the Chatkal Valley, located in Kanysh-Kiya, Chaarat hosts stakeholder meetings and maintains a grievance register. Chaarat operates two shops in the area selling products at a lower cost than most shops in the valley.

Due to the distance of the nearest communities from the Mine site, any negative socio-economic impacts to the Mine are likely to be limited. It is anticipated that the positive impacts from employment and community investment on the local communities will be significant.

1.10.2. Geochemistry

Preliminary geochemical characterisation studies on the Chaarat rock types have identified tectonic breccia as the only high-risk material in terms of potential acid rock drainage and metal leaching (ARD-ML). Based on the 2019 BFS this material makes up approximately 8% of the total pit volume, although the mine block model predicts that only approximately 51,000 tonnes of potentially Acid Generating (PAG) tectonic breccia material within the waste rock will be mined. The bulk of the waste rock is the largely benign Tulkubash Sandstone, which makes up 67% to 75% of the rock to be excavated, together with other siliceous/carbonate-rich lithologies.

As the cyanide heap leach process does not tolerate a high sulphur content, most of the PAG material from the deposit will be directed to the WRD, although some sulphide ore may be stored in a low-grade ore stockpile that may be established near the WRD.

Geochemical impact assessment has therefore identified possible acid rock drainage from PAG rock present in the pit walls; the waste dump and the low-grade ore stockpile. In the 2019 BFS, the risk is considered low given the 63 Mt of Non-Acid Generating (NAG) plus some specifically neutralising rock available to buffer any acid generating material.

Metal leaching may be more of an issue given that test work has indicated that some metals are soluble even in neutral water. The proposed ARD-ML management strategy is to intermix the PAG tectonic breccia material with NAG waste rock. Since PAG material is expected to be mined later in the mine plan, there is time to further define the geochemical risk, and if required, encapsulate the PAG within the WRD.

Whilst water monitoring samples from areas where there has been mining activity at the site have not shown evidence of ARD, they have shown some elevated metal levels, especially of arsenic and antimony, which suggests that metal leaching occurs when rock is disturbed, even in neutral to alkaline water.

It is also noted that test work to date has been completed on a limited number of samples that may not be a good representation of the rock to be excavated during the operation. Further studies are envisaged to ensure that the ARD-ML risk and PAG amount has not been underestimated.

1.11. WATER MANAGEMENT

1.11.1. GROUND WATER

Initial groundwater levels in the footprint of the Tulkubash pit are expected to be in the range of 2,300 masl and 2,500 masl. Initially, there will not be any groundwater inflow. When mine development reaches level 2500 masl during the third year, dewatering will be managed as indicated in Section 1.7.

However, depressurisation may be required to ensure that the pit walls are not subject to stability issues resulting from saturation. Groundwater levels in the high wall will be monitored and will most likely focus on the northwest wall where pre-mining groundwater levels are highest. A programme of installation of piezometers to monitor groundwater pressure conditions and the impact of the drains is envisaged. In addition, visual observation of seepage areas during excavation will indicate where additional horizontal drains are necessary.

GSPA (a local contractor) completed a set of ground water measurements in both the Dry Valley and Camp Area. Several months ground water measurements and other engineering analysis were done, and is currently under expertise approvals within Kyrgyz Government authorities. The elevation of the Sandalash Valley below the Dry Valley is approximately 2,150 masl, so given the likely high permeability of the rubbly infill there, it is likely that any groundwater recharge migrates towards a water table controlled by the geometry of the base of the infill and the level of the Sandalash River.

1.11.2. SURFACE WATER

The upstream catchments of the open pit mine and associated satellite pits will be diverted away from the pits to avoid inflows that would negatively impact mining operations, and to minimise the generation of contact water within the open pit.

For the HLF, surface water management measures include the following:

- The eastern collection drainage channel will collect surface water run-off from the east catchment slopes and divert it mainly to sediment pond 1 (and partly to the attenuation pond);
- The southern catchment channels will collect surface water run-off from the south catchment slopes mainly to the attenuation pond;

- The attenuation pond of 51,700 m^3 capacity at the south of the HLF will collect run-off from the catchment to the south and southwest of the HLF prior to discharge to settlement pond 2;
- Underliner drainage will collect and divert groundwater below the HLF; and
- A main collection pipe connecting the attenuation pond at the south of the HLF to settlement pond 2 at the north.

Catchments on the south side of the Sandalash River include those that surround other project infrastructure, such as the 360 Man Camp and the mine maintenance workshop.

For design flow purposes, 1-in-100, 24-hour flows were calculated for each of the sub-catchments to allow for the sizing of any diversion drains or road culverts to carry the design run-off.

1.11.3. Water Supply

Two pumping stations will supply raw water to the site:

- Boreholes, located west of the accommodation camp, will supply the accommodation camp and Mine Contractor`s Vehicle Maintenance Shop with raw water all year round.
- Boreholes to be located nearby Kumbeltash Stream, east side of the ADR plant, will supply the ADR plant, the crushers, administration and laboratory buildings and the gate house complex with raw water all year round.

1.12. Capital & Operating Cost Estimate

1.12.1. Capital Cost Estimate

1.12.1.1. Summary

The total estimated initial capital cost for the, construction, installation, and commissioning of all facilities and equipment is USD115.5 M with USD15.2 M deferred to when required. This cost estimate is consistent with a BFS accuracy range of –10% to +10%. This 2021 BFS shows an accuracy improvement over the 2019 BFS (-10% to +15%) due to the noted increase in designs completed, and the actualised costs that have made the cost estimations more accurate. This equates to an AACE Class 3 estimation.

LogiProc, as the lead consultant, developed the capital cost estimate with inputs from Chaarat. Table 1-6 presents the responsibility breakdown by area.

Table 1-6 Estimate Responsibility Matrix

Area	Company
Mining (Open Pit)	Chaarat
Processing	Chaarat / LogiProc
Infrastructure	Chaarat / LogiProc
Site Facilities	Chaarat / LogiProc
Indirect Costs	Chaarat
Owner’s Costs	Chaarat

Area	Company
Allowances	Chaarat / LogiProc

A proportional breakdown of the initial capital cost is provided in Table 1-7.

TABLE 1-7 INITIAL CAPITAL COST SUMMARY

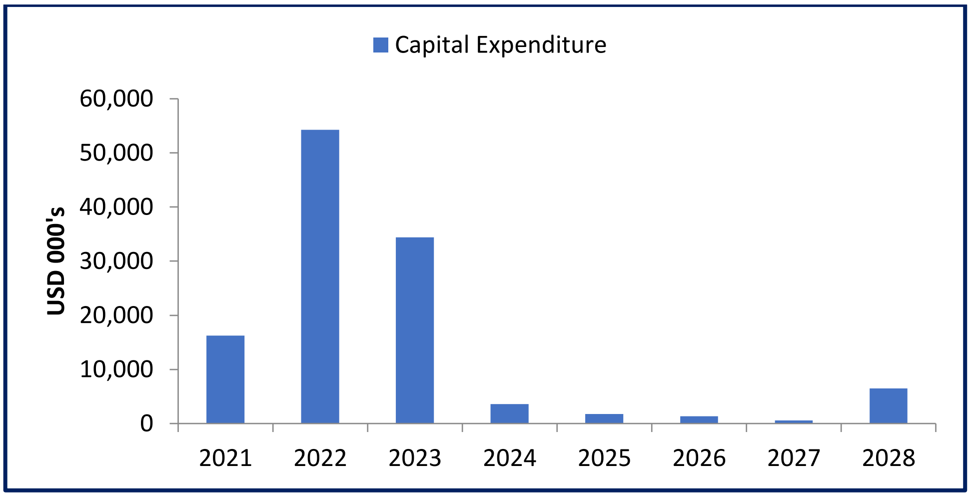
Area	Total (USD'000)
Mining	21,996
Infrastructure	4,236
Process Plant	59,807
Owners Cost	18,913
Contingency	10,496
Total Initial Capital Cost	**115,446**

This estimate includes the direct field costs required to execute the project, plus indirect cost overheads, commercial requirements, and management. This estimate is based on pricing in real 2021 terms, with no allowances for inflation or escalation for future periods. Amounts in this capital cost estimate are expressed in United States Dollars (USD), unless otherwise noted.

While a few of the estimates are based on information from the previous BFS in real 2019 terms, the majority of the quotations used in this updated BFS were obtained in March 2021.

Graph 1-1 provides a forecast of the capital expenditure over the LoM.

GRAPH 1-1 CAPITAL EXPENDITURE FORECAST OVER THE LOM

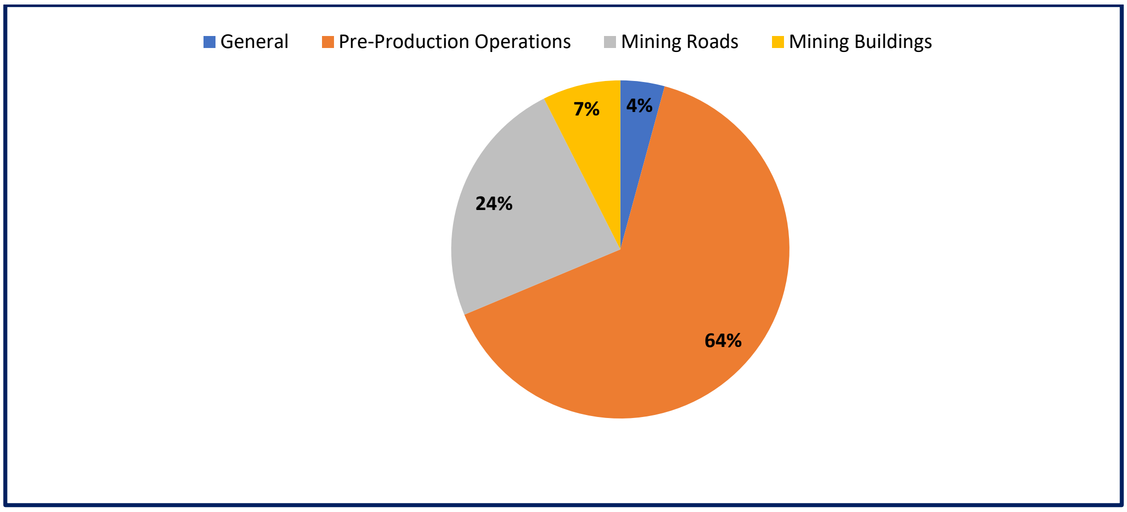


The Initial Cost estimate excluding the 10% contingency is divided into four categories: Mining, Infrastructure, Process Plant, and Owners Costs

- Mining (USD22.0 M)

Mining includes all pre-production cost items related to the mining site and activities, which include but are not limited to mobilisation of mining equipment, Pit and Waste dump development, Mining Roads, and Mining Buildings. Graph 1-2 illustrates the split in the mining capital cost estimate.

GRAPH 1-2 MINING CAPITAL COST ESTIMATE SPLIT

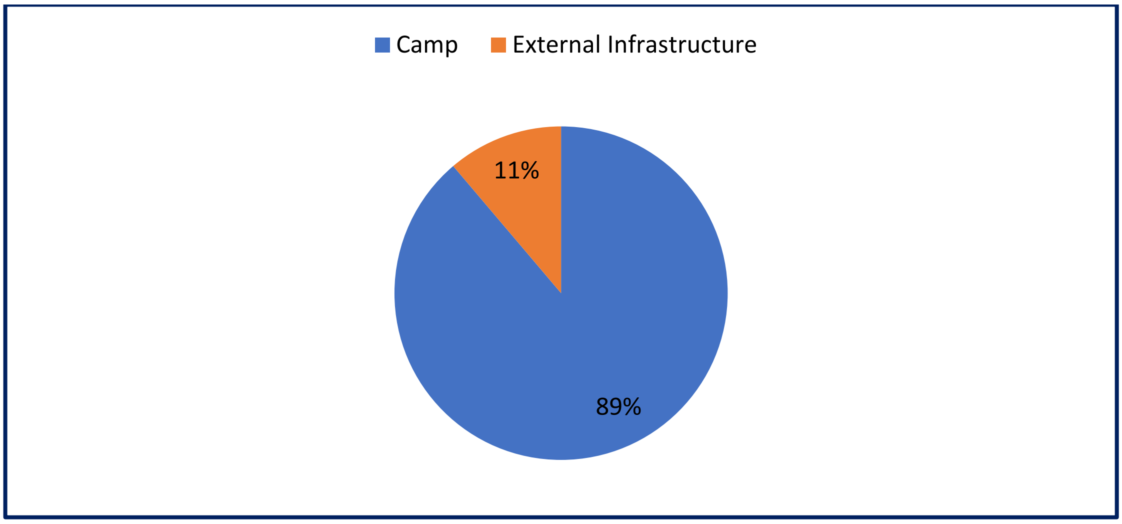


- Infrastructure (USD4.2 M)

The Infrastructure includes mainly the 360 Man Camp and the upgrade of the Kumbel pass to Tulkubash road.

The 360 Man Camp capital estimate is based on actual costs from the first phase construction. The 360 Man Camp will be a self-contained, multi-building facility that includes accommodation, mess, ablution, recreation and laundry services. Graph 1-3 illustrates the split in the infrastructure capital cost estimate.

GRAPH 1-3 INFRASTRUCTURE CAPITAL COST ESTIMATE SPLIT

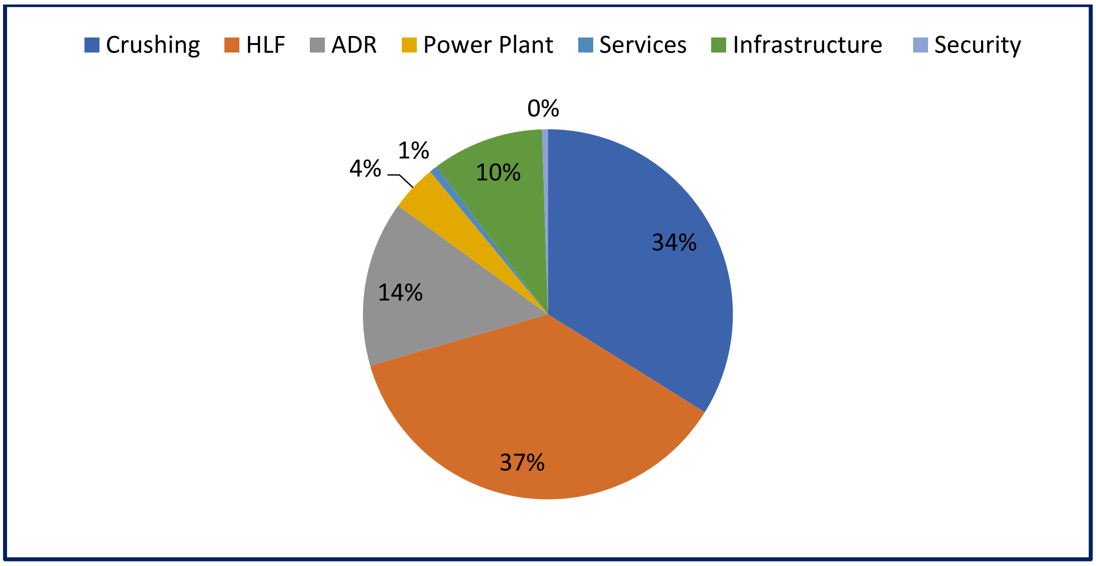


- Process Plant (USD59.8 M)

The Process Plant includes all cost items related to the process plant site, which include but are not limited to all process equipment, ie Crushing, Heap Leach, ADR, and Power Station, as well as Services (eg water), Infrastructure (eg Process area

Buildings and Roads), and Security. Graph 1-4 illustrates the split in the process plant capital cost estimate.

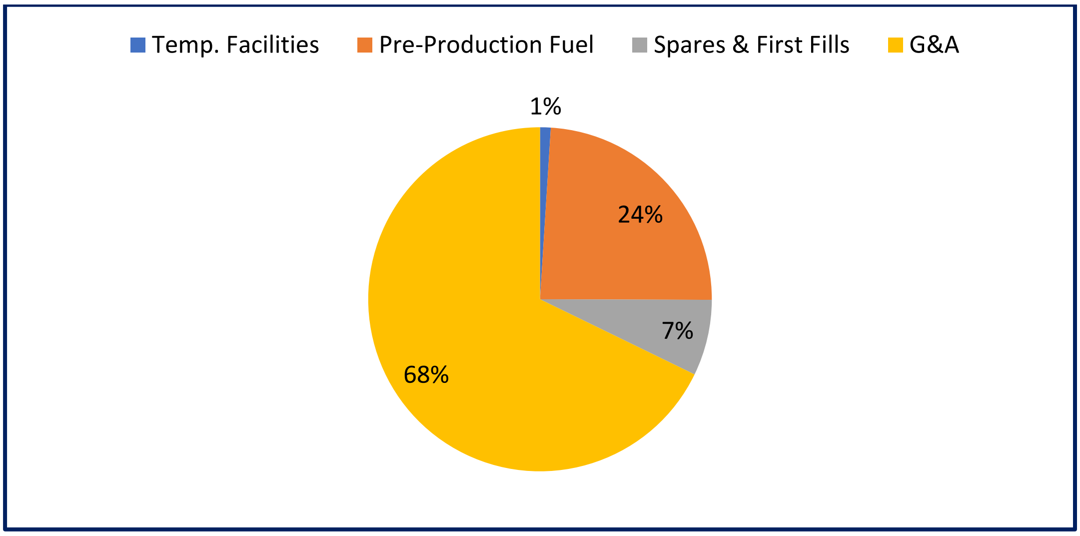
GRAPH 1-4 PROCESS PLANT CAPITAL COST ESTIMATE SPLIT



- Owners Cost (USD18.9 M)

Owners Costs includes all cost items related to temporary facilities, pre-production fuel, Spares & First fills, G&A. Graph 1-5 illustrates the split in Owners Cost Capital estimate.

GRAPH 1-5 OWNERS COST CAPITAL ESTIMATE SPLIT



Deferred costs cover Heap Leach Phases 2 and 3, Mine Closure and Deferred Equipment.

The Heap Leach will be constructed in 3 phases during the mine life. Phase one will form part of the initial capital cost and phase 2 to 3 will from part of deferred costs.

Construction phases:

- Phase 1 – 6.48 Mt, constructed in year 2021-2023; cost of USD8,490 M.
- Phase 2 - 10.29 Mt, constructed in years 2024-2025; cost of USD4,346 M.
- Phase 3 - 9,11 Mt, constructed in years 2026-2027; cost of USD1,938 M.

Once the mining operation has ceased and the remaining ore from the mine and ROM pad has been crushed and placed on the HLF for final leaching and rinsing, machinery and personnel will be reassigned to complete the earthworks required for mine closure. An estimate of USD6.5 M (including taxes) has been assigned for the labour and operating costs for the HLF during the flushing, drainage, and rehabilitation stages of the closure plan.

One tertiary crusher has been deferred; the plant will commence with two tertiary crushers, which will crush the ore to a P_{80} of 12.5 mm.

A Contingency of 10% is allowed for to cover uncertainties in both the Initial and Deferred Capital estimates. Such uncertainty could arise from interpretations related to VAT, Import Duties, escalation, foreign exchange, or undefined items that cannot be explicitly foreseen or described at the time the estimate is completed due to the lack of complete accurate and detailed information. The total initial capital contingency allowance is USD10.5 M, which is 10% of the initial capital cost estimate. The total deferred capital contingency allowance is USD1.4 M, which is 10% of the deferred capital cost estimate.

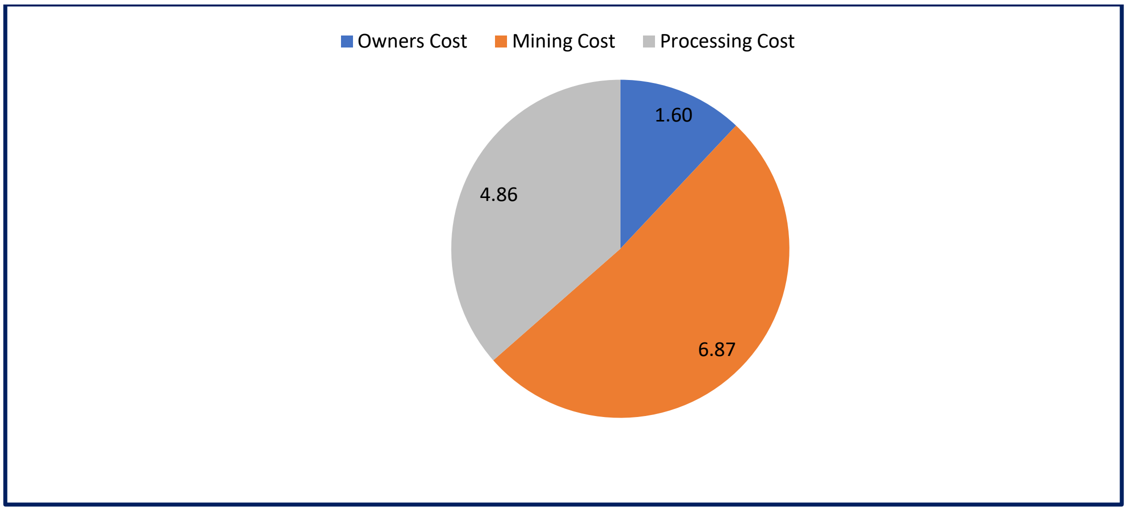
1.12.2. Operating Cost Estimate

The total operating cost estimate over the LoM (for this section LoM excludes pre-production as the pre-production costs are capitalized) is shown in Table 1-8 and operating unit costs are shown in Graph 1-6.

Table 1-8 LoM Operating Cost Estimate

Area	Total (USD 000's)
Owners Cost	32,456
Mining Cost	139,301
Processing Cost	98,579
Total LoM Operating Cost	**270,336**

Graph 1-6 LoM Unit Operating Costs (USD/t ore)



The mine cost comprises 52% of the total operating costs, whilst processing and owners costs make up 36% and 12% respectively.

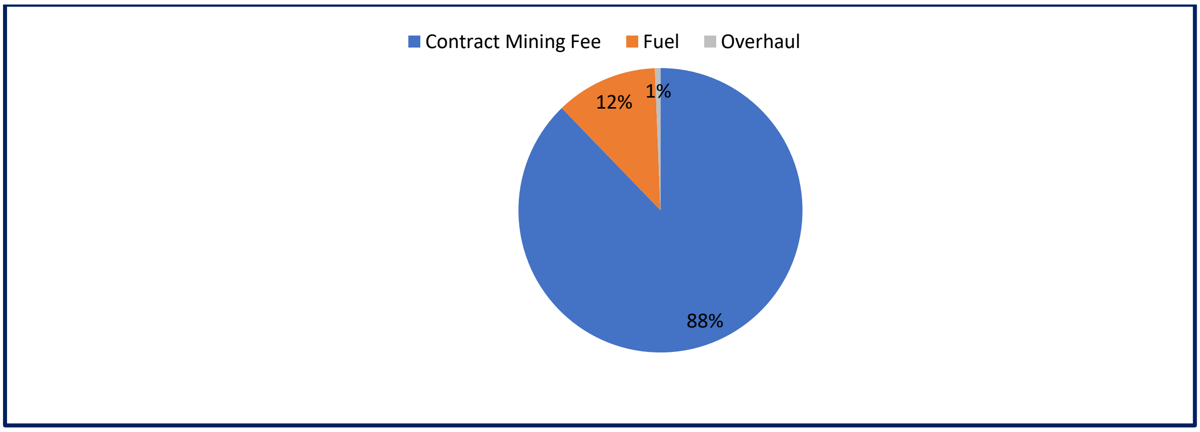
Owners costs comprise expenses related to the owners mining team (USD5.9 M) and general and administration costs for the operation of the site and for offices in Bishkek (USD26.6 M). The combined cost to the owner over the steady state period is USD32.5 M.

Mining costs are based on the production schedule, contract mining rates and an projected fuel price of USD0.60/ℓ.

The Contractor's mining fee is informed by agreed unit rates based on the cumulative material mined and hauled. The cost to supervise mining activities is included in the contract costs, but grade control, technical services, and management of the Contractor is provided for in the Owners mining cost.

The total contractors mining costs for the LoM, excluding the pre-production, amounts to USD139.3 M. This is composed of USD122.3 M for contract mining, USD16.2 M for fuel, and USD0.9 M for overhaul. Graph 1-7 illustrates the split in Contractor's Mining Cost over LoM.

Graph 1-7 Contractor's Mining Cost Split

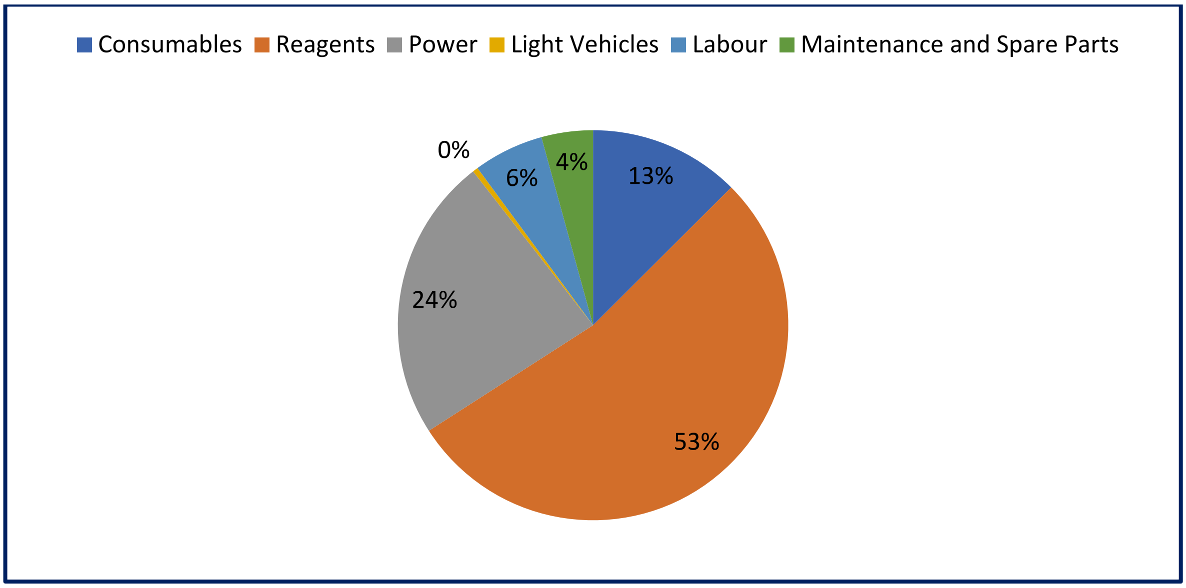


The total process operating cost for treating the oxide ore is estimated at USD4.47/t ore. The LoM cost will amount to USD91.4M.

The stacking cost for the heap leach is USD12.1M

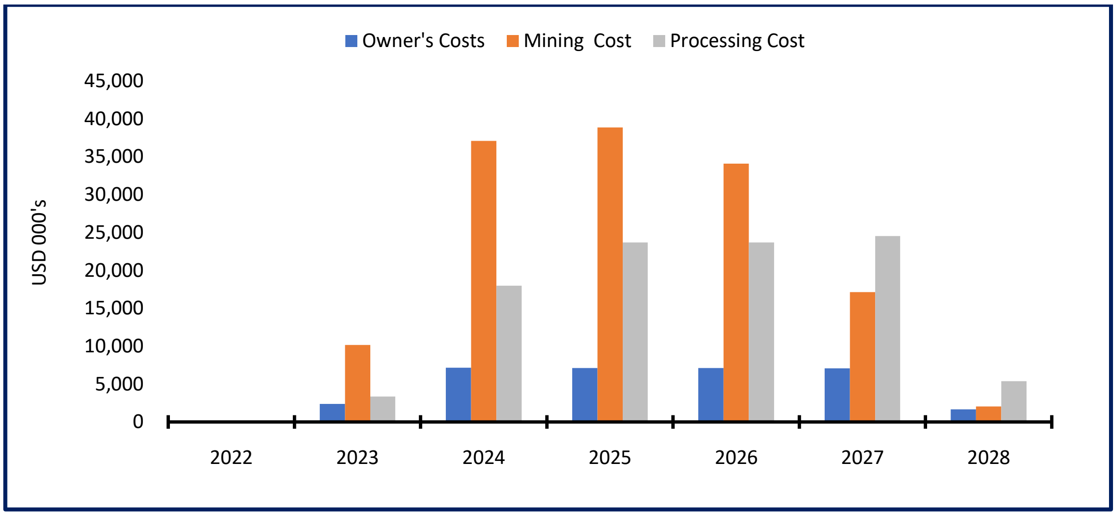
The process operating cost summary for the ore is presented in Graph 1-8.

GRAPH 1-8 PROCESS OPERATING COST SUMMARY



Graph 1-9 illustrates the operating cost forecast over the LoM, Pre-production operating costs are shown as zero as they will be capitalised.

GRAPH 1-9 LOM OPERATING COST FORECAST



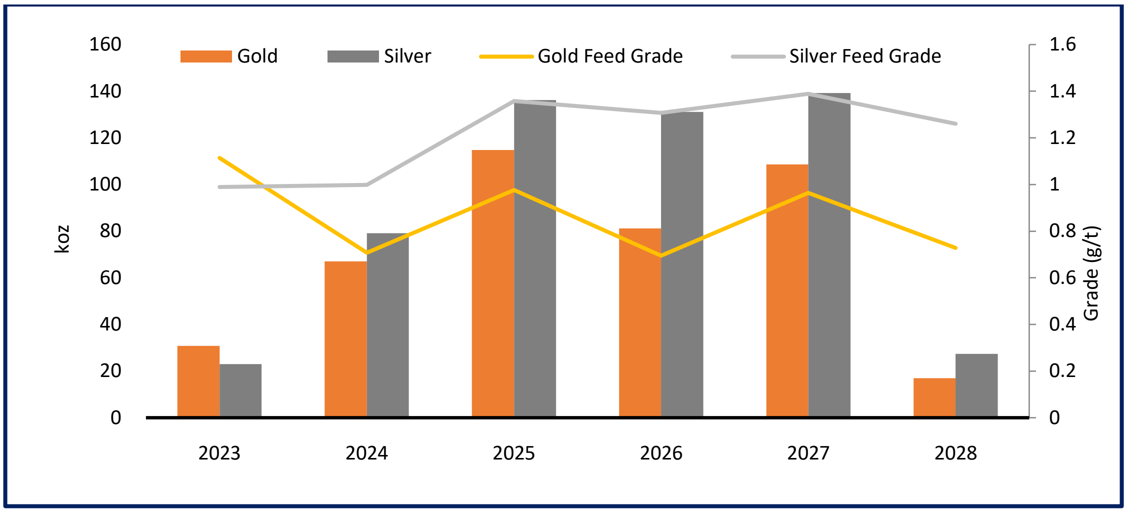
1.13. ECONOMIC ANALYSIS

The economic analysis has been accomplished through the construction of a Discounted Cash Flow (DCF) model based on the planned production data as set out in the LoM plan, with due regard to appropriate financial model inputs and reasonable assumptions informed by Chaarat and the Competent Persons responsible for the Mineral Resource and Reserve Statements. Metric units are used throughout this economic analysis and unless otherwise stated, monetary values are stated in United States Dollars (USD). The DCF model was established on a 100% equity basis, excluding debt financing and loan interest charges. It is not a complaint valuation of the project in terms of any of the international valuation codes for public reporting.

Its purpose is to assess the robustness of the project and to confirm the economic viability of the ore resources as stated herein.

The revenue forecast has been based on the mining of 20.9 Mt of gold bearing ore containing 418 koz of recoverable gold and 446 koz of recoverable silver. The metal has been sold at a gold price of USD 1,450/tr oz and a silver price of USD 17.50/tr oz. The forecast metal production over the LoM is illustrated in Graph 1-10.

GRAPH 1-10 COMMODITY RECOVERIES AND ASSOCIATED GRADE



The Gold *Doré* sales incur a State royalty together with refining and transport costs. These were subtracted from the gross revenue to establish a value for the metal sold at the mine gate.

The operating, capital and closure costs as described in this report have been used for the economic analysis. The average unit operating cost derived directly from the LoM is USD 13.59/t ore (in real terms). The average capital expenditure over the LoM is USD 6.21/t ore (in real terms), including a 10% contingency. This also includes a provision for mine closure of USD6.51 M.

Taxes in the form of value added tax and import duties are included in the forecast cash flows.

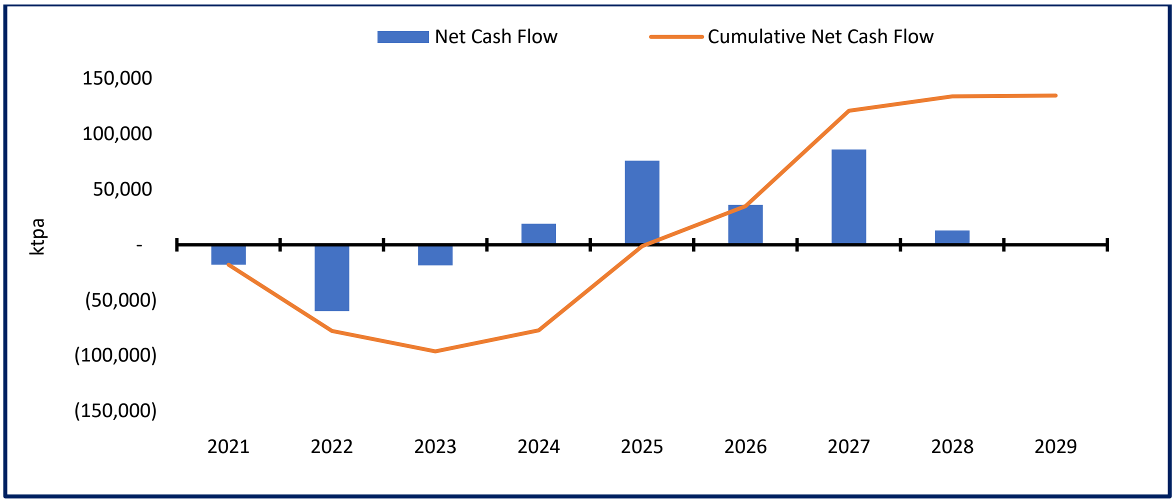
Operating costs for contract mining, owner mining, processing and G&A were deducted from the net revenue to derive the operating cash flows.

The initial capital, working capital, and closure costs were then also deducted from the operating cash flow to determine the net cash flow.

Initial capital expenditures include costs accumulated prior to the first production of gold.

The ungeared and undiscounted net cash flow (NCF) and cumulative net cash flow (CNCF) that result from the Project's post tax production forecast, operating cost forecast and capital expenditure forecast are illustrated in Graph 1-11.

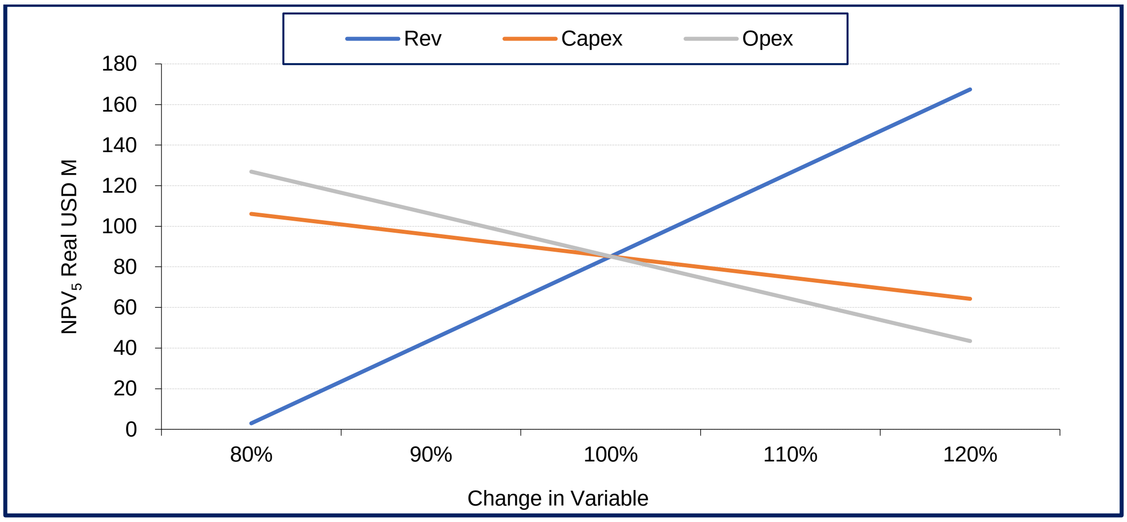
Graph 1-11 Forecast Cash Flows



The DCF results in an expected IRR of 25% and a net present value of USD85.2 M at a real discount rate of 5%, which reduces to USD51.4 M when the discount rate is increased to 10%. The predicted payback period is just over five years and the Maximum Cumulative Negative Cash Flow (MCNCF) or the peak funding requirement rises to USD96.1 M in 2023.

Graph 1-12 shows that increased revenue is clearly the biggest driver of value, but this factor needs to be carefully scrutinised at steady state production conditions since short term marginal increases in the gold price at various threshold limits can reduce value due to the State royalty equation.

Graph 1-12 NPV Sensitivities



1.14. PROJECT EXECUTION PLAN

The execution strategy for successful monitoring and control of the Project will be to use an Integrated Project Management Team (IPMT) approach. Chaarat will manage the Project with the support of various engineering companies, including LogiProc, Azmet, YPT, Ausenco and Ken-Too. The IPMT, led by Chaarat, will be responsible for the project management, procurement, and construction management using in-house resources. A flat organisation structure will favour the rapid decision making required to "fast-track" project. Table 1-9 outlines key milestone dates for the Project.

The project execution plan (PEP) is in part structured around the employment a single (known) contractor to perform mining and earthworks.

Table 1-9 outlines key milestone dates for the Project.

TABLE 1-9 TULKUBASH GOLD PROJECT KEY MILESTONES

Milestone	Date
Pamir Remobilization	15th May 2021
Project Full Financing	1st June 2021
Resume of HLF Bulk Earthworks	17th June 2021
Approval to Proceed with ADR Equipment Manufacturing	3rd August 2021
Approval to Proceed with Crushing Equipment Manufacturing - YPT	1st September 2021
Camp Construction Complete - Phase 1	8th October 2021
Approval to Proceed with Crushing Equipment Manufacturing - Crushers	5th November 2021
Camp Construction Complete - Phase 2 Kitchen and Dining Hall	27th November 2021
Liner Order	30th December 2021
Camp Construction Complete - Phase 2 Remaining Buildings	30th January 2022
Start of Pit Road Construction	1st April 2022
Site Batch Plant Installation Completed	26th April 2022
Start of Pre-stripping	30th June 2022
Haul Road Construction Complete	13th September 2022
Power Generation Facility Commissioned	30th December 2022
First Ore Stacking to Heap Leach	18th May 2023
Irrigation Start	24th June 2023
First Gold Dore Poured	24th August 2023

1.15. RISKS AND OPPORTUNITIES

The results of the economic assessment depend on inputs that are subject to a number of opportunities and uncertainties that may cause actual results to differ materially from those presented herein.

This project has significant upside potential due to the following specific opportunities to enhance project value: -

- Proactive and effective in-pit grade control measures have the potential to reduce the relatively high levels of dilution that has been included in the ROM material reporting to the processing facility. Dilution of 5-10%, as estimated for Tulkubash would be considered typical for an open pit mine;
- Additional exploration along the strike of the Tulkubash oxide orebody is likely to result in the definition of significantly larger mineral resource;
- Concurrent infill exploration drilling of the Tulkubash orebody will provide opportunities for introducing additional flexibility into mine planning and for extending the LoM. Inferred Mineral Resources could thereby also be upgraded to a higher level of confidence;
- The Kyzyltash (sulphide) deposit remains untouched and will benefit from the general infrastructure already provided for the Tulkubash Project; and
- The existence of an agreement with a seasoned mining contractor that shares in the risks associated with the Project, significantly reduces the exposure of new investors to uncertainties related to the overall operation.

However, cognisance needs to be taken of the following uncertainties:

- Commodity prices and exchange rates: A fluctuating gold price in the context of the state royalty equation poses a threat to optimal revenues;
- Mine Plan Flexibility: Insufficient flexibility in the mining plan could affect production rate in the context of the level of production envisaged. Flexibility in the mining plan would facilitate alternatives should the operation experience haulage constraints, such as relatively slow hauling, low truck availability or problems with the haul roads;
- Low Recoveries: The projected recovery rates may be negatively impacted by many variables. Excessive fines could result in gold lock up. The heap leach process is exposed to a wide range of temperature variations ranging from +38ºC to –35ºC. Heap leach kinetics slow down significantly below 7ºC, and production will be affected during the winter months;
- Local geohazards: This includes rock falls from upper mountain slopes, avalanches of debris, rock or snow, seasonal snow melt and stormwater run-off, with consequential impacts on the operations;
- Operational surprises: Geotechnical and hydrogeological considerations during mining may differ from what was assumed. Plant, equipment, or processes may not operate as anticipated, or accidents, labour disputes and other risks associated with day to day mining operations could occur;
- Logistical Problems: The site is remote and poor-quality access roads could pose a risk to the safe and efficient movement of personnel and matériel to site. Present upgrades are ongoing to mitigate this risk;
- Fluctuating fuel prices: This needs to be carefully managed if the Project's cash flow is to be adequately controlled;
- Changing legislative environment: This may create uncertainty regarding legal tenure which, if not managed proactively, may add to the overall risk ascribed to the project. Similarly, any delays to approvals or the receipt of

permits to operate, could adversely impact the revenue expectations. As an example, plant start-up could be delayed due to the late receipt of a sodium cyanide licence for procurement. The legalisation and adaptation process for mine design could also cause delay detail design;

- Tax in the Kyrgyz Republic: Kyrgyz tax legislation is at a developing stage and differing opinions regarding the correct legal interpretation of the various tax rules exist; Note however that Chaarat has a stability agreement with the government which defines the tax regime under which the project will operate; and
- Environmental and Social risks: Unforeseen events related to the environment and local population may occur. For example, any leakage from a damaged HLF pond would result in extra costs when dealing with the consequences. Uncertainty is this respect means that the mine rehabilitation provision may not be adequate.

1.16. Conclusions and Recommendations

The updated Tulkubash Mineral Resource estimate has resulted in a new mine plan, which has improved the financial outlook of the Project. The success of this Project over the short term will unlock the significant longer-term potential of the Kyzyltash deposit.

The studies reported on herein have confirmed that the orebody is amenable to a low-cost open pit mining and leaching operation that will deliver 418 koz of gold over the life of the mine. ROM ore will be crushed to 80% passing 12.5 mm, stacked, leached and the pregnant solution passed through carbon columns to extract the gold. The final product will be a Doré bar of gold and silver with minor impurities.

The geological interpretations, block modelling and subsequent mineral resource estimate were reviewed by Sound Mining, with no errors or red flags encountered. 78 km of exploration drilling has defined 660 koz of contained gold within 3.2 km of a 6 km long strike, and the mineralisation is evidently continuous along strike.

The latest mine plan and associated production schedule are achievable and conservative with respect to the modifying factors that were applied for the Mineral Resource estimate. The Mining Contractor, Pamir Mining, has extensive experience as a mining and civil engineering contractor in similar conditions and is well positioned to manage this type of mining operation.

Chaarat personnel are cognisant of the risks related to safety, health, and the environment. These have been identified and management procedures and preventative measures are already being implemented.

The risks associated with the project are all manageable and provisions have been included in the budget where appropriate for the envisaged mitigation measures. These include, in particular, those related to gold price variations, the availability of the road from the Kumbel Pass to the Project site, congestion of internal haul roads, fuel consumption and/or price fluctuations, avalanches, logistics and local population expectations.

In conclusion, the primary recommendation from this BFS is that Chaarat progresses the project to the commissioning phase and eventually to steady state production.

2. INTRODUCTION

Chaarat Zaav (CZ), a wholly-owned subsidiary of Chaarat Gold Holdings Ltd (CGHL), currently holds two Licences for the Property located in the Kyrgyz Republic.

1. **Licence I** – this Licence is **for the development of subsurface mineral resources**, consisting of the two mineralisation zones currently making up the Property: the Tulkubash zone and the Kyzyltash zone. This report summarises the Feasibility Study work completed for the Tulkubash Gold Project.
2. **Licence II** – this Licence is for the **subsoil use and geological exploration** of the property east of the presently designed mining pit.

In 2019, Chaarat retained LogiProc to update an existing BFS prepared by Tetra Tech (Tt) in April 2018, that detailed the scope, design features and economic viability of the Tulkubash Gold Project (the Project).

The main purpose of the 2019 update was to include new information relating to the resource and reserve, whilst at the same time updating other information where appropriate, for example, inclusion of the outcomes of the 'Value Engineering' study conducted on the Processing facilities.

During 2019/2020, further work was undertaken by Chaarat to:

- complete additional recovery test work in the Mid and Satellite/East zones;
- better define the resource; and
- update the project costs, to capture changes in development, construction, operating and in-country costs.

LogiProc (Pty) Ltd was retained to update the 2019 BFS with the above information.

Whilst many sections of the 2021 BFS document continue to reflect the 2019 BFS information, the following sections in particular have been updated – Sections 1, 3, 13, 14, 15, 16, 17, 18, 21, 22, 25, 26 and where appropriate, the Appendices.

The 2020 Mineral Resource Estimate was produced by Mr Viktor Usenco, and Mr Evgeny Fomichev, both competent persons as defined by the JORC Code.

The updated 2021 BFS has an effective date of 28 April 2021, with an effective date of the associated Mineral Resource Estimate stated as at 07 November 2020.

A summary of the Responsible Specialists and Editors responsible for the compilation and review of each section of the 2021 BFS report is provided in Table 2-1.

All currency is reported in US dollars, and all measurements are reported using SI units, unless otherwise noted.

TABLE 2-1 SUMMARY OF STUDY AUTHORS

	Report Section	Responsible Specialist	Editor
1.0	Summary	All – as per subsection	**All – edited by subsection**
2.0	Introduction	Richard Bewsey, BSc Chem Eng (Hons)	**Richard Bewsey, BSc Chem Eng (Hons)**
3.0	Reliance on Other Experts	Richard Bewsey, BSc Chem Eng (Hons)	**Richard Bewsey, BSc Chem Eng (Hons)**
4.0	Property Description and Location	Richard Bewsey, BSc Chem Eng (Hons)	**Richard Bewsey, BSc Chem Eng (Hons)**
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Richard Bewsey, BSc Chem Eng (Hons)	**Richard Bewsey, BSc Chem Eng (Hons)**
6.0	History	Richard Bewsey, BSc Chem Eng (Hons)	**Richard Bewsey, BSc Chem Eng (Hons)**
7.0	Geological Setting and Mineralisation	Victor Usenko, Principal Geologist, MAIG Evgeny Fomichev, Principal Geologist, MAIG	**Diana van Buren, BSc Geology (Hons)**
8.0	Deposit Types	Victor Usenko, Principal Geologist, MAIG Evgeny Fomichev, Principal Geologist, MAIG	**Diana van Buren, BSc Geology (Hons)**
9.0	Exploration	Victor Usenko, Principal Geologist, MAIG Evgeny Fomichev, Principal Geologist, MAIG	**Diana van Buren, BSc Geology (Hons)**
10.0	Drilling	Victor Usenko, Principal Geologist, MAIG Evgeny Fomichev, Principal Geologist, MAIG	**Diana van Buren, BSc Geology (Hons)**
11.0	Sample Preparation, Analyses and Security	Victor Usenko, Principal Geologist, MAIG Evgeny Fomichev, Principal Geologist, MAIG	**Diana van Buren, BSc Geology (Hons)**
12.0	Data Verification	Victor Usenko, Principal Geologist, MAIG Evgeny Fomichev, Principal Geologist, MAIG	**Diana van Buren, BSc Geology (Hons)**
13.0	Mineral Processing and Metallurgical Testing	Richard Bewsey, BSc Chem Eng (Hons)	**Richard Bewsey, BSc Chem Eng (Hons)**
14.0	Mineral Resource Estimate	Victor Usenko, Principal Geologist, MAIG Evgeny Fomichev, Principal Geologist, MAIG	**Vaughn Duke, PrEng, PMP, BSc Min Eng (Hons), MBA**

	Report Section	Responsible Specialist	Editor
15.0	Ore Reserve Estimates	Peter Carter, BSc (M.Eng), MBA, P.Eng	**Vaughn Duke, PrEng, PMP, BSc Min Eng (Hons), MBA**
16.0	Mining Methods	Peter Carter, BSc (M.Eng), MBA, P.Eng	**Vaughn Duke, PrEng, PMP, BSc Min Eng (Hons), MBA**
17.0	Recovery Methods	Richard Bewsey, BSc Chem Eng (Hons) Scott Elfen, BSc Civ Eng (Geotech)	**Richard Bewsey, BSc Chem Eng (Hons)** **Scott Elfen, BSc Civ Eng (Geotech)**
18.0	Project Infrastructure	Richard Bewsey, BSc Chem Eng (Hons) Scott Elfen, BSc Civ Eng (Geotech)	**Richard Bewsey, BSc Chem Eng (Hons)** **Scott Elfen, BSc Civ Eng (Geotech)**
19.0	Market Studies and Contracts	Marat Khasanov, MBA	**Marat Khasanov, MBA**
20.0	Environmental Studies, Permitting and Social or Community Impact	Alison Allen, MSc, BSc, CEnv, MIEMA, MIEEM, FIMMM	**Keith Raine, Environmental Specialist, PR SciNat, B Sc (Hons), B Sc Zoology**
21.0	Capital and Operating Cost Estimates	Ercan Unluyol, Civil Engineer - Bachelor Degree	**Richard Bewsey, BSc Chem Eng (Hons)**
22.0	Economic Analysis	Mark Turnbull (MSc)	**Vaughn Duke, PrEng, PMP, BSc Min Eng (Hons), MBA**
23.0	Adjacent Properties	Ercan Unluyol, Civil Engineer - Bachelor Degree	**Richard Bewsey, BSc Chem Eng (Hons)**
24.0	Other Relevant Data and Information	Ercan Unluyol, Civil Engineer - Bachelor Degree	**Ercan Unluyol, Civil Engineer - Bachelor Degree** **Richard Bewsey, BSc Chem Eng (Hons)**
25.0	Interpretation and Conclusions	All – as per subsection	**All – authored by subsection**
26.0	Recommendations	All – as per subsection	**All – authored by subsection**
27.0	References	All – as per subsection	**All – authored by subsection**

3. Reliance on Other Experts

The authors followed standard professional procedures in preparing the contents of this report. Data used in this report has been verified where possible and the authors have no reason to believe that the data was not collected in a professional manner.

Technical data provided by Chaarat or CGHL for use by the authors in this Feasibility Study is the result of work conducted, supervised, and/or verified by Chaarat or CGHL professional staff or their consultants.

In preparation of the updates to the relevant sections, LogiProc's review took into account new and updated technical and financial information relating to the project, and was reliant on the accuracy and integrity of the information provided by Chaarat.

When considering the updated design of the Processing Plant, LogiProc relied on the design input from plant supply specialists for layout and costing purposes:

- **Crushing Plant**. YPT (Yilmaz Proses Teknolojileri), based in Turkey, provided the basic design and costing for the Crushing Section of the Process Plant; and
- **ADR Plant**. Azmet Technology and Projects, based in South Africa, provided the basic design and costing for the ADR (Adsorption, Desorption, Regeneration) Section of the Process Plant.

The 2020 Mineral Resource Estimate was produced by Victor Usenko, Principal Geologist, MAIG, and Evgeny Fomichev, Principal Geologist, MAIG, competent persons as defined by the JORC code.

The 2021 Recovery Model was produced by Mr. Joe Hirst B.Sc. (Hons) M.Sc. EurGeol, CGeol, FGS, a competent person as defined by the JORC code.

Table 3-1 outlines the responsibilities of each company for the 2021 Update.

Table 3-1 Feasibility Study Scope of Work Performance

Company	Responsibility
LogiProc	Overall project management; mineral processing and metallurgical testing; recovery methods; project infrastructure; capital cost estimate, economic analysis, operating cost estimate, project execution plan.
Viktor Usenko Evgeny Fomichev	Geological block model and associated data integrity.
Peter Carter	Mining method review; and ore reserve statement. Competent person for ore reserves and Mining Engineering.
WAI	Environmental studies, permitting, and social or community impact; geochemistry; hydrology; hydrogeology.
Ausenco	Heap leach facility design.

4. Property Description and Location

The Project is located within the Chaarat Property at latitude 42°1'6.91" N and longitude 71°9'39.04" E, in the Sandalash Range of the Alatau Mountains, in the Jalal-Abad Province of north-western Kyrgyzstan, close to the border with Uzbekistan (Figure 4-1). The Property area is located approximately 300 km southwest of the capital Bishkek, 75 km upstream and northeast of the regional administrative centre of Kanysh-Kiya in the Chatkal Valley, and 300 km by road from the nearest railway station in Shamaldy-Say.

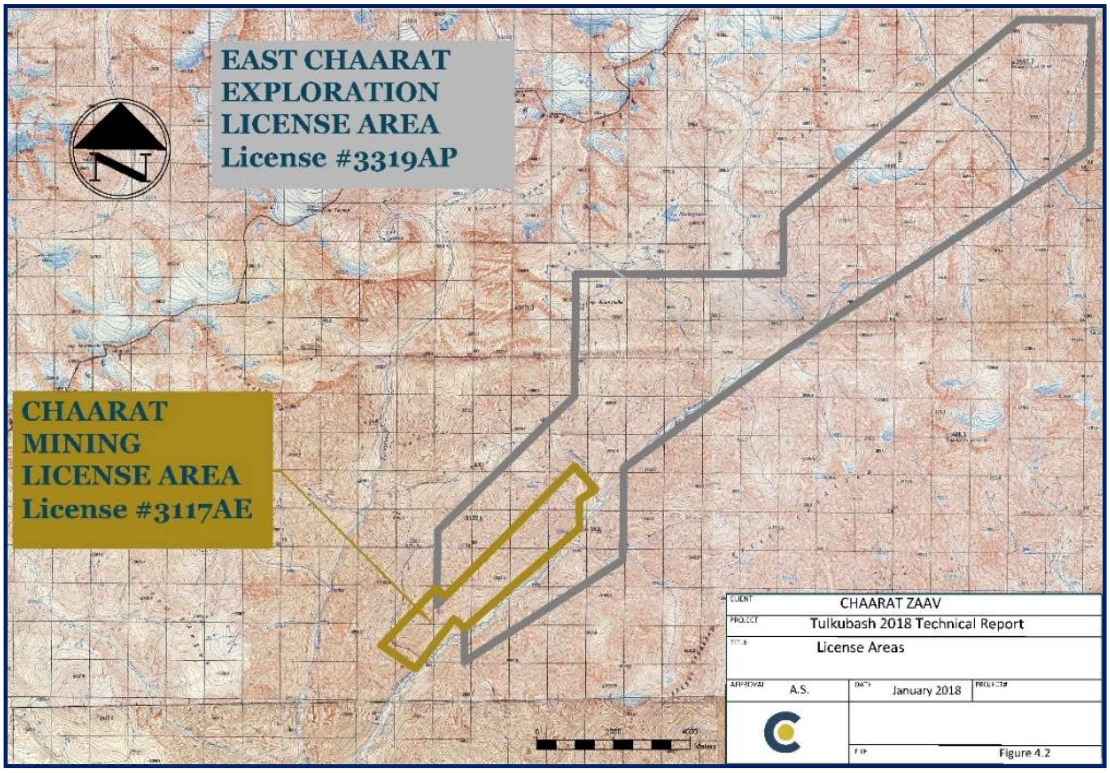
The Project site is situated adjacent to the Sandalash River, at an elevation of 2,100 to 3,600 masl.

Figure 4-1 Chaarat Property Location Map

4.1. Licensing & Ownership

Chaarat, a wholly-owned subsidiary of Chaarat established in the Kyrgyz Republic, currently holds two Licences controlling the Property, a mining (or production) Licence of 700.03 ha covering the defined Mineral Resources, and an exploration Licence of 6,776 ha covering prospective ground along trend to the northeast (Figure 4-2).

Figure 4-2   Chaarat Property Licence Areas



4.1.1. Chaarat Mining Licence 3117AE

Mining Licence 3117AE was renewed on 7th September 2017 and is valid until 25th June 2032.

The Licence coordinates are listed in Table 4-1. The coordinate system is Gauss Krueger Pulkovo 1942 Zone 12 and the size of the area is 700.03 ha.

Table 4-1   Mining Licence No. 3117AE Coordinates

Point No.	X	Y	Point No.	X	Y
1	126 77 600	46 55 400	6	126 82 728	46 59 261
2	126 79 000	46 56 900	7	126 82 757	46 58 554
3	126 79 264	46 56 711	8	126 79 776	46 55 887
4	126 82 604	46 60 152	9	126 79 487	46 56 116
5	126 83 150	46 59 556	10	126 78 500	46 54 800

There are certain conditions that need to be met to hold Mining Licence 3117AE, which include:

- Deposit development according to the Technical Project for the Chaarat Gold Deposit Development (Ken-Too 2015), which was approved by the State Committee for Industry, Energy and Subsoil Use of the Kyrgyz Republic (SCIES);

- Continuous work on development, detailed design and cost estimate documentation;
- Paying taxes on the right to use subsoil within the terms stipulated by Kyrgyz Republic legislation;
- Submitting a social package to SCIES, including an investment programme for improving conditions for local community development, which consists of training, providing jobs for residents of the local communities, and infrastructure development; and
- Opening a disturbed land rehabilitation account and accumulating funds defined by the Technical Project Report (Ken-Too 2015) for the Chaarat Gold Deposit Development.

4.1.2. East Chaarat Exploration Licence 3319AP

Exploration Licence 3319AP was renewed on 29th July 2016 and is valid until 7th October 2023.

The coordinates of the Licence are listed in Table 4-2. The coordinate system is Gauss Krueger Pulkovo 1942 Zone 12 and the Licence area is 6,776 ha.

The main conditions to hold Exploration Licence 3319AP include:

- Paying taxes and other payments for subsoil use per Kyrgyz Republic legislation;
- Informing SCIES on a quarterly basis about Licence retention fee payments and provide copies of all payment documents;
- Providing geological reports to the State Geological Fund, as required under Kyrgyz Republic legislation; and
- Opening a disturbed land rehabilitation account and accumulate the amount of funds as defined by the Technical Project Report (Ken-Too 2015) for the Chaarat Gold Deposit Development.

Table 4-2 Exploration Licence 3319AR Coordinates

Point No.	X	Y	Point No.	X	Y
1	12679775.83	4655000.00	10	12682571.49	4665177.33
2	12679775.83	4655886.65	11	12687993.31	4665260.71
3	12682757.12	4658554.26	12	12687993.31	4666816.98
4	12682728.12	4659260.70	13	12694125.98	4672000.00
5	12683149.87	4659555.94	14	12696000.00	4672000.00
6	12682604.22	4660151.66	15	12696000.00	4668607.81
7	12679035.11	4656474.48	16	12688029.05	4663211.98
8	12679035.11	4658418.95	17	12683893.61	4660127.56
9	12682571.49	4661982.42	18	12683893.61	4657717.98

4.2. Surface Land Use Permits

The general layout of the planned infrastructure located within the current permit boundaries is illustrated in Figure 4-3.

Chaarat obtained consents of the local state administration, and the local self-governments of the Chatkal Region, required to conduct exploration work under Exploration Licence 3319AP.

Chaarat, pursuant to Mining Licence Agreement No. 4 of Mining Licence 3117AE submitted to the SCIES, obtained temporary land-use rights to the land plots located within the coordinates indicated in Mining Licence Agreement No. 4 (see Appendix A), as well as the land plots located within the territory of Kanysh-Kiya Ayil Okmotu, for the construction of infrastructure facilities (also known as land allocation). The size, purpose, and expiry of each land plot is outlined in Table 4-3.

Figure 4-3 Permitted Surface Land Use

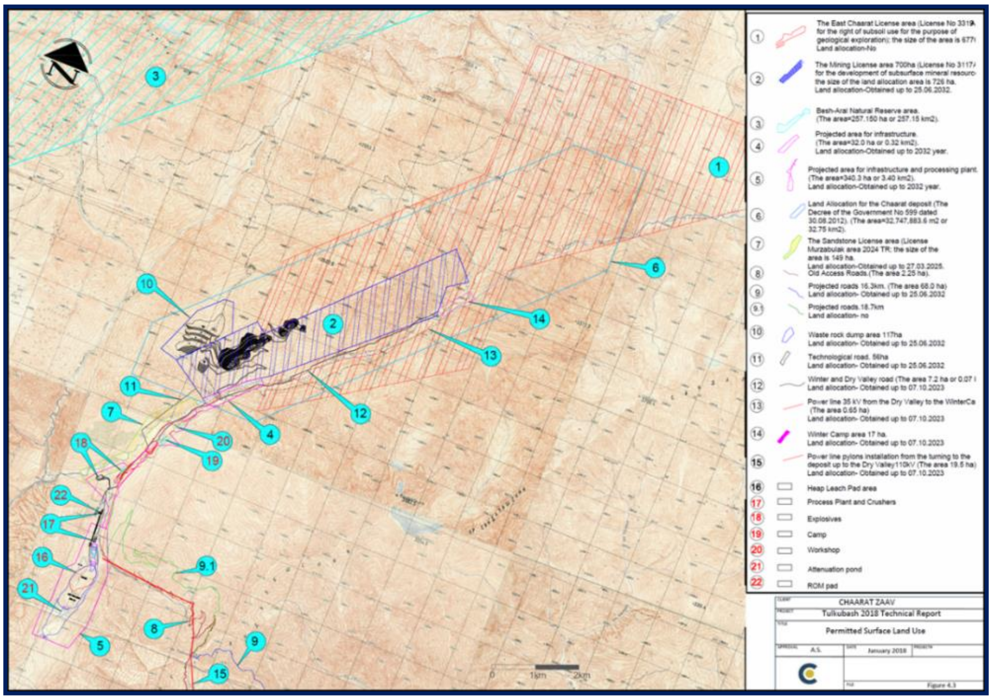


Table 4-3 Mining Licence Agreement No. 4 Land Plot Usage

№	Land Plot (ha)	Purpose	Expiry
1	899.000	726 ha for mining; 117 ha for blanket of Tulkubash area 56 ha for technological roads	For temporary use till 2032
2	384,586.000 (dry valley)	Construction of mining process plant and other supporting infrastructure.	For temporary use till 2032
3	68.000	Construction of access road along the southern slope of Kumbel pass.	For temporary use till 2032
4	32.000	Construction of infrastructure	For temporary use till 2032

№	Land Plot (ha)	Purpose	Expiry
5	17.440	Winter camp	For temporary use till 2023
6	7.200	Access roads from the dry valley to the summer camp	For temporary use till 2023
9	2.250	Access roads to the Chaarat Property area	For temporary use till 2023
Total	**1,431,143.317**	**-**	**-**

Chaarat is required to submit the following mandatory reports to SCIES:

- An annual report, as well as an operations programme for each new year, before 31 January of each new year. Note that the Annual Report and Operational Programme has been issued on 31 Jan 2021.
- Report of the established 5-GR form, before the 1st of March of each new year. Note that the Report of the established 5-GR form has been issued on 1 Mar 2020 and 2021.
- While the Semi-annual information on fulfilment of the Licence agreement terms, was issued on the 15th of July 2020, this requirement is no longer obligatory in Kyrgyz law.

The Mining Licence and surface rights are subject to the following taxes and royalties:

- Profit Tax for gold mining companies – from 1 to 20%, depending on the world price on gold. The profit tax is 3% for a gold price below USD 1,300/oz;
- Bonus – one-time payment while obtaining a Licence (the rate depends on the type and reported quantity of the Mineral Resource). Commercial discovery bonus is payable when officially reported to SCEIS. The current rates set by the Kyrgyz Republic government are USD 60,000/t of gold;
- Royalty 5% from gold sale proceeds;
- Land Tax, calculated depending on the size of the land area;
- Property Tax, calculated depending on the size of the property;
- Income Tax, (for individuals) 10%; and
- Value Added Tax 12%.

Non-tax Payments:

- Licence retention fees, the rates depend on the Mineral Resource and the year the Licenced area is used. A Special formula is applied per SCEIS guidelines; and
- 2% tax from revenue, for local infrastructure.

As per Kyrgyz Republic legislation on subsoil use, land allocation is granted for subsoil use (i.e., road construction, industrial sites, power lines, and other infrastructure facilities) by the state authorities or the local self-governing administrations for the term of validity of the Licence for the right to use subsoil.

4.3. PERMITTING REQUIREMENTS

In addition to Mining Licence 3117AE and surface rights, Chaarat needs to obtain additional Licences and permits to construct and operate the mine. The list includes, but is not limited to:

- Technical design that has passed the following expertise approvals:
  - Industrial safety;
  - Environment safety (i.e., environment impact assessment (EIA or OVOS); and
  - Subsoil use protection.
- Permit to perform mining works;
- In-country legalization of design documentation in case facilities are designed by a non-local organization;
- State construction expertise of all completed detailed design documentation;
- Commissioning of constructed facilities (government acceptance);
- Licence for water use from underground sources;
- Permit to release of pollutants into the air;
- Permit to discharge pollutants into the water;
- Permit for waste disposal;
- Licence to carry out activities for the utilization, storage, disposal, and destruction of toxic waste materials and substances;
- Licence for import, production and sale of explosive and pyrotechnic materials and products or permit to purchase explosive materials;
- Permit for the transportation of hazardous goods;
- Permit for the storage of explosive materials;
- Permit for blasting works;
- Approved emergency plan;
- Certification of machinery, plant and equipment; and
- Proper certifications for staff.

Chaarat has initiated a permitting process and believes the required permits will be granted under Kyrgyz legislation.

The status of the permits as of the latest revision of this BFS is seen in Table 4-4 below.

Table 4-4 Permits Status

No	Design Title	Local Designer	Expertise, date of obtaining				Licence	Licence agreement
			Industrial safety expertise	Environmental expertise	Construction expertise	Subsoil Expertise		
1	Haul road optimisation	Dortrans service	19.11.2019	20.12.2019	31.12.2019			
2	Culvert platform	Dortrans service	20.02.2020	15.09.2020	06.10.2020			
3	HLF design adaptation	Ken-Too	28.10.2019	09.12.2019	20.04.2020			
4	Camp waste water treatment plant design	Enkon		25.09.2020	25.01.2021			
5	Water supply wells for camp and plant	GSPA		29.10.2020			19.08.2019	16.04.2020 (#2)
6	Adaptation of Mining Works Design	Ken-Too	02.04.2020	14.09.2020		23.12.2020		
7	Platform Design	Dortrans service	01.02.2021					
8	OVOS (EIA)	Ken-Too		25.09.2020				
9	Permit for emission of pollutants into atmosphere	N/A					25.03.2020	
10	Expertise for Waste Management Standards	N/A					27.02.2020	
11	Expertise for Maximum Permissible Emissions Project	N/A					27.02.2020	
12	Expertise for Ecological Passport	N/A					27.02.2020	
13	GKZ Protocol (Approval of Reserves by State Reserve Committee)	N/A					12.11.2020	

4.4. ENVIRONMENTAL LIABILITIES

Chaarat bears full legal responsibility for compliance with environmental requirements under Kyrgyz Republic legislation and the approved design solutions, which includes, but is not limited to, air protection, protection of water resources, and land protection and rehabilitation. Chaarat is required to obtain the relevant environmental permits for the respective activities (EIA/OVOS), make quarterly payments for environmental pollution per Kyrgyz Laws, and submit reports on compliance with environmental requirements.

5. Accessibility, Climate, Location, Resources, Infrastructure and Physiography

5.1. Access

From the capital city of Bishkek, the Chaarat Property is accessible via 750 km of paved and unpaved roads, 240 km of which are gravel after the city of Ala-Buka (Figure 5-1). The M39 highway leads westward from Bishkek to Kara-Balta, connecting to the M41 highway south through the Too-Ashu Pass. The route continues westwards through Chichkan, and then around the Toktogul water reservoir along the Naryn River to Kara-Kul and Tash-Kumyr. After Tash-Kumir, the road continues northwest to the city of Ala-Buka and through Chapchima Pass to the village of Jany-Bazar at the intersection of the Chatkal and Sandalash rivers. The final part of the route continues south through the village of Kanysh-Kiya and through the Kumbel Pass to the Chaarat Property. Travel time from Bishkek is approximately 14 to 18 hr, with an overnight stay in the city of Ala-Buka.

This route provides virtual year-round access to the Chaarat Property area and, although longer, is the route favoured for future development, as it will be required to move hazardous goods. In addition, Ala-Buka is the nearest town to the Shamaldy-Say train station located approximately 300 km from the Property. The road over the Kumbel Pass is currently being upgraded to ensure all-season access. The upgrade is at 80% completion as of this BFS revision, with some road widening and drainage structures still required.

Currently during the summer months between April and October, 40 ft container trucks can travel on this road with the help of technical equipment, as some of the grades of the road do not allow these trucks to climb by themselves. From October till April during the wintertime, only 20 ft container trucks, which are equipped with winter gear can travel on the road, along with full time support by a grader or a loader.

There is an alternate access into the Chatkal Valley through Talas and Kyzyl Adyr (Kirovskoye) village. The distance from the capital city of Bishkek is 520 km of paved and unpaved roads, 150 km of which are gravel. The journey after Kyzyl-Adyr is via gravel roads, south through two high mountain passes: the Kara Bura Pass, with flatter areas through the Kara Bura and Chatkal valleys, and over the Sandalash range by the Kumbel Pass. The roads are generally in good condition, and the gravelled sections along the main roads are well maintained. The roads over the mountains are unsuitable for heavy vehicles greater than 10 t and are impassable during the winter and spring unless kept clear of snow. Seasonal access is between June and October. Travel time from Bishkek to the Chaarat Property using this route takes approximately 10 to 12 hours.

The railway station at Shamaldy-Say is currently not suitable for handling goods bound for Tulkubash. An alternative railway station is in Maymak, 195 km to the north on the international border with Kazakhstan; however, it is impractical to deliver hazardous goods to Maymak as the route from Maymak to the Property traverses through three high mountain passes (Otmok, Chapchyma, and Kumbel) and the narrow valley of Chichkan. Consequently, in the meantime, material and equipment shipped by rail will be directed to the Alamedin railway station in Bishkek, from where it will be transported by road to site.

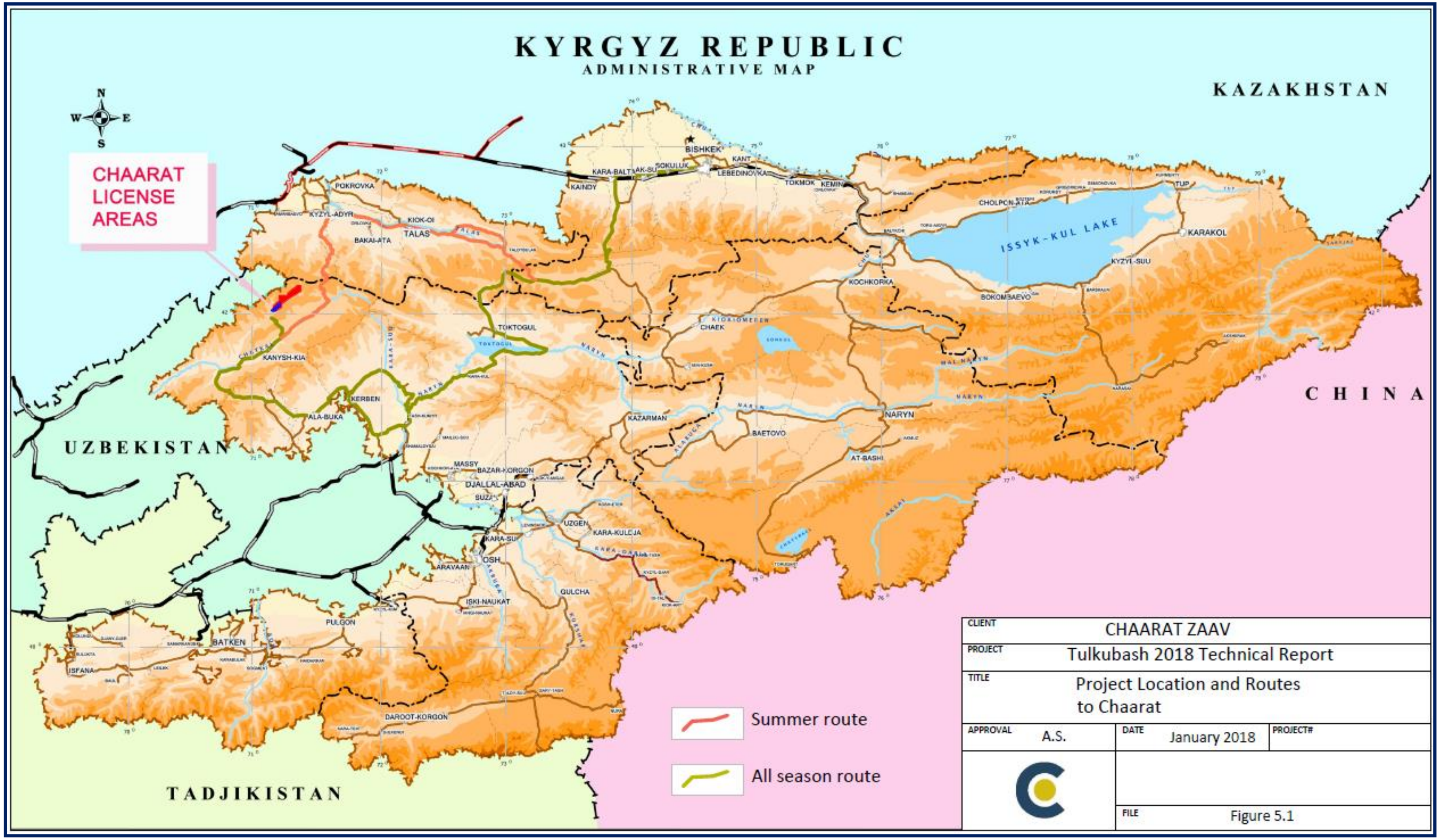
The nearest international airports to the Property are Manas International Airport in Bishkek (530 km northeast) and Osh International Airport (560 km southeast). Regional airports include Jalal- Abad Airport (400 km southeast) and Talas Airport (200 km northeast). There is a Soviet-era 800 m long airstrip in Kanyshkia (56 km southwest), which is currently not in use. Alternative airports are Namangan International Airport in Uzbekistan (360 km southeast) and Taraz Airport in Kazakhstan (200 km north).

5.2. CLIMATE

The climate is classified as semi-arid to temperate-humid in the lower part of the Property area. The high-alpine zones are subject to long severe winters, with frequent snowstorms and avalanches.

At lower elevations, the snow-free period lasts from March to December, and at higher elevations, from June to October, although the mountain peaks are covered by snow throughout the year. The average annual precipitation is 460 mm, with snow falling between October and February and rain between March and May. The dry season takes place from June to September. Temperatures in the Jalal-Abad Province range from an average high of +26°C in the summer months, to an average low of -20°C in the winter months (Anon. Chatkal weather data report 2012). Daily and seasonal temperatures are highly variable. The prevailing winds are north-westerly.

Figure 5-1 Property Location and Access Routes



5.3. Local Resources

The Chatkal Valley contains eight villages, with an estimated total population of 13,000 people. The area is isolated, and the economy is poorly developed, with most workers engaged in livestock breeding and hay production. There are no permanent residents in the Sandalash Valley. The area is not used for cultivation, but the treeless, grassy slopes are used during the summer for grazing sheep, horses, and cattle by the local people.

5.4. Infrastructure

The Property area lies in the uninhabited Sandalash Valley, and there is currently no electric power available within the immediate site area. The nearest power transmission line (10 kV) provides power to the Chatkal Valley villages and runs through the Chatkal Valley approximately 30 km from the Property. A 110 kV power transmission line runs from the Talas region to the Kuru-Tegerek deposit (China Gold owned mine) approximately 40 km away from the Property.

There are three potential sources of electricity for the Property:

- Power line connecting the Property to the national grid;
- Diesel generating capacity installed near the site; and
- Hydropower station located on the Sandalash River.

All three alternatives have been considered, as well as an optimal combination of all three alternatives. This study was done during the previous BFS. Further information on site power and additional infrastructure is available in Section 18.

5.5. Physiography

The Property area is characterised by extreme topography ranging from the Sandalash Valley at an elevation of 2,000 m, to the mountain ranges, which peak at an elevation of 4,200 masl. The Sandalash Valley is between 100 and 300 m wide, between steep slopes on either side. The Sandalash River follows a linear south-westerly trend, with a moderate gradient in the Property area, and intermittent rapids between swiftly flowing segments. The Sandalash River flows into the Chatkal River south of the Property area near the village of Jany-Bazar. These rivers normally flood in spring with snow melt and are intermittently impassable.

6. HISTORY

6.1. EARLY EXPLORATION

Antimony mineralisation in the Chaarat area was originally identified by Soviet-era geologists conducting a reconnaissance exploration programme prior to 1992. The North Kyrgyz Geological Expedition subsequently completed a regional stream sediment sampling programme, which identified antimony, arsenic, gold, silver, and tungsten anomalies in the Chaarat region. They identified significant antimony mineralisation in the Tulkubash and Main zone areas and developed three drifts totalling 660 m (Anon. 2004).

Following the breakup of the Soviet Union, Apex Asia acquired control of the Licence in 1996, and subsequently formed a joint venture with Newmont Overseas Exploration Limited. Newmont completed a geophysical survey and drilled seven holes totalling 1,803 m in the Shir Canyon area. Newmont terminated the joint venture in 2000, after which Apex sold its interest.

At the end of 2002, Chaarat was formed and acquired what is now known as the Chaarat Mining Licence. In 2003, Chaarat compiled historic data into a digital database and conducted mapping and sampling in the Shir Canyon area (Diner, pers. comm. 2017). This work identified targets that were followed up with mapping, trenching, and sampling in 2004. Five core holes totalling 857 m were completed during the 2004 field season. All the holes intersected significant gold mineralisation with drillhole CCH003 returning 8.3 m of 7.0 g/t of gold.

6.2. EXPLORATION AND DEVELOPMENT

Building on the success of the 2004 programme, drilling continued through 2006 to develop the Main and Contact zone mineralisation. In addition, in 2006 Chaarat collared an exploration adit to develop the C54 (now called the CP zone) area of the Contact zone. The purpose of this adit was to provide drill platforms to develop this zone down dip and to collect bulk samples for metallurgical testwork.

Concurrent with this work, soil sampling in the Tulkubash Formation was initiated in 2004. Soil samples were collected along spurs descending from the top of the ridge to the Sandalash River. The results of the soil survey exceeded expectations, generating large and extensive anomalies over 1 ppm of gold in the Tulkubash quartzite, with gold assays reaching up to 73 g/t of gold. Follow up trenches and detailed rock chip profiles were collected over what is now the Tulkubash deposit (variously called the T0700 and the Normat zone), which defined a large, coherent geochemical anomaly. In 2005, a single initial hole was drilled in this area which intersected 17.1 m that assayed 4.61 g/t of gold.

Systematic development drilling of the Main and Contact zones (also called the Kyzyltash mineralisation) continued through 2013, with underground Mineral Resources defined within nine ore bodies (the M2400, M3000, M3400, M3900, M4400, M5000, CP, C4000 and M6000) along the Main and Contact zones. Surface and underground drilling in the CP zone identified continuous mineralisation between the surface exposure at an elevation of 2,790 m, to a depth of 1,740 m, a vertical distance of over 1 km.

In 2010, early metallurgical testwork indicated that much of the Tulkubash mineralisation was free milling and could potentially develop into a low-cost, open pit, heap leach operation. This motivated an extensive development drilling programme concurrent with continued development of the refractory ores of the Kyzyltash mineralisation. This culminated with the completion of 128 holes totalling nearly 16,000 m in 2011.

Exploration and development programmes were modest from 2013 through 2016, with no drilling occurring in 2015. In 2017 and 2018, there was a renewed focus on the Tulkubash deposit as a potential starter mine for Chaarat, with approx. 17,400 m of drilling completed in 2017 and approx. 20,000 m of drilling completed in 2018, and 2019

A summary of drilling completed on the Property is shown in Table 6-1.

TABLE 6-1 DRILLING SUMMARY

Year	Kyzyltash Zones		Tulkubash Zone		Total Drilling		Geotechnical Drilling	
	No. of Holes	Total Metres	No. of Holes	Total Metres	No. of Holes	Total Metres	No. of Holes	Total Metres
2000	7	1,803.2	-	-	7	1,803.2	-	-
2004	5	856.8	-	-	5	856.8	-	-
2005	33	6,677.4	1	150.6	34	6,828.0	-	-
2006	23	4,592.5	7	1,393.6	30	5,986.1	-	-
2007	41	8,163,2	12	2,374.8	53	10,538.0	-	-
2008	71	16,051.4	-	-	71	16,051.4	6	839.4
2009	21	4,804.1	5	802.6	26	5,606.7	-	-
2010	28	5,597.0	37	4,271.8	65	9,868.8	-	-
2011	44	13,344.2	128	15,984.2	172	29,328.4	-	-
2012	31	3,884.3	39	6,842.0	70	10,726.7	-	-
2013	76	11,201.3	14	1,781.2	90	12,982.5	30	4,155.9
2014	-	-	48	5,813.6	48	5,813.7	-	-
2015	-	-	-	-	0	0.0	-	-
2016	-	-	12	1,185.8	12	1,185.8	15	951.1
2017	-	-	135	17,420.4	135	17,420.4	54	894.0
2018	-	-	122	19,924.8	122	19,894.5	-	-
2019	-	-	129	19,974.0	129	19,974.0	-	-
2020	-	-	21	2,434.3	21	2,434.3	-	-
Total	**380**	**76,975.4**	**710**	**100,353.7**	**931**	**177,329.1**	**105**	**6840.4**

6.3. RESOURCE AND RESERVE DEVELOPMENT

Over the course of developing the various deposits at the Property, Chaarat released a series of updated Mineral Resource reports, along with various scoping studies, prefeasibility studies, and definitive feasibility studies (Table 6-2). This work was completed by various international consulting companies and was generally stated as JORC compliant. As the level of detail of the work increased, Chaarat built a foundation of studies (geotechnical, hydrology, metallurgy, social, etc.) completed by international consultants that have been used, where appropriate, in the current feasibility.

TABLE 6-2 RESOURCE DEVELOPMENT HISTORY

	Press Release Date	Source	Cut-off Grade (Au g/t)	Measured			Indicated			Inferred			Measured & Indicated			Notes
				Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	
Main Zone	4/22/2008	Behre Dolbear	2.00	-	-	-	8,446	4.39	1,193	2,762	4.22	374	8,446	4.39	1,193	Cut-off not stated in press release
Contact Zone			2.00	-	-	-	5,286	4.48	761	3,503	4.33	488	5,286	4.48	761	
Tulkubash			2.00	-	-	-	1,642	4.70	248	473	4.66	71	1,642	4.70	248	
Total			**-**	**-**	**-**	**-**	**15,374**	**4.45**	**2,202**	**6,738**	**4.31**	**933**	**15,374**	**4.45**	**2,202**	
Main Zone	3/30/2009	SRK	3.00	-	-	-	6,531	4.30	904	4,992	4.33	693	6,531	4.30	904	
Contact Zone			3.00	-	-	-	3,673	4.18	493	6,831	4.23	928	3,673	4.18	493	
Tulkubash			2.00	-	-	-	1,642	4.70	248	473	4.67	71	1,642	4.70	248	
Total			**-**	**-**	**-**	**-**	**11,846**	**4.32**	**1,644**	**12,294**	**4.29**	**1,694**	**11,846**	**4.32**	**1,644**	
Main Zone	03/09/2010	SRK	2.00	-	-	-	8,600	4.05	1,127	5,400	4.28	744	8,600	4.05	1,127	
Contact Zone			2.00	-	-	-	8,000	4.12	1,061	5,600	4.13	741	8,000	4.12	1,061	
Tulkubash			2.00	-	-	-	-	-	-	2,500	4.18	338	-	-	-	
Total			**-**	**-**	**-**	**-**	**16,600**	**4.09**	**2,188**	**13,500**	**4.20**	**1,821**	**16,600**	**4.09**	**2,188**	
Main Zone	02/07/2011	WAI	2.00	-	-	-	5,155	4.40	731	9,239	4.20	1,261	5,155	4.40	731	Cut-off not stated in press release

	Press Release Date	Source	Cut-off Grade (Au g/t)	Measured			Indicated			Inferred			Measured & Indicated			Notes
				Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	
Contact Zone			2.00	-	-	-	7,864	4.30	1,078	7,671	4.10	1,015	7,864	4.30	1,078	Contact zone restated 7/7/2011
Tulkubash			2.00	-	-	-	219	4.60	32	2,280	3.90	289	219	4.60	32	
Total			**-**	**-**	**-**	**-**	**13,238**	**4.30**	**1,841**	**19,190**	**4.20**	**2,565**	**13,238**	**4.30**	**1,841**	
Main Zone	03/05/2012	WAI	2.00	-	-	-	7,136	4.23	971	9,051	4.26	1,240	7,136	4.23	971	
Contact Zone			2.00	-	-	-	12,463	4.30	1,721	8,045	4.25	1,109	12,463	4.30	1,721	
Tulkubash			1.00	180	3.07	18	2,145	2.80	196	2,987	2.99	287	2,325	2.84	214	
Total			**-**	**180**	**3.07**	**18**	**21,744**	**4.13**	**2,888**	**20,083**	**4.08**	**2,636**	**21,924**	**4.12**	**2,906**	
Main Zone	3/18/2013	Internal	-	-	-	-	-	-	-	-	-	-	-	-	-	
Contact Zone			-	-	-	-	-	-	-	-	-	-	-	-	-	
Tulkubash			-	-	-	-	-	-	-	-	-	-	-	-	-	
Total			**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	
Main Zone	04/01/2014	Gustavson	2.00	-	-	-	-	-	-	-	-	-	-	-	-	
Contact Zone			2.00	3,200	3.89	401	27,400	3.24	2,857	11,360	3.49	1,274	30,600	3.31	3,258	Main & Contact zones combined
Tulkubash			2.00	3,700	2.17	257	6,300	1.87	382	1,890	1.90	116	10,000	1.98	639	
Total			**-**	**6,900**	**2.97**	**658**	**33,700**	**2.98**	**3,239**	**13,250**	**3.26**	**1,390**	**40,600**	**2.98**	**3,897**	

	Press Release Date	Source	Cut-off Grade (Au g/t)	Measured			Indicated			Inferred			Measured & Indicated			Notes
				Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000 tr oz)	
Main Zone	11/11/2014	GSI	2.00	-	-	-	-	-	-	-	-	-	-	-	-	
Contact Zone			2.00	6,629	3.15	671	32,794	3.67	3,864	6,611	3.92	832	39,423	3.58	4,535	Main & Contact zones combined
Tulkubash			1.00	7,646	1.90	466	3,224	1.77	184	2,384	1.81	79	10,870	1.86	650	
Total			**-**	**14,275**	**2.48**	**1,137**	**36,018**	**3.50**	**4,048**	**8,995**	**3.36**	**911**	**50,293**	**3.21**	**5,185**	
Main Zone	6/23/2016	Internal based on GSI	1.0 OP	9,172	2.13	630	15,361	2.54	1,253	2,478	2.26	180	24,533	2.39	1,883	Main Zone open pit
Contact Zone			1.8 UG	3,215	3.05	315	25,844	3.63	3,013	6,068	3.79	740	29,059	3.56	3,328	Underground combined zones
Tulkubash			0.50	12,902	1.41	583	5,911	1.24	236	2,124	1.36	93	18,813	1.35	819	
Total			**-**	**25,289**	**1.88**	**1,528**	**47,116**	**2.97**	**4,502**	**10,670**	**2.95**	**1,013**	**72,405**	**2.59**	**6,030**	
Contact Zone	12/31/2018	Tetra Tech	2.00	6,722	3.26	681	32,794	3.79	3,864	6,611	4.05	832	39,516	3.70	4,545	Main & Contact zones combined
Tulkubash		Internal	0.30	5,660	1.35	246	36,300	1.18	1,378	2,330	0.46	33	42,000	1.20	1,624	
Total				**12,382**	**2.39**	**927**	**69,094**	**2.42**	**5,242**	**8,941**	**3.11**	**865**	**81,516**	**2.41**	**6,169**	

	Press Release Date	Source	Cut-off Grade (Au g/t)	Measured			Indicated			Inferred			Measured & Indicated			Notes
Contact Zone	02/19/2020	Tetra Tech	2.00	6,722	3.26	681	32,794	3.79	3,864	6,611	4.05	832	39,516	3.70	4,545	Main & Contact zones combined
Tulkubash		Internal	0.30	5,266	1.28	216	18,080	1.21	702	910	0,90	26	23,346	1.22	916	
Total				**11,988**	**2.33**	**897**	**50,874**	**2.79**	**4,566**	**7,521**	**3.55**	**858**	**62,862**	**2.70**	**5,461**	

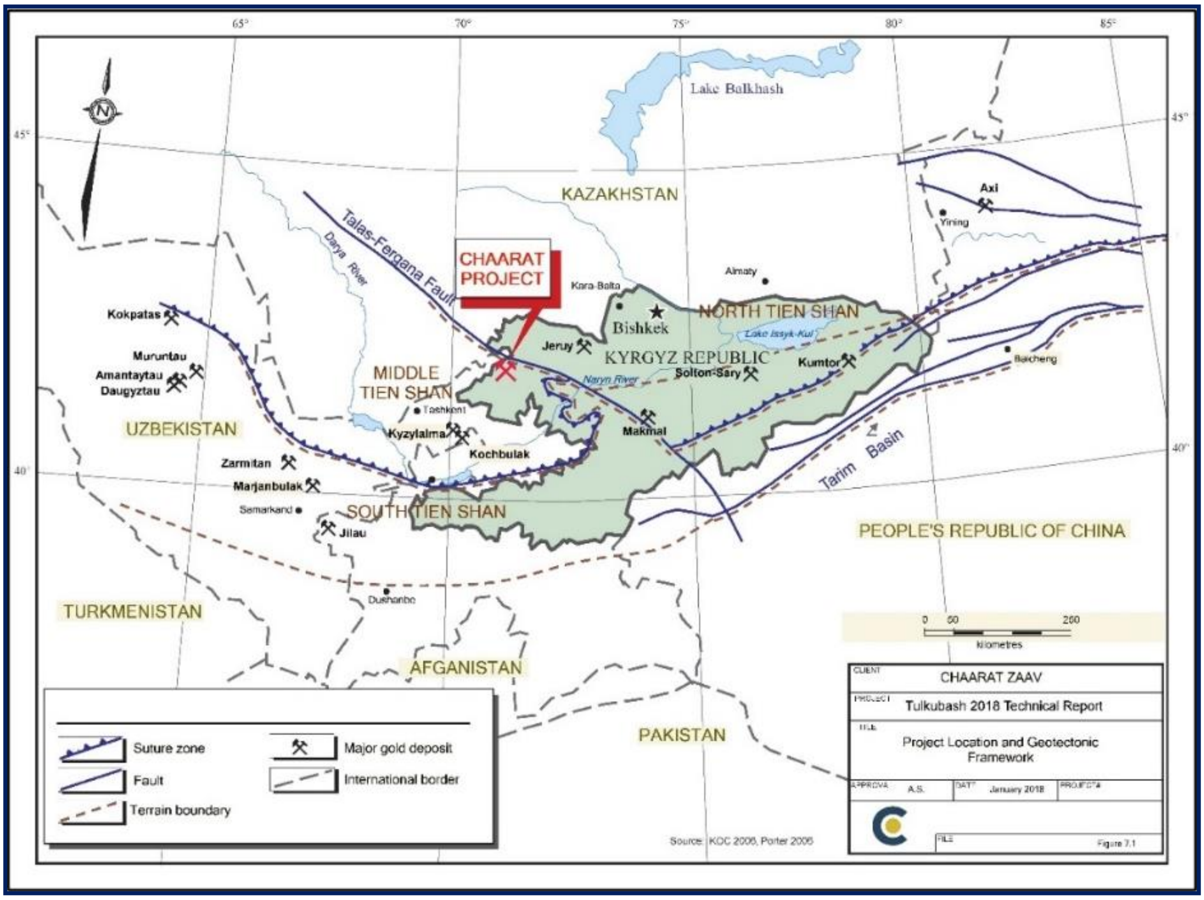
Information for Press Release Date 3/18/2013 unavailable.

7. Geological Setting and Mineralisation

7.1. Geological Setting

The Chaarat Property, located within the Middle Tien Shan Province, locates within the Tien Shan Metallogenic Belt, a Hercynian fold and thrust belt that crosses Central Asia, from western Uzbekistan in the west through Tajikistan and Kyrgyzstan into north-western China, a distance of more than 2,500 km (Figure 7-1). This belt contains many important gold deposits including the Muruntau (one of the largest gold deposits in the world), Zarmitan, Jilau, and Kumtor (Porter 2006). The Tien Shan Belt is divided into three, east-west-trending tectono-stratigraphic units: The Northern, the Middle, and the Southern Tien Shan. Each is separated by a major structural zone and are thought to represent accretionary prisms on the margin of the proto-Eurasian continent that was active from the Proterozoic to the end of the Permian.

Figure 7-1 Chaarat Project Geological Setting

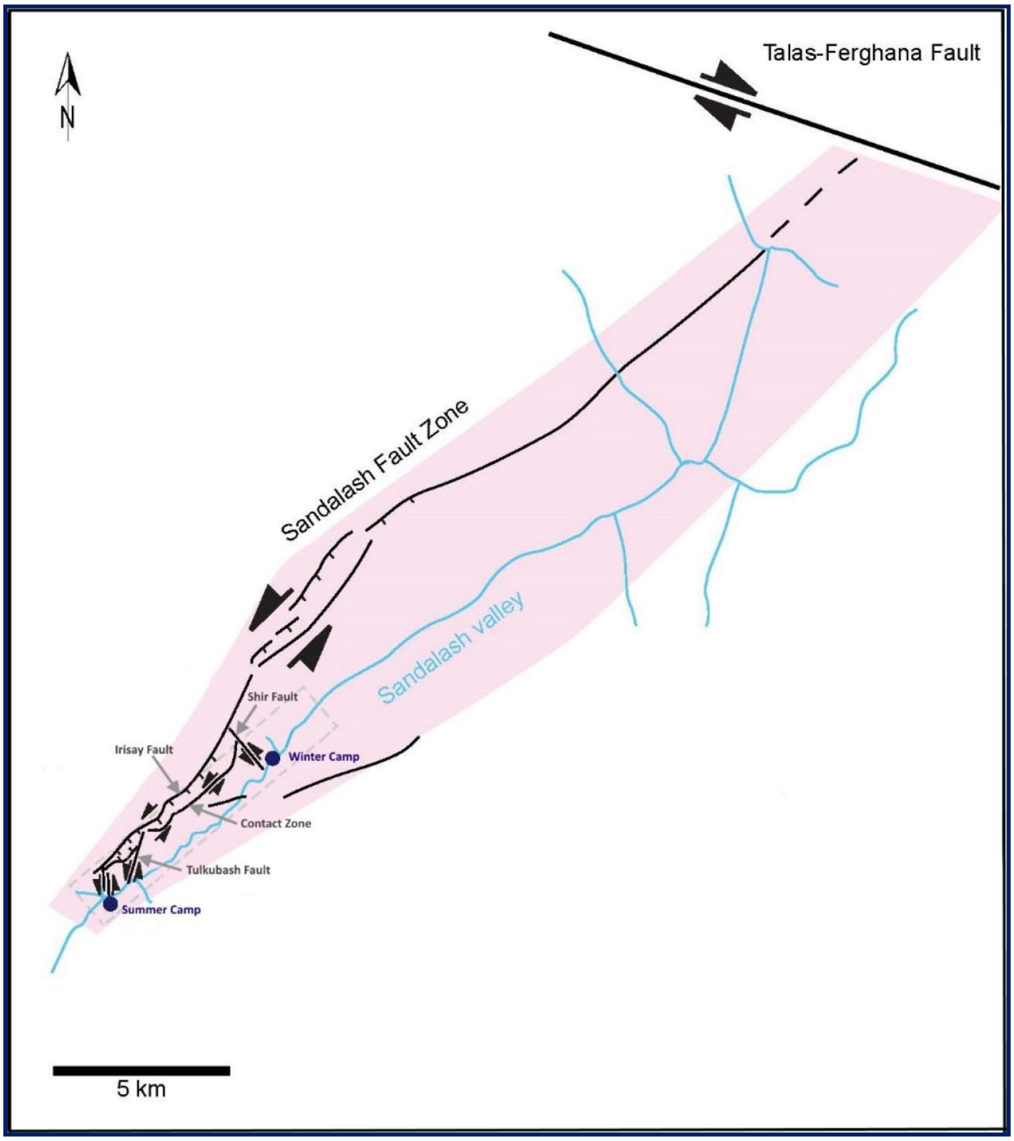


The Middle Tien Shan Province is made up of fragments of Late Devonian-Carboniferous rocks deposited in a forearc accretionary complex that was subsequently subjected to intense folding and thrusting during the upper Palaeozoic. The Middle Tien Shan hosts some of the largest orogenic gold deposits in the world with ages that range from Lower to Upper

Palaeozoic. These deposits are typically associated with Permian-age magmatism in carbon-rich sedimentary rocks (Cole and Seltmann 2000).

The structural evolution within the Chaarat District is closely linked to the tectonic history of the Talas-Fergana Fault (TTF). The TTF is the region's major structural feature extending northwest-southeast over a distance of 2,000 km and exhibits a maximum dextral offset of approximately 200 km (Rolland et al. 2013). The Chaarat District is located 35 km southwest of the TTF within the Sandalash Fault Zone (SFZ) (Figure 7-2). The Sandalash Fault Zone (SFZ) exhibits sinistral shearing which formed in response to displacement of the TTF.

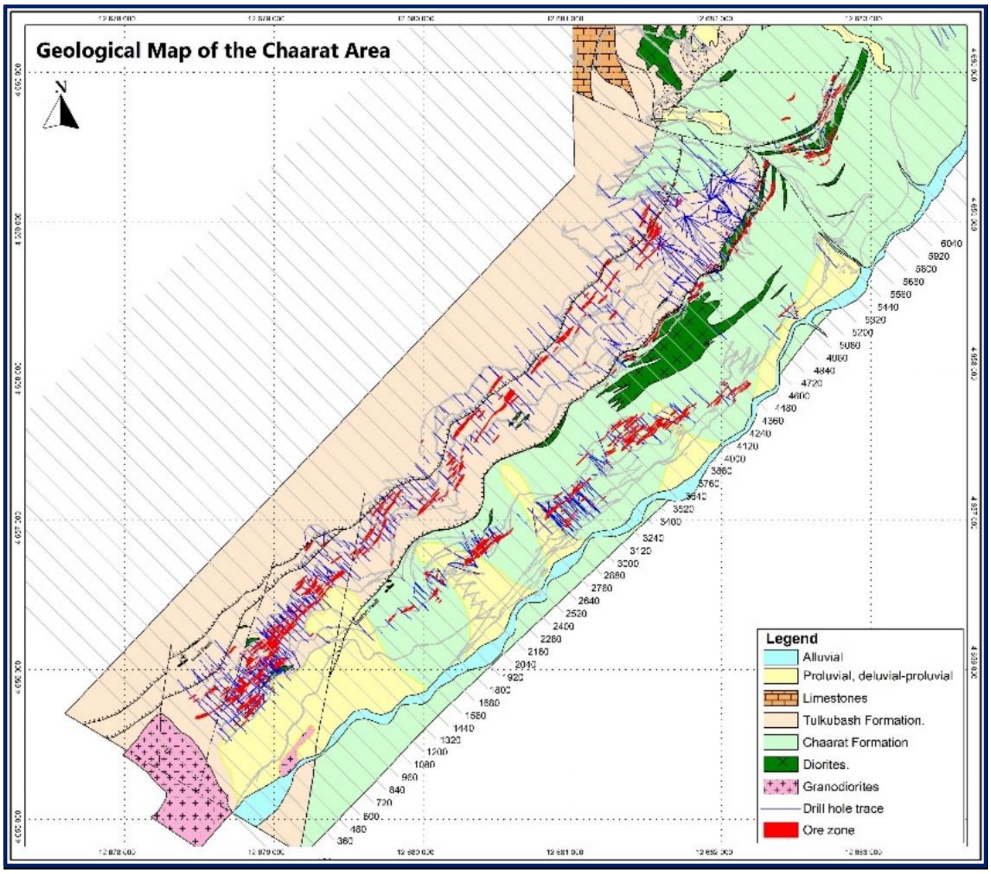
FIGURE 7-2 SANDALASH FAULT ZONE SCHEMATIC MAP



7.2. CHAARAT PROPERTY GEOLOGY

The Sandalash River valley down cuts a northeast-trending sequence of Cambro-Ordovician siliciclastic sediments which comprise the Chaarat Formation. This in turn is overthrust by a sequence of younger Devonian-age quartzites which make up the Tulkubash Formation (Figure 7-3). The sedimentary rocks hosting mineralisation strike north-easterly and exhibit dips between 40° and 75° to the northwest. Younger, Permo-Triassic-age granodiorite and diorite phases intrude the sediments and are closely associated with the gold mineralisation and, in some areas, are themselves mineralised.

FIGURE 7-3 CHAARAT PROPERTY AREA GEOLOGICAL MAP



7.2.1. CHAARAT FORMATION

The Chaarat Formation is made up of three members which exhibit a sequential package of alternating, moderately- to well-bedded, dark coloured, siltstones, shales, quartzites, and greywackes, with minor limestone interbeds (Cats et al. 2012).

The lower member is up to 170 m thick, consisting of grey siliceous siltstone interbedded with minor dark siltstone and shale.

The middle member is approximately 300 m thick. It consists of interbedded fine- and medium-grained sandstones, greywackes and siltstones, with a basal zone consisting of lenticular beds of polymictic gravely conglomerates and sandstones.

The upper member is dominated by shales and rhythmically interbedded siltstones and fine-grained sandstones which commonly exhibit graded bedding. The member is 70 m to 90 m thick whereas the thickness of individual beds ranges between 1 m and 2 m.

7.2.2. TULKUBASH FORMATION

The Tulkubash Formation is up to 1,000 m thick and consists of medium-grained to fine-grained quartzites and medium- to coarse-grained arkosic sandstones, with occasional thin interbeds of dark pyritic shales and siltstones. Quartzite beds range between 10 cm and 1 m in thickness, with the thicker beds predominating. Individual quartzite beds are generally massive and internally homogenous, with the occasional compositional layering of dark laminae alternating with lighter quartz-rich layers. The base of the Tulkubash Formation is generally identified by a conglomerate unit. Within the Chaarat Property area, the upper and lower contacts are faulted contacts.

7.2.3. Structure

The Chaarat Property lies within the Sandalash Fault Zone (SFZ) (Figure 7-2), a zone defined by a series of subparallel brittle shear zones that are the result of the local, predominantly sinistral strike-slip, displacement of the SFZ. The gold mineralisation occurs in various extensional structures, related to pressure relief during faulting (Kramer 2009; Jakubiak 2017). The SFZ comprises three mineralised fault zones, namely the Tulkubash Structural Zone, the Contact Fault, and the Main Zone Fault as well as one unmineralised zone called the Irisay Fault (Figure 7-2).

7.3. Mineral Deposits

Gold mineralisation within the Chaarat Property is divided into two styles of mineralisation:

- The Kyzyltash mineralisation, which is divided into the Main and Contact zones. This mineralisation is sulphide-rich and refractory; and
- The Tulkubash mineralisation, which is oxidised and can be processed through conventional heap leach methods.

The Tulkubash mineralisation is the primary subject of this feasibility study; however, the Kyzyltash mineralisation is briefly described for completeness.

7.3.1. Tulkubash Zone

The Tulkubash zone (Figure 7-4) is a mineralised structural zone that trends northeast-southwest and dips steeply 55° to 75° to the northwest. The Tulkubash zone is interpreted to be a brittle shear zone that developed as the result of predominately sinistral strike-slip motion within the SFZ. Gold mineralisation within the Tulkubash zone occurs within zones of intense silicification and quartz flooding, which form individual gold-bearing lodes that can range from 5 m to 45 m in true thickness. Where multiple lodes are present, the Tulkubash zone can have a width of up to 250 m with the individual lodes separated by unmineralised country rock (Figure 7-5). Development drilling of the Tulkubash deposit indicates that the zone is remarkably continuous, however its thickness does vary along strike.

Figure 7-4 Tulkubash Deposit Geology

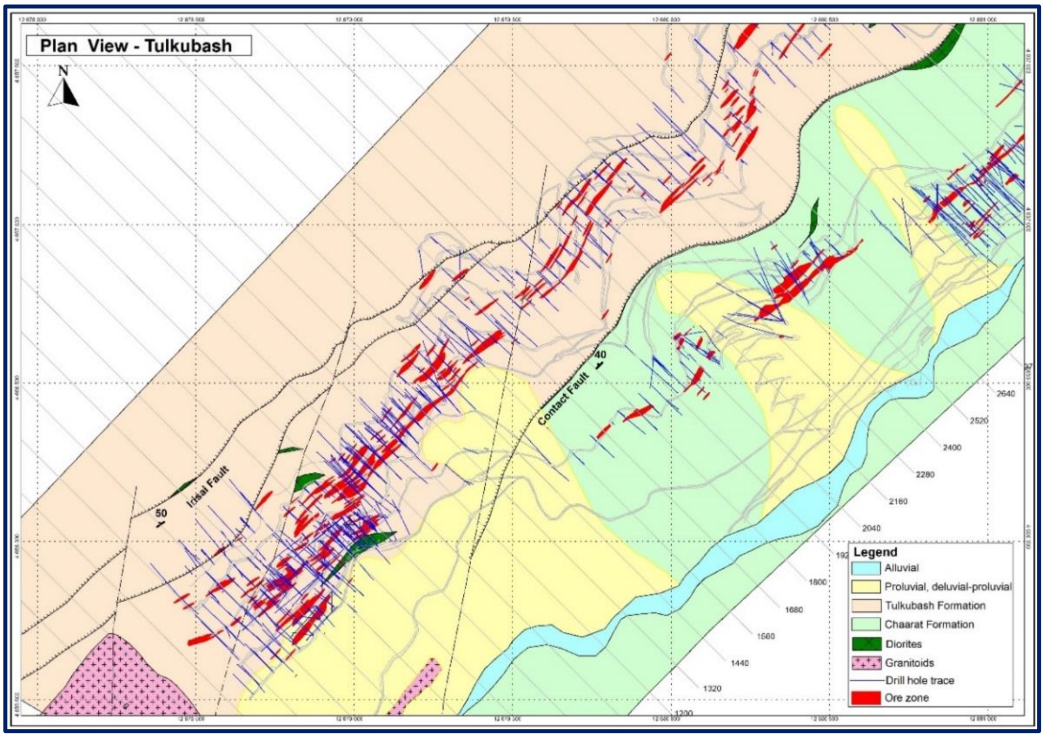
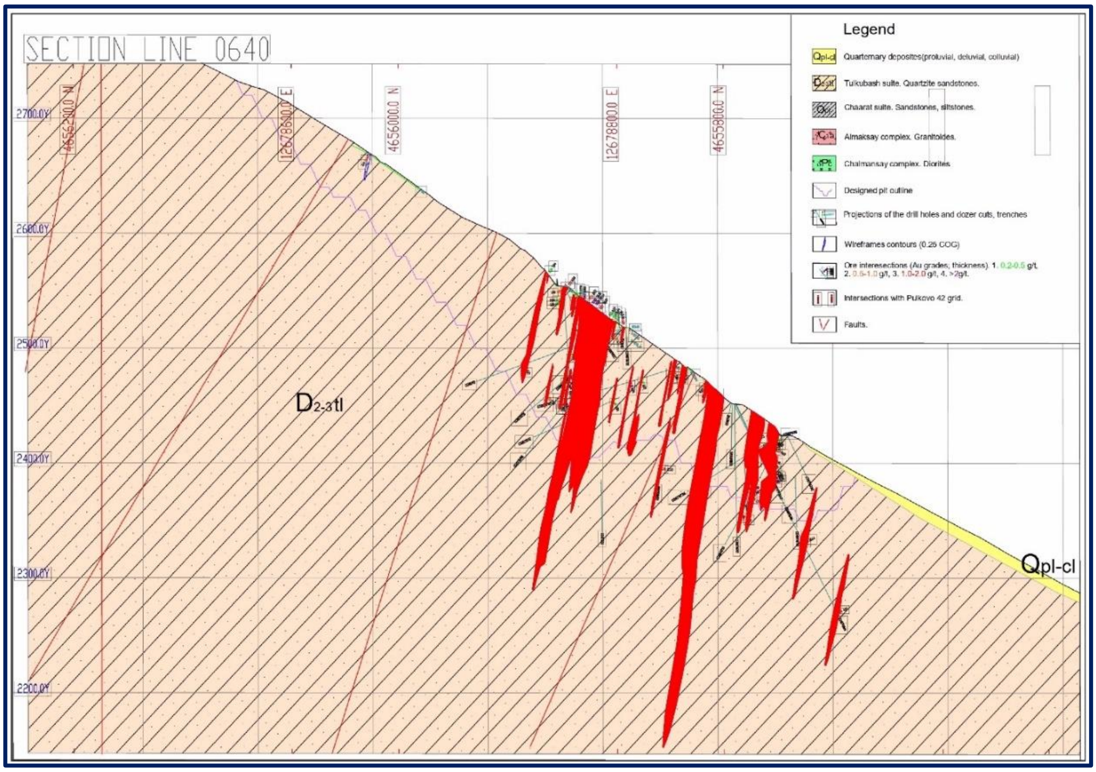


Figure 7-5 Section 640 Illustrating Multiple Lodes +0.25 g/t Gold (Wireframes in Red)



A distinctive feature present in areas of strong mineralisation are ovoid shaped hydrothermal breccias which are interpreted as fossilised steam vents. They form resistant spires up to 10 m high and 5 m to 10 m in cross section. The breccias are clast-supported with less than 5% carbonate cement and are easily identified in outcrop by the distinctive preferential growth of lichens on the carbonate cement.

The breccias are typically barren but occur within areas of strong gold mineralisation. Gold-bearing lodes are characterised by red and red-brown hematitic iron oxides, with minor yellow-brown limonite, and rarely occurring jarosite and stibiconite.

The Tulkubash zone is largely oxidised with low oxidation material occurring at the base and more strongly oxidised material at the top. The contact between unoxidised sulphide ore and oxidised ore can be gradational but is often observed with a sharp contact, suggesting at least some of the oxidation is hypogene.

Bulk flotation testwork, conducted on the Tulkubash sulphide ores, indicates that the main sulphide mineral is pyrite with subordinate arsenopyrite. All other sulphides occur in trace amounts and consist primarily of stibnite, molybdenite, sphalerite and galena (Sehlotho and Bryson 2012). The predominant gangue mineral is quartz with subordinate mica, dolomite, and ankerite. Metallurgical testing and cyanide soluble gold assays indicate that most of the developed mineralisation is amenable to extraction via heap leach.

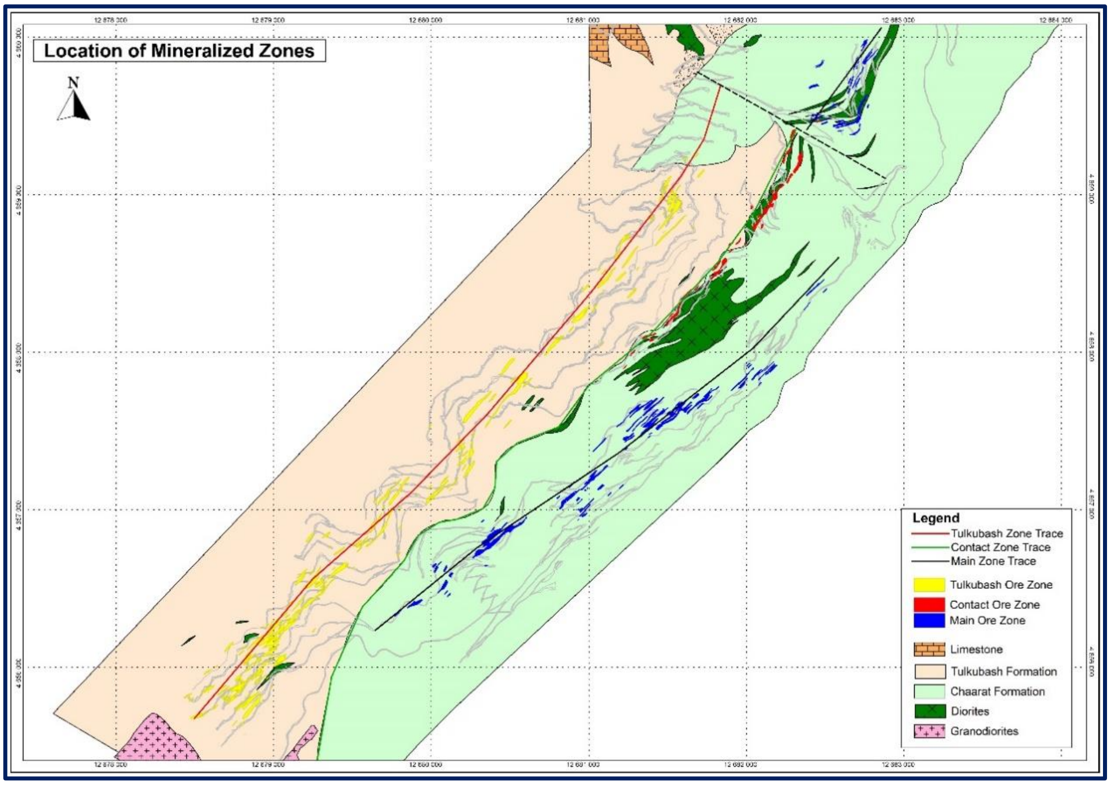
Using Tulkubash composites, gold particles are identified in heavy liquid separates (Kirchner and Coetzee 2011). The gold occurs as electrum containing a low silver content, typically ranging between 4% and 8%, with a few grains at 16% silver. Silver was also observed as silver-rich tetrahedrite and within a silver-rich lead-antimony-sulfosalt.

The widespread silicification and deep oxidation is in distinct contrast to the Kyzyltash zone, where minor quartz occurs in thin veinlets with no significant oxidation.

7.3.2. Kyzyltash Zone

The Kyzyltash zone is a series of sulphide-bearing ore bodies made up of the Main zone and Contact zone mineralisation (Figure 7-6) and locates to the East and northeast of the Tulkubash Zone. The mineralised zones occur within two subparallel northeast-trending structural zones that have been traced for 10 km along strike. The ore consists of gold-arsenopyrite-stibnite-tetrahedrite mineralisation occurring in sheared and altered wall rock. The ore exhibits strong sericitic alteration, with lesser amounts of quartz, quartz vein stockwork, ankerite, and calcite gangue. In some areas, antimony and silver are significant constituents of mineralisation, the latter particularly in the Contact zone and in the M7000 ore body (about 21 g/t silver average). Antimony, in stibnite and various sulfosalts, can locally reach values of 10% or more over 1 m to 2 m thick zones. Trace amounts of copper and molybdenum are also present in some of the ore.

FIGURE 7-6 LOCATION OF KYZYLTASH (MAIN ZONE AND CONTACT ZONE) ORE BODIES



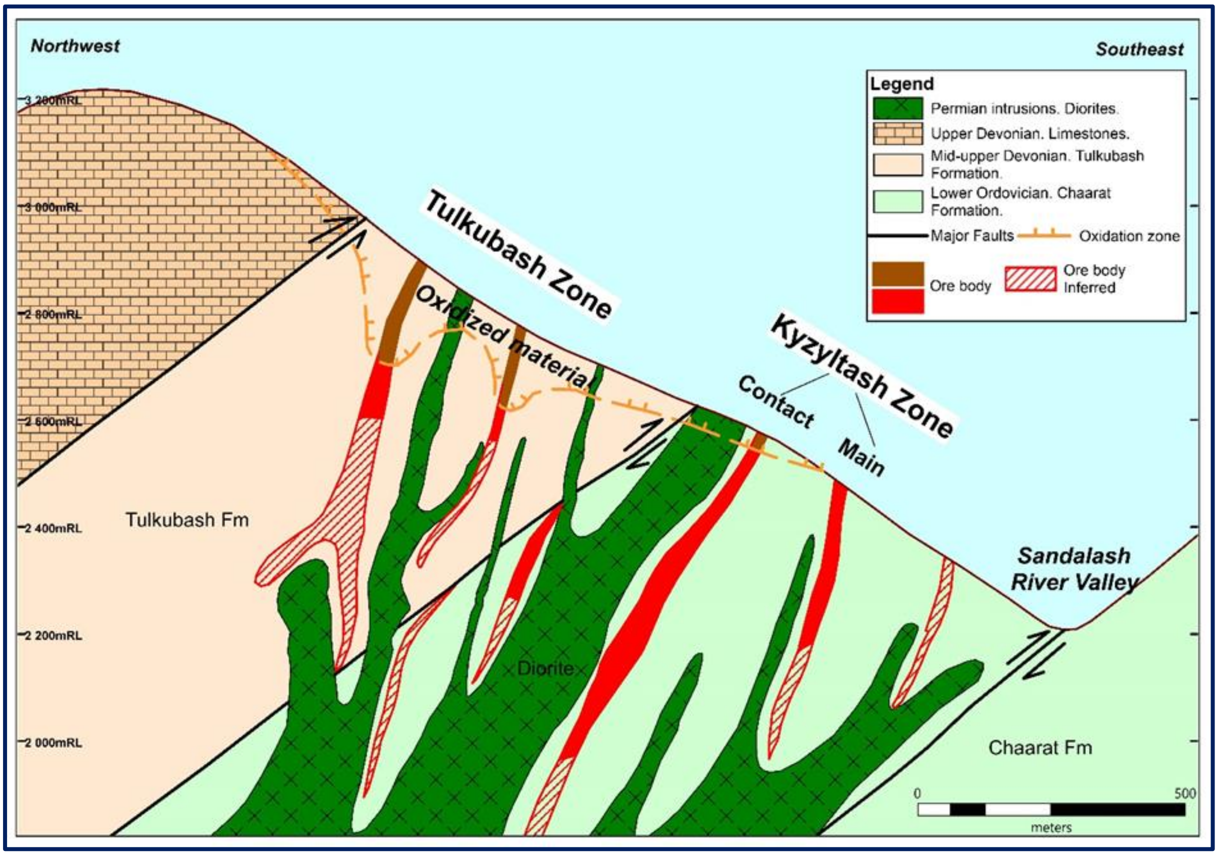
Petrographic work completed by Chaarat on more than 50 thin sections showed that free gold is present in the ore and occurs as inclusions in quartz and arsenopyrite. The gold mineralisation is, to some extent, correlated with arsenic, which mostly occurs as arsenopyrite. In some localised zones, there are very high silver values (greater than 400 g/t silver). The distribution of silver values is not fully understood, and transitions from silver-rich areas to silver-deficient areas can occur over distances of less than 20 m along strike.

Mining of the Kyzyltash Zone has not been investigated as part of this Feasibility Study, but is mentioned for completeness. With a strike in the order of 10 km and the deposit being open at depth based on drilling to date, this forms a significant target for future mining potential.

8. DEPOSIT TYPES

Mineralisation and associated hydrothermal alteration at Chaarat are genetically associated with igneous intrusive rocks along a system of regional-scale, sinistral, oblique-slip faults. Within this setting, there are two distinct types of mineralisation: the Tulkubash-type and the Kyzyltash-type. However, the proximity of the two types of mineralisation and the common structural controls suggest that both were the result of a common hydrothermal event (Figure 8-1).

FIGURE 8-1 CHAARAT CONCEPTUAL CROSS SECTION OF TULKUBASH AND KYZYLTASH MINERALISATION TYPES

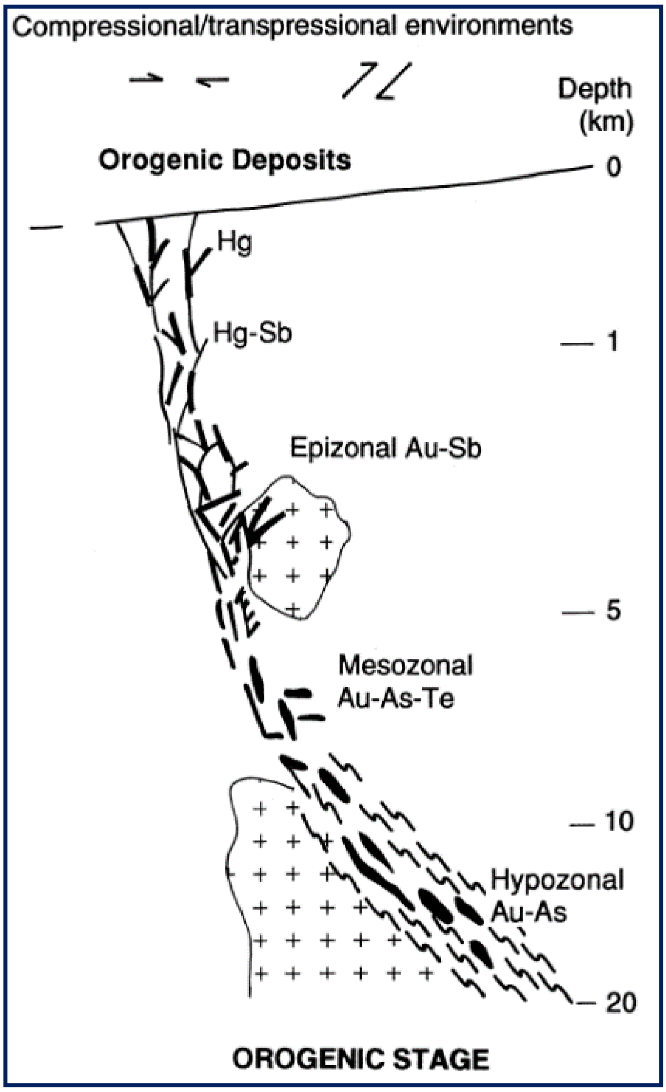


Colliform textures in the Tulkubash zone, along with widespread oxidation, silicification, and the geochemical association of gold with antimony and arsenic, indicate a shallow epithermal setting analogous to sediment-hosted deposits. According to Groves et al. (1998), the Tulkubash deposit is classified as an epizonal orogenic gold deposit (Figure 8-2).

Mineralisation in the Kyzyltash zone formed in a much deeper environment. The pervasive sericitization, disseminated sulphides and ankeritization within mineralised lodes, and the relative paucity of quartz veins (usually less than 5% of volume), indicate the prevalent mode of deposition was controlled by the reaction of reduced hydrothermal fluids with wall rocks. These zones are classified as mesozonal orogenic gold deposits. These deposits are formed in nearly isothermal conditions and can extend to great depths. Mineralisation in the Contact zone has been drilled over a vertical range of 1.3 km and is open at depth and along strike.

If the two types of mineralisation are related by a common hydrothermal system, it implies that the Tulkubash zone transitions to mesozonal-style mineralisation at depth and represents a deep, underground exploration target.

FIGURE 8-2 SCHEMATIC REPRESENTATION OF HYDROTHERMAL GOLD DEPOSITS AS A FUNCTION OF DEPTH IN THE CRUST (GROVES ET AL., 1998).



9. EXPLORATION

Mineralisation within the Chaarat Project area was first identified by Soviet-era soil and stream-sediment sampling, as part of a geochemically anomalous zone that extends for more than 40 km along the Sandalash Valley. Their work identified 28 separate areas of anomalous gold content and a similar number of tungsten (W), molybdenum (Mo), copper (Cu), lead (Pb), zinc (Zn), silver, arsenic, and antimony anomalies, most of which have not yet been investigated.

9.1. TULKUBASH ZONE EXPLORATION

In 2004, Chaarat completed a soil sampling programme along the strike of the Tulkubash zone. The survey consisted of soils collected every 40 m along irregularly spaced traverse lines that extended down the ridge. The results of the soil survey outlined numerous gold anomalies of greater than 1 g/t gold over a 4 km strike length, with the maximum value of 73 g/t in one sample. These anomalies range from 100 to 800 m in length (along strike) and 50 to 150 m in width.

In the Tulkubash deposit area, follow up trenches, and detailed rock chip profiles were collected, which defined a large, coherent geochemical anomaly. Subsequent drilling within the anomaly led to the discovery of the Tulkubash deposit.

Over the following years, additional rock chip, trench sampling, drilling and surface mapping has been completed along this trend. This work has continued to return positive results defining the so called Tulkubash Mid and Tulkubash East zones. (Figure 9-1).

9.1.1. TULKUBASH MID ZONE

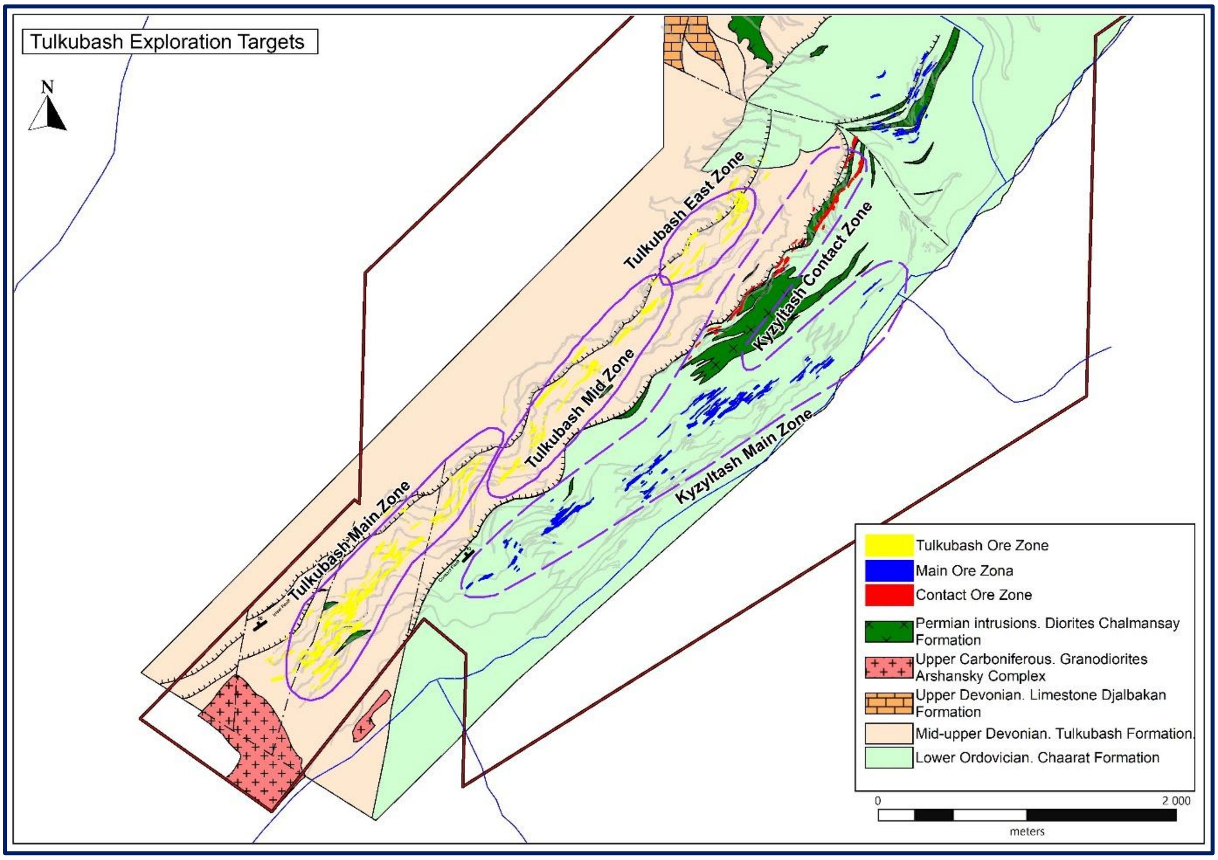
The Mid Zone is the natural northeast extension of Tulkubash, outlined based on assay results from soil sampling, trenching and diamond drilling in 2018 and 2019, supporting an estimation of Inferred Resources of the Tulkubash type. The contoured mineralisation is traced on about 1.5 km strike and presented by narrower individual oxidized loads, returning a reasonable leaching recovery above 70%.

The Zone is considered prospective for extending the Tulkubash resource/reserve and potentially increasing the life of mine.

9.1.2. TULKUBASH EAST ZONE

The East Zone is located 3 km northeast of the Tulkubash Main Zone and is outlined along 800 m of strike, confirmed by assay results in soil sampling, trenching and borehole intercepts mostly from the 2018 and 2019 exploration campaigns. Multiple high grade oxidized gold intercepts were outlined, confirming the Zone’s perspectivity and increasing the Tulkubash Mineral Resource. Additional drilling is required to improve the Mineral Resource definition.

FIGURE 9-1 TULKUBASH GEOLOGIC MAP WITH SUB SURFACE GOLD MINERALISATION AND THE OUTLINED EXPLORATION TARGETS



9.1.3. 2018-2020 TULKUBASH EXPLORATION PROGRAMS

A programme of exploration comprising 121 boreholes for a total of 19,821 m was completed during the 2018 season. The additional exploration added approximately 1 km to the explored strike of the Tulkubash deposit, taking the total explored strike to 3.3 km.

A field exploration programme of mapping and target generation was also completed in 2018, identifying further targets. Chip and grab sample results along this trend have continued to show anomalous values of potentially economic interest.

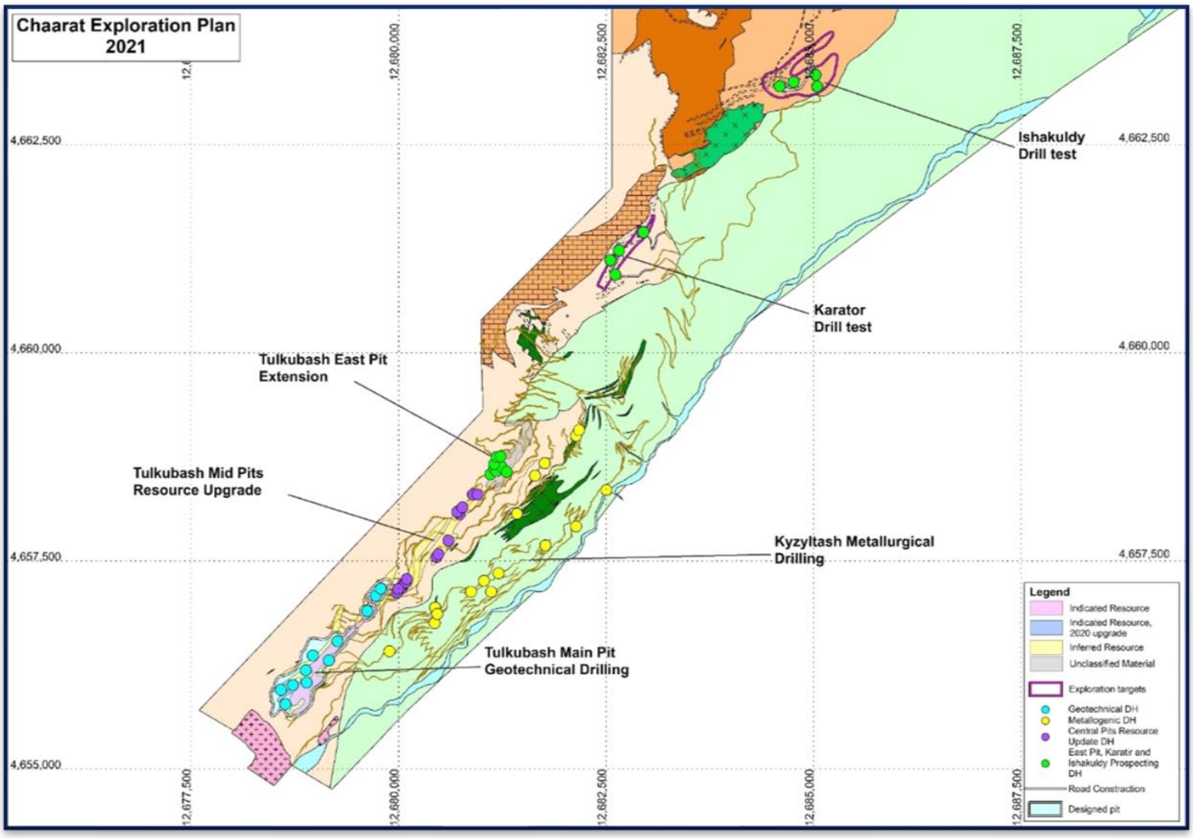
A programme of exploration comprising 130 boreholes for a total of 20,077 m was completed during 2019, while surface exploration continued with 86 ditches and 28 trenches.

Infill drilling of 21 RC boreholes comprising 2,432 m of sampling was undertaken during 2020 to confirm and upgrade the western portion of the deposit.

9.2. 2021 TULKUBASH EXPLORATION PLAN

The exploration potential of the outlined targets along the Tulkubash zone is considerable and may be equal to the currently known resources. Exploration and data collected has confirmed and outlined oxidized gold mineralisation of Tulkubash type, planned for resource upgrade and resource definition drilling in 2021, in Tulkubash Mid and Eastern Zones and initial drill testing of the recently outlined Mid Karator and Isakuldy targets (Figure 9-2 and 9-3).

FIGURE 9-2 TULKUBASH GEOLOGIC MAP AND 2021 EXPLORATION PLAN



Both the Mid Karator and Ishakuldy targets are located further northeast, approximately 5 km and 7 km respectively to the Tulkubash Main Pit and are considered to have the potential to host significant gold mineralisation of the Tulkubash type.

9.2.1. MID KARATOR

Mid Karator has a strike length of approximately 1,000 m by on average 70 m width and up to 150 m depth of expected reasonable oxidation.

The target is contour based on structurally complex shallow and steep dipping structures trending NE and ENE, overlapping a 100 m wide and NE striking Shear zone of intensive fracturing and brecciation. Grades of +0.5 g/t gold have been assayed in soil anomalies which overlap the NE striking shear zone, with high-grade samples up to 7.06 g/t being detected.

2018-2019 reconnaissance trenching returned consistent trench intercepts as: 21.7 m at 2.2 g/t gold in TR19T014; 8.2 m at 2.05 g/t gold in TR19T025; 16.7 m at 1.01 g/t gold in TR18T019 from oxidized silicified, brecciated sandstone.

The 2021 exploration program at Karator includes digging of 8 trenches across entire target width and drilling of 5 boreholes for testing.

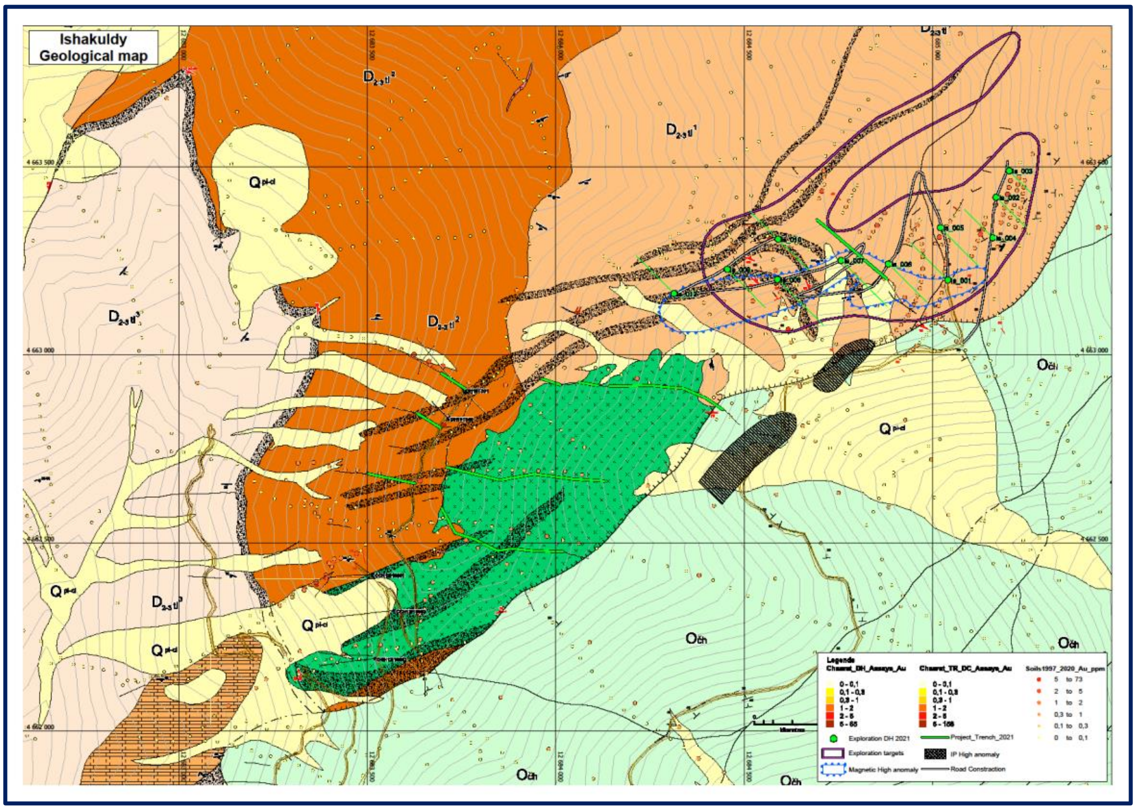
9.2.2. Ishakuldy

The Ishakuldy zone is located 7 km northeast of the Tulkubash Main Zone, close to the top of the ridge, approximately 1,000 m above the Sandalash River. Mineralisation was exposed by trenching and soil sampling over a strike length of approximately 2.5 km, with the highest gold values concentrated at the northeast and southwest ends of a tabular body of diorite, intruded along the contact between siltstones of the Chaarat Formation and quartzites of the Tulkubash Formation (Figure 9-3). Following these encouraging prospecting results, additional soil samples were collected over the northern end of the Ishakuldy zone, where the gold-in-soil anomaly, in excess of 0.5 g/t gold (up to 6.2 g/t), extends for more than 600 m along strike and 300 m across the strike. Reconnaissance soil sampling profiles along ridge-crest lines established the continuation of significant gold-in-soil anomalies (greater than 1 g/t gold) for a further 3 km north of Ishakuldy.

At Ishakuldy, gold mineralisation is associated with a 1,700 m by 500 m diorite stock intruded along the Contact zone, with the soil anomalies forming preferentially at the eastern and western ends of the diorite within the hanging wall of the Tulkubash Formation. Near the diorite contact, Trench 730-I contained 3.0 m at 16 g/t gold and 3.35% antimony, and Trench No.624 contained 3.0 m at 6.5 g/t gold, including 1 m at 15.8 g/t gold. The antimony and silver values in the rock samples are mostly very low (average 100 ppm antimony and 1 ppm silver), but the arsenic values were strongly anomalous (average 1,000 ppm arsenic) and showed a good correlation with the gold values.

The 2021 exploration plan includes digging of 6 trenches across the diorite intrusion and the entire target and the drilling of 4 boreholes for testing (Figure 9-3).

Figure 9-3 Ishakuldy Geology Map and 2021 Exploration Plan



9.2.3. 2021 Reconnaissance Exploration

Continued regional reconnaissance exploration has been undertaken since 2018 to evaluate the potential of north east strike mineralisation.

In 2021 is planned entire Chaarat exploration licence to be covered by drone based geophysical survey at 1:5000 scale including:

- Magnetic prospecting
- Gamma-ray surveying
- Resistivity prospecting

The main aim of that survey is generating quality and reliable geophysical anomalies to support:

- Understanding of Structural architecture of the ore field;
- Outline hydrothermal alteration zones, magmatic stocks & dyke swarms;
- Exploration target definition and prioritization;
- Focusing of surface & drilling exploration programs.

10. DRILLING

10.1. LOCATION OF DATA POINTS (SURVEY CONTROL)

All Project surveys use the Pulkovo 1942 datum and a Gauss Kruger projection. This is standard for Kyrgyzstan for consistency with government geologic and infrastructure databases. Appropriate conversions are available in the various commercial geographic information system (GIS) packages. All Project location data are in meters.

All surface boreholes have been surveyed by total station and are reportedly accurate to within centimeters. Underground drill collars have likewise been surveyed by total station with accuracies reportedly within centimeters.

All surface and underground boreholes have downhole surveys, typically taken at 15 m, and then every 50 m using REFLEX EZ SHOT™ electronic single-shot downhole survey equipment. Chaarat purchased the downhole survey equipment in 2013. Similar instrumentation had previously been rented on an annual basis from RELFEX™ UK. The equipment is serviced, and factory set for declination annually. Figure 10-1 shows equipment at an operational drill site.

FIGURE 10-1 SURVEY EQUIPMENT (2018)

Individual drillhole sample locations are assumed to be accurate to within a few meters or less, depending on depth downhole and relative deviation of boreholes. Underground sample line locations are surveyed using total station and are reportedly accurate to within tens of centimeters. There are minor differences between different software platforms in the handling of survey information for surface trenches, which can influence location accuracy. Because of this inconsistency, individual trench sample locations are assumed to be accurate to within a few meters.

Regional surface topography is derived from satellite data and shows significant variation (up to 50 m) from survey coordinates. The Mineral Resource area has been resurveyed using total station along roads, ridges, valleys, and additional traverses and the resulting surface elevation points have been contoured. Surface elevations from the resulting topographic surface correspond well to surveyed drill collar, trench sample, and drill road locations and are assumed to be accurate to within less than 5 m.

10.2. Data Spacing and Distribution

Drill spacing is variable depending on road construction and access, but is typically 30 m to 40 m where access was available, extending to 80 m spacing along the flanks of the deposit. There are specific areas where spacing is larger or smaller due to drill fans, lack of access to specific elevations, or lack of access due to the availability of drilling roads. There are also gullies that have been covered with alluvial material in which drilling is difficult or not possible.

Drill sampling is typically done on 1.5 m intervals, except where the interval length is adjusted to accommodate changes in lithology, alteration, or mineralisation. For early boreholes, only intervals designated by project geologists as mineralised intervals are sampled and assayed.

Intervals not sampled are treated in the database as having zero assay value, as the intervals were not sampled when the project geologists considered the material to be non-mineralised. It is possible that this practise results in some intervals treated as barren that actually contain grade.

10.3. Orientation of Data in Relation to Geological Structure

Drilling lines were angled with a 42° east rotation to correspond with the orientation of the strike of the deposit. The majority of the boreholes were drilled as inclined boreholes in order to cut the mineralised structures as close to right angles as possible. Underground drilling and some early boreholes at the Tulkubash deposit were drilled parallel to strike, as they were targeted to test silicified zones visible on surface that are perpendicular to the primary structures controlling mineralisation.

In most areas, there is sampling in both mineralised and adjacent non-mineralised material, so there are no biases or artefacts observed in the database or interpreted geometries related to sampling orientation.

10.4. Drilling Techniques

Diamond boreholes were drilled as HQ size, except where poor ground conditions required reducing to NQ size. Triple-tube has also been used in areas where recoveries are low, particularly where quaternary deposits are loose and unstable. RC drilling was undertaken using a borehole diameter of 124 mm.

Drilling campaigns for the entire Chaarat Project have been carried out using various contractors and Chaarat-owned equipment (Figure 10-2).

Figure 10-2 Typical Rig Site at Tulkubash (2018)

Table 10-1 Tulkubash Gold Project Surface and Underground Drill holes and Samples

Year	Tulkubash Zone	
	No. of Holes	Total Metres
2000	-	-
2004	-	-
2005	1	150
2006	7	1,393
2007	12	2,374
2008	-	-
2009	5	802
2010	37	4,271
2011	128	15,984
2012	39	6,842
2013	14	1,781
2014	48	5,813
2015	-	-
2016	12	1,185
2017	135	17,420
2018	121	19,822
2019	130	20,077
2020	21	2,434
Total	**710**	**100,348**

10.5. DRILL SAMPLE RECOVERY

Core recoveries have been recorded for all core intervals since the beginning of the Chaarat Project. Chaarat drill contracts require that drill recoveries remain in excess of 90%, and allow Chaarat to request re-drilling of the hole if this standard is not met. Sample recovery in some friable mineralisation may be reduced; however, it is unlikely to have a material impact on the reported assays for these intervals.

Diamond core recovery is maximised via the use of triple-tube sampling and additive drilling muds. Diamond core recovery is recorded as a percentage of total length drilled, estimated directly from core box observations.

Analysis of duplicate sample performance does not indicate any chemical bias as a result of inequalities in samples weights or core recovery. Core recovery is not expected to have any material impact on the Mineral Resource estimation.

An overall average recovery of 79.70% was achieved with the RC drilling, with higher grade samples (>1.0 ppm) displaying an average recovery rate of 78.95%. No correlation was observed between recoveries and gold grade.

10.6. LOGGING

All drill core has been logged for lithology, oxidation, veining, primary alteration, hardness, alteration intensity, fracture density, mineralisation (relative sulphide content), as well as graphical and descriptive logs. Core is inspected in the field at the rig site before transportation. (Figure 10-3).

FIGURE 10-3 CORE FIELD INSPECTION (2018)

The rock is described using a standardised set of alphanumeric and corresponding numeric codes. Logging is performed at nominal 1.5 m intervals, however when required, logging is done on shorter intervals, as well as across the mineralised zone's boundaries.

Descriptive logs contain a large amount of information which is often not recorded in a database format. Chaarat geologists photograph all drill core, and photographs are stored with the database for reference. Primary alteration, alteration intensity, fracture density, and relative sulphide quantity are recorded electronically from assay sheets to the database.

Subsequent data entry has added a relative oxidation code (from drill core photographs). GSI recommended during its site visit the incorporation of digital capture of all information potentially relevant to mineralisation, geotechnical characterisation, and geo-metallurgical characterisation.

Figure 10-4 Shows geologists logging core in the core shed.

FIGURE 10-4 LOGGING GEOLOGISTS (2018)

Rock chips from the RC drilling was collected as 1 m samples. Samples were split using a rifle splitter into samples of approximately 8 kg and duplicates.

Duplicate samples were collected for lithological logging and photography, with the samples being placed in trays marked with a permanent marker (Figure 10-5). Samples were logged for lithology, alteration intensity, alteration type, degree of disturbance, intensity of mineralisation, silicification and oxidation.

Figure 10-5 RC Samples (2020)

11. Sample Preparation, Analysis and Security

11.1. Sub-sampling and Sample Preparation

Prior to sampling of the core, project geologists designate and mark sample intervals. Samples are typically chosen at 1.5 m intervals, but the sample interval can be altered to fit structural, alteration, or lithological contacts. Core samples are split on site using a diamond saw (Figure 11-1) for competent core pieces, with highly-fractured intervals split manually.

Figure 11-1 Core Saw (2018)

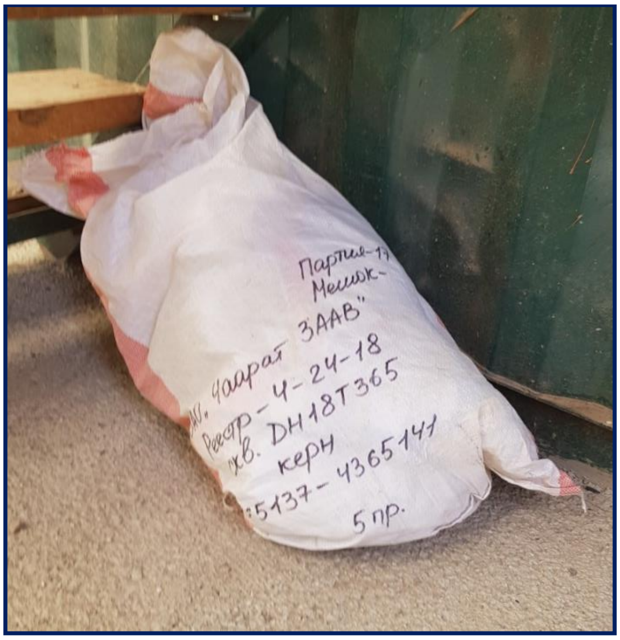
The saw uses fresh, clean, running water and is allowed to run to wash down between samples. The saw is thoroughly cleaned between batches.

One half of the core is selected and bagged for assay in individually labelled polyethylene bags (Figure 11-2). The polyethylene bags are top rolled and then stapled, weighed, and packaged in rice sacks, with five to six samples per rice sack.

Figure 11-2 Polythene Sample Bags (2018)

Individual rice sacks are tied with wire, labelled, and set aside for pickup by project logistics personnel for transport to the laboratory (Figure 11-3).

FIGURE 11-3 BULK SAMPLE SACK (2018)



The second half of the core is retained in numbered and labelled wooden core boxes for future reference and possible additional analysis. These core boxes are picked up by project logistics personnel and transported to the Chaarat core storage facility in Bishkek. (Figure 11-4).

FIGURE 11-4 CORE IN STORE, BISHKEK (2018)

The RC drilling is sampled in 1 m intervals and split into 8 kg samples and duplicates. The samples for analysis are individually sealed in polyethylene bags for transport to the laboratory.

Five laboratories have been used for assaying during the life of the Project: IRC in Kara Balta, a Kyrgyz laboratory that is International Organization for Standardization (ISO) 9001:2008 certified by Bureau Veritas; Stewart Assay and Environmental Laboratories in Kara Balta, a subsidiary of ALS (ALS-Stewart), which is also ISO 9001:2008 certified; and Genalysis Laboratory Services Pty Ltd. (Genalysis) of Perth, Western Australia, a subsidiary of Intertek, which is ISO/International Electrotechnical Committee (IEC) 17025 Certified by the National Association of Testing Authorities (NATA). The fourth laboratory is Central Scientific Research Laboratory (CSRL) located in Kara Balta; however, this laboratory has not been used since 2007. The fifth laboratory, SGS Vostok Limited located in Chita, Russia (ISO/IEC 17025) has been used as a check laboratory since 2017.

IRC was used until 2017 as the preparation laboratory and to pre-screen mineralised intervals. ALS-Stewart is currently the main assay laboratory, handling all sample preparation and analysis and prior to 2017 analysing all mineralised samples assaying above 0.3 g/t of gold from the IRC laboratory. Selected samples were submitted for referee assay by Genalysis, and since 2017 SGS Vostok. A review of laboratory duplicate samples shows good agreement between different assay laboratories.

Upon arrival at the laboratory, samples were dried and crushed in a laboratory jaw crusher to 100% passing 2 mm (10 mesh), and two, 120 g to 150 g manual splits were taken. These subsamples were pulverised to –0.075 mm (200 mesh) in a ring and puck pulveriser and stored in numbered paper packets. One packet was sent to the IRC assay laboratory where a 2 g split was assayed for gold using aqua regia digestion with an atomic absorption (AA) finish.

Assay values from IRC were reported electronically to Chaarat. The second packet, along with coarse reject material, was shipped to the Chaarat's core storage facility in Bishkek. For those samples over 0.3 g/t of gold from IRC, the sample packets were transported to ALS-Stewart, also in Kara Balta (approximately ½ mile by road). ALS-Stewart logged the samples into the Laboratory Information Management System (LIMS), re-labelled the sample packets with ALS-Stewart barcoded labels, and assayed for gold using 30 g charge FA, aqua regia digestion, and AA finish. ALS-Stewart procedures appear generally more automated than those of IRC laboratories. ALS-Stewart assay values are also reported electronically to Chaarat.

11.2. Sample Security

Samples were collected by project logistics personnel for transport directly to the IRC preparation laboratory in Kara Balta (since 2017 samples are transported directly to ALS-Stewart laboratories for analysis). Project logistics personnel also collect the core boxes on site and transport them to the core storage facility in Bishkek. Laboratory personnel transport samples between IRC and ALS-Stewart laboratories. Project logistics personnel pick up samples from IRC and/or ALS-Stewart for transport to the core storage facility in Bishkek. No special arrangements for sample security were noted during the site visit, but samples remain under direct control of Chaarat staff from drilling through delivery to the assay laboratory, and from pickup at the assay laboratories until delivery to the core storage facility.

12. DATA VERIFICATION

No twinned drilling has been conducted at Tulkubash. However, there are areas in Tulkubash where different drillhole and sampling orientations make statistical analyses comparing gold grades from different samples at a very close distance (less than or equal to 1 m) possible. These analyses were completed for the 2014 Mineral Resource Model update (GSI 2014) and showed very good correspondence and reproduction between samples from different boreholes, and also when comparing samples from surface boreholes, underground boreholes, and channel samples.

There is a QA/QC process in place that must be followed prior to accepting a batch of assay results from the laboratory.

Significant intercepts are routinely re-assayed to confirm higher-grade intercepts. Sample blanks are inserted into the sample stream at site at a rate of one blank per 18 samples. The blank material used is non-mineralised limestone; however, it is preferable that the blank material has the same matrix as the regular mineralised samples.

Standards are inserted into the sample stream at a rate of one standard per 18 samples. Pulps used by Chaarat are commercially certified reference material from Geostats (Pty) Ltd, Malcolm Smith Reference Materials, and Rocklabs. A review of the available blanks, duplicates, and standards data for the entire Chaarat Project has been undertaken.

The laboratories send the information back to Chaarat electronically, which is stored in secured Microsoft® Excel spreadsheets. It is recommended that Chaarat implement an SQL-based, relational database for the Chaarat Project to enhance further data quality and security.

12.1. QUALITY CONTROL AND QUALITY ASSURANCE OF ASSAY DATA AND LABORATORY TESTS

12.1.1. BACKGROUND AND HISTORICAL QUALITY ASSURANCE/QUALITY CONTROL

The Chaarat database prior to the 2018 Mineral Resource update has been constructed such that the Genalysis data take precedence over other data, the ALS-Stewart data take second precedence, and local (IRC) laboratory data are used when neither of the other laboratory data is available. In instances where there are multiple assay values for a given sample interval from the same laboratory, the available assay values are averaged to generate a final value.

These procedures follow Kyrgyzstan standards and are not consistent with generally accepted best practices in two regards. First, standard industry practice would use the first response from the laboratory, except in cases where there is a demonstrable problem with the initial assay. This differs from the methodology used in that all available assay values from a single sample are averaged to select a grade. Selection of samples for duplicate analysis tends to be driven by a minimum grade threshold.

The second difference is in the use of referee laboratory data to replace primary laboratory data. Standard industry practice is to confirm the primary laboratory results with a check laboratory, but not to replace the data. Both the selection of samples for ASL-Stewart and the selection of samples for Genalysis are based on a minimum grade threshold.

This practise was discontinued since the 2018 Resource Model update.

In the 2017 season, a small number of pulp duplicates were sent to SGS Vostok.

12.1.2. 2014 Quality Assurance and Quality Control

Sample blanks were inserted into the sample stream at site at a rate of one blank per 18 samples. The blank material was non-mineralised limestone. Sample blanks were used as a check to ensure there was no contamination between sample intervals in the preparation laboratory. Using easily distinguishable limestone blank material makes it obvious to the preparation laboratory which samples are blanks and which samples are drill core. Preferably, blanks should have the same matrix as the regular mineralised samples.

Pulp standards were inserted into the sample stream upon transport from IRC to Stewart. Standards were submitted at a rate of one standard per 18 samples. Chaarat used pulps from commercially certified reference material from Geostats (Pty) Ltd, Malcolm Smith Reference Materials, and Rocklabs.

Review of the available blanks and standards data for the entire Chaarat Project reveals very good results. Only two blanks (of over 2,201) were not recorded as “below detection limit” (less than 0.05 g/t of gold). One of these had a value of 0.073 g/t of gold (less than two times the detection limit) and the other had a value of 0.126 g/t. It was recommended that Chaarat use blank material in the sample stream which is less readily identifiable, to confirm the blanks performance.

It was apparent from the certified reference materials (standards) that there has been significant improvement in laboratory performance, particularly when the primary laboratory used was shifted from CSRL to ALS-Stewart in 2008. The outlier standards in 2011, 2012, and 2013 are Rocklabs standard G901-10, which Gustavson (2014) has seen to return erratic values in previous standards evaluation work, according to the comment made in his report. GSI could not confirm the comment, but recommends that in future drilling standards from multiple sources be used.

12.1.3. 2017 Quality Assurance and Quality Control

GSI completed a thorough review of the 2017 quality assurance (QA)/quality control (QC) programme. In general, the results of the QA/QC were considered good to very good and although no reason for concern was raised, improvements in methodologies were recommended including:

- Blank samples showed anomalous results in August of 2017, likely due to contamination. These samples were not re-assayed according to the QA/QC protocols and it was reported that the laboratory was not notified;
- A lower grade standard sample, more appropriate with the average grade, than those used should be procured and inserted into the sample stream alternately with the high-grade standard;
- Use differing sample numbers in the sample stream, even for duplicate samples; and

- Batches of samples which have presented major failures should be sent for analysis by the control laboratory, including QA/QC samples, not only select samples.

12.2. 2018 Quality Assurance and Quality Control

Chaarat completed a thorough internal review of the 2018 quality assurance (QA)/quality control (QC) programme. This section summarises three main aspects of the overall QA/QC programme, - Standards, Blanks and Round Robbins.

12.2.1. Standards

Standard Reference Materials are inserted into the sample stream to test to accuracy of the analyses against known values. The following Standards were used for the Tulkubash 2018 QA/QC programme:

Table 12-1 QA/QC Programme Standards, 2018

Reference Material	Grade Au (g/t)
Rocklabs SE86	0.595
Rocklabs OxH139	1.312
Rocklabs OxF142	0.805
Rocklabs OxD127	0.459

The standards have been selected to reflect the typical mineralisation grades encountered at Tulkubash.

12.2.1.1. SE86

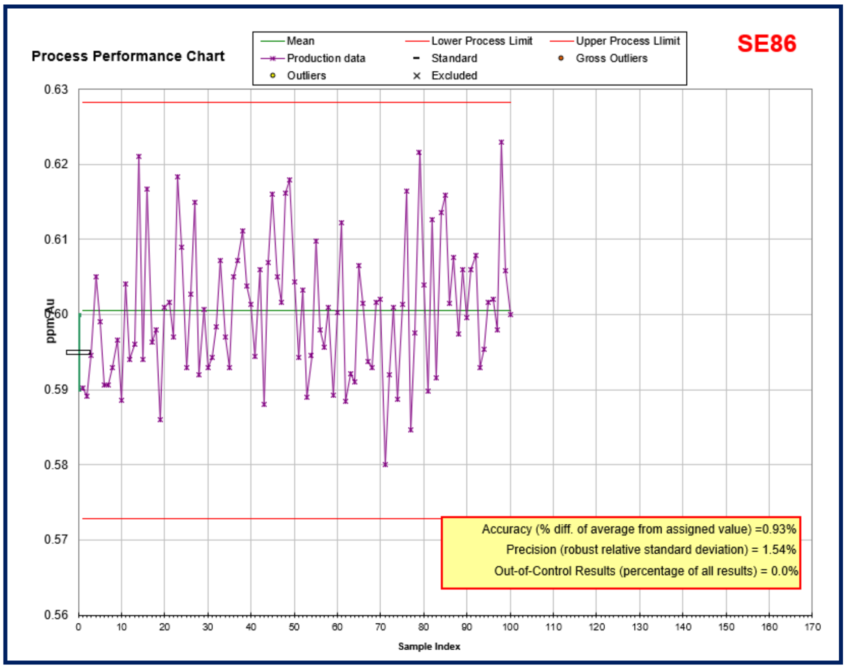
100 analyses were completed on standard SE86. Table 12-2 shows the statistics for the standard. Figure 12-1 shows the performance of the standard.

Table 12-2 SE86

Description	All Results	Gross Outliers Excluded	User Outliers Excluded	Comments
Number of Results	100	100	100	
Average	0.6005	0.6005	0.6005	
Accuracy: (% Difference of Average from Assigned Value)	0.9%	0.9%	0.9%	
Precision: Relative Standard Deviation (Robust)	1.5%	1.5%	1.5%	Good
Number of Outlying results (Outside Process Limits)	0	0	0	
Percentage of Outlying Results			0.0%	Good

SE86	
Assigned Value of Standard	0.595
95% Confidence Limits Standard (+/-)	0.005

FIGURE 12-1 SE86 PERFORMANCE



Zero failures were recorded.

12.2.1.2. OX127

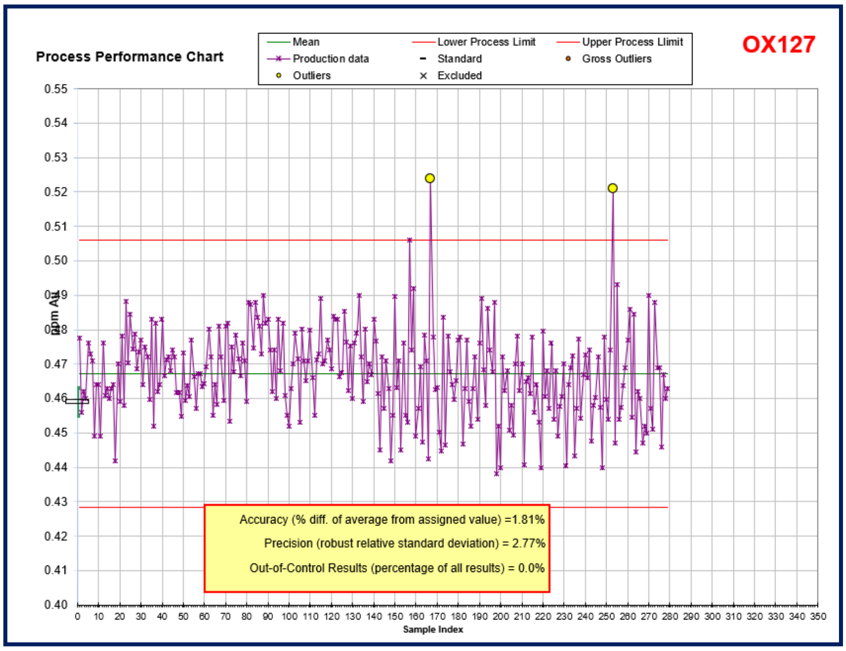
279 analyses were completed on standard OX127. Table 12-3 shows the statistics for the standard. Figure 12-2 shows the performance of the standard.

TABLE 12-3 OX127

Description	All Results	Gross Outliers Excluded	User Outliers Excluded	Comments
Number of Results	279	279	279	
Average	0.4673	0.4673	0.4673	
Accuracy: (% Difference of Average from Assigned Value)	1.8%	1.8%	1.8%	
Precision: Relative Standard Deviation (Robust)	2.8%	2.8%	2.8%	Good
Number of Outlying results (Outside Process Limits)	0	0	0	
Percentage of Outlying Results			0.0%	Good

OX127	
Assigned Value of Standard	0.459
95% Confidence Limits Standard (+/-)	0.004

FIGURE 12-2 OX127 PERFORMANCE



Two failures were recorded for OX127, with overall excellent conformance.

12.2.1.3. OX139

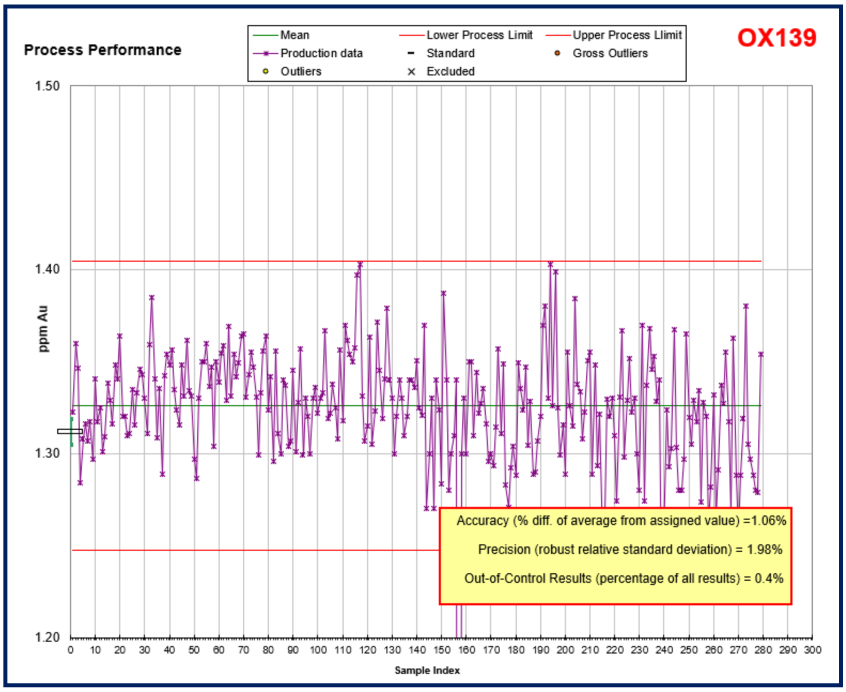
279 analyses were completed on standard OX139. Table 12-4 shows the statistics for the standard. Figure 12-3 shows the performance of the standard.

TABLE 12-4 OX139

Description	All Results	Gross Outliers Excluded	User Outliers Excluded	Comments
Number of Results	279	278	278	
Average	1.3220	1.3259	1.3259	
Accuracy: (% Difference of Average from Assigned Value)	0.8%	1.1%	1.1%	
Precision: Relative Standard Deviation (Robust)	2.5%	2.0%	2.0%	Good
Number of Outlying results (Outside Process Limits)	0	1	1	
Percentage of Outlying Results			0.4%	Good

OX139	
Assigned Value of Standard	1.312
95% Confidence Limits Standard (+/-)	0.007

Figure 12-3 OX139 Performance



Zero failures were recorded for OX139.

12.2.1.4. OX142

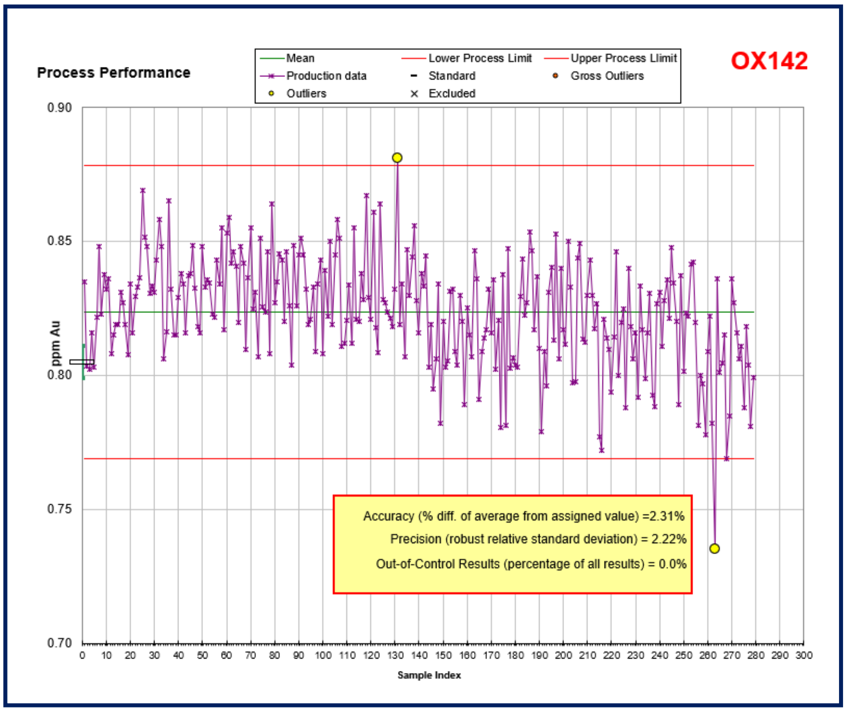
279 analyses were completed on standard OX142. Table 12-5 shows the statistics for the standard. Figure 12-4 shows the performance of the standard.

Table 12-5 OX142

Description	All Results	Gross Outliers Excluded	User Outliers Excluded	Comments
Number of Results	279	279	279	
Average	0.8236	0.8236	0.8236	
Accuracy: (% Difference of Average from Assigned Value)	2.3%	2.3%	2.3%	
Precision: Relative Standard Deviation (Robust)	2.2%	2.2%	2.2%	Good
Number of Outlying results (Outside Process Limits)	0	0	0	
Percentage of Outlying Results			0.0%	Good

OX142	
Assigned Value of Standard	0.805
95% Confidence Limits Standard (+/-)	0.006

FIGURE 12-4 OX142 PERFORMANCE



Two failures were recorded for OX142, with overall excellent conformance.

12.2.1.5. SUMMARY

The overall standard performance for the 2018 sample analysis was excellent, with four recorded failures from a total of 937 analyses. The standard performance is suitable to support a Mineral Resource estimate.

12.2.2. DUPLICATES

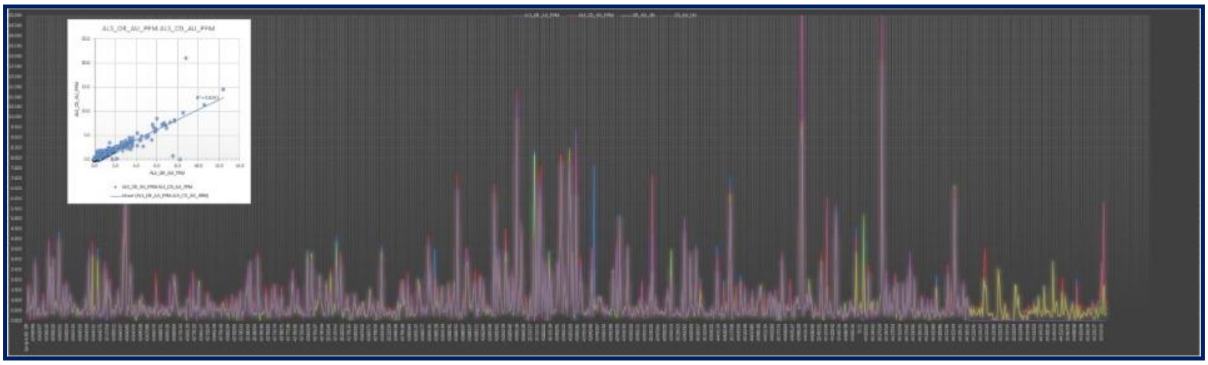
Coarse duplicates and pulp duplicates are analysed for performance to ensure that the analysis method is repeatable and accurate.

12.2.2.1. COARSE DUPLICATES

965 sample pairs were analysed from coarse sample. Coarse duplicate performance is good. There is natural variability of the grades within coarse duplicates due to factors such as the nugget effect and how the gold is distributed within a sample.

Figure 12-5 shows the coarse duplicate performance.

FIGURE 12-5 COARSE DUPLICATE PERFORMANCE

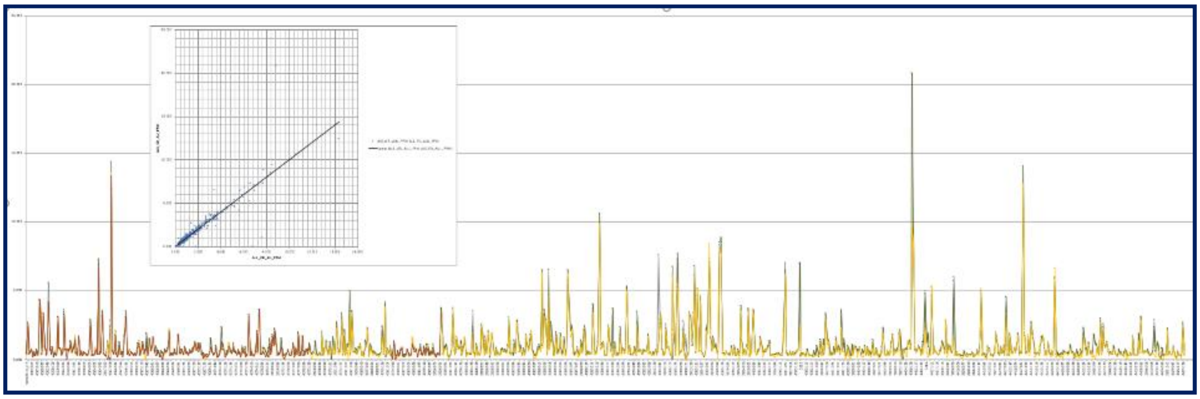


12.2.2.2. PULP DUPLICATES

886 sample pairs were analysed from pulps. Pulp duplicate performance is good. Assessment of homogenised pulp sample reduces the natural variability of the grades within coarse duplicates due to factors such as the nugget effect and how the gold is distributed within a sample.

Figure 12-6 shows the pulp duplicate performance.

FIGURE 12-6 PULP DUPLICATE PERFORMANCE

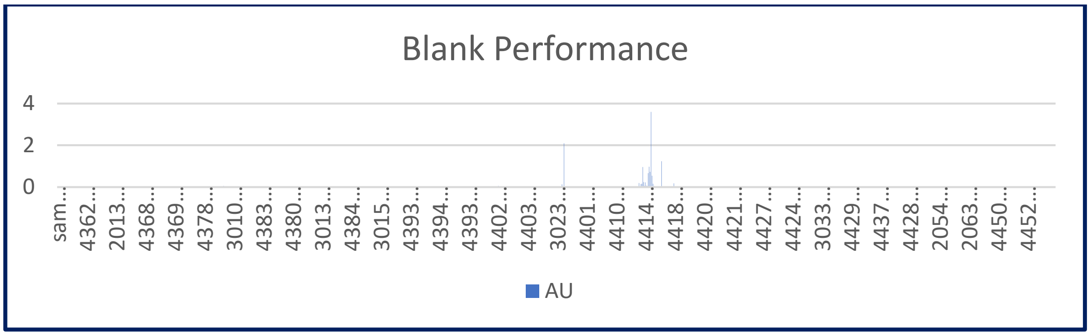


12.2.3. BLANKS

1053 blank samples have been analysed for the 2018 sampling data.

Blank performance has been excellent, with 21 fails in 1,053 analyses. (Figure 12-7)

FIGURE 12-7 BLANK PERFORMANCE



The fails may be attributed to contamination from previous high-grade samples. Procedures allow for thorough cleaning between analyses and where appropriate, materials are re-run on receipt of the results.

12.2.4. ROUND ROBIN

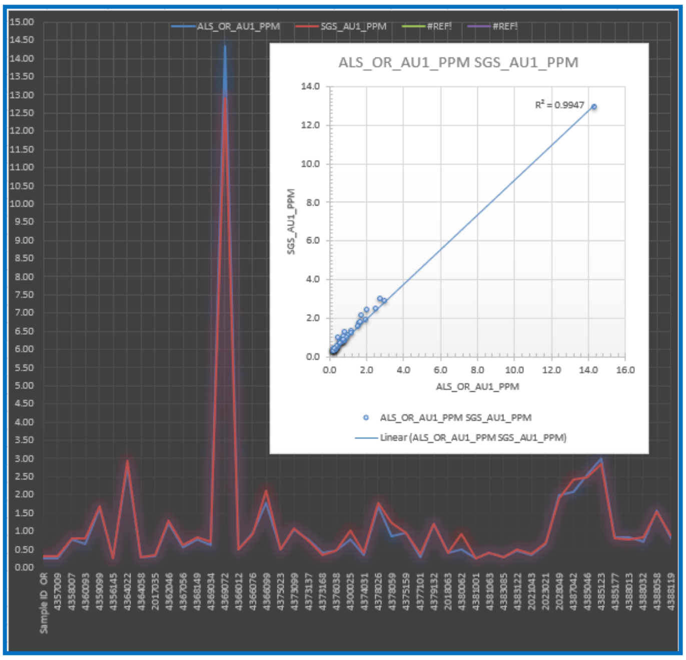
SGS and ALS-Stewart labs have been assessed against each other to test for general conformance to the analyses results.

Only 83 samples were tested which represent only 0.5% of the total number of ordinary samples. Table 12-6 shows the statistical assessment between ALS-Stewart and SGS and Figure 12-8 shows the conformance graphically.

TABLE 12-6 STATISTICAL CONFORMANCE

Basic Statistics	ALS_OR_AU1_PPM	SGS_AU1_PPM
Mean	1.19	1.22
Standard Error	0.31	0.28
Median	0.74	0.79
Mode	#N/A	0.31
Standard Deviation	2.10	1.90
Sample Variance	4.41	3.59
Kurtosis	35.94	33.47
Skewness	5.72	5.45
Range	14.10	12.63
Minimum	0.25	0.27
Maximum	14.35	12.90
Coef.Var	1.76	1.55
Sum	54.82	56.09
Count	46	46
Confidence Level (95.0%)	0.62	0.56
Quartile 1	0.40	0.41
Quartile 3	1.15	1.22
2Q Box	0.34	0.38
3Q Box	0.41	0.43
Усы -	0.15	0.14
Усы +	13.20	11.68

Figure 12-8 Graphical Conformance



12.2.5. Conclusion

The 2018 QA/QC programme continues to be excellent and is suitable to support a mineral Resource estimate.

12.3. 2019 Quality Assurance and Quality Control

A review of the QA/QC protocols employed for the 2019 drilling programme indicates a total of 24.9% for control testing, including duplicates, standards and blanks. These control samples are summarised in Table 12-7.

Table 12-7 Summary of 2019 Control Sampling

Parameters	Sample Count	Proportion of the Total (%)
Core Samples	19,072	
Coarse Duplicates	1,182	6.2
Pulp Duplicates	1,204	6.3
Blanks	1,196	6.3
Reference Materials	1,056	5.5
External Control	119	0.6

12.3.1. Standards

Standard Reference Materials are inserted into the sample stream to test to accuracy of the analyses against known values. The following Standards were used for the Tulkubash 2019 QA/QC programme.

Table 12-8 QA/QC Programme Standards, 2019

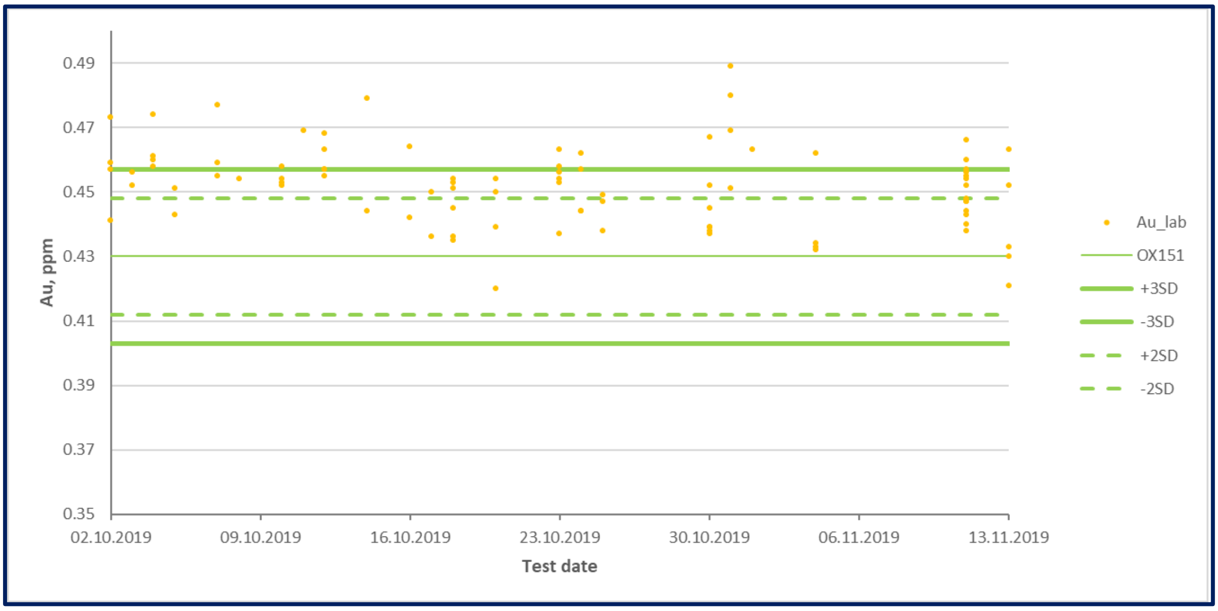
Reference Material	Grade Au (g/t)
Rocklabs OxD151	0.430
Rocklabs OxD127	0.459
Rocklabs OxF142	0.805
Rocklabs OxF162	0.832
Rocklabs OxH149	1.279
Rocklabs OxH139	1.312

The standards have been selected to reflect the typical mineralisation grades encountered at Tulkubash.

12.3.1.1. OxD151

90 analyses were completed on standard OxD151. These analyses indicated a pass rate of 71.1% with 26 samples failing (differed by greater than three standard deviations). Figure 12-9 shows the performance of the standard.

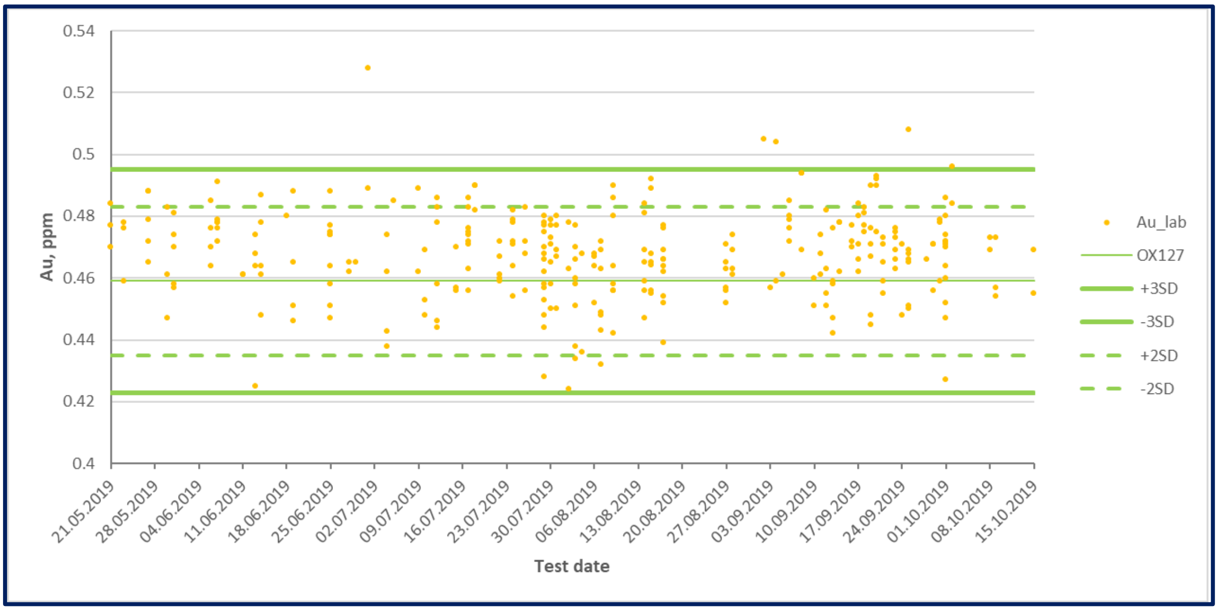
Figure 12-9 OxD151 Performance



12.3.1.2. OxD127

298 analyses were completed on standard OxD127. These analyses indicated a pass rate of 98.3% with five samples failing (differed by greater than three standard deviations). Figure 12-10 shows the performance of the standard.

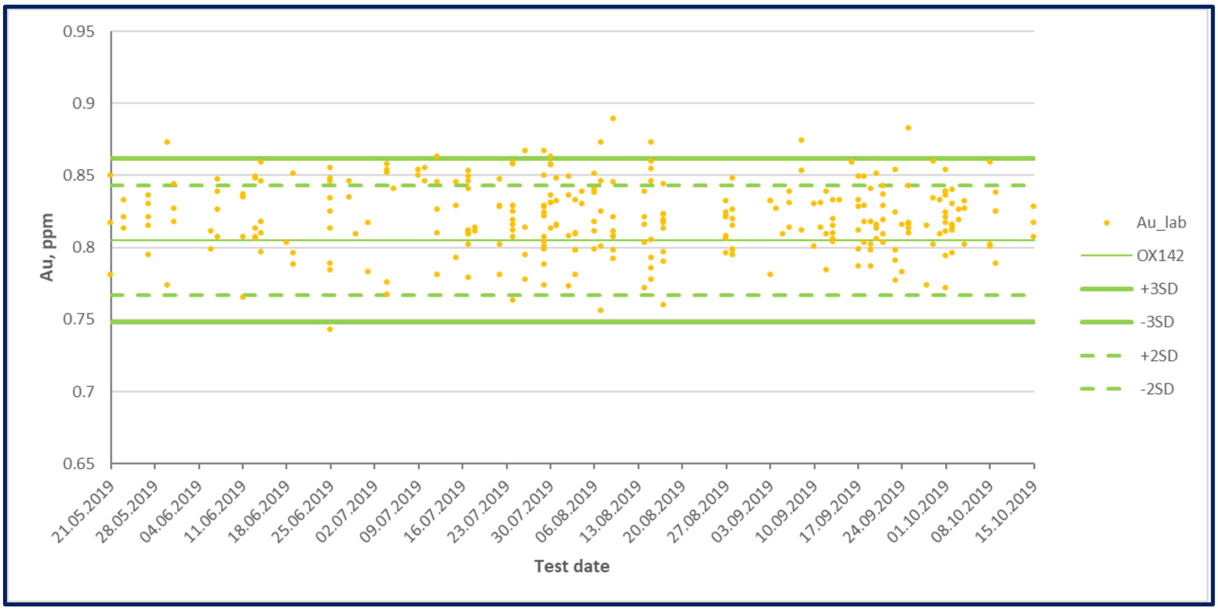
FIGURE 12-10 OXD127 PERFORMANCE



12.3.1.3. OxF142

296 analyses were completed on standard OxF142. These analyses indicated a pass rate of 96.3% with 11 samples failing (differed by greater than three standard deviations). Figure 12-11 shows the performance of the standard.

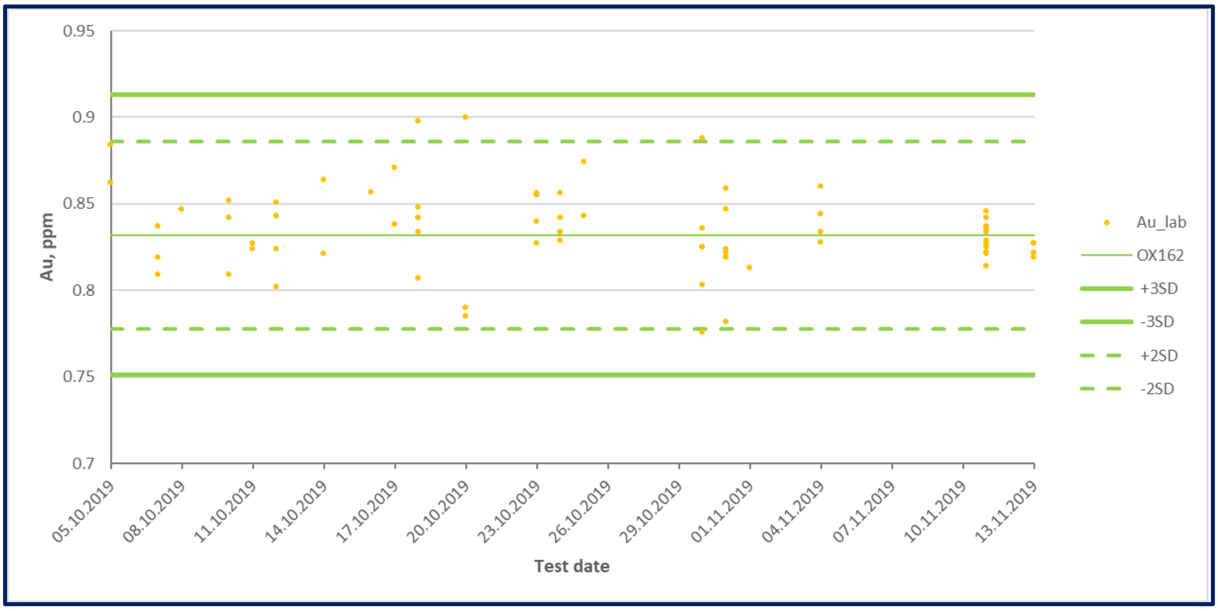
FIGURE 12-11 OXF142 PERFORMANCE



12.3.1.4. OxF162

73 analyses were completed on standard OxF162. These analyses indicated a pass rate of 100% with zero samples failing (differed by greater than three standard deviations). Figure 12-12 shows the performance of the standard.

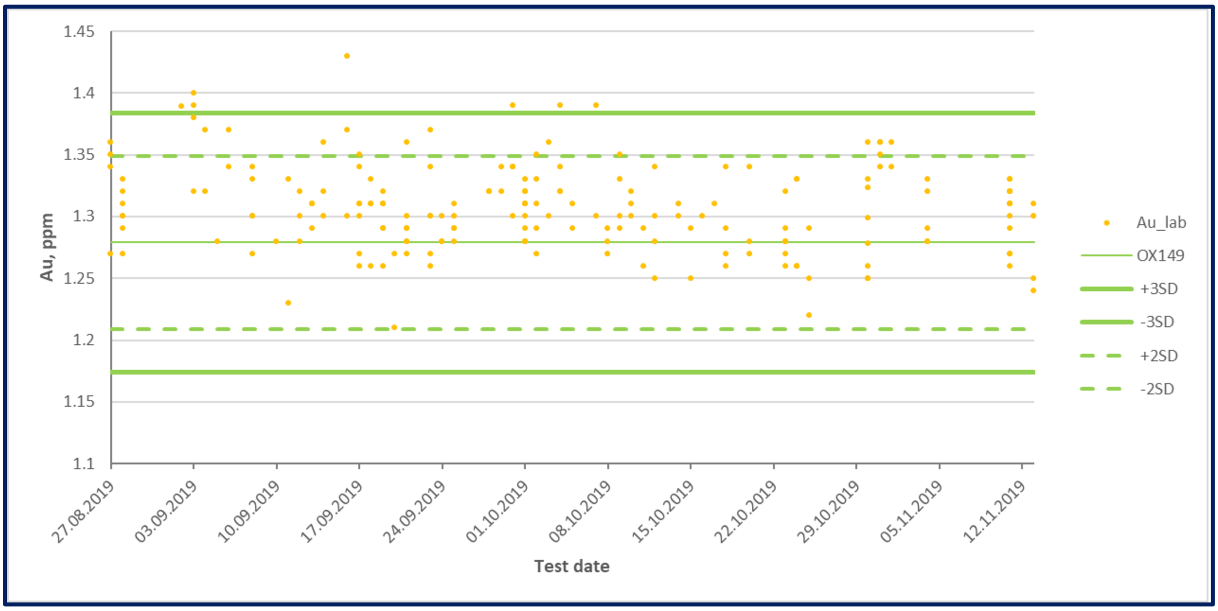
Figure 12-12 OxF162 Performance



12.3.1.5. OxH149

194 analyses were completed on standard OxH149. These analyses indicated a pass rate of 96.4% with seven samples failing (differed by greater than three standard deviations). Figure 12-13 shows the performance of the standard.

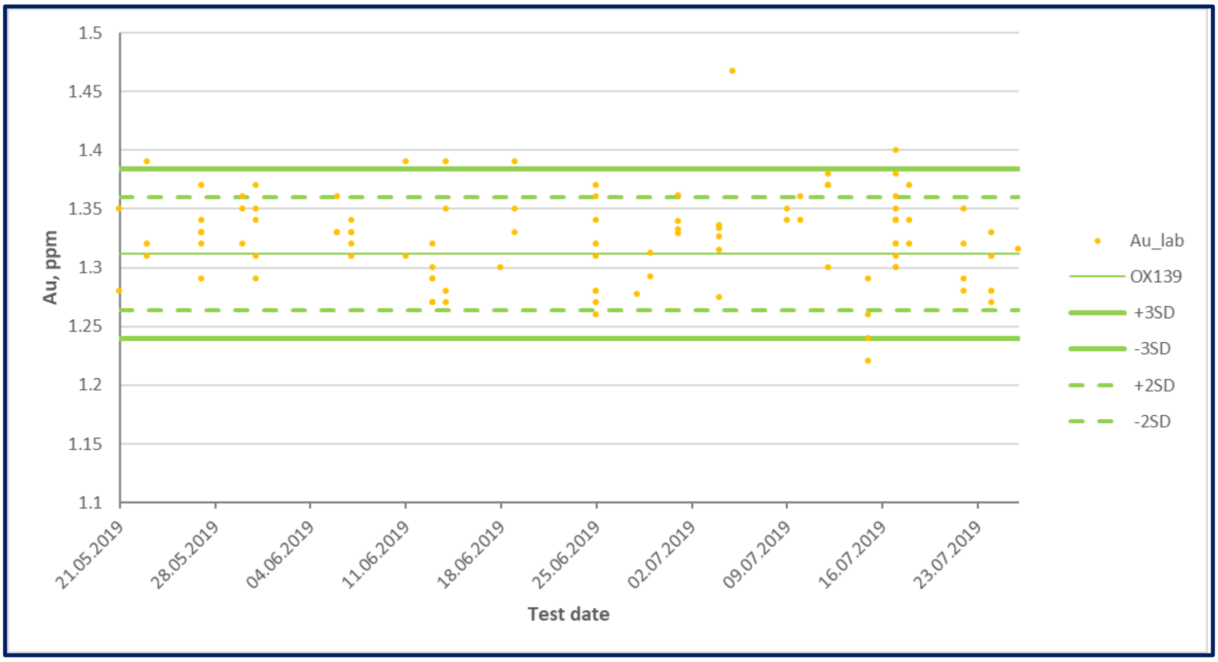
Figure 12-13 OxH149 Performance



12.3.1.6. OxH139

105 analyses were completed on standard OxH139. These analyses indicated a pass rate of 93.3% with seven samples failing (differed by greater than three standard deviations). Figure 12-14 shows the performance of the standard.

FIGURE 12-14 OXH139 PERFORMANCE



12.3.1.7. SUMMARY

The overall Standard performance for the 2019 sample analysis indicated a 5% failure rate, however if the lowest grade Standard, OxD151 which showed a 28.9% failure, is excluded, this decreases to a 3% failure rate. This may indicate a problem with the analyses of the lower grade samples. If one considers Figure 12-9 and Figure 12-13, sample grades appear to be biased high. Standard OxD151 grade is lower than Standard grades from the 2018 dataset.

12.3.2. DUPLICATES

Coarse duplicates and pulp duplicates are analysed for performance to ensure that the analysis method is repeatable and accurate.

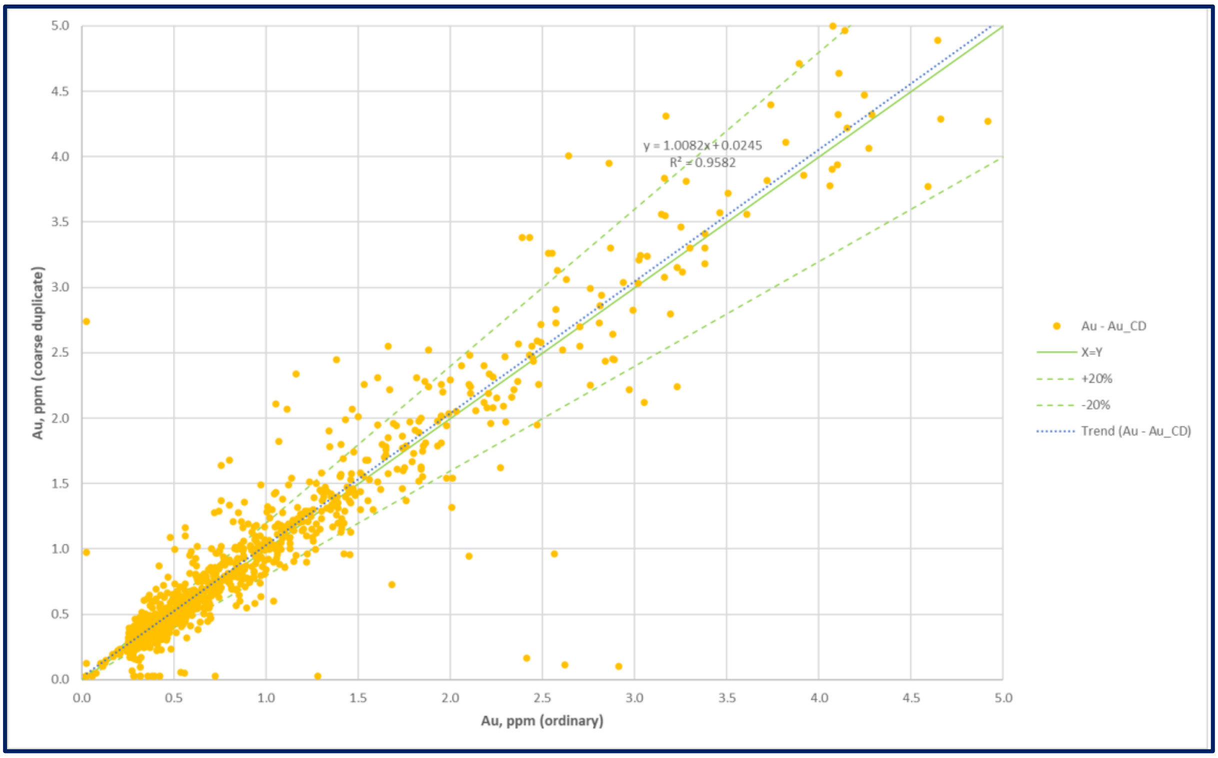
12.3.2.1. COARSE DUPLICATES

1,182 sample pairs were analysed from coarse sample, selected after the first stage of crushing to 2 mm. There is natural variability of the grades within coarse duplicates due to factors such as the nugget effect and how the gold is distributed within a sample.

23.0% of the pulp duplicates, 272 samples, show deviations greater than 20% (failures are considered as deviations >= 10%). Higher deviations are seen within the low-grade samples (< 1 g/t gold).

Figure 12-15 shows the coarse duplicate performance for samples 0-5 g/t.

FIGURE 12-15 COARSE DUPLICATE PERFORMANCE (0-5 G/T)



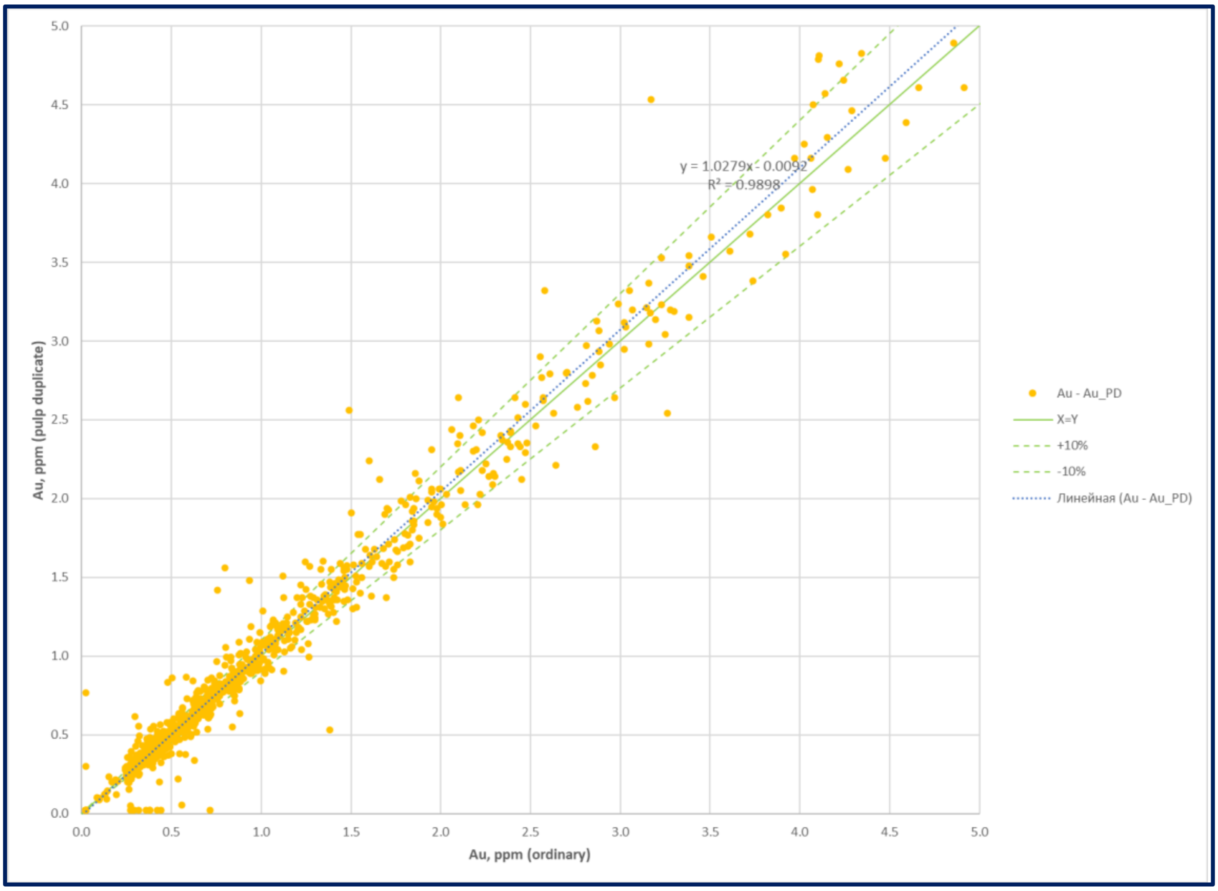
12.3.2.2. PULP DUPLICATES

1,204 sample pairs were analysed from pulps. Assessment of homogenised pulp sample reduces the natural variability of the grades within coarse duplicates due to factors such as the nugget effect and how the gold is distributed within a sample.

23.9% of the pulp duplicates, 288 samples, show deviations greater than 10% of which 207 (17% of pulp duplicates) are from low grade samples (< 1 g/t gold).

Figure 12-16 shows the pulp duplicate performance for sample grades 0 g/t to 5 g/t.

FIGURE 12-16 PULP DUPLICATE PERFORMANCE (0-5 G/T)

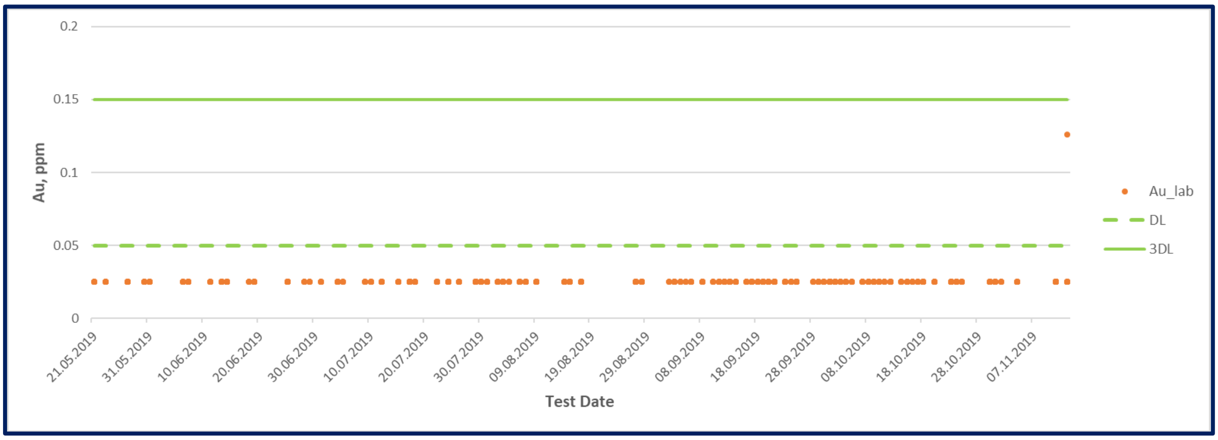


12.3.3. BLANKS

1,196 blank samples have been analysed for the 2019 sampling data.

Blank performance has been excellent, with only a single sample showing gold content above the gold sensitivity threshold. (Figure 12-17).

FIGURE 12-17 BLANK PERFORMANCE



The increased gold content within this single sample has been attributed to substandard sample material.

12.3.4. ROUND ROBIN

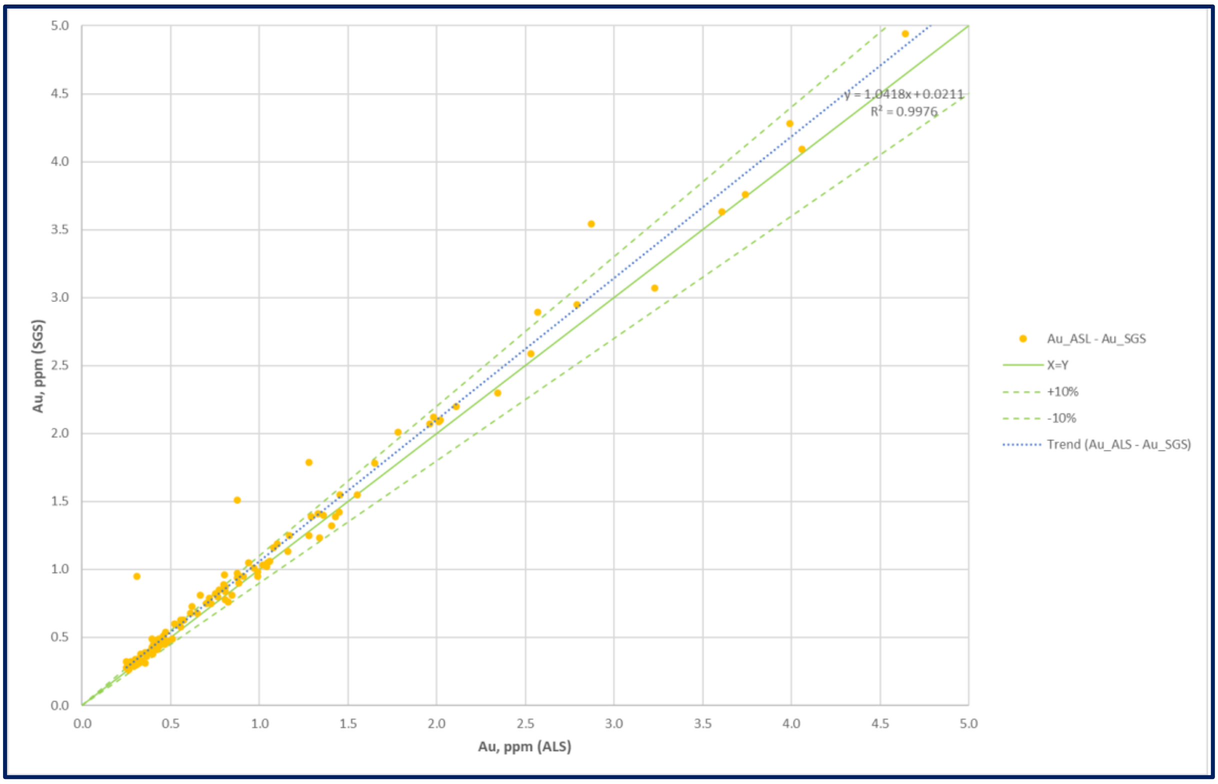
SGS and ALS-Stewart labs have been assessed against each other to test for general conformance of the 2019 analyses results. The trend line is considered within acceptable limits indicating no significant systematic deviation.

Table 12-9 shows the statistical assessment between ALS-Stewart and SGS and Figure 12-18 shows the conformance of samples 0 g/t to 5 g/t graphically.

TABLE 12-9 STATISTICAL CONFORMANCE

Basic statistics	ALS_AU1_PPM	SGS_AU1_PPM
Mean	1.770	1.865
Standard Error	0.331	0.345
Median	0.800	0.820
Mode	0.805	0.310
Standard Deviation	3.608	3.763
Sample Variance	13.017	14.161
Minimum	0.250	0.260
Maximum	29.500	30.700
25^{th} percentile	0.425	0.450
75^{th} percentile	1.406	1.420
Coefficient of Variation	2.038	2.018
Correlation Coefficient	0.999	
Coefficient of Determination	0.998	
AMPRD ≥ 20	5.0%	
AMPRD ≥ 10	26.1%	
AMPRD ≥ 5	56.3%	
Sum	210.6	222.0
Count	119	119

FIGURE 12-18 GRAPHICAL CONFORMANCE (0-5 G/T)



12.4. 2020 QUALITY ASSURANCE AND QUALITY CONTROL

Control samples employed for the 2020 Exploration programme are summarised in Table 12-10.

TABLE 12-10 SUMMARY OF 2020 CONTROL SAMPLING

Parameters	Sample Count	Proportion of the Total (%)
RC Samples	2,430	
Coarse Duplicates	163	6.7
Pulp Duplicates	151	6.2
Blanks	182	6.3
Reference Materials	147	6.0
External Control	-	-

12.4.1. STANDARDS

Standard Reference Materials were inserted into the sample stream ready for delivery directly to ALS-Stewart. A total of 147 Standard samples were assayed showing adequate laboratory results. The following Standards were used for the Tulkubash 2020 QA/QC programme:

Table 12-11 QA/QC Programme Standards, 2020

Reference Material	Grade Au (g/t)
Rocklabs OxD151	0.430
Rocklabs OxF162	0.832
Rocklabs OxH149	1.279

Figure 12-19 OxD151 Performance

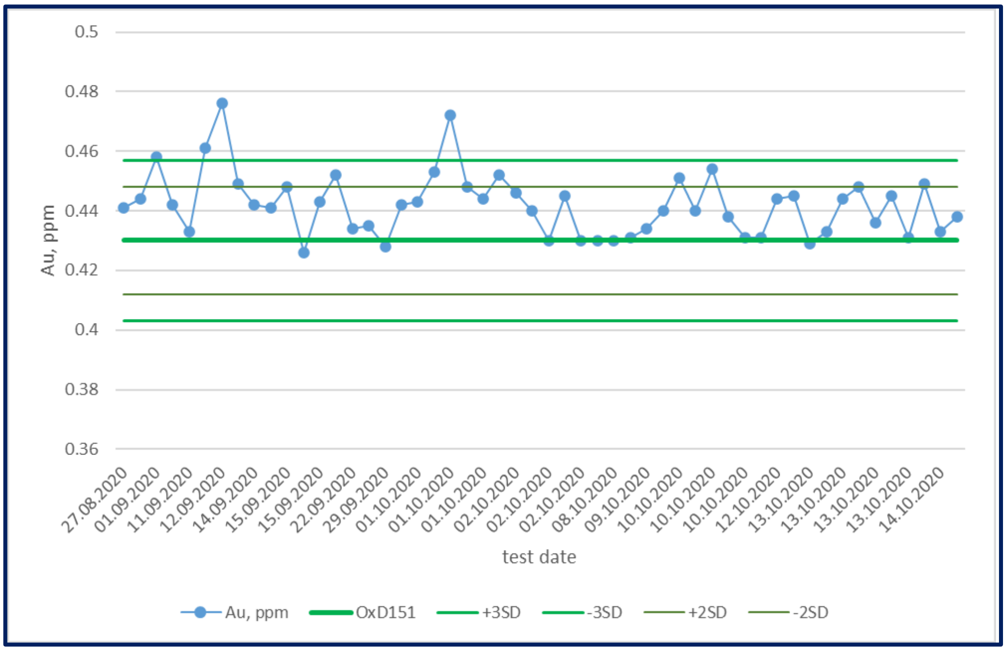


Figure 12-20 OxF162 Performance

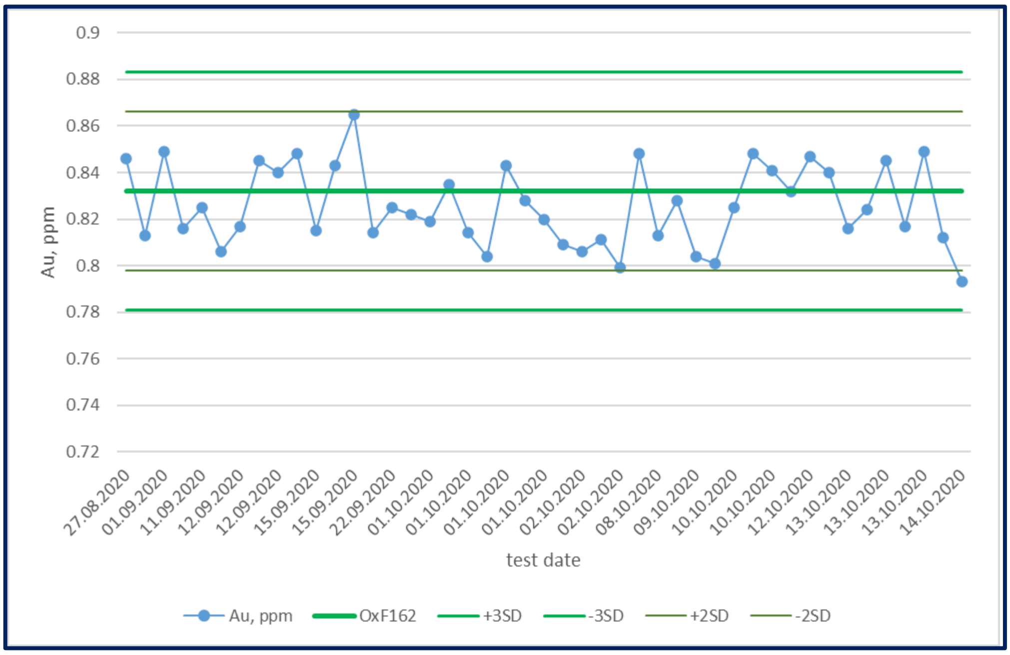
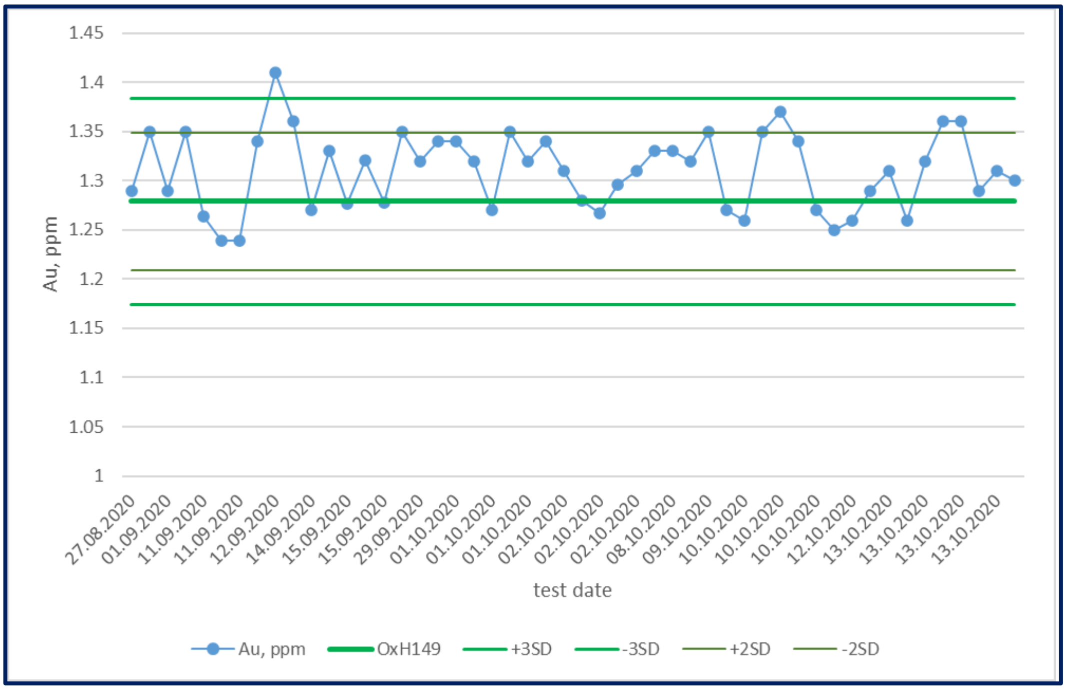


Figure 12-21 OxH149 Performance



12.4.2. Duplicates

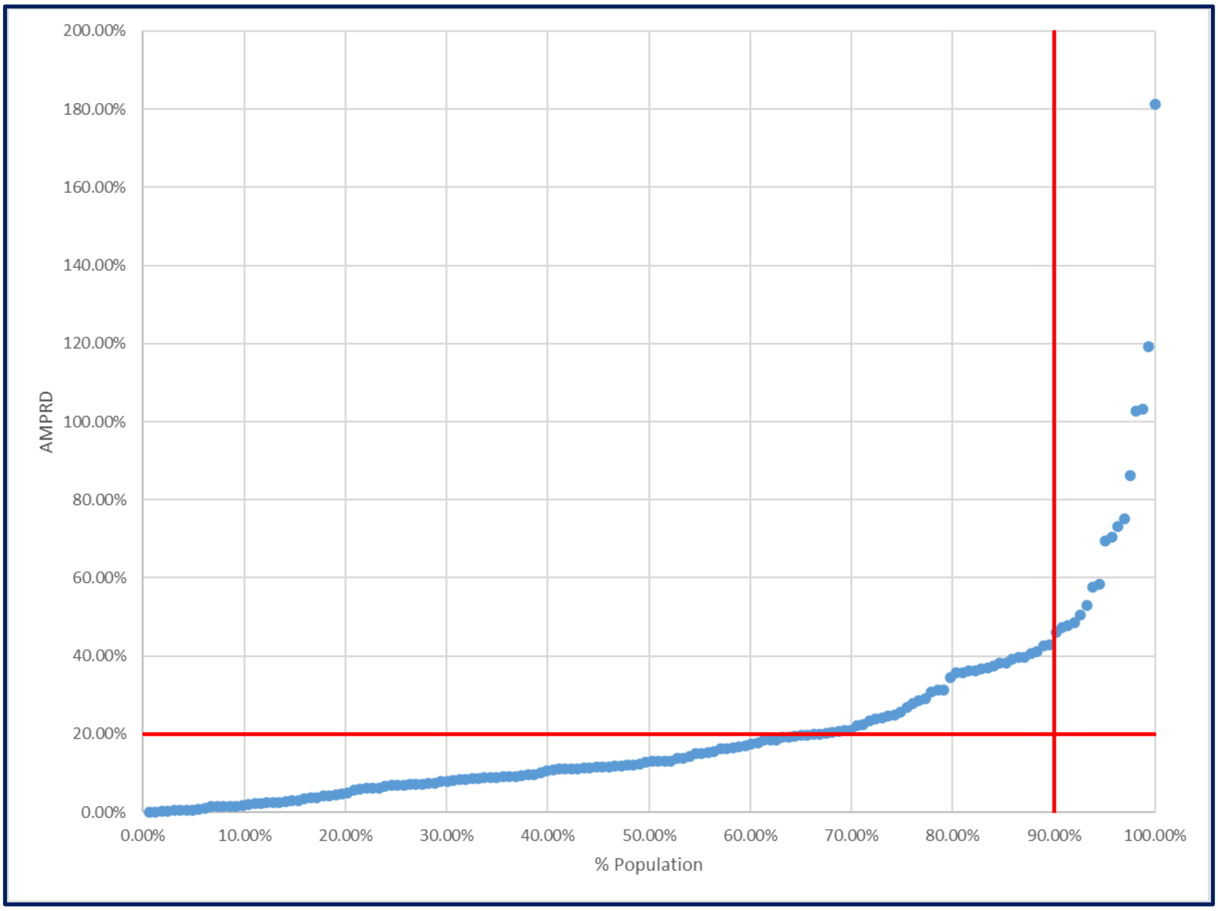
Coarse duplicates and pulp duplicates are analysed for performance to ensure that the analysis method is repeatable and accurate.

12.4.2.1. Coarse Duplicates

Analysis of 163 coarse duplicate sample pairs shows higher average grades in the coarse duplicate samples. Only 19% of pairs show differences less than 10%, with 27% of samples having differences below 20%.

Figure 12-22 shows the Absolute Mean Relative Deviation of Pairs (AMRDP) as a function of percent Population.

FIGURE 12-22 COARSE DUPLICATE PERFORMANCE

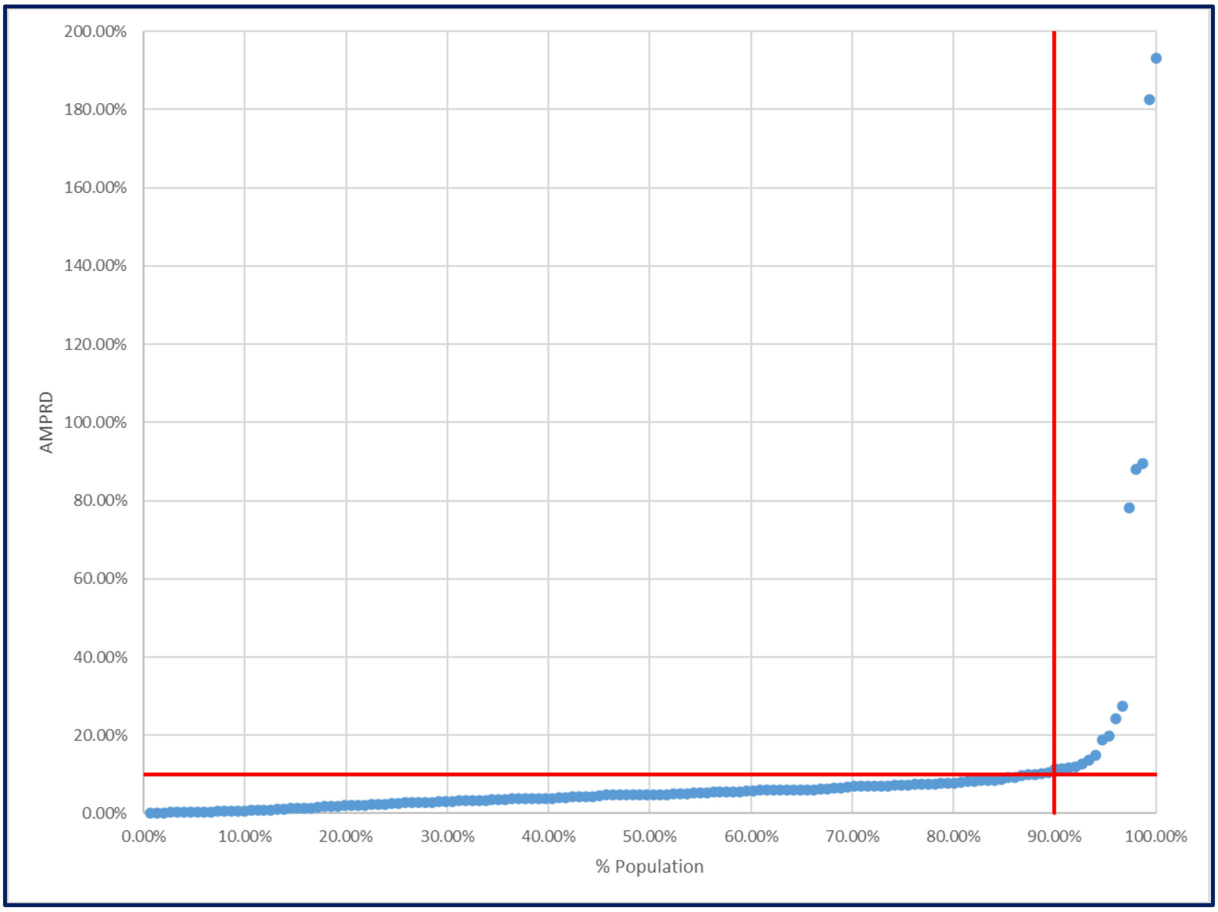


12.4.2.2. PULP DUPLICATES

Analysis of 151 pulp duplicate sample pairs shows significant difference between analyses of duplicate and ordinary samples, with deviations in individual samples being attributed to extremely uneven gold distribution in the host rock. Only 36% of pairs show differences less than 10%.

Figure 12-23 shows the AMRDP as a function of percent Population.

FIGURE 12-23 PULP DUPLICATE PERFORMANCE

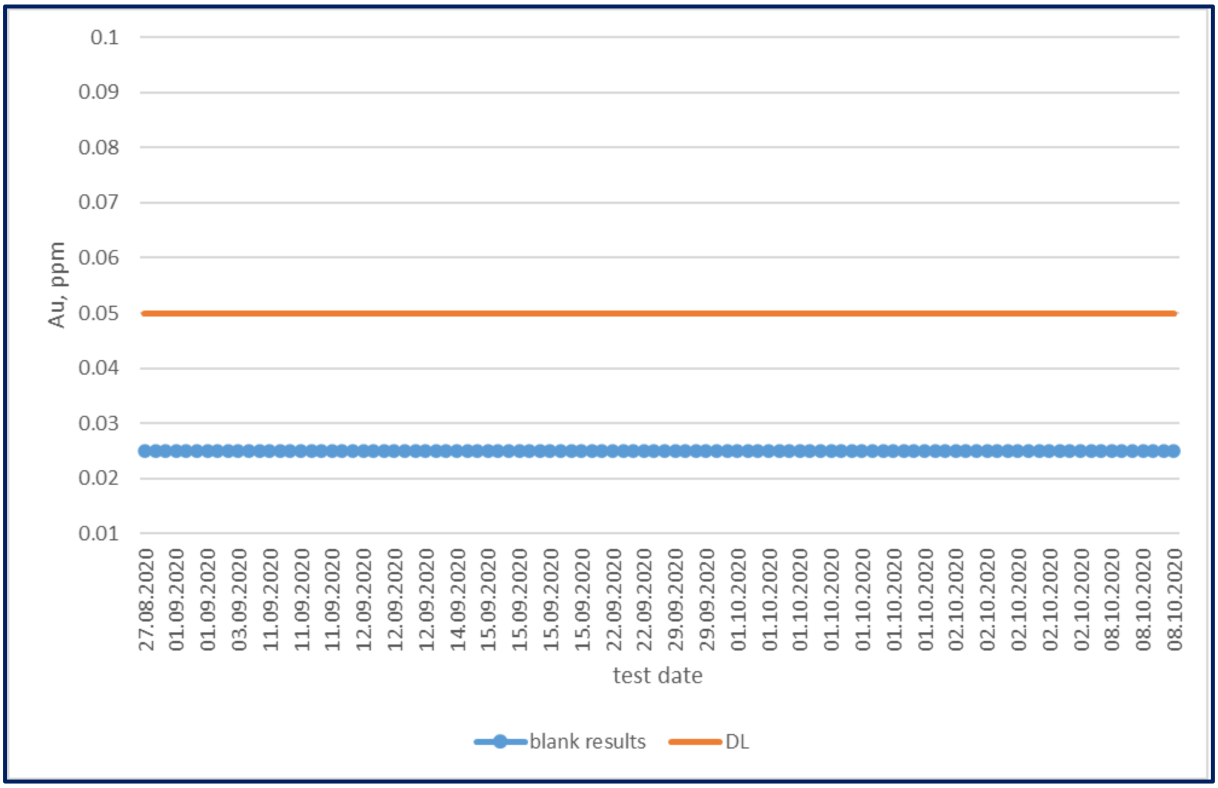


12.4.3. BLANKS

182 blank samples have been analysed for the 2020 sampling data.

No samples returned gold content above the gold sensitivity threshold. (Figure 12-24).

Figure 12-24 Blank Performance



13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1. SUMMARY

This section details the mineralogical and metallurgical testwork completed to date on the Project ore samples. A sample suitable for heap leach testwork was defined as any material within the Feasibility Study pit shell that had a total sulphur (S_{TOTAL}) content of 0.5% or less (S_{TOTAL} ≤0.5%) and above a nominal cut-off grade of 0.2 g/t gold. A detailed description of the metallurgical testwork for samples taken only in the areas to be mined is expanded on in detail. This is metallurgical testwork completed by WAI, UK (2017), MLI (2018), and ALS-Stewart (2019) on the applicable samples.

Conventional and block model recovery averaging methods of obtaining the Au and Ag recovery were investigated in this report. It was established that the block model recovery averaging method is more accurate than conventional recovery methods, and is therefore used for the final recovery. The block model uses the applicable inputs from the testwork completed.

Six organisations have conducted several historical metallurgical investigations:

- Resource Development Inc., USA (RDI) (2005) & (2007);
- Mintek Johannesburg, SA (MINTEK) (2009);
- Resource Development Inc., USA (RDI) (2010);
- SGS South Africa Pty. Ltd. (SGS-SA) (2011);
- Mintek Johannesburg, SA (MINTEK) (2011/2012);
- WAI, UK (2012);
- Beijing General Research Institute of Mining and Metallurgy, China (BGRIMM) (2013);
- Hazen Research Inc., USA (Hazen) (2013); and
- Resource Development Inc., USA (RDI) (2014).

A high-level summary of the above investigations is presented in Section 13.2.

As part of the Feasibility Study, three commercial laboratories completed additional metallurgical testwork: WAI, UK (2017), MLI (2018), Reno, Nevada, USA, and ALS-Stewart (2019).

WAI tested 23 variability composite samples collected from dedicated metallurgical drillholes within the zone of mineralisation, but these were not restricted to the proposed Feasibility Study pit. WAI also tested two master composites; the first master composite consisted of sub-samples from all variability samples, and the second master composite consisted of selected variability samples representing the heap leach ore within the Feasibility Study pit. WAI completed the testwork between October 2016 and March 2017.

MLI completed a separate testwork programme in 2018, which included a variability test programme consisting of 48 coarse ore bottle roll tests, followed by 11 column leach tests simulating heap leach conditions. MLI began the testwork in December 2017 with the results available in July 2018, which are included in this report.

ALS-Stewart (Stewart being SAEL, Stewart Analytical and Environmental Labroratories) has completed the testwork on 22 composites that come from the defined mining area (middle zone and Satellite zone). ALS-Stewart completed the testwork in 2019 and is included in this report.

LogiProc analysed all the metallurgical testwork results with the objective of identifying optimal heap leach conditions. The ALS-Stewart, WAI and MLI metallurgical studies indicate that the oxide ore is amenable to cyanide heap leaching and can be efficiently processed using a heap-leach based flowsheet.

The report describes the block model recovery as well as the less accurate conventional recovery estimate.

Based on the block model recovery averaging method, the expected LoM recovery for gold and silver is estimated to be 73.6% and 63.4%, respectively.

13.2. LoM Historical Testwork Review

Resource Development Inc., USA (RDI) (2005) & (2007) testwork was conducted on the Chaarat Zaav Kyzyltash deposit, and is therefore not relevant to this report.

Mintek Johannesburg, SA (MINTEK) (2009) testwork was an extension of the testwork completed by RDI in 2005 and 2007. Therefore, it is not relevant to this report.

Resource Development Inc., USA (RDI) (2010) had one relevant sample. This relevant sample was crushed too small for use in the current heap design, and is therefore not relevant to this report.

SGS South Africa Pty. Ltd. (SGS-SA) (2011) testwork was completed on samples outside of the present mining area. Therefore, it is not relevant to this report.

Mintek Johannesburg, SA (MINTEK) (2011/2012) is not relevant as the testwork was completed on a sulphide ore not suitable for heap leaching.

WAI, UK (2012) is not relevant as the testwork was completed on a sulphide ore that is not suitable for heap leaching.

Beijing General Research Institute of Mining and Metallurgy, China (BGRIMM) (2013) testwork was completed on samples outside of the mining pit.

Hazen Research Inc., USA (Hazen) (2013) is not relevant as the testwork was completed on a sulphide ore that is not suitable for heap leaching.

Resource Development Inc., USA (RDI) (2014) had two relevant samples – T-O and T-T. These samples were crushed too small in order to represent the current heap design, and are therefore not relevant to this report.

13.3. Detailed Testwork Review

The WAI (2017), MLI (2018), ALS-Stewart (2019) testwork were completed as part of the feasibility study and are used to determine the design criteria. Due to the progression and optimization of the pit design, some of the samples taken are no longer representative of the pit. This statement becomes more relevant as newer testwork is completed and fewer changes to the pit design occur.

13.3.1. WAI (2017) Testwork

13.3.1.1. Testwork Sample

LogiProc was not involved with the sample selection for the metallurgical testwork completed by WAI. Between October 2016 and March 2017, Chaarat submitted to WAI a total of 4,847 kg of sample material, collected from 12 metallurgical drillholes within the Tulkubash deposit mineralisation zone (Figure 13-1).

From Figure 13-2, Figure 13-3 and Figure 13-4, it can be seen that some samples were taken from outside the pit. The WAI testwork was done early on in the project and the mining pit has been optimized as the project has progressed, therefore only 52% of the samples in this report represent the current pit. These representative samples were selected for use in this report as described in the below sub-sections.

Figure 13-1 Metallurgical Drill holes

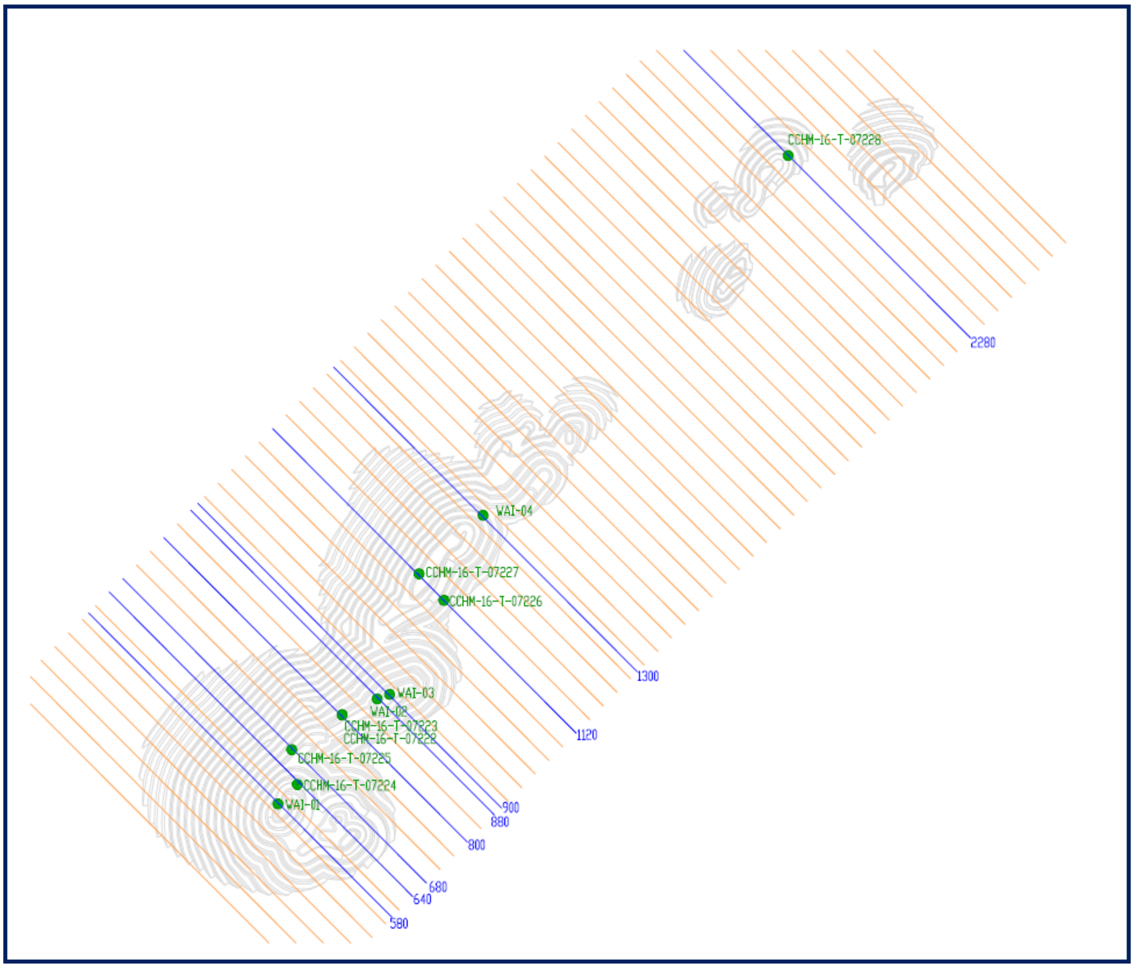


FIGURE 13-2 SAMPLES 16 AND 17 FROM DRILL HOLE CCHM16WAI01 LOCATED OUTSIDE THE PIT

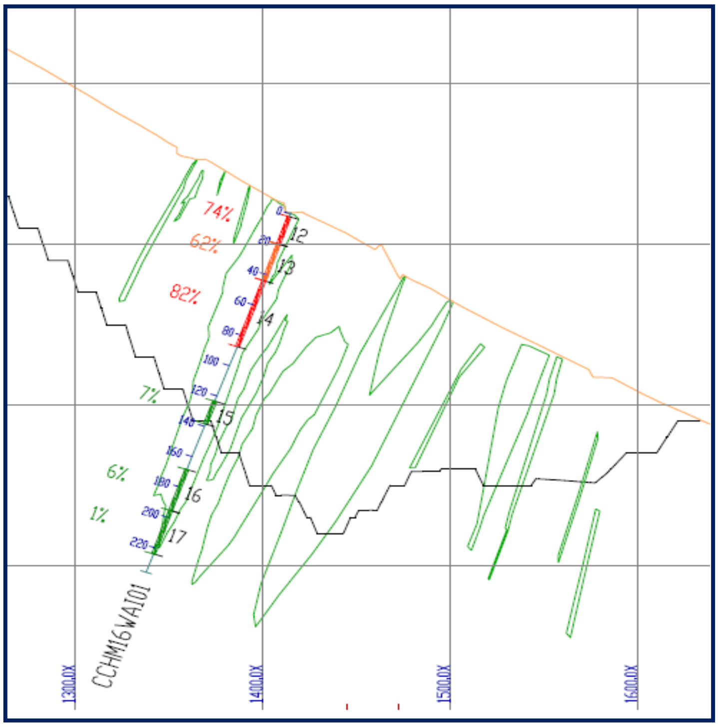


FIGURE 13-3 SAMPLES 21 AND 22 FROM DRILL HOLE CCHM16WAI03 LOCATED OUTSIDE THE PIT

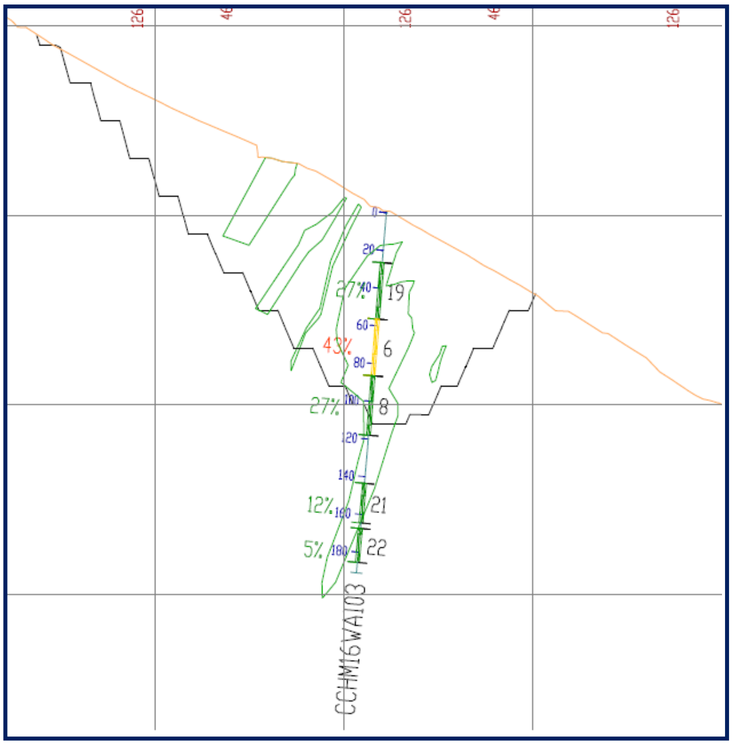
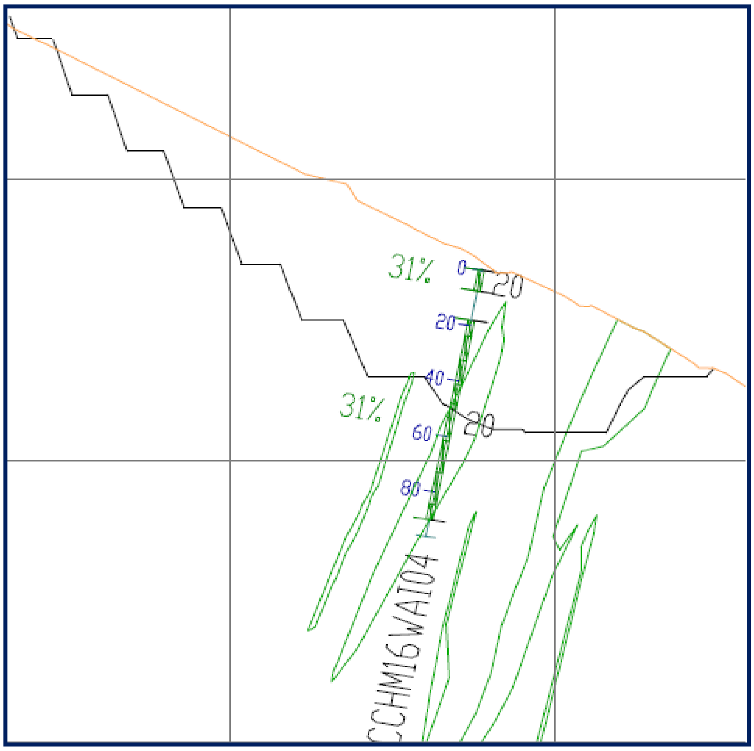


FIGURE 13-4 PORTION OF SAMPLE 20 FROM DRILL HOLE CCHM16WAI04 LOCATED OUTSIDE THE PIT



13.3.1.2. COMPOSITE METHODS

13.3.1.2.1. VARIABILITY COMPOSITES

Following receipt, WAI crushed the samples to 100% passing 25 mm and blended them to prepare 23 variability composites (Table 13-1).

TABLE 13-1 VARIABILITY COMPOSITE SAMPLE DETAILS

Composite ID	Bore Hole ID	Sample Interval (m)		Sample Mass (kg)	S_{TOTAL} (%)
		From	To		
1	CCHM16T07222	57.5	86.0	310.50	0.930
2	CCHM16T07223	25.5	39.0	148.46	0.970
3	CCHM16T07223	46.5	61.5	178.02	0.390
4	CCHM16T07224	13.5	60.0	494.30	0.320
5	CCHM16T07225	39.0	78.0	201.18	0.450
6	CCHM16WAI03	56.5	86.5	170.82	0.760
8	CCHM16WAI03	86.5	118.0	165.58	0.880
9	CCHM16T07228	0.0	18.0	79.90	0.083
10	CCHM16T07228	18.0	36.0	76.71	0.072
11	CCHM16T07228	36.0	55.5	97.50	0.045

Composite ID	Bore Hole ID	Sample Interval (m)		Sample Mass (kg)	S_{TOTAL} (%)
		From	To		
12	CCHM16WAI01	2.0	20.0	83.22	0.160
13	CCHM16WAI01	20.0	44.0	122.60	0.460
14	CCHM16WAI01	44.0	87.5	240.81	0.170
15	CCHM16WAI01	123.5	138.5	85.04	0.840
16	CCHM16WAI01	168.5	195.5	160.83	0.790
17	CCHM16WAI01	195.5	224.0	187.25	0.820
18	CCHM16WAI02	40.0	68.5	158.70	0.880
19	CCHM16WAI03	26.5	56.5	162.47	0.810
20	CCHM16WAI04	0.0	7.5	423.92	0.450
		18.0	90.0	-	
21	CCHM16WAI03	143.5	164.5	130.30	0.700
22	CCHM16WAI03	167.5	185.5	118.89	0.830
226	CCHM16T07226	46.5	84.0	193.58	0.380
227	CCHM16T07227	28.0	43.0	281.56	0.100
	CCHM16T07227A				

Table 13-2 shows all of the relevant samples that can be used to determine the process parameters. The composites that WAI did testwork on, were rejected, either, due to their sulphur content being higher than the intended cut-off value (S_{TOTAL} ≤0.5%) for heap leach processing or their drillhole location being outside of the mining pit.

Therefore, it was decided to exclude the results from variability composites 1, 2, 6, 8, 15, 16, 17, 18, 19, 20, 21, and 22 for the Feasibility Study as these composites are not representative of the planned heap leach feed. These composites are excluded to avoid unnecessary interference in the testwork results from material that is not relevant to proposed heap leach operation.

TABLE 13-2 RELEVANT VARIABILITY COMPOSITE SAMPLE DETAILS

Composite ID	Bore Hole ID	Sample Interval (m)		Sample Mass (kg)	S_{TOTAL} (%)
		From	To		
3	CCHM16T07223	46.5	61.5	178.02	0.390
4	CCHM16T07224	13.5	60.0	494.30	0.320
5	CCHM16T07225	39.0	78.0	201.18	0.450
9	CCHM16T07228	0.0	18.0	79.90	0.083
10	CCHM16T07228	18.0	36.0	76.71	0.072
11	CCHM16T07228	36.0	55.5	97.50	0.045
12	CCHM16WAI01	2.0	20.0	83.22	0.160
13	CCHM16WAI01	20.0	44.0	122.60	0.460
14	CCHM16WAI01	44.0	87.5	240.81	0.170

Composite ID	Bore Hole ID	Sample Interval (m)		Sample Mass (kg)	S_{TOTAL} (%)
		From	To		
226	CCHM16T07226	46.5	84.0	193.58	0.380
227	CCHM16T07227 CCHM16T07227A	28.0	43.0	281.56	0.100

13.3.1.2.2. MASTER COMPOSITES

WAI prepared two master composites (Table 13-3) by blending varying quantities of the variability composites (Table 13-1).

TABLE 13-3 MASTER BLENDED COMPOSITE COMPOSITIONS

Composite ID	Master Blended Composite (Mass%)	New Blended Master Composite (Mass%)
1	4.0	-
2	10.0	-
3	3.0	9.1
4	6.0	9.1
5	8.0	9.1
6	6.0	-
8	2.0	-
9	1.0	9.1
10	1.0	9.1
11	2.0	9.1
12	1.0	9.1
13	1.0	9.1
14	4.0	9.1
15	1.0	-
16	8.0	-
17	8.0	-
18	8.0	-
19	8.0	-
20	5.0	9.1
21	7.0	-
22	6.0	-
226	-	-
227	-	9.1

It is understood that the master composite was originally prepared by combining various mass fractions of all the available 21 composites at the time of testing, without the knowledge of the sample source or its relevance to the heap leach operation.

A new master composite was later prepared, with the understanding of the relevance of each composite in the heap leach operation and the effect of sulphide material on the heap leach. Two additional variable composites (226 and 227) were delivered to WAI before preparation of the new master composite and were also included in the new master composite.

The new master composite comprises composite 20 which is partially outside of the mining pit. However, the mass of this portion is negligible when compared to the total mass of the new composite. Therefore, the testwork on the new master composite is applicable.

The results from the new master composite were included for the Feasibility Study, but the leach extraction results from the master composite (originally prepared with all available 21 samples) were discounted for the same reasons namely, certain variability composites were excluded.

13.3.1.3. HEAD ASSAYS

Table 13-4, Table 13-5, and Table 13-6 show a summary of the WAI head assay analysis.

TABLE 13-4 VARIABILITY COMPOSITE HEAD ASSAYS

Variability Composites	Au (g/t)	As (%)	S_{TOTAL} (%)	$S_{SULPHIDE}$ (%)	C_{TOTAL} (%)
3	1.58	0.06	0.390	0.37	0.65
4	1.97	0.08	0.320	0.29	0.95
5	2.67	0.15	0.450	0.43	0.25
9	2.92	0.13	0.083	0.07	0.20
10	2.02	0.13	0.072	0.05	0.26
11	2.04	0.06	0.045	0.02	0.36
12	1.22	0.11	0.160	0.13	0.14
13	5.42	0.14	0.460	0.43	0.31
14	3.08	0.07	0.170	0.15	0.69
226	2.55	0.1	0.38	0.37	0.50
227	1.02	0.05	0.100	0.09	0.29

Note: $S_{SULPHIDE}$ – sulphur sulphide

TABLE 13-5 MASTER COMPOSITE HEAD ASSAYS

Au (g/t)	Ag (g/t)	As (%)	S_{TOTAL} (%)	$S_{SULPHIDE}$ (%)	C_{TOTAL} (%)
1.49	1.55	0.099	0.64	0.61	1.07

TABLE 13-6 NEW MASTER COMPOSITE HEAD ASSAYS

Au (g/t)	Ag (g/t)	As (%)	S_{TOTAL} (%)	$S_{SULPHIDE}$ (%)	C_{TOTAL} (%)	Hg (ppm)
2.03	0.5	0.092	0.22	0.2	0.44	0.573

This report notes that the total sulphur content of the master composite (0.64%) is higher than the cut-off value for heap leaching (S_{TOTAL} ≤0.5%). The higher sulphur content supports the view regarding the suitability of the master composite sample for use in the Feasibility Study.

13.3.1.4. Comminution Testwork

WAI completed the Bond crusher work index, Bond abrasion index, and Specific Gravity determinations (Table 13-7).

Table 13-7 Summary of Comminution Results

Bond Crusher Work Index kWh/t)	Bond Abrasion Index	Specific Gravity
10.2	0.4645	2.73

The results indicate that, for crushing purposes, the Tulkubash ore is moderately hard and moderately abrasive.

13.3.1.5. Optimisation of Heap Leach Parameters

WAI conducted coarse ore bottle roll leach tests to optimise the leach parameters. Table 13-8 summarises the test conditions.

Table 13-8 Coarse Ore Bottle Roll Leach Test Conditions

Parameter	Unit	Value
Sample Weight	kg	2
Cyanide Concentration	g/ℓ	2
pH	-	10.5-11.0
Pulp Density	% w/w	40
Leach Time	d	21

13.3.1.6. New Master Composite

Table 13-9 shows the results of the new master composite optimisation tests.

Table 13-9 New Master Composite Leach Optimisation Results

CrushSize (P100) (mm)	Cyanide Consumption (kg/t)	Lime Consumption (kg/t)	Extraction (Au%)
-25.0	0.94	0.16	70.8
-12.5	1.24	0.12	71.4

The results indicate that finer crushing increases the gold extraction by approximately 0.6% for the new master composite sample.

13.3.1.7. Variability Leach Tests (Coarse Ore Bottle Roll Leach Tests)

WAI conducted coarse ore bottle roll tests on the variability composites at a crush size of 12.5 mm. All other test conditions remained the same as shown in Table 13-8.

Table 13-10 summarises the results of the variability leach tests.

Table 13-10 Variability Leach Results Selected for Analysis

Composite ID	Reagent Consumption (kg/t)		Extraction (Au%)
	NaCN	Lime	
3	0.98	0.15	62.6
4	1.37	0.29	60.8
5	2.11	0.44	56.0
9	1.10	0.14	83.8
10	1.21	0.32	83.8
11	0.98	0.1	83.5
12	1.92	0.20	79.1
13	1.81	0.21	70.9
14	1.17	0.10	75.6

The selected variability leach results indicate that gold extraction ranged from 56.0% to 83.8%, with an average of 72.9%. The results also show that the average cyanide and lime consumptions were 1.4 kg/t and 0.2 kg/t, respectively.

13.3.1.8. Agglomeration and Percolation Tests

WAI completed a series of agglomeration and percolation tests, with the objective of determining the natural drainage characteristics of each of the samples. The target average drainage flowrate for percolation testing was 10,000 ℓ/m^2/h.

The effect of agglomeration on drainage flowrates through the addition of cement was also investigated for samples that demonstrated a flow rate below the desired target of 10,000 ℓ/m^2/h.

Table 13-11 shows a summary of the results for the natural (un-agglomerated) percolation testing of the master composite sample.

Table 13-11 Percolation Test Results for Master Composite Sample

P_{80} Crush Size (mm)	Average Drainage Flowrate (ℓ/m2/h)
25.0	33,760
12.5	13,763

P_{80} Crush Size (mm)	Average Drainage Flowrate (ℓ/m2/h)
6.3	2,523

The results show that the average drainage flowrate, at crush sizes of 25.0 mm and 12.5 mm are both above the target value of 10,000 ℓ/ m^2/h. However, the average drainage flow rate for the 6.3 mm crush size was below the target value.

Based on these results, WAI undertook further testing to investigate the effect of cement addition on drainage characteristics for the 6.3 mm crush size.

Table 13-12 shows the summary of the percolation tests with cement agglomeration.

Table 13-12 Summary of the Percolation Test Results with Agglomeration

Cement Addition (kg/t)	Average Drainage Flowrate (ℓ/ m^2/h)
2.5	2,545
5.0	7,455
7.5	14,915
10.0	14,748

The results indicate that a minimum of 7.5 kg/t of cement is required to achieve an average drainage flowrate in excess of the target value of 10,000 ℓ/m^2/h, and a further increase in the amount of cement added (to 10 kg/t) provided no further improvement in the average drainage rate of the master composite material at a crush size of (P_{100}) 6.3 mm.

The agglomeration requirement at the finer crush size of 6.3 mm supports the optimal crush size of 12.5 mm identified during the bottle roll leach tests.

In this context, WAI completed all the remaining testwork at a crush size of 12.5 mm.

Table 13-13 and Table 13-14 show the results of un-agglomerated percolation testing at a crush size of 12.5 mm for the new master composite, and the variability composites, respectively.

Table 13-13 Percolation Testing Results for the New Master Composite

Crush Size (mm)	Average Drainage Flowrate (ℓ/m2/h)
12.5	13,763

The results indicate the average drainage flow rate for the new master composite is above the target value of 10,000 ℓ/m^2/h.

TABLE 13-14 PERCOLATION TESTING RESULTS FOR VARIABILITY COMPOSITES

Composite ID	Drainage Flowrate (ℓ/ m^2/h)
3	14,645
4	19,170
5	9,457
9, 10, 11*	25,389
12, 13*	30,546
14	34,240
226	24,958
227	16,027

*Note: *combined samples*

The results of the percolation testing show drainage flowrates ranging from 9,457 ℓ/m^2/h for variability composite 5 to 34,240 ℓ/m^2/h for variability composite 14. The unweighted average drainage flowrate across the 8 variability samples tested was 21,804 ℓ/m^2/h.

Of the 8 samples tested only one sample, variability composite 5, shows a drainage flowrate slightly below the target level of 10,000 ℓ/m^2/h; however, the drainage flowrate achieved was comparable to the target flow rate of 10,000 ℓ/m^2/h. The decision was taken to proceed with further testing without agglomeration of the sample.

13.3.1.9. COLUMN LEACH TESTS

WAI conducted column leach tests on all the composites to optimise the heap leach operational parameters. Table 13-15 summarises the tests conditions.

TABLE 13-15 COLUMN LEACH TEST CONDITIONS

Parameter	Unit	Value
Sample Weight	kg	40-50
Column Diameter	m	0.15
Column Height	m	2
Retention Time	d	57-70
pH	-	10.5-11
Cyanide Concentration	g/ℓ	2
Irrigation Rate	ℓ/m^2/h	14
Water Type	-	tap water

The new master composite and variability composites were tested at a top size of 12.5 mm.

Table 13-16 and show the results of the column leach tests.

Table 13-16 New Master Composite Column Leach Test Results

Crush Size (P100) mm	Cyanide Consumption (kg/t)	Lime Consumption (kg/t)	Extraction (Au%)	Extraction (Ag%)
-12.5	1.24	0.03	72.2	48

Gold extraction from the new master composite column test (72.2%) is slightly higher than the coarse ore bottle roll test gold extraction (71.4%) on the same sample.

Table 13-17 summarises the results of the remaining samples.

Table 13-17 Column Leach Results of the Selected Variability Composites

Composite ID	**Reagent Consumption (kg/t)**		**Extraction (Au%)**
	NaCN	**Lime**	
3	1.65	0.16	65.9
4	1.78	0.07	63.9
5	2.56	0.06	56.2
9, 10, 11*	1.60	0.02	86.2
12, 13*	1.87	0.04	75.7
14	1.36	0.05	70.5
226	1.21	0.02	33.2
227	1.62	0.01	74.6

*Note: *combined samples*

The selected variability leach results indicate that gold extraction ranged from 33.2% to 86.2% with an average of 65.8%. The results also show that the average cyanide and lime consumptions were 1.7 kg/t and 0.05 kg/t, respectively. Since no bottle roll tests for composite 226 and 227 were completed, the column tests are excluded in the extraction adjustment factor calculation. Due to the observed extraction recovery of the column leach test for composite 226, it can be deduced that the sample was refractory and therefore not representative of the feed, and was not used in the sample set

$$Extraction\ Adjustment\ Factor = \frac{Column\ Extraction}{Bottle\ Roll\ Extraction}$$

A summary of the extraction adjustment factors and recoveries for the bottle roll tests can be seen below in Table 13-18 for Au. No Ag data was recorded for the individual composite tests; therefore, no extraction adjustment factor for Ag is calculated.

Table 13-18 Au Extraction Adjustment Factor Results

Composite ID	Equivalent Bottle Roll Extraction (Au%)	Column Extraction (Au%) Overall (57-70d)	Extraction adjustment factor
3	62.6	65.9	1.05
4	60.8	63.9	1.05
5	56.0	56.2	1.00
9, 10, 11*	83.7	86.2	1.03
12, 13*	75.0	75.7	1.01
14	75.6	70.5	0.93
New Master Composite	71.4	72.2	1.01
Average	64.54	64.45	0.99

Note an average bottle roll extraction for composite 12 and 13 as well as 9,10 and 11 is used.

This could be interpreted as the overall gold extractions during the heap leach operation could be approximately 1% higher than the bottle roll extractions reported. Therefore, the expected recovery is 70.1%.

13.3.1.10. Carbon Adsorption Testwork

WAI undertook carbon loading capacity testing, to determine the amount of gold and silver that can be loaded onto samples of activated carbon at varying carbon concentrations.

Table 13-19 summarises the results of the carbon loading tests.

Table 13-19 Carbon Loading Test Results

Solution Concentration (Au mg/ℓ)	Equilibrium Carbon Loading (Au g/t)	Solution Concentration (Ag mg/ℓ)	Equilibrium Carbon Loading (Ag g/t)
1.0	2,540	0.2	234
3.0	3,668	1.0	246
5.0	4,352	2.0	252

The results show that, at equilibrium, the amount of gold that can be loaded onto carbon ranges from 2,540 to 4,352 g/t at solution concentrations ranging from 1.0 to 5.0 mg/ ℓ.

Silver loadings ranged from 234 to 252 g/t at solution concentrations ranging from 0.2 to 2.0 mg/ ℓ.

13.3.1.11. Interpretation of WAI Testwork

It is understood that the WAI testwork samples were selected prior to gaining a good understanding of the proposed pit for the heap leach operation. The sample selection was primarily driven by gold grade, as opposed to a combined approach that would include gold grades along with the heap leach amenability of the ore zones and sulphur grades. Based on the sample selection and the Feasibility Study pit limits, it is concluded that approximately 48%

of the samples used in the WAI testwork were not representative of the heap-leach-amenable ore within the pit limits.

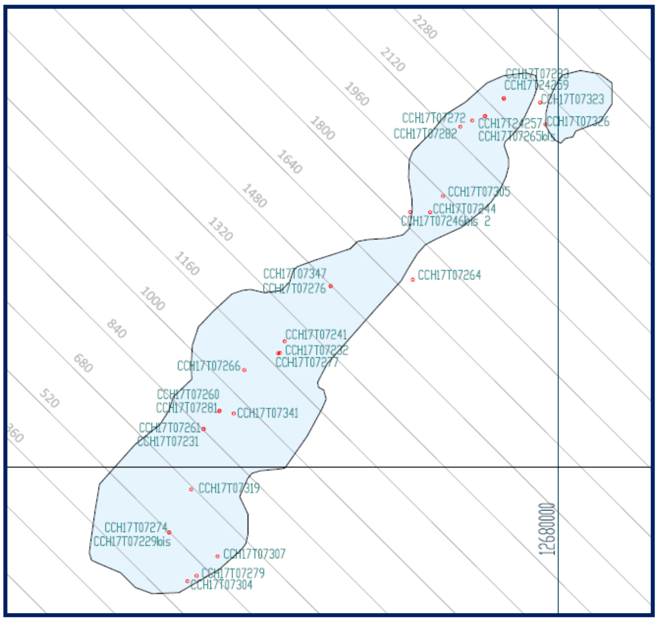
Therefore, it was decided to exclude those samples for the metallurgical recovery estimate and for the basis of forming the process design. The exclusion of the non-representative samples resulted in relying on only 52% of the samples which were representative of the proposed heap leach operation. All of the relevant samples were used in the process design along with the other relevant testwork from other laboratories.

13.3.2. MLI (2018) Testwork

13.3.2.1. Testwork Sample

A total of 619 drill core interval samples, weighing approximately 3,000 kg, and collected from 32 metallurgical drillholes (Figure 13-5) within the Tulkubash mineralisation, were submitted to MLI in five shipments between October 2017 and January 2018 for testing.

Figure 13-5 Metallurgical Drill holes (2017)



13.3.2.2. Compositing Methods

Each sample was weighed upon receipt at the laboratory. Selected drill core intervals were then combined in their entirety, according to compositing instructions provided by Chaarat, to produce 48 composites (designated composites 1 to 48) for bottle roll testing.

Each composite (ranging in weight from 17 to 131 kg) was stage crushed to 80% 9.5 mm in size, and each crushed composite was then thoroughly blended and split to obtain 2 kg for a bottle roll test and 1.0 kg for head analyses. The remainder of the sample was stored in the laboratory for the preparation of column leach composites.

13.3.2.2.1. Variability Composites

Table 13-20 shows a summary of the variability composites.

Table 13-20 Summary of the Variability Composites

Composite ID	Borehole ID	Sample Interval (m)		Sample Mass (kg)
		From	To	
1	CCH17T07229bis	98.5	121.0	95.6
2	CCH17T07229bis	121.0	142.0	91.6
3	CCH17T07261	55.5	78.0	87.9
4	CCH17T07261	91.5	103.5	47.4
5	CCH17T07232	27.0	70.5	50.8
6	CCH17T07241	70.5	90.0	43.5
7	CCH17T07264	96.0	114.0	72.5
8	CCH17T24257	68.5	88.0	52.9
9	CCH17T24257	88.0	103.5	58.0
10	CCH17T24257	104.5	115.0	39.4
11	CCH17T07244	94.0	105.0	27.9
12	CCH17T24259	45.5	60.5	51.9
13	CCH17T24259	69.5	80.0	37.1
14	CCH17T07245	12.5	29.0	58.2
15	CCH17T07245	29.0	64.0	69.4
16	CCH17T07245	64.0	96.0	48.2
17	CCH17T24254	96.0	117.0	36.1
18	CCH17T24254	117.0	139.2	37.9
19	CCH17T07263	31.5	42.0	31.6
20	CCH17T07260	26.0	63.5	76.7
21	CCH17T07260	63.5	83.0	79.9
22	CCH17T07229bis	7.0	16.0	38.3
23	CCH17T07231	45.0	54.0	34.9
24	CCH17T07260	107.0	115.0	33.7
25	CCH17T07279	82.0	121.0	130.7
26	CCH17T07276	33.0	57.0	75.5
27	CCH17T07276	175.5	184.5	26.6
28	CCH17T07281	22.5	55.5	107.6
29	CCH17T07282	57.0	70.5	30.1
30	CCH17T07283	44.0	56.0	16.9
31	CCH17T07277	58.5	75.0	44.6
32	CCH17T07274	106.5	120.0	47.7
33	CCH17T07265bis	75.5	89.0	39.3
34	CCH17T07266	117.0	126.0	37.1
35	CCH17T07272	40.5	54.0	47.5

Composite ID	Borehole ID	Sample Interval (m)		Sample Mass (kg)
		From	To	
36	CCH17T07246bis2	205.5	214.5	39.9
37	CCH17T07304	54.0	81.0	84.4
38	CCH17T07305	114.0	126.0	28.2
39	CCH17T07319	1.5	31.5	87.1
40	CCH17T07319	31.5	60.0	86.1
41	CCH17T07319	60.0	88.5	93.8
42	CCH17T07319	88.5	135.0	126.8
43	CCH17T07323	63.0	96.0	88.3
44	CCH17T07326	85.5	106.5	48.5
45	CCH17T07307	119.5	134.5	45.3
46	CCH17T07341	172.5	186.0	34.4
47	CCH17T07341	186.0	202.5	49.6
48	CCH17T07347	40.5	75.0	98.4

Composites 4, 7, 10, 11, 13, 14, 15, 16, 17, 18, 25, 34, 36, 37, 38, 43, 44, 45, 46, and 47 from Table 13-20 are outside the mine pit shell and their testwork was disregarded.

13.3.2.2.2. Column Composites

MLI prepared eight column leach composites by blending equal mass fractions of different variability composites. Table 13-21 shows a summary of the compositing criteria.

Table 13-21 Summary of Column Leach Composites

Column Composite	Variability Composite	Mass (%)	Composite Weight (kg)
A	1	25.0	50
	2	25.0	
	22	25.0	
	32	25.0	
B	3	14.3	50
	4	14.3	
	20	14.3	
	21	14.3	
	23	14.3	
	24	14.3	
	28	14.3	
C	5	20.0	50
	6	20.0	
	26	20.0	

Column Composite	Variability Composite	Mass (%)	Composite Weight (kg)
	27	20.0	
	31	20.0	
D	7	14.3	50
	11	14.3	
	14	14.3	
	15	14.3	
	16	14.3	
	19	14.3	
	36	14.3	
E	8	9.1	50
	9	9.1	
	10	9.1	
	12	9.1	
	13	9.1	
	17	9.1	
	18	9.1	
	29	9.1	
	30	9.1	
	33	9.1	
	35	9.1	
F	8	16.7	50
	12	16.7	
	13	16.7	
	29	16.7	
	30	16.7	
	35	16.7	
G	6	25.0	50
	7	25.0	
	14	25.0	
	15	25.0	
H	39	25	60
	40	25	
	41	25	
	42	25	

It was assumed that if any composite contained less than 60% of ore from the mined area it was deemed as a non-representative sample. From Table 13-21 it can be seen that composites A, B, C, E, F, and H are mainly composites of the mined areas.

13.3.2.3. Head Assays

MLI completed head assays of the variability composites. The gold and silver assays were completed using FA method in triplicate; the total sulphur analysis was completed using LECO analysis. Table 13-22 shows a summary of the head assays for the relevant samples.

Table 13-22 MLI Head Assays

Composite No	Head Grades			Composite No	Head Grades		
	Au (g/t)	Ag (g/t)	S_{TOTAL} (%)		Au (g/t)	Ag (g/t)	S_{TOTAL} (%)
1	1.49	0.40	0.29	26	1.02	0.40	0.12
2	1.28	0.60	0.43	27	2.19	0.50	0.40
3	0.68	0.70	0.29	28	0.56	0.30	0.13
5	0.60	0.30	0.09	29	1.32	0.70	0.09
6	0.46	0.50	0.11	30	0.70	0.20	0.03
8	0.56	1.30	0.05	31	0.66	0.20	0.07
9	1.15	0.60	0.21	32	3.59	1.10	0.42
12	1.44	1.40	0.04	33	1.75	0.70	0.12
19	1.14	0.30	0.05	35	1.00	1.00	0.05
20	0.91	0.80	0.09	39	0.92	3.30	0.04
21	0.77	0.30	0.09	40	1.99	1.10	0.49
22	1.50	2.10	0.07	41	1.26	0.30	0.22
23	0.52	1.30	0.23	42	1.88	1.90	0.32
24	1.35	0.30	0.06	48	1.18	0.70	0.09

13.3.2.4. Coarse Ore Bottle Roll Leach Tests

MLI has completed coarse ore bottle roll leach tests on all of the 48 variability composites. The test conditions are shown in Table 13-23.

Table 13-23 Coarse ore bottle roll test conditions

Parameter	Unit	Value
Crush Size	P_{80} mm	9.5
Sample weight	kg	2
Cyanide Concentration	g/ℓ	2
pH	-	10.8-11.2
Pulp Density	% w/w	40
Leach Time	d	16-18

The results of the variability composite leach tests are shown in Table 13-24.

Table 13-24 Variability Leach Test Results

Composite ID	Reagent Consumption (kg/t)		Extraction (%)	
	Lime	CN	Au	Ag
1	0.4	0.3	57.2	42.5
2	0.3	0.61	42.4	45.0
3	0.4	0.71	61.5	47.1
5	0.3	0.42	79.9	66.7
6	0.3	0.45	67.2	80.0
8	0.3	0.69	87.5	82.3
9	0.4	0.91	78.7	66.7
12	0.2	0.85	81.3	83.6
19	0.2	0.39	91.6	56.7
20	0.2	2.21	85.4	62.5
21	0.3	0.49	81.7	43.3
22	0.2	0.33	82.3	60.5
23	0.2	0.65	63.0	53.8
24	0.2	0.4	73.7	23.3
26	0.2	0.54	63.1	75.0
27	0.3	0.52	51.0	80.0
28	0.3	0.68	79.3	66.7
29	0.3	0.43	87.5	85.7
30	0.2	0.33	86.5	50.0
31	0.3	0.52	75.9	50.0
32	0.3	0.57	48.5	72.7
33	0.2	0.61	83.6	81.4
35	0.2	0.29	83.5	77.0
39	0.2	0.69	87.0	81.8
40	0.2	0.9	62.3	81.8
41	0.2	0.57	69.8	66.7
42	0.2	0.93	66.5	68.4
48	0.2	0.44	74.6	57.1

Note: cyanide - CN

It is noted that the composites that are outside the mining pit or have a higher than the intended cut-off value (S_{TOTAL} ≤0.5%) for heap leach processing (as stated previously) were excluded from the analysis.

The results of the remaining 28 variability samples indicate that the gold extraction ranged from
42.4% to 91.6%, with an average of 73.3%. The silver extraction ranged from 23.3% to 85.7%, with an average of 64.6%.

The results also show that the average cyanide and lime consumptions were 0.62 kg/t and 0.26 kg/t, respectively.

13.3.2.5. Column Leach Tests

MLI completed eleven column leach tests. These included composites A to H and composite 37, 38, and 48. However, only seven column leach tests are applicable, composites: A, B, C, E, F, H, 48. A summary of the tests conditions and results are shown in Table 13-25 and Table 13-26.

Table 13-25 Column Leach Conditions

Parameter	Unit	Value
Sample Weight	kg	20-50
Column Diameter	m	0.15
Column height	m	1.8
Leach Retention Time	d	60-80
pH	-	10-11
Cyanide Concentration	g/ℓ	1
Solution Application Rate	$t_{SOLUTION}$:t_{ORE}	2.5:2.8
Irrigation Rate	ℓ/m^2/h	12

Note: $t_{SOLUTION}$ – total solution; t_{ORE} – total ore

Table 13-26 Column Leach Test Results

Test Ref	Assay Head (g/t)		Leach/Rinse Cycle (d)	Extraction (%)		Reagent Consumption (kg/t)	
	Au	Ag		Au	Ag	CN	Lime
A	2.21	1.73	81	51.1	35.3	1.03	0.35
B	0.72	0.50	69	77.8	60.0	0.71	0.35
C	0.63	0.23	69	84.1	100	0.72	0.35
E	0.95	1.27	81	96.8	46.2	1.04	0.35
F	0.81	0.60	81	90.1	83.3	0.96	0.35
H	1.49	1.50	91	65.1	73.3	1.15	0.35
48	1.18	0.70	91	69.5	85.7	1.57	0.35

The column leach results for the applicable tests indicate that the gold extraction ranged from 51.1% to 96.8%, with an average of 76.4%. The silver extraction ranged from 35.3% to 100%, with an average of 69.1%.

The results also show that the average cyanide and lime consumptions were 1.03 kg/t and 0.35 kg/t, respectively.

13.3.2.6. Comparison of Bottle Roll and Column Leach Results

Table 13-27 and Table 13-28 shows a summary of the comparison between the bottle roll and column results for Au and Ag extraction for the applicable samples.

Table 13-27 Comparison of Bottle Roll and Column Leach Results for Gold

Test Ref	Equivalent Bottle Roll Extraction (Au%)	Column Extraction (Au%) Overall (69/70/91 d)
A	57.6	51.1
B	72.7	77.8
C	67.4	84.1
E	78.8	96.8
F	86.5	90.1
H	71.4	65.1
48	74.6	69.5

In terms of overall column gold extractions, the columns A, H, and 48 have worse Au extractions. Columns B, C, E, and F have indicated better column extractions compared to bottle roll extractions.

Table 13-28 Comparison of Bottle Roll and Column Leach Results for Silver

Column	Equivalent Bottle Roll Extraction (Ag%)	Column Extraction (Ag%) Overall (69/70/91d)
A	55.2	35.3
B	51.8	60
C	70.3	100
E	70.0	46.2
F	73.1	83.3
H	74.68	73.3
48	57.1	85.7

The silver extraction comparison data indicates that the overall column extractions are higher for Columns B, C, F, and 48. The bottle roll extractions are higher for Columns A, E, and H.

The comparison data shown in Table 13-27 and Table 13-28 indicates that there is no clear trend between the column and bottle roll work (some columns provided better extraction than the bottle rolls, but the others provided lower extractions than bottle rolls).

The extraction adjustment factors (Table 13-29) are calculated for each of the columns by expressing the overall column extraction as a fraction of the relevant bottle roll extraction. The extraction adjustment factor equation is:

$$Extraction\ Adjustment\ Factor = \frac{Column\ Extraction}{Bottle\ Roll\ Extraction}$$

Table 13-29 Extraction Adjustment Factors

Column	Extraction Adjustment Factor	
	Au	Ag
A	0.89	0.64
B	1.07	1.16
C	1.25	1.42
E	1.23	0.66
F	1.04	1.14
H	0.91	0.98
48	0.93	1.50
Average	1.05	1.07

Table 13-29 shows that gold extraction adjustment factors range from 0.89 to 1.25, with an average of 1.05. This could be interpreted as the overall gold extractions during the heap leach operation could be approximately 5% higher than the bottle roll extractions reported.

Table 13-29 also shows that silver extraction adjustment factors range from 0.64 to 1.5, with an average of 1.07. This could be interpreted as the overall silver extractions during the heap leach operation could be approximately 7% higher than the bottle roll extractions reported.

13.3.3. ALS-Stewart (2019) Testwork

13.3.3.1. Testwork Sample

The ALS-Stewart testwork comprised of 214 core samples from exploration drilling of the Mid and Satellite Zones, situated north east of the main zone. These core samples were split into 22 composites for testing.

FIGURE 13-6 OUTLINE OF USD1200 WHITTLE SHELL CENTERED ON THE NE PIT

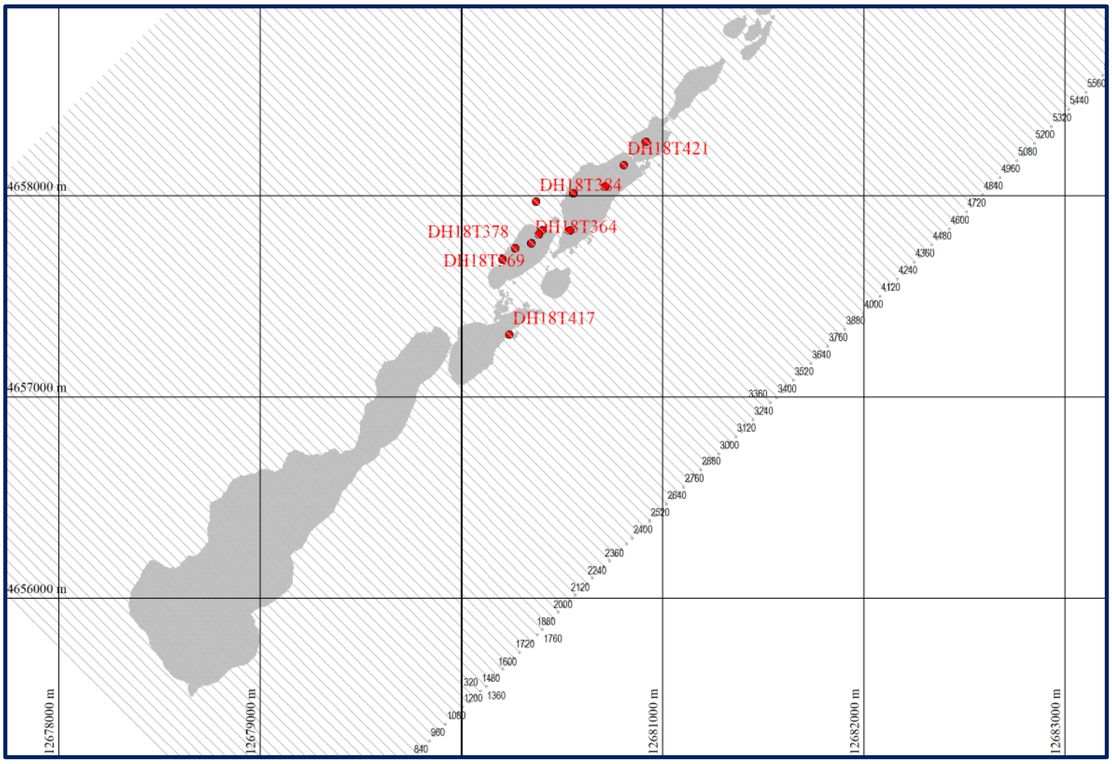


Figure 13-7 Metallurgical Drill Holes Supporting the Recovery Estimate

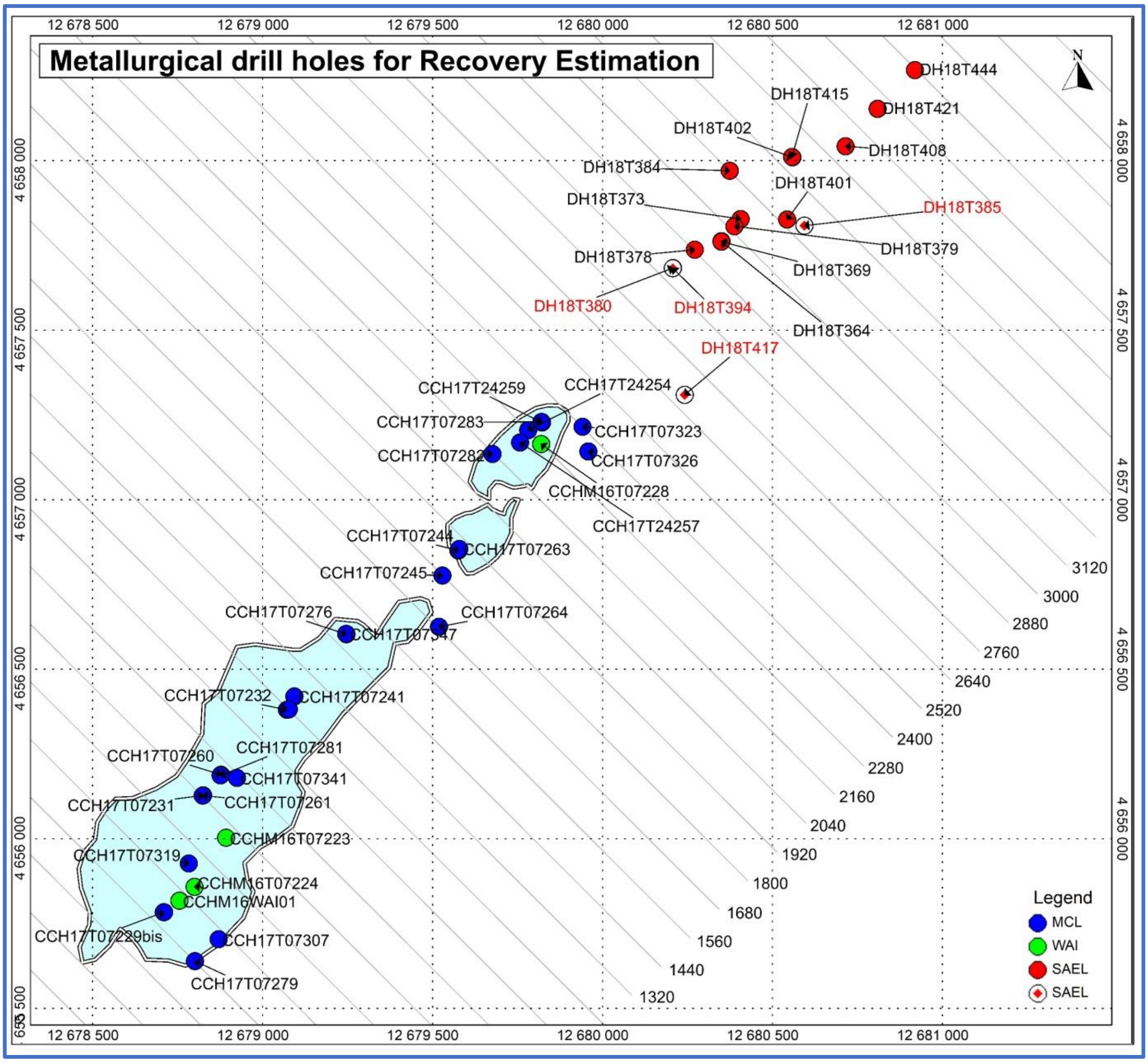
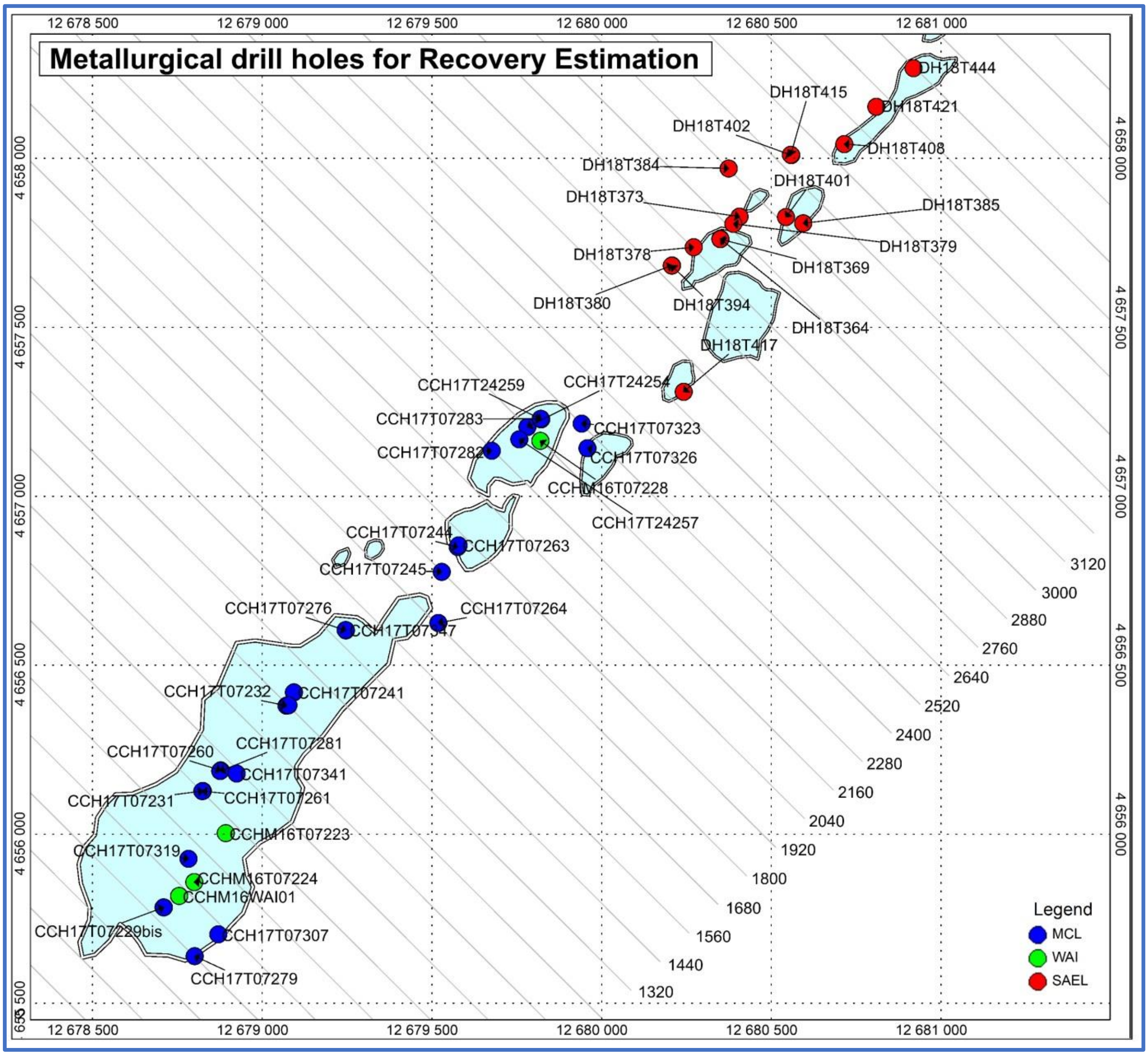


FIGURE 13-8 METALLURGICAL DRILL HOLES FOR RECOVERY IN RELATION TO THE PITS



22 composites from the drillholes, as seen in Figure 13-6, Figure 13-7, Figure 13-8, and Figure 13-9 were used as samples for the ALS-Stewart testwork.

13.3.3.2. COMPOSITING METHODS

The individual samples were received, in bags marked with the composite sample ID, by the laboratory. The samples were weighed before and after drying as it was determined that composites 10, 11 and 21 were wet on arrival. Selected drill core intervals were then combined in their entirety, according to compositing instructions provided by Chaarat, to produce 22 composites (designated composites 1 to 22).

Each composite (ranging in weight from 15 to 130 kg) was stage crushed to 80% 9.5 mm in size, and each crushed composite was then thoroughly blended and split to obtain 2 kg for a bottle roll test and 1.0 kg for head analyses. The remainder of the sample was stored in the laboratory storage facilities.

13.3.3.3. Variability Composites

Table 13-30 shows a summary of the variability composites and the compositions of the composite samples.

Table 13-30 Variablility, Weight and Composition of ALS-Stewart Samples

Composite sample	Wet weight	Dry weight	Bag ID	Drill Hole-ID
comp_1	70.98	69.67	1	DH18T415
			2	DH18T415
			3	DH18T415
			4	DH18T415
comp_2	100.80	98.84	5	DH18T402
			6	DH18T402
			7	DH18T402
				DH18T385
			8	DH18T385
			9	DH18T385
comp_3	27.23	27.20	10	DH18T401
comp_4	19.24	18.87	11	DH18T384
comp_5	27.79	27.19	12	DH18T379
comp_6	64.00	62.83	13	DH18T373
			14	DH18T373
			15	DH18T373
comp_7	79.59	78.08	16	DH18T373
			17	DH18T373
			18	DH18T373
			19	DH18T373
comp_8	33.43	32.47	20	DH18T369
			21	DH18T369
comp_9	33.42	32.77	22	DH18T369
			23	DH18T369
comp_10	15.59	13.82	24	DH18T364
comp_11	28.71	25.75	25	DH18T364
			26	DH18T364
comp_12	36.39	35.79	27	DH18T444
			28	DH18T444
comp_13	26.00	25.58	29	DH18T421
comp_14	26.54	26.14	30	DH18T408
comp_15	12.28	12.04	31	DH18T378
comp_16	28.15	27.54	32	DH18T378
			33	DH18T378

Composite sample	Wet weight	Dry weight	Bag ID	Drill Hole-ID
comp_17	9.43	9.13	34	DH18T380
comp_18	13.29	13.00	35	DH18T380
comp_19	18.93	18.50	36	DH18T394
comp_20	24.80	24.26	37	DH18T394
			38	DH18T394
comp_21	22.23	21.17	39	DH18T394
			40	DH18T394
			41	DH18T394
comp_22	13.94	13.57	42	DH18T417

The minor losses in the combined weights are assumed to be due to dust and sieving losses as well as rounding errors.

The following composites consist of or include drillholes taken outside of the pit and will be excluded from the results: Comp-2, Comp-17, Comp-18, Comp-19, Comp-20, Comp-21, and Comp-22. This corresponds to the red font indicating the drillholes that are outside the pit as seen in Figure 13-7 and Figure 13-8.

13.3.3.4. Head Assays

ALS-Stewart completed head assays of the variability composites. The gold assays were completed using FA method with atomic absorption finish in quadruplicate; separate silver and total sulphur analysis were completed while arsenic, antimony, and 33 other elements were analysed using multi-element ICP-OES analysis. Table 13-31 shows a summary of the head assays for the composite samples.

Table 13-31 ALS-Stewart Head Assay

##	Sample ID	Au (g/t)	Ag (g/t)	S_{TOTAL} (%)
1	Comp-1	0.722	2.2	0.54
2	Comp-2	4.94	1.2	0.52
3	Comp-3	0.668	<1.0	0.16
4	Comp-4	1.87	3.2	0.06
5	Comp-5	0.834	<1.0	0.11
6	Comp-6	0.620	<1.0	0.06
7	Comp-7	0.617	<1.0	0.44
8	Comp-8	0.66	<1.0	0.02
9	Comp-9	0.717	<1.0	0.02
10	Comp-10	1.92	<1.0	0.03
11	Comp-11	1.59	<1.0	0.03
12	Comp-12	1.43	<1.0	0.02
13	Comp-13	1.04	<1.0	0.10
14	Comp-14	1.76	3.7	0.45
15	Comp-15	1.15	<1.0	0.03

##	Sample ID	Au (g/t)	Ag (g/t)	S_{TOTAL} (%)
16	Comp-16	0.828	<1.0	0.02
17	Comp-17	0.489	<1.0	0.04
18	Comp-18	1.09	1.9	0.04
19	Comp-19	1.18	<1.0	0.05
20	Comp-20	1.01	<1.0	0.04
21	Comp-21	0.342	<1.0	0.07
22	Comp-22	1.04	1.3	0.03

Due to composite 1 and composite 2 having a sulphur content higher than the intended cut-off value ($S_{TOTAL} \leq 0.5\%$) for heap leach processing these composites are excluded from the results of this report. Therefore, the composites excluded from the ALS-Stewart results are: Comp-1, Comp-2, Comp-17, Comp-18, Comp-19, Comp-20, Comp-21, and Comp-22.

13.3.3.5. Coarse Ore Bottle Roll Leach Tests

ALS-Stewart has completed coarse ore bottle roll leach tests on all of the 22 variability composites. The test conditions are shown in Table 13-32.

Table 13-32 Coarse Ore Bottle Roll Test Conditions

Parameter	Unit	Value
Crush Size	P_{80} mm	9.5
Sample weight	kg	2
Cyanide Concentration	g/ℓ	2
pH	-	10.8-11.2
Pulp Density	% w/w	40
Maximum Allowable Leach Time	d	17.2

The results of the variability composite leach tests are shown in Table 13-33.

TABLE 13-33 RECOVERY AND REAGENT CONSUMPTION PER COMPOSITE

Sample ID	Reagent Consumption (kg/t)		Recovery of elements into leaching solution	
	Lime	CN	Au (%)	Ag (%)
Comp-1	1.97	1.82	71.0	N/A
Comp-2	1.79	3.28	65.5	N/A
Comp-3	1.48	1.90	71.3	N/A
Comp-4	1.51	2.74	89.7	N/A
Comp-5	1.32	2.04	76.8	N/A
Comp-6	1.31	2.39	79.8	N/A
Comp-7	1.52	2.64	56.7	N/A
Comp-8	1.69	2.56	93.6	N/A
Comp-9	1.26	2.61	86.7	N/A
Comp-10	1.43	2.68	97.9	N/A
Comp-11	1.76	2.73	92.3	N/A
Comp-12	1.63	2.37	88.0	N/A
Comp-13	1.69	3.13	94.5	N/A
Comp-14	1.93	2.00	77.0	N/A
Comp-15	1.52	2.12	91.9	N/A
Comp-16	1.37	1.72	91.8	N/A
Comp-17	1.34	1.50	86.1	N/A
Comp-18	1.46	2.89	94.9	N/A
Comp-19	1.26	1.66	81.4	N/A
Comp-20	1.49	2.17	92.2	N/A
Comp-21	1.23	1.42	80.8	N/A
Comp-22	1.36	1.48	85.1	N/A

Note: cyanide - CN

It is noted that the composites that are outside the mining pit or have a higher than the intended cut-off value (S_{TOTAL} ≤0.5%) for heap leach processing (as stated previously) were excluded from the analysis.

However, in the operating mine, BR testing for grade control which indicated that material like Comp-1 and 2 with above cutoff sulfur had recoveries of 60-70%, and would be treated as ore.

The results of the remaining 14 variability samples indicate that the gold extraction ranged from 56.7% to 97.9%, with an average of 84.9%. The silver extraction was not tested due to silver being a by-product.

The results also show that the average cyanide and lime consumptions were 2.40 kg/t and 1.53 kg/t, respectively.

13.4. Metallurgical Recoveries

Bottle roll leach gold and silver extractions were calculated based on the applicable variability leach testwork described in the above sections. Table 13-34 shows the summary of the applicable testwork estimated average bottle roll leach extractions.

Table 13-34 Bottle Roll Leach Extractions

Report and Composite	Au (%)	Ag (%)	Cyanide consumption (kg/t)	Lime consumption (kg/t)
Extraction WAI (2017) composite 3	62.6	N/A	0.98	0.15
Extraction WAI (2017) composite 4	60.8	N/A	1.37	0.29
Extraction WAI (2017) composite 5	56	N/A	2.11	0.44
Extraction WAI (2017) composite 9	83.8	N/A	1.1	0.14
Extraction WAI (2017) composite 10	83.8	N/A	1.21	0.32
Extraction WAI (2017) composite 11	83.5	N/A	0.98	0.1
Extraction WAI (2017) composite 12	79.1	N/A	1.92	0.2
Extraction WAI (2017) composite 13	70.9	N/A	1.81	0.21
Extraction WAI (2017) composite 14	75.6	N/A	1.17	0.1
Extraction WAI (2017) composite 20	31.2	N/A	1.55	0.05
Extraction WAI (2017) new master composite	71.4	N/A	1.24	0.12
Extraction MLI (2018) Composite 1	57.2	42.5	0.3	0.4
Extraction MLI (2018) Composite 2	42.4	45	0.61	0.3
Extraction MLI (2018) Composite 3	61.5	47.1	0.71	0.4
Extraction MLI (2018) Composite 5	79.9	66.7	0.42	0.3
Extraction MLI (2018) Composite 6	67.2	80	0.45	0.3
Extraction MLI (2018) Composite 8	87.5	82.3	0.69	0.3
Extraction MLI (2018) Composite 9	78.7	66.7	0.91	0.4
Extraction MLI (2018) Composite 12	81.3	83.6	0.85	0.2
Extraction MLI (2018) Composite 19	91.6	56.7	0.39	0.2
Extraction MLI (2018) Composite 20	85.4	62.5	2.21	0.2
Extraction MLI (2018) Composite 21	81.7	43.3	0.49	0.3
Extraction MLI (2018) Composite 22	82.3	60.5	0.33	0.2
Extraction MLI (2018) Composite 23	63	53.8	0.65	0.2
Extraction MLI (2018) Composite 24	73.7	23.3	0.4	0.2
Extraction MLI (2018) Composite 26	63.1	75	0.54	0.2
Extraction MLI (2018) Composite 27	51	80	0.52	0.3
Extraction MLI (2018) Composite 28	79.3	66.7	0.68	0.3
Extraction MLI (2018) Composite 29	87.5	85.7	0.43	0.3
Extraction MLI (2018) Composite 30	86.5	50	0.33	0.2
Extraction MLI (2018) Composite 31	75.9	50	0.52	0.3

Report and Composite	Au (%)	Ag (%)	Cyanide consumption (kg/t)	Lime consumption (kg/t)
Extraction MLI (2018) Composite 32	48.5	72.7	0.57	0.3
Extraction MLI (2018) Composite 33	83.6	81.4	0.61	0.2
Extraction MLI (2018) Composite 35	83.5	77	0.29	0.2
Extraction MLI (2018) Composite 39	87	81.8	0.69	0.2
Extraction MLI (2018) Composite 40	62.3	81.8	0.9	0.2
Extraction MLI (2018) Composite 41	69.8	66.7	0.57	0.2
Extraction MLI (2018) Composite 42	66.5	68.4	0.93	0.2
Extraction MLI (2018) Composite 48	74.6	57.1	0.44	0.2
Extraction ALS-Stewart (2019) Comp-3	71.3	N/A	1.9	1.48
Extraction ALS-Stewart (2019) Comp-4	89.7	N/A	2.74	1.51
Extraction ALS-Stewart (2019) Comp-5	76.8	N/A	2.04	1.32
Extraction ALS-Stewart (2019) Comp-6	79.8	N/A	2.39	1.31
Extraction ALS-Stewart (2019) Comp-7	56.7	N/A	2.64	1.52
Extraction ALS-Stewart (2019) Comp-8	93.6	N/A	2.56	1.69
Extraction ALS-Stewart (2019) Comp-9	86.7	N/A	2.61	1.26
Extraction ALS-Stewart (2019) Comp-10	97.9	N/A	2.68	1.43
Extraction ALS-Stewart (2019) Comp-11	92.3	N/A	2.73	1.76
Extraction ALS-Stewart (2019) Comp-12	88	N/A	2.37	1.63
Extraction ALS-Stewart (2019) Comp-13	94.5	N/A	3.13	1.69
Extraction ALS-Stewart (2019) Comp-14	77	N/A	2	1.93
Extraction ALS-Stewart (2019) Comp-15	91.9	N/A	2.12	1.52
Extraction ALS-Stewart (2019) Comp-16	91.8	N/A	1.72	1.37
Average	**75.46**	**64.58**	**1.25**	**0.58**

Two calculation methodologies were considered to determine the overall gold recovery for the life cycle of the mine.

1. **Block Model Recovery Averaging**. This methodology is based on the geological block model that is generated by the geological department to define the ore body. The theoretical recovery generated is based on the above metallurgical testwork, however it is inserted into the block model to provide a weighted average of the recoveries based on the grade in each special position in the mining pit. The recoveries in the block model are used to define the economic pit limits and provide the basis for recoveries in the process production plant.
2. **Conventional Recovery Averaging**. This is the traditional methodology which is used by the Metallurgical teams to determine the recovery of gold that should be expected through the Process Plant.

The samples that are tested are examined to determine suitability for inclusion into the sample average. For the samples to be considered, the samples must fall within the mining pit and must be seen to have the metallurgical characteristics that are at or above the nominal cut-off grade. In this case a mining cut-off of 0.5% S_{TOTAL} (Total Sulphur) and a nominal cut-off grade of 0.2 g/t gold will be implemented. The assumption is made that the mining plan will only allow for ore that abides with the above to be delivered to the plant for processing. A column extraction adjustment factor is calculated and used to interpret final recoveries.

13.4.1. Block Model Recovery Averaging

Utilizing the Block Model Recovery Averaging methodology (via a geo-metallurgical model), a model is developed in conjunction with the mineral Resource model to estimate the recoveries of gold and silver. This is used to inform the reserve model and financial model derived from the Resource. Recovery has been estimated per (selective mining unit) SMU scale block, based on the testwork data available, to reflect the variability in the potential to recover metal by heap leach process. The model is developed by first defining the oxide-sulfide boundary for the ore bodies and then applying inverse distance weighting square (IDW2) to extraction data in the oxide zone to generate block estimates. The resulting estimate compared favourably with the results of test work extraction.

Data from the bottle roll leach tests conducted by WAI (2017), MCL (2018), and ALS-Stewart (2019) was used for the recovery model. As seen in Section 13.3, the testwork comprises of 78 composite tests on samples collected from drill cores that are spatially evenly distributed throughout and around the main pit area. The composites cover a range of depths, strike locations, and oxidation states. Material in a given oxide class is geologically similar throughout the currently planned and future potential mining area. For this reason, the average recovery for each oxide class is based on the test results from all potentially minable samples, both inside and outside the pit..

The premise for the modelling of recovery on a block by block basis consisted of the following:

- Oxidation is the primary driver of leachability
- The degree of oxidation can be distinguished qualitatively, not quantitatively
- Oxidation intensity for samples is based on oxidation code and can be related to bottle roll extraction results
- Oxidation states in the deposit are mixed making traditional domaining impractical
- Sample data is representative of potential ore in the deposit

13.4.1.1. Sample Selection

Samples were selected for the various test programs based on the following criteria:

- Au grade near or above 0.2 g/t
- Degree of oxidation
- % Total Sulfur near or below 0.5%
- Leachability using hot cyanide shake test.

Gold grade was the initial criteria used to determine whether material could be potential leach feed. % Total Sulfur and degree of oxidation provided guidance on whether the material could be expected to leach. Cyanide solubility was used to confirm leachability.

The application of the sample selection criteria resulted in 95% of the test composites proving suitable as leach feed. These results indicated that observed oxidation and cyanide solubility provide a good guide to identifying heap leach feed.

13.4.1.2. Sample Characteristics

Of the 991 samples composited for the three phases of metallurgical testing, about 10% were not originally assigned oxidation states and another 5% were later found to be refractory and considered not representative of leach feed. Criteria were applied to assign oxidation states to the samples not originally classified based on percent extraction as shown in Table 13-35.

Table 13-35 Criteria Used to Classify Unassigned Samples

Oxidation State	Au Extraction %
0	0-39%
1	40-59%
2	60-79%
3	>79%

Table 13-36 shows the results on the entire data set of classifying unassigned samples. For the samples considered representative of heap leach feed, over 90% were visually identifiable as being moderately to highly oxidized.

Table 13-36 Breakdown of Metallurgical Samples by Oxidation State

Oxidation State	Total	%
Ox0	49	--- (1)
Ox1	80	8
Ox2	305	32
Ox3	557	60
Total	**991**	**100**

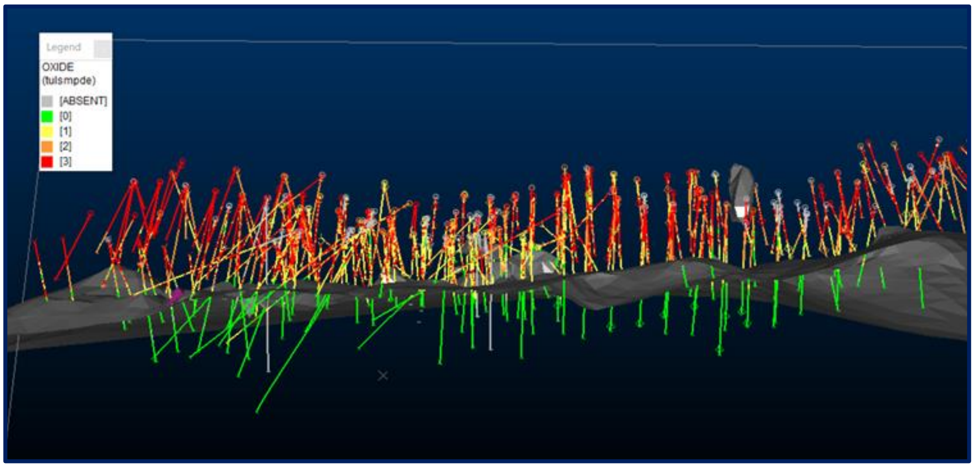
1. Ox0 samples were refractory and therefore not included with leach feed samples

13.4.1.3. Data Handling

The testwork results were back flagged to the drillhole data to be used in the model estimate. The composites were all comprised of single consecutive runs of core of varying lengths to achieve a target sample weight. The length of each composite varied according to the competency of the core. Where core was less competent longer runs were used to achieve the desired weight.

LeapFrog software was used to generate a boundary between the oxide (heap leachable) zones in the upper parts of the deposit from the un-weathered sulfide (refractory) below. Figure 13-9 shows how oxide states for drillhole samples were used to define the oxide-sulfide boundary.

Figure 13-9 Oxide – Sulphide Boundary



The intermixed nature of the material in the upper oxidized zone made it impractical to define "hard" boundaries for potentially leachable material. Instead the average extraction for each oxide code was applied to each sample of that corresponding state. For example, as Ox2 had an average extraction of 72.8%, every Ox2 sample was assigned an extraction of 72.8%. Note that Figure 13-9 is not a histogram, but is merely a picture which shows the trends in the data.

13.4.1.4. Data Assessment

Once the composite extraction values were back-flagged to the drilling data, an assessment through logged geological parameters was possible. Since there are many influences on the recovery of the metals a range of recoveries for each oxidation state was first identified as seen in Table 13-37.

Table 13-37 Range of Recoveries for each Oxidation State

Extraction	Oxidation State			
(%)	Ox0	Ox1	Ox2	Ox3
Mean	15.3	54.9	72.8	77.6
Low	9.8	10.6	42.9	42.9
High	65.9	80.8	88.7	97.9
Count	49	80	305	557

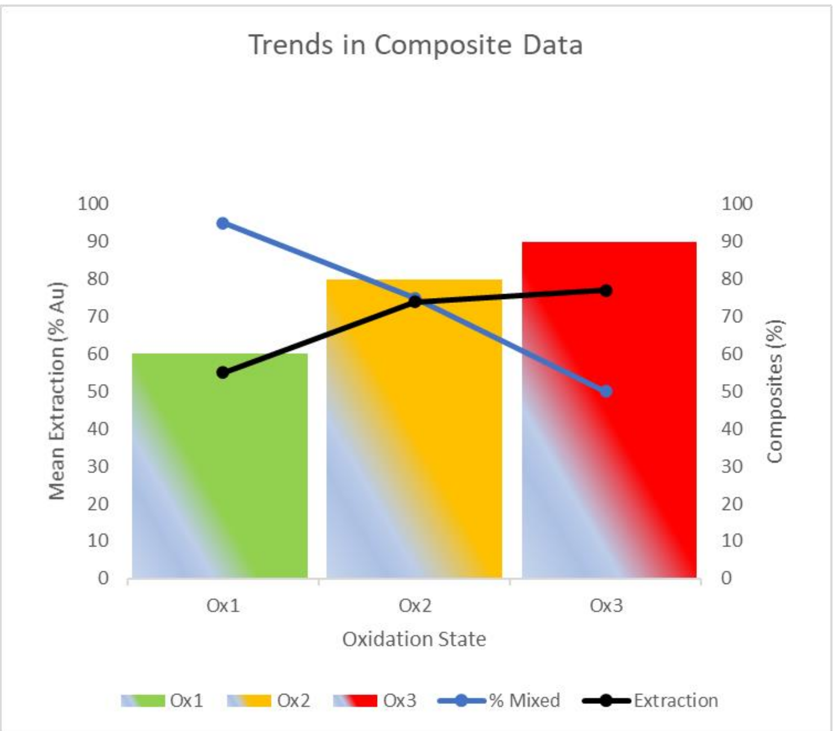
It can be seen that there is a substantial range of recoveries per oxidation state. Due to the increase in average recovery as the oxidation state increases, the main area of assessment is the oxidation state. Sulphur grades were also assessed for their contribution to extraction. Recovery was estimated for each block in the model using the assigned average values as source data for the oxidation state and through the IDW2 method. Only oxidation states, Ox1, Ox2, and Ox3 were used to estimate oxide recoveries. Table 13-38 shows that the recovery estimate using IDW2 by oxide class is remarkably similar to the results of test work.

TABLE 13-38 ESTIMATED RECOVERY (IDW2) VS BOTTLE ROLL EXTRACTION BY OXIDE CLASS

Oxide Class	Estimated Recovery (%)	Bottle Roll Extraction (%)
1	55	57
2	73	75
3	78	78
Weighted Average	**74**	**75**

Two trends can be noted. The first trend indicates how mean extraction increases with intensity of oxidation. Material with a low degree of oxidation, Ox1, has a mean estimated recovery of 55%. This increases to 73% for moderately oxidized Ox2 material and 77% for highly oxidized Ox3 material.

FIGURE 13-10 EXTRACTION TRENDS ACCORDING TO OXIDATION LEVELS



The second trend indicates how the degree to which different oxidation states are mixed increases as oxidation intensity declines. For example, about half of the composites which contain highly oxidized Ox3 material are mixed with other material types, three quarters of the composites which contain moderately oxidized Ox2 material are mixed with other material types, and almost all the composites which contain poorly oxidized Ox1 material are mixed with other material types.

For the unmixed Ox3 composites, it is interesting to note that if the bottom 20% of the samples in these composites is removed, the mean only increases about 3%. This suggests that highly oxidized material is relatively insensitive to the effects of other factors which may influence recovery. A similar change to the unmixed Ox2 composites results in a 7% change in the mean implying that the lower degree of oxidation makes the extraction of this material more apt to be influenced by other factors.

Based on the preceding observations, even if Ox2 and Ox3 have similar recoveries when unmixed, Ox2 is more likely to occur mixed with lower extraction material and therefore exhibit lower recovery when processed.

It appears that this method offers the most reliable method of estimating recoveries. This is due to the block model scale reflecting the testwork accurately as the averages for each zone are consistent with the input data. The weighted average, excluding completely fresh material which is not considered recoverable, is acceptably close to the raw data. This method produces results closer to the raw data than other methods investigated, such as sulphur regression.

Estimating metallurgical recovery using IDW2 applied to test work extractions grouped by oxide class generated an average gold recovery for ore within the 2020 EOY pit limits of 73.6%. As a by-product with only nominal value, the recovery for silver used in the study was based on the average result from the MLI test work in 2018, being 63.4%.

13.4.2. Conventional Recovery Averaging

A mathematical average of the testwork recoveries that conform to the three criteria of:

- The sample must fall within the mining pit;
- The sample must have a total sulphur below 0.5%;
- The sample must have a minimum gold grade of 0.2 g/t.

The extraction adjustment factors (EAFs) relate the bottle roll tests to the column leach tests. The EAFs are calculated in Sections 13.3.1 and 13.3.2

This conventional model only considers the drillholes that underwent relevant test work and does not predict the ore recoveries between the drillholes taken for sampling. Therefore this is less accurate than the block model.

Extraction adjustment factors (Table 13-39) were applied to the bottle roll leach extractions, to estimate the heap leach extractions under the proposed operating conditions. The adjusted bottle roll leach extractions were used to predict the heap leach extraction as these more accurately reciprocate a heap leach. An average of the extraction recoveries of gold and silver in Table 13-34 was calculated as 75.46% and 64.58%.

Table 13-39 Extraction Adjustment Factors

	Au	Ag
Adjustment Factor from WAI (2017)	0.99	N/A
Adjustment Factor from MCL (2018)	1.05	1.07
Average	**1.02**	**1.07**

Heap leach operational extractions (Table 13-40) were calculated by adjusting the bottle roll extractions.

Table 13-40 Heap Leach Operational Extractions

	Unit	Au	Ag
Average Bottle Roll Extraction	%	75.46	64.58
Extraction Adjustment Factor	-	1.02	1.07
Heap Leach Operational Extractions	%	76.97	69.10

The metallurgical recovery was calculated by assuming 99.2% adsorption efficiency, and 99.2% elution efficiency (Table 13-41).

Table 13-41 Metallurgical Recovery Calculation

	Au	Ag
Recovery	75.74	68.00

Based on the above, the recoveries generated by inverse distance weighting applied to bottle roll extractions grouped by oxide class can be considered to reflect appropriate adjustments for operating efficiencies and be somewhat conservative.

13.5. Reagent Consumption

Similar to the extraction factors used for Au and Ag, a consumption adjustment factor (CAR) for reagent consumption in the leach is calculated.

Consumption adjustment factor for Cyanide and Lime consumption between the column and bottle roll tests can be seen in Table 13-42.

Table 13-42 Consumption Adjustment Factor for NaCN and Lime

Composite	Bottle Reagent Consumption (kg/t)		Column Reagent Consumption (kg/t)		CAF	
ID	NaCN	Lime	NaCN	Lime	NaCN	Lime
WAI (2017) Composite 3	0.98	0.15	1.65	0.16	1.68	1.07
WAI (2017) Composite 4	1.37	0.29	1.78	0.07	1.30	0.24
WAI (2017) Composite 5	2.11	0.44	2.56	0.06	1.21	0.14
WAI (2017) Composite 9, 10, 11*	1.10	0.19	1.60	0.02	1.46	0.11
WAI (2017) Composite 12, 13*	1.87	0.21	1.87	0.04	1.00	0.20
WAI (2017) Composite 14	1.17	0.10	1.36	0.05	1.16	0.50
WAI (2017) Composite 20	1.55	0.05	1.57	0.02	1.01	0.40
WAI (2017) New master composite	1.24	0.12	1.24	0.03	1.00	0.25
MLI (2018) Composite A	0.45	0.30	1.03	0.35	2.28	1.17
MLI (2018) Composite B	0.83	0.27	0.71	0.35	0.86	1.29
MLI (2018) Composite C	0.49	0.28	0.72	0.35	1.47	1.25
MLI (2018) Composite E	0.65	0.26	1.04	0.35	1.60	1.33

Composite	Bottle Reagent Consumption (kg/t)		Column Reagent Consumption (kg/t)		CAF	
ID	NaCN	Lime	NaCN	Lime	NaCN	Lime
MLI (2018) Composite F	0.53	0.25	0.96	0.35	1.81	1.40
MLI (2018) Composite H	0.77	0.20	1.15	0.35	1.49	1.75
MLI (2018) Composite 48	0.44	0.20	1.57	0.35	3.57	1.75
Average					**1.53**	**0.86**

Therefore, using the average cyanide and lime consumption for the applicable bottle roll test, the reagent consumption can be calculated. The WAI and MLI test results are used to estimate the reagent consumption of the main pit while the ALS-Stewart reagent consumption is used to calculate the reagent consumption of the mid pits. These reagent consumptions are then weighted against mined ore to predict an average reagent consumption across the LoM. The reagent consumptions can be seen in Table 13-43.

Table 13-43 Adjusted Column Leach Reagent Usage for Main Pit

	Unit	NaCN	Lime
Average Bottle Roll reagent usage	Kg/t	0.84	0.24
Extraction Adjustment Factor	-	1.53	0.86
Column Leach reagent usage	Kg/t	1.28	0.21

The reagent consumption for the ALS-Stewart testwork was flagged as being out of the acceptable 'normals' for similar applications, and was investigated further. A decision was made to reject this data (CN and Lime reagent consumption) due to:

- The testwork results showed a significant increase in reagent consumption compared to the results from the other two laboratories (up to 300%).
- The samples that ALS-Stewart used have little mineralogical variance compared to the WAI and MLI samples.
- The ALS-Stewart did not provide a final recommendation on reagent consumption.

Due to some reagents being lost and replaced when sampling is conducted during the testwork phase; a factor, considered from industry practices, is used to determine the heap leach reagent usage for cyanide. This factor can range from 0.25 to 0.33 for clean non-reactive ores. For other types of ores, a higher factor is used. This report uses a factor of approximately 0.5 given the presence of sulphur in the ore.

This factor, applied to the cyanide consumption in Table 13-43 results in a projected cyanide consumption of 0.60 kg/t. Lime consumption was estimated at 0.50 kg/t from experience.

14. MINERAL RESOURCE ESTIMATES

14.1. BACKGROUND

The Tulkubash Mineral Resource estimate is based on geological logging and interpretations, as well as grade and other information recorded from boreholes, channel samples, trench samples, and road cut samples.

While the high continuity of the host shear zone is evident, gold grade is much less continuous. The mineralised volume at Tulkubash was generated by applying wireframes that use a 0.7 g/t gold threshold for the higher-grade portion of the deposit and 0.2 g/t threshold for the lower grade portion, with the higher grades locating within lower grade halos. These wireframes were constructed with a view to provide grade and thickness continuity within the deposit.

A number of factors were taken into consideration when choosing the Mineral Resource estimation method:

- The number of samples available within the Tulkubash deposit;
- The statistical characteristics of the available sample information;
- The spatial distribution of gold mineralisation; and
- Constraining the grade estimation within geologically based domains while limiting the effects of high-grade samples so as not to overestimate grade estimation within the low grade halos.

While gold is the most significant Mineral Resource at Tulkubash, silver was also estimated, as it is expected to constitute a valuable by-product of the mineral process.

14.1.1. SUMMARY OF ESTIMATION TECHNIQUES

An updated Mineral Resource estimate of the Tulkubash zone was undertaken by Victor Usenko and Evgeny Fomichev of IGT-service LLC with the effective date of 07 November 2020. All drilling and exploration data available up to 07 November 2020 was incorporated. The methods employed for this latest update are consistent with previous methodologies used in the July 2020 Mineral Resource update.

Geological modelling and Mineral resource estimation were done using the Micromine software. Wireframes were created to represent mineralisation above 0.7 g/t and the low-grade (0.7 g/t > gold > 0.2 g/t) mineralised corridor. Grades were estimated independently within their respective wireframe envelopes using Ordinary Kriging.

Statistical and grade continuity analyses were completed in order to characterise the mineralisation and were subsequently used to develop grade interpolation parameters.

14.1.2. DATABASE

The database consisting of all data for drillholes, underground workings and trenches was received by IGT in early 2020 for the July 2020 Mineral Resource update. This was checked for technical issues including:

- Duplicate drillhole, underground workings and trench IDs;

- Missing collar coordinates;
- Depth FROM or TO absent in the sample assay file;
- Missing intervals within the sample assay file;
- Overlapping intervals within the sample assay file;
- Downhole surveys:
- Multiple surveys for the same depth in a given drillhole;
- Azimuth not between 0° and 360°;
- Dip not between 0° and 90°;
- Azimuth or dip is absent; and
- Correspondence between the total depth of the drillhole and depth of the last sample.

No critical errors were found that would materially influence the Mineral Resource estimate.

A summary of the database can be seen in Table 14-1.

TABLE 14-1 SUMMARY OF THE DATABASE SUPPLIED FOR THE TULKUBASH DEPOSIT

Category	Drillholes 2005 - 2019
Workings / Drillholes	689
Metres Driven / Drilled	97,918.9
Trace / Survey Records	3,496
Assay Intervals	64,070
Including:	
Values Au = 0 g/t (fire assay)	0
Values Au < 0.025 g/t (fire assay)	6
Values Au = 0.025 g/t (fire assay)	45,298
Values Au > 0.025 g/t (fire assay)	18,187
Values Au = “” g/t (fire assay)	579
Lithology Intervals	64,070

An updated database including the RC drilling completed in 2020 and additional trenches was then received. A summary of the database can be seen in Table 14-2.

TABLE 14-2 SUMMARY OF THE SUPPLIED DATABASE FOR THE TULKUBASH DEPOSIT FOLLOWING 2020 RC DRILLING

Category	Drillholes 2020
Workings / Drillholes	21
Metres Driven / Drilled	2,434.3
Trace / Survey Records	3,035
Assay Intervals	2,433
Including:	
Values Au = 0 g/t (fire assay)	0

Category	Drillholes 2020
Values Au < 0.025 g/t (fire assay)	0
Values Au = 0.025 g/t (fire assay)	1,703
Values Au > 0.025 g/t (fire assay)	727
Values Au = “” g/t (fire assay)	3
Lithology intervals	2,433

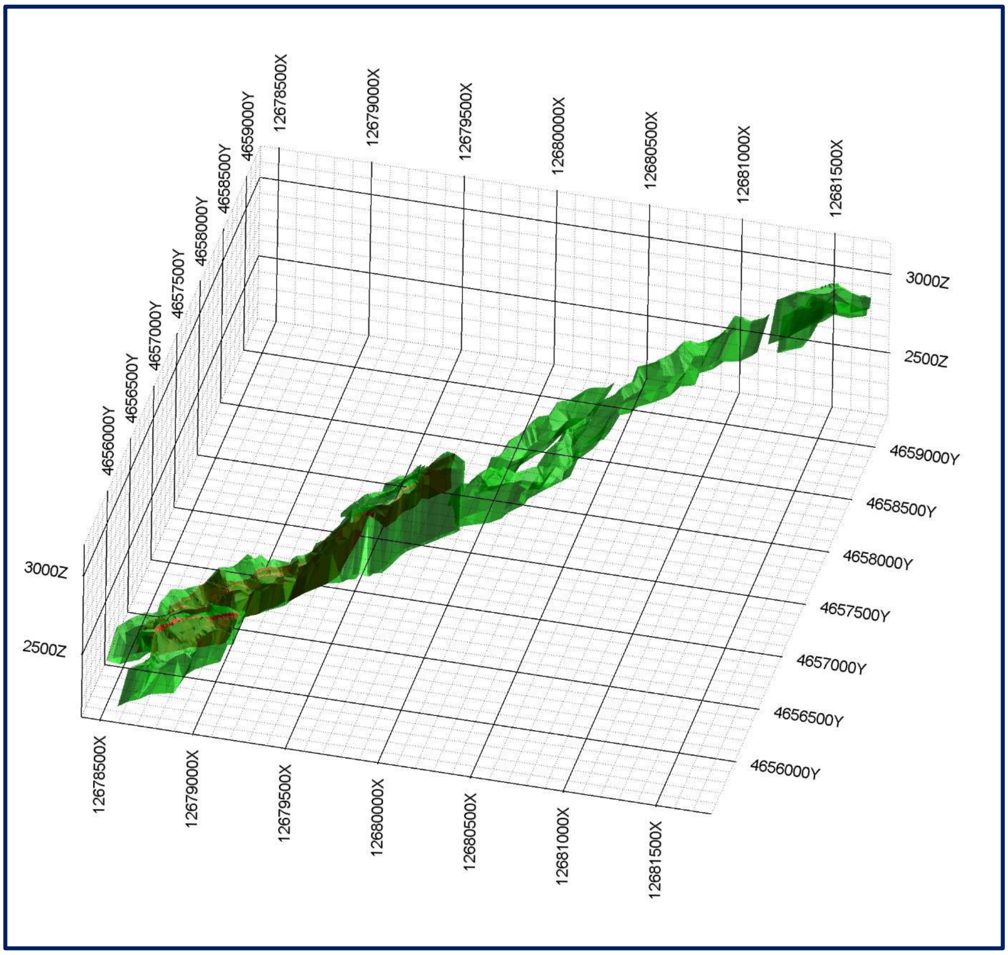
Whereas data from trenches and underground workings was used in the interpretation of mineralised zones, only drillhole data was used in grade interpolation.

14.1.3. Geological Interpretation

The Tulkubash deposit is interpreted to have formed in a shallow epithermal setting and has been classified as an epizonal orogenic gold deposit. The deposit is thought to be a brittle shear zone formed through sinistral strike-slip motion within the SFZ.

Figure 14-1 illustrates the surface expression of mineralised domain wireframes that have been modelled for Tulkubash.

Figure 14-1 Isometric View of Mineralised Domains at Tulkubash – Looking Northwest



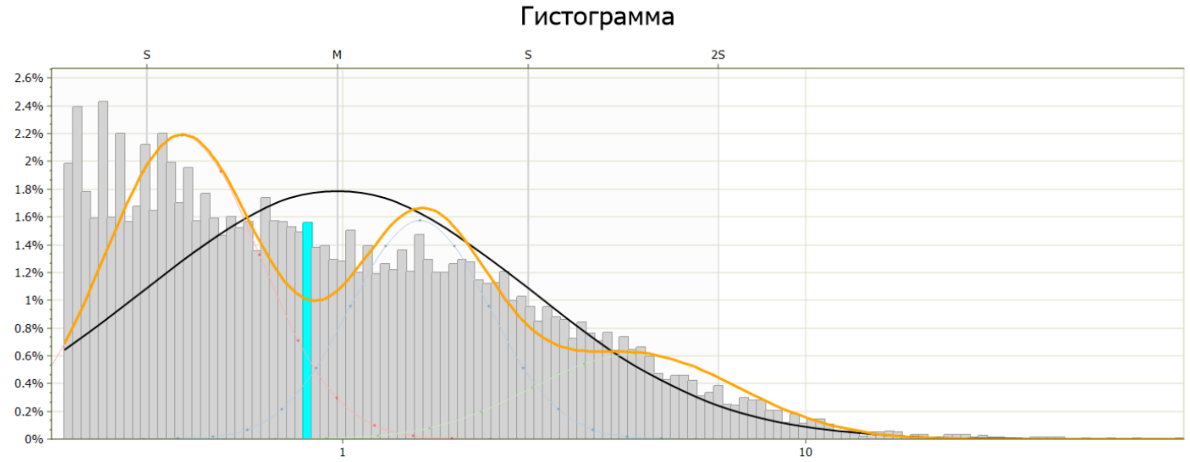
Note: Not to scale

14.1.4. WIREFRAME MODELLING

Wireframe models were created by visual inspection of drillhole section lines after statistical analysis to determine the grade cut-offs to be used for the high-grade and low-grade mineralised envelopes.

Figure 14-2 indicates two major grade populations depicting high-grade and low-grade sample assays, with a third population related to the extreme high-grade results. This analysis depicts the grade cut-off for the high-grade zones to be 0.8 g/t gold, however during interpretation of the section lines a cut-off of 0.7 g/t was applied to provide for better grade and thickness continuity.

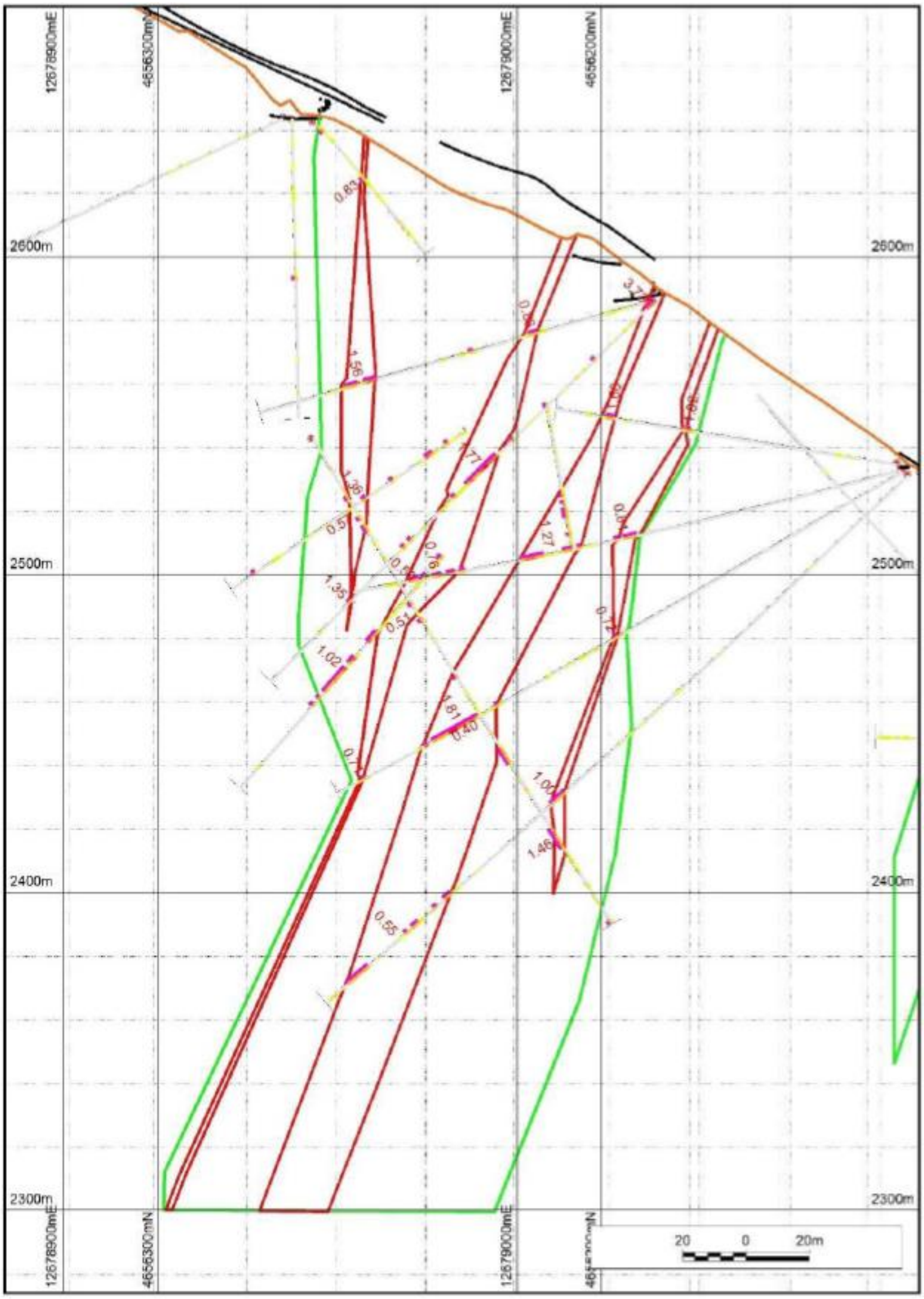
FIGURE 14-2 HISTOGRAM OF ASSAY DATA



Минимальная величина	0.25000	Кол-во	21378	Медиана	0.84000	Оценка Сишеля	1.52556
Максимальная величина	65.50000	Среднее	1.62556	Ln среднее	-0.02743	V Сишеля	0.89960
2-е наибольшее	51.17200	Дисперсия	5.33539	Ln стд. отклон.	0.94850	Гамма Сишеля	1.56798
3-е наибольшее	45.00200	Стандартное отклонение	2.30985	Геометрическое среднее	0.97295	Хи-квадрат	5050.967
4-е наибольшее	40.89000	Коэф. вариации	1.42095	Геометрич. стд. откл.	2.58182	Степени свободы	100

(Figure 14-3) shows a typical cross section of the mineralisation interpretation.

FIGURE 14-3 CROSS SECTION THROUGH TULKUBASH MINERALISATION



14.1.5. Top Cuts

In order to restrict the effect of significantly high-grade samples top cuts for grade interpolation were determined for each individual domain wireframe by analysing log probability plots of the sample grades (Figure 14-3).

Table 14-3 Top cut grades per domain

Domain	Cap Grade (g/t Au	Probability (%)
cut1_07	12	99.6
cut2-7_07	10	98.8
cut3_07	6.5	98.4
cut4_07	2	92.2
cut5_07	1.8	83.6
cut6_07	2.4	92.8
cut1_02_ver2	10	99.9
cut2_02_ver2	3.6	99.7
cut3_02_ver2	6.4	99.8
cut4_02_ver2	4	99.6
cut5_02_ver2	2.2	99.0
cut6_02_ver2	1.0	94.4
cut7_02_ver2	1.3	94.9
cut8_02_ver2	2.2	97.8
cut9_02_ver2	1.62	91.2

14.1.6. Sample Length and Compositing

All samples were coded within the wireframe models and indexed according to their relevant domain. All assay intervals from 2005 to 2019 were then composited to a standard length of 1.5 m. Samples from the 2020 RC drilling were composited to standard lengths of 1.0 m.

14.1.7. Density

1,409 specific gravity measurements were received for the Tulkubash deposit. The density was interpolated into the block model using the inverse distance squared (IDW2) method resulting in an average density for the deposit of 2.64 t/m^3.

14.1.8. Variography

Variography analysis was undertaken for high-grade and low-grade domains separately and as a result differing statistical parameters were used during gold grade estimation. Parameters used in the grade estimation can be seen in Table 14-4.

Table 14-4 Variogram Parameters

Direction	Nugget Effect	Range	Sill	Model	Az	Plunge
Variogram Models for 0.2 g/t Au Cut-off Grade Domains						
1st	0.27	19/37	0.54/0.19	spheric	150	-20
2nd	0.27	29/40	0.54/0.19	spheric	60	0
3rd	0.27	12/40	0.54/0.19	spheric	150	70

Direction	Nugget Effect	Range	Sill	Model	Az	Plunge
Variogram Models for 0.7 g/t Au Cut-off Grade Domains						
1st	0.31	13/20	0.37/0.32	spheric	150	-20
2nd	0.31	18/60	0.37/0.32	spheric	60	0
3rd	0.31	15/60	0.37/0.32	spheric	150	70

Normal score Variogram models for the 0.2 g/t gold and 0.7 g/t gold cut-off domains are shown in Figure 14-5 and Figure 14-6.

FIGURE 14-4 NORMAL SCORE VARIOGRAM MODELS FOR 0.2 G/T CUT-OFF DOMAINS

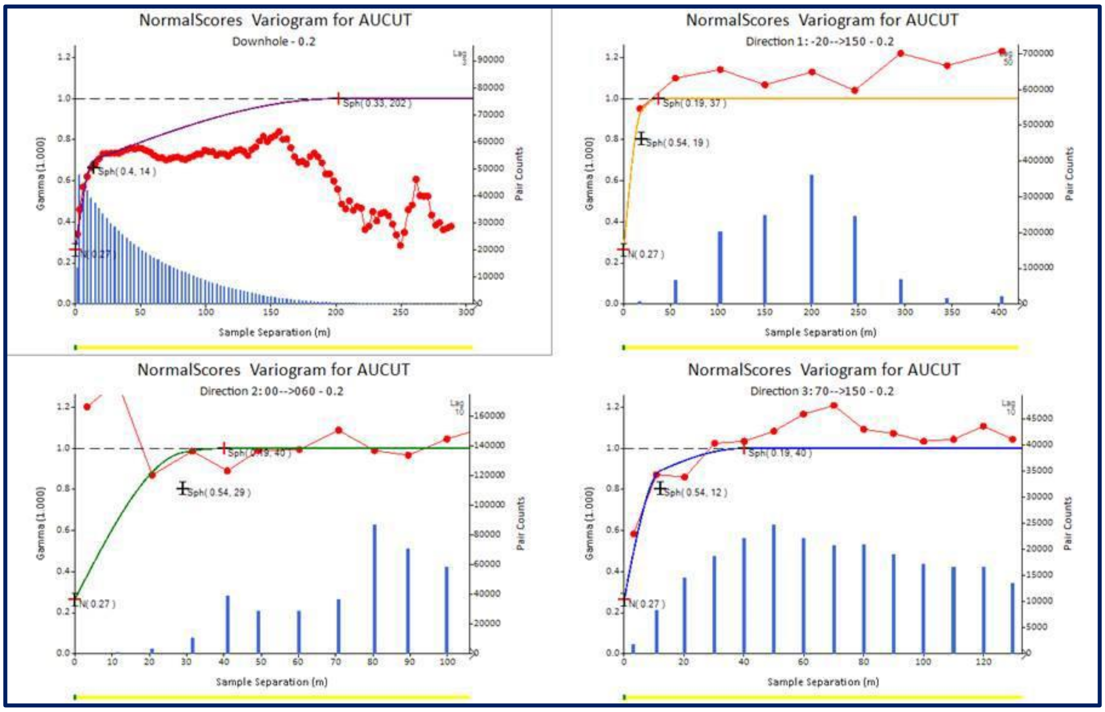
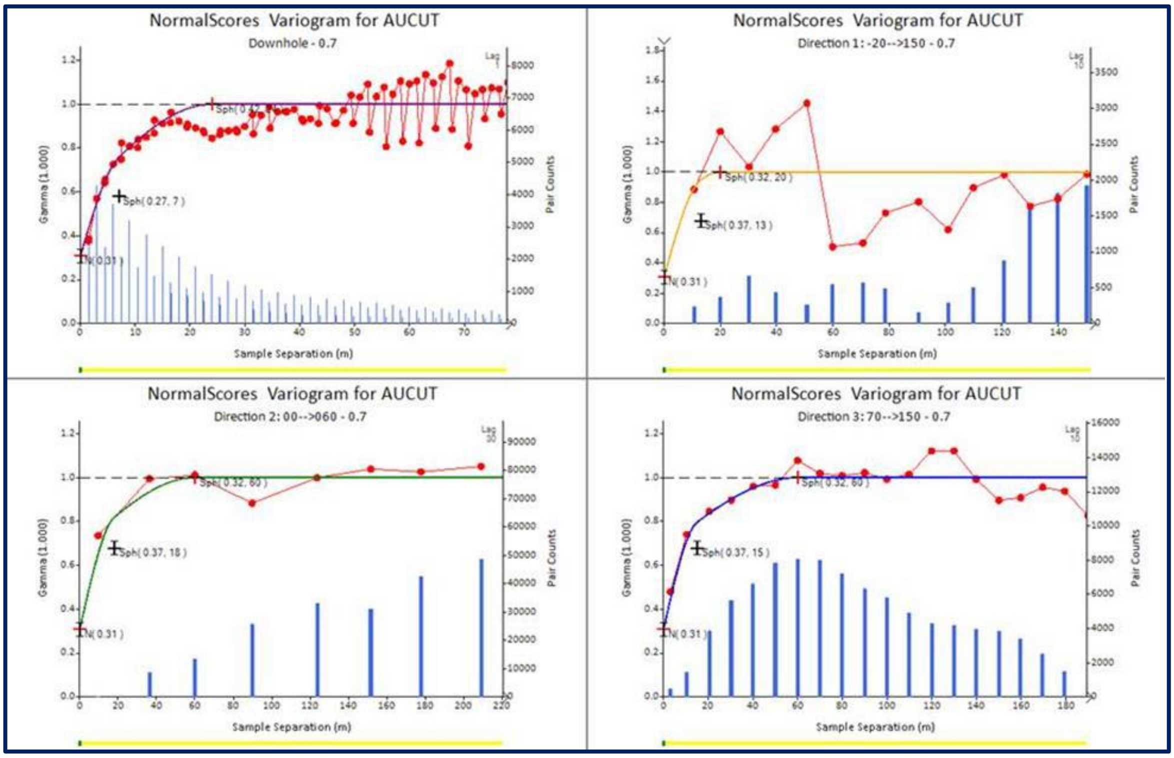


FIGURE 14-5 NORMAL SCORE VARIOGRAM MODELS FOR 0.7 G/T CUT-OFF DOMAINS



14.1.9. MINERAL RESOURCE BLOCK MODELS

The block model was constructed using 5 m x 5 m x 5 m cell size as this was considered to best reflect gold distribution and is similar to the SMU.

14.1.10. INTERPOLATION STRATEGY

Grades were estimated by OK using dynamic anisotropy during the interpolation process with search ellipse parameters determined during the geostatistical analysis. Multiple passes were run to interpolated into all blocks. Ellipse orientations per domain are shown in Table 14-5 with interpolation parameters shown in Table 14-6.

A constant ellipse parameter of 40.6 m x 18.5 m x 27.1 m was used.

TABLE 14-5 ELLIPSE ORIENTATIONS – JULY 2020 DOMAINS

Sub-Domain	Azimuth	Plunge	Rotation
2-7_07	41	-20	-20
3_07	41	-20	-20
6_07	41	-20	-20
1_07	41	-20	-20
2-7_07_3	37	-20	-8
1_07_2	60	-20	-13
4_07	41	-20	-20
5_07	41	-20	-20

Sub-Domain	Azimuth	Plunge	Rotation
2-7_07_2	15	-20	-8
1_02_1	42	-20	-20
1_02_1_1	42	-20	-20
1_02_2	44	-20	-20
1_02_3	55	-20	-20
1_02_4	63	-20	-20
1_02_5	45	-20	-20
1_02_6	44	-20	-20
1_02_7	46	-20	-20
1_02_8	43	-20	-20
1_02_9	52	-20	-20
1_02_10	52	-20	-20

TABLE 14-6 INTERPOLATION PARAMETERS – JULY 2020 DOMAINS

Run	Minimum Drillholes	Minimum Points	Maximum Points	Radius Factor
1	1	2	5	0.05/0.07
2	3	6	30	0.67
3	2	4	30	1
4	1	1	30	1.5
5	1	1	30	2
6	1	1	30	20

14.1.11. BLOCK MODEL VALIDATION

Upon completion of the grade interpolation the block model was checked visually as well as statistically. Review of the grade distribution in sections considers the block model to correspond well with the assay results.

Figure 14-6 displays a typical cross section through the interpolated block model and corresponding samples. Figure 14-7 to Figure 14-9 show swath plots comparing the composited samples to the grade interpolation within the block model.

FIGURE 14-6 CROSS SECTION DEPICTING THE TULKUBASH BLOCKMODEL AND CORRESPONDING SAMPLES

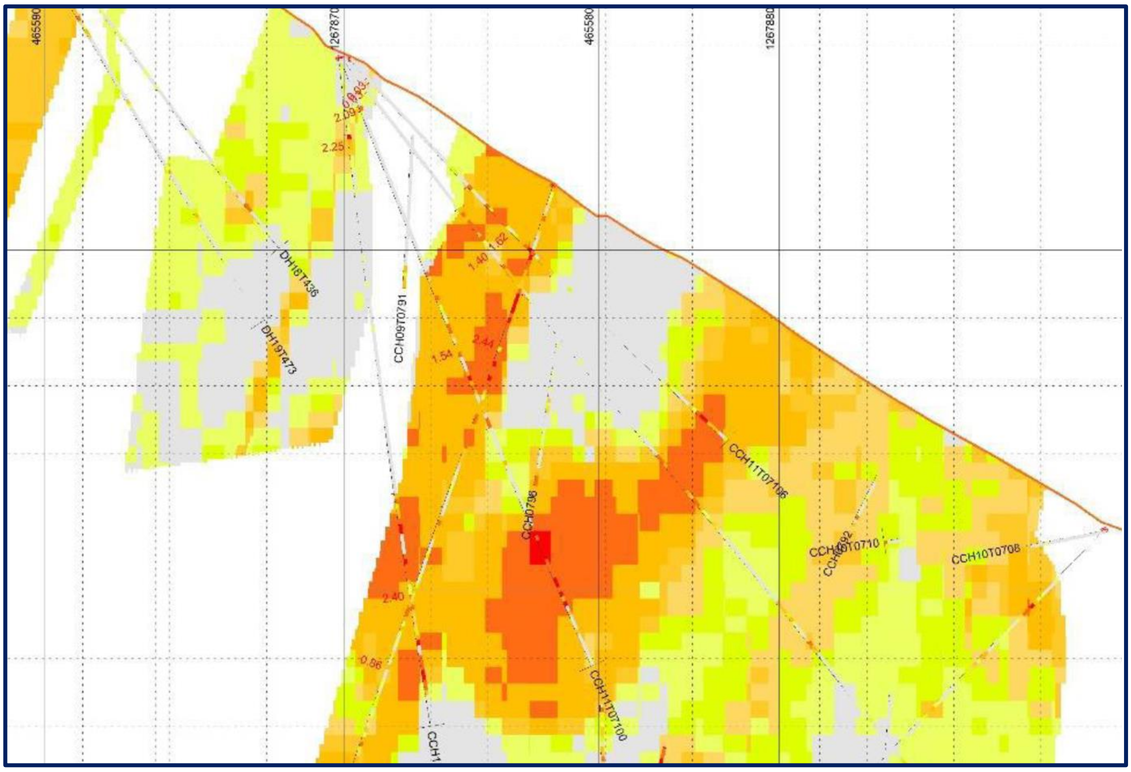


FIGURE 14-7 SWATH PLOT FOR GRADE IN COMPOSITE FILE COMPARING TO BLOCK MODEL (AZ42)

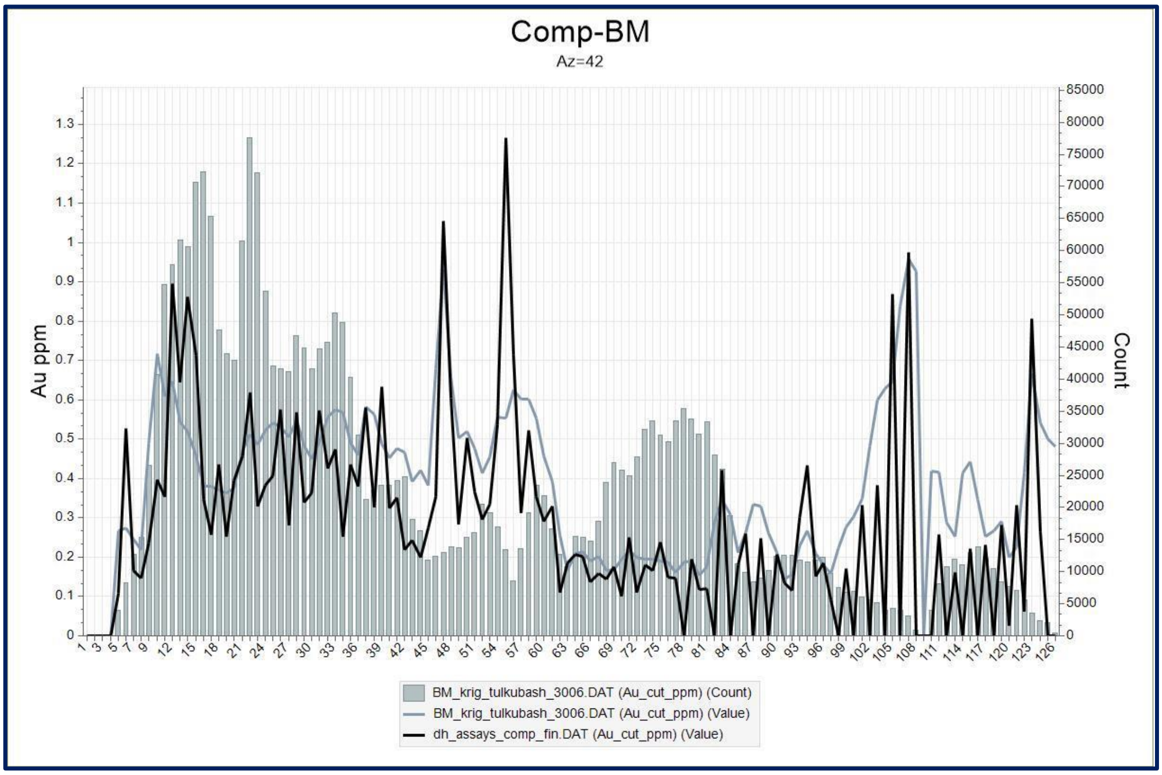


Figure 14-8 Swath Plots for Grades in Composite File Compared to Block Model (AZ132)

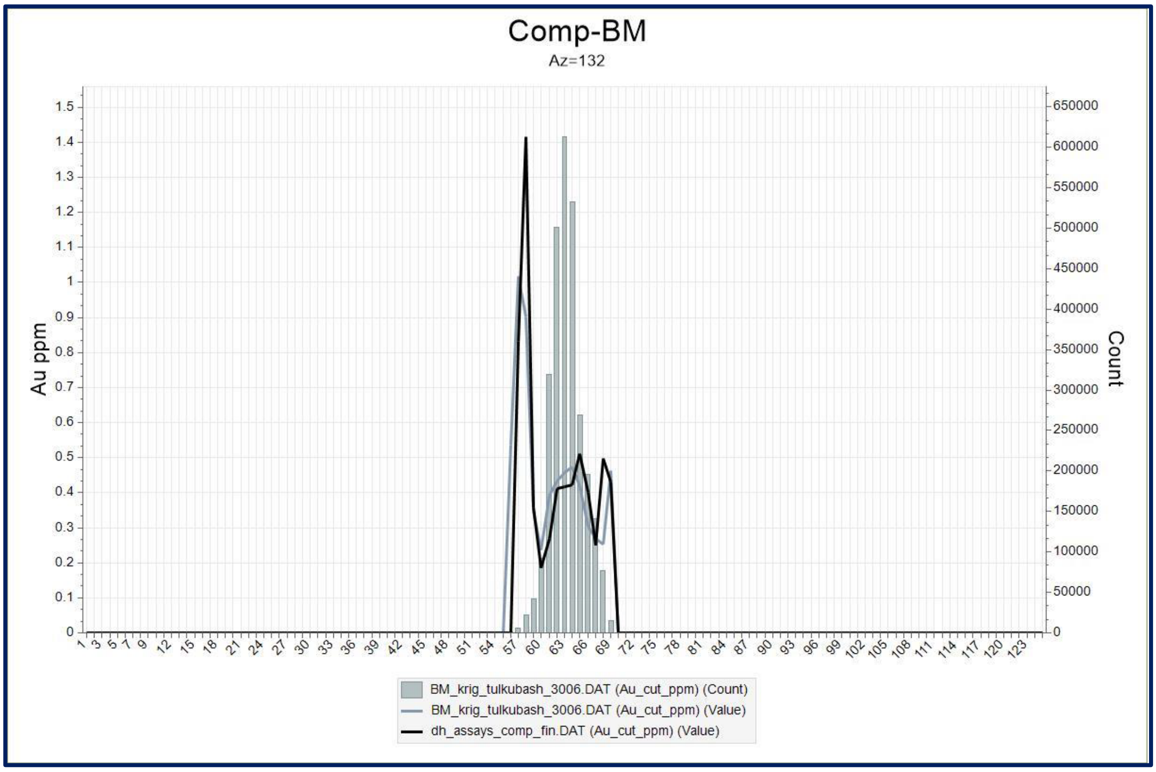
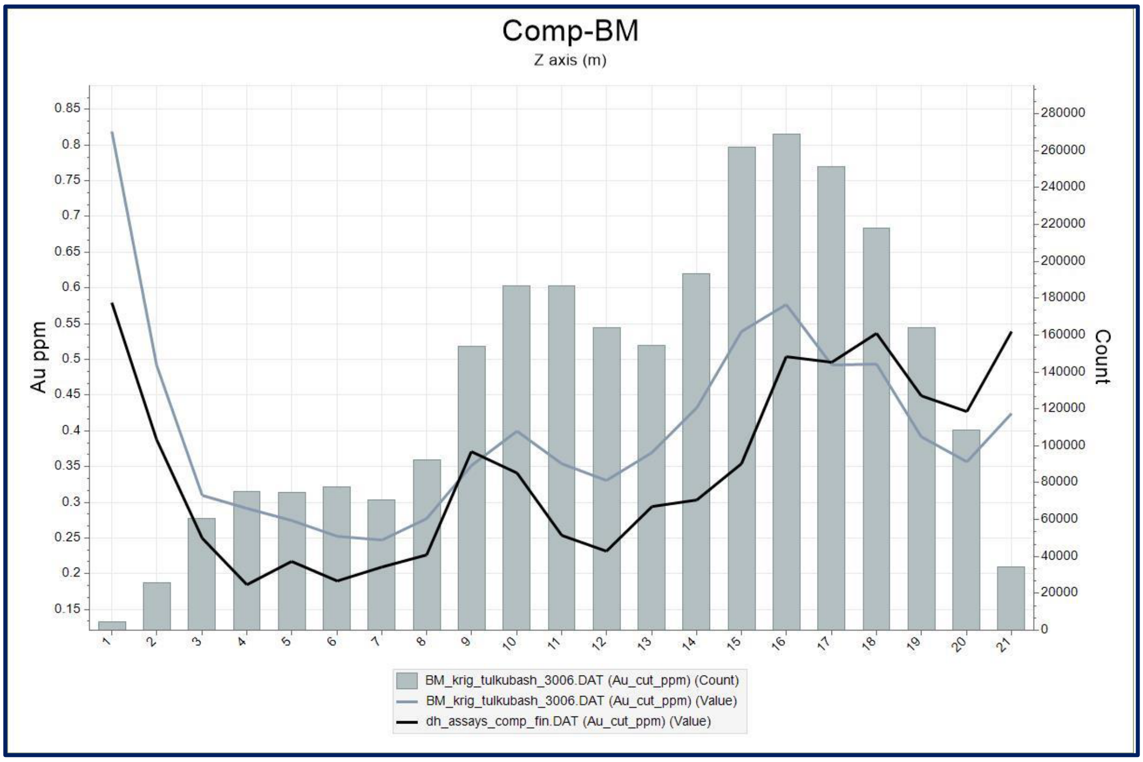


Figure 14-9 Swath Plots for Grades in Composite File Compared to Block Model (with Depth)



14.1.12. Mineral Resource Classification

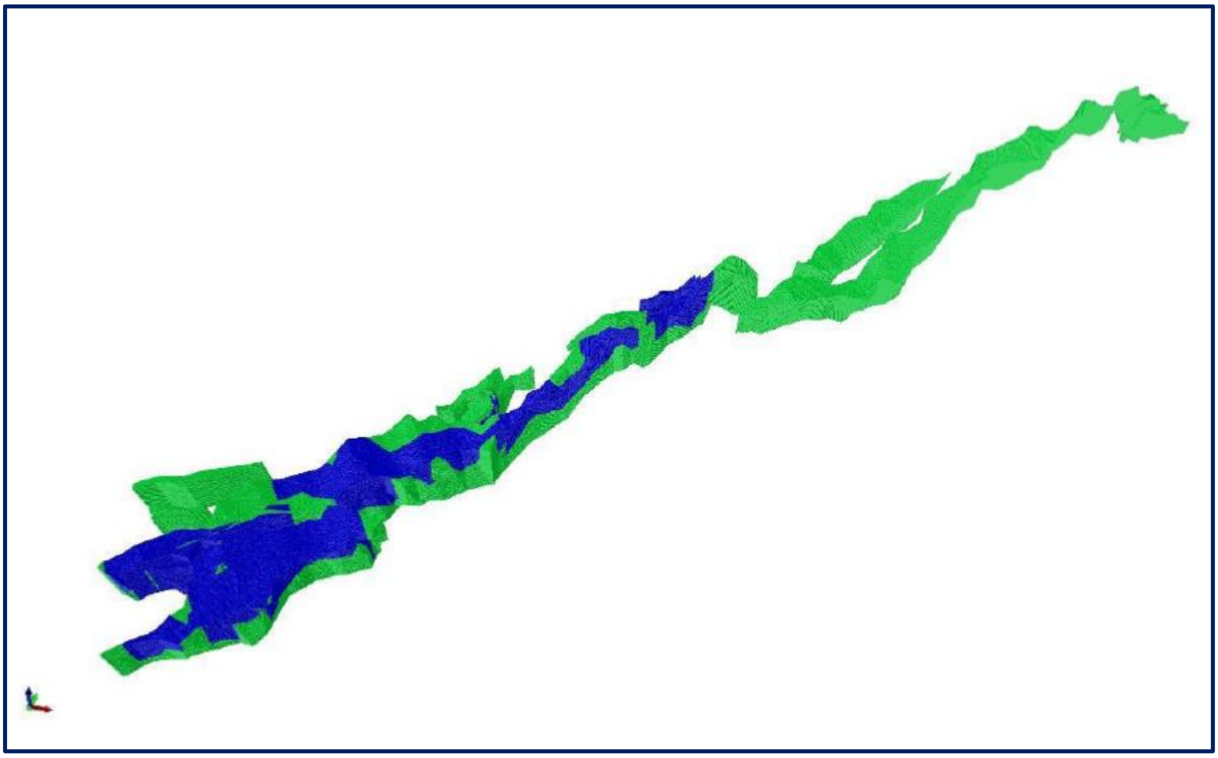
Mineral Resource classification was undertaken manually, section by section. No Measured Mineral Resource has been declared with the current Mineral Resources being within the Indicated and Inferred categories.

The Mineral Resource classification for the Tulkubash deposit considers the following criteria:

- Variography results;
- Grade and thickness variability and/or continuity; and
- Confirmation of grade at surface (trench and road sampling).

Indicated Mineral Resources have been constrained to the area covered by drilling on the 40 m grid spacing. Inferred Mineral Resources were constrained to 80 m, along strike and down-dip, from the furthest drillhole data point.

FIGURE 14-10 INDICATED MINERAL RESOURCE CLASSIFICATION FOR TULKUBASH DEPOSIT



14.1.13. Mineral Resource Tabulation

The economic parameters considered for the Mineral Resource declaration were obtained from the Client and include:

- Gold price of USD1,800/tr oz;
- Gold recovery of 72.6%;
- Mining cost of USD1.89/t;
- Operating cost of USD7.24/t; and

The updated Mineral Resource for Tulkubash is summarised in Table 14-7 at a cut-off grade of 0.21 g/t. The definitions of Mineral Resources as outlined within the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC code (2012)) were adopted in order to classify the Resources. Classification of the Mineral Resource considered the following aspects:

- Variography results indicate a 40 m distance along strike for Indicated resources;
- Grade and thickness continuity and variability;
- Confirmation of grade at surface; and
- A maximum length along strike and down dip of 80 m from the last drillhole;

The effective date of the updated Mineral Resource is 07 November 2020.

TABLE 14-7 TULKUBASH MINERAL RESOURCE STATEMENT (EFFECTIVE 7 NOVEMBER 2020)

Classification	Quantity (Mt)	Grade Au (g/t)	Contained Metal Au (koz)
Measured	-	-	-
Indicated	28,505	0.86	789
Inferred	21,412	0.56	388

Notes:

1. *Numbers are rounded in accordance with disclosure guidelines and may not sum accurately;*
2. *The Mineral Resource has been estimated using 5.0 m x 5.0 m x 5.0 m (x, y, z) blocks;*
3. *The estimate was constrained to the mineralised zone using wireframe solid models;*
4. *The wireframes were sub-domained to isolate the strongly mineralised main zone from the gold mineralisation in the main structural corridor;*
5. *Grade estimates were based on 1.5 m composited assay data; and*
6. *The Mineral Resource estimate has been reported to 0.21 g/t cut-off grade.*

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

15. ORE RESERVE ESTIMATES

The Ore Reserves for the Tulkubash Gold Project have been updated according to the code prescribed by the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ('the JORC Code'), 2012. The Ore Reserves have been estimated by considering only the Measured and Indicated Mineral Resources that can be exploited economically. The Ore Reserve estimate has been based on the latest geological block model, which included processing recovery data in each of the 5 m by 5 m by 5 m blocks that informed the pit optimisation and subsequent final open pit design.

The 2020 End-of-Year (EOY) Ore Reserves stated in this section, supersede the 2018 EOY Ore Reserves which served as the basis for the 2019 Feasibility Study (FS) update. The 2020 EOY Ore Reserve is based on a revised Resource model which incorporates the results of exploration drilling up to the end of 2020, a new geological interpretation, and technical and economic parameters established in the 2019 BFS or modifications based on subsequent work.

The Ore Reserve estimate has considered hydrogeology, geotechnical criteria and various other modifying factors which are described at an acceptable level of accuracy in the Mine Design Section (Section 16). Only Measured and Indicated Mineral Resources were used to generated Ore Reserves. Inferred Mineral Resources were treated as waste. The mine production schedule in Section 16 is based upon the Ore Reserve presented here.

An economic assessment of the Ore Reserve was conducted prior to declaring the Ore Reserve statement.

The 2020 EOY Ore Reserve estimate is stated in Table 15-1, which reports a contained gold content of 571 koz, all of which have been categorised as Probable.

TABLE 15-1 TULKUBASH ORE RESERVES AS AT 2020 EOY

Category	Quantity (Mt)	Grade (g/t)	Content (kg)	Content (koz)
Proven	-	-	-	-
Probable	20.9	0.85	17,760	571
Total	**20.9**	**0.85**	**17,760**	**571**

Source: *Chaarat, 2021*

Notes:

1. *This statement of Ore Reserves has been prepared by Mr Peter C Carter, an independent consulting mining engineer, based on a review of work performed by Chaarat Gold technical staff;*
2. *Mr Carter is a member of the Association of Professional Engineers and Geoscientists of British Columbia and is qualified as a Competent Person under the JORC Code, 2012;*
3. *The Ore Reserve has been reported in accordance with the classification criteria of the JORC Code, 2012 and is 100% attributable to Chaarat;*
4. *Any apparent computational errors are due to rounding and are not considered significant;*
5. *Ore Reserves are reported with appropriate modifying factors of mining dilution (8%) and mining recovery (97.5%);*
6. *Ore Reserves are reported at the head grade delivered to the leach pad;*
7. *The Ore Reserves are stated at a price of USD1,450/tr oz as at 2020 EOY;*
8. *Although stated separately, the Mineral Resources are inclusive of Ore Reserves;*
9. *No Inferred Mineral Resources have been included in the Ore Reserve estimate;*
10. *Quantities are reported in metric tonnes; grades are reported in grams per metric tonne = ppm (parts per million);*

11. *The input studies are to the prescribed level of accuracy; and*
12. *The Ore Reserve estimates contained herein may be subject to legal, political, environmental or other risks that could materially affect the potential development of such Ore Reserves.*

Table 15-2 shows a re-statement of the Ore Reserve. Approximately 95% of the contained gold is associated with the MZ Pit area.

Table 15-2 Tulkubash Ore Reserves as at 2020 EOY

Zone	Ore				Waste Quantity (Mt)	Total Quantity (Mt)	SR (t:t)
	Quantity (Mt)	Au (g/t)	Gold Content (oz)	Au Recovery (%)			
Main Zone	19.4	0.86	537,815	73%	50.3	69.8	2.59
Mid Zone	1.4	0.72	33,324	77%	3.7	5.1	2.54
East Zone	-	-	-	0%	-	-	-
Total	**20.9**	**0.85**	**571,139**	**73%**	**54.1**	**74.9**	**2.59**

Source: Chaarat, 2021

Table 15-3 provides a comparison of the 2020 EOY Ore Reserve to the previously reported 2018 EOY Ore Reserve. This Shows that the 2020 EOY Ore Reserves represent a 6% decrease in ore tonnage and an 8% decrease in grade compared to the 2018 EOY Ore Reserves. Overall, these changes result in a 13% decrease in contained ounces of gold.

The Inferred Resources within the pit limits, which are currently treated as waste, offer the potential to increase ore tonnage and contained ounces, along with decreasing the Strip Ratio (t:t) in the order of 5% to 10%.

Table 15-3 Comparision of Tulkubash Ore Reserves as at 2018 EOY and 2020 EOY

Parameter	Units	2018 EOY	2020 EOY	Variance
Ore	Mt	22.2	20.9	-6%
Grade (Au)	g/t	0.92	0.85	-7%
Metal (Au)	koz	658	571	-13%
Waste	Mt	58.6	54.1	-8%
Total	Mt	80.8	74.9	-7%
Strip Ratio	t:t	2.64	2.59	-2%
Recovery	%	68.9	73.6	7%
Recovered Au	koz	453	419	-7%

Source: Chaarat, 2021

16. Mining Methods

16.1. Hydrogeology

The hydrogeology for the open pit designs have been informed by field investigations conducted by SRK Consulting and Tetra-Tech Engineering in 2010 and 2014 respectively. This data was subsequently used to generate a finite-element groundwater model developed by Wardell Armstrong International (WAI) in 2017.

Measurements were taken from borehole KP103 in the Main Zone (MZ) at depths correlating to an elevation of around 2,500 masl. The information showed that the discharge ranged between 4 m^3/hr and 6 m^3/hr (or 1.0 ℓ/s and 1.5 ℓ/s). Hydraulic conductivity and transmissivity were between 1.0 m/d and 3.0 m/d, and 6 m^2/d and 13 m^2/d, respectively.

A groundwater model used field data to simulate the seasonal rise and fall of the water table over a five-year period. It indicates static groundwater levels during winter, rapid recharge in spring from snow melt, and a slow decline over the summer and autumn. This suggests that groundwater levels are largely a function of local recharge infiltrating rapidly through fractures connected to surface. Pre-mining water levels were modelled at between 2,340 masl and 2,500 masl.

16.2. Geotechnical Aspects

Kyrgyzstan is a seismically active region. Studies have been conducted to establish the technical parameters which appropriately reflect seismic conditions at the site. The primary criteria is Peak Ground Acceleration (PGA) which was determined to be 0.157 G's based on a 10% probability in a 50-year return period.

The Peak Ground Accelerations (PGA) for a 10% probability of exceeding these measurements were calculated for 5, 8, 10, 12, 15 and 50 years as shown in Table 16-1 for rock and soil.

Table 16-1 Median and Mean PGA with 10% Probability of Exceedance during Time Interval

Material/Time		Peak Ground Acceleration (PGA) G					
		5 Years	8 Years	10 Years	12 Years	15 Years	50 Years
Rock	Median	0.0426	0.0553	0.0624	0.0687	0.0770	0.138
	Mean	0.0492	0.0631	0.0709	0.0778	0.0872	0.157
Soil	Median	0.0703	0.0904	0.100	0.109	0.121	0.200
	Mean	0.0718	0.0916	0.102	0.112	0.125	0.217

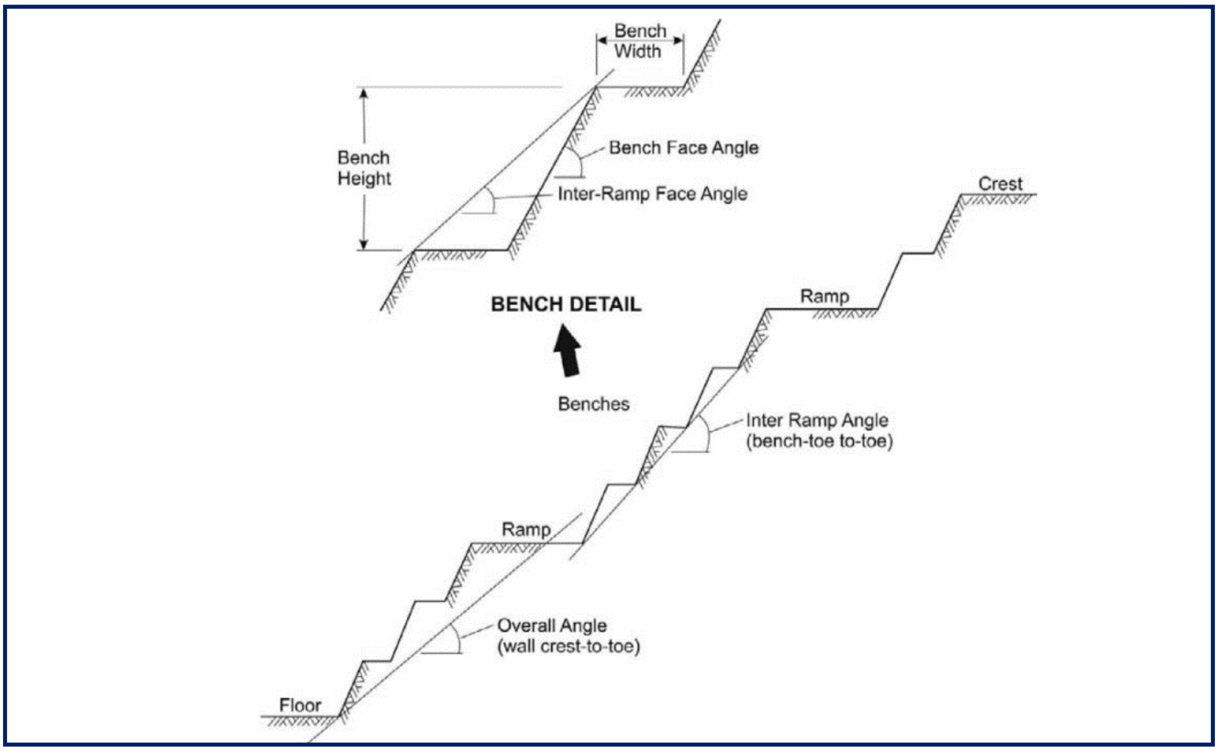
Source: Chaarat, 2020

Note: 10% Probability of exceedance based on time period.

The multiple interacting joint sets form a highly blocky rock mass and as would be expected, kinematic analysis identifies planar, wedge and toppling type structural failures as a risk to open pit mining in the MZ. Excavated bench faces are likely to unravel where relatively small blocky formations prevail, while this is less likely in the case of large block along the entire joint length. It is therefore important that careful blasting techniques be adopted for the final benches and that appropriate scaling is practised to maintain the integrity of a bench's rock mass to minimise risk.

Limit equilibrium stability analyses was used to examine the overall slope stability of the pit design. A minimum factor of safety (FoS) of 1.2 was targeted for the inter-ramp angles (IRA), and 1.3 for the overall pit slopes (Figure 16-1).

FIGURE 16-1 PIT WALL TERMINOLOGY



Source: Chaarat, 2020

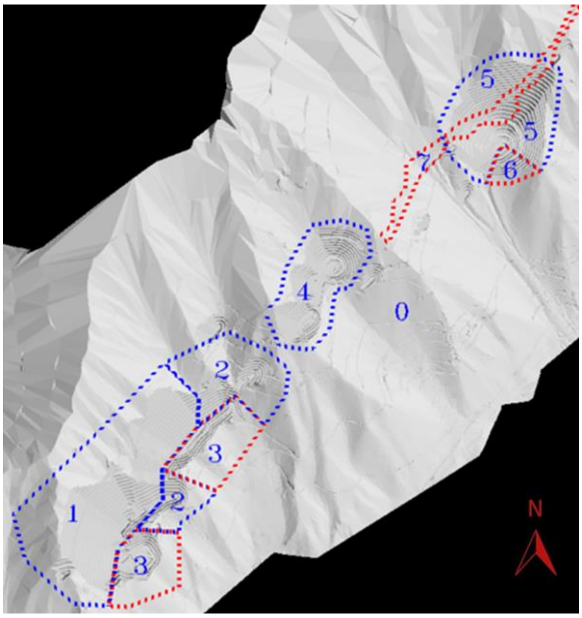
The above work showed that in all instances the FoS remains above 1.3. However, a decrease in the FoS was indicated with increasing hydro-geological influence. Horizontal drains and/or vertical pumping, depending on the weather conditions, may be required to reduce in-situ water pressures due to increased water levels. This option should be carefully assessed during the early stages of mining while ramping up to steady state operations. This work also allowed geotechnical design parameters to be developed for each sector of the MZ, and these parameters were also used for the relatively shallow open pit designs North-East of the MZ in the Satellite Zone (SZ).

The following associated overall slope geometry was recommended by WAI:

- Berm width: 5.5 m;
- Bench height: 15 m;
- Bench face angle: 66° and 75°;
- Inter-ramp angle: 51° and 58°; and
- Geotechnical berm: 9.5 m.

The bench face angles in the final designs vary between 60° and 75°, with 8 m berm widths to comply with local regulations and to allow mechanised cleaning. The inter-ramp angles (IRA) of around 51° and 58° for the various design sectors (i.e., Sectors 1 to 4), were flattened slightly to accommodate bench width and aligned with the WAI bench geometry recommendations. The IRA for the fault zone was reduced to 45° (Figure 16-2). It is noted that design sectors 5 to 7 were not used in the open pit design since the revised 2020 design did not extend beyond Sector 4.

FIGURE 16-2 PIT OPTIMISATION INTER-RAMP ANGLE AREA



Inter-Ramp Angles

Design Sectors	IRA (°)
0 (Default Value)	55.5
1	48.5
2	55.5
3	46
4	49
5	50
6	45
7 (Fault Zone)	45

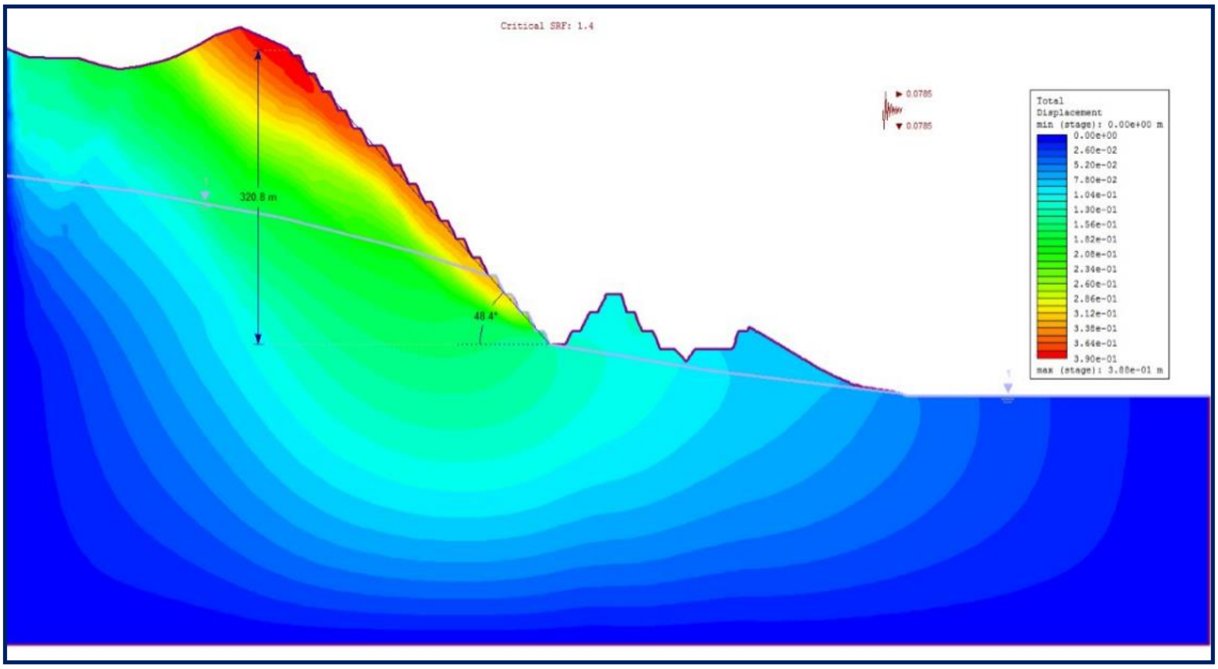
Source: Chaarat, 2020

In the previous pit design study done in 2019, a numerical modelling approach was undertaken using appropriate modelling software (RS2 Rocscience Inc) to test the appropriateness of the slope angles previously designed for the MZ Pit where the final highwall reached a height of 370 m. The software uses finite element analysis by increasing gravity field stress loading until the slope becomes unstable and/or decreasing the shear strength of the materials, until the slope becomes unstable. It takes account of the rock mass properties, the groundwater line, seismic acceleration and the highest anticipated pit wall.

In the revised 2020 pit design it was noted that the final highwall reaches a height of 375 m, a moderate highwall height increase of approximately 1.4%. This increase is not considered significant when compared to the earlier 2019 design and it is considered acceptable that the results of the earlier 2019 slope stability analysis are applied to the revised 2020 design.

Figure 16-3 presents the earlier 2019 results of this modelling.

Figure 16-3 Resulting Ground Displacement



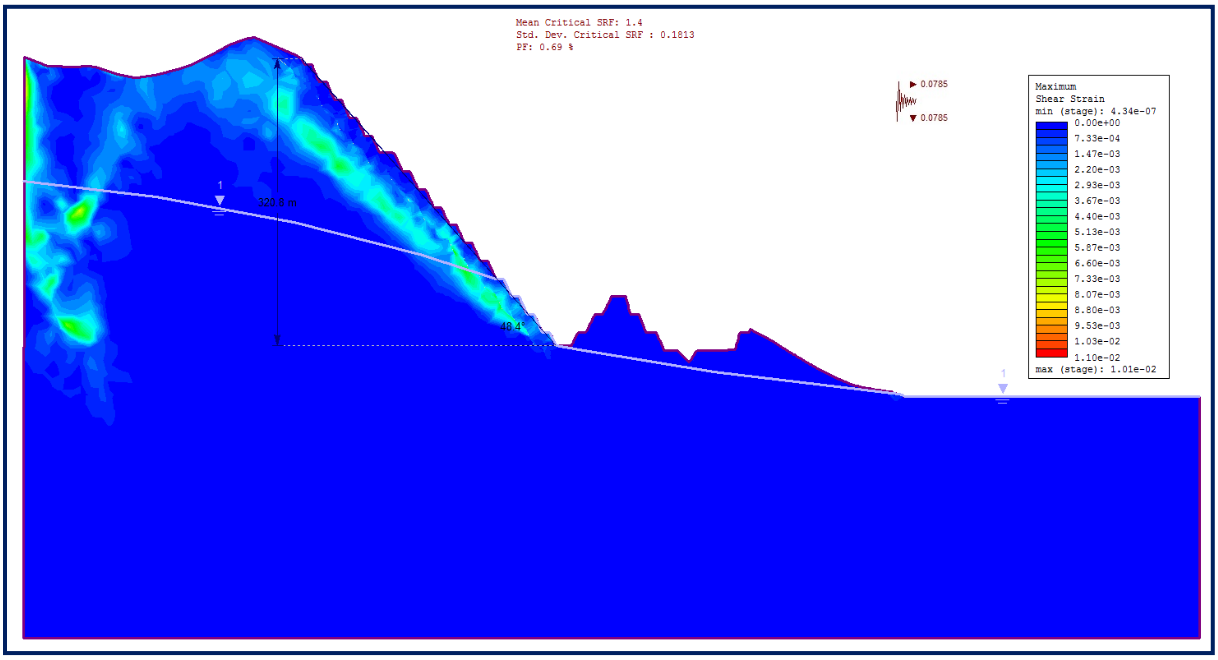
Source: RS2 Software

Figure 16-3 shows the earlier (2019) analysis and refers to a critical Shear Strength Reduction Factor (SRF) of 1.4 which is essentially the FoS of the slope. The results of this modelling confirm that the designed slope geometry will be stable for the MZ Pit. The SRF of 1.4 is larger than the threshold limit of 1.3 as suggested by Stacey (2002).

Unconfined compressive strength (± 10Mpa) and Hoek-Brown material constant (± 0.3) where used as variables for a probabilistic analysis.

The results of the earlier 2019 analysis are presented in Figure 16-4, which indicates that the probability of a FoS of less than or 1.4 would be around 0.7%, which means that the designed MZ Pit with its adopted geometry, is likely to remain stable.

FIGURE 16-4 RESULT OF PROBABILISTIC ANALYSIS, SRF=1.4 AND PROBABILITY OF SRF<=1.4: 0.69%



Source: RS2 Software

Nevertheless, the following inherent risks remain, and if not addressed with appropriate mitigating controls could result in unplanned high-risk events:

- Planner, Wedge or Toppling failure;
- Ground water seepage;
- Friable ground / loose material on crest or slope face;
- Water ponding on slope crest
- Blast damaged rock (back-break and undercutting);
- Excessive bench heights; and
- Incorrect bench slope angles.

The following mitigating measures are recommended:

- Implement effective ground control by leveraging off and carefully managing the four basic disciplines in an open pit mine; geology, planning, geotechnical and production;
- Drilling of horizontal drain holes where necessary to reduce highwall pore water pressure and enhance slope stability;
- Implementing a proper water management system that also controls water run-off;
- Smooth blasting techniques to minimise blast damage to final walls;
- Regular inspection of highwall, scaling and clean up of benches and slope faces to reduce the rockfall hazard; and

- Ongoing monitoring and geotechnical mapping to detect the slope instability, deformation, and structures which could lead to failure.

16.3. Mine Planning, Design, Sequencing and Scheduling

Mine planning was based on the 7 November, 2020 Mineral Resource model. The model used a parent block size of 5 m x 5 m x 5 m and sub-blocks as small as 1 m x 1 m x 1 m to model mineralised structures in detail for a more accurate Mineral Resource estimate.

This block model was re-blocked to the parent block size of 5 m by 5 m by 5 m for the pit optimisation and subsequent final open pit design, sequencing and scheduling. By weight averaging the component blocks and sub-blocks into a consistent block size, the approach of excavating in 5 m lifts is more appropriately simulated. This also facilitates mining selectivity and a lower mining dilution which will be possible through better grade control practices on the 5 m loading benches.

The mine design was guided by the results of a pit optimisation exercise. Suitable software which uses a Lerchs-Grossman an algorithm to generate a series of nested pit shells, was used to identify an optimal pit and a starting location for each open pit (i.e., the lowest cost, near-surface ore). The algorithm calculates the economic value of each block in the model based on a series of technical and economic parameters (Table 16-2).

Table 16-2 Pit Optimisation Economic and Technical Parameter

Parameter	Unit	Amount	Comments
Economic Parameters/Price			
Metal Price - Gold	USD/tr oz	1,450	See Section 19
Metal Price - Silver	USD/tr oz	17.50	See Section 19
Transport and Refining Cost	USD/tr oz	9.82	See Section 19
Royalty	%	12%	See Section 22
Discount Rate	%	5%	See Section 22
Processing			
Production Rate	tpa	4,927,500	Notes 1 and 2.
Plant Recovery - Gold	%	73.6% over the LoM	Defined in geological block model 4 (Note 3)
Plant Recovery - Silver	%	63.4%	See Section 13 and Note 3
Processing Cost	USD/t ore	4.23	See Section 21
Stacking Cost	USD/t ore	0.59	See Section 21
Owner's Cost	USD/t ore	0.29	See Section 21
G&A Cost	USD/t ore	1.27	Updated figure from Chaarat
Mining Cost			
Mining Cost	USD/t mined	1.83	Mining costs are based on the Contract Mining Agreement and the mine plan
Extra Ore Mining	USD/t ore	0.72	Extra cost from overhaul and additional fuel for Ore
Pit Geometry			
Inter-Ramp Angle	°	Variable by location	See Section 16.2

Parameter	Unit	Amount	Comments
Bench Height	m	20m pit design 5m production fliches	See Section 16.2
Density			
Average	t/m3	2.64	Defined in geological block model
Ore	t/m^3	Variable from 2.10 to 3.18	Defined in geological block model

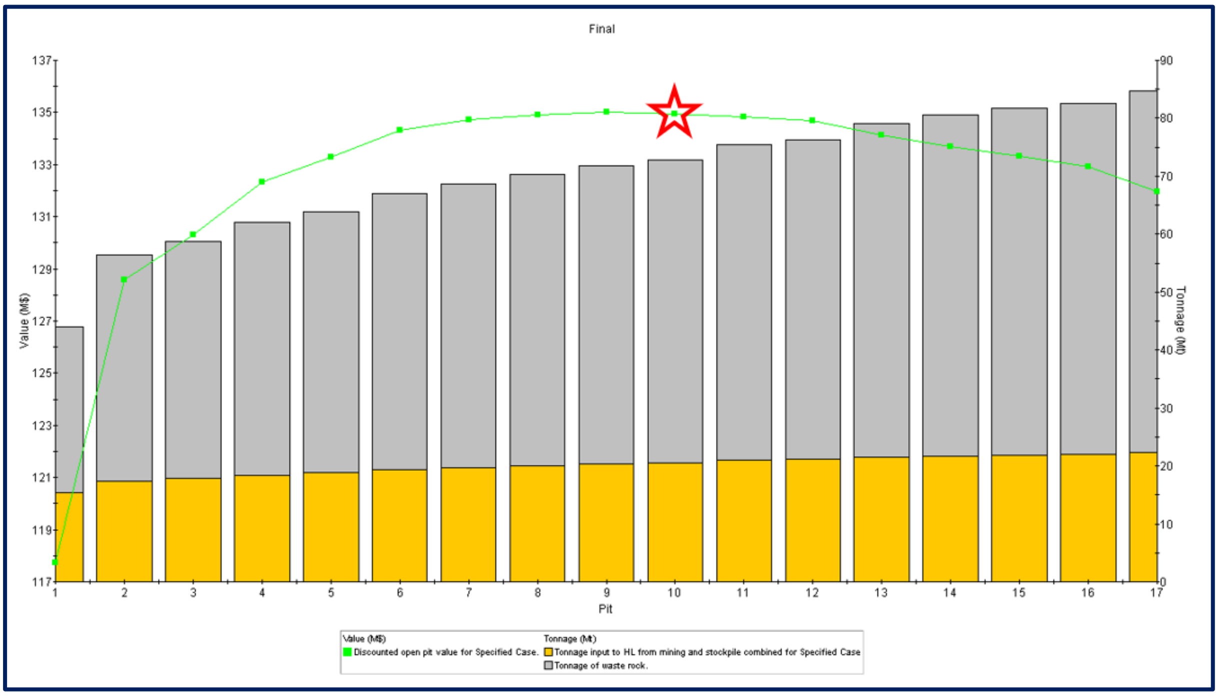
Source: Chaarat, 2021

Notes:

1. *The in-situ density is obtained from the block model and varies between 2.10 t/m^3 and 3.18 t/m^3, depending on the rock type and oxidation state. The average density is approximately 2.64 t/m^3.*
2. *Gold and Silver recovery both based on results of metallurgical test work. Au recovery was estimated geostatistically while Ag recovery was the average from the 2018 test programme.*

The selected final pit shell encompasses the set of blocks which have the highest relative net present value over the LoM given the constraints applied. Graph 16-1 highlights Pit No. 10 as the preferred final pit upon which the final open pit design has been based.

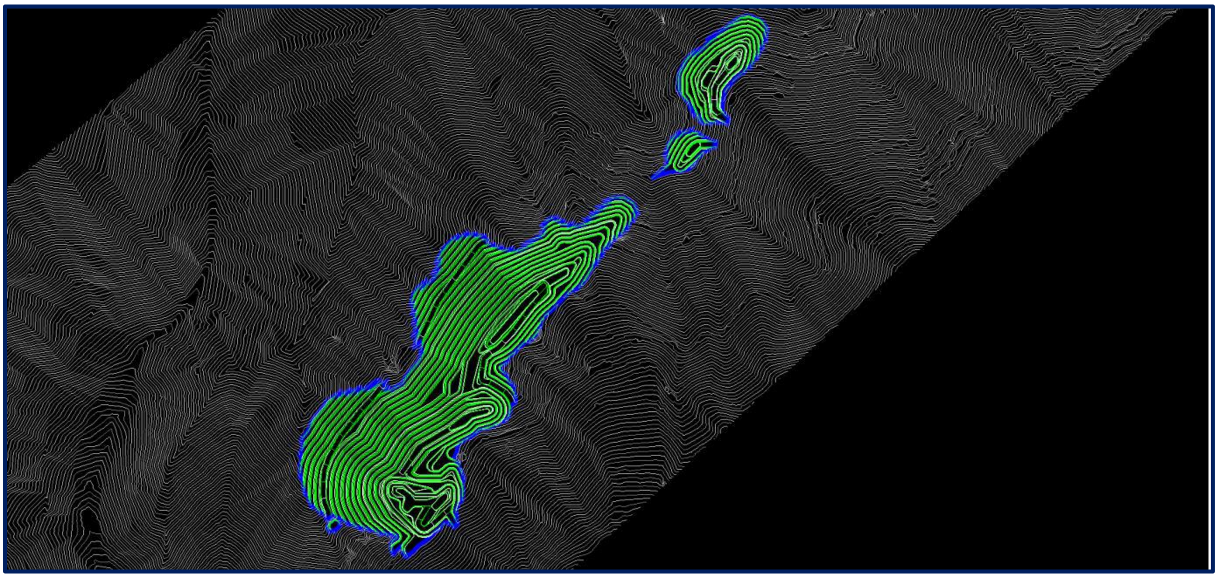
GRAPH 16-1 PIT OPTIMISATION RESULTS



Source: Chaarat, 2021

The pit optimisation shell results presented in Graph 16-1 is used as the basis for the pit design process with which the pit design criteria, together with the adjustments for dilution and mining losses enable the ore reserve to be declared. The final pit designs are illustrated in Figure 16-5.

FIGURE 16-5 DESIGNED PITS



Source: Chaarat, 2021

The Tulkubash 2020 EOY open pit design is composed of three separate pits arranged along the strike of the orebody over 2 km. The pits are situated in steep, mountainous terrain at elevations of 2,300 masl to 2,800 masl. The deposit is divided up into two zones, the Main Zone and the Mid Zone. The following provides a brief description of the pits in each zone:

The Main Zone Pit is situated at the southwestern end of the mining area. It is the single largest pit accounting for over 90% of the reserve by both tonnage and contained gold. The Main Zone Pit hosts a reserve of 19.4 Mt ore grading 0.86 g/t Au, containing 538 Koz Au. Associated with the ore is 50.4 Mt of waste resulting in a strip ratio of 2.6:1 (t:t).

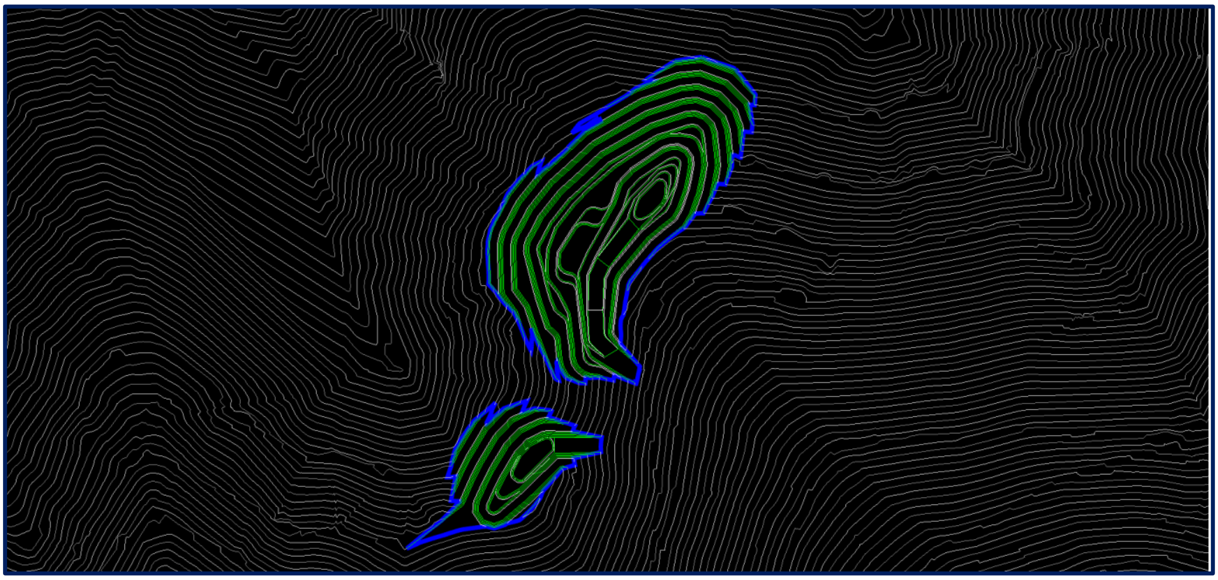
The Main Zone Pit is approximately 1.3 km in length. The width of the pit varies from 530 m at the south end and 370 m in the central portion before narrowing to 130 m at the northeast end. The crest of the final pit lies at an elevation of 2,740 masl while the elevation of the final pit bottom is 2,365 masl resulting in a maximum vertical extent of 375 m at the south end. Overall, the final highwall ranges between 250 m to 300 m in height.

The Main Zone Pit exhibits a single pit bottom at the south end of the pit and two other lenticular bottom benches arranged along strike as the pit moves to the northeast. Most of the benches in the pit intersect surface contours except for the bottom 40 m to 50 m. The Main Zone Pit design can be seen in Figure 16-5.

The Mid Zone Pit design is composed of two separate small open pits. These pits are arranged along strike length about 150 m northeast of the Main Zone Pit. The Mid Zone accounts for approximately 7% of the reserve by tonnage and 6% of the contained gold. The Mid Zone Pits host a reserve of 1.4 Mt ore grading 0.72 g/t Au, containing 33 Koz Au. Associated with the ore is 3.7 Mt of waste resulting in a strip ratio of 2.6:1 (t:t).

The first Mid Zone pit is roughly circular with a diameter of about 150 m and a depth of 120 m. The second Mid Zone pit, located immediately to the northeast is bigger, being about 350 m in length, 150 m wide, and 150 m deep. Although small and lower grade than the Main Zone, the Mid Zone Pits offer the highest metallurgical recovery in the reserve, over 76%. The Mid Zone Pit designs can be seen in Figure 16-6.

FIGURE 16-6 MID ZONE PIT DESIGNS



Source: Chaarat, 2021

16.3.1. MINE DEVELOPMENT

The deposit will be developed and mined using conventional hard rock open pit mining techniques. Mine development will entail establishing large enough working areas in the open pit to enable safe and efficient mining operations at production rates that are high enough to support a steady state supply of ore for processing.

All vegetation and organic material will be cleared and deposited in designated stockpiles (SP) to be used in the future for rehabilitation and mine closure. Topsoil will similarly be stripped and stockpiled separately and will be used to rehabilitate the area once mining is finished.

The existing roads can be used to move equipment up the hillside from where additional new roads will be constructed to access the top benches of the pit. These roads will be widened where necessary to accommodate the mining equipment. Once access to the initial bench elevation is established, dozers will level a large enough area to allow blasthole drilling. This initial platform will then be drilled and blasted, dozed down, and the process repeated until a bench wide enough (15 m) to accommodate single-side truck loading is established. Steady state production benches will be at least 25 m wide. The drilling and blasting of 5 m benches will commence and waste rock will be used to widen haul roads pioneered to the waste dumping areas.

Once the initial working areas are sufficiently developed to support steady state production, mine development work will progress along strike. It is in this manner that the open pits will be developed along strike across the hillside with access development, bench development, and steady state mining following each other in a continuous sequence.

Haul roads connecting the open pit area to the Sandalash River Bridge and the waste dump will be constructed during pre-production. The cost of these roads has been designated as part of the project capital expenditure.

Table 16-3 shows the LoM mine production schedule. The mining plan calls for 4.6 years of production mining preceded by 13 months of pre-production stripping, a total of 68 months. Total mined tonnage over the LoM, including pre-stripping, is 74.9 Mt with an average mining rate of 13.0 Mtpa or about 37,000 tpd. The mining rate peaks in 2025 at 18.5 Mtpa or about 53,000 tpd.

During the 13-month pre-production period, 7.4 Mt of material is mined including 600 kt of ore. 409 Kt of this ore is sent to the HLF, including the material for the overliner. The remaining approximately 185 Mt of ore is stockpiled for processing later in the LoM.

TABLE 16-3 SUMMARY OF LOM PRODUCTION SCHEDULE

Description		Units	Year							Total
			2022	2023	2024	2025	2026	2027	2028	
Mining										
Ore Mined		kt	38	1,560	3,933	4,469	5,371	4,844	644	20,859
Au Grade		g/t	0.5	0.91	0.70	1.04	0.67	0.97	0.94	0.85
Ag Grade		g/t	0.52	1.00	0.99	1.40	1.32	1.39	1.14	1.26
Waste Mined		kt	658	11,307	14,292	14,060	10,653	2,860	217	54,048
Total Rock Mined		kt	696	12,868	18,225	18,529	16,024	7,704	861	74,907
Strip Ratio		t:t	17.33	7.25	3.63	3.15	1.98	0.59	0.34	2.59
Processing										
Process Feed		kt		1,138	3,893	4,920	4,920	4,920	1,068	20,859
Au	Grade	g/t		1.11	0.71	0.98	0.69	0.96	0.73	0.85
	Contained Metal	koz		41	88	154	110	153	25	571
	Recovery	%		75.80%	75.90%	74.40%	74.20%	71.20%	68.50%	73.60%
	Recovered Metal	koz		31	67	115	82	109	17	420
Ag	Grade	g/t		0.99	1.00	1.36	1.31	1.39	1.26	1.26
	Contained Metal	koz		36	125	215	207	220	43	846
	Recovery	%		63.40%	63.40%	63.40%	63.40%	63.40%	63.40%	63.40%
	Recovered Metal	koz		23	79	136	131	139	27	536

Source: Chaarat, 2021

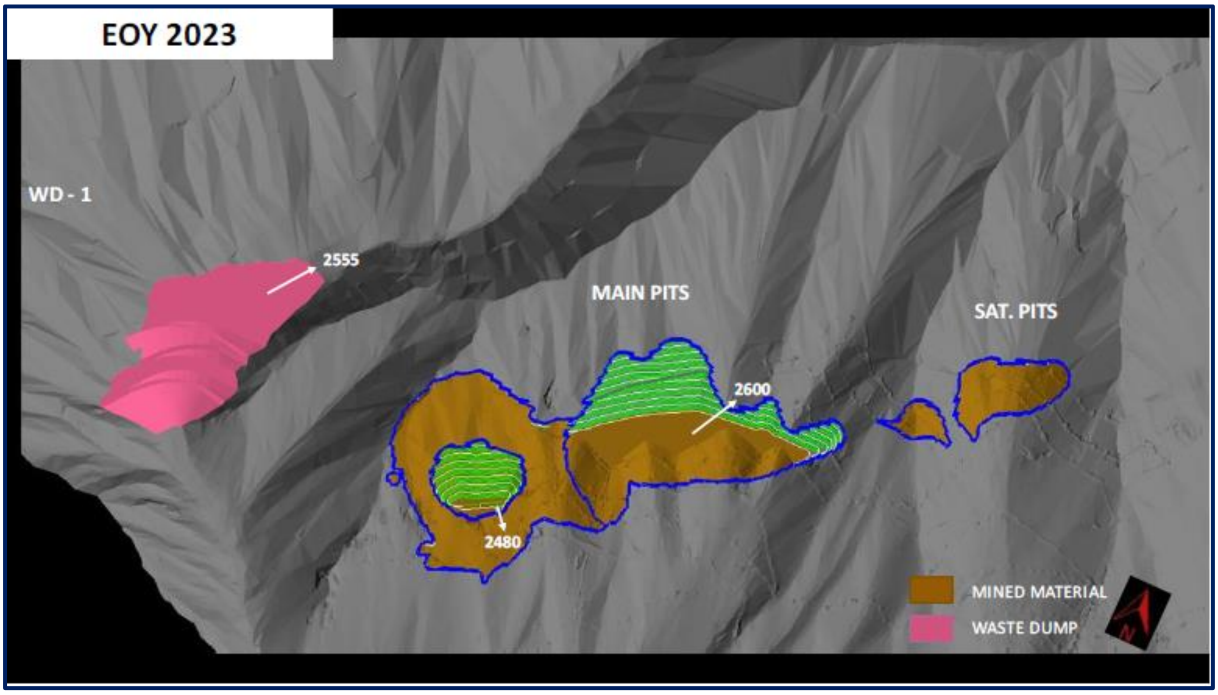
The ore process rate at full production is 4.9 Mtpa. In years where mining exceeds this figure, ore is stockpiled. In years where ore mined is less than the process rate, ore is reclaimed from the stockpile. Stockpiling and reclaiming ore allow the schedule to manage annual variations in ore and waste mining from the open pits.

Figure 16-7 to Figure 16-13 illustrate how mining progresses over the LoM.

FIGURE 16-7 SCHEMATIC 3D VIEW OF YEAR END OF 2022 (YEAR 1)

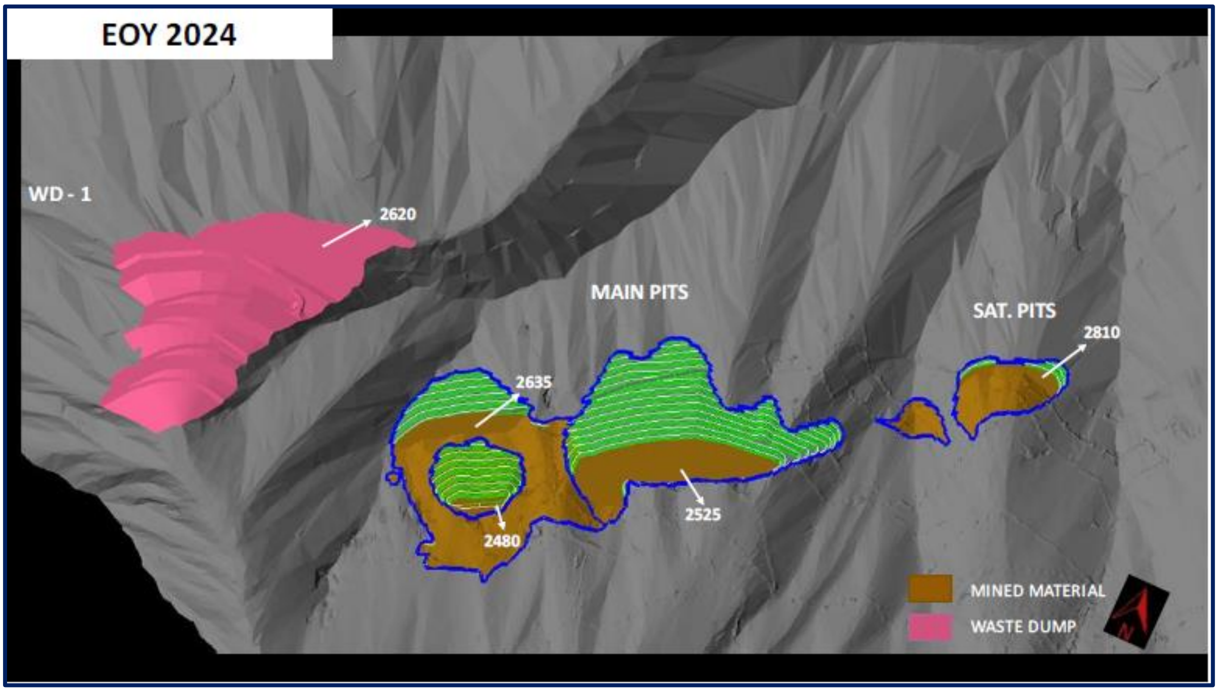
Source: Chaarat, 2020

FIGURE 16-8 SCHEMATIC 3D VIEW OF YEAR END OF 2023 (YEAR 2)



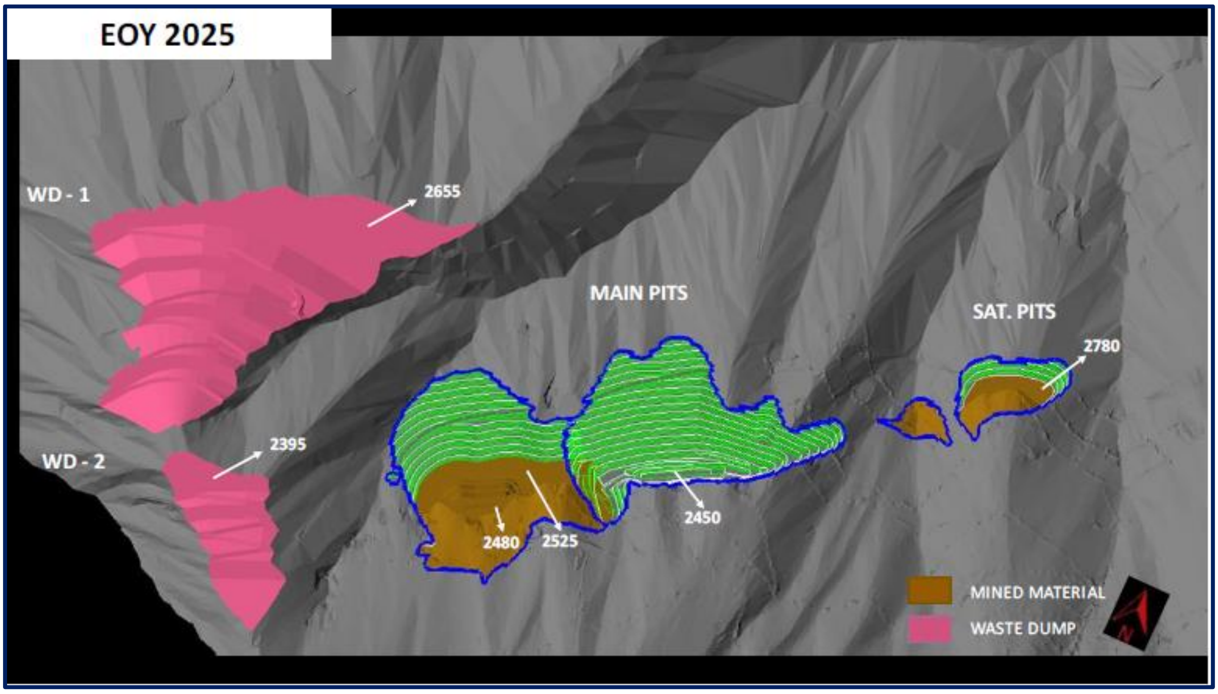
Source: Chaarat, 2020

Figure 16-9 Schematic 3D View of Year End of 2024 (Year 3)



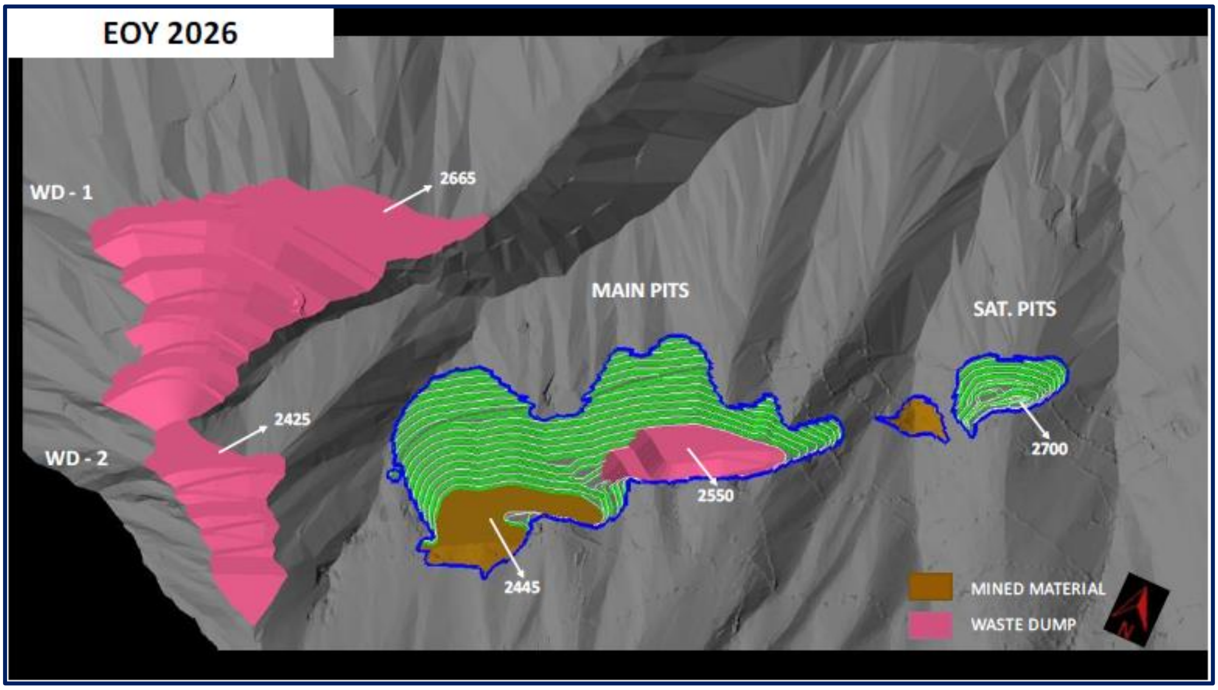
Source: Chaarat, 2020

Figure 16-10 Schematic 3D View of Year End of 2025 (Year 4)



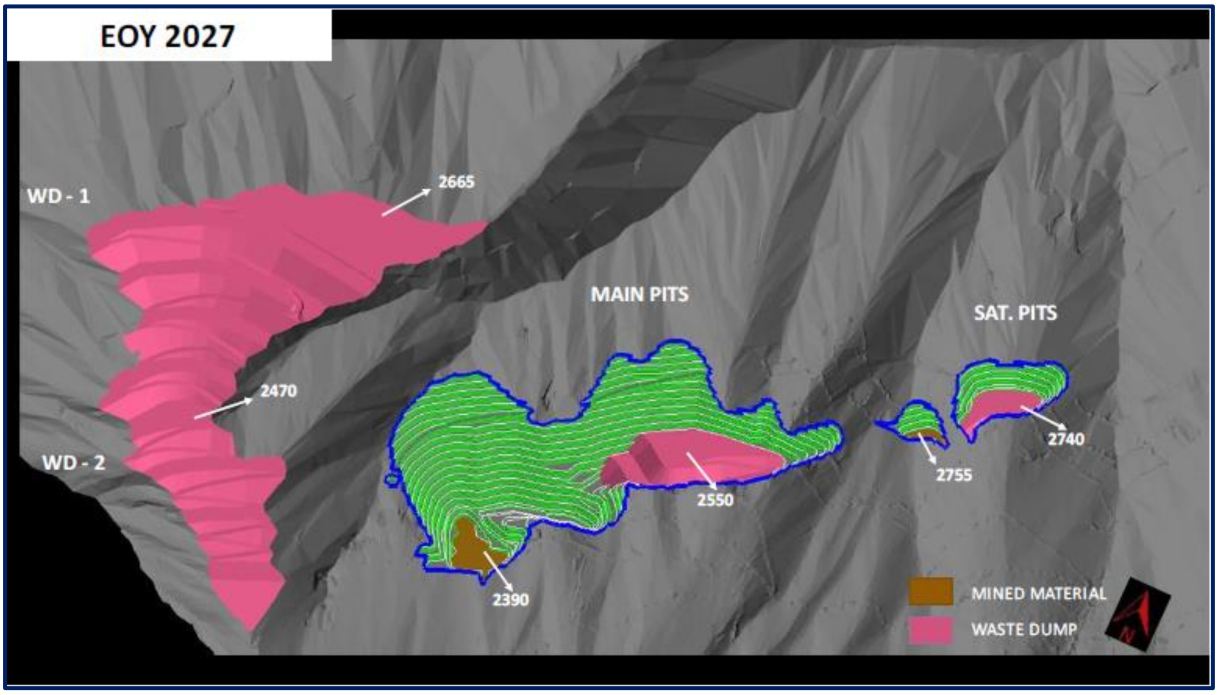
Source: Chaarat, 2020

FIGURE 16-11 SCHEMATIC 3D VIEW OF YEAR END OF 2026 (YEAR 5)



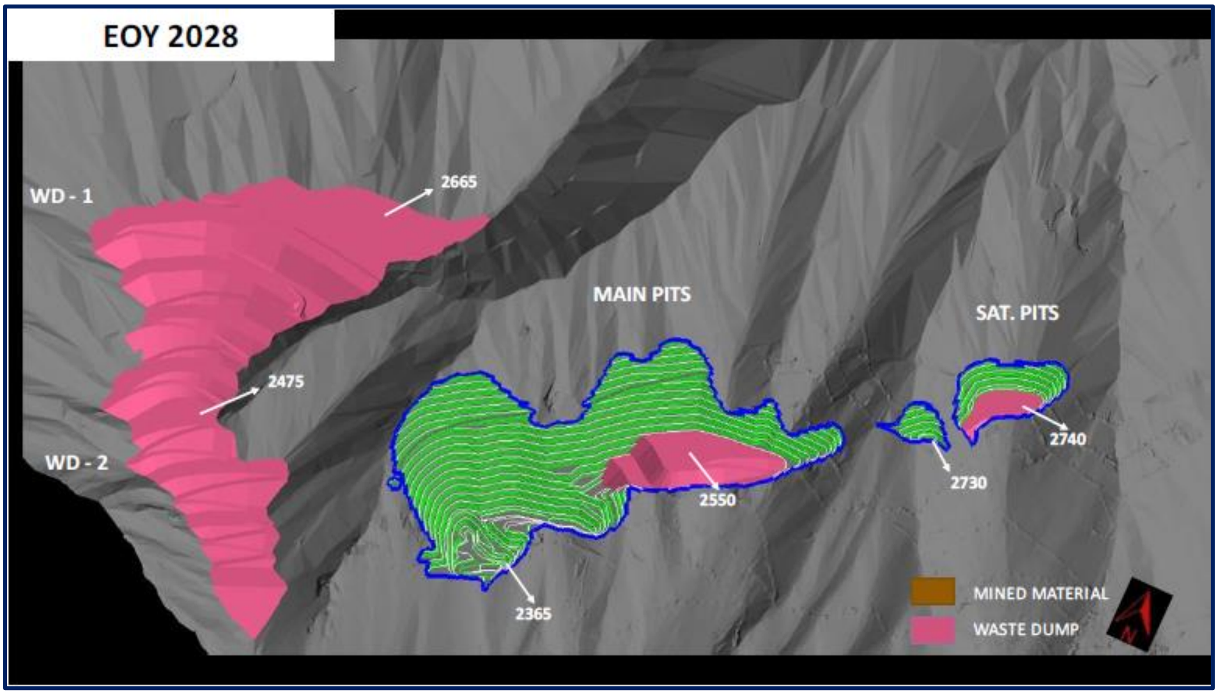
Source: Chaarat, 2020

FIGURE 16-12 SCHEMATIC 3D VIEW OF YEAR END OF 2027 (YEAR 6)



Source: Chaarat, 2020

Figure 16-13 Schematic 3D View of Year End of 2028 (Year 7)



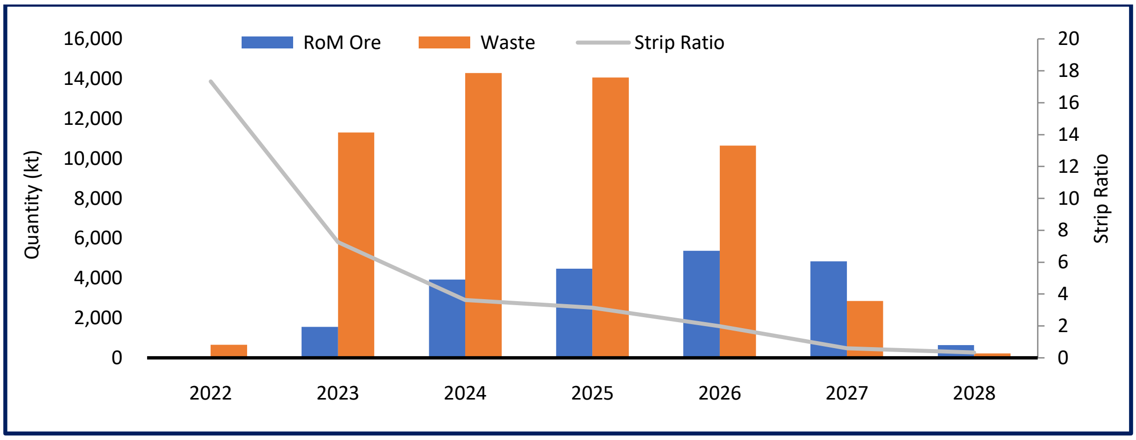
Source: *Chaarat, 2020*

16.3.2. Waste Mining and Waste Rock Dump

An average of 2.59 tonnes of waste will be removed and stored on a waste dump for every tonne of ore mined over the life of mine. The strip ratio during pre-production is 11.3:1 as an average. The maximum strip ratio is 17.3:1 in 2022, during the first 5 months of pre-stripping

Graph 16-2 illustrates the LoM production schedule aligned with the strip ratio.

Graph 16-2 LoM Production Schedule and Strip Ratio



Source: *Chaarat, 2021*

Waste rock will be stored on a Waste Rock Dump (WRD) in the adjacent Irisay Valley west-southwest of the mine area and used to backfill a portion of the mined-out pits. Table 16-4 shows the waste dumping schedule for the LoM and Figure 16-14 shows the waste dumps in their final configuration.

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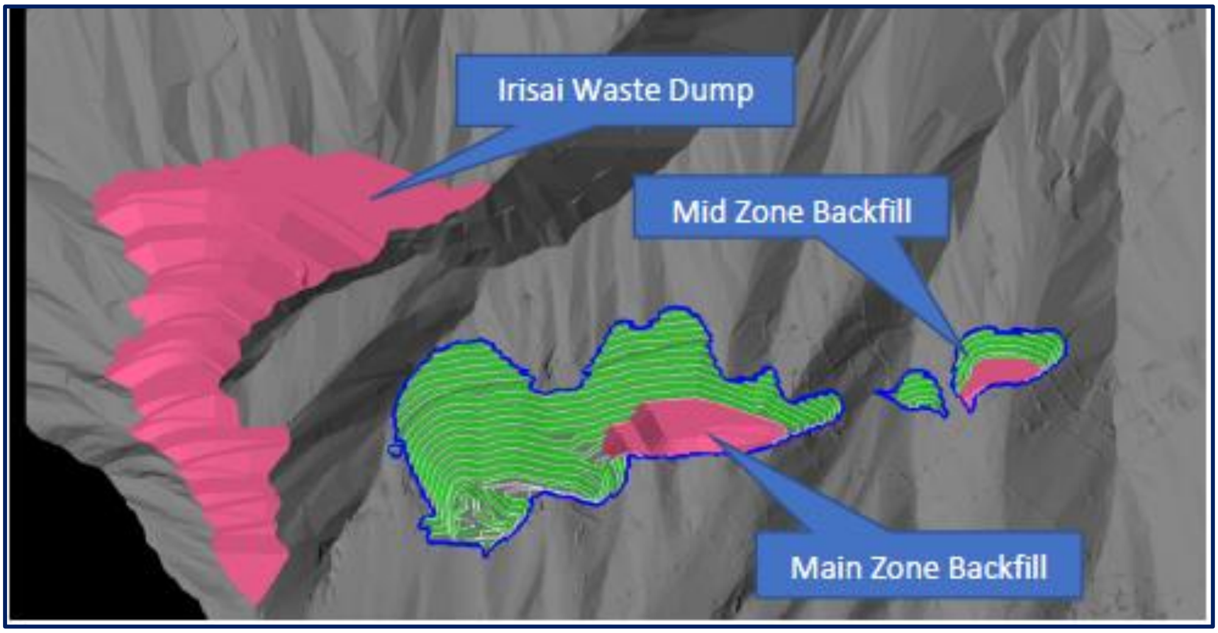
TABLE 16-4 WASTE TONNAGE SCHEDULE BY YEAR AND LEVEL

Location	Unit	Year								Total	Capacity	Remaining
		2021	2022	2023	2024	2025	2026	2027	2028			
Waste Dump Phase 1	LCM		346,452	5,951,151	7,522,322	4,721,175	898,516			19,439,616	42,860,908	23,421,291
	Level (masl)		2,455	2,555	2,620	2,655	2,665					
Waste Dump Phase 2	LCM					2,678,951	1,028,940	1,186,713	62,036	4,956,640	5,129,407	172,767
	Level (masl)					2,395	2,425	2,470	2,475			
Main Pit In-pit Dump	LCM						3,679,471			3,679,471	3,679,471	0
	Level (masl)						2,550					
Satellite Pit In-pit Dump	LCM							318,428	52,252	370,681	642,620	271,940
	Level (masl)							2,740	2,740			
Total Waste Dumped	LCM	0	346,452	5,951,151	7,522,322	7,400,127	5,606,927	1,505,141	114,289	28,446,408	52,312,406	23,865,998

Source: Chaarat, 2021

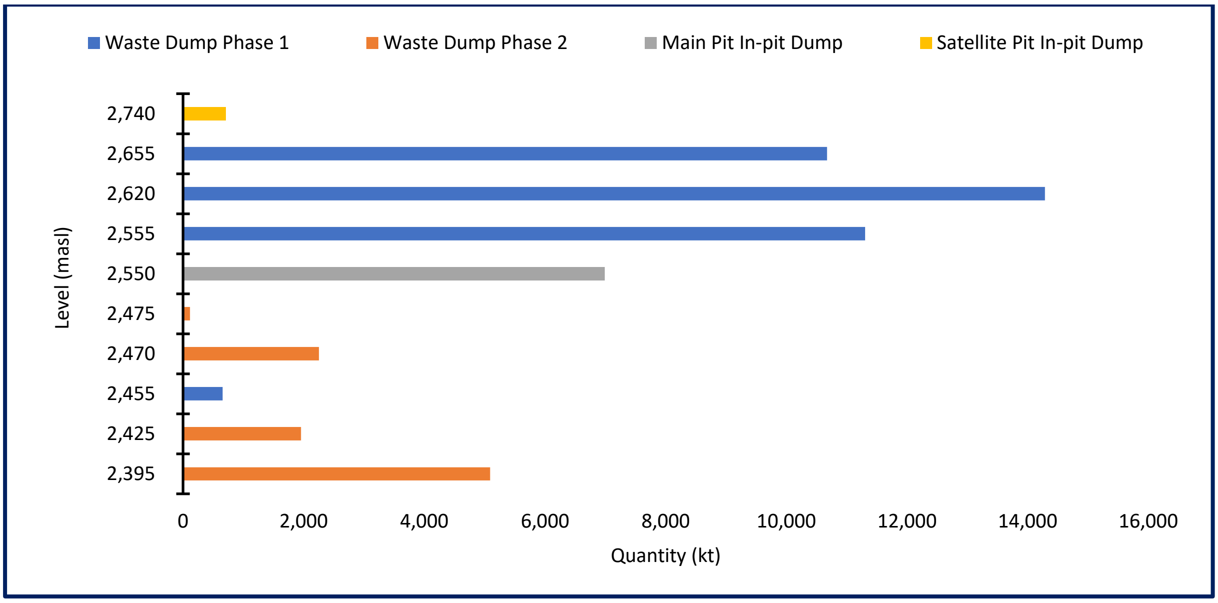
Figure 16-14 Final Waste Dump Configuration



Source: Chaarat, 2020

The WRD will be built from the bottom up, in roughly 50 m benches. Dumping will occur on more than one bench at a time, separated by sufficient room for safe operations, and snow will not be dumped with waste rock. Graph 16-3 shows the waste tonnage by the location and the level of waste dump.

Graph 16-3 Waste Tonnage by Location and Level



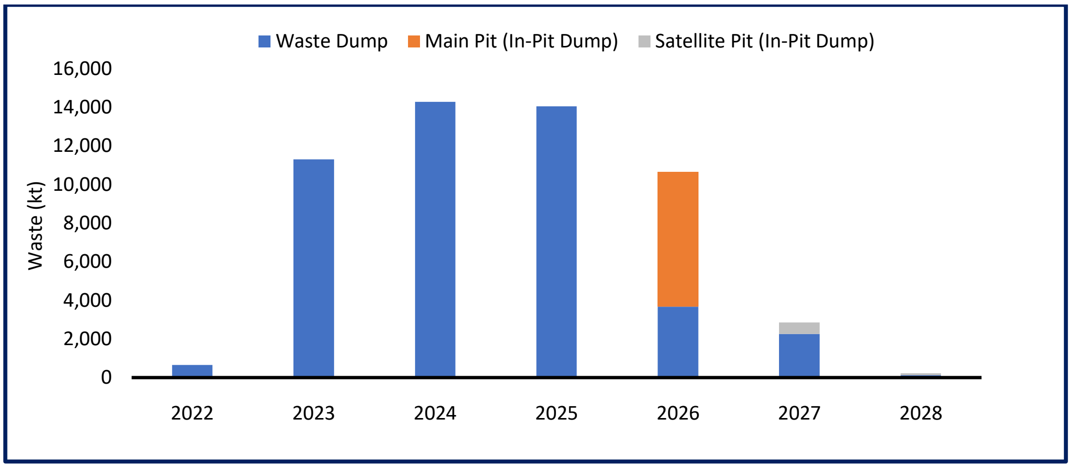
Source: Chaarat, 2021

The waste rock will be dumped at angle of repose. Wide flat structures will be maintained on the WRD so the overall angle does not exceed 22°. Surface drainage will be intercepted up canyon and maintained in drainage ditches above the WRD surface. A settling pond located below the WRD will collect surface run-off and seepage.

Waste dumps will be constructed with berms of a minimum height equal to half the height of the tire of the trucks dumping there. Dumps will be manned with a dozer to ensure that berms are maintained properly. Dump platforms will be sloped upwards slightly within 3 m of the berm. No dumping will be permitted over an open edge. If the berm is absent, trucks will stop short of the edge and dump on top.

Graph 16-4 illustrates the WRD schedule during the LoM.

GRAPH 16-4 WASTE DUMP SCHEDULE



Source: Chaarat, 2021

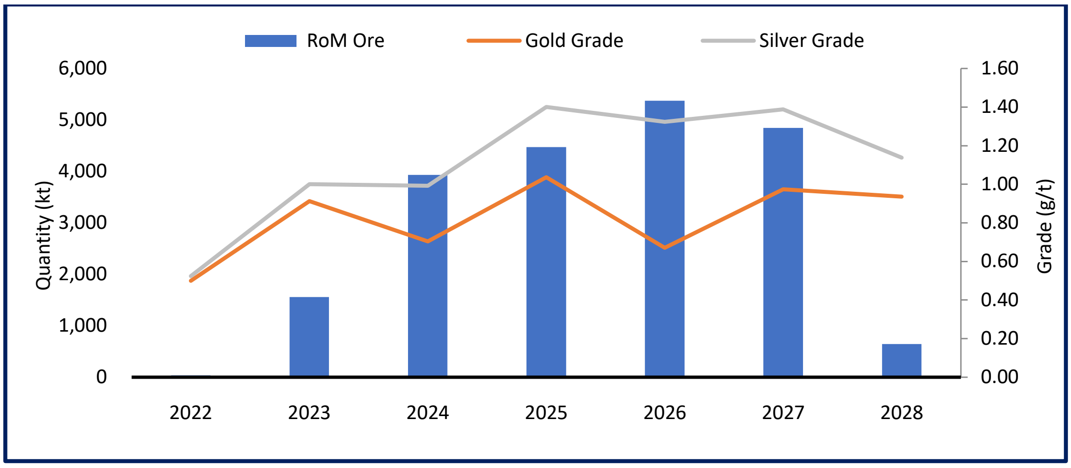
Dumps will be monitored for settlement. Areas subject to excessive settlement will be closed and given appropriate danger signage until monitoring determines it is safe to work there. Trucks will approach the berm from left to right so the drivers can see if the edge is cracking. No equipment shall be parked in the "caving prism", the area subject to sloughing near the crest of the dump.

In-pit dumping occurs from 2026 - 2028. Backfilling the pits represents good industry practice and reduces costs.

16.3.3. ORE MINING AND ORE STOCKPILE

The average metal grades in the 20.9 Mt of ore mined over the LoM are 0.85 g/t for gold and 1.26 g/t for silver (Graph 16-5).

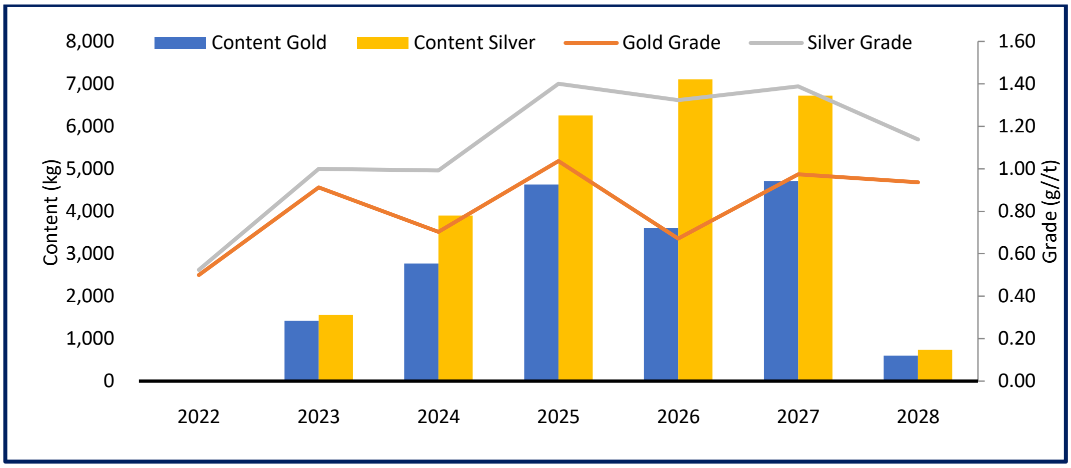
Graph 16-5 ROM Ore Mined and Grades



Source: *Chaarat, 2021*

This provides for contained metal of 571.1 koz of gold and 845.7 koz of silver respectively (Graph 16-6).

Graph 16-6 ROM Content and Grades



Source: *Chaarat, 2021*

A portion of the initial 600kt ore generated is used as overliner for the heap leach pad. Overliner is the material used to bury the distribution piping on top of the liner, which is the material that the ore is dumped on.

The rate of mining increases steadily once production starts. In the last four months of 2023 approximately 730kt of ore are mined. Ramp-up of ore mining and processing continues until mid-2024. At steady-state, annual ore production is 13.5 ktd or 4.92 Mtpa.

Table 16-5 shows the details of ore tonnage and grade mined over the LoM.

TABLE 16-5 LoM ORE MINING PRODUCTION SCHEDULE

Description	Units	Year							Total
		2022	2023	2024	2025	2026	2027	2028	
Mining									
Ore Mined	kt	38	1,560	3,933	4,469	5,371	4,844	644	20,859
Au Grade	g/t	0.5	0.91	0.70	1.04	0.67	0.97	0.94	0.85
Ag Grade	g/t	0.52	1.00	0.99	1.40	1.32	1.39	1.14	1.26
Waste Mined	kt	658	11,307	14,292	14,060	10,653	2,860	217	54,048
Total Rock Mined	kt	696	12,868	18,225	18,529	16,024	7,704	861	74,907
Strip Ratio	t:t	17.33	7.25	3.63	3.15	1.98	0.59	0.34	2.59

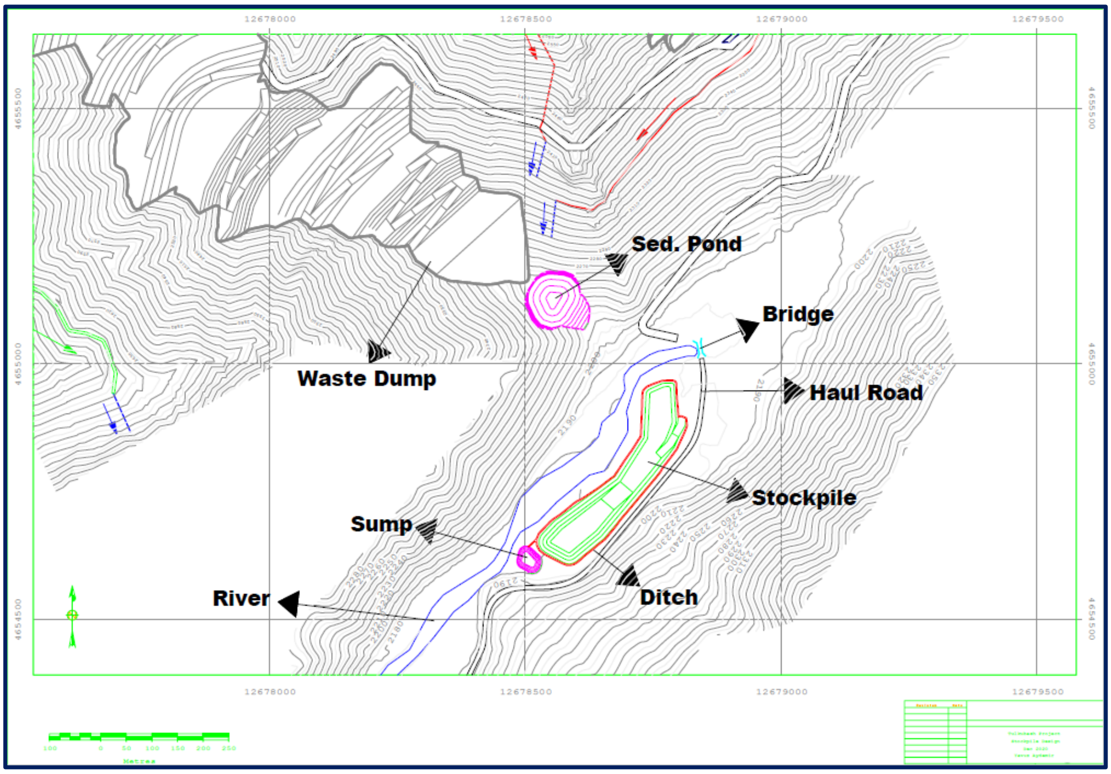
Source: Chaarat 2021

A small SP facility (~500 kt) has been allowed for at the site of the current Summer Camp.

The planned 500 kt SP represents about five weeks of processing and is < 1% of all material to be mined. It is noted that the SP will be fully depleted at the end of the LoM.

Figure 16-15 shows a schematic view of the designed ore stockpile.

FIGURE 16-15 SCHEMATIC VIEW OF DESIGNED STOCKPILE



Source: Chaarat, 2021

16.4. MINE OPERATION

This section summarises the open pit mining operation and the type of equipment to be used by the contract mining company (Pamir Mining, which is 100% owned by Çiftay).

16.4.1. Working Arrangements

The open pit mining operation will operate continuously for 350 days per year, with ten days lost due to bad weather or supply-related issues. The mining crews will work 12-hour shifts on a 15-day rotation (15 days on, and 15 days off), rotating cycle to facilitate a continuous operation.

An operating efficiency of 83% is planned with 8.3 hours of productive work per shift or 16.6 hour per day. One hour will be allowed for lunch with a second hour allowed for the shift change, safety meetings, blasting delays, and other planned delays.

Standard equipment hours used to calculate productivities and costs are 5,390 Gross Operating Hr (GOH) per year. This operating time includes both productive work and delays. Theoretical equipment productivities are based on 4,474 Net Operating Hr (NOH) per year. Table 16-6 shows the mine operating hours.

Table 16-6 Operating Hr for the Mining Operation

Description	Hr Per Year	Comment
Scheduled Working Hr	8760	350 days per year
Equipment Down Hr	1,260	85% mechanical availability
Equipment Available Hr	7,140	
Idle	350	One Hr per Day
Lunch	700	One Hr per Shift
Shift Change	700	One Hr per Shift
Gross Operating Hr (GOH)	5,390	75% Utilisation
Delay Hr	916	83% Efficiency
Net Operating Hr (NOH)	4,474	

Source: Chaarat, 2021

16.4.2. Drilling and Blasting

All of the material to be mined from the open pits will require blasting prior to loading. Surface crawler-type drill rigs (i.e. Atlas Copco T35 or Sandvik D65) will drill 5 m benches at a penetration rate of 27 m/NOH. The design is for 127 mm blastholes to be drilled with a 10% subgrade on a 3.5 m burden and 4.2 m spacing for the normal production patterns. Similary,102 mm blastholes will be drilled on a 3.2 m burden and a 3.5 m spacing with an average subgrade of 5% for controlled blasting of the final wall. The plan includes a provision of 10% for re-drills. Table 16-7 shows the assumptions behind the drill and blast calculations.

Table 16-7 Drill and Blast Planning Assumption

Drilling Design					
Drilling Type	**Burden (m)**	**Spacing (m)**	**Bench Height (m)**	**Volume (bcm/hole)**	**Density (t/m^3)**
Production	3.5	4.2	5.0	73.5	2.64
Wall Control	3.2	3.5	5.0	56.0	2.64
Drilling Type	**Tonnage (t/BH)**	**Bench height (m)**	**Subgrade (%)**	**Subgrade (m)**	**BH Depth (m)**

Drilling Design					
Production	194	5.0	10%	0.5	5.5
Wall Control	148	5.0	5%	0.25	5.25
Type	**Rate (m/NOH)**	**Efficiency (%)**	**Rate (m/GOH)**	**Production Rate (BH/GOH)**	**Production Rate (t/GOH)**
Production	27.0	83%	22.4	3.7	719
Wall Control	27.0	83%	22.4	3.9	575
Blast Design					
Blasting Type	**Hole Diameter (in)**	**Hole Diameter (mm)**	**Nominal (mm)**	**Volume (m^3/m)**	**Explosive**
Production	3.5	127.0	127.0	0.0127	ANFO and Emulsion
Wall Control	3.2	102.0	102.0	0.0082	
Blasting Type	**Explosive Density (t/m^3)**	**Explosive Load (kg/m)**	**Column Height (m)**	**Material Volume (bcm/hole)**	**Powder Factor (kg/bcm)**
Production	0.85	10.77	3.3	73.5	0.48
Wall Control	1.20	9.81	1.3	56.0	0.23

Source: Chaarat, 2021

Steady state operations will require that approximately between 510,000 m and 590,000 m be drilled annually to generate approximately between 95,000 and 105,000 blastholes. A maximum of five drill rigs will be required to achieve these production targets. Details of the drilling plan is shown in Table 16-8, which indicates that the steady state total consumption of explosive will be in the order of 3.2 ktpa. The blasting will be accomplished with shock-tubes (i.e., non-electric detonation) and ANFO. A powder factor of 0.48 kg/bcm will apply for the usual production patterns while a reduced powder factor of 0.23 kg/bcm will be loaded for controlled blasting to minimise the impact on final walls.

TABLE 16-8 DRILL AND BLAST PLANNING

Description	Unit	2022	2023	2024	2025	2026	2027	2028
Production D-B Summary								
Total	Mt	0.70	12.87	18.23	18.53	16.02	7.70	0.86
Production Drilling								
Tonnage	Mt	0.63	11.58	16.41	16.68	14.42	6.93	0.77
Drilling	m	19,643	361,054	511,650	520,068	449,603	216,071	24,008
	BHs	3,571	65,646	93,027	94,558	81,746	39,286	4,365
Hours	GOH	877	16,111	22,831	23,207	20,063	9,642	1,071
Units	Calculation	0.5	3.0	4.2	4.3	3.7	1.8	0.8
	Required	1.0	3.0	5.0	5.0	4.0	2.0	1.0
Production Blasting								
Explosive	t	318	2,121	3,004	3,054	2,640	1,269	142
Wall Control D-B Summary								
Total	Mt	0.70	12.87	18.23	18.53	16.02	7.70	0.86
Wall Control Drilling								
Tonnage	Mt	0.07	1.29	1.82	1.85	1.60	0.77	0.09
Drilling	M	2,734	50,391	71,094	72,266	62,500	30,078	3,516
	BHs	521	9,598	13,542	13,765	11,905	5,729	670

Description	Unit	2022	2023	2024	2025	2026	2027	2028
Hours	GOH	122	2,249	3,172	3,225	2,789	1,342	157
Units	Calculation	0.1	0.4	0.6	0.6	.05	0.2	0.1
	Required	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Wall Control D-B Summary								
Wall Control Blasting								
Explosive	t	6	109	154	156	135	65	7
Total Drilling								
Tonnage	Mt	0.7	12.9	18.2	18.5	16.0	7.7	0.9
Drilling	M	22,377	411,445	582,744	592,334	512,103	246,149	27,524
	BHs	4,092	75,244	106,569	108,323	93,651	45,015	5,035
Hours	GOH	999	18,360	26,003	26,432	22,852	10,984	1,228
Units	Calculation	0.2	3.4	4.8	4.9	4.2	2.0	0.9
	Required	1	4	5	5	5	3	1
Total Explosive								
Explosive	T	324	2,230	3,158	3,210	2,775	1,334	149

Source: Chaarat, 2021

The blastholes will be double-primed with solid, cast PETN boosters. Each booster will be detonated with a 500 millisecond DTH delay. They will be initiated individually using shock-tube lead-in line from a safe distance with 17 milliseconds between holes. Provision has been included in the planning for 5% of all blastholes to be loaded with waterproof emulsion explosive. Wet blastholes that can be pumped will be lined with 127 mm plastic sleeves and loaded with ANFO.

16.4.3. Loading

Digging and loading will occur on 5 m lifts to match the height of the working face to the size of the equipment and to facilitate digging selectivity when separating ore and waste.

CAT 374 hydraulic excavators (Figure 16-16) with 5.0 m^3 buckets will load 34.5 t trucks in 2.5 minutes. A fully trucked excavator will load approximately 3.71 Mtpa. Smaller excavators with hydraulic rock breakers will be used to clean walls and break oversize rock at the face to maximise excavator loading productivity. CAT 980 front-end loaders or similar units, will be used to support the primary excavators for truck loading, clean-up, and snow removal.

FIGURE 16-16 CAT 374 EXCAVATOR LOADING MERCEDES ACTROS 3340 TRUCKS

Source: Chaarat, 2020

This mix of front-end loaders and excavators facilitates mobility and provides more flexibility during steady state mining operations. Table 16-9 shows the production calculation for the primary loading units.

TABLE 16-9 PRODUCTION CALCULATION FOR CAT 374 EXCAVATOR

Shovel:	**CAT 374F**	**Backhoe**				
Truck:	**Mercedes**	**3340**	**(30t)**			
In-situ Density (t/m^3)	**Swell Factor (ICM/BCM)**	**Loose Density (t/m^3)**	**Bucket Size (m^3)**	**Bucket Fill (%)**	**Bucket Capacity**	
					Volume (ICM)	**Quantity (t)**
2.64	1.4	1.89	5.2	88%*	4.6	8.6
Loading Passes (No.)	**Total Load**		**Time per Pass (sec)**	**Spot Time (sec)**	**Load Time (sec)**	**Load Time (min)**
	Volume (ICM)	**Quantity (t)**				
4.0	18.3	34.5	30*	30	150	2.5
Theoretical Productivity			**Operating Efficiency %**	**Planned Productivity**		
Number of Loads	**Vol (ICM/NOH)**	**Quantity (t/NOH)**		**Number of Loads**	**Vol (ICM/GOH)**	**Quantity (t/GOH)**
24	439	728	83%	19.9	365	688
Planned Production						
GOH per Year	**Volume (M ICM/year)**	**Quantity (Mt/year)**				
5,390	1.97	3.71				

Source: Chaarat, 2021

** Talpac software associated recommendation*

Earlier work has verified the number of excavator and loader units as originally estimated by Chaarat (Table 16-10). This earlier work used a more conservative 30 second duration per pass and a higher bucket fill factor of 90%, for this estimate.

The results confirm that four to five excavator units will be sufficient to achieve the planned mining schedule, if supported by a single FEL during the mining plan for peak production during 2024 to 2026.

TABLE 16-10 EXCAVATOR CALCULATION

Description	Unit	2022	2023	2024	2025	2026	2027	2027	Total
Production Schedule									
Ore	Mt	38	1,560	3,933	4,469	5,371	4,844	644	20,859
Waste	Mt	358	11,307	14,292	14,060	10,653	2,860	217	54,048
Total	Mt	696	12,868	18,225	18,529	16,024	7,704	861	74,907
Excavator Productivity									
Gross Operating Hours	GOH/year	5,390	5,390	5,390	5,390	5,390	5,390	5,390	n/a
Excavator Production Rate	Mtpa per unit	3.71	3.71	3.71	3.71	3.71	3.71	3.71	n/a
Excavator Quantity									
Excavator Quantity (Calculation)	Quantity	0.45	3.47	4.92	5.00	4.32	2.08	1.39	n/a
Actual Excavators Required	Quantity	1	3	5	5	4	2	1	n/a
Total Fleet Capacity	**Mt**	**3.71**	**11.13**	**18.55**	**18.55**	**14.84**	**7.42**	**3.71**	n/a

Source: Chaarat, 2020

16.4.4. HAULING

Mercedes Actros 3340 dump trucks with a capacity of 34.5 t (Figure 16-17) will be used to transport all blasted material to the WRD, ore stockpile or the ROM Pad, as the case may be. The average hauling distance for the ore covers a haul from the open pit to the ROM Pad.

FIGURE 16-17 MERCEDES ACTROS 3340 TRUCKS

Source: Chaarat, 2020

The estimated required truck fleet using Talpac software for the contract mining is shown in Table 16-11. The GOH and associated fleet size assume average ore hauling and waste hauling speeds of 23 kph and 19 kph respectively, and load and dump times of two and a half minutes and one minute respectively.

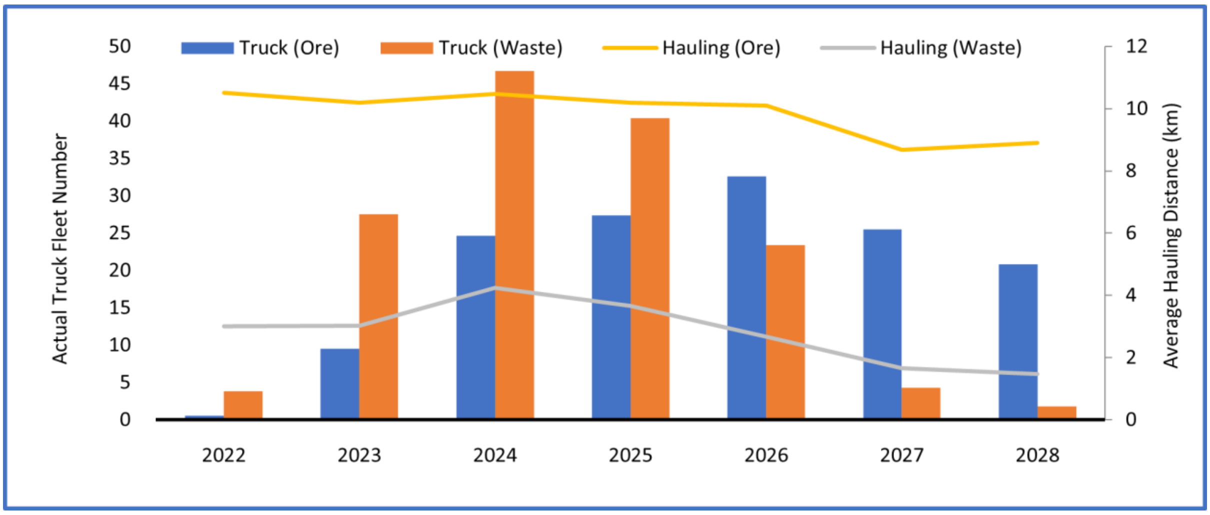
Table 16-11 Truck Fleet Haulage Estimate

Rock Type	Unit	2021	2022	2023	2024	2025	2026	2027	2028
Production Schedule									
Ore	kt	-	38	1,560	3,933	4,469	5,371	4,844	644
Waste	kt	-	658	11,307	14,292	14,060	10,653	2,860	217
Average Hauling Distance									
Ore	m	-	10,515	10,186	10,466	10,198	10,105	8,677	8,903
Waste	m	-	3,000	3,020	4,238	3,660	2,671	1,657	1,467
Average Cycle Time (Average Speed: 23 kph for Ore and 19 kph for Waste)									
Ore	hr	-	1.0	0.9	1.0	0.9	0.9	0.8	0.8
Waste	hr	-	0.4	0.4	0.5	0.4	0.3	0.2	0.2
Truck Productivity (Efficiency: 83%)									
Ore	t/GoH	-	29.4	30.3	29.6	30.3	30.6	35.2	34.4
Waste	t/GoH	-	76.5	76.1	56.8	64.6	84.4	123.0	134.6
Truck Fleet Number (5,390 GOH per Year)									
Ore	No.	-	0.6	9.5	24.7	27.4	32.6	25.5	20.9
Waste	No.	-	3.8	27.6	46.7	40.4	23.4	4.3	1.8
Total Required Trucks	**No.**	**0**	**5**	**38**	**72**	**68**	**57**	**30**	**23**

Source: Chaarat, 2021

Table 16-11 indicates that the mining operation during steady state will require a maximum truck haulage fleet of 72 trucks in order to support the production peak forecast during 2024. Graph 16-7 presents the number of trucks required to support the mining schedule as the average hauling distances changes for ore and waste over the LoM.

Graph 16-7 Estimated Truck Requirement vs. Average Hauling Distance



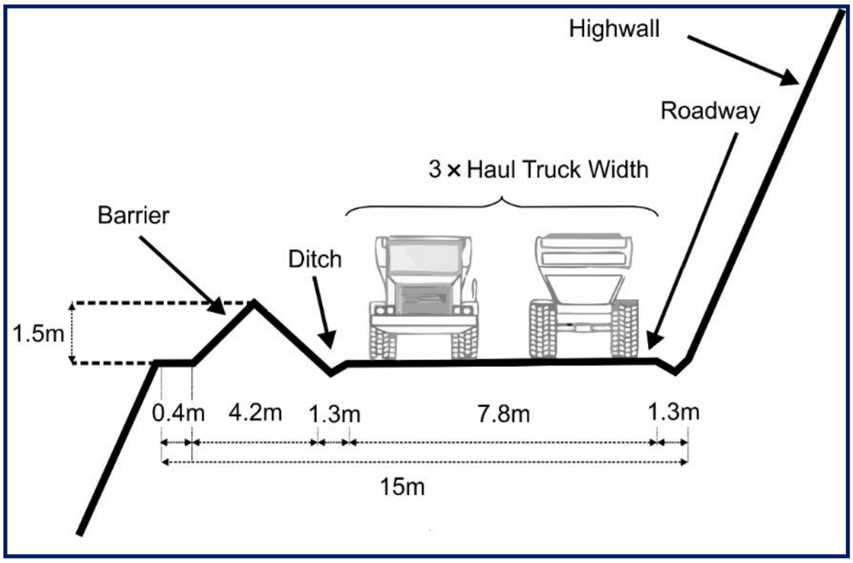
Source: Chaarat, 2021

An analysis of the haulage fleet size requirements has demonstrated that the maximum truck fleet size of 72 could be reduced by optimising some of the turns in the road to enhance the average speed and by re-considering the current sequencing and scheduling, which has resulted in a relatively high production demand from a single area during 2024 to 2025.

16.4.5. Haulage Road and ROM Pad

A typical cross section of a mine haul road is shown in Figure 16-18.

Figure 16-18 Typical Mine Haulage Road Cross-Section



Source: Chaarat, 2020

The haul roads will be 15 m wide inclusive of berm, ditches, and carriageway, which will be a minimum of three times the width of the trucks to permit dual-lane traffic. The preferred dump trucks are 2.6 m wide and so the carriage way will be at least 7.8 m wide.

In winter, the carriageway is planned to be widened to enhance safety in icy conditions. The carriageway will be widened by making use of the space designed for ditches. During the spring and summer, the carriageway will be narrowed to provide ditches to drain water from the roadway.

A barrier (berm), 1.5 m in height, will be constructed on the outside edge of the road. The berm will be 1.5 m in height in order to stop or deflect a vehicle from going over the edge. In some instances, the berm may be smaller but not less than half the height of the tire of the largest vehicle travelling on the road.

Special purpose, single-lane roads may be used in certain circumstances such as pioneering initial benches. Such roads will have a total width not less than 10 m but the same size berm as a dual-lane road. Single-lane roads will have pull-outs to allow equipment to pass and will only be used under written order by the Mine Manager.

The surface of haul roads will be maintained in a smooth and level condition. The carriageway will be sloped slightly towards the berm to facilitate drainage. Ditches will be cleaned during spring and summer to ensure roads drain freely.

Material sloughing from highwalls or berms will be removed, and berms maintained in good repair. In summer, roads will be watered to suppress dust. In winter, roads will be cleared of snow and sanded with material (i.e., 5 mm to 25 mm) to provide traction and a safe running surface.

Trucks loaded with ore will haul an average of 4.3 km before crossing the Sandalash River bridge from where they will continue to haul a further 5.6 km to the ROM Pad. The trucks will then dump directly into the primary crusher feed bin. If the crusher is down or material is not suitable as immediate feed it will be stockpiled until required. A FEL will be available to maintain the ROM Pad and the crushed ore truck load-out station and to keep both areas clean and clear of spillage.

16.4.6. Ore Control (Grade Control)

The effective identification and separation of ore and waste will be critical to the economic success of the mine. Grade control in the open pits will involve the sampling of blasthole cuttings after drilling. These will be assayed for gold, silver, carbon, sulphur, and cyanide solubility. The cuttings will also be mapped to identify the types of alteration consistent with ore-type mineralisation.

This information will allow the blastholes to be classified as either ore or waste and blasting will be managed to minimise the lateral movement of material. Boundaries will be marked on the muckpiles to delineate ore and waste zones. The location of boundaries will be surveyed using GPS technology and marked with wooden stakes tagged with fluorescent ribbon.

A map of the working benches, working faces, zones of ore and waste, and the plan for digging, will be provided to the mining crews on a daily basis. The ore and waste will then be excavated according to the daily and weekly mine plan. The excavator operators will use horn signals to alert the truck drivers as to the type of material that they are to transport.

16.4.7. Drainage and Water Management

General inflows of groundwater will be routed to in-pit sumps for onward pumping via pipelines, to a holding pond from where the water can either be used for dust suppression or discharged. The dewatering arrangements are planned to be sufficiently mobile to be able to address dewatering wherever necessary. More specifically, any in-pit water prior to 2022, will be diverted to the southeast side of the MZ Pit and transported via a catch ditch below the mining operations. After 2022, the pit becomes closed to topography, and any water collected in the pits after 2022 will be diverted to a sump and pumped to the collection ditch below the mine. The collection ditch will deliver the mine water to a settling pond located at the base of the WRD, southwest of the MZ Pit.

It is possible that drain holes may be needed to manage groundwater pressures, however this is deemed to not be likely given the arid conditions and lack of evidence of aggressive groundwater conditions during exploration drilling. A hydrogeology study has been planned during 2021 which will quantify any further drainage hole requirements necessary to depressurise pit walls.

16.4.8. Maintenance

Mine Operations will conduct maintenance on the mining equipment fleet so that sufficient equipment hours are available to meet safety standards and production requirements on an ongoing basis. Average equipment availability over the LoM is planned to be 85%.

Preventative maintenance will be conducted on a regular basis. This will include the changing of oils and filters at intervals recommended by the manufacturer. Fuelling will be done in the open pit at lunch time or shift change. Operators and service personnel will inspect equipment each shift to identify potential problems before a breakdown occurs.

Components will be changed out based on the hourly life-cycle parts schedule for each type of equipment. An adequate stock of spare parts will be maintained on site. Suppliers will be encouraged to stock parts on site on consignment to ensure availability and reduce inventory costs. Some spare parts will be stored in the workshop, however, most spares inventory will be stored in containers near the workshop.

The maintenance focus will be to prevent breakdowns by fixing problems before they impact safety or production. Repairs will be performed on an as-needed basis. Major overhauls will initially be performed at site, however, as the fleet ages and overhauls become more frequent, this work may be performed at off-site supplier facilities.

A temporary maintenance workshop will be installed near the Sandalash River Bridge at the start of mining. The permanent maintenance workshop will comprise a building on a 25 m x 30 m concrete floor.

The building will have 6 m of headroom inside and will be equipped with a 10 t overhead crane. Four overhead doors will provide access to 12 service bays. The building will also house a machine shop and a tool crib. Lubricants will be stored next to services bays in concrete containment.

Two associated structures will house a vehicle wash bay and a welding shop. Both buildings will have concrete floors. The wash bay will be equipped with a hot water boiler and pressure washer. Dirty water will be drained to an oil-water separator. Contaminated waste will be disposed of appropriately. Waste water will be recycled or disposed of through the waste water treatment plant at the camp. The welding shop will permit arc welding to be conducted without fumes and noise affecting other maintenance workers.

Vehicles parked at the workshop will be parked in a designated gravel parking area equipped with plug-ins to keep equipment ready to start in cold weather and lighting.

A list of all equipment to be operated in the mine is provided in Table 16-12.

TABLE 16-12 CONTRACT MINING EQUIPMENT LIST OVER LOM

Year	2022	2023	2024	2025	2026	2027	2028
Production							
Excavator	1	3	5	5	4	2	1
Loader	-	1	-	-	1	1	1
Haul Truck	5	38	72	68	57	30	23
Drill	1	4	5	5	5	3	2
Support							
Dozer	1	2	3	3	3	3	2
Auxiliary Loader	1	1	1	1	1	1	1
Grader	1	2	2	2	2	2	2
Water Truck	1	2	3	3	3	2	2

Source: Chaarat, 2021

16.5. Fuel Consumption Requirements

Table 16-13 contains the assumptions used by Chaarat to estimate the fuel consumption for the contract mining.

Table 16-13 Assumptions for Fuel Calculation

Equipment	Item		Unit	Quantity
Drills	Production Rate	Production Drilling	t/GOH	719
		Wall Control Drilling	t/GOH	587
	Fuel Consumption		ℓ/hr	30
			ℓ/t	0.04
Excavator	Production Rate		t/GOH	688
	Fuel Consumption		ℓ/hr	50
			ℓ/t	0.07
Trucks	Average Speed		kph	19 for Waste and 23 for Ore
	Load Cycle		min	2.5
	Dump Cycle		min	1.0
	Availability		%	85%
	Efficiency		%	83%
	Payload		t	34.5
	Fuel Consumption		ℓ/hr	13
Support and Other	Provision set to 10 % of total litres		%	10%
All	Fuel Price		USD/ℓ	0.60

** Source: Chaarat*

Table 16-14 shows the estimated fuel consumption over the LoM.

Table 16-14 Equipment's Fuel Consumption

Description	Unit	Total	2021	2022	2023	2024	2025	2026	2027	2028
Excavator	kℓ	5,194	-	51	809	1,325	1,347	1,078	539	45
Loader	kℓ	284	-	-	143	-	-	98	24	20
Truck	kℓ	18,761	-	129	2,600	5,003	4,749	3,926	2,090	265
Drill	kℓ	3,197	-	30	549	778	791	684	329	37
Support and Other (10% of Total)	kℓ	2,744	-	21	410	711	689	579	298	37
Total	**kℓ**	**30,179**	**-**	**230**	**4,510**	**7,816**	**7,576**	**6,365**	**3,279**	**403**

Source: Chaarat, 2021

For the estimation of fuel quantity and costs over the LoM, fuel consumption estimates of 13 ℓ/hr, 50 ℓ/hr and 30 ℓ/hr have been used for haul trucks, excavators and drills respectively.

16.6. Mining Personnel

A Mining Contractor will be employed to hire the workforce, train operators, provide mining equipment, and conduct all of the activities necessary to meet the planned production targets. The contract will also cater for the housing and feeding of all mining personnel.

The Owner's Mining Manager will administer the mining contract and provide mine planning and other technical services to enable the Mining Contractor to execute the mine plan effectively and efficiently. These technical services will inter alia include short and long-term planning, mine design, geology, and grade control.

16.6.1. Contractor's Personnel

It is estimated that the Mining Contractor will employ a maximum of 524 persons, which ranges from 133 to 524 with an average of 365. They will cover administration, blasting, maintenance, operations, supervision and training. Technical services, including mine engineering, will fall under the Owner's scope of work. Most of these employees will cover the operations where there will be a peak of truck drivers during 2025 of some 284 truck drivers. The number of drivers will vary as the truck complement changes to meet the production requirements. Table 16-15 shows the Mining Contractor's manpower.

TABLE 16-15 MINING CONTRACTOR'S MANPOWER

Position	2022	2023	2024	2025	2026	2027	2028
Mine Superintendent	1	2	2	2	2	2	1
Mine General Foreman	1	2	2	2	2	2	1
Mine Foreman	2	4	4	4	4	4	4
Mine Planner	1	2	2	2	2	2	1
Surveyor 1	2	2	2	2	2	2	2
Surveyor 2	2	4	4	4	4	4	2
Training Foreman	1	2	2	2	2	2	1
Equipment Trainer	4	8	8	8	8	8	4
Excavator Operator	4	12	20	20	16	8	4
Loader Operator	0	4	0	0	4	4	4
Drill Operator	4	16	20	20	20	12	8
Drill Helper	4	8	8	8	8	8	4
Haul Truck Driver	22	160	300	284	240	128	96
Dozer Operator	4	8	12	12	12	12	8
Grader Operator	4	8	8	8	8	8	8
Water/Sand Truck Operator	4	8	12	12	12	8	8
Auxiliary Operator	4	4	4	4	4	4	4
Blasting Foreman	2	2	2	2	2	2	2
Blaster 1	2	2	2	2	2	2	2
Blaster 2	2	4	4	4	4	4	2
Blaster Helper	2	4	4	4	4	4	2
Bulk Truck Operator	2	2	2	2	2	2	2
Maintenance General Foreman	1	2	2	2	2	2	1

Position	2022	2023	2024	2025	2026	2027	2028
Maintenance Foreman	2	4	4	4	4	4	2
Maintenance Planner	2	2	2	2	2	2	2
Mechanic 1	8	16	16	16	16	16	8
Mechanic 2	8	16	16	16	16	16	8
Welder	6	12	12	12	12	12	6
Electrician	2	4	4	4	4	4	2
Fuel/Lube Operator	4	8	8	8	8	8	4
Crane Operator	2	4	4	4	4	4	2
Warehouse Foreman	2	2	2	2	2	2	2
Warehouseman	8	8	8	8	8	8	8
Site Services Foreman	2	2	2	2	2	2	2
Site Services Operator	8	12	12	12	12	12	8
Labourer	4	8	8	8	8	8	4
Total	**133**	**368**	**524**	**508**	**464**	**332**	**229**

Source: Chaarat, 2021

Senior roles will be filled by a single person who can cover another similar position when that person is absent or on their off cycle, for example, the Project Manager and the Mine Superintendent. Positions where the total number of people represent a multiple of two are generally dayshift only, for example technical positions and blasting. Positions where the total number of people is a multiple of four are on continuous shift, for example, operators.

As Çiftay will perform a number of contracts on site (mining, camp services, crushed ore haul), the Project Manager and his staff are not specifically part of the Contract Mining.

A total of 35 persons are involved in training. This allows for seven trainers on each crew to monitor operation of critical production equipment including trucks, excavators, dozers, graders, loaders, and drills. These personnel also provide a manpower reserve to replace employees who are sick or otherwise absent.

16.6.2. Owner’s Personnel

The owner’s personnel will amount to 22 positions, with about half of these being associated with grade control activities. The Mining Manager and Chief Engineer will cover for each other over their off cycles. The geologists, mining engineers, and surveyors will work dayshift only. The grade control technicians and samplers will work on a continuous shift schedule. Details of the Owner’s mining manpower is shown in Table 16-16.

Table 16-16 Owner’s Mining Manpower

Position	Number per Shift	Shift Number	Total
Mine Manager	1	1	1
Chief Engineer	1	1	1
Mining Engineer	2	1	4*
Geologist	1	1	2*

Position	Number per Shift	Shift Number	Total
Ore Control Technician	1	2	4*
Sampler	1	2	4*
Surveyor	1	1	2*
Survey Technician	1	1	2*
Clerks	1	1	2*
Total		**22**	

Source: Chaarat, 2021

**Requirement doubled to accommodate the15 days on and a 15 days off, rotating cycle.*

16.7. MINING COST ESTIMATES

Table 16-17 and Table 16-18 presents the estimated owner related mining costs over the pre-production and steady state periods respectively.

TABLE 16-17 OWNER MINING COST OVER PRE-PRODUCTION

Description	Units	Year		Total
		2022	2023	
Pre-strip – Fixed Costs				
Months	Qty	5	8	13
Labour	USD 000's	290	464	754
Expenses	USD 000's	148	237	385
Total	USD 000's	438	701	1,139
Pre-strip - Mining				
Ore	Mt	0,04	0,56	0,59
Waste	Mt	0,66	6,15	6,81
Total	Mt	0,70	6,71	7,40
Pre-strip – Assay Costs				
Ore Sampled	Mt	0,04	0,56	0,59
Waste Sampled	%	11%	11%	11%
Waste Sampled	Mt	0,07	0,68	0,75
Total Sampled	Mt	0,11	1,23	1,34
Rate	t/sample	194	194	194
Samples	Qty	569	6,350	6,919
Rate	sample/d	4	26	532
Cost	USD/sample	41,61	41,61	41,61
Total	USD 000's	24	264	288
Pre-strip – Total Costs				
Total	**USD 000's**	**462**	**965**	**1,426**
Unit Rate	**USD/t ore**	**12,15**	**1,74**	**2,40**
Unit Rate	**USD/t mined**	**0,66**	**0,14**	**0,19**

Source: Chaarat, 2021

Table 16-18 Owner Mining Cost over the Production Period

Description	Units	Year						Total	LOM
		2023	2024	2025	2026	2027	2028		Total
Production – Fixed Costs									
Day	Qty	4	12	12	12	12	2	54	67
Labour	USD 000's	232	696	696	696	696	116	3,132	3,886
Expenses	USD 000's	118	355	355	355	355	59	1,598	1,982
Total	USD 000's	350	1,051	1,051	1,051	1,051	175	4,730	5,868
Ore	Mt	1,00	3,93	4,47	5,37	4,84	0,64	20,27	20,86
Waste Mined	Mt	5,16	14,29	14,06	10,65	2,86	0,22	47,24	54,05
Total	Mt	6,16	18,23	18,53	16,02	7,70	0,86	67,51	74,91
Production – Assay Costs									
Ore	Mt	1,00	3,93	4,47	5,37	4,84	0,64	20,27	20.86
Sample Ratio	t:t (w:o)	11%	11%	11%	11%	11%	11%	11%	11%
Waste Sampled	Mt	0.57	1.57	1.55	1.17	0.31	0.02	5.20	5.95
Total	Mt	1,57	5,51	6,02	6,54	5,16	0,67	25,46	26,80
Rate	t/sample	194	194	194	194	194	194	194	194
Samples	Qty	8,102	28,372	31,002	33,718	26,584	3,444	131,221	138,139
Rate	sample/d	68	79	86	94	74	57	2,430	2,062
Cost	USD/sample	8.64	8.64	8.64	8.64	8.64	8.64	8.64	10.29
Total	USD 000's	70	245	268	291	230	30	1,134	1,422
Production – Total Costs									
Total	USD 000's	420	1,296	1,319	1,342	1,281	205	5,863	7,290
Unit Rate	USD 000's	0.42	0.33	0.30	0.25	0.26	0.32	0.29	0.35
Unit Rate	USD 000's	0.07	0.07	0.07	0.08	0.17	0.24	0.09	0.10

Source: Chaarat, 2021

Table 16-19 shows the expected contract mining costs over the LoM. The fuel costs will be borne by the owner (Chaarat).

Table 16-19 Contractor Mining Cost over LoM

Description	Units (USD 000's)	Total	2022	2023	2024	2025	2026	2027	2028
Base Mining Cost									
Base Mining Cost	Ore	37,557	59	2,420	6,736	8,041	9,985	9,088	1,227
	Waste	96,455	1,109	17,890	25,095	26,208	20,327	5,410	416
	Total	**134,011**	**1,168**	**20,310**	**31,831**	**34,249**	**30,312**	**14,498**	**1,642**
Fuel Cost									
Fuel Cost	Ore	7,092	13	538	1,400	1,557	1,856	1,520	208
	Waste	11,015	125	2,168	3,290	2,989	1,963	448	34
	Total	**18,108**	**138**	**2,706**	**4,690**	**4,546**	**3,819**	**1,968**	**242**
Overhaul Cost									
Overhaul Cost	Ore	8,060	17	641	1,715	1,855	2,159	1 466	207

Description	Units (USD 000's)	Total	2022	2023	2024	2025	2026	2027	2028
	Waste	(8,274)	(131)	(2,108)	(1,173)	(1,801)	(2,206)	(791)	(64)
	Total	**(214)**	**(115)**	**(1,466)**	**542**	**54**	**(48)**	**675**	**143**
Total Cost									
Total Cost	Ore	52,708	89	3,600	9,851	11,453	14,000	12,074	1,642
	Waste	99,196	1,102	17,950	27,212	27,396	20,084	5,066	386
	Total	**151,905**	**1,191**	**21,550**	**37,064**	**38,849**	**34,084**	**17,140**	**2,027**

Source: Chaarat, 2021

17. Recovery Methods

17.1. Summary

The process design for the Tulkubash Project is based on the testwork presented in Section 13.0, the geological information presented in Sections 7.0-14.0, and the mining plan presented in Section 16.0. A successful process design is one that results in a flowsheet of a plant that is as simple as possible to supply, operate, and maintain, whilst maximising gold and silver recoveries and minimising power requirements.

Ore that is suitable for heap leach processing is defined as any material identified within the Feasibility Study pit shell above the cut-off grade of 0.2 g/t, and that has a total sulphur content of 0.5% or less (S_{TOTAL} ≤0.5%).

The Tulkubash processing facilities comprise typical components of a conventional heap leach operation namely, crushing, truck load-out and stacking, valley fill heap leach and an ADR circuit based on split-AARL elution.

The LoM gold and silver recoveries have been calculated to be 73.6% and 63.4%, respectively, as reported in Section 13.

17.2. Process Design Criteria

The process design criteria have been derived from the following information sources:

- Testwork results, as reported in Section 13;
- The mine production plan, as described in Section 16;
- Equipment manufacturers' recommendations;
- Previous studies completed on the Project; and
- Information published in the public domain, industry standard assumptions, and knowledge gained from similar projects/unit operations.

Table 17-1 lists a summary of the principal process design criteria established for the Project

Table 17-1 Process Design Criteria

Description	Unit	Value	Source
General			
Ore Characteristics			
Ore Relative Density	-	2.64	7,11
RoM Ore Bulk Density	t/m^3	1.90	2
Crushed Ore Bulk Density	t/m^3	1.75	2
As-delivered Ore Moisture	%	3.8 - 4.7	1
Operating Schedule			
Shifts/Day	shifts/d	2	1

Description	Unit	Value	Source
Hr/Shift	h	12	1
Hr/Day	h	24	1
Days/Year	d	365	1
Plant Availability/Utilization			
Crushing Plant Availability (planned down time)	%	85	6
Crushing Plant Utilization (unplanned down time)	%	82	6
Crushing Plant Effective Utilization	%	70	2
Nominal Plant Throughput	Dry t/d	13,500	1
Nominal Processing Rate	Dry t/h	804	3
Annual Design Processing Rate	Dry t/a	4,927,500	3
Days/Year Heap Leach Ore Stacking	d/a	350	3
Bond Work Index			
Crusher Work Index	kWh/t	10	5
Abrasion Index	N/A	0.465	5
Production			
Head Grade, Gold	g/t	0.85	11
Head Grade, Silver	g/t	1.26	11
Product Size	P_{80}, mm	12.5	5, 8
Gold Recovery	%	73.6	5,11
Silver Recovery	%	63.4	5,11
Gold Production	tr oz/a	99,109	3
	kg/a	3,083	3
Silver Production	tr oz/a	126,554	3
	kg/a	3,936	3
100 - Crushing			
Primary Crushing			
Primary Crusher Nominal Processing Capacity	t/h	398	7
Maximum Feed Particle Size	mm	700	7
F_{80} Feed Particle Size	mm	430	7, 9
Primary Crusher	type	Jaw - C150	7, 1
Secondary/Tertiary Crushing			
Scalping Screen	-	Double Deck, banana, Vibrating Screen	7
Product Screen Nominal Processing Capacity	t/h	810	7
Number Needed	-	1	3
Top Deck Aperture Size	mm	100	7
Bottom Deck Aperture Size	mm	75	7

Description	Unit	Value	Source
Secondary Nominal Processing Capacity	t/h	517	7
Number Needed	-	1	3
Secondary Crusher	Type	GP500S Extra Coarse	7
Tertiary Surge Bin	m^3	70	7
Product Screen	-	Double Deck, banana, Vibrating Screen	7
Product Screen Nominal Processing Capacity	t/h	792	7
Number Needed	-	2	3
Top Deck Aperture Size	mm	35	7
Bottom Deck Aperture Size	mm	14	7
Tertiary Crusher	-	HP800 Extra Coarse	10, 7, 1
Nominal Processing Rate	t/h each	387	7
Number Needed	-	2	3
Heap Leach Feed Particle Size (P_{100})	mm	15	2
200 - Leaching			
Stacking System			
Type	-	Truck Stacking	1
Total Stacking Capacity	Dry t/h	587	3
Stack Height	m	7	2
Heap Leach Pad			
Type	-	Permanent Multi-lift	2
Total Capacity	Mt	25.88	2
Maximum Heap Height	m	90	2
Nominal Stacking Rate	Dry t/d	13,500	1
Average In-Situ Ore Density	t/m^3	1.75	2
Volume of Daily Production	m^3/d	7,714	3
Area of Daily Production	m^2/d	1,102	3
Area of Irrigation	m^2	79,600	3
Pan Evaporation	mm/a	532	Other Sources
Average Precipitation Rate	mm/a	470	Other Sources
Saturated Moisture	%	7	7
Net Evaporation	m^3/hr	3.3	3
Residual Moisture Content	%	6.6	7
Drain-down Moisture Content	%	5	6
Solution Application			
Leach Pad Cycle	d	60	5
Solution Flowrate	$\ell/m^2/h$	10	5, 10

Description	Unit	Value	Source
Nominal Flowrate	m^3/hr	723	3
Design Flowrate	m^3/hr	796	3
Solution pH	pH	10.5	5
Cyanide Solution Strength	g/ ℓ	0.5 – 1.0	5
Solution Heating	-	No	1
Solution Ponds			
Pregnant Leach Solution (PLS) Pond			
Type	-	In Heap	2
Operation Storage Capacity	hr	8	6
Operation Volume - Nominal	m^3	6,368	2
Operation Volume - Maximum	m^3	9,552	2
Emergency Drain-down	hr	24	2
Emergency drain-down Volume	m^3	28,656	3
Storage Capacity	m^3	38,208	3
Storage Capacity (Rock+Liquid)	m^3	101,888	3
Pond Freeboard	m	1.5	6
Number of Pumps	-	1	2
Pump Type	-	Vertical Turbine	2
300 - ADR Plant			
ADR Plant Availability (planned down time)	%	95	6
ADR Plant Utilization (unplanned down time)	%	95	6
ADR Plant Effective Utilization	%	90	2
Adsorption			
Location	-	Inside insulated Building	6
Processing Method	-	Carbon Adsorption Columns	10
Type	-	Carousel	6
Number of Trains Required	-	1	6
Number of Tanks per Train	-	6	3
Nominal Flowrate	m^3/hr	800	2
Design Flowrate	m^3/hr	850	2
Carbon Density	t/m^3	0.5	Other Sources
Carbon Consumption Rate	t/a	24.6	2
Carbon Tonnage	t/column	4	3
Carbon Advance Rate	t/d	8	3
Loaded Carbon	g/t (total)	2,800	3
Adsorption Efficiency - Gold	%	98.0	6
Adsorption Efficiency - Silver	%	98.0	6
Elution			

Description	Unit	Value	Source
Type	-	Split AARL	2
Elution Column Capacity	t	4	3
Number of Elution Columns	-	1	3
Strip Schedule	batches/wk.	14	2
Design Strip Temperature	°C	140	1
Strip Pressure	kPa	350	2
Strip Solution Rate	bv/h	2.50	2
Electrowinning			
Number of Cells	-	2	3,7
Type	-	Atmospheric Sludge Tank	7
Cell Temperature	°C	65	10
Acid Wash			
Type	-	Diluted Nitric Acid (3% HNO_3)	2
Acid Column Capacity	t	4	3
Number	-	1	3
Wash Schedule	-	Every Batch	10
Acid Wash Time	h	2	10
Carbon Regeneration			
Type	-	Diesel-Fired Rotary Kiln	1
Capacity	kg/h	200	2
Regeneration Temperature	°C	700	1, 2
Max Operating Temperature	°C	750	2
Smelting			
Type	-	Diesel-Fired Tilting Furnace	1, 2
Smelting Temperature	°C	1,200	2

Notes: Information Sources: 1 – client; 2 – engineering design; 3 – calculation; 4 – mass balance; 5 – metallurgical testwork; 6 – assumption; 7 – vendor; 8 – GBM (2018); 9 – IMC; 10 – Industry Standard; 11 – block model

Detailed design criteria for each heap leach component are presented in the following sections.

17.2.1. Leach Pad Design Criteria

The primary design objectives for the proposed heap leach facility are as follows:

- Provide a stable and cost-effective configuration for staged heap development;
- Effectively collect and convey Pregnant solutions to the process plant or the PLS overflow and Emergency Ponds while ensuring maximum recovery;
- Provide for secure containment of pregnant solution and run-off up to the design flood event, while monitoring and eliminating losses due to leakage;

- Minimize surface run-off entering the leach pad area while providing for the collection of direct run-off from behind the heap area;
- Provide for staged development of heap leach stacking and leaching operations;
- Ensure that the Heap Leach Facility can be operated satisfactorily under year-round conditions, and
- Effective decommissioning and reclamation of all heap leach facility components.

The HLF specific design criteria were developed in conjunction with Chaarat. The parameters adopted for the feasibility design are summarised in Table 17-2.

17.2.1.1. HLF Design Standards

The HLF has been designed in accordance with international best available technology (BAT). The table below summaries the standards used. (Table 17-2)

Table 17-2 Design Guidelines Utilized for the HLF Design

Refer	Author	Standard
Slope stability, Pond, and Dam Design	Canadian Dam Association	Dam Safety Guidelines (2007/2014)
Designing Geosynthetics	Robert M Koerner	Designing Geosynthetics 6th Ed. Vol. 1 & 2 (2016)
Haul and Perimeter Road Design	Dwayne D. Tannant and Bruce Regensburg	Guidelines for Mine Haul Road Design (2001)

Standards referenced during the preliminary design are summarized in below.

The leach pad for this project is combined with the PLS pond. The leach pad has a solution containment system (liner) on to which the ore is stacked, and a solution collection system that directs pregnant solution to the PLS Pond. The pad construction will proceed in three phases until the completion of the 25.88 Mt footprint. Table 17-3 summarises the Heap Leach Pad design criteria.

Table 17-3 Leach Pad Design Criteria

Parameter	Unit	Design Input Value	Reference/ Notes
General			
Number of Phases	No.	3	Chaarat
Phase 1 Ore Storage Capacity	t	6,480,000	Ausenco
Phase 2 Ore Storage Capacity	t	10,290,000	Ausenco
Phase 3 Ore Storage Capacity	t	9,110,000	Ausenco
Underdrain System			
Underdrain	Yes/No	Yes	Ausenco

Parameter	Unit	Design Input Value	Reference/ Notes
Pipe Type	type	Dual Wall Perforated	Ausenco
Pipe Diameter	mm	varies	Ausenco
Drainage Gravel	Yes/No	Yes	Ausenco
Minimum Pipe Gradient	%	1	Ausenco
Solution Collection System			
Solution Collection System	Yes/No	Yes	Ausenco
Pipe Type	type	HDPE Dual Wall Perforated and Solid Wall Non-Perforated Pipe	Ausenco
Pipe Diameter	mm	varies	Ausenco
Drainage Gravel	Yes/No	No[1]	Ausenco
Minimum pipe Gradient	%	1	Ausenco
Liner System			
Bedding Layer (Entire Liner Area)	mm	150-400	Ausenco
Transition Layer (Scree Slopes)	mm	Geo-composit	Ausenco
Liner Type	Single/Double	Single	Ausenco
-Liner Materials			
-GCL	Type	High Strength	Ausenco
-Geomembrane	Type	LLDPE	Ausenco
	Thickness (mm)	2.0	Ausenco
	Textured/Smooth	SST	Ausenco
Heap			
Overliner	m	1	Ausenco
Lift Height	m	7	Chaarat
Global Exterior Heap Slope	H:V	3:1	Ausenco
Lift Exterior Slope	H:V	1.5:1	Ausenco
Bench Width	m	11	Calculated
Maximum Heap Height	m	90	Ausenco
Design Seismic Event	1/year	MCE	CDA
Leach Pad during Construction and Operation Minimum Static Factor of Safety (FOS)	unitless	1.3	CDA
Leach Pad Long-term Minimum Static FOS	unitless	1.5	CDA
Leach Pad Minimum pseudo-static FOS	unitless	1.0	CDA
Leach Pad Minimum Post-earthquake FOS	unitless	1.2	CDA
Design Storm Event	1:year	200	Ausenco

Note *1. Drainage gravel eliminated due to placement of overliner.*

17.2.2. Pregnant Leach Solution Pond Design Criteria

The PLS Pond is part of the Heap Leach Pad and will be filled with ROM to prevent freezing of the solution during the winter. The ROM will be of graded, coarse particles layered above the overliner material. The PLS embankment will include a spillway to direct overflow into the PLS overflow Pond. In addition, a portion of the heap will be stacked over the south section of the PLS Pond Table 17-4 summarises the PLS Pond design criteria.

Table 17-4 PLS Pond Design Criteria

Parameter	Unit	Design Input Value	Reference/ Notes
General			
Dam Classification		Very High	CDA
Design Storm Event	1:year	n.a.	Ausenco
Operation Storage Capacity	hr	8	Ausenco
Operation Volume - Nominal	m^3	6,368	Ausenco
Operation Volume - Maximum	m^3	9,552	Ausenco
Emergency Drain-down	hr	24	Ausenco
Emergency drain-down Volume	m^3	28,656	Ausenco
Storage Capacity	m^3	38,208	Ausenco
Storage Capacity (Rock+Liquid)	m^3	101,888	Ausenco
Pond Freeboard	m	1.5	Ausenco
Underdrain			
Underdrain	Yes/No	Yes	Ausenco
Pipe Type	type	HDPE Dual Wall Perforated and Solid Wall Non-Perforated Pipe	Ausenco
Pipe Diameter	mm	Varies	Ausenco
Drainage Gravel	Yes/No	Yes	Ausenco
Minimum Pipe Gradient	%	1	Ausenco
Solution Collection System			
Solution Collection System	Yes/No	Yes	Ausenco
Internal Caissons Risers	No.	2	Ausenco
Caisson Material	Concrete/Steel/Plastic	Steel	Ausenco
Caisson Size - Diameter	mm	914	Ausenco
Caisson Size - Wall Thickness	mm	14	Ausenco
Pipe Type	type	Dual Wall Perforated	Ausenco
Pipe Diameter	mm	Varies	Ausenco
Drainage Gravel	Yes/No	No[1]	Ausenco
Minimum Pipe Gradient	%	1	Ausenco
Liner System			
Bedding Layer (Entire Liner Area)	mm	150-400	Ausenco
Transition Layer (Scree Slopes)	mm	none	Ausenco
Liner Type	Single/Double	Double	Ausenco
Liner Materials			

Parameter	Unit	Design Input Value	Reference/ Notes
GCL	Type	High Strength	Ausenco
Secondary Geomembrane	Type	LLDPE	Ausenco
	Thickness (mm)	1.5	Ausenco
	Textured/Smooth	DST	Ausenco
Geonet	Type	PET/PP	Ausenco
Primary Geomembrane	Type	LLDPE	Ausenco
	Thickness (mm)	2.0	Ausenco
	Textured/Smooth	SST	Ausenco
Leak Detection and Recovery System	Yes/No	Yes	Ausenco
Embankment			
Structural Fill	Type	Colluvial	Ausenco
Transition Zone	Type	Sandy Gravel	Ausenco
Crest Width	m	6	Ausenco
Downstream Dam Slope	H:V	2.5:1	Ausenco
Upstream Dam Slope	H:V	2:1	Ausenco
Design Seismic Event	1:year	MCE	CDA
Dam during Construction and Operation Minimum Static Factor of Safety (FOS)	unitless	1.3	CDA
Dam Long-term Minimum Static FOS	unitless	1.5	CDA
Dam Minimum Pseudo-static FOS	unitless	1.0	CDA
Dam Minimum Post-earthquake FOS	unitless	1.2	CDA
Spillway Design Storm Event	1:year	2/3 between 1,000 and PMF	CDA
Heap			
Over-liner	m	1	Ausenco
Over-liner Size P_{80}	mm	35mm	Ausenco
Pond Rock Fill Size	mm	150-250	Ausenco
Rock Fill Elevation	masl	2381	Ausenco
Rock Fill Void Space	%	37.5	Ausenco

Drainage gravel eliminated due to placement of overliner and rock fill.

17.2.3. PLS Overflow Pond Design Criteria

The PLS Overflow Pond will provide operational surge volume capacity to accommodate the annual rainfall (mainly occurring in Spring) and annual snow melt events. The PLS Overflow Pond will contain any surplus liquid from the PLS Pond and any excess barren solution that exits the ADR plant but is not delivered to the irrigation system. Any flow greater than the annual run-off will overflow to the Emergency Pond through a spillway. Table 17-5 describes the PLS Overflow Pond design criteria.

TABLE 17-5 PLS OVERFLOW POND DESIGN CRITERIA

Parameter	Unit	Design Input Value	Reference/ Notes
General			
Dam Classification		Very High	CDA
Design Storm Event	1:year	1	Ausenco
Catchment Area (Leach Pad, PLS Pond and PLS Overflow Pond)	Ha	40.0	Ausenco
Storage Capacity	m^3	89,290	Ausenco
Pond Freeboard	m	1.5	Ausenco
Underdrain			
Underdrain	Yes/No	Yes	Ausenco
Pipe Type	type	HDPE Dual Wall Perforated and Solid Wall Non-Perforated Pipe	Ausenco
Pipe Diameter	mm	varies	Ausenco
Drainage Gravel	Yes/No	Yes	Ausenco
Minimum pipe Gradient	%	1	Ausenco
Liner System			
Bedding Layer (Entire Liner Area)	mm	150-400	Ausenco
Transition Layer (Scree Slopes)	mm	none	Ausenco
Liner Type	Single/Double	Double	Ausenco
Liner Materials			
GCL	Type	High Strength	Ausenco
Secondary Geomembrane	Type	LLDPE	Ausenco
	Thickness (mm)	1.5	Ausenco
	Textured/Smooth	DST	Ausenco
Geonet	Type	PET/PP	Ausenco
Primary Geomembrane	Type	LLDPE	Ausenco
	Thickness (mm)	2.0	Ausenco
	Textured/Smooth	SST	Ausenco
Leak Detection and Recovery System	Yes/No	Yes	Ausenco
Embankment			
Structural Fill	Type	Colluvial	Ausenco
Transition Zone	Type	Sandy Gravel	Ausenco
Crest Width	m	6	Ausenco
Downstream Dam Slope	H:V	2.5:1	Ausenco
Upstream Dam Slope	H:V	2:1	Ausenco
Design Seismic Event	1:year	MCE	CDA

Parameter	Unit	Design Input Value	Reference/ Notes
Dam during Construction and Operation Minimum Static Factor of Safety (FOS)	unitless	1.3	CDA
Dam Minimum Long-term Static FOS	unitless	1.5	CDA
Dam Minimum Pseudo-static FOS	unitless	1.0	CDA
Dam /Minimum Post-earthquake FOS	unitless	1.2	CDA
Spillway Design Storm Event	1:year	2/3 between 1,000 and PMF	CDA

17.2.4. Emergency Pond Design Criteria

The Emergency Stormwater Pond will collect any overflow from the Overflow PLS Pond from the run-off greater than average annual run-off up to the design storm event. The pond is designed to capture and retain the 1 in 200-year inflow design event. The Emergency Pond is meant to store excess run-off for short periods of time and is not meant to be used for any purpose other than as temporary emergency storage volume. Table 17-6 describes the Emergency Pond design criteria.

Table 17-6 Emergency Pond Design Criteria

Parameter	Unit	Design Input Value	Reference/ Notes
General			
Dam Classification		Very High	CDA
Design Storm Return Period	1:year	200	Ausenco
Design Storm Duration	hr	24	Ausenco
Design Equivalent Rainfall	mm	105.6	Ausenco
Catchment Area (Leach Pad, PLS Pond and PLS Overflow Pond)	Ha	52.1	Ausenco
Storage Capacity	m^3	55,000	Ausenco
Pond Freeboard	m	1.5	Ausenco
Underdrain			
Underdrain	Yes/No	Yes	Ausenco
Pipe Type	type	HDPE Dual Wall Perforated and Solid Wall Non-Perforated Pipe	Ausenco
Pipe Diameter	mm	varies	Ausenco
Drainage Gravel	Yes/No	Yes	Ausenco
Minimum pipe Gradient	%	1	Ausenco
Liner System			
Bedding Layer (Entire Liner Area)	mm	150-400	Ausenco
Transition Layer (Scree Slopes)	mm	none	Ausenco

Parameter	Unit	Design Input Value	Reference/ Notes
Liner Type	Single/Double	Single	Ausenco
- Liner Materials			
-GCL	Type	High Strength	Ausenco
-Primary Geomembrane	Type	LLDPE	Ausenco
	Thickness (mm)	2.0	Ausenco
	Textured/Smooth	SST	Ausenco
Leak Detection and Recovery System	Yes/No	No	Ausenco
Embankment			
Structural Fill	Type	Colluvial	Ausenco
Transition Zone	Type	Sandy Gravel	Ausenco
Crest Width	m	6	Ausenco
Downstream Dam Slope	H:V	2.5:1	Ausenco
Upstream Dam Slope	H:V	2:1	Ausenco
Design Seismic Event	1/year	MCE	CDA
Dam during Construction and Operation Minimum Static Factor of Safety (FOS)	unitless	1.3	CDA
Dam Minimum Long-term Static FOS	unitless	1.5	CDA
Dam Minimum Pseudo-static FOS	unitless	1.0	CDA
Dam Minimum Post-earthquake FOS	unitless	1.2	CDA
Spillway Design Storm Event	1:year	2/3 between 1,000 and PMF	CDA

17.2.5. Attenuation Pond Design Criteria

The Attenuation Pond (AP) is designed to capture surface run-off arising from areas upstream of the HLF and from diversion channels located along the eastern and western sides of the leach pad. In addition, a south-eastern diversion channel will be constructed to convey surface run-off out of the valley to reduce the size of the Attenuation Pond and of the Attenuation Pipelines required.

The primary function of the AP is therefore to prevent clean surface run-off entering the HLF and ponds and to reduce the peak flows that need to be directed underneath the leach pad and ponds thereby reducing the of costs of the Attenuation pipeline. The Attenuation dam will be located just outside of the boundary of the 30 Mt HLF footprint. Run-off will be collected in the AP and discharged via the Attenuation drainage system to a sediment pond just upstream of the Kumbeltash Stream north of the HLF footprint. Table 17-7 describes the Attenuation Pond design criteria.

Table 17-7 Attenuation Pond Design Criteria

Parameter	Unit	Design Input Value	Reference/ Notes
General			
Dam Classification		Significant	CDA
Design Storm Return Period	1 year	200	Ausenco
Design Storm Duration	hr	24	Ausenco
Design Equivalent Rainfall	mm	105.6	Ausenco
Catchment Area	Ha	239	Ausenco
Storage Capacity	m^3	51,700	Ausenco
Pond Freeboard	m	1.5	Ausenco
Underdrain	Yes/No	No	Ausenco
Attenuation Pipelines			
Number of Attenuation Pipelines	No.	2	Ausenco
Attenuation Pipe Type	type	Solid Wall Non-Perforated Pipe	Ausenco
Minimum pipe Gradient	%	1	Ausenco
Liner System			
Bedding Layer (Entire Liner Area)	mm	150-400	Ausenco
Transition Layer (Scree Slopes)	mm	none	Ausenco
Liner Type	Single/Double	Single	Ausenco
Liner Materials			
GCL	Type	High Strength	Ausenco
Primary Geomembrane	Type	LLDPE	Ausenco
	Thickness (mm)	2.0	Ausenco
	Textured/Smooth	SST	Ausenco
Leak Detection and Recovery System	Yes/No	No	Ausenco
Embankment			
Structural Fill	Type	Colluvial	Ausenco
Transition Zone	Type	Sandy Gravel	Ausenco
Crest Width	m	6	Ausenco
Downstream Dam Slope	H:V	2.5:1	Ausenco
Upstream Dam Slope	H:V	2:1	Ausenco
Design Seismic Event	1/year	between 100 and 1,000	CDA
Dam during Construction and Operation Minimum Static Factor of Safety (FOS)	unitless	1.3	CDA
Dam Minimum Long-term Static FOS	unitless	1.5	CDA

Parameter	Unit	Design Input Value	Reference/ Notes
Dam Minimum Pseudo-static FOS	unitless	1.0	CDA
Dam Minimum Post-earthquake FOS	unitless	1.2	CDA
Spillway Design Storm Event	1:year	between 100 and 1,000	CDA

17.2.6. Sediment and Underdrain Pond Design Criteria

These ponds are small and do not fall under CDA guidelines. The smaller of the two sediment ponds captures sediment from the HLF eastern perimeter diversion channel and the larger Sediment Pond captures any sediment from the Attenuation Pipeline. The Underdrain Pond captures near surface groundwater beneath the HLF and is also used for monitoring for any leaks from the HLF. Table 17-8 describes the two sediment and underdrain ponds design criteria.

Table 17-8 Sediment and Underdrain Ponds Design Criteria

Parameter	Unit	Design Input Value	Reference/ Notes
Sediment Ponds			
Design Storm Return Period	1:year	200	Ausenco
Design Storm Duration	hr	24	Ausenco
Design Equivalent Rainfall	mm	105.6	Ausenco
Catchment Area	Ha	239	Ausenco
Storage Capacity	m^3	51,700	Ausenco
Pond Freeboard	M	1.5	Ausenco
Underdrain	Yes/No	No	Ausenco
Underdrain Pond			
Number of Attenuation Pipelines	No.	2	Ausenco
Attenuation Pipe Type	type	Solid Wall Non-Perforated Pipe	Ausenco
Minimum pipe Gradient	%	1	Ausenco
Spillway Design Storm Event	1:year	between 100 and 1,000	CDA

17.3. Availability and Utilisation

The crushing facility will be designed to have 85% availability, and 82% utilisation, to yield a 70% effective utilisation. However, higher crushing facility runtimes are expected to be achieved during the steady state operation of the plant.

The ROM pad will have a capacity of 33,122 m^3 based on a stockpile height of 12 m and an angle of repose of 37 degrees. This corresponds to approximately 63,000 tonnes at an average RoM bulk density of 1.90 t/m^3. This will be sufficient for routine storage purposes and for a major interruption to mine supply of up to about 4 days, based on a processing rate of 13,500 t/d.

The Fine Ore stockpile with the material bulk density of 1.75 t/m³, is designed with live capacity of 4,500 tonnes and total capacity of 11,500 tonnes. This will provide approximately 6 hours live storage and 16 hours of total storage.

The ADR plant design is based on an availability of 95%. The plant utilisation is expected to be 95%, which corresponds to an effective utilisation of 90%. Actual runtimes after initial stabilisation are expected to be much higher.

17.4. PROCESS DESCRIPTION

A conventional three-stage crushing circuit will crush ROM ore to a P_{80} of 12.5 mm, at a rate of 13,500 t/d. Lime is added to the crushed ore before it is transported to the heap leach pad. Trucks will haul the crushed ore to the heap leach pad where it will be stacked in 7 m lifts.

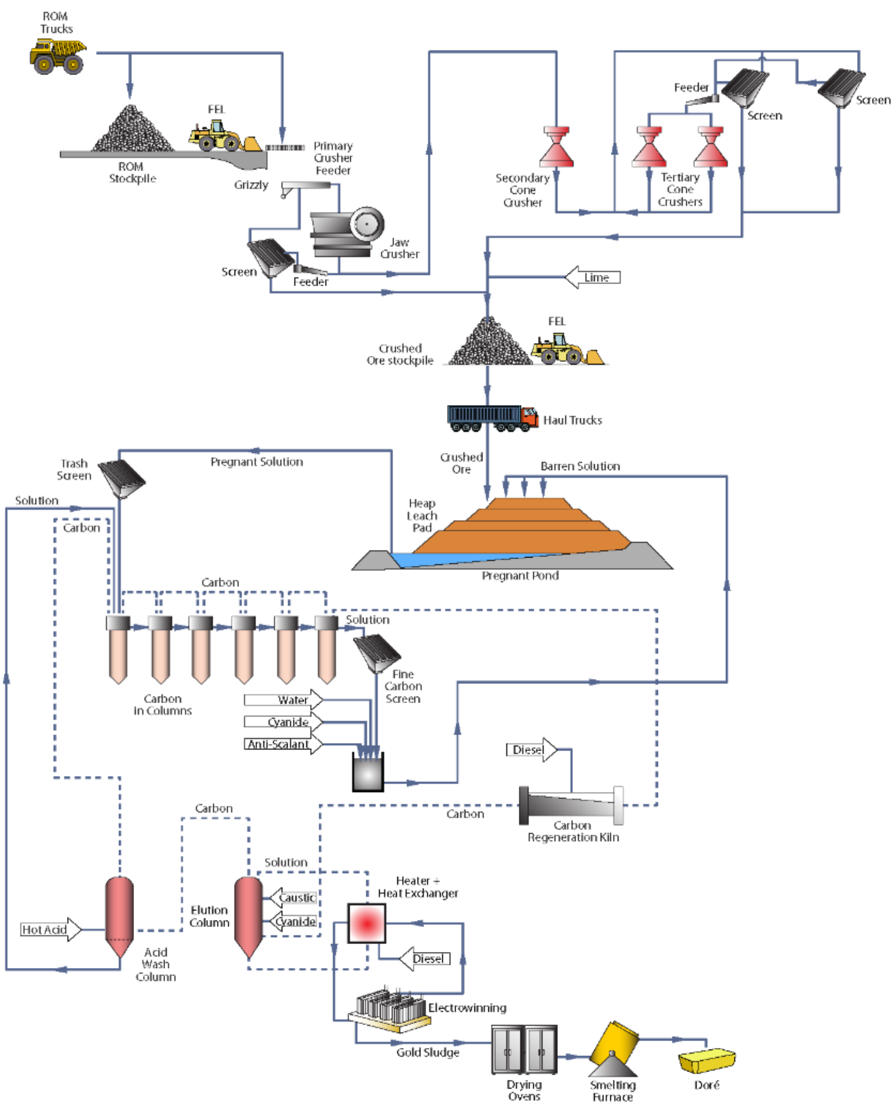
The prepared areas of ore on the heap leach will be irrigated with a dilute cyanide solution at a rate of 10 ℓ/m^2/h to dissolve the gold and silver from the ore into the solution. Once the solution percolates through to the base of the pad, it gravitates to the pregnant leach solution (PLS) pond. From there, it is gravity fed to the ADR plant for gold and silver recovery; however, a pump is used to begin the siphon process. The precious metals from the pregnant solution adsorb on to granular activated carbon in the CIC circuit ('Carbon in Column') of the ADR plant. After passing through the CIC tanks, the solution now depleted in gold (barren solution), is recirculated back to the heap leach pad, after being dosed with the required amount of make-up cyanide.

The loaded carbon is then pressure stripped (eluted) with a hot caustic solution to re-dissolve the precious metals into the pregnant solution. This pregnant solution is treated by conventional electrowinning to produce a gold-rich sludge suitable for direct smelting on site into gold Doré. The gold Doré bars produced are transported off-site to a suitable refinery.

At the end of its production life, the heap leach pad will be subjected to an extended water-rinse programme to ensure environmental compliance and to recover any residual precious metals.

Figure 17-1 shows a conceptual block flow diagram of the of proposed process plant. Process flow diagrams of the individual areas can be found in Appendix C.

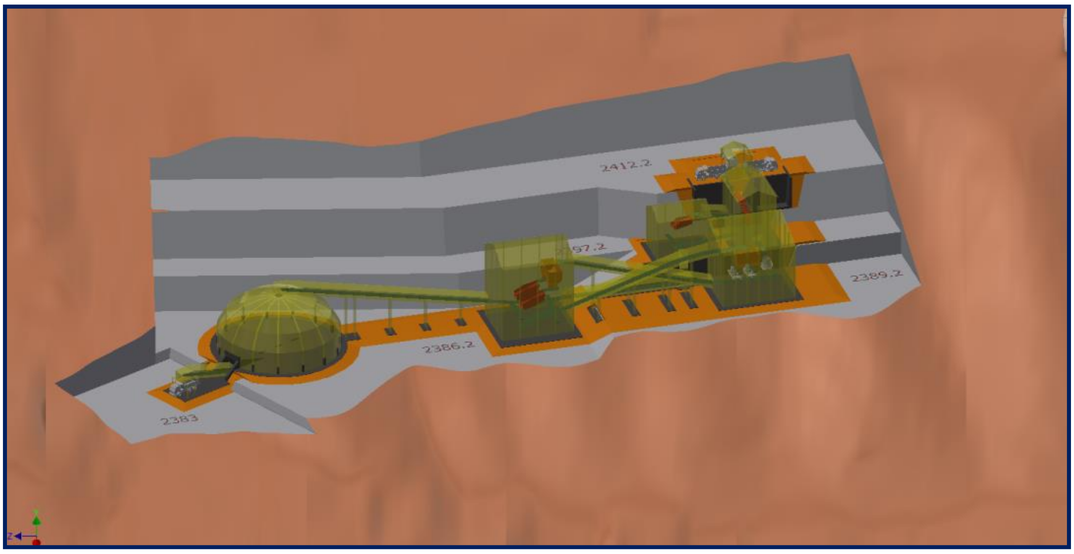
FIGURE 17-1 CONCEPTUAL BLOCK FLOW DIAGRAM



17.4.1. Crushing and Screening

A general view of the crushing circuit including screening and load-out is shown in Figure 17-2.

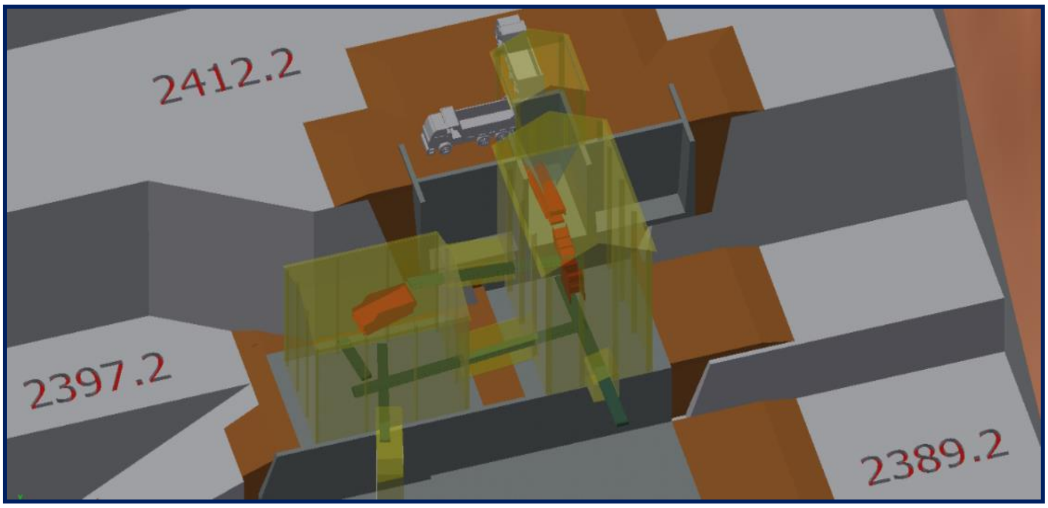
Figure 17-2 Crushing Circuit Overview



Run-of-Mine (ROM) ore is delivered by 30 tonne haul trucks to the primary crusher. The trucks dump on to 700 mm aperture stationary grizzly installed over the truck dump hopper. This hopper has a live capacity of 70 tonnes. Oversize rocks will be broken by a mobile rock breaker. Space has also been allowed for a future fixed rock breaker.

The hopper has two-sided access. One side will be in continuous use by trucks and the other will be for a mobile rock breaker and may be used by trucks if necessary. The primary crushing and bypass screen buildings are shown in Figure 17-3 below.

Figure 17-3 Primary Crushing Facility plus bypass screen



The ore is withdrawn from the dump hopper via an 1800x8800mm apron feeder, which supplies material to a 1.6 m x 4.5 m vibrating grizzly. The vibrating grizzly oversize is directed to a C150 jaw crusher, which reduces the rock size to 80% passing 180mm prior to being conveyed by the secondary cone crusher feed conveyor to the secondary crusher.

A self-cleaning belt magnet is installed over the conveyor belt which feeds the secondary crusher. The vibrating grizzly screen undersize is conveyed to the by-pass screen building. The bypass screen is a low head high “G” double deck screen with an upper deck aperture size of 35mm and bottom deck selected to produce a 14 mm separation, corresponding to the desired product size of 80% passing 12.5 mm. Both decks comprise modular panels made of abrasion resistant rubber.

Screen undersize (minus 12.5 mm material) is conveyed by a transfer conveyor to the tail end of the final product conveyor, which feeds the fine ore stockpile. Bypass screen oversize material (i.e. plus 12.5 mm) is conveyed back to the secondary crusher feed conveyor upstream of the belt magnet. Both primary crushing and by-pass screening take place in enclosed buildings.

Primary crusher product and by-pass screen oversize material are combined and fed directly to the secondary cone crusher. The secondary cone crusher discharge product is transported to the screen feed bin in the screen house building by the screen feed conveyor.

The screen feed bin supplies two 3 m x 7.2 m double deck banana screens which screen at a separation of 35 mm (top deck) and 15 mm (bottom deck). The oversize material from the product screens reports to the crusher feed conveyor via a transfer conveyor. This conveyor discharges into the tertiary crusher feed bin, which has a live capacity of 125 tonnes.

Two belt feeders installed under tertiary crusher feed bin supply two parallel tertiary cone crushers. Space has been allowed for a future third tertiary crusher. The discharge material from the tertiary cone crushers reports to the screen feed conveyor, where it combines with the secondary crusher discharge and delivers crushed ore to the product screen feed bin.

The secondary and tertiary crusher building is shown in Figure 17-4. A third tertiary crusher is shown, but this is ‘future’ and will not be installed initially.

FIGURE 17-4 SECONDARY AND TERTIARY CRUSHING FACILITY

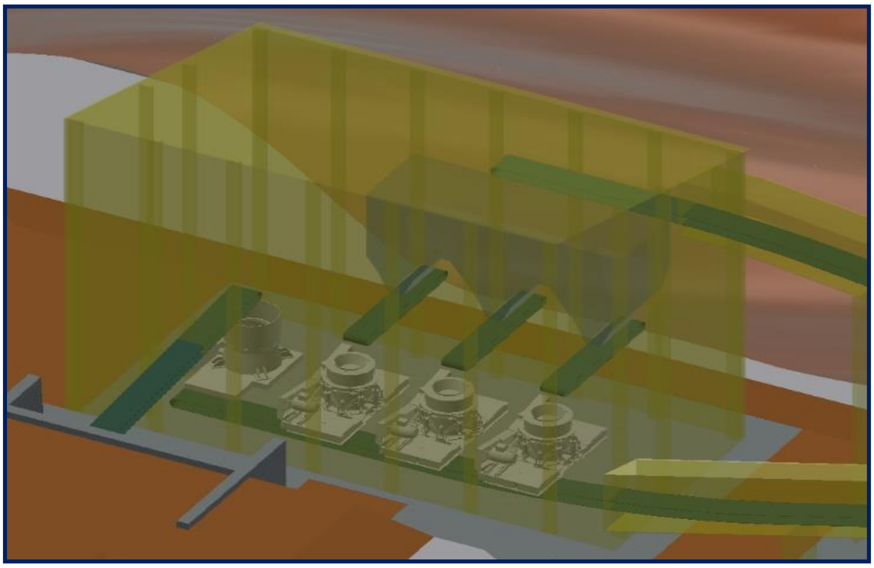
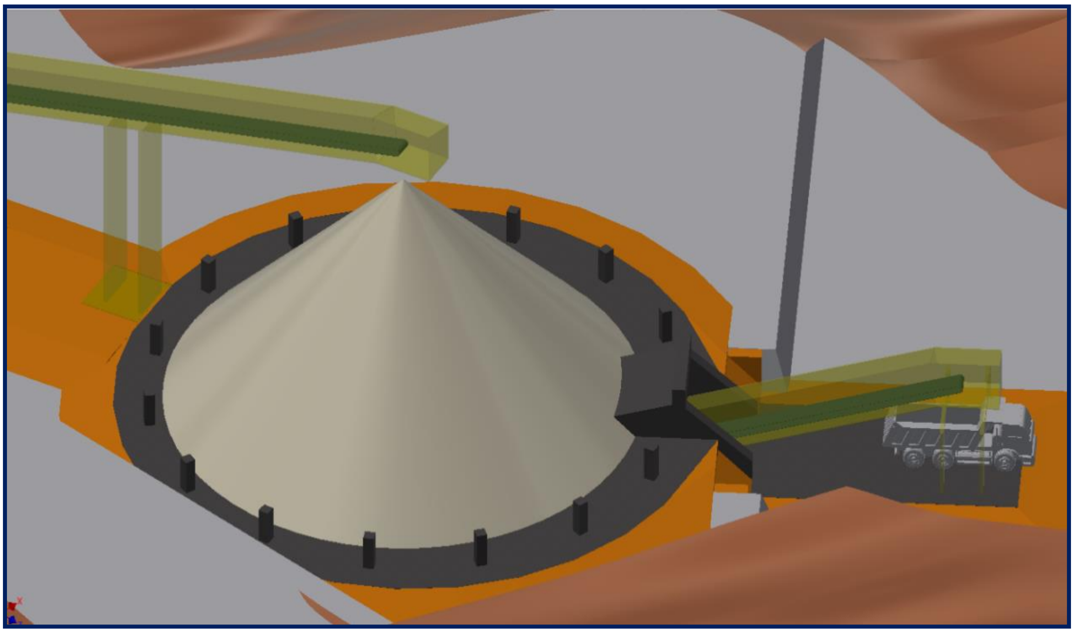


Figure 17-5 depicts the fine ore stockpile and load-out facilities.

FIGURE 17-5 TRUCK LOADOUT FACILITY



The fine ore stockpile is an open stockpile and it is designed for live capacity of approximately 10,000 tonnes. A reclaim tunnel underneath the stockpile is of a multi-plate steel culvert type construction. Three belt feeders in the tunnel withdraw material from the stockpile and discharge onto the truck loading conveyor belt. Truck loading will be controlled by the appropriate instrumentation.

Dust collection units are installed throughout the crushing and screening facilities, for example at the discharge of the crushers, in the screen building, and at transfer points of the crushing and screening conveyors. In addition, a separate dust suppression system will be provided at the ROM pad. A compressor will provide air to meet process and instrumentation requirements as well as maintenance tools. Raw water will be distributed where required.

The crushing, screening and ore handling facilities will be operated from an operator hut located in the primary crusher building and a control room situated in the secondary crusher building.

17.4.2. ORE STACKING

Crushed ore will be stacked on the heap leach pad using a combination of haul trucks and a bulldozer. The heap leach pad will be constructed in twelve 7 m lifts, to a maximum design height of 90 m.

The HLF has been designed to contain approximately 25.88 Mt with the potential to expand to 30 Mt. The heap leach pad has been designed in three phases. This is to suit operational requirements, to make optimum use of the summer construction windows, and to provide the opportunity for deferral of capital expenditures. Table 17-9 summarises the general operating design criteria.

Table 17-9 Summary of Process Design Criteria Relevant to HLF Design

Parameter	Unit	Design Input Value	Reference/ Notes
Climate			
Average Monthly Temperatures Below 0ºC	-	Nov.-Mar.	Ausenco
Average Annual Equivalent Rainfall	mm	470	Ausenco
Average Annual Evaporation	mm	532	Ausenco
Average Annual Wind Speed	m/s	2.2	Ausenco
General			
Total Stacked Ore	Mt	25.88	Chaarat
Ore Stacking Schedule	Days/year	350	Chaarat
Ore Leaching Schedule	Days/Year	365	Chaarat
Operational Life of HLF	Years	5	Chaarat
Closure Ore Rising	Years	1 to 2	Industry Standard
Nominal Ore Stacking Rate	t/d	13,500	Chaarat
Annual Ore Stacking	t	4,927,500	Chaarat
Average *in situ* Ore Density	t/m^3	1.75	Geo-Logic Assoc
ROM Moisture Content	%	3.8 – 4.7	Chaarat
Residual Moisture Content	%	6.6	Ausenco
Drain-down Moisture Content	%	5	Geo-Logic Assoc.
Ore Gradation	P_{80}	12.5	Chaarat
Maximum Fines Content	%	5	LogiProc
Leach Cycle time	d	60	Chaarat
Ore irrigation Rate - Nominal	$\ell/m^2/h$	10	Chaarat
- Maximum	$\ell/m^2/h$	15	Chaarat
Irrigation Area on HLF (maximum)	m^2	79,600	Calculated
Cyanide Solution Irrigation Rate	m^3/hr	796	Calculated
Pregnant Solution Inflow Rate to PLS Pond - Nominal	m^3/hr	796	Calculated
Pregnant Solution Inflow Rate to PLS Pond - Maximum	m^3/hr	1194	Calculated

Once stacking is complete, irrigation piping (‘drippers’) will be laid beneath the surface using pipe laying equipment mounted on the back of a dozer or tractor. The drippers will thus be covered by a sufficiently thick layer of crushed ore to prevent the solution in the pipes from freezing. Dilute cyanide leach solution will percolate through the ore dissolving the precious metals, and will collect above the liner at the base of the heap. The pregnant solution will report to the internal PLS pond by means of a solution collection system comprising of a network of dual wall perforated pipes to ASTM standards (ADS or equivalent). Vertical steel caissons at the toe of the PLS pond allow for collection of the pregnant solution by vertical turbine pumps for delivery to the CIC circuit.

More details about the collection system can be found in Section 18.5.2.8.8.

17.4.3. Stacking Plan

The ore stacking schedule for the heap leach pad has been designed in three Phases, with each Phase requiring advance expansion of the leach pad footprint. Each Phase requires pad foundation preparation, and the installation of the underdrain, geomembrane liner, solution collection, and overliner systems. The duration for each stacking Phase ranges from one to one and half years.

The Phase 1 development of the HLF will include the construction of the PLS Pond (part of the leach pad), the PLS Overflow Pond, the Emergency Pond, the Attenuation Pond, the Sediment Ponds, Underdrain Pond, perimeter access roads, and diversion channels prior to commencement of ore stacking and leaching. Table 17-10 presents the HLF phases and corresponding stacked tonnages.

Table 17-10 HLF Phases

Phase	Start (Year)	End (Year)	Total Period (Days)	Incremental Tonnes (Mt)	Accumulative tonnes (Mt)
1	0	1.33	480	6.48	6.48
2	1.33	3.45	762	10.29	16.77
3	3.45	5.33	675	9.11	25.88

Note Table 17-10 reflects the design capacity of the HLF (25.88 Mt) and not the operating plan (20.86 Mt).

The phased construction of the Heap Leach Facility is informed by several factors, including but not limited to:

- The benefit of distributing capital expenditure over the LoM – initial capital and deferred capital;
- The limited summer window for construction and installation;
- The undesirability of leaving liner exposed to weather; and
- The undesirability of having a large footprint of exposed liner to collect rain and snow melt.

Figure 17-6 presents a plan view of the HLF pad and ponds. All the main components of the HLF are present in this view – HL Pad, Attenuation pond, Diversion channels, PLS pond, PLS Overflow pond, Emergency pond, and Sedimentation ponds.

FIGURE 17-6 HLF PLAN VIEW

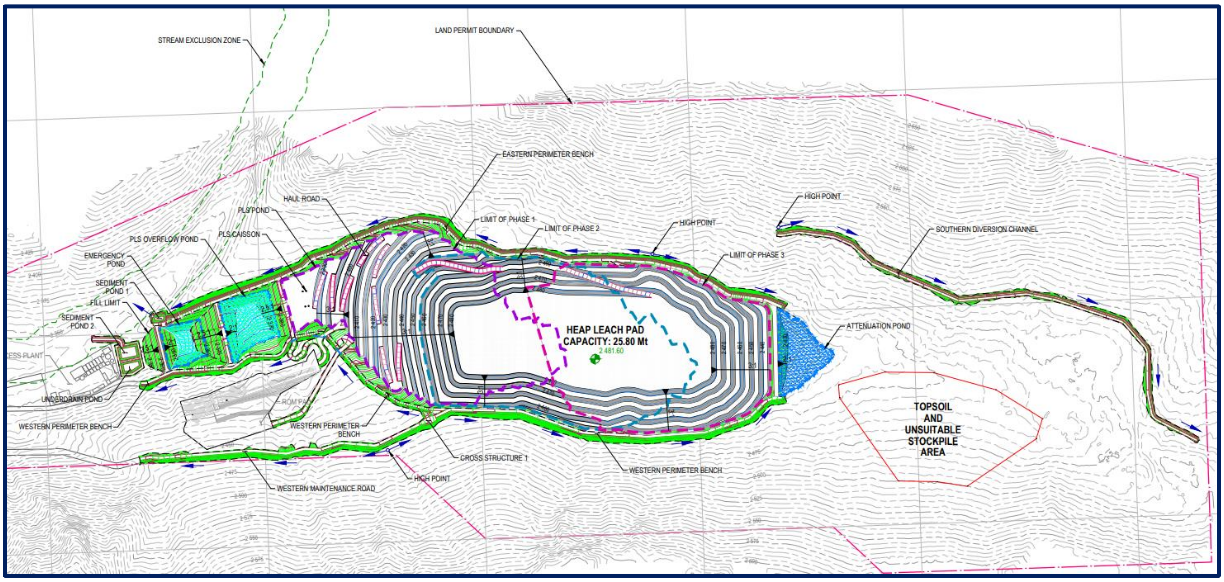
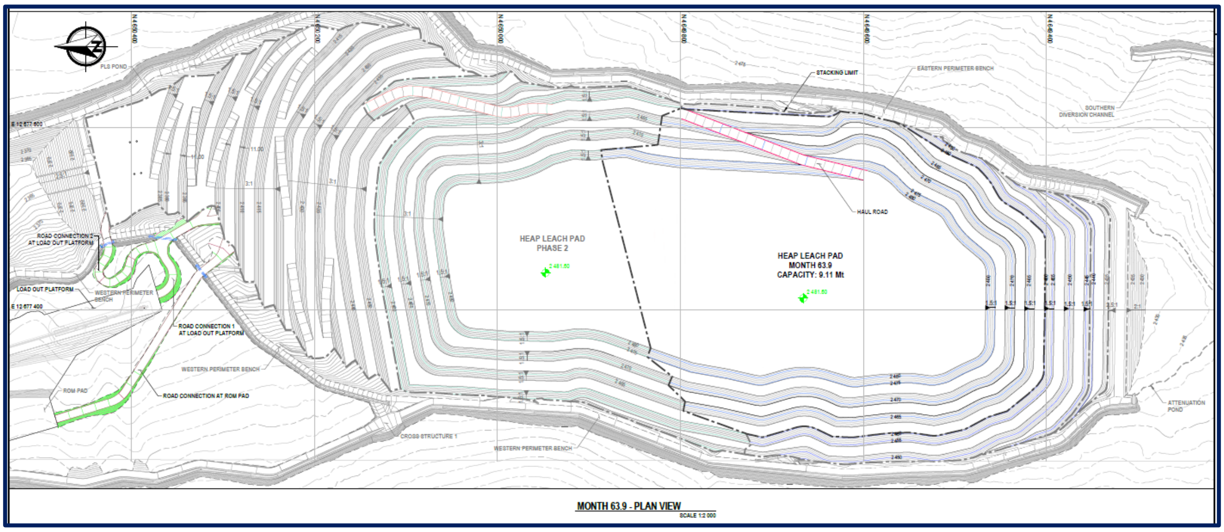


Figure 17-7 presents the staging and the development progress.

FIGURE 17-7 HLF PHASE DEVELOPMENT SHOWING FINAL LIFT



17.4.4. WATER BALANCE

An operational average monthly water balance analysis was undertaken for the leach pad and ponds using GoldSim software. The intent of the modelling was to estimate the magnitude and extent of any water surplus or deficit conditions in the HLF based on annual average climatic conditions. The modelling timeline was for 5 years of HLF operations (covering the 6-month ramp-up and 5.33 years of operations, consistent with the mine production schedule). The model incorporates the following major project components:

- Heap Leach Facility;
- Fresh water supply, and

- PLS, PLS Overflow, Emergency, and Attenuation Ponds.

The water balance analysis results indicate that the HLF will operate with a water deficit. The deficit is most pronounced in the early years and diminishes later in the operation, as the water stored within the ore is released from the earlier leaching phases. The total make-up required by the HLF ranges from 12,300 to 389,000 m^3 annually, and during the final years the site will be in surplus.

It may be noted also that the PLS Overflow Pond together with the Emergency Pond constitute a large storage capacity for spring rain and snow melt excess reporting to the PLS. In normal years, this source has the potential to provide the majority of HLF makeup water requirements during the dry summer months.

Table 17-11 and Table 17-12 shows the average results for the leach pad water balance and operational ponds water balance (PLS, PLS Overflow and Emergency Ponds) these results are the average of the 5 years using an average monthly total precipitation and evaporation.

Table 17-11 Leach Pad Balance results

Inflow	**(m^3/hr)**
Ore moisture	13.7
Leach flow	643.2
Precipitation	12.5
Outflow	**(m^3/hr)**
Moisture retained	21.9
PLS Flow	635.1
Evaporation	3.3
Change storage	9.1

Table 17-12 Pond Balance results

Inflow	**(m^3/hr)**
PLS moisture	635.1
Water demand	10.4
Precipitation	1.0
Outflow	**(m^3/hr)**
Leach Flow	643.2
Treatment Plant	0.0
Evaporation	0.3
Change storage	3.0

17.4.5. Collection Ponds

The collection ponds were sized to store the operational and drainage flows, and the excess water volumes predicted by the water balance calculations.

Preliminary design of the pregnant leach solution (PLS), emergency, PLS overflow, attenuation, two sediment, and underdrain ponds were undertaken based on the following:

- Location of ponds fixed based on valley topography;
- Pregnant solution pond dam to be completed during the initial construction phase;
- Geotechnical design of dams assumes that the ponds will be lined per the lining system specification;
- The design criteria summarised in Section 17.3 are implemented;
- The PLS pond will capture the PLS from the heap leach pad. Excess PLS will pass over the PLS pond embankment spillway into the PLS overflow pond. The PLS overflow pond will also capture rainfall or snow melt events that the PLS pond is unable to handle. The emergency pond will act as a tertiary storage if the PLS pond and PLS overflow pond reach maximum capacity; and
- The HLF will have diversion channels around the perimeter of the HLF to capture run-off water not associated with the external to the HLF. Run-off to the north will report to the sedimentation pond, whilst run-off to the South will report to the attenuation pond.

17.4.6. Surface Water Management

In general, the diversion channels and sedimentation ponds take care of run-off water and prevent ingress into the HLF. The ponds take care of the abnormal precipitation (generally Spring rain) and snow melt (also during Spring). Table 17-13 summarises the water management design criteria.

Table 17-13 Surface Water Management Summary

Parameter	Unit	Design Input Value	Reference/ Notes
Design Storm Return Period	1:year	200	Ausenco
Design Storm Duration	hr	24	Ausenco
Design Equivalent Rainfall	mm	105.6	Ausenco
Rainfall Intensity	mm/h	4.4	Ausenco
Minimum Channel Gradient	%	0.5	Ausenco
Channel Type	V-Notch/Trapezoid	Trapezoid	Ausenco
Channel Depth	m	Varies	Ausenco
Channel Bottom Width	m	Varies	Ausenco
Channel Side Slopes	H:V	1:1	Ausenco

17.4.7. Cold Weather Considerations

A review and comparison of heap leaching operations in cold climates indicates that year-round leaching is feasible. Design provisions have been incorporated to add and maintain heat in the process solutions applied to the heap.

The Project has adopted the following mitigation measures:

- Selection of an in-valley heap configuration to create a heat sink;
- Employment of an 'in-heap' solution pond for pregnant solution storage;
- Burying drip emitter lines (drippers) with up to 1 m of ore; and
- Provision of generators for backup power supply to pumps and emergency process equipment.

17.4.8. Cyanide Destruction

The HLF is designed as a contained, zero-discharge system, where leach solutions are maintained within a lined leach pad and pond areas. However, if a 200-year, 24-hour storm event is exceeded, the pond levels could exceed capacity. Normal operational procedure would be to pump additional solution back to the pad as a pre-emptive measure as the pond levels rise. However, when all normal avenues have been exhausted, the solution in the Emergency Pond will be treated with hydrogen peroxide from IBC's to neutralise any residual cyanide. Once the level of the cyanide is below the environmental regulation limit for safe discharge to the environment, the solution will be discharged.

17.4.9. Carbon Adsorption

The carbon adsorption section of the ADR plant comprises six up-flow, open-top, mild steel CIC columns installed at the same elevation. Each column contains 4 tonnes of activated carbon for the adsorption of precious metals from the heap leach pregnant solution. Pregnant solution from the PLS pond is delivered to the CIC adsorption columns at a design flow rate of 850 m^3/hr via a stationary trash screen to remove foreign material.

Pregnant solution will flow through the columns until the carbon contained in the first column achieves the required precious metal loading of approximately 2 800 g/t (based on an 'upgrade ratio' of approximately 2,300). The nominal daily carbon movement will be 8 t/d, which corresponds to two 4 tonne transfers per day. Each 4 tonne batch is acid washed followed by stripping. Wire samplers for continuous sampling of the pregnant and barren solution are installed.

The CIC columns are configured in a 'Carousel' arrangement instead of in the more common 'Cascade' arrangement. The solution is pumped between the columns, and the CIC column sequence is managed by operating the appropriate valves. As the carbon remains in each column until fully loaded, carbon wear is minimised. Loaded carbon is transferred twice per day from the column with the highest carbon loading to the acid wash vessel using an induced flow pump.

The carbon columns are designed for 50% carbon expansion at full capacity. The columns will only achieve 20% expansion at half flow, but because residence time and solution-carbon contact time double, the adsorption efficiency will still be high.

17.4.10. Carbon Acid Washing

An acid wash is required to remove inorganic fouling caused by calcium and magnesium thereby restoring the activity of the carbon. The loaded carbon slurry is pumped from the CIC columns to the acid wash vessel for descaling. The acid wash vessel is made from rubber lined carbon steel, and is fitted with internal strainers to drain the transfer water from the carbon contained in the acid vessel. The acid wash vessel holds 4 tonnes of carbon.

The acid used in the Acid Wash process to remove inorganic scaling from the carbon is Nitric Acid. Dilute acid will be circulated from the dilute acid wash tank upwards through the carbon in the acid wash column. A vent fan will be installed on the acid wash tank to vent acid fumes and carbon dioxide out of the building. The acid wash system is contained in its own bunded area and provided with a sump pump. Any spillage will be neutralized before being pumped to the carbon safety screen or the carbon sizing screen. Concentrated nitric acid will be pumped directly from an IBC container to the acid mix tank and a pH of between 0 and 2 is maintained to minimise acid consumption.

The acid washed carbon is then rinsed with transfer water and transferred hydraulically to the elution vessel.

17.4.11. Carbon Elution (Stripping)

The Split AARL process is used to remove gold from the loaded carbon. The elution system is sized to treat 4 tonne carbon batches twice per day to make a total of 8 tonnes of carbon per day. The elution circuit comprises of an elution tank, a lean tank, two pregnant tanks, a strip vessel, a diesel-fired strip solution heater and heat exchangers. Carbon elution takes place in a 304 stainless steel pressure vessel at 120°C.

One bed volume (BV) of strip solution (eluent) composed of fresh water with 3% sodium hydroxide and 1% sodium cyanide is pumped through the column at a nominal rate of 2 BV/h to the eluent tank while the solution is being heated. Once the solution in the column reaches 120°, the solution is transferred to the pregnant solution tank. 4 bed volumes of lean solution, heated to 120°C, passes through the column and into the pregnant solution tank.

Once the lean solution is complete, the column is rinsed with 4 bed volumes of heated (120°C) softened water which will then constitute the lean solution for the next elution. The column is then cooled with one bed volume of room temperature water, to reduce the temperature of the carbon for carbon transfer. Stripped carbon is then pumped from the elution vessel to the eluted carbon hopper in the carbon regeneration area.

Pressure in the system is maintained by means of a pressure control valve located on the eluate pipe downstream of the recovery heat exchanger. The pregnant solution is cooled to approximately 65°C in the recovery heat exchanger before being discharged into the pregnant or lean solution tanks.

17.4.12. Electrowinning

The pregnant solution from elution is pumped from the pregnant solution tank to a splitter box, where the flow will be split between two stainless steel electrowinning cells. The electrowinning cells are installed on the top floor of a civil goldroom to allow barren eluent to gravitate back to the pregnant tanks. The electrowinning cells are equipped with stainless steel mesh cathodes to allow for multiple use. Separate rectifiers power each cell with direct current (DC) up to 1,500 A at a voltage of between 3 V and 5 V.

A fan-induced ventilation system is installed above the cells. The loaded stainless-steel cathodes are cleaned every 3-4 days, depending on production, by lifting each cathode above a purpose-built wash tank to remove the precious metal rich product attached to the stainless-steel cathode mesh with a high-pressure water spray. The sludge resulting from cleaning is recovered from the sludge holding tank and filtered in a pressurised pot filter.

17.4.13. Smelting

The goldroom will be a civil building. The gold/silver rich cake from the pot filter will be dried in the drying oven. Dried gold/silver fines will be mixed with a combination of fluxes before being charged into the diesel fired tilting furnace. The charge is melted at 1,200°C to ensure separation between the metal and slag. The molten metal is poured into cascading bullion moulds with a slag pot at the end. The metal will solidify in the bullion moulds to form Doré bars, whilst the less dense slag will overflow into the slag pot.

The Doré bars will be cleaned using a bar cleaner before being sampled and transferred to a vault for later transport to the refinery. Slag arising from this operation can be retreated to recover entrained precious metal. This slag and other rejects (such as used crucibles) will be stored until required for re-processing.

17.4.14. Carbon Reactivation

Activated carbon can be fouled either by inorganic or organic substances. Inorganic fouling is removed by acid washing as described in Section 17.4.10, whilst organic fouling is removed in a diesel-fired carbon regeneration kiln, by heating the carbon in a non-oxidising atmosphere to volatilise the organics. Depending on the temperature employed, some of the carbon may also react with the steam produced from the residual moisture in the carbon, reactivating the carbon.

Since the organic fouling on carbon in a heap leach operation is significantly less than in CIL/CIP circuits, allowance has been made to regenerate only every second batch of carbon through a 200kg/hr kiln. Carbon will be transferred from the elution column into the kiln feed hopper, which is fitted with internal strainers and a dewatering screw feeder to remove any of the transfer water. The off-gas from the kiln will exit the building through two separate stacks and does not require induced draft fans as the natural draft created by the hot flue is sufficient.

Regenerated carbon from the kiln is discharged on to quench pan where it is continuously flushed with transfer water to quench the carbon before passing to a vibrating screen to remove carbon fines. The sized carbon will be held in the carbon holding tank for transfer using a induced flow pump and transfer water to the last CIC column in the train.

Fresh carbon will be agitated in a conical tank with the agitator located above the carbon sizing screen. This is done to detach fine carbon particles – a process known as ‘attritioning’. The attritioned carbon will be discharged on to the carbon sizing screen to remove the carbon fines generated. The system is fitted with its own electric hoist.

The carbon stripping and reactivation area will be provided with its own sump pump that will transfer any spillage to the quench pan.

17.5. Utilities and Reagents

Reagents will be delivered to site in 20 ft shipping containers. The shipping containers will be utilised as on-site storage where possible to limit the requirement for a reagent building on site. Containers will be handled with a site reach stacker. The quantities stored on site will meet strategic and operational requirements. Empty containers will be taken away by the supplier for reuse when full container deliveries are made.

The approximate numbers of containers required in the reagent storage areas are shown in Table 17-14.

Table 17-14 Number of Containers Required for Reagent Storage

Reagent	Package Type within Container	Strategic 10-day	Working 4-day	Empty	Total
Cyanide	1 t boxed bags	84	34	10	128
Lime	1 t bags	70	28	0	98
Anti-scalant	1 m³ IBC	1	1	1	3
NaOH	800 kg bulk bags	3	2	1	6
HNO_3	1 m³ IBC	9	6	1	15
Activated Carbon	25 kg bags on a pallet (40 bags per pallet)	1	1	1	3
Smelting Fluxes	25 kg bags on a pallet (40 bags per pallet)	-	-	-	1

Note IBC – intermediate bulk container

17.5.1. Power

Electricity will be supplied from on-site diesel generators. A more detailed description can be found in Section 18.5.4.

17.5.2. Fuel

Almost all equipment in the ADR area requiring energy supply for heat will be diesel fired – for example the carbon regeneration kiln, the elution heaters and the gold furnace. A day tank for diesel storage will be provided in the ADR building.

17.5.3. Water

17.5.3.1. Raw Water

A supply of raw water will be directly pumped from bores located near the ADR plant, adjacent to the Kumbeltash stream, through a buried pipeline to a dedicated 1000m^3 raw water tank. Raw water will be delivered to the processing facilities as required, principally as solution make-up for the heap leach operation. The raw water tank will be dual purpose with one suction nozzle located at the base of the tank and one half-way up. The upper section will supply raw water for process requirements, whilst the lower section will supply the fire water ring main in the ADR plant.

The raw water will be pumped by centrifugal pumps around the ADR area and reports back to the raw water tank. Raw water will be drawn from the ring main as required by the process. In addition to barren solution makeup, a significant consumer is the cyanide mixing. Continuous circulation of the raw water reduces the risk of freezing in winter, but other means to prevent freezing will also be required.

17.5.3.2. Potable Water

A small potable water facility may be installed in the ADR facility. However, the default strategy will be to transfer potable water daily by truck to the potable water located in the ADR area. Other areas in the process area that will require water are the laboratory, the administration building, the power station, and the site main gate.

17.5.4. Fire Water System

The fire water system will supply the ADR area only and will consist of the fire water section of the raw water tank, fire water pumps, fire water ring main and distribution points. An electric fire water pump, a diesel fire water pump and a jockey pump make up the fire water pump arrangement. Provision for fire hoses will be made at intervals around the ring main.

The lower section of the raw water tank will be dedicated to fire water. The suction spigot for the raw water pumps will be located part way up the raw water tank such that there will always be a dedicated volume of fire water in the tank.

17.5.5. Laboratory

A fully equipped laboratory will be available on site and will have the following separate sections:

- A sample preparation sector with space/bench area for sample receiving, drying ovens, size reduction equipment, and adequate bench space for the preparation of mine and geological samples;
- An assay laboratory with separate sample preparation and storage areas for samples sourced from different zones of the orebody to avoid cross contamination, fire assay equipment, scale room, chemical laboratory analysis, and chemical storage; and
- A metallurgical laboratory including pressure filters, leach columns, bottle roll leaching test facilities, and other miscellaneous metallurgical laboratory equipment as required.

The laboratory sample schedule can be seen in Table 17-15.

Table 17-15 Laboratory Sample Schedule

Type of Sample	Per Shift	Per Day	Per Week	Per Month
Grade Control	-	85	1,400	5,600
Crushed Ore	2	4	30	130
Heap Leach Pad Samples	2	4	30	130
Solution Samples (pregnant and barren)	2	4	30	130
Carbon Samples (pregnant and barren)	2	4	30	130
Smelter Slag Samples	2	4	30	130
Total Metallurgical Samples	10	20	150	650

17.5.6. CYANIDE

Cyanide will be delivered in bulk bags containing sodium cyanide briquettes. Cyanide bags are normally delivered in wooden crates in a shipping container, a number of which will be stored on a concrete platform, with suitable access for a forklift.

The cyanide system allows for preparation of cyanide from bulk bags in a 10 m³ (50 m³ for optional design) mixing tank, fitted with jet mixers. The solution is made up to a strength of 25% (m/m). The systems include an electric hoist and bag breaker with dust extraction and filtration. The prepared solution is pumped to a storage/dosing tank of 110 m³ (50 m³ for optional design) where the cyanide solution is pumped in a ring main to the various required locations. The cyanide area is bunded and fitted with a sump pump to return spillage to the mixing tank.

17.5.7. LIME

Burnt lime (CaO) will be used to adjust the pH of the irrigation solution for the HLF. Lime will be delivered in one tonne bulk-bags. A lime handling and storage facility will be installed over the fine ore stockpile feed conveyor or the load-out conveyor, whichever is most suitable. The facility will be sized to store approximately 3 days' supply. The facility will be equipped with a dust collection system, bag breaking system, and a discharge arrangement on to the conveyor. Screw feeders will add the required amount of lime onto the conveyor at a rate of 0.5 kg/t of ore.

17.5.8. SODIUM HYDROXIDE

The caustic system allows for preparation of NaOH solution from bulk bags (1 tonne) or 25 kg bags in a 10 m³ mixing tank, fitted with jet mixers. The systems include an electric hoist and bag breaker. The solution is made up at a strength of 20% to minimise the need for heat tracing of the tank, equipment and piping. The dilute solution is pumped directly from the mixing tank to the relevant areas since all consumers are on a batch basis and not for continuous use. The main users of sodium hydroxide are Elution, Acid Wash, Adsorption and cyanide mixing. If required, small amounts of sodium hydroxide can also be used for acid spill neutralization.

17.5.9. NITRIC ACID

Concentrated nitric acid (55-65%) will be delivered to site in 1000 ℓ intermediate bulk containers (IBC's). As required, an IBC will be transported into the process area by forklift truck and placed into a bunded area on the tiled platform next to the mixing tank. The acid will be pumped undiluted, directly from the IBC tank into the acid mixing tank as required.

The mixing tank will always be filled with water prior to acid addition, to achieve a target pH of 0 to 2. The diluted acid will be used to remove scale and other contaminants contained in the loaded carbon that would inhibit gold desorption in the elution step.

17.5.10. ANTI-SCALANT

Anti-scalant will be delivered to site in 1000 ℓ IBC containers and pumped undiluted at a controlled rate to the barren solution tank using a metering pump. The purpose of the anti-scalant is to retard the formation of calcium carbonate scale and other deposits that may foul the activated carbon or plug the system drip emitters. The consumption rate is projected to be 20 g/t of ore.

17.5.11. Activated Carbon

Fresh granular carbon will be delivered to site in 800 kg bulk bags. The fresh carbon will be added in the attritioning tank, and screened prior to transfer to the carbon storage tank. Carbon will be transferred to the adsorption circuit as required. Carbon consumption is projected to be approximately one inventory turnover per year, i.e. about 25 t/yr.

17.5.12. Smelting Fluxes

Flux material required in the smelting process will comprise a combination of sodium borate (borax), silica, sodium nitrate (nitre), and sodium carbonate (soda ash). Fluxes will be delivered in 25 kg bags. During smelting, the flux constituents combine with base metal oxides present to form silicates and borates in the slag, promoting a higher gold content in the Doré. It is estimated that approximately 150 g of flux per kilogram of dried gold and silver sludge will be required.

17.5.13. Hydrogen Peroxide

Hydrogen peroxide may be used to remove any cyanide contained in any solution which has overflowed from the heap into the emergency event pond before discharging it. The solution containing 50% by weight of hydrogen peroxide will be delivered to site in IBC containers. The hydrogen peroxide will remain in these containers until required. If required, the IBC container will be taken to the emergency event pond, and a sufficient amount of solution will be added to the pond water, to break down the any residual cyanide

17.6. Control Philosophy

In alignment with the underlying design criteria selected for the Tulkubash project and Processing Facilities, the control strategy for the processing facilities will be functional and minimalistic. An integrated control system will not be employed. Rather, each of the main functional areas of the process plant will employ its own stand-alone process control software.

A preferred platform for the crushing and ADR areas could be Siemens S7-1500 PLC's for areas and S7-1200 for field PLC's, together with TP1900 Comfort HMI's for operator control. Communication to a central location for data collection and processing would be provided.

An objective will be to integrate vendor software into the area packages as far as possible. This applies in particular to secondary and tertiary crusher operation but, in addition, interfaces with other vendor equipment in the Crushing and ADR areas will be necessary – such as the dust extraction, and regeneration kiln.

17.7. Process Plant Environmental Issues

An environmental discussion relating to the Project can be found in Section 20.

Section 17.7 outlines a summary of environmental issues from a process plant perspective.

17.7.1. Sodium Cyanide

The primary environmental focus of any Gold Processing Facility using cyanide as the leaching agent is sodium cyanide itself. Whilst Chaarat is not a signatory to the International Cyanide Management Code (ICMC), Chaarat has committed to follow the guidelines of the Code in several important respects.

- First and foremost, the design of the ADR facilities has been in accordance with ICMC code requirements;
- Cyanide will be sourced from ICMC compliant manufacturers;
- Transport of Cyanide will be to ICMC standards; and
- ICMC Procedures for Storage and Handling will be followed.

Supporting strategies include:

- Development of a cyanide management strategy as part of the site's environmental management plan for implementing best practice;
- Implementation of cyanide safety and management training for all personnel employed in areas where cyanide is used - including contractors;
- Implementation of safe procedures for cyanide handling including transport, storage, containment, use and disposal;
- Integration of the Site cyanide and water management plans;
- Implementation of Procedures for disposing of residual cyanide from bags and boxes; and
- Organisation of periodic third-party cyanide audits of the Tulkubash Processing Facilities and subsequent revision of cyanide management procedures where appropriate.

17.8. Process Plant Health and Safety

A health and safety policy will be drafted in accordance with the Kyrgyz Republic health and safety legislation.

Best practice safety management procedures will be established and employed throughout the operation.

Given the inherently hazardous nature of Adsorption, Desorption and Refining (ADR), and Assay Laboratory operations, special safety provisions will apply to these activities including but not limited to:

- Risks related to – Sodium Cyanide:
  - Restricted access for personnel wherever cyanide is used;
  - Special cyanide training for all personnel working in these areas;
  - Employment of portable hydrogen cyanide monitors for personnel working in high risk areas;
  - Employment of fixed hydrogen cyanide monitors in known high risk areas; and
  - Colour coding of all tanks, pipes, etc. containing cyanide.

- Risk related to – Temperature and pressure (Desorption area, gold refining):
    - Provision of special safety equipment;
    - Special Training; and
    - Hire of personnel with higher level of education.
- Risk related to – Mercury:
    - Whilst Mercury removal facilities have not been installed in the initial installation (analysis results have not indicated an immediate requirement), space has been provided to allow for future installation;
    - Mercury testing equipment will be purchased; and
    - Medical testing programme will be instituted for Gold Room employees.
- Risk related to – Lead (Laboratory):
    - Provision of Special safety equipment for fire assay;
    - Provision of suitable HVAC facilities; and
    - Routine medical testing for Lead for Fire Assay employees.

Supporting Strategies will include:

- Hazard and operability (HAZOP) studies for relevant equipment and processes during the design phase;
- Periodic risk assessments for process equipment as required during the operational phase; and
- Establishment of a procedure for the management and control of substances that are hazardous to health (COSHH).

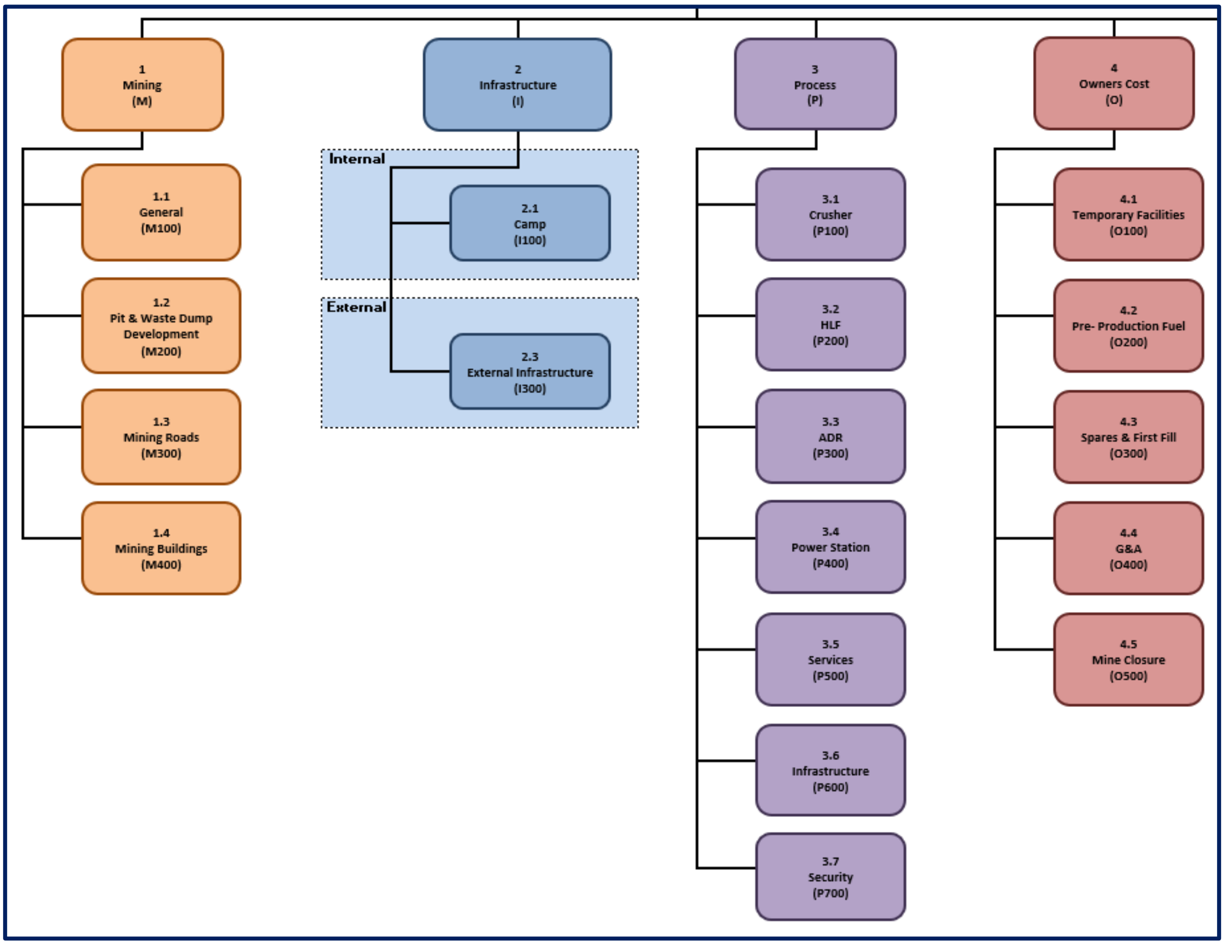
18. PROJECT INFRASTRUCTURE

18.1. OVERVIEW AND SITE LAYOUT

Infrastructure is generally understood as the services and utilities supporting a project, industry or country. However, in this document it will be used in a more general sense to cover all man-made interventions on the property on the Tulkubash site, including earthworks, utilities, buildings and equipment.

The locations of project facilities and other infrastructure items have been selected to take advantage of local topography, accommodate environmental considerations, and ensure efficient and convenient operation of the mine haul fleet.

FIGURE 18-1 PROJECT STRUCTURE SUMMARY



18.1.1. MINING COMPRISES -

18.1.1.1. MINING ROADS

- Roads and platforms;
- Haul roads;

18.1.1.2. Mining Buildings

- Explosives Storage;
- Ammonium Nitrate (AN) Storage; and
- Mine Maintenance Workshop

18.1.2. Infrastructure comprises -

18.1.2.1. Internal

- Accommodation Camp.

18.1.2.2. External

- Off-site infrastructure, including the Chatkal Station and Kumbel Pass checkpoint; and
- Site access road (Kumbel pass to Security Gate).

18.1.3. Processing Facilities comprises -

18.1.3.1. Crushing Facility

- Including ROM pad, primary crushing facility, secondary and tertiary crushing facility, and loadout station.

18.1.3.2. Heap Leach Facility (HLF)

- Comprising a heap leach pad; liner system with overliner drainage; catchment drains and underliner drainage; stormwater diversion channels; and pregnant leach solution pond (PLS), PLS overflow pond, emergency pond, attenuation ponds and sedimentation ponds.

18.1.3.3. ADR Pad

- ADR plant, electrowinning and goldroom, cyanide storage facility, reagent storage facility; and
- Support Services such as laboratory, clinic and administration building.

18.1.3.4. Power Supply

- Diesel-generator power station;
- Diesel fuel farm;
- Internal utilities; and
- Power distribution to all facilities via two 10kV feeder circuits.

18.1.3.5. Process Services

- Raw water, fire water, and potable water;
- Sewage;

18.1.3.6. Process Infrastructure

- Process buildings:
  - Warehouse/workshop; Site admin building, and laboratory.
- Process Roads:
  - Heap Leach (west, east, central);
  - Load out, Crusher Plant and Power Station, and ADR; and
  - Haul Road Extension.
- Process General:
  - Process IT and communications infrastructure; and
  - Tools and Equipment, special safety, and training.

18.1.3.7. Process Area Security

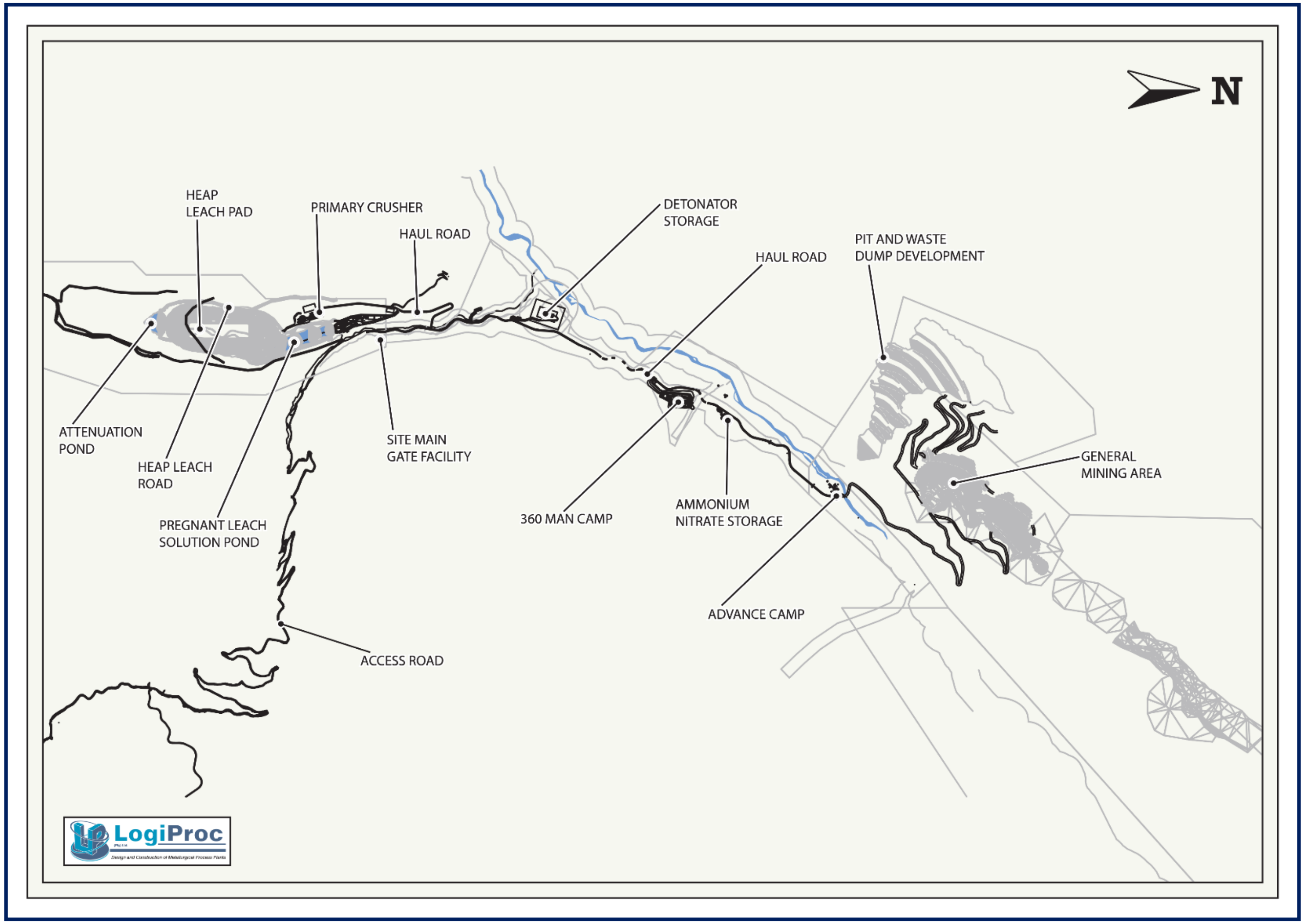
- Site gatehouse, ADR entry, and Gold Room.

18.1.4. 'Owner's' Facilities comprises -

- Temporary facilities:
  - Batch plant; and
  - Borrow Pits, Laydowns.
- Mobile Equipment; and
- Radio Communication.

The main components described above are shown on the general arrangement map in Figure 18-2 below (refer to Appendix E for additional GA drawings).

FIGURE 18-2 GENERAL SITE LAYOUT



18.1.5. Basis of Site Layout

Site layout is based on topography derived from a LIDAR Survey commissioned by Chaarat, publicly available satellite imagery, and where available, site observations.

The site facilities layout is informed primarily by:

- The site topography, geohazards such as avalanches and rockfalls,
- The 100 m water exclusion zone on either side of the Sandalash River
- The 50 m water exclusion zone on either side of the Kumbeltash stream, has been removed from the requirements, as noted in a July 2020 notification from the Kyrgyz Local Water Authority regulations.

All site facilities are situated between the Mine in the south and the HLF in the Dry Valley in the north. Limited areas exist on site which have a sufficiently low-geohazard level and are outside of the water exclusion zone, and on reasonably level terrain.

Consequently, many facilities and associated access roads have been positioned on the lower slopes of the surrounding mountains, necessitating substantial earthworks.

A detailed geohazard assessment was conducted in the dry valley. This assessment highlighted several areas of rockfall and avalanche risks around the HLF, process area, and ROM pad locations.

Key geohazard hazards identified within the local area include:

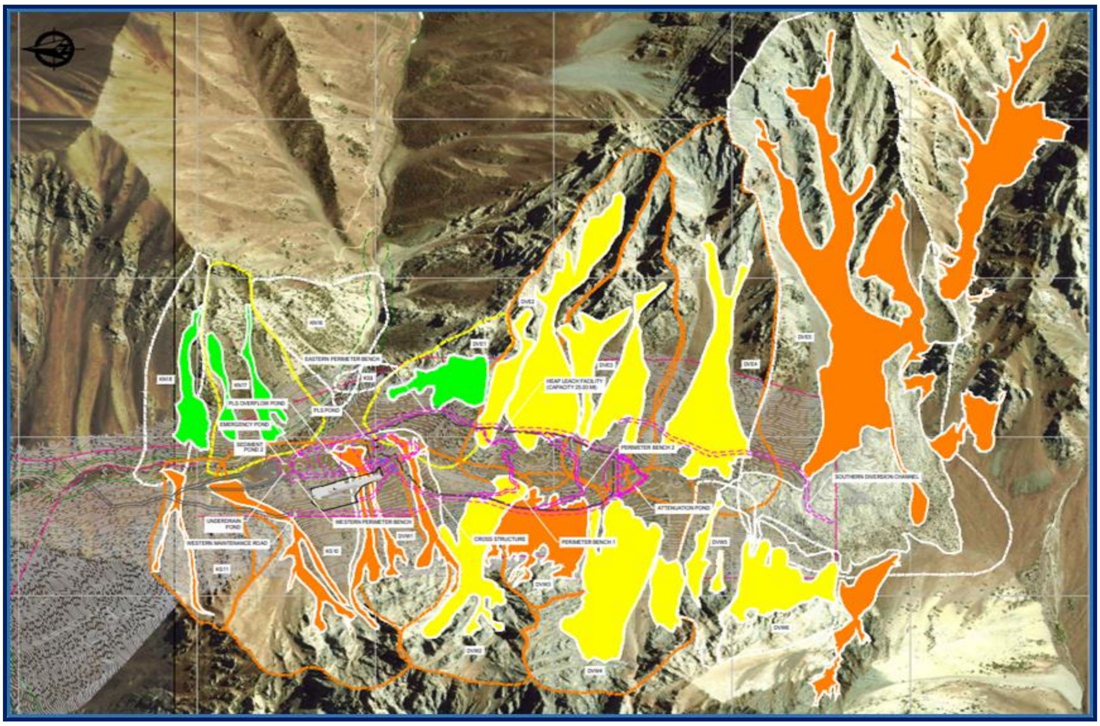
- Rockfalls from upper mountain slopes (rock crags), as isolated block fall events;
- Rock avalanches;
- Large-scale active/seasonal debris flows
- Snow avalanches, which include dry and wet snow avalanches, and slush flows; and
- Seasonal snow melt and stormwater run-off.

Significant geohazards to site infrastructure will be managed by:

- Avoiding the risk through the relocation of facilities;
- Preventing the risk by stabilizing the area that poses the danger, or
- Controlling the impact of the event through protective measures.

A drawing of the geohazards in the Process Area and proposed mitigation measures is shown in Figure 18-3 below.

FIGURE 18-3 PROCESS AREA GEOHAZARD RISK AND MITIGATION PROPOSAL



18.1.6. LOCATION OF FACILITIES

18.1.6.1. MINE

The open pit mine and Waste Rock Dump will be located at the north end of the site, on the eastern side of the Sandalash River, halfway up the valley side. A suitable location, near the permanent site of the mine maintenance workshop, has been identified as a safe zone to park trucks during blasting and shift change.

18.1.6.2. MINE SITE ROADS

The main mine road is the Mine Ore Haul Road extending from the Pit to the ROM pad at the Crushing Facility.

The East Pit Road from the 2019 BFS Update has been removed and replaced by 3 off sub-roads, being:

- Pit Road to Main Zone, constructed in Phase-1;
- Pit Road from Sandalash Bridge to Main Zone, constructed in Phase-2; and
- Pit Road to Waste Dump.

Other roads in the mining area are:

- The Ammonium Nitrate Platform Access Road;
- The Detonator Platform Access Road; and
- The Camp Platform Access Road.

Roads are not considered permanent structures with regards to the water exclusion regulations, so the routing of site roads makes use of the water exclusion zone in the valley floor, where required.

18.1.6.3. Mine Maintenance Workshops

A transitional facility will be established on the flat ground near the bridge below the open pit. This site is outside of the 100 m exclusion zone for the river and will be equipped with appropriate size of storage, tyre and welding shops and other infrastructures as required.

The permanent mine maintenance workshop will be located adjacent to the Waste Rock Dump, a minimum of 500 m from the open pit. This location eliminates the need for significant preparatory earthworks and long-term exposure to geohazards in the Sandalash Valley. The pad for the mine maintenance workshop will be constructed later in the project.

18.1.6.4. Detonator Storage

The location of the explosive's storage is limited by the requirement of a 500 m exclusion zone. The explosives storage will sit on a cut-and-filled pad on the eastern side of the Sandalash Valley, upstream of the confluence of the Sandalash River and Kumbeltash Stream. The geohazard assessment identified minor risks of rock fall and boulder debris, but the location is considered acceptable provided some mitigation measures are put in place.

18.1.6.5. Ammonium Nitrate Storage

Ammonium Nitrate (AN) is not considered explosive until it has been mixed with fuel oil. An explosive exclusion zone around the stores is not required. However, according to Kyrgyz regulations, AN must be stored separately from the detonator storage area, fenced, and guarded.

The AN storage will be located on the eastern side of the Sandalash Valley, upstream of the confluence of the Kumbeltash Stream and Sandalash River, accessible via a spur road off the main north-south haul road. The AN storage will be located on a cut-and-filled, bunded pad. The site, which is large enough for the storage of AN shipping containers, is located outside the river exclusion zone on the lower slopes of the valley.

18.1.6.6. Infrastructure

18.1.6.6.1. 360 Man Camp

The camp will be located in the Sandalash Valley, halfway between the open pit mine and the process area, to limit disturbances from the mine and process operations. Suitable locations for the camp were restricted due to several boulder gullies and avalanche risks. The high-level geohazard assessment concluded that while the camp is not in an ideal location, mitigation measures could be put in place to protect it, and therefore Chaarat has built several protection berms above the camp platform as per the specific recommendation of avalanche experts, based on regular site surveys.

18.1.6.6.2. Water Source

The borehole installation supplying the camp will be located adjacent to the Sandalash River as close as possible to the camp. Significant progress has been made with the on-site installation which is covered in Section 18.4.1.6.

18.1.6.7. Processing Facilities

18.1.6.7.1. Crusher

- Rom Pad
    - A trade-off study was conducted to assess the relative benefits of the ROM pad and crusher facility locations. The proposed location is identified as north, up the valley. The ROM pad location is selected halfway up the valley side to enable the topography to be utilised for dumping into the crusher feed bin.
- Crushing Facility
    - A trade-off study identified conventional crushing closer to the HLF as opposed to in-pit crushing (Appendix G). The primary crusher location is determined by the ROM pad location and the requirement to have the ROM bin as close as possible, and below, the ROM pad.
- Fine Ore Stockpile (Loadout)
    - A trade-off study showed it would be more advantageous to truck the ore over a greater distance rather than use an extended conveyor to transport the ore closer to the HLF (Appendix G).

18.1.6.7.2. HLF Area

- Heap Leach Facility
    - The Heap Leach Pad will be located at the southern end of the site in the dry valley. The dry valley was selected as the only suitable area for the HLF, with no active watercourses and enough reasonably level ground to build the heap. The location makes use of the valley sides and the slope of the valley floor to direct the pregnant leach solution via a series of drainage pipes to the vertical Caisson within the PLS pond area.
    - The location of the Heap Leach Pad determined the position of other related facilities such the PLS ponds, attenuation pond, emergency pond and sedimentation ponds, as well as the diversions channels.
- ADR Area
    - The process area pad will be located on the eastern side of the Kumbeltash Valley, running lengthwise along the contours of the valley side, between the outer perimeter of the water exclusion zone and the steeper section of the valley side. The exact location was selected to minimise geohazard risks.
    - Based on the topography of the site, the ADR area's proximity to the ponds allows for the efficient pumping arrangement of the PLS from the PLS pond to the ADR plant.

- Power Station and Fuel farm
    - As the power station will be the major user of fuel, the power station and fuel farm will be located adjacent to one another.
    - The power station is located within reasonable proximity to its main users - crusher and ADR.
    - The power station location is satisfactory with respect to geohazards.

18.1.6.7.3. SERVICES

- Water
    - Water for the processing facilities will be supplied from bores located near the plant and adjacent to the Kumbeltash stream. Studies have confirmed year-round supply. Significant on-site progress has been made with this installation as noted in Section 18.5.5.1.
    - The main water storage tank will be located in the ADR plant.

18.1.6.7.4. PROCESS INFRASTRUCTURE

Fabricated steel buildings will be located in the ADR Plant and in the crushing plant, which has four separate buildings – primary crusher building, bypass screen building, secondary and tertiary crusher building, and screen house.

Portable buildings will be used for the admin offices and clinic, the laboratory, the ADR offices, and the crusher offices.

The reagent storage building in the ADR area and the workshop/warehouse building may be portable or fixed depending on final Kyrgyz authority decision.

18.1.6.7.5. SECURITY

- Site Gatehouse
    - The gatehouse will be located at the point where the new Kumbel Pass road enters the mine site, close to the Process area. The gatehouse will be used as an assembly point in the event of an emergency, such as an avalanche in the Process or HLF areas.

18.1.6.7.6. OWNER

- The batch plant will be located in the Dry Valley.
- The mobile crusher will be positioned at the mine.
- The potential location of borrow pits is shown on the relevant HL drawing, Reference 103822-01-B-1107 Rev0 .

18.2. DESIGN BASIS

18.2.1. INTRODUCTION

The Mine is located in a seismically active area, in steep terrain, with high risk of geohazard events and with a severe climate. Therefore, the environmental conditions have a strong influence on the design.

Furthermore, for this feasibility study and this phase of the project, modest mine grades and mine life together with relatively unskilled workforce plus a minimum expatriate operational contingent mandate that –

- Processing facilities shall be technologically 'low to medium' and 'fit for purpose;
- Site communications and process control strategy shall be minimalist and limited in scope; and
- Whilst for example, the temporary mine workshop, Crusher and ADR buildings will be fabricated as enclosed structures, many of the other buildings on site will be prefabricated portable type buildings, delivered for example as 'flatpacks'. Exceptions may or may not include the process workshop/warehouse and the CN storage area, depending on regulatory requirements.

Obtaining Kyrgyz Government approval for the design of Site and Process facilities in common with all former Soviet Republics entails a process of 'Adaptation and Legalisation'. The success and timeliness of the process requires adherence to Russian and Kyrgyz design standards, including GOST National Standards, SNiP Construction Codes and Regulations, and PUE - the Electrical Installation Design Code.

The site's remote location and relatively long (and high risk) supply chain informs equipment selection criteria. In general, major critical equipment such as crushers, regen kiln, electrowinning cells, elution heaters, 'E-houses' and other major electrical equipment will be sourced from reputable international suppliers. Steel fabrications and other equipment will be sourced more cost effectively from countries such as Kazakhstan, Russia and Turkey.

Finally, an optimisation exercise ('Value Engineering') conducted after the initial BFS and basic engineering phase, generated some significant changes and improvements, notably –

- Optimising the location of facilities especially in the crusher area to provide the most cost-effective earthworks design;
- Opting for a slightly more robust crushing design to allow greater flexibility in ore feed and increased reliability;
- Establishing a more flexible, cost effective and operable ADR facility; and
- Reengineering the HLF to provide phased construction for operability and deferred capital.

18.2.2. CIVIL GEOTECHNICAL

Four phases of ground investigation were undertaken at the site by EcoServices (2010); KyrgyzGIIZ (2011); NK Group (2016); KyrgyzGIIZ (2017 and 2018).

The ground investigations focused mainly in the areas of the proposed site infrastructure: mine maintenance workshop, accommodation camp, AN storage, explosives storage, fuel farm and power station, ROM pad, loadout station, and the HLF.

Table 18-1 summarises the ground conditions from available site ground information.

The information in Table 18-1 was reviewed to determine the preliminary ground conditions of the site. The most relevant information for the site infrastructure is contained within the KyrgyzGIIZ (2017) report, which has the largest number of exploratory drillholes located at the proposed locations of infrastructure. The ground at the proposed site infrastructure locations comprises three different types of soils:

- Loam, classified as clayey silt;
- Gruss, classified as clayey, silty, sandy gravel; and
- Rubble ground, classified as clayey, silty, sandy gravel.

Loam material is generally encountered at surface to relatively shallow depths and will, in most cases, be removed during site preparation and topsoil stripping.

The gruss and the rubble ground generally classify as coarse soils, which are cohesionless. The foundation material for the site infrastructure will include these materials, and the preliminary bearing capacity assessment (Section 18.5.2) is based on typical strength parameters for these types of soils.

The groundwater level was generally encountered at depths greater than 6 m and in most cases much deeper.

TABLE 18-1 GROUND CONDITION INFORMATION

Author	Date	Document	Description	Relevant Information
EcoServices	2010	GI Report	Factual GI Report for the HLF	Boreholes and laboratory test results
KyrgyzGIIZ	2011	GI Report for Former Planned Processing Area	Factual GI Report for the area south of the dry valley	Borehole logs and report text, including summary of ranges of laboratory test results and soil parameter values
Kyrgyz National Academy of Science	2016	Report of Research for the Study of Mechanical Properties of Ground at Chaarat Mine Heap Leaching Facilities	Factual GI Report for the HLF	Report only
Kyrgyz National Academy of Science	2016	Report on the Determination of Ground Physical and Mechanical Properties for Heap Leach Pad, Dry Valley	Factual Report for the HLF, specifically to evaluate suitability for reuse as fill material	Report and partial field results only, plus borehole logs
NK Group	2016	Engineering Study of Chaarat Mine Facilities	Factual Report: RC and VES Seismic Survey	Tabulated geophysics results
WAI	2017	Design Report for Heap Leach Facility	DRAFT Feasibility Study by WAI	Relevant drawings and report extracts
KyrgyzGIIZ	2017	GI Report	Factual report for the 2017 HLF GI	Borehole logs and laboratory test results
GBM	2017	Borehole Location Plan, including Coordinates of all Previous GI On Site	Borehole Location Plan	Borehole locations
Chaarat	2017	Geological Map of the Area	Geological Map	Regional geology
KyrgyzGIIZ	2018	GI Report	Overall site additional GI works	Several new BHs and new SPT tests completed

It should be noted that due to the relocation of a number of the pads, some of the proposed locations for site infrastructure are not covered by the current suite of exploratory ground investigation. Further, no in situ testing, such as standard penetration tests or cone penetration tests, was undertaken during the ground investigation work. The assumptions relating to the allowable bearing pressure made in this report will need to be verified at the detailed design stage through further investigation work comprising the aforementioned tests or similar.

The civil geotechnical aspects of the Project will also need to be reviewed in the light of the detailed geohazard assessment.

18.2.2.1. BEARING CAPACITY

Table 18-2 shows the typical allowable bearing capacities of various square pad foundation sizes. The bearing capacities have been calculated based on the Brinch Hansen equation for shallow foundations. A factor of safety (FOS) of 3 was applied to the ultimate bearing resistance calculated from the Brinch Hansen equation to limit likely settlement below the foundation.

Strain softening (or loosening) of the soils due to earthquake loading was allowed for in the calculation by selecting reduced effective stress parameters. A preliminary information assessment indicates that liquefaction due to earthquake loading is unlikely to be an issue at this site. However, a detailed assessment is required at the detailed design stage and following the further investigation work previously discussed in this section.

TABLE 18-2 TYPICAL ALLOWABLE BEARING PRESSURE ON PAD FOUNDATIONS

Pad Size (m)	Allowable Bearing Pressure* (kPa)
1 by 1	271
2 by 2	275
3 by 3	297
4 by 4	324
5 by 5	352

*Note: *pads to be founded at least 1m below the surrounding ground level.*

The allowable bearing pressure on pad foundations detailed in Table 18-2 assumes that the pads are located on horizontal ground. This assumption remains valid where the pad foundations are located at a minimum of twice the width of the pad foundation away from the edge of a slope. For example, a 2 m by 2 m pad foundation located at least 4 m away from the crest of the sloping ground. Where this is not possible, a detailed bearing capacity assessment will be required at the detailed design stage.

18.2.2.2. CUT-AND-FILLED SLOPES

Cut-and-filled slopes are expected to be stable in the long term at slopes of 1(V):1(H) and for excavations less than 12 m deep 1(V):0.75(H). As legally required for the haul road excavations, construction of filled slopes should be undertaken in accordance with good earthworks practice as outlined in the earthworks specification. A detailed seismic stability assessment of cut-and-filled slopes has not been undertaken as part of the Feasibility Study and should be investigated during detail design. Furthermore, detailed slope stability calculations have to be completed during the detail design phase to finalize the slope cut methodology.

18.2.3. Civil and Structural

The minimum and maximum temperatures experienced on site are the main parameters that control the design slab foundations for the Project. Because there is an optimal temperature range for pouring concrete, site temperatures have the potential to both degrade concrete once poured and disrupt pouring schedules. Outside this temperature range, various remedial measures must be taken.

If the temperature at the time of pouring is expected to be below 5°C for three days or more, freezing and cracking will occur and produce unsatisfactory concrete in terms of consistency and strength. Remedies include the provision of windbreaks, blankets, or even heated enclosures. Utilizing heated enclosures is the most expensive option. If this option is chosen, care must be taken not to allow the concrete to cool too rapidly after the heating is removed as this can lead to cracking.

If the temperature at the time of pouring is over 25°C, cooling is required by providing sunshades or washing the aggregate in cold water prior to mixing. For extreme temperatures, the remedy is to reschedule pours to the early morning or late evening.

The following parameters control the design of the buildings and slab foundations on the Project. These are shown as pre-specified values and designer-chosen values.

18.2.3.1. Temperatures

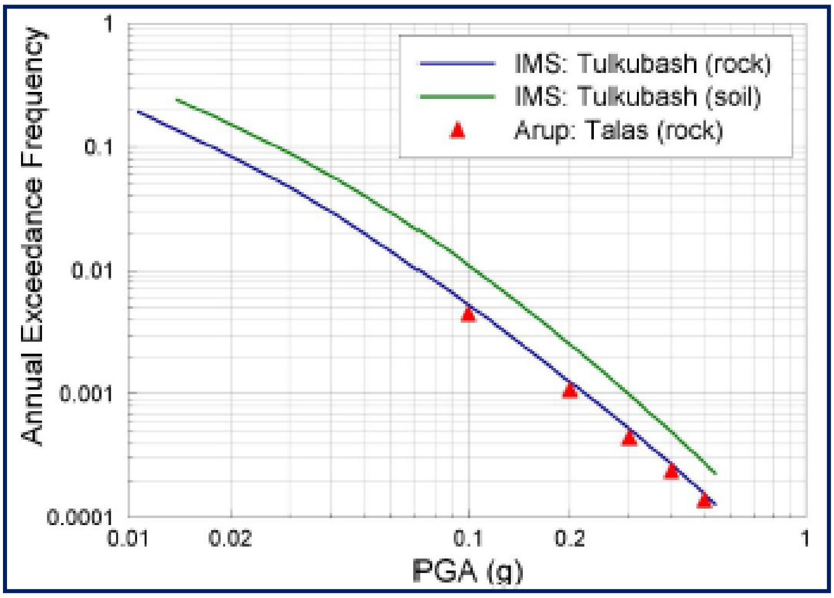
Design maximum/minimum	+38°C/–55°C
Design maximum/minimum indoor temperature	+35°C/+7°C

18.2.4. Seismicity

The Central Asian region in which the Tulkubash Gold Project is located is one of the world's most seismically active regions. Several seismic hazard assessments (SHA's) have been conducted, which cover the site of the proposed Tulkubash gold mine. Many of the studies have followed the now superseded practise of modelling the hazard in terms of macro-seismic intensity and are therefore unsuitable for quantifying the hazard in terms of peak ground acceleration (PGA) at the site because of the tenuous relationship between PGA and intensity. However, the results of those hazard studies all suggest that the official seismic zoning map for the Kyrgyz Republic is very conservative, at least for a gold mine site.

Among the hazard studies performed in terms of PGA, two are judged to be reliable probabilistic seismic hazard analyses (PSHA). One of these is a study undertaken by the Institute of Mine Seismology (IMS) for the Tulkubash site called Probabilistic Seismic Hazard Analysis for a Site in Kyrgyzstan (2017). The other is a study for Kyrgyz Republic called Measuring Seismic Risk in Kyrgyz Republic (2017) undertaken by the World Bank led by Arup & Partners with the participation of CAIAG, GEM and GFZ. The Arup study provides a hazard curve for rock sites in Talas, as the closest location to the mine site, for which hazard estimates appear comparable to those at the mine site itself. Neither study provides sufficient details of the hazard model nor includes systematic treatment of all uncertainties but the similarity of the results obtained (refer to Figure 18-4) suggest that these are reasonably stable and reliable estimates of the seismic hazard at this site.

FIGURE 18-4 SEISMIC HAZARD CURVES: ANNUAL EXCEEDANCE FREQUENCY VS. PGA



In view of the consistency of results between the two hazard studies and given the more complete nature of the hazard information provided by Malovichko & Calixto (2017), it is proposed that the IMS study was selected as the basis for inferring the return periods associated with the targeted PGA values. These results are summarised in Table 18-3.

TABLE 18-3 RETURN PERIOD FOR PGA VALUES ON ROCK AND SOIL

Return Period (yr)	Rock (G)	Soil (G)
100	0.073	0.106
475	0.16	0.23
975	0.22	0.31
2,475	0.33	0.45
4,750	0.45	0.60
10,000	0.60	0.76

None of the seismic hazard studies reviewed was a site-specific assessment of the Tulkubash site. A distinguishing feature of site-specific hazard studies, apart from the incorporation of local site conditions is the delineation with greater attention to the seismic sources in the area surrounding the site. The most important feature of such a local seismic source model will be the inclusion of geological faults as specific sources rather than only using area sources of diffuse activity.

For this feasibility study (FS), the IMS study was taken as the basis for estimating return periods and associated PGAs. A site-specific SHA was performed by Ausenco in 2019 which validated the values used in the 2019 BFS. If there are any significant deviations in the calculated PGAs, the design of the HLF will be adjusted to compensation for any potential minor impacts.

An estimation of the Maximum Credible Earthquake (MCE) by a deterministic method was performed in this section, considering the tectonic settings of the region developed by Arup. Figure 18-5 and Figure 18-6 present seismogenic sources and active faults of the Central Asia region. The seismogenic sources 3, 6, 7, 20, 21, 22 in the figure below and the Talas Ferghana Fault were selected in view of their proximity to the project location. Table 18-4 presents the results of this assessment. The MCE corresponds to a seismic event of M7.5 with its origin in source 22, obtaining a peak ground acceleration considering 84th percentile on the Tulkubash project of 0.49 G.

FIGURE 18-5 SEISMOGENIC SOURCES FOR THE KYRGYZ REPUBLIC (ARUP)

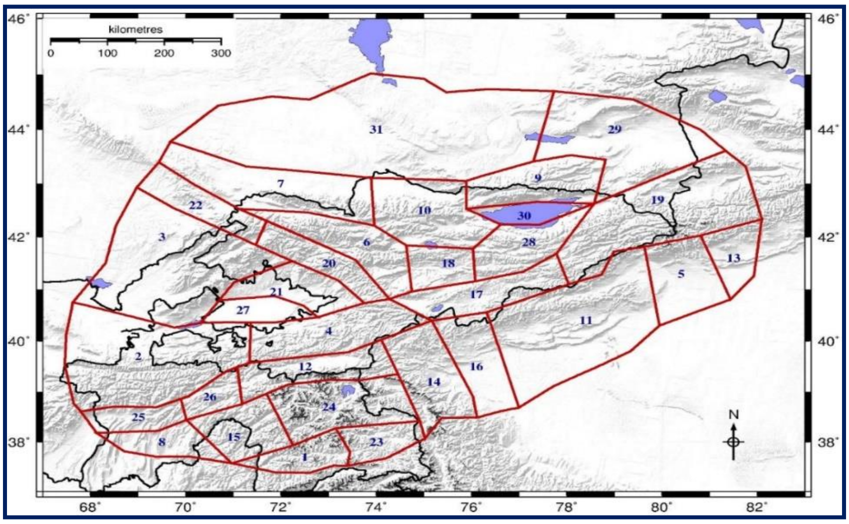


FIGURE 18-6 ACTIVE FAULTS PRESENTED IN ARUP

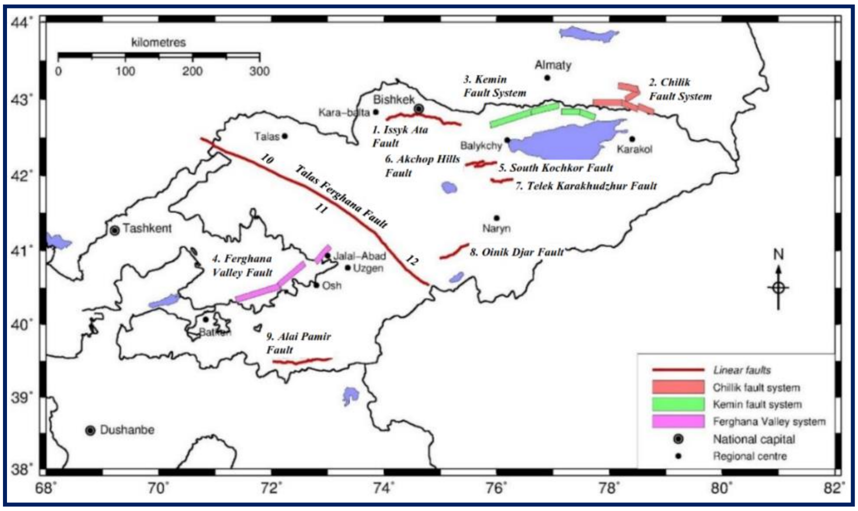


TABLE 18-4 MAXIMUM CREDIBLE EARTHQUAKE (MCE) – DETERMINISTIC ANALYSIS

Source Name/Number	Rx (km)	Rrup	PGA (P84)
22	0	25	0.49
3	8	26	0.42
20	34	42	0.27
6	44	50	0.36
7	48	54	0.34
21	81	85	0.15
Talas-Fergana Fault	28	26	0.40

Note: the following ground motion prediction equation equally weighted were used:

Abrahamson & Silva & Kamai 2014 NGA West-2 Model

Boore & Stewart & Seyhan & Atkinson 2014 NGA West-2 Model

Campbell & Bozorgnia 2014 NGA West-2 Model

Chiou & Youngs 2014 NGA West-2 Model

Rx: Horizontal distance to the source

Rrup: Rupture distance

18.2.5. GENERAL

18.2.5.1. SNOW

Maximum snowfall is 600 mm. A density value of 400 kg/m^3 is assumed.

18.2.5.2. ROADS

Site roads have been designed in accordance with the following general criteria that considers road use, vehicle type, and speed limit. Road-specific criteria are outlined in the following sub-sections.

- Maximum design speed: 30 km/h;
- Minimum sight distance: 15 m;
- Minimum road width for single-lane roads: two times the width of the largest vehicle using the road;
- Minimum road width for two-lane roads: three and a half times the width of the largest vehicle using the road;
- Minimum safety berm height: two thirds of the diameter of the largest wheel using the road with slopes of 1(V):1(H);
- Roads to be widened by 1.18 times on curves;
- Drainage ditches to be provided on either side of the road, with a minimum depth of 0.6 m, maximum side slope of 1(V):2(H) adjacent to the carriageway, and 1(V):1(H) on the external side;
- A maximum gradient of 10% where haul trucks will use the roads and up to 20% where light vehicles will use the roads;
- A maximum gradient of 4% on curves and 0% on switch back corners;

- A minimum camber (cross-fall) of 3%; and
- Cut-and-fill slopes, with a maximum slope of 1(V):1(H).

Table 18-5 Main Haul Road Specific Design Criteria

Criteria	Unit	Value
Largest Vehicle	-	Mercedes Actros 3340
Number of Traffic Lanes	-	2
Minimum Road Width	m	15
Minimum Safety Berm Height	m	1.3

The mined ore haul road to the ROM pad and the crushed ore haul road from the loadout station to the heap leach pad, have been designed with a width of 15 m, including berm and ditch, to accommodate dual-lane traffic for dump trucks in accordance with the mine plan description.

Table 18-6 Accommodation Camp Access Road Specific Design Criteria

Criteria	Unit	Value
Largest Vehicle	-	Mercedes Benz Actos (or similar)
Number of Traffic Lanes	-	1
Minimum Road Width	m	5.0
Minimum Safety Berm Height	m	0.75

Table 18-7 Power Station, Fuel Farm, AN and Explosive Storage Access Roads Specific Design Criteria

Criteria	Unit	Value
Largest Vehicle	-	Mercedes Benz Actos (or similar)
Number of Traffic Lanes	-	1
Minimum Road Width	m	5.0
Minimum Safety Berm Height	m	0.75

18.2.5.3. Ground Bearing Capacity

Ground bearing capacity is 300 kPa. Compaction will increase this value.

18.2.5.4. Designer Chosen Values

- Due to the potential funnelling effects in the valley, the design wind speed is 20 m/s;
- Unless more accurate values for equipment can be made, floor live loads are:
  - 20 kN/m^2 in the crushers;
  - 10 kN/m^2 where there are heavy wheel loads; and

- 5 kN/m^2 elsewhere.

- Concrete strength of C35;
- Rebar yield strength of 420 MPa; and
- Codes and standards will be Eurocodes or their equivalent.

18.2.5.5. LEGALISATION

The legalisation process represents a significant design factor and criterion. Procedures have been established to ensure that external designs are prepared in accordance with local and Russian norms and standards before they are submitted to the local Kyrgyz design Institute.

18.2.5.6. STRUCTURAL CONSIDERATIONS

It is envisaged that all steelwork structural connections will be bolted.

Design for operational, seismic, serviceability and any accidental load cases will be to a consistent set of design codes such as Eurocodes or ISO codes where different. These dictate recommended loadings and load combinations, plus consequential allowable stresses and deflections limits.

Typical primary steel shall be grade S355 or similar; with metric bolts of grade 8.8 or higher.

Typical standard live loads vary by location from 5 to 15 kPa, with more extreme values considered in the area of the crushers (a minimum of 20 kPa).

Typical allowable deflections as a function of member length vary between 1/90 and 1/800 depending on the application.

All steel fabrications for the crusher and ADR will be sourced from neighbouring countries such as Kazakhstan to minimise transportation costs.

18.2.5.7. ARCHITECTURAL

The architectural design basis for the facilities within the current scope are as follows:

- Modular steel-framed clad buildings will be provided for major processing area buildings – ADR plant building, Primary, Bypass, Secondary, Screenhouse, Mine Maintenance Workshop;
- The detonator store will be located in containers;
- Warehouse/workshop will be containerised or steel according to Kyrgyz requirements;
- Insulated cladding will be used where applicable e.g. ADR building;
- Roll-over metal doors suitable for vehicle entry will be provided for steel buildings; and
- External personnel access doors will be provided as appropriate, as well as for internal partitioning.

18.2.6. ELECTRICAL AND INSTRUMENTATION

18.2.6.1. POWER STATION

Electrical power generation will be on site using stand-alone, diesel-powered generators. The power station will be a modular, containerised design with sufficient redundancy to obtain at least 99.5% runtime.

The gensets will be derated for altitude, and continuous operation.

Power will be generated at 400 V, 50 Hz, 3 Phase. Each generator will have an individual step up 0.4/10 kV transformer which will connect into a common 10 kV switchboard for medium voltage distribution at 10 kV.

Diesel fuel will be reticulated to the power station from the on-site fuel farm.

18.2.6.2. MEDIUM-VOLTAGE POWER DISTRIBUTION

Main power will be supplied to the following users via a 10kV ring main from the Power Station

- The Main Gate Area;
- Kumbeltash bores;
- The Crushing and load-out facility;
- The ADR Facility;
- The Office and lab area; and
- The Heap Leach Return Pumping Station.

All facility loads will be supplied from individual areas. Each area will be fed either by a packaged substation/switchgear/transformer unit that will step down the voltage from 10 kV to 400 V for low-voltage distribution.

The 10 kV distribution system will be a cable system. Cables will be installed on a cable ladder/tray and utilize conveyor, building, and plant support structures for reticulation, where possible.

18.2.6.3. LOW-VOLTAGE POWER DISTRIBUTION

Low-voltage power distribution will be at 400 V. Feeders for the packaged substations will supply individual motor control centres (MCCs) and distribution boards (DBs) in the load areas. MCCs and DBs will supply and control the individual items of process area.

18.2.6.4. CONTROL SYSTEM AND COMMUNICATION

Site will be divided into 2 control areas that can run independently of each other - Material handling and ADR (including water). The system will consist of PLC's and HMI's/ SCADA. As the project is relatively small, the cost and complexity of a DCS package is not warranted.

For surface communications, a mobile hand-held radio system and/or mobile phone communication will be utilised.

Cellular communication will be made available on site by others.

18.2.6.5. Instrumentation

Electrical supply for field instruments, relays, solenoids, control systems, panels, and back of panel instruments, where required, will meet the design requirements of the instruments. A supply of 24 V is preferred.

Instrument brands will be limited and standardised as far as possible. Only established brands will be used.

All field electrical instrumentation will be wired to field mounted junction boxes. Single pair cables will be used between field junction boxes and individual instruments or electrical devices. Multi-conductor cables will be used between the field junction boxes and the control system input/output panels.

18.2.7. Mechanical

18.2.7.1. Process Equipment

Critical equipment such as crushers, barren solution pumps, regeneration kiln, electrowinning cells shall be obtained from reputable international manufacturers. Other equipment may be sourced more cost effectively and nearer to Kyrgyz Republic – Kazakhstan, Russia, and Turkey.

As far as possible, equipment has been selected to have maximum dimensions of 12 m (L) x 3 m (W) x 2.6 m (H), or the ability to be broken down into subcomponents of less 12 m (L) x 3 m (W) x 2.6 m (H), being standard container dimensions

18.2.7.2. Pumping, Piping and utilities

Piping material has been selected to be practical and economical, based on application, operating conditions, climate conditions, and industry common practice. Typically, high-density polyethylene (HDPE) pipe will be used for most solution systems and stainless-steel pipes will be used in the corrosive reagent systems.

All pipe sizes are expressed in nominal diameter (DN) using the SI unit (metric) system. Piping has been sized for designed flowrates and conditions stated on the PFDs (Appendix C). Non-standard nominal diameters (32, 65, 90, 125, 175, 225, 325, 550, 800 and 850 mm) have not been used, except where required to connect to equipment.

All piping has been designed to be self-draining. Drain and vent connections will be provided at low and high points of the piping system, to facilitate maintenance and hydro testing, as well as process requirements.

Provision to protect against freezing will be made where necessary, by burying, insulation or heat tracing.

Fire water supply and pump flow shall be in accordance with the applicable fire protection codes. Fire water will be supplied using a combined raw/fire water tank and a back-up fire water tank.

18.2.8. Buildings And Related

Steel fabricated buildings will be prefabricated in Kazakhstan.

Structural steel will be prefabricated to minimise construction work on site. Pre-engineering buildings may be used where cost effective.

The site infrastructure design basis includes the following:

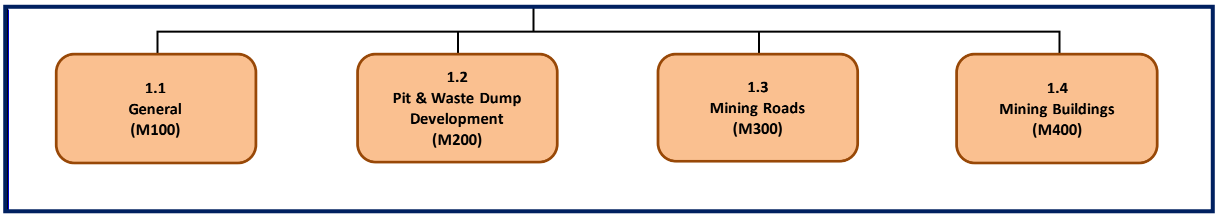
- Major process buildings will be clad steel fabricated structures – ADR;
- As far as possible all other buildings will be modularised, and preferably containerised i.e. all offices, laboratory, security, site main gate;
- The gold room will be concrete and brick;
- The cyanide storage building and workshop/warehouse may be steel or modular portable depending on legalisation requirements;
- Buildings will be installed on concrete foundations;
- Parking and outdoor storage facilities will be installed on compacted hard stands;
- Buildings will use cladding and if appropriate insulation as well, e.g. ADR;
- Overhead cranes will be provided in the crusher buildings for maintenance; and
- Roller doors will be used to provide access and ventilation.

The design basis for the site utilities includes the following:

- Power generation has been de-rated for altitude and continuous operation;
- The power station has been sized for high runtime (>99.5%);
- The fuel farm has been sized for 10-days reserve capacity and 4-days operational capacity; and
- Water will be sourced from the near available bore locations.

18.3. MINE

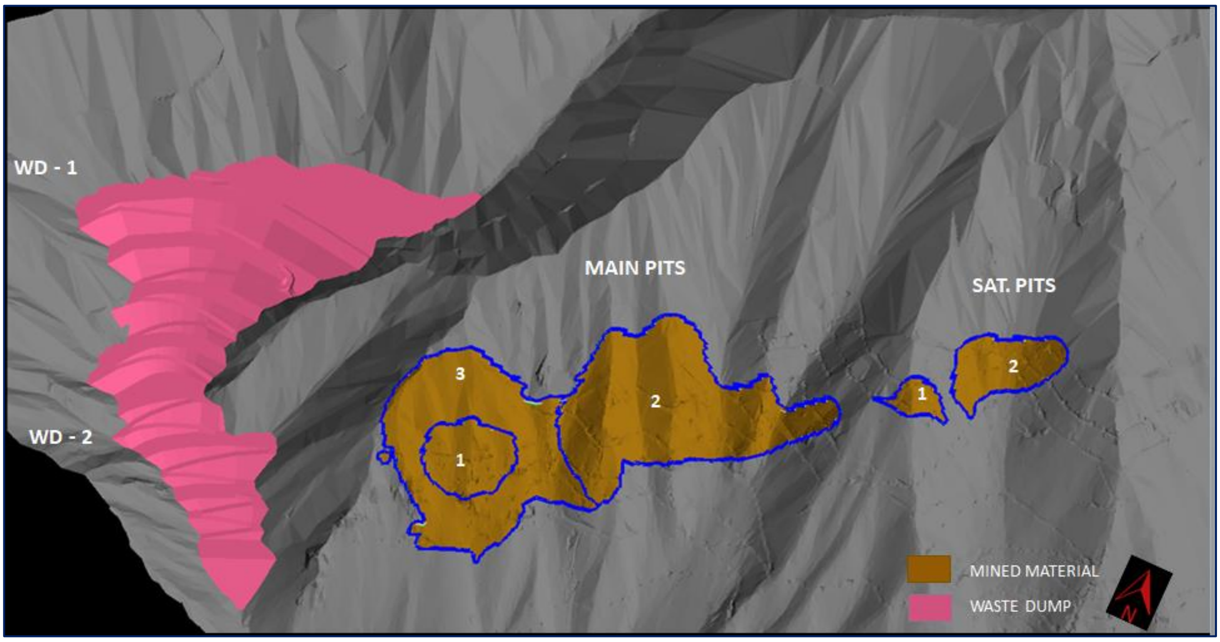
FIGURE 18-7 MINE EXTRACT FROM FIGURE 18-1



18.3.1. PIT AND WASTE ROCK DUMP

The mining contractor will be employed to undertake all mining activities. These include drill, blast, load, haul, support, and supervision. In addition, the mining contractor will provide equipment and services enabling the Owner to manage the site and main access road and to haul crushed ore from the loadout station to the heap leach pad. The mining contractor will perform all vehicle and mobile equipment maintenance on site for both his fleet and vehicles belonging to the Owner. The general layout of the Mine is shown in Figure 18-8.

FIGURE 18-8 MINE AREA LAYOUT



18.3.2. WASTE ROCK DUMP(WRD)

The mining contractor will develop, manage and operate the WRD that will be located approximately 1 km south west of the main pit.

The WRD has been designed to accommodate waste rock according to the production and mining schedules. Provision has been made in the pre-stripping budget for the preparation of the WRD. This will consist of clearing and grubbing, topsoil removal, unsuitable soil removal, low permeable soil placement in low lying areas, and ripping and compacting to create a firm and dense platform to dump the waste rock.

The principle considerations for the WRD Feasibility Study design include minimising infiltration into the foundation, controlling surface run-off, and location of the WRD to reduce fuel consumption by reducing haul times and distances.

18.3.3. ORE STOCKPILE

The Ore stockpile will be located near the current summer camp site as described in Section 16.

Provision for Preparation of the Ore stockpile has been allowed for in the pre-stripping budget. Preparation will be completed by the mining contractor and will consist of clearing and grubbing; topsoil removal; unsuitable soil removal; low permeable soil placement; and ripping, moisture conditioning, and compacting. Once all organic material, topsoil, and unsuitable material has been removed and any low points have been filled with low permeability material, the foundation will be shaped to promote adequate drainage towards designated low points.

18.3.3.1. Advance Camp Area

The advance camp area will be prepared for the following uses (pre-strip budget):

- Storage of topsoil;
- Location of Ore stockpile;
- Temporary location of mobile crusher; and
- Temporary storage of overliner material if required.

18.3.4. Mine Roads

The main mine roads outside of the pits themselves are the:

- Main Haul Road;
- AN Pad Access Road;
- Detonator Pad Access Road; and
- Accommodation Block Access Road.

There are three pit roads namely:

- Pit Road to Main Zone, constructed in Phase-1
- Pit Road from Sandalash Bridge to Main Zone, constructed in Phase-2
- Pit Road to Waste Dump

Two water filling stations for haul road dust suppression will be installed near the main camp.

18.3.4.1. Main Haul Road

The main haul road will link the open pits to the ROM pad. The location of the road has been selected to minimise the associated cut-and-fill earthworks within reason noting the variable topography of the valley. The road is located on the south side of the Sandalash River and generally follows the river alignment in a south-westerly direction until it meets the Kumbeltash Stream where there is a crossing to the proposed ROM pad location. Vehicles using the crusher haul road to the ROM pad will be Mercedes-Benz Actros 3340 30 t trucks (Table 18-5).

Haul road construction was commenced in May 2019, with 606,000 m^3 of excavation and 84,000 m^3 of backfilling and compaction being completed. Additionally, 81,000 m^3 of topsoil stripping has been completed. The total progress of haul road earthworks is 71%.

18.3.4.2. Pit Roads

The mine to be constructed during 2023 are as follows:

- The Pit Road to Main Zone is located in the main pit and 99,225 m^3 of excavation and 58,021 m^3 of backfilling is required.
- Pit Road from Sandalash Bridge to Main Zone is located optimally to connect the main pit to the Sandalash Bridge. The majority of this road can be constructed without blasting, therefore limiting the amount of backfilling required. 340,281 m^3 of excavation is required white 189,874 m^3 of backfilling

is required for the portion of the road that needs to undergo blasting to be constructed.

- Pit Road to Waste Dump connects the main pit to the waste dump. 64,238 m^3 of excavation and 70,384 m^3 of backfilling is required.

18.3.4.3. Ammonium Nitrate Pad Access Road

The ammonuim nitrate access road connects the ammonium nitrate pad with the main haul road. Delivery trucks and light vehicles will use this road.

18.3.4.4. Detonator Pad Access Road

The detonator storage access road connects the detonator pad with the main haul road. Delivery trucks and light vehicles will use this road.

18.3.4.5. Accommodation Block Access Road

The access road to the accommodation block will be mainly traversed by light vehicles and personnel buses. Occasionally delivery trucks will require access to the accommodation block which are anticipated to be the largest vehicles to traverse the road. The access road to the accommodation block is anticipated to be low volume and therefore a single lane road is proposed (Table 18-6).

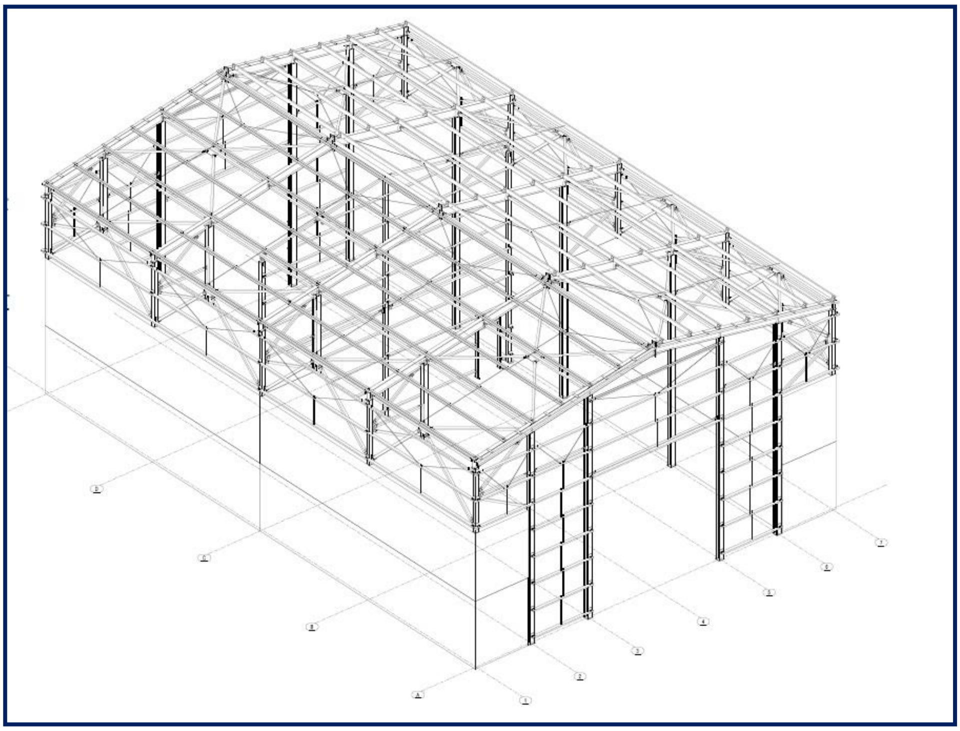
18.3.4.6. Mine Area Buildings

18.3.4.6.1. Mine Maintenance Workshop

The mine maintenance workshop will be provided and operated by the mining contractor. Initially, a temporary maintenance building (see Figure 18-9 below), will be constructed on flat ground by the Sandalash River crossing. It will include a warehouse, car wash facility, tire workshop and welding workshop.

A permanent facility may be constructed adjacent to the WRD in the mining area. The workshops will be for repair and service of mobile equipment and vehicles. A covered wash pad, welding shop, parts storage area, and offices will be located nearby.

FIGURE 18-9 TEMPORARY MINE MAINTENANCE WORKSHOP BUILDING



18.3.4.7. DETONATOR BUILDINGS

The detonators and explosives are stored on the same platform in separate structures.

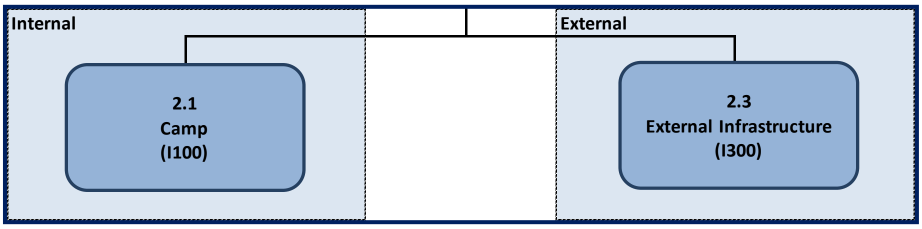
The detonator storage facility will be constructed using portable buildings or containers. The storage area will be constructed by the Mining Contractor in collaboration with the explosive's supplier. The detonator storage facility will have some additional infrastructure such as office containers.

The explosives storage facility will be contained within a high-security fence and an exclusion zone. The access will be via boom gate which will be manned by security guards. Only authorised personnel will be permitted to enter the facility.

The earthworks for the detonator storage facility has commenced, with 36,500m^3, being moved. The platform is 75% complete.

18.4. INFRASTRUCTURE

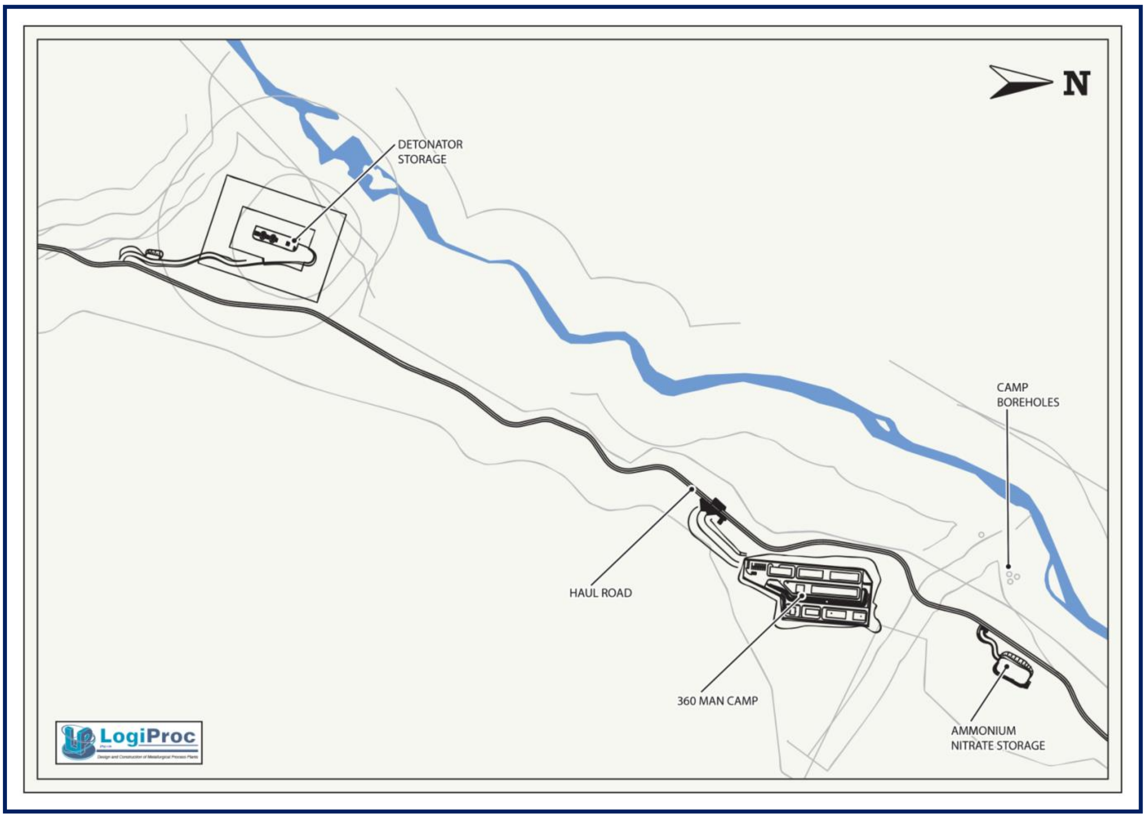
FIGURE 18-10 INFRASTRUCTURE EXTRACT FROM FIGURE 18-1



18.4.1. ON SITE

On site infrastructure facilities relate only to the 360 Man Camp and associated facilities.

FIGURE 18-11 CAMP, DETONATOR AND AMMONIUM NITRATE LOCATIONS



18.4.1.1. 360 MAN CAMP

The original concept of a fully constructed camp has been replaced by a portable camp facility, and this is reflected in the budget. The 360 Man Camp will be constructed in line with the minimum requirements of Kyrgyzstan law with respect to temporary facilities.

This scope of supply covers all the requirements for a 360 man accommodation camp. However, this may be enlarged to cover up to 450 people if required.

The accommodation camp will include accommodation for management, supervisors and general labourers, as well as services buildings that will house facilities such as kitchen, laundry, reception, and recreational area.

The mining contractor will construct, manage and operate the accommodation camp, including clearing and grubbing, earthworks to prepare the pad and foundations where required. The contractor's scope includes all facilities with the exception of the borehole water.

The earthworks for the 360 Man Camp has commenced, with 87,000 m^3 being excavated, and 103,000 m^3 being backfilled. The platform is 98% complete.

18.4.1.2. Buildings and Fence

The accommodation camp will have three types of accommodation as well as services buildings, which will include facilities such as kitchen, clinic, reception and recreational area. There will be security fencing around the whole accommodation camp with pedestrian and vehicular access gates that will control access to the camp.

The Main Camp construction is being undertaken in two phases:

- Phase 1 : The design and manufacturing of the workers dormitory consisting of 80 beds and associated ancillaries is complete, with 60% delivery to site, and 50% installation completed.
- Phase 2 : The design of the kitchen, canteen, management accommodation, and recreation facilities is 80% completed.

18.4.1.3. Power Supply

Accommodation, Construction & others will be powered from two existing 400 kVA diesel generator sets. Remote sites that require power shall be supplied by stand-alone, self-contained generator packages.

18.4.1.4. Potable Water System

An 80 m^3/day treatment system has been proposed as shown below. The plant is sized to provide potable water for the whole site.

A treatment plant with the following units has been proposed:

- Chlorine Dosing Unit;
- Raw Water Tank;
- Sand Filter;
- Active Carbon Filter; and
- Ultraviolet disinfection system.

The water system will be contained in 20 ft container and can be relocated if necessary.

Design values are summarized in Table 18-8 and Table 18-9.

The design packages for pump house and water purification system have been completed, while the internal camp water supply grid design is 75% complete

Table 18-8 Project Flow Rate

Parameter	Value	Unit
Flowrate	80	m^3/Day

Table 18-9 Water Quality

Parameter	Influent	Effluent	*Desired Effluent	Unit
Bods	200	<10	4	mg/ℓ
TSS	200	<10	0,75	mg/ℓ
Ph	6-9	6-9	6-9	

18.4.1.5. Waste Water System

Sewage will be collected from various sites around the site by vacuum truck and transported to a treatment plant located at the camp. An 80 m^3/day treatment system has been specified as shown below:

- Physical treatment:
  - Oil separation; and
  - Equalization.
- Biological treatment:
  - MBR.

Equalization: Wastewater is delivered to an equalization tank for hydraulic and organic equalization and homogenization. Mixing is done via blower.

Biological Treatment: The system used for biological treatment is Membrane Bioreactor (MBR) technology, which is a 'fill and draw' activated sludge system. During aeration, oxygen for biological treatment is provided by blower-diffuser system. The waste sludge is collected in the sludge unit for disposal.

Design values are summarized in Table 18-8 and Table 18-9.

The Waste-Water Treatment Plant has been designed and constructed by a specialist company in Lithuania. The local adaptation and permitting was completed, and the WWTP plant has been delivered to site, awaiting installation.

The associated sewage line has been designed to local regulations, and is awaiting the required permitting.

18.4.1.6. Water Supply

Chaarat will install a bore with a pump near the Sandalash River to which the contractor will connect the supply pipeline.

Analysis of the 2 boreholes has confirmed 4,23 m^3/hr water supply was confirmed, which is adequate for the 85 m^3/day required for the camp.

18.4.2. Off-site Infrastructure

Off-site infrastructure covers proposed facilities at the Shamaldy-Say rail head, Chatkal Station, Kumbel Pass Checkpoint and the mine access road from Kumbel Pass to site. No budget provision has been made for the upgrading of the rail head at Shamaldy-Say, which will therefore not be used in the construction phase of the project. A logistics hub will be constructed by UDK-Group LLC by July 2021 to service the mines in the area which Chaarat can join onto.

The Chatkal Station and the Kumbel pass checkpoints are existing, and no additional budget provision has been made for them.

A budget allowance has been made to upgrade the Kumbel pass to site access road, though a new road is no longer envisaged for this phase of the project.

Maintenance of the Kumbel Access Road by Charaat, has continued since the 2019 BFS.

18.4.2.1. Chatkal Station

The existing Chatkal Station comprises secure fenced parking area with pedestrian and access gates, and some basic facilities. It is a compacted hard stand area, measuring 100 m by 60 m with 2.4 m high chain link security fence.

18.4.2.2. Kumbel Pass Checkpoint

The Kumbel Pass road checkpoint is located along the access road at the boundary of the mine lease and is to ensure that unauthorized vehicles do not gain access to the site.

The Kumbel Pass road checkpoint consists of a guard house with an ablution block, and site gate. Two security personal will be based at the checkpoint on both day and night shift.

The guardhouse is pre-existing and will be equipped with communication (radio) equipment in order to communicate with the main gatehouse at the process area entrance and the Chatkal Station. A portable generator will provide power for heat and light.

18.4.2.3. Kumbel Pass to Site Section of Road

The final section of the existing Kumbel Pass Access Road, which leads down from the mountain and into the mine site, is in an area of high avalanche and rockfall activity, which leads to frequent road closures.

Budget provision has been made to substantially upgrade this road so that it can be negotiated by trucks transporting 40 ft containers by modifying the switchbacks. Provision will also be made for additional avalanche protection.

During the springtime, rock falls are becoming more frequent, and mainly they block the road ditches and sometimes block the road, sharp edged rocks create risk to damage to tires. Road ditches and road base must be kept clean for the coming winter.

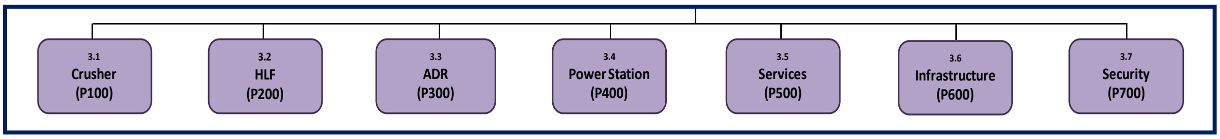
During winter operation of the mine, an avalanche safety and mitigation programme is in place to ensure additional safe operation of the road.

The Kumbel Pass ugrade is 80% complete. A few drainage structures and gradient alterations of sections are still required and the road needs to be widened in certain sections.

18.5. PROCESS

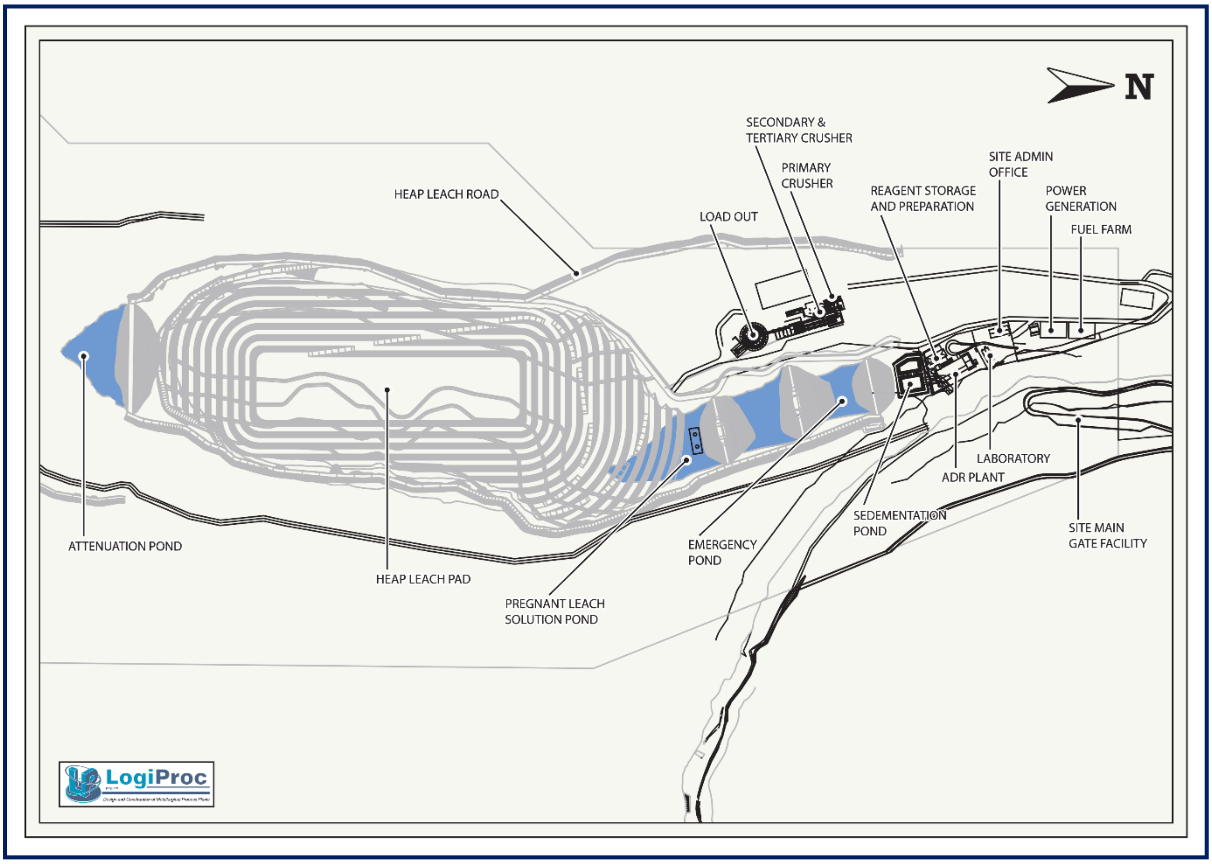
The following facilities are located within the processing area footprint in the Dry Valley.

FIGURE 18-12 PROCESS EXTRACT FROM FIGURE 18-1



The layout of the process area can be seen in Figure 18-13.

FIGURE 18-13 PROCESS AREA LAYOUT



18.5.1. CRUSHING FACILITY

The crushing circuit will be located to the north of the process area, approximately 5 km south of the open pit.

The positions and orientations of the ROM pad, crushing facility, and loadout facility were chosen to make the most efficient use of cut-and-fill, valley contours and the proximity to the HLF.

The buildings will be steel clad but unheated.

Dust collection units are provided at appropriate locations throughout the facility.

18.5.1.1. Earthworks and ROM Pad

The ROM pad size is based on an approximate capacity of four days, which equates to 63,000 t or 33,122 m^3 (based on a bulk density of 1.90 t/m^3). This is based on an estimated stockpile size of 130 m in length, 40 m wide, and 12 m high at an angle of repose of 37°. The ROM pad has been reshaped to accommodate less earthworks and as a result tapers to a narrower section to the south of the stockpile.

The ROM pad will be at 2,425 masl, with the dump pocket located in the north of the ROM pad.

18.5.1.2. Primary Crusher

The primary crushing facility will be enclosed in a metal clad building and will include dust suppression (as described in Section 0.).

The primary crusher structures will be cladded steel construction. The building is 29 m (L) x 14 m (W) x 21 m (H). The crusher complex is divided into three distinct levels i.e.:

- **ROM Bin:** Level 2,425 masl, is designed for a two-way tip.
- **Apron Feeder:** Level 2,414 masl; the apron feeder feeds a vibrating feeder into the Primary crusher;
- **Primary Crusher:** Level 2,418 masl; the primary crusher discharge reports to the secondary crusher building.

18.5.1.3. Secondary and Tertiary Crushing Facility

The secondary and tertiary crushing facility will be enclosed and will include dust suppression, as described in Section 0.

The secondary and tertiary crushing building will be a clad portal framed structure with an overall footprint of 24 m (L) x 18 m (W) x 24 m(H) m and is located at 2,405 masl, the structure will have a mono pitch roof sloping to the East.

This building houses a Secondary Crusher, and two Tertiary Crushers, with space for a third.

18.5.1.4. Screen house and Lime Facility

The screenhouse contains two vibrating screens to sort particle size. This building will be steel clad with a mono pitch roof sloping to the East, and an overall footprint of 24 m (L) x 21 m(W) x 30 m(H).

18.5.1.5. Conveyors

Table 18-10 shows the capacities and lengths of the conveyors used in the crushing section.

Table 18-10 List of Conveyors in the Crushing Section

Equipment Name:	Capacity Range (t/h)		Length (m)
Secondary Crusher Feed Conveyor	741	889	25
Screen Feed Conveyor	1378	1654	77
Screen oversize Conveyor	637	764	11,5
Tertiary Cone Crusher Feed Conveyor	637	764	80
Product Screen-1 undersize transfer Conveyor	371	445	11,5
Product Screen-2 undersize transfer Conveyor	371	445	11,5
Product transfer Conveyor	741	889	13
By-Pass Screen Feed Conveyor	353	424	25,5
By-Pass Screen Oversize Conveyor	285	342	26,5
By-Pass Screen Underflow Conveyor	69	83	12
Fine Ore Stockpile Feeding Belt Conveyor	810	972	138,5
Loadout Belt Conveyor	810	972	50,5

18.5.1.6. Fine Ore Stockpile and Truck Load-out.

The crushed material reports to an open stockpile located at 2,400 masl. The Fine Ore stockpile is designed with a live capacity of 4,500 tonnes and total capacity of 11,500 tonnes. This will provide approximately 6 hours of live storage and approximately 16 hours of total storage.

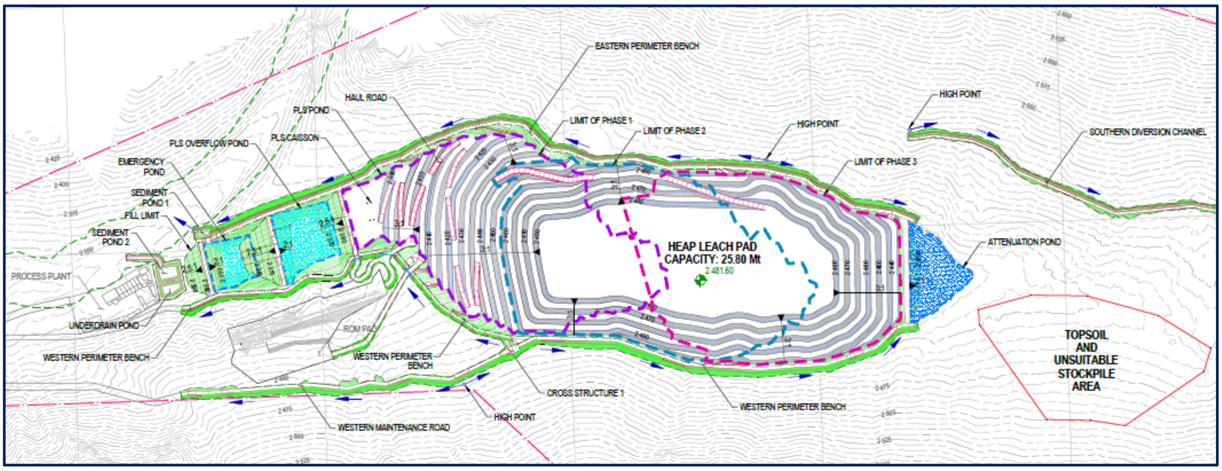
There is a multiplate steel culvert type tunnel underneath the stockpile in which 3 belt feeders withdraw material from the stockpile and discharge on to the truck loading conveyor belt. Truck loading will be controlled by appropriate instrumentation. Truck loading can take place from the fine ore stockpile with loaders if necessary.

18.5.2. Heap Leach Facility

The heap leach facility is designed to store and process 25.88 Mt of ore (expandable to 30 Mt) at a nominal rate of 13,500 t/d. The HLF is located south of the ADR Plant in the Dry Valley. The HLF main components are the following (refer to Figure 18-14):

- Heap Leach Pad;
- Pregnant Leach Solution (PLS) Pond;
- PLS overflow Pond;
- Emergency Stormwater Pond;
- Attenuation Stormwater Pond;
- Perimeter Access Roads; and
- Stormwater Diversion Channels.

Figure 18-14 Heap Leach Facility General Layout



Run-of-Mine (ROM) ore will be processed through a conventional three-stage crushing circuit to a size of P_{80} of 12.5 mm. The crushed ore will be stacked on the heap leach pad using a combination of haul trucks and a bulldozer. The heaps will be constructed in 7 m lifts, to an average heap height of approximately 90 m. Heap leach operations will commence during pre-production stripping of the open pit. The heap leach pad will be stacked with ore and leached continuously from Year 1 through to Year 5 of mine operations.

18.5.2.1. HLF Siting Study

A siting study was performed for the heap leach facilities in previous studies by other consultants. The final location of the HLF in the Dry Valley was largely driven by the local topography, geohazards, such as avalanche and rock fall, and the 100 m water exclusion zone on either side of the Sandalash River, as per Kyrgyz Republic Regulations.

18.5.2.2. Previous Studies

A number of studies have been prepared for the development of a heap leach facility for the Tulkubash Gold Project. TetraTech (2018), Wardell Armstrong International (WAI) (2017) and SLR (2015) undertook feasibility-level studies for the proposed HLF. In addition, a couple of studies were completed on the geologic hazards for the mine facilities, including the heap leach facilities, e.g. Vershina (2018) and Dynamic Avalanche Consulting LTD. (2018). Ausenco has reviewed these reports in the course of acquiring information for the engineering of the HLF.

18.5.2.3. Site Conditions

The U-shaped dry valley in which the HLF will be situated includes glacial detrital valley infill material, interpreted as a combination of moraine detritus and colluvium, formed by recent (post-glacial) erosion of the steep valley sides in debris flows and landslides.

The valley is approximately 5 km in length, trending generally in a north-south direction. The northern valley extent is bordered by the Kumbeltash Stream, which flows in a northwest to north-northwest direction towards the Sandalash River. A steep, east-west trending v-shaped valley borders the southern extent of the valley, which drains towards the Sandalash River. The valley floor in the proposed HLF area typically falls to the north towards the Kumbeltash

Stream; however, the southernmost area of the valley drains to the south into the east-west trending valley into the Sandalash River, at which point this change in fall direction of the valley floor is marked by a substantial boulder field.

The base of the valley is typically narrow, ranging from 50 to 100 min width, with a north to south elevation change of approximately 150 m, from 2,300 to 2,450 masl in elevation. The valley sides, which vary in slope angle, extend up to 3,000 m in elevation.

The lower valley slopes have moderate slope angles of 20 to 25° becoming progressively steeper (30 to 40°) in the mid-slope areas, before becoming very steep (greater than 60°) in areas of exposed bedrock. However, historical and ongoing geological and geomorphological processes have created a varied and complex ground profile along the valley.

Groundwater flows in the direction of the valley slopes, towards the Kumbeltash Stream and the Sandalash River. The elevation of the Sandalash Valley below the dry valley is approximately 2,150 masl. It appears that below 2,245 masl the general permeability in the dry valley rubbly infill is such that any groundwater will rapidly seep into the water table below 2,245 masl. This will be controlled by the geometry of the infill base and the level of the Sandalash River, towards which groundwater will migrate.

18.5.2.4. Geotechnical Considerations

Several geotechnical field investigations have been performed in the area of the HLF. The most recent testing was performed by KyrgyzGIIZ OJSC in 2018. Testing completed in the heap leach area included 21 boreholes to a depth of 30 m, 9 cone penetration tests (CPTs), 30 test pits to a depth of 4 to 6 m, and 189 physical points of vertical electric sounding (VES). In addition, 7 piezometers were installed in 7 boreholes drilled to monitor the groundwater.

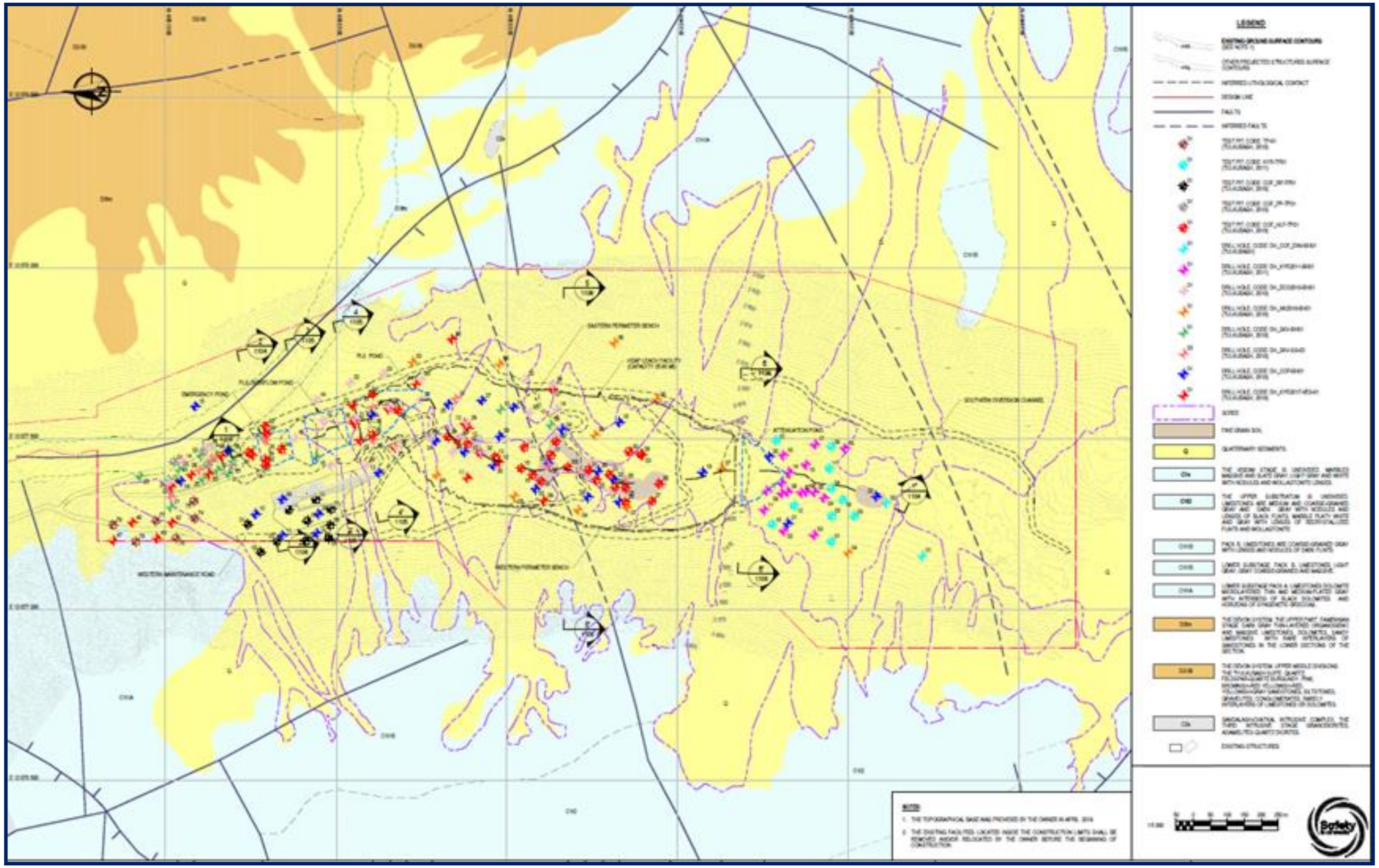
A Laboratory programme was performed on soil samples from the field programme to obtain both physical and mechanical properties. As part of this laboratory programme to look at the mechanical interface properties between various liner interfaces (geosynthetic against geosynthetic and geosynthetic against soil/gravel) were tested for the leach pad and ponds to be utilized in the design (Refer to Figure 18-15).

Based on the geotechnical investigations and laboratory programmes, the geological units at HLF were found to comprise of:

- Coarse valley infill, which is coarse, highly-variable, detrital material draping the valley sides and the valley floor;
- Fine valley infill, which is highly-variable, clay-rich (typically 35 to 40% clay matrix) material in the base of the valley, interbedded with the coarse, highly-variable, detrital material to a depth of approximately 18 m;
- Loose talus/scree slopes are located within the leach pad footprint on the west side towards the back end of the pad and on the east side near the center of the pad on the side slopes of the valley;
- Weathered rock (5 to 10% silty clay filler) in the base of the valley, to a depth of approximately 70 m;
- Rock outcrops on adjacent tops of slopes and valley sides, with bedrock at depth;

- Groundwater along the valley floor ranging in depth from 5 to 25 m below the surface;
- Potentially compressible silty soils are located in the base of the dry valley toward the south end of the leach pad; and
- Avalanche and debris flow deposits were identified and classified above the HLF on both the eastern and western slopes.

FIGURE 18-15 GEOTECHNICAL CONDITIONS PLAN HLF



18.5.2.5. GEOHAZARD INVESTIGATION

Significant snow avalanches, rockfall and debris flow hazards are present in the project area, including numerous avalanche paths with the potential to influence project facilities and mine personnel. In addition, there are several relic rock avalanche and debris flows.

Two geohazard studies were performed: Vershina (2018) and Dynamic Avalanche Consulting LTD. (2018).

Key geohazard hazards identified within the HLF area include:

- rock fall from upper mountain slopes (rock crags) as isolated block fall events,
- rock avalanche, they are usually rare,
- large-scale active/seasonal in channels or open slope debris flows,
- snow avalanches, which include dry avalanches, wet snow avalanches, and slush flows,
- seasonal snow melt and stormwater run-off, and
- avalanche hazards are extensive and complicated and require a comprehensive avalanche risk management programme, such as an avalanche safety plan.

A preliminary plan has been developed by Dynamic Avalanche Consulting but needs further development. A summary of their findings and recommendations are presented below.

A total of 10 avalanche paths have been identified that intersect the leach pad and ponds. All of these paths are capable of producing Size 3 avalanches, which are capable of affecting workers and light infrastructure. In addition, two paths; one on the east side leach pad and one on the west side of the plant, Crusher, Underdrain Pond and Emergency Pond are capable of producing Size 4 avalanches, which are also capable of affecting workers and infrastructure.

It is expected that the leach pad and ponds will not be vulnerable to the avalanche hazards and that the risk will be related to the workers and ancillary infrastructure, such as pump houses and buildings. Risk to workers can be managed by temporary closures and worksite safety procedures including triggering avalanches in a controlled manner. Vulnerable infrastructure, such as pump houses, etc, will need structural protection. Avalanches can also create large waves when they hit open water, such as in ponds. However, during the avalanche season the Attenuation, PLS Overflow, and Emergency Ponds are dry and the PLS Pond is covered with gravel so it will not be impacted by avalanches.

18.5.2.6. DAM HAZARD CLASSIFICATION

According to international design standards for heap leach facilities, any pond that has a dam wall higher than 2.5 m, contains more than 30,000 m^3, or its failure of which is likely to be unacceptable to the public, requires a dam hazard classification. There are several guidelines to analyze the hazard classification of impounding structures. The HLF has 4 ponds that require a dam hazard classification; the PLS Pond, The PLS Overflow Pond, the Emergency Pond, and the Attenuation Pond. For this project, the hazard classification for the four ponds was assessed using the Canadian Dam Association “Dam Safety Guidelines (2007/2013)”.

Consequence categories are based on the incremental losses that a failure of a dam might inflict on downstream or upstream areas, or at the dam location itself. Incremental losses are those over and above losses which might have occurred in the same natural event or condition had the dam not failed. The classification assigned to a dam is the highest rank determined among the 3 loss categories; loss of life, Environmental & Cultural, and Infrastructure & Economic Losses.

Since the ADR plant is below the PLS, PLS Overflow, and Emergency Ponds, the loss of life category will govern the dam hazard classification for these 3 facilities. Since the potential loss of life is greater than 10 and less than 100, the dam classification is "Very High". For the Attenuation Pond the dam classification is "Significant" since there are no permanent population at risk and the Environmental, Cultural, Infrastructure and economics are low. The design target criteria for seismicity and inflow design flood (IDF) for the 4 ponds is presented in Table 18-11.

TABLE 18-11 TARGET DESIGN LEVELS FOR FLOOD AND EARTHQUAKE HAZARDS

Dam Classification	Annual Exceedance Probability - Earthquakes	Annual Exceedance Probability - Floods
PLS Pond	½ between 1:2475 and 10,000 or MCE	2/3 between 1:1,000 and PMF
PLS Overflow Pond	½ between 1:2475 and 10,000 or MCE	2/3 between 1:1,000 and PMF
Emergency Pond	½ between 1:2475 and 10,000 or MCE	2/3 between 1:1,000 and PMF
Attenuation Pond	Between 1:100 and 1:1,000	Between 1:100 and 1:1,000

Note: PMF = Probable Maximum Flood

Taking into account the site seismic framework, the MCE (PGA=0.49g) which corresponds to an M7.5 event with origin in source 22 was chosen for the PLS, PLS Overflow, and Emergency Ponds.

For the Attenuation Pond, the 475-return period was chosen with a PGA equal to 0.23 G.

18.5.2.7. HEAP LEACH FACILITY ANALYSIS AND DESIGN

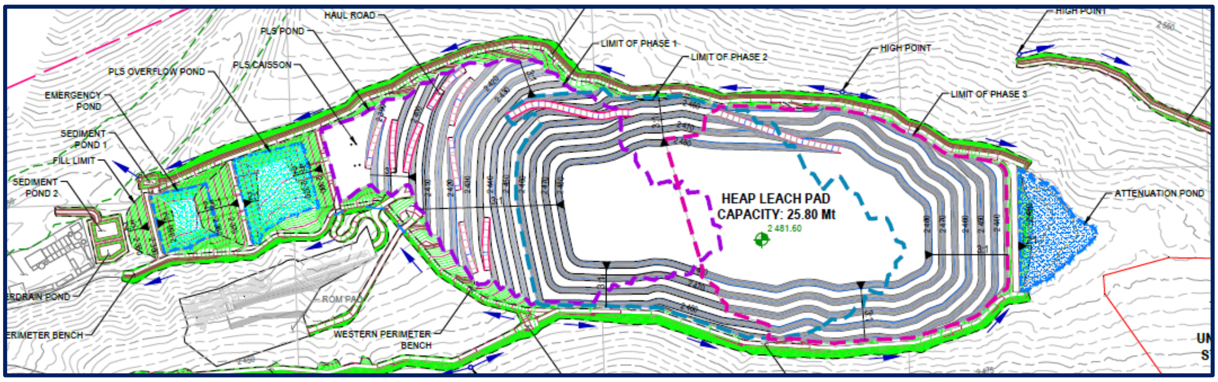
The Heap Leach Facility (HLF) consists of the following components:

- Heap leach pad
- PLS, PLS Overflow, Emergency, and Attenuation ponds
- Liner Systems
- Underdrain System
- Solution Collection System
- Stormwater management system

18.5.2.8. HEAP LEACH PAD AND PLS POND

The heap leach pad consists of a confining embankment (PLS Pond Embankment), underdrain, pad liner system, and solution collection system to collect and convey the pregnant solution to the gold extraction plant located to the north of the HLF (Refer to Figure 18-16).

Figure 18-16 Heap Leach Pad and PLS Pond



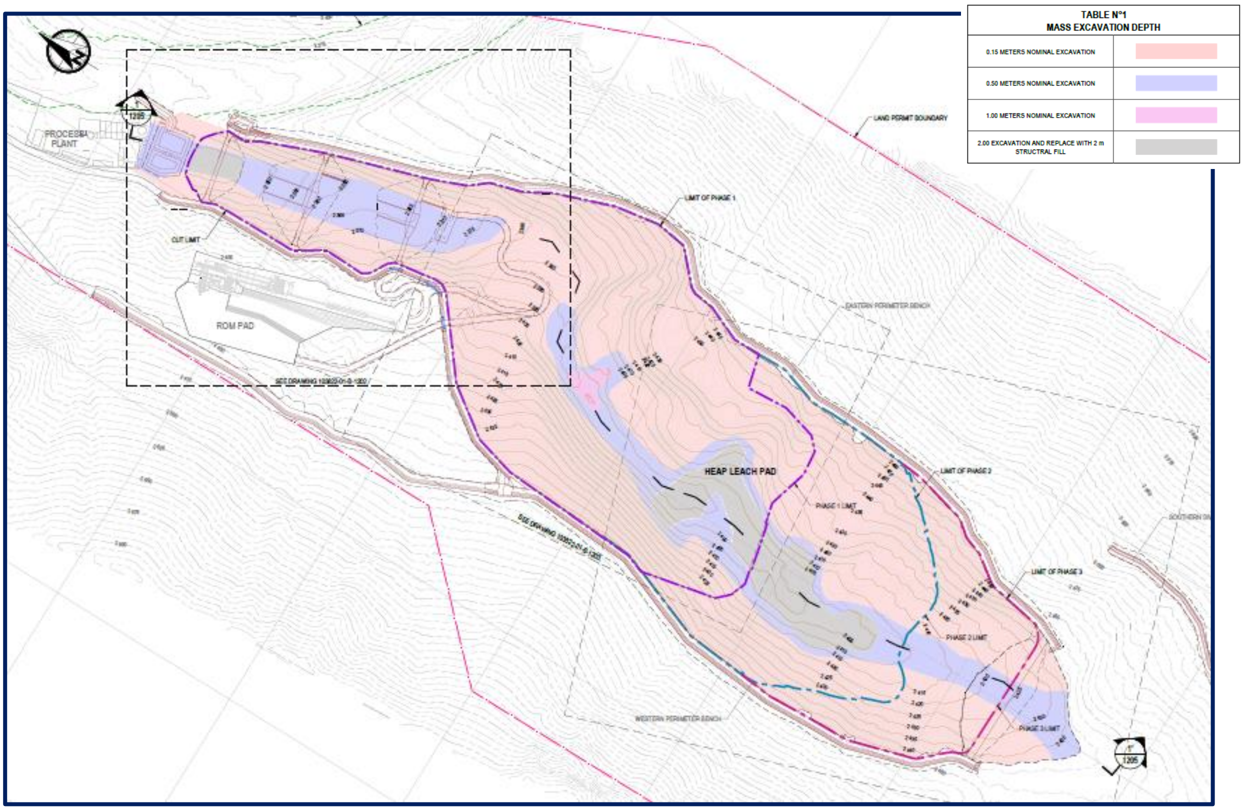
The pad is located in a u-shaped dry valley with steep side slopes ranging from approximately 7H:1V (15%) to 2.5H:1V (42%) and has an approximate footprint area of 472,400 m^2. The heap leach pad is designed to be operated predominantly as a dry/wet heap-leach facility with the majority of the facility having less than 2 m of solution over the liner (dry), and pregnant solution stored behind the confining PLS Pond embankment (wet).

The following sections outline the general design features and construction aspects for each of the main components of the heap leach pad.

18.5.2.8.1. Foundation

At the start of each of the phases, preparation of the pad foundation is required. Foundation preparation entails clearing and grubbing surface organics, stripping of approximately 0.5 m of topsoil and vegetation, and the removal of any large boulders. The unsuitable soil and topsoil will be stockpiled at a location south of the HLF (see Figure 18-17) and the topsoil will be used for reclamation of the HLF at closure.

Figure 18-17 Unsuitable Soil and Topsoil Stockpiles



The underlying colluvial and residual soils will be excavated down to a competent, stable foundation. A one-meter excavation depth has been estimated for foundation preparation to competent ground.

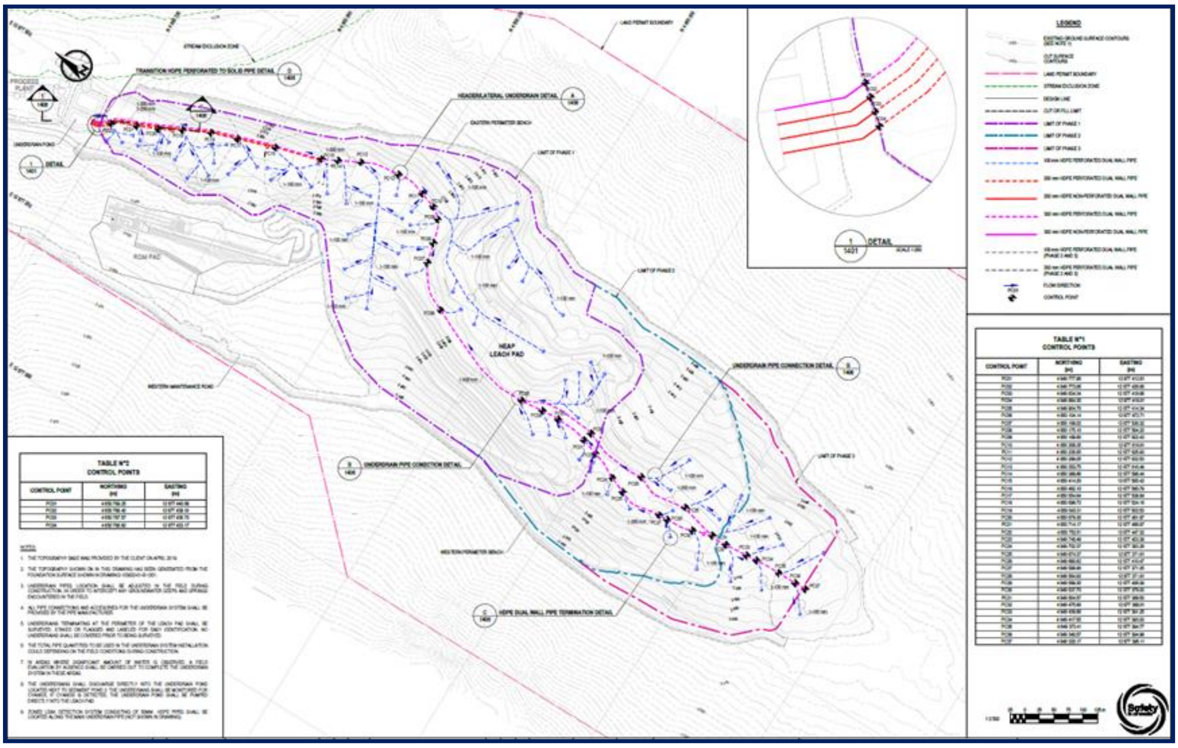
In the central southern section of the leach pad there is a zone of soft soils, which is over 9 m thick. This zone will consolidate (settle) during loading of the pad and could potentially place the liner in tension and rip it. Therefore, instead of removing and replacing the soft soil, a structural wedge fill will be placed over this zone to mimic that amount of calculated settlement to prevent the liner system from going into tension after consolidation.

In order to provide a uniformly and positively graded surface to place the pad liner system on, rough grading and backfill will be used to level the naturally undulating bedrock surface and to ensure that the pad grading will promote solution flow to be positively draining towards the solution collection piping system and caisson sump located at the centre of the upstream embankment toe. A minimum pad grade of 1% is required.

18.5.2.8.2. Underdrain

An underdrain will be installed below the HLF to drain near surface groundwater from below the liner system to ensure that groundwater pressure cannot be developed under the liner system. The system consists of polyethylene dual wall perforated pipe placed in a trench filled with drain rock (Refer to Figure 18-18). The outlet of the underdrain will discharge into the underdrain pond and monitored for constituents of concern. If the water exceeds project water quality monitoring standards then the water will be pumped to the PLS pond or treated and released till the water quality exceeds project standards.

Figure 18-18 Underdrain Collection System Layout



18.5.2.8.3. PLS Embankment And Construction

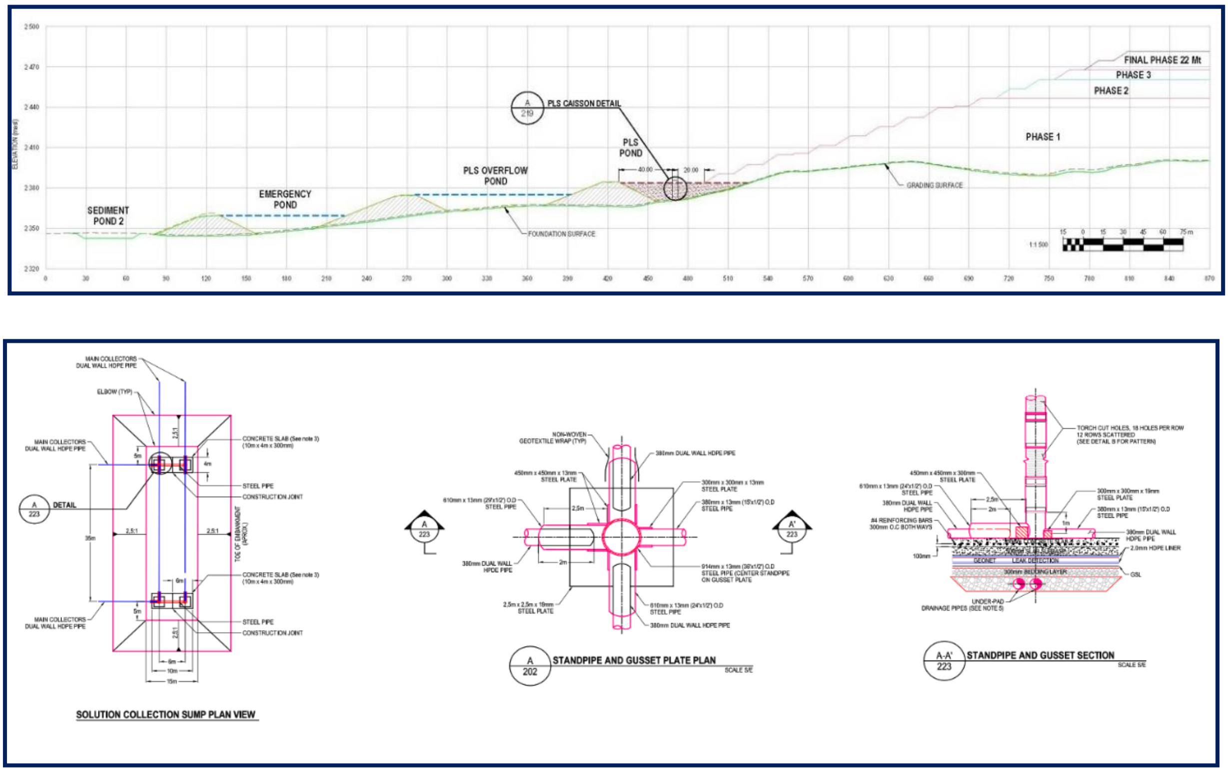
The PLS embankment constructed at the toe of the proposed pad will provide stability to the heap leach pad and provide in-heap storage for solution. As presented on Figure 18-19, the embankment will have a final crest elevation of 2,386 masl and a crest width of 6 meters. The embankment will be constructed with structural and a transition zone on upstream slope with an upstream slope of 2H:1V and downstream slope of 2.5H:1V.

The PLS Pond is designed to meet the following design criteria:

- Storage capacity to contain 8 hour of operation Pregnant Solution flow plus 24 hour of drain-down flow without discharging into the PLS Overflow Pond. Any significant snow melt or rain may overflow into the PLS Overflow Pond, and
- Spillway is designed to discharge the 2/3 between the 1:1,000-year event 24-hour and PMF 24-hour storm event with a minimum embankment crest freeboard of 1.0 meter.

Solution will be stored behind the dam and after significant periods of rainfall, snow melt or during a shut-down process in-heap storage will be utilized. The PLS embankment has been designed with an in-heap storage capacity of approximately 38,208 m^3 (approximately 8 hour of irrigation volume plus 24 hour of drain-down). If the storage requirement is greater than this, excess solution will pass over the PLS embankment spillway (invert elevation 2384.5 masl) into the PLS Overflow Pond.

Figure 18-19 Caisson and Dam Section



Preparation of the embankment foundation will be undertaken in the same manner as the foundation preparation for the heap leach pad and will involve stripping the topsoil and excavating the underlying colluvial and residual soils down to component material. The main embankment body will be constructed from structural fill which will consist primarily of locally sourced earth fill. The embankment will be constructed by placing the fill in lifts and compacting to a specified density. The earth fill will be sourced from borrow areas in and adjacent to the HLF (including the ROM Pad excavation) and the rock fill will be obtained from talus deposits and new local rock quarries. A 0.3 to 0.6-meter-thick bedding layer will be placed in preparation for installation of the liner system. In areas with no scree, only silty/clayey gravel (20 to 30% fines and 60% sand and gravel with less than 30% gravel <5 mm) will be placed and a transition zone (gravel with a P_{80} of 30 mm) will be placed before the silty/clayey gravel in areas with scree slopes.

18.5.2.8.4. Liner System

The liner system is used to maximize pregnant solution recovery and minimise environmental operational impacts by minimizing leakage losses of pregnant solution through the bottom and sides of the heap leach pad. The composite liner consists of ‘barrier’ and ‘drainage’ layers using a combination of synthetic and natural materials to provide solution containment which meets the required ‘best practice standards’ for leach pad design. While the Heap Leach Pad Area is designed to operate as a ‘dry’ pad with minimal solution occurring above the liner (i.e.

less than 2 m) during normal operating conditions, the PLS Pond area liner system is designed to meet the required performance standards assuming fully saturated solution storage conditions behind the PLS embankment to a maximum depth of 15.5 m.

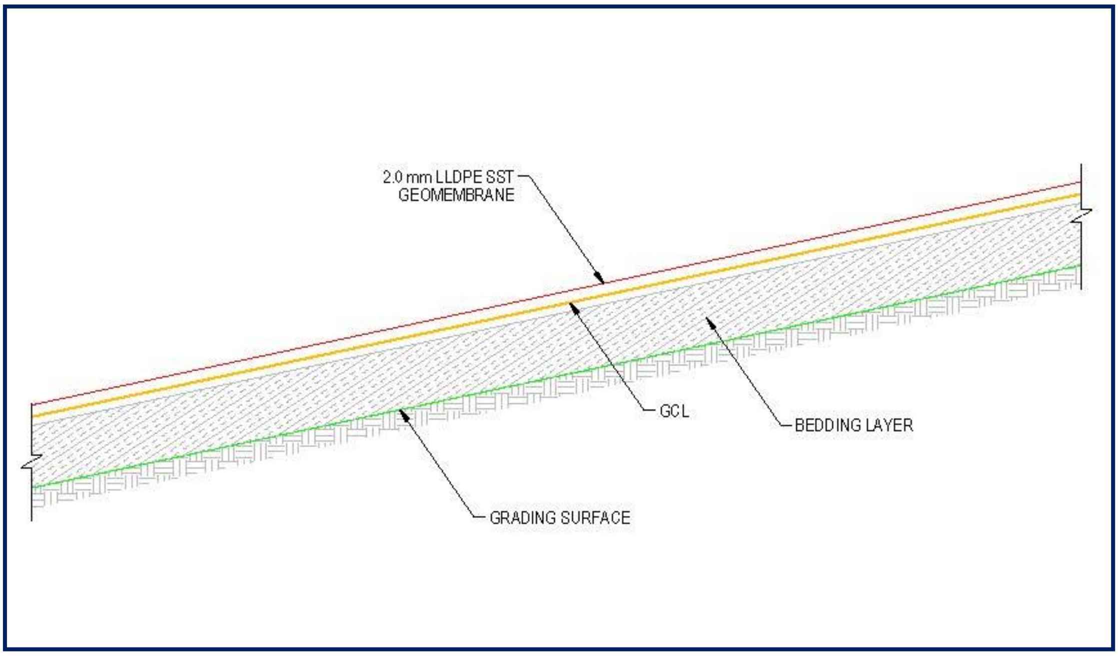
18.5.2.8.5. Liner Design

Two liner systems have been developed for the Heap Leach pad, an engineered single liner design for the upper portion of the leach pad (above the impounding solution level in the PLS Pond) and a composite double liner design for the PLS Pond which will have solution storage.

The single liner system is designed to be installed on the heap leach pad's area that positively drain towards the collection pipes and then to the caisson sump area. The liner system consists of the following components (Refer to Figure 18-20):

- 1.0-meter-thick overliner (minus 38 mm with less than 5% fines content) along the valley floor,
- 2 mm linear low-density polyethylene (LLDPE) geomembrane,
- High Strength Geosynthetic Clay Liner (GCL), and
- 0.3 to 0.6-meter-thick compacted bedding layer (0.3 m layer of silty/clayey gravel on non-scree slopes and 0.3 m transition material between the scree slope and the silty/clayey gravel).

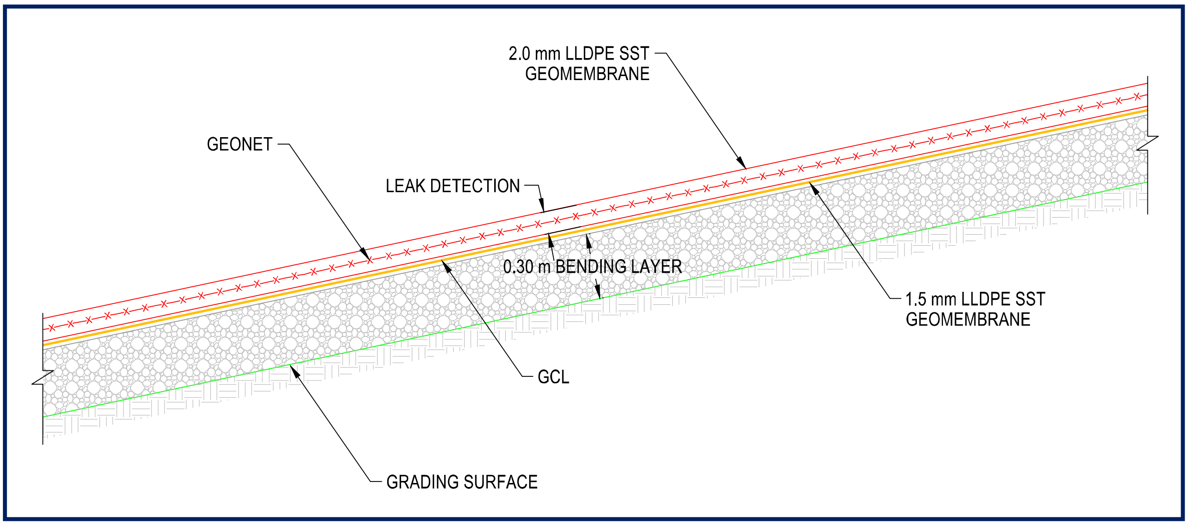
Figure 18-20 Single Liner System



The double liner system is designed to be installed in the PLS Pond Area which will experience hydraulic loading from in-heap solution storage. Whilst still positively draining towards the leachate collection pipes and sump, the surface grades under the double lined portion may be as low as 1%. The double liner system consists of the following components (Refer to Figure 18-21):

- 1.0-meter-thick overliner (38 mm minus with less than 5% fines content) within the entire PLS Pond area,
- 2 mm linear low-density polyethylene (LLDPE) geomembrane,
- High Capacity Geonet,
- 1.5 mm linear low-density polyethylene (LLDPE) geomembrane,
- Leak Detection and Recovery System (LDRS),
- High Strength Geosynthetic Clay Liner (GCL), and
- 0.3 to 0.6-meter-thick compacted bedding layer (0.3 m layer of silty/clayey gravel on non-scree slopes and 0.3 m transition material between the scree slope and the silty/clayey gravel).

FIGURE 18-21 DOUBLE LINER SYSTEM



Laboratory direct shear testing has been completed by Ausenco to determine the interface shear strength of the liner materials and confirmed that the strengths are sufficient to provide long-term stability of the HLF.

18.5.2.8.6. CONSTRUCTION

Development of the heap leach liner system will be constructed in three phases, with annually proposed liner expansions to meet the ore stacking requirements.

The liner system will be constructed with both synthetic and natural layers extending to the top of the confining embankment and perimeter berms to provide full containment. The synthetic materials will be anchored and backfilled in trenches along the heap leach pad perimeter and PLS Pond embankment crest to ensure that ore loading does not compromise the liners within heap leach pad footprint by pulling the liner into the pad.

A small perimeter berm will also be constructed as part of the liner tie-in around the perimeter of the pad footprint to ensure that heap solution is contained within the pad footprint and to also prevent surface run-off from the adjacent slopes entering the leach pad solution collection system.

18.5.2.8.7. OVERLINER

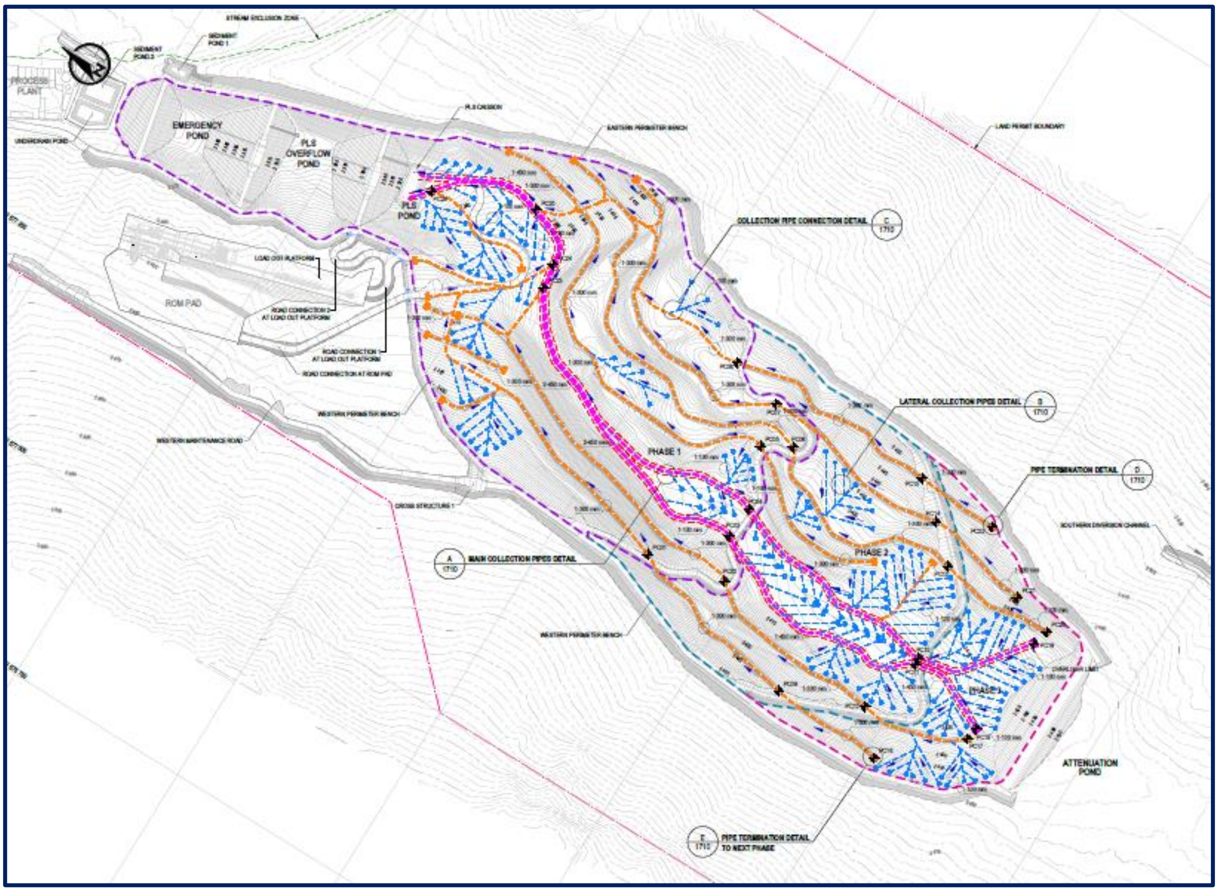
A one-meter thick protective layer of crushed ore or gravel will be placed over the geomembrane liner in the heap leach pad area and the lined area of the PLS Pond footprint. This protects the liners from damage during rock fill placement on the PLS Pond and initial ore placement on the base of the leach pad. The overliner also serves as a drainage layer, directing pregnant solution into the piped solution collection system, therefore reducing head loading on the liner system and maximizing solution recovery.

18.5.2.8.8. SOLUTION COLLECTION SYSTEM

The solution collection system to recover the pregnant solution comprises a network of dual wall perforated piping (ADS® or Equivalent) plus four steel caissons which work in conjunction with the leach pad liner, overliner, and leak detection and recovery systems. The collection system consists of the following (refer to Figure 18-22):

- HDPE dual wall perforated lateral collection pipes,
- HDPE dual wall perforated solution collection header pipes,
- Overliner in designated zones,
- Vertical steel caissons, and
- Rock fill in the PLS Pond.

FIGURE 18-22 SOLUTION COLLECTION SYSTEM LAYOUT



The solution collection system is designed to streamline solution collection and facilitate solution conveyance off the pad as quickly as possible, thereby reducing the potential risk of leachate solution losses through the liner system. The entire piping system is constructed with perforated HPDE dual wall pipes which are embedded within the one-meter thick overliner layer

The lateral solution collection pipes, which are under the entire valley floor, feed directly into the solution collection header pipes. The solution collection header pipes are positioned along the valley floor of the leach pad and terminate in the vertical caissons at the upstream toe of the PLS Pond embankment.

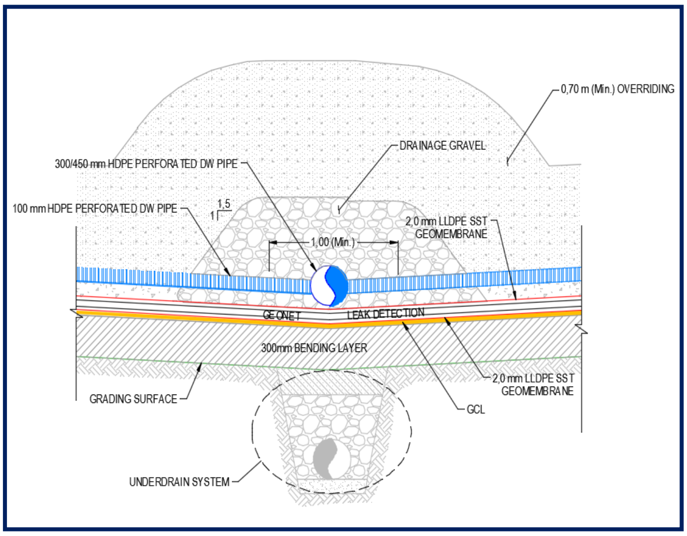
Two solution collection vertical caisson sumps are located at the upstream toe of the PLS Pond embankment, spaced equally across the bottom of the valley. The 964 mm (outer) and 910 mm (inner) steel caissons consist of several telescoping steel sections with perforations. The telescoping sections allow them to move independently down due to drag forces from consolidation of the rock fill and ore and to prevent puncturing of the liner system below the caissons. The area around the caissons will consist of a cone with the top 10 m in diameter with 2H:1V slopes of clean well grade rock fill with no fines (approximate diameter of 50 mm to 300 mm) placed around the perforated steel vertical pipe. The remainder of the PLS pond rock fill can be smaller rock fill (50 mm to 150 mm). The compaction of the conical rock fill is critical to ensure that there is minimal settlement around the vertical caissons and prevent damage to the liner system.

Pregnant solution storage is allowed in the rock fill-pore spaces behind the PLS pond area up to a maximum elevation of 2384.5 masl before discharging through the PLS spillway.

18.5.2.8.9. Leak Detection and Recovery System

The Leak Detection and Recovery System (LDRS) in the PLS Pond is designed to capture and convey any solution that leaks through the primary geomembrane. The LDRS in the double lined area is present in Figure 18-23.

Figure 18-23 The LDRS for Double Liner System



The LDRS in the double lined area consists of a geonet sandwiched between 2 geomembranes. In addition, there are a series of pipes with drainage gravel to facilitate drainage of leaks. Any leakage recovered by the LDRS will be conveyed into the LDRS sump at the upstream toe of the PLS Pond embankment. A level-switch controlled submersible sump pump will transfer the recovered solution up the embankment slope via a solid wall pipe installed within the LDRS gravel layer, and pumped back into the PLS Pond. Monitoring of the leakage recovery will be undertaken through continuous monitoring of the pump hour records.

18.5.2.9. HLP PLS Overflow Pond

The PLS Overflow pond is designed to provide storage for excess solution and run-off which is generated as a result of rainfall and snow melt events that cannot be accommodated by the PLS Pond storage capacity. The pond is situated immediately downgradient of the PLS Pond embankment and pond flows are conveyed via the PLS spillway. The plan layout of the PLS Overflow Pond is included on Figure 18-16. A typical cross-section through the PLS Overflow pond and embankment is included on Figure 18-16.

The PLS Overflow pond is designed to meet the following design criteria:

- Storage capacity to contain the excess PLS Pond solution and surface run-off from the annual event without discharging into the Emergency Pond, and
- Spillway is designed to discharge the 2/3 between 1:1,000-year 24-hour storm event and PMF 24-hour storm event with a minimum embankment crest freeboard of 1.0 meter.

18.5.2.9.1. Storage Requirements

The storage requirement for the PLS Overflow Pond was established based on containment of the entire estimated surface run-off generated from the HLF during an average year.

Modelling of the average annual run-off was undertaken using the Hydrologic Modelling System GoldSim. The model uses site specific data to accurately capture the specific climate and catchment conditions at Site, including storm precipitation intensity distribution, snowmelt, catchment slope, drainage and precipitation losses. Based on the surface run-off results generated by the model, the storage requirements for the PLS Overflow Pond is 82,290 m^3.

Solution stored in the PLS Overflow Pond will be pumped back to the Heap Leach Pad using the PLS Overflow Pond pump station. The pump station is designed to be able to empty the average annual run-off volume (approximately 82,290 m^3) over ten days. Depending on the solution processing rate of the gold extraction plant and the available solution storage capacity behind the confining embankment, the actual pump rate will likely need to vary.

18.5.2.10. Liner System

The engineered double liner system designed for the PLS Overflow Pond uses the same design principles as the PLS Pond liner system. The liner consists of the following layer configuration (Refer to Figure 18-21):

- 2 mm linear low-density polyethylene (LLDPE) geomembrane,
- High Capacity Geonet,

- 1.5 mm linear low-density polyethylene (LLDPE) geomembrane,
- Leak Detection and Recovery System (LDRS),
- High Strength Geosynthetic Clay Liner (GCL), and
- 0.3 to 0.6-meter-thick compacted bedding layer (0.3 m layer of silty/clayey gravel on non-scree slopes and 0.3 m transition material between the scree slope and the silty/clayey gravel).

Careful preparation of the ground surface interfacing with the GCL is required to ensure that the ground surface conditions are acceptable to install the liner system without compromising the liner system integrity.

18.5.2.10.1. Embankment

The embankment will be constructed of structural fill borrow materials. Fine grained residual soils will be selectively utilized on the upstream face to provide a transition zone for geosynthetic liner installation.

The embankment is designed with a 2.5H:1V downstream slope and a 2.0H:1V upstream slope. These slopes ensure the embankment stability.

18.5.2.10.2. Construction and Operation

The PLS Overflow Pond will be constructed to full size prior to commencement of the HLF operations. Construction of the earth fill embankment for the PLS Overflow Pond involves stripping approximately 0.5 m of topsoil and 1 m of overburden beneath the embankment and ponding footprint. The earth fill embankment, pond liner and LDRS system will be constructed directly on a bedding layer. A solution return pump station will be constructed adjacent to the PLS Overflow Pond embankment to pump solution back to the HLF in-heap storage for gold-extraction processing or re-use in leaching.

Under typical operating conditions the PLS Overflow Pond will be operated as a dry pond to ensure that the maximum pond capacity is available for storage of excess HLF surface run-off from snowmelt and storm events. During snowmelt and storm event, solution and run-off exceeding the PLS Pond storage capacity will flow into the PLS Overflow Pond via the PLS spillway to be stored in the PLS Overflow Pond. This water will then be transferred back to the HLF in-heap storage as required. During storm events greater than the annual run-off, water volumes exceeding the PLS Overflow Pond storage capacity will be conveyed to the Emergency pond via the PLS Overflow Spillway.

18.5.2.11. HLF Emergency Pond

The Emergency Pond is designed to provide storage for excess solution and run-off which is generated as a result of the design storm events that cannot be accommodated by the PLS and the PLS Overflow Ponds storage capacities. The pond is situated immediately downgradient of the PLS Overflow Pond, and pond flows are conveyed via the PLS Overflow spillway. The plan layout of the Events Pond is included in Figure 17-6. A typical cross-section through the pond and embankment is included in Figure 17-6.

The Emergency Pond is designed to meet the following design criteria:

- Storage capacity to contain the excess solution and surface run-off from the 1:200 year 24-hour storm event without discharging into the environment, and
- Spillway is designed to discharge the 2/3 between the 1:2,475 year 24-hour storm event and PMF 24-hour storm event with a minimum embankment crest freeboard of 1 meter.

18.5.2.11.1. Storage Requirement

The storage requirement for the Emergency Pond was established based on containment of the entire estimated surface run-off generated from the HLF during the 1 in 200 year 24-hour storm event.

Modelling of the average annual run-off was undertaken using the Hydrologic Modelling System GoldSim. The model uses site specific data to accurately capture the specific climate and catchment conditions at Site, including storm precipitation intensity distribution, snowmelt, catchment slope, drainage and precipitation losses. Based on the surface run-off results generated by the model, the storage requirements for the Emergency Pond is 51,700 m^3.

Solution stored in the Emergency Pond will be pumped back to the Heap Leach Pad using the Emergency Pond pump station. The pump station is designed to be able to empty the 1 in 200-year storm run-off volume (approximately 51,700 m^3) over ten days. Depending on the solution processing rate of the gold extraction plant and the available solution storage capacity behind the confining embankment, the actual pump rate will likely need to vary.

18.5.2.11.2. Liner System

The engineered single liner system designed for the Emergency Pond uses the same design principles as the leach pad liner system. The liner consists of the following layer configuration (Refer to Figure 18-20):

- 2 mm linear low-density polyethylene (LLDPE) geomembrane,
- High Strength Geosynthetic Clay Liner (GCL), and
- 0.3 to 0.6-meter-thick compacted bedding layer (0.3 m layer of silty/clayey gravel on non-scree slopes and 0.3 m transition material between the scree slope and the silty/clayey gravel).

Careful preparation of the ground surface interfacing with the GCL is required to ensure that the conditions are acceptable to install the liner system without compromising its integrity.

Installation of a Leak Detection and Recovery System (LDRS) is not required for the Emergency Pond as the pond is operated as a dry-facility and will only receive and store run-off water during significant storm events. In the event that leakage does occur through the single liner system, this water will be conveyed via the underdrain to the underdrain pond that is routinely monitored.

18.5.2.11.3. Embankment

For the emergency pond, the embankment will be constructed of colluvial soil borrow materials. Fine grained soils will be selectively utilized on the upstream face to provide a transition zone that is acceptable for GCL installation.

The embankment is designed with a 2.5H:1V downstream slope and a 2.0H:1V upstream slope. These slopes ensure the embankment stability.

18.5.2.11.4. Construction And Operation

The emergency Pond will be constructed to full size prior to commencement of the HLF operations. Construction of the earth fill embankment for the Emergency Pond involves stripping approximately 0.5 m of topsoil and 1 m of overburden beneath the embankment and ponding footprint. The earth fill embankment, pond liner and underdrain system will be constructed on competent soils. A solution return pump station will be constructed adjacent to the Emergency Pond embankment to pump solution back to the PLS Pond for gold-extraction processing or re-use in leaching.

Under typical operating conditions the Emergency Pond will be operated as a dry pond to ensure that the maximum pond capacity is available for storage of excess HLF surface run-off from snowmelt and storm events. During a storm event, solution and run-off exceeding the PLS and PLS Overflow storage capacities will flow into the Emergency Pond via the PLS Overflow spillway to be stored in the Emergency Pond. This water will then be transferred back to the HLF in-heap storage as required. During storm events greater than 1:200 year 24-hour, water volumes exceeding the Emergency Pond storage capacity will discharge into the river through the Emergency Pond Spillway.

18.5.2.12. Instrumentation and Monitoring

Instrumentation and monitoring will be carried out on an on-going basis to ensure the safe and effective operation of the HLF. Recommendations for instrumentation and monitoring are summarized below:

18.5.2.12.1. Instrumentation

Geotechnical instrumentation will be installed to monitor the performance of the HLF during the construction stage and throughout the life of the facility. The purpose of the instrumentation will be to provide data to assess the stability of the leach pad and ponds and to evaluate the effectiveness and performance of the facilities.

The following instrumentation is recommended for installation at the HLF:

- Vibrating wire piezometers in the foundation of all pond dams and pad
- Inclinometers and settlement cells in all pond dams and pad
- Survey monuments in all pond dam crests, and
- Accelerometer for closure stage

18.5.2.12.2. Monitoring

Preliminary recommendations for monitoring are summarized below:

- Surface water quality sampling at the underdrain pond and sediment ponds locations downstream of the HLF,
- Installation of monitoring wells around the facility to monitor groundwater quality during operations and at closure. These wells would be installed prior

to development of the facilities to obtain baseline information for comparative assessment,

- Slope movement monuments and survey control points installed and monitored to ensure the integrity and stability of the ore heap, and
- Installation of flow monitoring devices in diversion ditches and creeks to confirm design flows.

Details of the instrumentation and monitoring plan will be developed during the detail design phase.

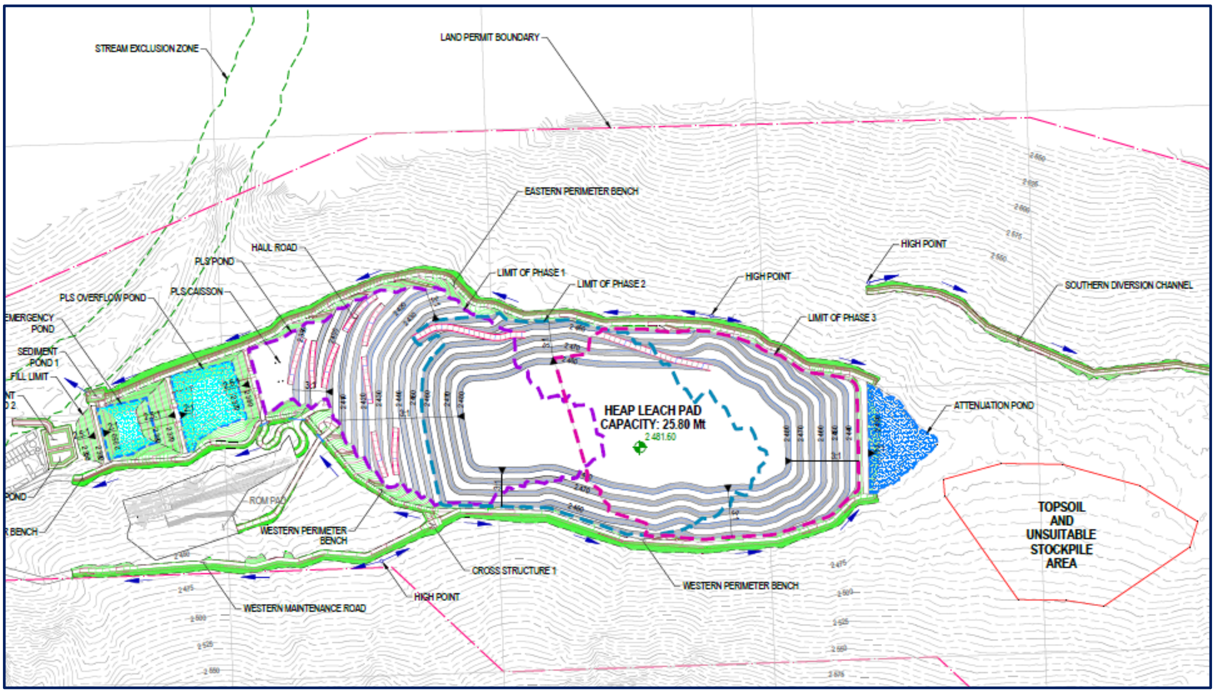
18.5.2.13. Surface Run-off Management

The surface water management system for the site, as presented in Figure 18-24, consists of a series of ditches constructed around the perimeter of the HLF to intercept overland surface run-off around the HLF pad and ponds and to convey flows to the Attenuation Pond or to the stream below the Emergency Pond. The channels are designed to meet the following design criteria:

- Conveys the 1:200 year 24-hour duration storm event,
- Minimum freeboard = 0.2 m,
- Minimum ditch grade = 0.01 m/m,
- Side slopes = 1H:1V, and
- Channel shape = V-notch and trapezoidal.

Lining and protection of the diversion channels from erosion and scouring is required for all permanent ditches due to the steep channel gradients associated with the natural topography and the anticipated run-off flow rates. The alignments of the HLF diversion ditches are shown in Figure 18-24. At start-up, (Year -2), a temporary ditch approximately 1 meter wide will be constructed which will divert surface run-off from the HLF pad footprint. This temporary ditch will be decommissioned after the start-up.

Figure 18-24 Surface Water Management Map



As part of the surface water management structures, an Attenuation Pond needs to be constructed upstream of the ultimate leach pad footprint due to the HLF blocking the exit of the valley. Any surface run-off upstream of the HLF would enter the back of the leach pad and create additional contact water to be treated. Therefore, the Attenuation Pond captures surface run-off from snowmelt and storm event and passes these reduced peak flows under the HLF in a HDPE solid Attenuation Pipeline.

The Attenuation Pond is designed to meet the following design criteria:

- Storage capacity to contain surface run-off from the 1:200 year 24-hour storm event without discharging into the environment, and
- Spillway is designed to discharge between the 1:200 year 24-hour storm event and 1:1000 24-hour storm event with a minimum embankment crest freeboard of 1 meter.

18.5.2.13.1. Storage Requirements

The storage requirement for the Attenuation Pond was established based on containment of the entire estimated surface run-off generated from the contributing sub-watersheds during the 1:200 year 24-hour storm event.

Modelling of the average annual run-off was undertaken using the Hydrologic Modelling System GoldSim. The model uses site specific data to accurately capture the specific climate and catchment conditions at site, including storm precipitation intensity distribution, snowmelt, catchment slope, drainage and precipitation losses. Based on the surface run-off results generated by the model, the storage requirements for the Attenuation Pond is 51,700 m3.

Run-off stored in the Attenuation Pond will drain underneath the leach pad, PLS Pond, PLS Overflow Pond and Emergency Pond to the Sediment Pond. The Attenuation Pipeline is designed to empty the 1:200-year storm run-off volume (approximately 51,700 m3) over 24 hours.

18.5.2.13.2. Liner System

The engineered single liner system designed for the Attenuation Pond uses the same design principles as the leach pad liner system. The liner consists of the following layer configuration (Refer to Figure 18-20):

- 2 mm linear low-density polyethylene (LLDPE) geomembrane,
- High Strength Geosynthetic Clay Liner (GCL), and
- 0.3 to 0.6-meter-thick compacted bedding layer (0.3 m layer of silty/clayey gravel on non-scree slopes and 0.3 m transition material between the scree slope and the silty/clayey gravel).

Careful preparation of the ground surface interfacing with the GCL is required to ensure that the conditions are acceptable to install the liner system without compromising its integrity.

Installation of a Leak Detection and Recovery System (LDRS) is not required for the Attenuation Pond as the pond is operated as a dry-facility and will only receive and store run-off water during significant storm events and the water captured by the pond in non-contact water.

18.5.2.13.3. Embankment

For the surface run-off, the embankment will be constructed of colluvial soil borrow materials. Fine grained soils will be selectively utilized on the upstream face to provide a transition zone that is acceptable for GCL installation.

The embankment is designed with a 2.5H:1V downstream slope and a 2.0H:1V upstream slope. These slopes ensure the embankment stability.

18.5.2.13.4. Construction And Operation

The Attenuation Pond will be constructed to full size prior to commencement of the HLF operations. Construction of the earth fill embankment for the Attenuation Pond involves stripping approximately 0.5 m of topsoil and 1 m of overburden beneath the embankment and ponding footprint. The earth fill embankment and pond liner will be constructed on competent soils.

Under typical operating conditions the Attenuation Pond will be operated as a dry pond to ensure that the maximum pond capacity is available for capture of excess surface run-off from snowmelt and storm events. During a storm event surface run-off, water will flow to the attenuation pond. A 2,900 mm HDPE pipe will guide the water from the bottom level of the attenuation pond to the stream.

18.5.2.14. HLF and Ponds Stability Assessment

Analyses have been carried out to examine the stability of the HLF to the final ore heap elevation of 2,482 masl and the HLF ponds. The stability analyses were carried out using the limit equilibrium method in SLOPE/W. In the analyses, a systematic search was performed to obtain the minimum factor of safety from several potential slip surfaces. Factors of safety were calculated using the Spencer method.

Analyses have been performed to investigate the stability of the final heap leach pad and ponds under both static and seismic conditions. A typical cross-section of the heap leach pad and ponds was used in the analyses. The minimum acceptable factor of safety for the heap leach pad under static conditions is 1.3 for short-term operating conditions and 1.5 for long-term (post-closure) of the Leach Pad and Ponds.

The Leach Pad has been designed for closure stage; therefore, a design earthquake corresponding to the Tr = 475 years event was adopted for the design. the corresponding mean peak ground acceleration is 0.23 G.

The consequences of failure of the Pond(s) during an earthquake event are likely to be very high and for the Attenuation Pond is Significant. There would also be significant impact on the Process Plant. A conservative design earthquake corresponding to the MCE event was adopted for the design of the pond(s); accordingly, the corresponding mean peak ground acceleration is 0.49 G. A design earthquake magnitude of 7.5 is selected based on a review of regional tectonics, potential seismic source zones in the region, and historical seismicity.

18.5.2.14.1. Material Parameters

The following parameters and assumptions were incorporated into the stability analyses:

- Unit weights for the heap leach pad and foundation materials were based on typical values for similar materials. Adopted values are included in Table 18-12.

Table 18-12 Stability Analysis: Material Properties

Material	γ (kN/m^3)	γ_{Sat} (kN/m^3)	C (kPa)	Φ (°)
Heap Leach Ore	17.5	19.0	0	34
PLS Pond Rock Fill	18.0	20.0	0	34
Weakest Liner Interface[1]	20.0	21.0	-	-
Dam Fill	18.0	19.0	0	36
Granular Soil	22.0	23.0	0	36

Note The weakest interface was between the GLC and Bedding Interface. In addition, the interface shear strength is a non-linear envelope.

A phreatic surface within the lower portion of the leach pad was modelled with a constant head at elevation 2,384.5 masl to represent maximum potential solution storage during shut-down or snow melt and rainfall. The phreatic surface modelled in the upper portion of the leach pad was assumed to be 2 m over the liner. This is a reasonable assumption since adequate drainage will be provided at the base of the heap to minimise the build-up of pore water pressures within the heap.

The stability of the heap leach pad is controlled by the interface shear strength between the various components of the liner system (overliner, geomembrane liners, GCL, Geonet (PLS and PLS Overflow Ponds only)). It is anticipated that the liner interface between the silty gravel soil and the GCL is the one with the lowest shear strength, and therefore, controls the heap stability.

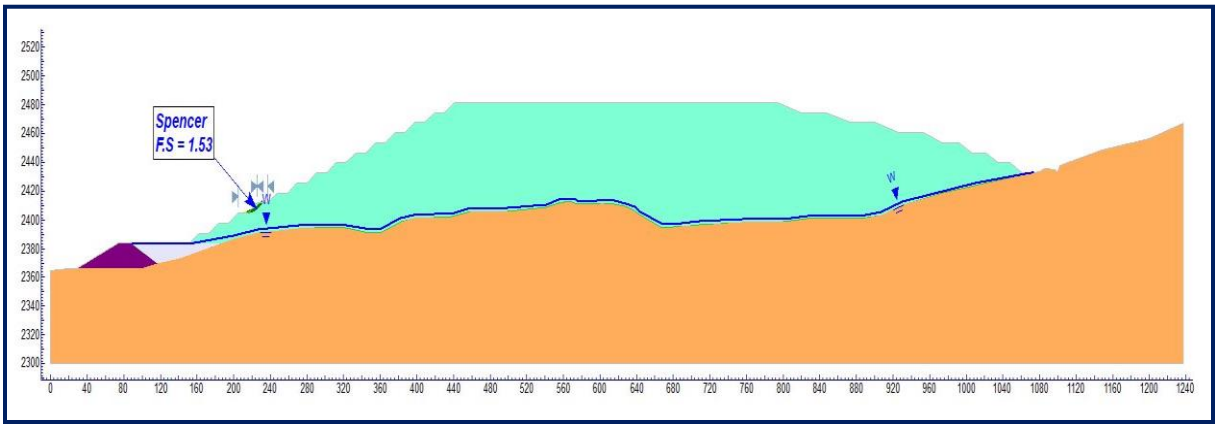
The interface shear strength between the silty gravel soil and GCL liner is defined using the results of laboratory direct shear strength testing conducted on similar geosynthetic, soil and rock materials to be used on site. Laboratory test results (stress-strain plots) were used to define relationships between interface shear strength and normal confining stress for peak, post-peak (15 mm strain) and residual (> 60 mm strain) strength conditions. The post-peak relationship was used for static stability conditions, as it is recognized that some relative displacement between the layers may occur during the construction or operation of the pad. The residual strength value was used for seismic loading conditions from the design earthquake data

18.5.2.14.2. Results Of Stability Analyses

The stability analyses look at both circular and block type failures along potential weak zones within the leach pad and ponds.

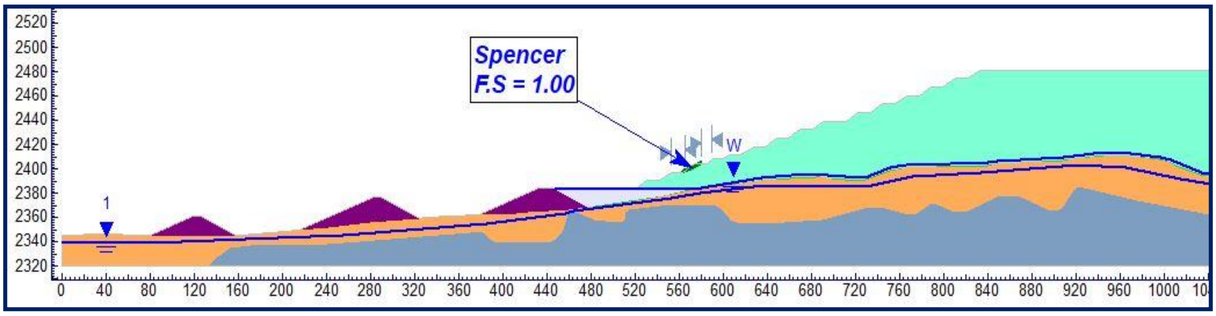
The results of the stability analyses indicate that the leach pad and ponds are stable with minimum static factor of safety of 1.53 for all facilities. The potential slip surface and calculated static factor of safety are shown in Figure 18-25.

Figure 18-25 Minimum HLF Static Factory of Safety



The results of the pseudo-static Stability analysis of the HLF and ponds during earthquake loading indicate that the stability of all facilities with minimum factor of safety of 1.00. The potential slip surface and calculated static factor of safety are shown in Figure 18-26.

FIGURE 18-26 MINIMUM HLF PSEUDO-STATIC FACTOR OF SAFETY



The stability of the HLF is sensitive to the interface shear strengths associated with the liner system. Therefore, laboratory shear strength tests for each of the liner interfaces within the composite liner systems are recommended for detailed design studies, once all material sources have been confirmed.

Ausenco completed additional testwork on the stability analysis in 2020. From the testwork, Ausenco confirmed that no changes to the design of the heap leach pad and ponds would be required. In addition the permeability results for the clay illustrate that this material would make a good low permeability soil layer.

18.5.2.15. CONSTRUCTION AND OPERATION

18.5.2.15.1. LIFT PLACEMENT AND LEACHING

The sequence of lift placement and leaching will be as follows:

- Placement of 38 mm minus overliner material on the surface of the geomembrane liner to provide liner protection and base drainage,
- Placement of 7 m lifts of ore, as required, by haul trucks,
- Spreading the ore with dozers to establish an evenly graded surface for leaching,
- Laying out the irrigation lines for drip leaching. Sprinkler leaching may also be possible during summer months as part of a rotational, cell-type leach operation if there is excess water in the PLS Overflow and Emergency Ponds,
- Burying Irrigation lines (‘drippers’) below surface to depth greater than the depth-of-freeze to prevent freezing during winter operations, and
- Leaching the stacked ore below the irrigation lines prior to loading with the next 7 m lift.

18.5.2.15.2. BORROW MATERIALS

Borrow materials will be required for the construction of the leach pad foundation and ponds. Random fill may also be required for foundation earthworks levelling prior to placing transition and bedding layers prior to the placement of liner system. Borrow material for the overliner may be obtained by screening the ore at the crusher plant. Rockfill for embankment construction may be obtained from local rock excavation, scree slopes or local quarry sources.

Materials for the soil liner and random fill may be obtained from borrow areas within the HLF or areas surroundings the HLF. Additional laboratory tests have been completed to confirm the suitability, availability and quantity of borrow materials for earthwork construction. Several borrow areas have been identified. The borrow areas will be developed by dozing and ripping material down-slope to refusal. Fine grained residual silty sands near surface will be stockpiled or be used as bedding layer materials. Coarse grained material will be utilized as random fill and transition layer in construction of the HLF. The material may also be used as general backfill over bedrock for the leach pad foundation. Ripping or blasting in rock may provide rockfill for the rock fill the construction of the pond embankments, as required.

18.5.2.15.3. Cold Weather Considerations

The Project area's average monthly temperature between November and March is below freezing, with temperatures as low as -35.2 °C. In order to enable year-round operation of the HLF, the Site's winter conditions (snowfall and cold temperatures) were considered during the preparation of the design.

Ore stacking will be conducted a minimum of 330 days/year (design 350 days/year) and leaching 365 days/year. Stacking of the ore will stop during heavy snowfall to reduce the risks and challenges of stacking. Challenges of leaching during winter include reduced leaching efficiency, freezing solution and leachate lines, and freezing ponds. In order to overcome the risk of freezing ponds during winter, all pregnant solution collection will be conducted within the heap leach pad and stored within the PLS Pond within the ore void spaces rather than in an external free-surface pond. The irrigation drip lines will be buried.

The PLS Overflow and Emergency Ponds are designed as dry-ponds to ensure that freezing and associated ice-damage to the pond liner does not occur. Winter operations for the leach pad will likely be modified during extreme cold events to include 'ripping' the frozen ore stacks to promote improved infiltration through the ore.

18.5.2.16. HLF Summary Observations

The heap leach pad will be developed in three phases by loading in successive lifts upslope from the PLS Pond embankment. Bench lift heights of approximately 7 m will be constructed at bench face angles of 1.5H:1V. Benches approximately 11 m wide will be left at the toe of each lift to establish a final overall slope angle of approximately 3H:1V. This will provide stability of the heap and allow for on-going reclamation during operations.

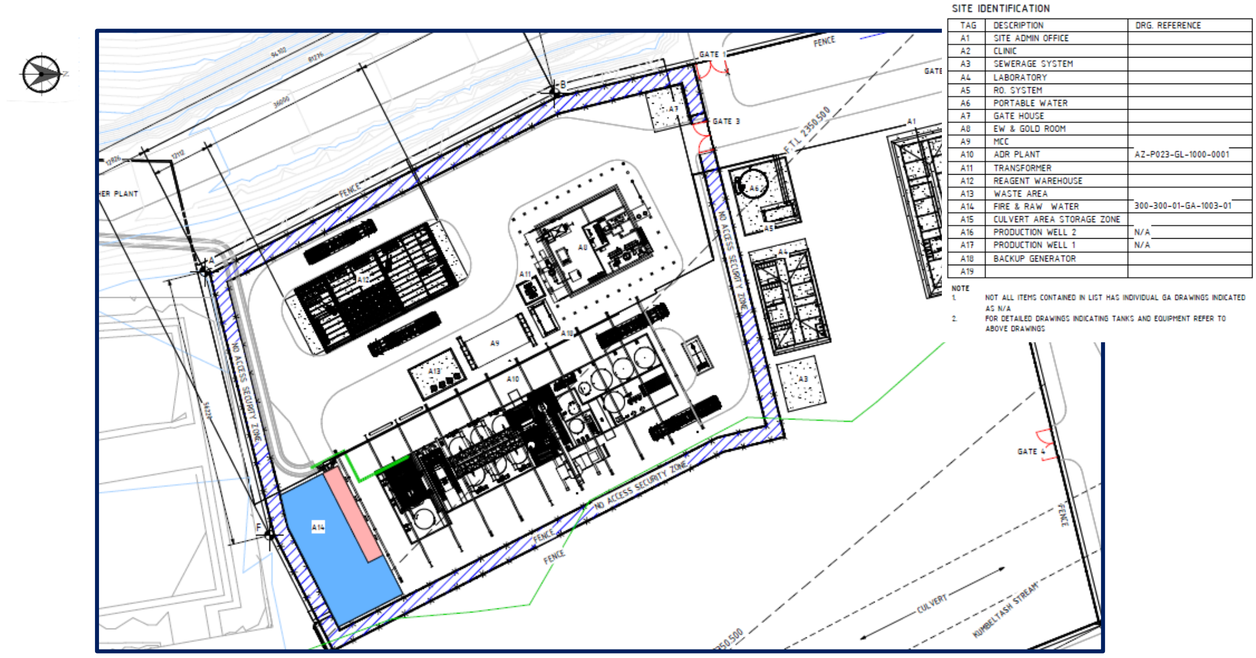
18.5.3. ADR Platform

The process area location was chosen to be as close to the HLF as possible, while also making use of the elevation change to allow for gravity flow of the pregnant solution to feed the process area. The process area pad design was oriented for the most efficient cut-and-fill, and to make use of the valley contours.

The ADR Platforms comprises two parts. The main, northern section, area is a secure zone accessed via a guard house by pedestrian traffic only. In this zone are located the ADR building, gold room, reagent mixing and reagent storage and water storage.

In the Southern section, support services such as administration / management, the clinic and the laboratory are located in portable buildings.

FIGURE 18-27 GENERAL LAYOUT OF THE ADR AREA SHOWING ADR ZONE AND SUPPORT ZONE



SITE IDENTIFICATION

TAG	DESCRIPTION	DRG. REFERENCE
A1	SITE ADMIN OFFICE	
A2	CLINIC	
A3	SEWERAGE SYSTEM	
A4	LABORATORY	
A5	RO. SYSTEM	
A6	PORTABLE WATER	
A7	GATE HOUSE	
A8	EW & GOLD ROOM	
A9	MCC	
A10	ADR PLANT	AZ-P023-GL-1000-0001
A11	TRANSFORMER	
A12	REAGENT WAREHOUSE	
A13	WASTE AREA	
A14	FIRE & RAW WATER	300-300-01-GA-1003-01
A15	CULVERT AREA STORAGE ZONE	
A16	PRODUCTION WELL 2	N/A
A17	PRODUCTION WELL 1	N/A
A18	BACKUP GENERATOR	
A19		

NOTE
1. NOT ALL ITEMS CONTAINED IN LIST HAS INDIVIDUAL GA DRAWINGS INDICATED AS N/A
2. FOR DETAILED DRAWINGS INDICATING TANKS AND EQUIPMENT REFER TO ABOVE DRAWINGS

The ADR pad houses the administration building, laboratory, goldroom, ADR plant and building, reagent storage areas, raw/fire water tank, barren solution sump and pumps, and parking areas.

The ADR Pad is accessed by a road from the north of the pad, entering the facility adjacent to the administration building and parking area (see Figure 18-27).

18.5.3.1. EARTHWORKS AND GENERAL

The process area pad will be a level, graded and compacted fill, approximately 165 m by 90 m, located to the north of the HLF and ponds. The northeast corner of the process area will intersect with a water exclusion zone that prohibits the construction of permanent facilities. The graded compacted fill is not considered a permanent structure, and the area falling within the water exclusion zone will be used as parking for process and administration staff vehicles and buses.

As external temperatures on site can reach – 35 °C during winter, allowance has been made to ‘winterise’ pipes and tanks as necessary, especially equipment located outside buildings. By ‘winterise’ is meant burying (pipes), insulation or heat tracing or a combination, to prevent them from freezing.

18.5.3.2. ADR PLANT

The ADR plant is situated in a north-south direction to account for topography and prevailing winds running through the valley. The portal frame will be clad with sandwich panels to provide insulation and weather protection.

The ADR plant building consists of a portal framed building with an overall footprint of 70 m by 29 m, clad with sandwich panels. The building will be 20 m high to the eaves, with a double-sloping roof. The floor of the building will be bunded, with bunds, catchment sumps and sump pumps collecting area specific spillage and returning it to process. The building will not have an overhead crane, but instead will rely on monorails, hoists and mobile cranes to service equipment.

There will be four external personnel access doors and five external vehicle access roller doors and two internal vehicle access doors. Internally, there will be a structural space frame approximately 25 m by 12 m by 12 m (H) to support the ADR process equipment.

The building will house the CIC circuit, elution, acid wash, carbon regeneration, reagent mixing, compressor, and ‘E-House’.

The ADR plant will be a secure area. The area will be fenced, and access controlled by a manned security gate, allowing access for pedestrians only, with the exception of escorted reagent deliveries, and maintenance equipment/spares

Within the secure ADR plant area, the goldroom will have further security controls and will be fenced. The goldroom will have a double fence with no-man’s land between the fences, and will have a further security checkpoint to access the building. The reagent mixing and storage tanks may be fenced.

18.5.3.3. Goldroom

The Goldroom will be a double story flat concrete building and concrete roof with a footprint of 15 m by 15 m. The building will be 7 m high. There will be two vehicle access doors on the ground floor with one personnel access door on the ground floor one on the top floor with an external steel access staircase.

The goldroom will have its own perimeter fence and security guard room. The goldroom will have a bullion vault to store the Doré bars produced prior to collection.

The goldroom building will have two access doors; one pedestrian access doorway and an interlocked double roller shutter door for vehicle access. The pedestrian door from outside the building will lead to the ablutions and changeroom, with access into the goldroom through a security checkpoint and metal detector. The change room will be equipped with washing facilities for both personnel and PPE.

The vehicle access interlocked roller shutter doors allow the security gold transporter to park up in a secure area before the external roller shutter door is closed and the driver and security staff pass through the security checkpoint. Once inside the building and with the external roller shutter door closer, the internal roller shutter door can be opened to allow the Doré bars to be loaded into the vehicle.

18.5.3.4. Reagent Mixing and storage.

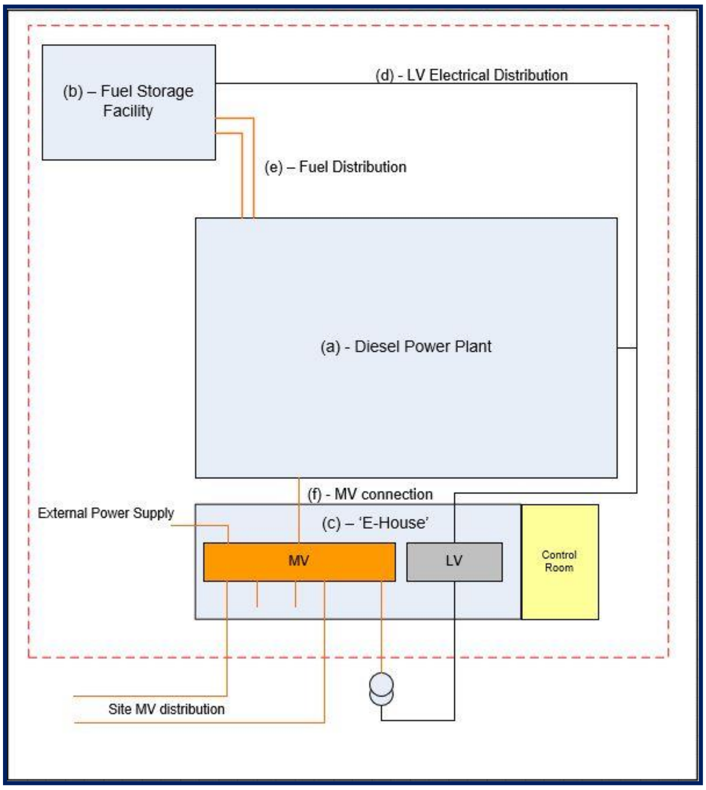
The cyanide mixing area will be in an enclosed, ventilated building within the fenced cyanide bund to the southern end of the process facility. A steelwork structure will support the mixing tank, bag breakers, cyanide storage tank and monorail.

The storage areas for cyanide and for the other reagents will be separately bunded and fenced. Each area will sit on its own dedicated concrete pad with a catchment sump to contain any spillage. Budget provision has been made for a cyanide reagent storage building if regulations require this.

18.5.4. Power Station

The Power Station pad houses a number of different facilities as shown by the diagram in Figure 18-28.

Figure 18-28 Main Components of the Power Station Pad



18.5.4.1. Power Generation

Electrical power for the Project will be provided by a modular diesel generating station. The generator station will be supplied by a third-party power generation contractor to be appointed at a later date and shall be a stand-alone package equipped complete with engine generators, protection and controls, step-up transformers, and a 10 kV distribution switchgear board for interface to the rest of the facility. This will include fuel piping systems and pumps to generator sets and a lubrication piping, pumping clean and waste lubrication oil storage system. The electrical load is summarised in Table 18-13.

The diesel fired power station will be located to the east of the secondary and tertiary crushing facility. The gensets will have individual 0.4/10 kV step up transformers, which will connect onto a common 10 kV switchgear assembly. This will be located in an air-conditioned pre-fabricated container included within the power station package scope of supply.

The power station has sufficient redundant capacity to ensure that runtimes at or above the specified 99.5% are achieved, and to cover maintenance.

The power station comprises containerised generating sets, step-up transformers and 10 kV switchgear room, based around a 20 ft container footprint, as shown in Figure 18-29.

FIGURE 18-29 PROPOSED GENERATOR AND ELECTRICAL LAYOUT

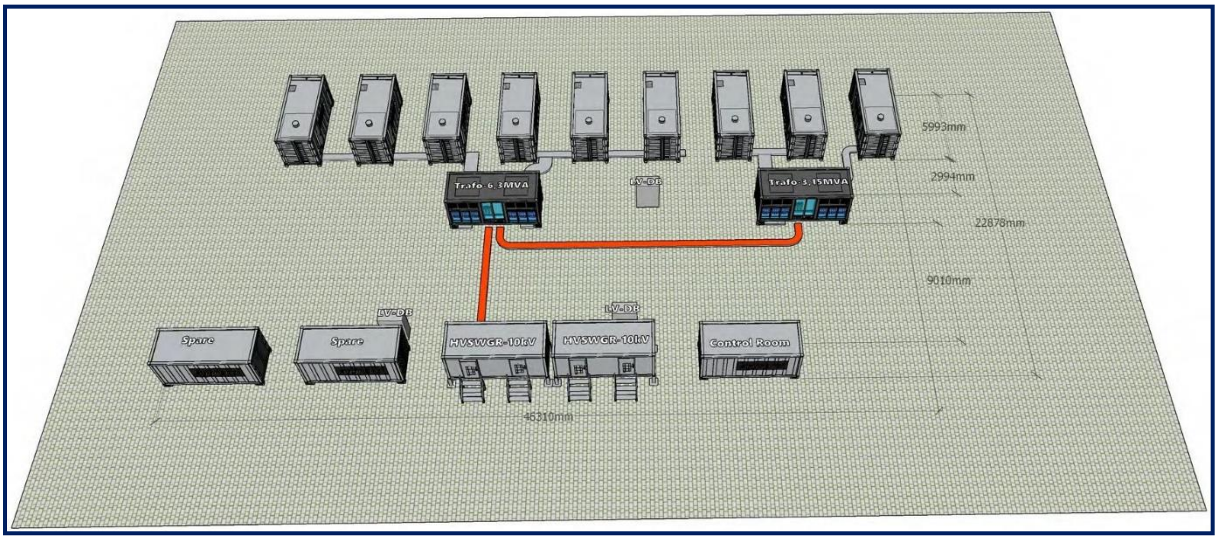


TABLE 18-13 ESTIMATED ELECTRICAL LOADS SUMMARY

Description	Units	Quantity
Estimated Average Power	kVA	4,464
Estimated Maximum Demand	kVA	7,6000
Generator Capacity	kVA	8750
Generators Required for Normal Operation (includes spinning reserve)	-	4
Cold Reserve/Standby	kVA	1,750
Total Number of Generators Required	-	5
Estimated Average Power	kVA	4,464

18.5.4.2. FUEL FARM

A fuel farm will be constructed to Kyrgyz standards next to the power generation sets. Storage capacity will meet the requirement of 4 days operating and 10 days reserve. The fuel farm will be connected by fuel lines, instrumentation and power to the main generation facility. The fuel usage has been estimated as shown in Table 18-14.

TABLE 18-14 FUEL USAGE ESTIMATE

Consumer	Units	Value
Power Station	ℓ/d	21,200
Mining Equipment	ℓ/d	23,000

Consumer	Units	Value
Other Vehicles	ℓ/d	2,730
Eluant Heating	ℓ/d	1,430
Regen Kiln	ℓ/d	600
Smelting Furnace	ℓ/d	240
Camp Heating	ℓ/d	2,500
Total	**ℓ/d**	**51,700**
Strategic 10-day storage	m^3	517
Working 4-day storage	m^3	207
Total/2 weeks	**m^3**	**724**

18.5.4.3. 'E-House' and LV Distribution

18.5.4.3.1. Motor Control Centres

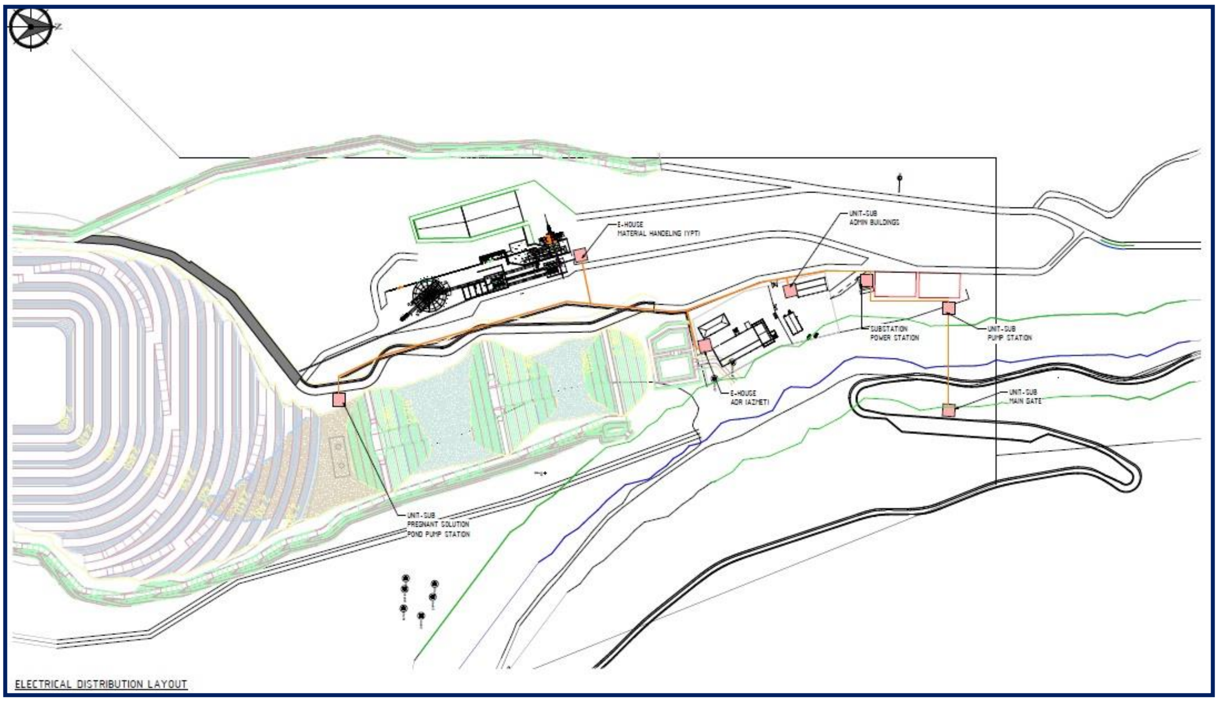
The low-voltage MCCs and distribution boards will be fully installed in air-conditioned E-houses prior to shipment to site. Each room will be maintained at temperatures in the range of 10°C (winter) to 30°C (summer) with suitable air conditioning and heating. The E-house buildings for the MCCs and low-voltage distribution boards will be mounted at a minimum of 1,500 mm from the finished ground level to allow bottom entry access for all cabling.

Low-voltage variable speed drives (VSDs) and soft starters will be supplied as part of the low-voltage MCC package. All VSDs supplied will eliminate low-order harmonics using either an active rectifier or a mains passive filter.

18.5.4.4. Power Distribution

MV power will be distributed to consuming facilities as shown in Figure 18-30 below in a ring main configuration to provide redundancy.

FIGURE 18-30 MV ELECTRICAL RETICULATION



There will be two outgoing 10 kV radial feeder circuits which will form a ring feed to the following areas:

- Power Station;
- Main Gate;
- Water Pumping Area;
- Crushing Plant;
- ADR Facility;
- Offices; and
- Heap Leach Return Pumping Station.

The outgoing feeders will supply local area substations which will house switchgear and step-down transformers to feed MCCs and facilities at 400 V.

Power will be reticulated from the radial feeders via a 10 kV wooden pole OHL transmission system.

Conveyor support structures, building support structures and process equipment structures will be utilised for medium-voltage and low-voltage cable reticulation in the wet and dry process areas as far as practicable. Cable ladder shall be utilised for all above ground cabling with segregation between different category cables. Buried cable will be minimised.

Three phase low-voltage supply to MCCs, motors and other services will be 400 V.

18.5.4.5. Other Electrical

18.5.4.5.1. Earthing

The earthing system will be a “TN-C-S” system as defined in IEC 60364.

The main earthing electrode system comprises localised earth grids and electrodes around the generators and main switchroom tied by means of polyvinyl chloride (PVC) insulated 120 mm^2 copper cables to the switch room main earth bar.

The underground earth grid will be installed around and below the generators to prevent dangerous step and touch potentials developing under fault conditions.

Rods will be driven in sufficient numbers and to sufficient depth to achieve less than 1 ohm for substations/MCC rooms, and less than 10 ohms at connection points for lightning protection.

A Neutral Earthing Resistor (NER) will be installed at the power station to protect against and limit fault current scenarios.

18.5.4.5.2. Cables

Cables will be sized and selected according to IEC 60364. Cables will be installed on heavy duty, galvanised cable ladder mounted on building and support structures.

18.5.4.5.3. Lighting and Small Power

A limited number of road lights may be placed in strategic locations around the processing area.

Small power outlets will be provided throughout the process area. Lighting and small power distribution boards will be located in electrical rooms and process areas and these will distribute power at three phase and single-phase voltage as required.

18.5.4.5.4. Vendor Packages

Where Vendor packages are installed in the plant, equipment will conform to site standard and software will be integrated into the area process control systems.

18.5.5. Process Services (Utilities)

18.5.5.1. Raw Water Supply

A borehole pump in a borehole located adjacent to the Kumbeltash Stream will pump fresh water through a buried pipeline to the raw water tank at the process area. The pipe will be buried to prevent the water from freezing during winter. The raw water tank will be a fabricated steel tank, 9.5 m in diameter and 15.0 m high, with a capacity of 1,000 m^3.

Exploration wells have been drilled, and extended ground water tests, followed by water pump tests have been completed. The indicated available water availability of up to 2,590 m^3/hr, significantly exceeds the plant requirement of 1,850 m^3/day.

The raw/fire water tank, will be located outside the ADR plant building at the southern end.

18.5.5.2. Fire Water

The fire water tank will be a dedicated lower section of the raw water tank. The fire water tank is the feed for the fire water system, which consists of a fire water ring main, fire water pump, jockey pump and diesel-powered backup pump. The fire water ring main will run through the process area building and goldroom and will be provided with fire hose reels at the required locations.

18.5.5.3. Potable Water

The main potable water plant is located at the 360 Man Camp.

The ADR Plant will have a potable water tank and its own small treatment equipment. Potable water will be distributed around ADR as required.

The Admin area, the laboratory buildings, and other locations such as the main gate will have their own tanks which will be filled periodically by the potable water truck from the camp.

18.5.5.4. Sewage Treatment

The admin and lab area will be provided with a buried septic tank adjacent to the administration building. As necessary, a sewage tanker will empty the septic tank and transport the sewage to the sewage plant located at the camp for processing.

A septic tank will also be installed in the ADR and Crusher areas and emptied periodically

18.5.5.5. Compressed Air

Plant air and instrument air will be provided by a compressor both in the ADR plant and in the Crushing area, and distributed as required. The laboratory will have a dedicated duty/standby compressor. Air receivers will be provided as necessary.

18.5.6. Process Infrastructure

18.5.6.1. Buildings

18.5.6.1.1. Site Administration Building

The site administration offices are located to the southern end of the ADR pad. They will be sized to accommodate on-site administration requirements and will provide office space, meeting rooms and clerical space for the site-based G&A staff. The offices will be constructed from portable buildings, and will be insulated and heated to provide a comfortable working environment.

The site administration modular buildings will provide offices for staff, as well as meeting rooms, boardroom, kitchenette, toilet facilities and a medical facility.

The medical facility will consist of a reception area, a consulting room, an emergency room, a surgery (minor procedures) room, storage facility for medical equipment and medical waste, and pharmaceutical drugs store. The ambulance will have a designed parking space with unimpeded access in all direction.

18.5.6.1.2. Crusher and ADR

Both the Crusher and ADR plants will be equipped with offices (portables) and toilet facilities in portable containers.

18.5.6.1.3. Process Workshop / Warehouse

The ADR area will be provided with temporary containers for maintenance repairs and for parts storage.

The crushing area has by far the highest demand for maintenance and parts and will be equipped with a reasonably substantial combined workshop and warehouse. This will either be a small steel building with cladding and insulation or a temporary facility constructed from containers.

18.5.6.1.4. Laboratory

A fully equipped laboratory will be established on the ADR Platform in the Southern half of the footprint,

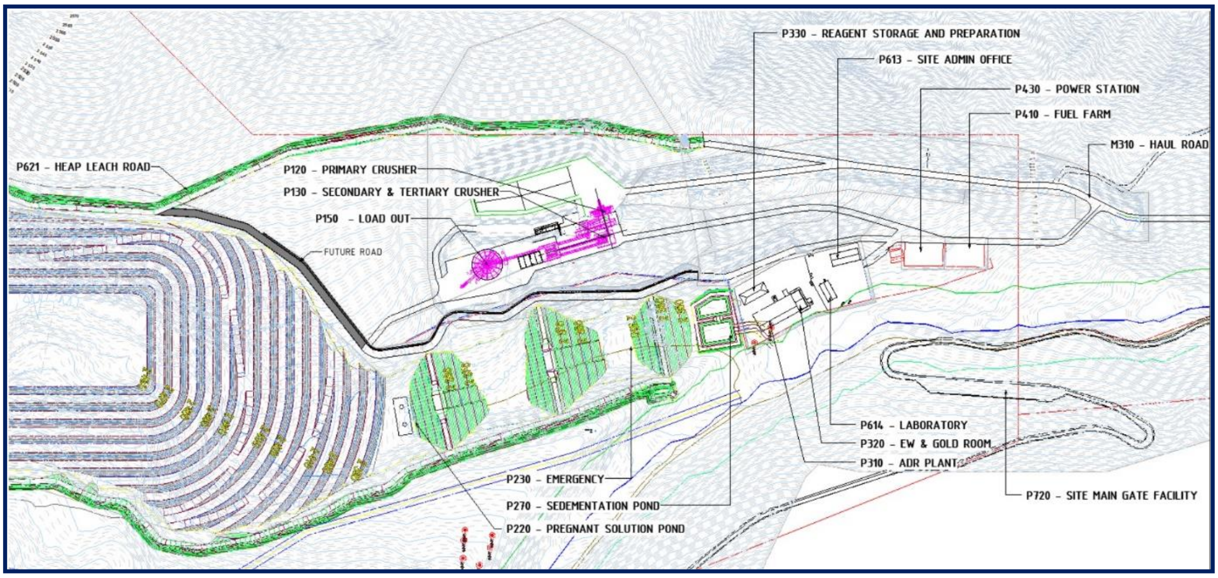
The laboratory will be containerised similar to the admin facility. It will handle both mine grade control samples and metallurgical samples from the ADR and crusher. It will include a small metallurgical laboratory.

18.5.6.1.5. Process Roads

The location of the principal roads in the process area is shown in Figure 18-31:

- HL Roads – East, West, Central;
- Load out Road;
- Crusher and ADR access Roads;
- Ponds Road; and
- Mine Haul Road Extension.

FIGURE 18-31 LOCATION OF THE PROCESS AREA ROADS



18.5.7. GENERAL

18.5.7.1. PROCESS CONTROL

The budget allocation for process control covers the process control system and all other IT type requirements in the process area, such as fire, gas, and emergency alarm systems

The Process area will be divided into two control areas that will run independently of each other – Crusher and ADR (including water). The system will consist of PLC's and HMI's/ SCADA. As the project is relatively small the cost and complexity of a DCS package is not warranted

PLC make, family and programming software will be established as a site standard that would simplify maintenance and reduce training requirements. As an example, Siemens S7-1500 PLC's for areas and S7-1200 for field PLC's along with TP1900 Comfort HMI's for visualization all programmed in the Siemens TIA portal software package.

The visualization system can be migrated from standalone HMI to a fully redundant SCADA system if required in the future. Using the Siemens system all HMI work can be reused and incorporated in the Scada system thereby saving on engineering.

Fibre-optic networking will be required to RIO panels, inter-plc communications and management visualization view only access.

18.5.7.2. TOOLS AND EQUIPMENT

Budget provision has been made for crusher workshop tools and maintenance personnel tool boxes.

18.5.7.3. Special Safety Equipment

Budget provision has been made for specialised safety equipment, including but not limited to HCN Monitors, Mercury Detector, CN Mixing PPE, and Smelting PPE.

18.5.7.4. Training

Training for operators and maintenance personnel represents a critical success factor for start-ups.

Budget provision has been made for the hire of external resources for this phase.

18.5.7.5. Security

18.5.7.5.1. Site Gatehouse

Once vehicles pass the Chatkal Station yard, vehicles can only enter site via the mine access road. The site gate house is located where the access road enters the site.

The site main gate will house Security, the emergency response team, induction room, truck weigh scale, and firetruck.

An area will be designated as suitable for a helicopter to land on and take off from, safely and easily, in case a medical emergency evacuation is required. The helipad will be in a flat sectioned off area and clearly marked with an "H" on the ground for easy visibility from the air.

18.5.7.5.2. Process Gate

The ADR area is a secure area with pedestrian access only, via a security container provided with turnstiles and search rooms in each direction.

18.5.7.5.3. ADR/Gold Room Security

Budget provision has been made for all the additional items required for a secure gold room operation excluding the building which is covered elsewhere. This includes vault door, safe, stand-alone CCTV cameras etc.

18.6. Owner

18.6.1. Temporary Facilities

A batch plant will be required for concrete supply on site for foundations for structures and mechanical equipment, as well as slabs, bunds and mass concrete as required. The batch plant will be supplied erected and operated by the civil contractor.

Construction water will be available from the Sandalash River which is perennial and thus has water all year round. Construction water can be drawn from the river and transported via bowser to temporary tanks also supplied by the contractor(s) and placed strategically as required.

A construction laydown area will be prepared by the civils contractor and shared by all the contractors. There will not be a fence around the laydown area, but the contractors may supply security personnel to safeguard their plant and materials if deemed necessary.

18.6.2. Site Vehicles

There will be three general categories of vehicles on the mine site:

- Mining vehicles will all be provided by the mining contractor and will include all vehicles required to operate the pit, haul and heap leach operation. This will include, but is not limited to, haul trucks, excavators, wheel loader, dozers, graders, compactors, drilling machines, HIAB trucks, forklift, maintenance, water and fuel trucks, lowbed trailers and tractor units and 4x4 pickups;
- Plant area vehicles will all be provided by the Owner and will include all vehicles required to operate the plant area (both crushing and process). Vehicles include 1.5 t forklifts, 35 t mobile crane, container handler, skid steer, telehandler, 4x4 pickups; and boom truck;
- Vehicles required for site-wide services will be provided by the Owner. These include fire truck, ambulance. Mountain rescue vehicle, 22-seater buses for personnel transport, vacuum truck (honey sucker), and 4 x 4 pickups.

18.6.3. Communications

The site wide data and voice telecommunications for the project is provided by a cellular network provider. This comprises a telecommunications tower at the Chatkal, another at the Kumbel Pass and a final tower at the main gate house, which house the hardware for cellular communication.

A site local area network (LAN) to connect to the cellular network will not be installed.

The communications systems for the Project will include:

- Site-wide Mobile phone system;
- Camera surveillance system; and
- Site-wide radio network, including installation transmitters on the cell phone masts.

19. MARKET STUDIES AND CONTRACTS

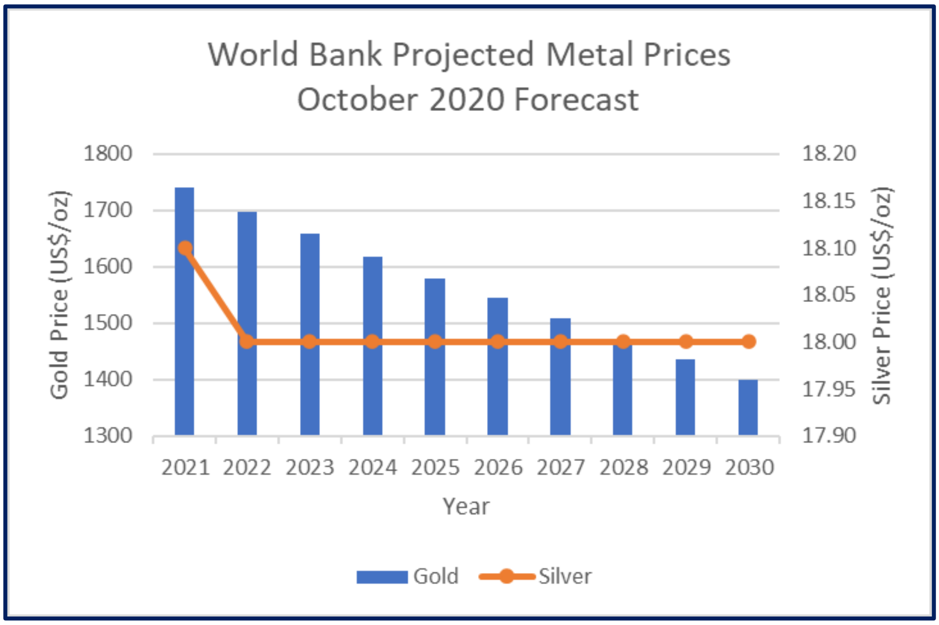
19.1. METAL PRICES

The metal prices used for the Tulkubash 2021 Feasibility Study Update were USD1,450/oz for gold and USD17.50/oz for silver. The prices are aligned with Chaarat Gold's long-term outlook for gold and silver and are based on forecasts by Bank of America – Merrill Lynch in H2 2020.

Graph 19-1 shows metal price forecasts by year from 2021 to 2030 from the World Bank Commodity Forecast issued in October 2020. The average gold prices over the planned life-of-mine are forecast to be about USD1,565/oz. This indicates that the gold price applied in the feasibility study could be considered conservative.

The current outlook contrasts with the prevailing consensus forecast at the time of the 2019 Feasibility Study, when gold price over the life-of-mine was projected to average USD1,300-USD1,320/oz Au.

GRAPH 19-1 WORLD BANK PROJECTED METAL PRICES OCTOBER 2020



The average World Bank forecast price of by-product silver is USD18.00/oz over the life-of-mine. This indicates the silver price used in the study is also conservative. As a by-product, however, the value of silver has little influence on the value of the project.

19.2. DORÉ

19.2.1. SALES

Under Kyrgyz legislation, gold and silver produced in the Kyrgyz Republic must be offered for sale to the National Bank of the Kyrgyz Republic (NBKR) at spot market prices. If the NBKR declines to purchase the metal, Chaarat is free to sell it on the international market.

Tulkubash Doré (contained gold and silver) will be delivered to the Kyrgyzaltin JSC (Kyrgyzaltin) refinery at Kara Balta, Kyrgyzstan, for refining. Kyrgyzaltin is a state-owned gold company and a member of the London Bullion Market Association. For Doré deliveries of more than 50 kg, a refining charge of USD0.26/g or USD8.06/oz is charged. Full payment is anticipated within 7 days of dispatch from the gold room.

The average monthly Doré production from Tulkubash will be more than 100 kg.

19.2.2. Transport

A secure carrier Licenced in the Kyrgyz Republic will transport the Doré from Tulkubash to Kara Balta monthly. Monthly shipments will have a fixed cost of USD1,400 per trip plus a variable cost equal to 0.03% of the value of the metal transported. All prices are subject to VAT. This estimate is based on a quotation received in 2018.

19.3. Contracts

Chaarat will have several significant Service and Supply contracts, the largest of which is the Mining Contract. There will also be an important service contact for Power supply, as well as some high value Supply Contracts such as Fuel, Cyanide and some other consumables.

19.3.1. Service Contacts.

19.3.1.1. Mining Contract (Pamir Mining)

Chaarat Gold (CG) has engaged Pamir Mining, designated entity of Çiftay Inşaat (Çiftay) as its mining and construction earthworks contractor. Çiftay is a major construction and mining contractor in Turkey. Çiftay will provide services to Chaarat through its 100% owned Kyrgyz-based subsidiary, Pamir Mining. Chaarat's senior management has previous experience of Çiftay's capabilities from the development of the Alacer Gold Project in Turkey. The mutual confidence developed during this project provides a basis for Çiftay's decision.

19.3.1.1.1. Scope

Çiftay has two main contracts with Chaarat for earthworks construction and contract mining. The earthworks contract includes construction and improvement of site roads, facility platforms, and the heap leach pad.

The mining contract includes mine development, pre-stripping, and production mining of ore and waste. It also includes ore hauled from the mine area to the ROM Pad, and ROM Pad management. The mining contract is estimated to be worth more than USD140 M over the approximate 5-year mine life.

In addition to the earthworks and mining contracts, Çiftay will perform a number of other activities on a contract basis. These include haulage and stacking of crushed ore on the heap, maintenance of on and off-site roads, and management of the permanent camp.

19.3.1.1.2. Contractor background

Çiftay Inşaat is a large Turkish construction company specializing in the contract mining, construction, energy, and the hospitality industries. Annual turnover is more than USD100 M. The origin of the company's business is in contract mining and construction. Çiftay's current mining and bulk materials handling contracts include -

- Çöpler Gold Mine, Alacer Madencilik SA, Erzincan;
- Erdenmir Iron Mine, Erdenmir Madencilik SA, Sivas;

- Öksüt Gold Mine, Centerra Gold, Kayseri;
- Kurumu Coal Mine, Turkiye Komur Islemeleri Kurumu SA, Manisa; and
- Clay Handling Operations, Tracim Cimento, Kirklareli.

Çiftay also has considerable construction experience related to the establishment of new Mines. At the Çöpler and Öksüt mines, Çiftay was responsible for construction earthworks, erection of buildings and structures, and construction of lined leach pads and tailings facilities. Çiftay has also erected and operated man-camps at various sites.

19.3.1.1.3. CONTRACTOR CAPABILITY

Çiftay's operating capability is well suited to the development of the Tulkubash deposit. Çiftay has a demonstrated ability to employ small construction-size equipment to deliver high rates of production. At the Çöpler Gold Mine, Çiftay operates more than one hundred (100) 30 t and 40 t trucks to move over 100,000 tpd of ore, waste, and stockpiled plant feed.

FIGURE 19-1 ÇIFTAY TRUCK FLEET AT ÇÖPLER GOLD MINE

Çiftay's ability to operate small equipment at high efficiency will allow Chaarat to minimise strip ratios despite the steep terrain at Tulkubash, and to deliver feed to the plant at higher rates and lower cost than would be typical for most operations in these circumstances. Furthermore, Çiftay's operating experience at the Çöpler Mine has provided Çiftay with an appreciation for the importance of ore control in gold mining operations, which distinguishes them from many other contract earthmovers.

Çiftay is Caterpillar's largest customer in Turkey and, as such, commands preferential pricing for capital equipment purchases. Çiftay has similar relationships with its preferred truck supplier, Mercedes Benz, and drill supplier, Atlas Copco. Çiftay can leverage its long-term relationships with suppliers to offer competitive rates which reduce capital costs for Chaarat.

Turkey and Kyrgyzstan have many cultural similarities which will allow Çiftay to communicate with, train, and relate to the local workforce.

19.3.1.1.4. Contract Structure

The earthworks contract is a standard 'form' agreement based on unit rates for excavation, transport, and placement of different types of material. Payment is based on volumes moved at the applicable rates. There is a mobilization cost for the contractor to establish the required equipment and facilities on site.

The mining contract is structured differently to the earthworks contract. The mining contract is based on a variable unit rate which increases with each additional 5 M bcm for the first 25 M bcm. Above 25 M bcm, the unit rate remains constant, though the total cost may be adjusted up or down depending on the length of hauls.

The unit rates for both earthworks and mining include the cost of all administration, maintenance, and management of the contractor's workforce. The unit rates do not include fuel and VAT which are added to establish the total cost to Chaarat. Chaarat will provide fuel to the Contractor on a free-issue basis.

19.3.1.2. Power Station Contract

Chaarat has selected contract (rental) power supply as the preferred method of power generation on site. The main supply parameters are shown in Table 19-1 below.

Table 19-1 Power Supply Parameters

Parameter	Value
Duration	5 Years
Power Capacity (Installed)	11,250 kVA (including reserve to achieve >99.5% reliability)
Power Capacity (Continuous)	~6,840 kW (depending on ambient temperature and elevation)
System Voltage & Steady State Accuracy	10 kV
System Frequency & Steady State Accuracy	50 Hz
Operating Configuration	Baseload
Site Ambient Temperature	Maximum +38 Minimum -35 ^{0}C
Site Altitude	2,500 masl
Applicable site rating of the Generator Sets (Genset)	KTA50-G3 1,250 kVA Diesel Genset
Redundancy %	47% (for service and emergency purposes)
Total Number of Genset	9

Guaranteed fuel consumption is 0.261 ℓ/kW. Rental cost is USD2,790 per day with an option to defer the first 6 months' rental costs. This corresponds to a cost of USD0.193/Kwh. If the deferred cost option is taken, then the daily fee would be adjusted to USD3,100 for the remaining 4.5 years. Site mobilisation and de-mobilization costs are included in the daily fee.

19.3.1.3. Logistics Contract

Chaarat has well-established relationships with several regional and international freight forwarders, including Globalink, Move One, and TransAsia Logistics.) These service providers will be on an as-needed basis.

19.3.1.4. Supply Contracts

19.3.1.4.1. Fuel

The fuel will be procured from several sources and will cover all requirements on site, i.e. - mining equipment, power generation and drill rigs. The diesel classes of EURO 3 and above will be used.

The fuel supply contractor has established a fuel storage facility (mobile fuel tank, 50+ t) on the Chatkal side of the Kumbel pass. Larger fuel trucks (up to 40 t) deliver diesel to this storage facility. Smaller fuel trucks then transport fuel to the fuel storage facility at the Tulkubash site. The capacity of the storage facility can be expanded as needed. After the upgrade of the Kumbel Pass to Site Access Road, this storage facility may become redundant.

19.3.1.4.2. Contract Terms

The Contract will be min 1-year duration, with an allowance for monthly price revisions according to an agreed procedure. The cost of fuel for the BFS is USD0.60/ℓ delivered including VAT. To the extent possible, hedges will be initiated to manage the variation in cost between winter and summer fuel. Fuel quality control conditions will be included in the contract.

While the intent is to source fuel from a single supplier, Chaarat will negotiate contract terms with other suppliers for immediate replacement of supply routes in case the main contract is terminated for cause – for example due to persistent quality problems.

Diesel consumption during operations is estimated to be about 87 m^3 per day.

19.3.1.4.3. Cyanide

Cyanide will be sourced from Russia, China or other cyanide producing countries delivered to site in 980 kg wooden IBC containers.

The intent is to negotiate a one-year renewable Contract with an annual review provision whereby revisions and extensions may be negotiated. Termination may be invoked in the case of poor performance, for example failure to maintain adequate quality, and failure to meet delivery schedules.

Estimated cyanide consumption is 0.60 kg/t ore, which corresponds to about 3,000 tpy. An operating supply of approximately 2 weeks will be maintained on site.

20. Environmental Studies, Permitting, and Social or Community Impact

Chaarat appointed WAI to carry out an ESIA analysis of the Project as part of the Feasibility Study. Work carried out thus far includes site visits by several members of WAI's technical team and review of the Project's environmental and social baseline documentation, and laboratory test work.

WAI's review of the Project environmental and social performance is based on data provided by Chaarat and Tetra Tech, as well as from observations during several WAI site visits to the Project area during Q3 2016, including by WAI's Environmental and Social team. No updated site visit has been undertaken in 2018 due to seasonal weather conditions preventing access to site.

A preliminary ESIA was completed in Q3 2017 and updated in June 2018, when the results were disclosed to the community at a public hearing. This further updated ESIA was completed in Q3 2020 and is based upon the LogiProc 2019 BFS project description. Documents and reports were inspected on site and within an extensive data room, which continues to be updated. Key documents used in the production of this report are noted in Section 27-References.

While WAI believes it has gained sufficient insight into the key issues and performance, there may be additional information that was not seen, or variations in interpretation of the available data that could not be explored further. These missing data relate to aspects that have yet to be determined within the Project design. The engineering design that was provided, had the following progress:

- Principal Engineering 47% complete;
- ADR Plant 90% complete;
- Crusher Plant 60% complete; and
- Heap Leach 100% complete.

The remaining design is not expected to have a significant impact on the outcomes of the BFS.

20.1. Environmental Setting

20.1.1. Location

The Chaarat Property is located at latitude 42° 1' 6.91" N and longitude 71° 9' 39.04" E in the Sandalash Range of the Alatau Mountains in the Chatkal district of the Jalal Abad Province of western Kyrgyzstan, close to the border with Uzbekistan and approximately 300 km southwest of the Kyrgyzstan capital, Bishkek.

20.1.2. Topography, Land Use and Land Cover

The Chaarat Property area is characterised by extreme topography ranging from the Sandalash Valley, running along the Property at an elevation of 2,200 masl to 2,100 masl to the mountain ranges on both sides, which peak at an elevation of 3,800 masl. The Sandalash River runs through a relatively narrow valley (100 m to 500 m wide at river level). There is no permanent population residing within 20 km of the Project area. The Sandalash River follows a linear south-westerly trend, with a moderate gradient in the Property area, and with intermittent rapids between swiftly flowing segments. The Sandalash River flows into the Chatkal River, south of the Property area at Jany-Bazar.

The dry valley lies between the eastern and western slopes of the Sandalash Mountain Range, which in this area winds south, and then east, before continuing in its general southwest orientation. Topographically, the Dry Valley is an open space varying in width from 150 m to 300 m, and is not altogether level due to raised sections along the sides of the valley reaching heights of tens of meters. The eastern slopes of the valley are steep, exposed rock, with no evident surface water channels. The western slopes are also steep, but still have some soil and vegetation. These western slopes also have no evident water flows, except for their northern edge.

The Project's total land allocation is 1,394,000 ha.

20.1.3. CLIMATE

Climate records are available from weather stations located in the exploration summer and winter camps at the proposed mine site. The temporary camps have been constructed within the Sandalash Valley and are approximately 5 km apart, with the winter camp furthest to the north. There are two additional weather station locations: one at Jany-Bazar Village, which is located within the Chatkal Valley, 38 km southwest of the proposed mine site at the confluence of the Sandalash and Chatkal rivers, and the other within the dry valley.The climatic characteristics of the area have been derived from the Chatkal weather station (1,937 masl), which is located in the middle part of Chatkal valley, on the left-bank of the river (latitude 41º 0' 54" / longitude 71º 0' 19") and are summarised below:

TABLE 20-1 CLIMATIC DATA FOR THE CHAARAT PROJECT

Item Description	Unit	Detail
Climate description	-	Typical inland climate
Annual average precipitation	mm	553 to 1,000
Annual average snow fall	mm	500 to 600
Minimum monthly average precipitation	mm	9
Maximum monthly average precipitation	mm	69
Rainy season	-	March to May
Snowy season	-	October to February
1:100 year return, 24-hour precipitation event[1]	mm	4.1
1:100 year return, 3-hour precipitation event[1]	mm	11.6
Annual average temperature	˚C	2.8
Coldest month	-	January
Hottest month		July
Minimum design temperature	˚C	-35
Maximum design temperature	˚C	38
Days below 0 degrees per year	-	200
Annual average humidity	%	61
Monthly average maximum humidity	%	75
Minimum average monthly humidity	%	40

Item Description	Unit	Detail
Average wind speed	m/s	2
Typical wind direction	-	Southeast
Design wind speed (VR)		Zone III
Snow loads		Zone II

Source: Chaarat, ESIA, 2020

Note: [1] The 1 in 100 year rainfall record is appropriate to use to represent a design flood risk event. However, dependent on engineering requirements and catchment definition i.e., for hydraulics and channel/pond sizing, a more intense event of less duration than 24 hours should be used i.e., the design 3-hour storm.

The nominal depth of seasonal soil freezing in the horizontal areas is as follows: sandy loam - 158 cm, fine and silty sands - 193 cm; gravel sand, large and medium grain size - 207 cm and macro-fragmental soil - 234 cm.

The climate of the district is typical of mountainous regions of Kyrgyzstan. It is a 'continental' climate with hot summers and severe winters and significant day to night temperature fluctuations.

WAI analysed Meteorological data from the Chatkal weather station in 2017. Monthly precipitation is shown in Table 20-2 Summary of Adjusted Daily Precipitation Record for the Mine Site and the calculated periods of rainfall recurrence in Table 20-3. Most precipitation between November and March occurs as snow.

Table 20-2 Summary of Adjusted Daily Precipitation Record for the Mine Site

Record	Monthly Averages for Period (mm)												
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Mine Site[1] (1933 to 2015)	8	21	122	124	55	40	24	14	17	51	48	24	548
Raw Summer Camp[2] (2010 to 2017)	7	22	57	78	58	49	27	22	30	71	66	32	556
Raw Jany-Bazar[2] (1993 to 2015)	41	42	69	52	36	30	18	9	10	32	50	51	440

Source: Chaarat, ESIA, 2020

Notes: [1] Sequence adjusted to reflect correct timing of snowmelt and the spatial variation between Jany-Bazar and the mine site.

[2] Daily record without any adjustment for timing of snowmelt.

Table 20-3 Estimated Periods of Rainfall Recurrence

ARI (years)	PMAP (%)	Precipitation Intensity (mm/h)								
		15 min	30 min	45 min	60 min	2 h	3 h	24 h	48 h	72 h
5	20	27.5	19.4	15.6	12.9	8.6	7.2	2.4	1.8	1.4
10	10	32.2	22.6	18.2	15.0	10.4	8.5	2.8	2.1	1.7
20	5	36.5	25.7	20.7	17.1	11.9	9.6	3.2	2.4	2.0
50	2	42.2	29.7	23.9	19.7	13.7	11.1	3.7	2.8	2.3
100	1	46.4	32.6	26.3	21.7	15.1	12.2	4.1	3.1	2.6
200	0.5	50.7	35.6	28.7	23.7	16.4	13.3	4.4	3.3	2.9
500	0.2	56.3	39.6	31.9	26.4	18.3	14.8	4.9	3.7	3.2
PMAP	0.001	82.0	62.0	50.0	41.0	28.0	23.0	8.0	6.5	5.5

Source: Chaarat, ESIA, 2020

Notes: ARI- average repetition interval; APE - annual probability of excess; PMAP - the probable maximum amount of precipitation (equal to ARI 1 per 100,000 years).

The climate of the region favours the formation of frozen soils. Sub-zero temperatures are present at the depth of 2.4 m on the northern slopes all year round.

20.1.4. GEOLOGY

The Tulkubash Zone is hosted in quartzites. The Chaarat Property mineralisation shows persistent silicification and pyritization (quartz-pyrite) with local argillic and sericitic alteration, and is characterised by the presence of arsenopyrite, stibnite and tetrahedrite. Gold and silver have some correlation with arsenic, mostly in the form of arsenopyrite.

Five main rock types have been identified within the Tulkubash zone that will be excavated during open pit mining. The bulk of the target gold mineralisation occurs in the tectonic breccia, which makes up less than 8% of the pit volume. Tulkubash sandstone will make up 67% to 75% of the pit rock, constituting most of the waste rock, with Chaarat Sandstone, diorite, and travertine accounting for 6% to 8% each.

20.1.5. GEOCHEMISTRY

The Chaarat Property environmental setting has influence on the geochemical outcomes of mining the Project area. Climate data shows that there is sufficient seasonal rainfall and snow melt water to recharge groundwater, to wash open pit walls and accumulate in pit sumps, and to generate run-off and seepage drainage from the WRD and low-grade stockpile. However, as much of the precipitation falls as snow, water is frozen for approximately five months of the year, and annual evaporation is greater than precipitation. ARD risk is reduced given the need for water/rock interaction. Groundwater recharge will come from infiltration of precipitation and levels respond sharply in spring, with discharge to the river and valley alluvials, as well as seasonal seepage into the open pits.

Results of sampling at 17 surface monitoring points across the Property from 2010 to 2013, show above neutral pH, many exceeding pH 8. However, neutral water samples taken downstream from mining activities still showed higher electrical conductivity and ion concentrations, with elevated metal readings, particularly aluminium, arsenic, cadmium, and antimony, from two or three monitoring points associated with existing mining. Groundwater sampling shows arsenic and antimony is present both suspended and dissolved in the deeper aquifer in contact with the ore zone. Samples from the Sandalash River, springs and Adit 2 are dominated by cadmium, magnesium, and bicarbonate (HCO_3) indicating chemical alteration of meteoric water reacting with carbonate minerals, while higher sulphate from Adit 4 samples suggest some oxidation/dissolution of sulphides in the ore zone. All underground and borehole samples are circum-neutral, although slightly lower pH from deeper underground water indicates longer contact with mineralisation in the ore zone may increase the risk of acid generation and metal leaching.

There have been a number of different geochemical characterisation, metallurgical, and leaching test work studies undertaken on the Chaarat lithologies, although on a limited number and distribution of samples, and have given often contradictory results. Analyses of samples for lithology characterisation and of ore material used for testing, gave sulphur content ranging from 0.00% to 2.94% sulphur, and carbonates 1% to 15% (calcite and dolomite). The differences suggest a high degree of variability within, and between, deposit rock types. Most samples have arsenic and antimony values above average crustal abundances, and many with significant manganese, thallium, zinc, lead, cobalt, copper, and molybdenum.

SRK undertook geochemical studies of Chaarat waste rock in 2010 (SRK 2010). The screening level study covered the Main and Contact zones of the Chaarat deposit but did not include the Tulkubash Zone. Twelve samples composited from five lithologies in the Main zone and seven from the Contact zone were used for a test schedule that included acid-base accounting (ABA) and mineralogical and chemical characterisation. The study concluded that the main rock types from the Contact and Main zones contain concentrations of sulphides which when oxidised are likely to generate acidity; however, contact leaching resulted in neutral pH and low salinity suggesting acid production had not occurred. All samples had a sulphide sulphur content above 0.50%, up to 2.94%, and calculations assumed that all sulphur was in the form of pyrite, so acid producing potential (AP) may have been over-estimated.

In 2012, AMEC carried out a review of the waste characterisation programme and cross-referenced Tulkubash deposit rock types with the previous tests on the Kyzyltash samples (AMEC 2012). AMEC concluded that waste rock from the Project was likely to be PAG, with metal leaching of arsenic and antimony.

MINTEK (2011) conducted geochemical studies on Tulkubash zone cyanidation test work tailings and concluded low to negligible potential for ARD, with high calcite and dolomite likely to neutralise any acid production.

Inconclusive results and data gaps identified from these studies, together with Tulkubash field data, were used to develop a geochemical programme for the WAI ESIA. Samples of each of the five main lithologies from the Tulkubash zone, were used for x-ray diffraction (XRD) analysis, paste pH, ABA; net acid generation tests; and humidity cell testing (HCT) over 20 weeks. However, only one composite sample for each rock type was provided and no information was available on sample locations.

XRD results showed significant pyrite content in the tectonic breccia and some in Chaarat sandstone, while the travertine had high calcite and aragonite carbonates. Diorite also contains calcite and Chaarat sandstone carbonate is in the form of dolomite. The ABA results suggested that only the tectonic breccia is likely to generate acid, but net acid generation tests indicate that none of the samples generate sufficient acid to quantify, although the final net acid generation pH for tectonic breccia and Tulkubash Sandstone samples were below pH 5.

Kinetic HCT leachate from all samples gave fairly constant pH results throughout the 20-week testing, with both sandstones and diorite samples neutral, travertine around pH 8, and the tectonic breccia at pH 6.5. Of the 31 elements analysed for, the most notable in HCT leachates were aluminium, antimony, arsenic, cadmium, manganese, nickel, strontium, and zinc, with lesser lead, magnesium, calcium, barium, and boron. All other elements were either very low or below detection limits. In almost all cases, the highest values were from the first flush, decreasing dramatically in subsequent weeks. Exceptions were for arsenic and antimony from the tectonic breccia sample, at or above WHO standard limits. HCT results suggest that acid generation from these samples is not significant, but that mobilisation of arsenic and antimony can be expected, particularly from the tectonic breccia, even in neutral pH.

To supplement the geochemical laboratory testing, field barrel tests were carried out on the five main rock types. The test barrels were exposed to the local climate and sampled monthly when there was sufficient liquid for analysis. As with the HCT tests, the tectonic breccia sample had the highest results in terms of metal leaching, but significantly, the leachate pH was much lower at 2.5 in the first flush leach extraction, only increasing to pH 3.5 five months later. The concentrations of solutes released during the barrel tests provide good approximations of the geochemical process and leaching of waste rock materials under field conditions over an extended period. Results from the test barrels verify the predictions from the other test work that the tectonic breccia is likely to be the only high-risk rock type.

Tectonic breccia makes up less than 8% of the pit volume, but much of it will be mineralised and taken as ore, where the sulphur percentage is low enough to be amenable to cyanide heap leaching. As this rock type is identified as the most likely to be PAG, and has the highest leachable metal content, the breccia characteristics were used in the mine block model to define the likely PAG material in the pits. Based on criteria of weak oxidation - greater than 40% silicification; high alteration; greater than or equal to 1% sulphur; and inside the greater than 0.25 g/t gold shell, which largely describes the tectonic breccia rock type - the block model has predicted only 51,000 t of gold-bearing PAG tectonic breccia material within the pits. If the gold shell criteria are removed from the model, the amount of PAG tectonic breccia increases to approximately 208,000 t. This is still less than 0.33% of the total waste rock to be mined. Confidence levels for these criteria are premised on 22% of all core samples being analysed for sulphur, including almost all samples with greater than 0.25 g/t gold.

Of the 600 kt of pre-production mined material, 410 kt of ore will be stockpiled until the HLF is constructed, of which approximately 140 kt will be required for overliner for the heap leach pad. According to the latest mining plan, no low-grade ore is expected to be stockpiled. Strategic placement of identified higher risk ore material within the stockpile may be required the stockpile storage capacity is 530,000 t. The high mining rate of this Project has the positive effect of quickly covering any PAG material, whether waste or low-grade ore, reducing the opportunity for oxidation of sulphide minerals.

Following recommendations given in the 2017 WAI Framework Geochemistry Management Plan for more comprehensive sampling and testing, a new geochemical ARD study plan was designed and instigated in June 2020. Sample selection was based on testing all rock types to be encountered in the proposed mine plan - the five units identified previously, Quartzite (Tulkubash Sandstone), Tectonic Breccia, Traventine, Chaarat Sandstone and Diorite, together with ore material. The selection program also further defined the predominant Quartzite into HW, FW and internal waste material.

The NAG test results showed that all but one sample had final NAG pH above 4.5, that is, are theoretically non-acid producing (NAP), where the acid generated through oxidation of sulphides is countered by the neutralising minerals in the sample. The single sample below the NAG pH 4.5 was a waste Quartzite material, but at pH 4.48 is only just below the standard NAP definition.

While 33% of the tested ore samples fit into the ABA PAG or uncertain category; less than 18% of waste samples are so categorised; and almost all of these are marginal with very low net acid producing potential. The combined results from this recent and comprehensive sample testing suggests that acid generation is likely to be limited and will largely be neutralised by the contained carbonates. However, it is also noted that PAG material can occur in any of the rock types.

Geochemical impact assessment has identified the following potential risks from the project:

- Pit: ARD-ML from exposure of PAG material in the pit walls. Orebody definition is still being refined and modelling will determine where and how the different rock types will appear in the pit shell walls and floor at various times during excavation for ore production;
- WRD: Aside from the identified PAG material, some samples of the sandstone rocks have shown high pyrite, and arsenic, antimony, and other metals have been identified in neutral leachate from different waste rock material. All of the field evidence and most testing results indicate that the bulk of the waste rock is NAG, and some may provide neutralising capacity, but metal leaching is a risk that should be further investigated and used to guide WRD management;
- Stockpiles: Ore stockpiles, by definition, will be mineralised and likely to have higher metal content. While Tulkubash rock taken as HLF ore is unlikely to have significant sulphidic mineralisation, gold association with arsenopyrite and relative abundance of stibnite and other potentially soluble metals are of concern; and
- HLF: Sulphur analyses of ore samples have been variable and ARD test work uncertain, but as high sulphur percentage is unacceptable for cyanide heap leaching, ARD is unlikely to be an issue at the HLF. However, even after gold extraction through cyanidation, high metal contents could result in metal leaching from spent heap leach residue at the end of mine life. Planned water management at the HLF is designed to contain all seepage and run-off from this material both during production and post-closure.

20.1.6. Ecology and Biodiversity

Eleven ecology and biodiversity surveys were carried out between 2009 and 2019, principally by Davletbakov et al. The surveys were conducted in the drainage basin of the Sandalash River, along a stretch of 15 km matching the boundaries of the “area of direct influence” of the Project. Stage I involved recording plant and animal species present (both terrestrial and aquatic) and mapping the habitat types in the Project area. Eight representative "habitat-type" sampling sites were mapped for Stage II of the survey. In Stage II of the survey, the local plant populations were recorded for each habitat type assigned in Stage I.

The mountain slope elevations at the study site varied between 2,170 masl to 3,250 masl on both sides of the Sandalash River. The flora of the mountain slopes comprised mainly herbaceous and ephemerid plants with scattered shrubs and trees. The appearance of shrubs and trees and perennial herbaceous vegetation was found to be dependent on both elevation, slope, direction and positioning, and soil stability. Shrubs and trees, as well as perennial herbaceous vegetation, were found at a higher altitude on the southern ridge (north-facing slope) and at a lower elevation. Perennial vegetation shifts to ephemerid herbaceous dominance from an elevation of 2,900 masl.

There are more than 500 kinds of vertebrates in the Kyrgyz Republic, including 83 species of mammals and 388 species of birds. The fauna of the survey area includes vertebrates of steppe and mountain ecosystems of western Tien Shan.

In 2018, a group of scientists from the National Academy of Sciences carried out studies on the flora and fauna of the Chaarat deposit and adjacent areas. Research data on the current state of flora and fauna are reflected in the Report “Baseline study of the flora and fauna of Chaarat deposit”.

According to the findings of scientists of the National Academy of Sciences of the Kyrgyz Republic reflected in the Report, the following species had been identified in the surveyed area:

- one species of amphibians and five species of reptiles; all species observed on the Project site belong to widespread species in the Kyrgyz Republic; Agkistrodon (Gloidius) halis (Pallas, 1776) - Halys viper; Elaphe dione (Pallas, 1773) - Pallas' coluber; Natrix tessellate (Laurenti, 1768) - dice snake; Eremias (s. Str.) Nikolskii Bedriaga - Kirghiz racerunner; Asymblepharus alaicus (Elpat.) - Alai snake-eyed skink;
- 87 species of birds, representatives of 11 units and 25 blood-lines were encountered: common kestrel, chukar, common sandpiper, eastern turtle dove, tree pipit, rock pipit, gray wagtail, masked wagtail, common myna, magpie, carrion crow, dipper, whitethroat, chiffchaff, stonechat, Indian redstart, blackbird, whistling thrush, rufous-naped tit, rock bunting and red-headed bunting. Himalayan whistling thrush and white-winged grosbeak are rather small in numbers;
- In the area of the Chaarat deposit and adjacent areas, vegetation is represented by the following types: Petrophylous plants represented by juniper and bush association (Juniperus pseudosabina Fisch. et C.A.Mey. + J. semiglobosa Regel + Lonicera spp.), large grass semi-savannahs, represented by prangos and gramineous association (Prangos pabularia Lindl. + Festuca valesiaca Gaudin + Poa pratensis L.) and ferulic and prangos association (Ferula tenuisecta Korov. +Prangos pabularia Lindl.), whitewood (flood-plain forests), represented by birch and willow association (Betula pendula Roth + Salix spp.), meadow steppe, represented by bluegrass-helictotrichon-mixed grass association (Poa relaxa Ovcz. + Helictotrichon hookeri (Scribn.) Henrard). and Mesophilic mountain grasslands represented by three associations: sea flower-alliaceous-knotweed association (Anemone protracta (Ulbr.) Juz. + Allium hymenorhizum Ledeb. + Polygonum coriarium Grig.), alliaceous association (Allium fedtschenkoanum Regel. + Allium hymenorhizum Ledeb.) and tubuliflorous-alliaceous association (Solenanthus karateginus Lipsky + Allium hymenorhizum Ledeb.). The following can be observed in the study area: one species listed in the “Red Book of Kyrgyzstan” (2007) - Tulipa kaufmanniana Regel - Kaufman tulip; a number of ornamental, food and medicinal plants; a number of especially valuable tree species - J. semiglobosa Regel - hemispheric juniper and Juniperus pseudosabina Fisch. et C.A. Mey. Turkestan juniper; one endemic plant species - Ferula renardii was found along the river banks and the river plains and the wetland; and
- no species of plants (algal flora), invertebrates and fish listed in Red Book of the Kyrgyz Republic were found in water bodies in Sandalash river basin; the presence of rare species such as the caddis Himalopsyhe gigantea and fish from the genus Phoxinus and Cottus (are Central Asian endemics) had been identified.

According to publicly available United Nations Educational, Scientific, and Cultural Organization (UNESCO) maps, the Project site is located within Besh-Aral State Nature Reserve – Sandalash Area, one of the territories within the Western Tien Shan UNESCO World Heritage Site. Inscribed in 2016 (UNESCO ID: 1490-010), this transnational territory (Kazakhstan, Kyrgyzstan, Uzbekistan) is deemed by UNESCO to be of, "global importance as a centre of origin for a number of cultivated fruit crops and is home to a great diversity of forest types and unique plant community associations."

Based on additional information provided by the Chaarat, the publicly available UNESCO map depicts incorrect delineations for the Besh-Aral State Nature Reserve – Sandalash Area. To confirm this, Chaarat secured an official letter from the government of the Kyrgyz Republic correcting UNESCO's territorial delineation and stating that the Chaarat Property does not lie within the Besh-Aral State Nature Reserve. In this context, the nearest official nature reserve, as recognised by the Kyrgyz Republic government, lies approximately 7.5 km northwest of the Sandalash River. It is understood that the Kyrgyz government has recently submitted an official letter to UNESCO reiterating this position.

20.2. PROJECT ENVIRONMENTAL CONSIDERATIONS

20.2.1. ROCK MINE WASTES

Given the presence of sulphides in the Chaarat deposit, coupled with potentially high levels of soluble metals, the Project is exposed to the risk of acid generation and/or metal leaching. Apart from the specific ARD studies, there is currently little analytical data on the likely open pit waste rock. In theory, the bulk of the waste rock should not be mineralised.

However, 208,000 t of PAG waste rock tectonic breccia has been predicted, and some sandstone waste rock may have high pyrite content. While most of the leachable metals are associated with the ore that will report to the HLF, arsenic, antimony, and various other metals have been identified in neutral leachate from other Chaarat rock materials.

Proposed WRD ARD-ML management is to intermix the PAG tectonic breccia material with over 63 Mt of NAG waste rock. It is assumed that PAG material will be encapsulated in the waste dump by neutralising material. A settlement pond at the base of the WRD is however planned and will allow runoff to be monitored prior to discharge.

20.2.2. WATER MANAGEMENT AND EFFLUENTS

The Project's raw water requirement for the camp and the process area is met by boreholes adjacent the Sandalash and Kumbeltash water courses.

20.2.2.1. INFRASTRUCTURE FOR RAW WATER SUPPLY

Two pump stations will supply raw water to the site:

- Boreholes located to the west of the 360 Man Camp will supply the camp and the Mine Contractor's Mine Maintenance Shop with raw water all year round; and
- Boreholes located adjacent to the Kumbeltash Stream to the east of the ADR platform will deliver water to a tank at the ADR plant, which will be used to supply the ADR and Crusher platforms as well the administration building,

laboratory and gate house. The analysis of the two boreholes showed that 4.23 m^3/hr was confirmed.

Raw water will be stored in insulated raw water tanks at the camp and the ADR plant. These will also be used for fire water.

20.2.2.2. Process Water

The main water requirement for the Project will be for leaching of the ore after stacking the crushed ore on to the heap leach pad and to replace solution which evaporates during the leaching process. This water will be used to ensure that there is sufficient water for irrigating the heap.

Water will be pumped from the Kumbeltash boreholes to a raw water tank at the ADR plant. The location of the bores will ensure a water supply all year round. Duty and stand-by pumps will be used to pump the raw water to users.

Raw water will mainly be used for HLF make-up water. After engineering calculations it showed that two water wells nearby the plant has 1,980 m^3/hr to 2,590 m^3/hr (depending on season) water capacity which is more than enough for the process plant as indicated in water balance report (1,850 m^3/day).

20.2.2.3. Heap Leach Water Circulation

The barren solution will be pumped from the barren solution tank in the ADR building to the heap, where the solution will flow through a series of irrigation pipes. These pipes will have drip emitters, which will let the solution flow out at a controlled rate into the heap. The solution will flow through the heap, down to the pad liner, and then to the pregnant solution pond. The solution will then be pumped to the CIC circuit. Once the solution has passed through the CIC circuit it will finish up in the barren solution tank. There will be no discharge of solution from the heap leach pad or ADR.

20.2.2.4. Potable Water Supply

Potable water will be produced from water taken from the raw water tank at the camp site. The raw water will be treated by filtration and UV treatment or chlorination before being transferred to a potable water tank at the camp. This water will be pumped to the camp buildings.

As Chaarat will construct 360 Man shift camp; total water consumption is calculated 85 m^3/day based on the number taken as 220 ℓ/day average water consumption per person according to the local norms. Therefore, it is concluded that two water wells located nearby the Sandalash River have more than enough water for the whole camp.

The ADR plant will also have a similar setup and this potable water will supply the ADR plant, administration building and the laboratory. Potable water will be taken to other locations, for example, the mine buildings, by a dedicated water tanker. Insulated holding tanks will be used at these locations.

20.2.2.5. Drainage and Sewage Treatment

Site drainage will be designed and implemented for different locations and will consist of diversion drains, discharging uncontaminated water to natural streams. Run-off from active mine facilities, such as WRDs, stockpiles, and open pits, will be collected through internal ditches and collection sumps the sump capacity next to the stockpile is 4,500 m^3. Where practical, water will be recycled to meet demands for dust suppression and process plant requirements. Due to the dispersal of mine infrastructure along both sides of the Sandalash Valley, any excess water once complaint with applicable surface water quality standards, will be discharged to the downstream environment at a number of separate locations to. The HLF has been designed with catchment diversion drains and containment ponds to provide a closed system with no outflows to the environment, with the exception of evaporation.

A standard treatment plant will be installed for sewage disposal in the camp area, see Section 18.4.1.5. Wastewater will be treated to a standard that is acceptable under Kyrgyz regulations to allow it to be disposed of into the environment. Detailed design of the method of disposal of effluent (treated wastewater) will be in accordance with Kyrgyz Republic regulations.

Each area will have its own septic tank. These tanks will be emptied periodically by the site vacuum truck ("honey sucker") and delivered to the camp sewage treatment plant for purification.

20.2.3. EMISSIONS TO AIR

Emissions to air, including Greenhouse Gas (GHG) emissions, are based on the finalised project design. Main emissions are expected to be dust from haulage vehicles operating on unpaved roads and combustion emissions from site operations and haulage vehicles. The intensity and duration of air emissions will also be dependent on local weather conditions. Combustion engines are expected to be the main emitters of GHGs from the Project, including through emissions of carbon dioxide (CO_2), methane (CH_4) and nitrous oxide (N_2O).

20.2.4. NOISE

Noise impacts on local communities associated with peak operations of the Project are predicted to be significantly below noise levels recommended by the IFC, WHO, and in Kyrgyz Republic guidelines. No specific noise mitigation beyond standard mitigation measures and best practices will be adopted to protect receptors in the settlements of Chatkal Valley.

The workers accommodation is a potentially sensitive noise receptor; however, the haul road will be designed and constructed to mitigate noise related issues at the camp and associated facilities.

20.2.5. NON-MINING WASTE

Non-mining waste that will be produced on the Project site includes:

- Domestic waste;
- Clinic waste from the first-aid station;
- Spent oil and grease from the process area, generators, and maintenance facilities;
- Waste oils from oil/water separators;
- Materials contaminated with oil and grease, such as cleaning materials and oil filters, which are combustible;
- Reagent packaging; and
- Metal scrap, scrap tyres, and batteries.

Non-mining waste will be collected, segregated, and transported to the waste management facility for appropriate re-use, storage, or disposal, as applicable. Wastes will be segregated so that:

- Opportunities for recovery of salvageable wastes are maximised;
- Non-hazardous wastes are not mixed with hazardous wastes; and
- The wastes can be readily transferred to the correct sections of the waste management facility.

Chaarat has waste management plan in place for the Project. Wastes will be characterised, handled, and disposed of in accordance with the waste management plan.

20.2.6. Hazardous Materials Storage and Handling

All chemicals will be stored and handled in accordance with specifications on their Material Safety Data Sheets (MSDS). They will be stored such that there is no potential for mixing of spilled materials that may cause reactions. The storage areas will include full spill containment to prevent contamination of any sub-surface water and minimise closure costs from having to treat contaminated soils. Waste oil will be collected from the site for off-site disposal at a suitable facility. Hazardous materials and waste infrastructure on site include:

- Incinerators;
- A landfill area;
- Laydown areas;
- Additional storage area;
- Open burning area contained by concrete walls on three sides; and
- A hazardous waste storage unit.

Hazardous waste will be stored in the waste management facility in suitable closed containers that prevent accidental releases to soil, air, or water. Appropriate secondary containment will be provided for liquid wastes stored in volumes greater than 220 ℓ. The hazardous waste storage unit in the waste management facility will be developed to store waste with enough capacity for the Project LoM.

Clinical waste from the first-aid station will be incinerated in a clinic incinerator, which is located at the clinic and is dedicated to incineration of medical waste.

Reagent containers will be rinsed, if appropriate or necessary, and then compacted prior to landfilling. Sodium cyanide boxes will be burned (the plastic liners will be rinsed and buried in landfill and the boxes will be burned).

20.2.6.1. Storage and Handling of Reagents

Process reagents will include sodium cyanide, lime, nitric acid, sodium hydroxide, and anti-scalant. Hydrogen peroxide will be used for cyanide neutralisation in an emergency. All reagents will be delivered to site in meeting international standards containers and stored in secure dedicated storage areas. Cyanide consumption will be 0.60 kg/t of ore, or approximately 3,000 t/a. In the process area, containers will be stored in an enclosed, gated, and security controlled bunded area. When required, the container will be moved to the ADR plant building and the bags will be offloaded by forklift to the cyanide reagent mixing area.

Lubricants, engine coolant, and other fuels will be kept at the site. These materials will normally be stored in barrels or metal or plastic jerrycans in areas providing secondary containment.

Small quantities of paints, solvents, cleaners, insecticides, rodenticides, and herbicides will be retained on site to support routine site maintenance and will be transported to the site by truck in steel intermodal containers. These items will be kept in controlled warehouse locations.

20.2.6.2. Fuel Farm

Fuel storage tanks will be bunded to contain any spillages. Any spills and/or stormwater captured within containment will pass through oil separators prior to disposal. Fuel storage tanks will be located immediately adjacent to the power plant to supply the generators and will have sufficient capacity to hold a strategic and operational reserve (approximately 14 days). There will be a fuelling station adjacent to the fuel storage tanks. Additionally, there will be a fuel day tank at the ADR plant for processing and heating requirements, as well as at the camp and the vehicle workshop.

A fuelling station will be required for both light vehicles and mining fleet vehicles. The fuelling station will be used to refill vehicles and refuelling tankers, which will refuel the mining fleet at the mine. The refuelling tanker will also supply the localised fuel tanks.

Diesel fuel will be used during construction for power generation, construction equipment and machinery, and vehicles. Diesel fuel consumption during operations will be used to fuel the mining equipment and process operations.

The primary sources of fuel consumption at the mine will be the power generators, designed to meet most of the mine’s energy requirements, and the mine production vehicles and mobile equipment.

20.2.7. General Housekeeping

WAI performed a site visit and reported that the project site and staff facilities were generally tidy, and well maintained and managed.

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20.3. Permitting and Documentation

20.3.1. ESIA/OVOS and other Environmental and Social Reviews

In 2015, Ken-Too developed an Environmental Impact Assessment (EIA or OVOS) to local standards. Academy of Sciences of KR (Davletbekov) carried out an environmental review (technical assessment of the environmental baseline), presenting results of a field survey of soil, flora, and fauna developed in and around the Project area. A social review of local communities was carried out by Leshem Scheffer in 2011.

A final updated ESIA according to international standards was completed in October 2020 by Wardell Armstrong International.

20.3.2. Environmental Permits and Licences

Chaarat received its initial exploration licence (Au-174-02) on 10 December 2002. The exploration area was later enlarged to cover the Chaarat, Kashkasu, and Minteke prospects. The current exploration licence (3319AP) was issued on 07 October 2013 and is valid until 21 April 2023 and covers 6,776 ha. A mining licence (3117AE) was issued on 22 January 2014 covering the core 700.03 ha of the Chaarat Property. This licence is valid until 25 June 2032.

The Project will require several state-approved licences and other authorisations, these are summarised in the Table 20-4.

Table 20-4 Permit/Licence Status

Licence or Permit	Authority	Project Status
Environmental Impact Assessment); and - Subsoil use protection.		
Mining Licence №3117 AE for the development of the gold deposit	State Committee on Energy, Industry and Subsoil Use	Licence Agreement № 4 to Mining Licence 3117AE was signed on 07 September, 2017 and is valid till 2032.
Licence for gravel/rock borrow source	State Committee on Energy, Industry and Subsoil Use	Obtained. Valid till 27 March 2025.
Land permits (certificates for temporary use) i.e., land allocation	Local administration bodies and Regional Department of State Registration Office	Up to 90 % of lands obtained (main areas) - all valid through 2032 Tulkubash and Kyzyltash - 899 ha – 16 March, 2016 Dry Valley - 384,586 ha – 14 June, 2013 Access Road - 68 ha – 5 April, 2017 Bridge Area - 32 ha – 11 November, 2017
Architectural and Planning specifications and Engineering and Planning specifications	Regional Department on Architecture and Construction (in Chatkal District)	14 April 2020 for Check Point Platform Construction
Designing/Legalization of design documentation in case if designed by non-local organization	International/Kyrgyz designing companies having proper licences and permits	Designing process started (Design adaptation will happen in parallel with design process).
Industrial safety expertise	GosGorTechNadzor (branch monitoring the industrial safety) under the State Committee on Energy, Industry and Subsoil Use	All designs shall pass industrial safety expertise. To be completed after designs are finalized prior to legalisation.
Ecological safety expertise	State Agency of Environmental Protection and Forestry under the KR Government	All designs shall pass ecological safety expertise. To be completed after designs are finalized prior to legalisation.
State Construction Expertise of all design documentation	Department of the State Expertise of the State Agency for Architecture and Construction under the Government of	Expertise will start after submission of completed/legalised detailed design documentation.

Licence or Permit	Authority	Project Status
	the Kyrgyz Republic and its regional branch	
Permits for the construction of facilities	Regional Department on Architecture and Construction (in Chatkal District)	Immediately after all above expertise approvals obtained.
Commissioning of constructed facilities	The act is signed by the Commission including applicant, the author of the design, the general contractor and the authorized officials of the territorial body of state architectural and construction that carried out the supervision.	After completion of construction.
Permit for mining works	State Committee on Energy, Industry and Subsoil Use	Will be obtained within 7-10 days after submitting of completed mine design drawings.
Licence for water use from underground sources	State Committee on Energy, Industry and Subsoil Use	Will be obtained after design of water wells completed.
Permit to release of pollutants into the air	Regional Department of Environmental Protection and Forestry of the State Agency of Environmental Protection and Forestry.	During the construction period, the permit is issued on the base of the OVOS (EIA). Within the period of operation - according to the norms of maximum allowable discharges. Chaarat will obtain this permit prior to start of operations. Contractors will obtain this permit after Chaarat receives construction permit.
Licence for carrying out activities for the utilization, storage, disposal, destruction of toxic waste materials and substances	State Agency of Environmental Protection and Forestry under the KR Government	Chaarat will obtain this permit prior to start of operations. Contractors will obtain this permit after Chaarat receives construction permit.
Permits for the transportation of hazardous goods	Ministry of Internal Affairs	Transportation Company will have this permit.
Permit to purchase explosive materials	Ministry of Internal Affairs	Mining/Earthworks Contractor will have this permit.
Permit for the storage of explosive materials	Ministry of Internal Affairs	Mining/Earthworks Contractor will have this permit.
Permit for blasting	State Committee on Energy, Industry and Subsoil Use	Mining/Earthworks Contractor will have this permit.
Permit for the waste disposal	State Agency of Environmental Protection and Forestry under the KR Government	Will be obtained together with construction permit.

Source: Chaarat, 2021

20.4. ENVIRONMENTAL MANAGEMENT

20.4.1. ENVIRONMENTAL POLICY AND COMPANY APPROACH

Chaarat has a range of environmental, social, and human resource policies designed to guide management of the Project. Several systems and work operational procedures are currently in place, relating to the environmental, social aspects, security, and health and safety. These include:

- Health and Safety Policy dated 11 May 2019;
- Regulation on industrial safety, labour protection and ecology dated 04 March 2019;
- Community Development Policy 15 September 2020;
- Donations Policy, under development;
- Procedure for Formation of a Community consultation group, 15 September 2020;
- Local Purchasing Procedure dated 19 May 2020;
- Regulation on shift work dated 31 May 2019;
- Training Policy dated 17 June 2020;
- Human Recourses Policy 17 June 2020;
- Contracting procedure dated 26 January 2020;
- Procedure on Formation of a community consultation group dated 14 June 2018;
- Grievance procedure dated 04 May 2018;
- Local Recruitment Procedure Chatkal district, Jalal-abad oblast dated 14 September 2020;
- Probation period procedure dated 01 November 2018;
- Internal Company Regulations dated 22 October 2018;
- Recruitment Procedure dated 25 January 2019;
- Legislation process for Tulkubash design packages dated 15 July 2019;
- Drilling and Blasting procedure under development;
- Supplier/Contractor Post Performance evaluation procedure dated 06 January 2020;
- Warehouse and Inventory management procedure dated 06 April 2020;
- Security Procedure dated 08 May 2018;
- Workplace Alcohol & Drugs Policy dated 11 May 2018;
- Regulation on Provision of PPE to Employees dated 30 May 2018;
- Regulation on Vehicle–Pedestrian Interaction dated 13 September 2018;
- Commercial confidentiality procedure dated 05 June 2020.
- Project Approvals and Payment for Completed Designs, dated 07 August 2012;
- Chaarat Commissioning Procedure, dated 27 July 2012;
- General Procedure, effective 01 February 2014;
- Chaarat Environmental Policy effective 09 July 2018;
- Chaarat Health and Safety Policy effective 11 May 2018;
- Chaarat Anti-Drug and Anti-Alcohol Policy effective 11 May 2018;

- Relations with Suppliers Policy, dated 01 March 2012;
- Provision on Tender Procedure, dated 17 February 2014;
- Purchasing Procedure, dated 15 January 2014;
- Site Materials Registration and Write-off Procedure, effective 01 March 2014;
- Receipt and Issue of Inventory from the Company's Warehouses, effective 01 July 2014;
- Procedure on Fuel Consumption Recording OM/S/22, undated;
- Structure of Security Department, dated 04 December 2015 (Russian);
- Standard Operational Procedures (SOP) for Head of Security Department, dated 2016 (Russian);
- On the Organisation of Support and Maintenance, Security of Individuals, dated 2016 (Russian);
- Anti-corruption and Bribery Policy, dated 01 January 2012;
- Instructions on the Organisation and Access Control at Sites, dated 2016 (Russian);
- On Security at 'Office 2' facility in Bishkek, dated 2016 (Russian);
- On the Organisation of Check-points at Accommodation Facilities and Routes for Workers and Vehicles, dated 2016 (Russian);
- Chaarat deposit closure and land reclamation project, 2018. It had been developed and adopted by the relevant state agencies;
- Cultural and historical heritage objects protection plan at Chaarat deposit and adjacent areas 2018, had been developed and adopted by the Ministry of Culture of the Kyrgyz republic;
- Guidelines #1. Top soil determination and protection process management. 2019; and
- Guidelines #2. Waste collection, accounting, temporary storage and disposal management. 2019.

A number of management plans are currently in development, or are being considered by Chaarat to conform to international best practice, including:

- Air quality management plan;
- Biodiversity management plan;
- Health and safety (human resources) management plan;
- Emergency preparedness and response plan;
- Mine closure and rehabilitation plan (it had been developed and included in the above section);
- Noise and vibration management plan;
- Soils management plan (it had been developed and included in the above section. Guidelines #1);
- Updated stakeholder engagement plan;

- Traffic and Transportation Management Plan;
- Waste and wastewater management plan (waste management plan developed, included in guidelines #2. Waste water management plan still needs to be developed); and
- Chance finds procedure. (developed and included in the section above).

These management plans will be implemented as part of the Project's broader ESMS, which will formalise best practice approaches to environmental and social aspects.

Chaarat currently has the following plans:

- Stakeholder Engagement Plan 2020;
- Community Development Management Plan 21 September 2020
- Emergency Response Plan 2019 dated 22 April 2019; and
- COVID-19 Emergency Response Plan 2020.

20.4.2. Emergency Preparedness and Response

Emergency preparedness and response management systems are currently being developed:

- SOPs for contractor health and safety responsibilities, general traffic rules, operational practises; and
- Instructions on Security and Maintenance: Explosive Materials During Transportation, dated 2016 (Russian).

Chaarat has Occupational Health, Safety and Environmental department with HSE Manager, safety engineers, doctors, environmental specialists, avalanche management specialists and emergency rescue team.

Chaarat is in process of updating an EPRP 2020 for the Project. At this time the project construction works had started on the haul roads and the earthwork. The EPRP will be updated for 2019 year and further annually as it required by KR law. The EPRP, along with a hazardous materials management plan, which is also under consideration, is also a key element within cyanide management systems.

20.4.3. Training

Training activities for employees and visitors should be adequately monitored and documented (curriculum, duration, and participants). Emergency exercises, including fire drills, should be documented adequately. Service providers and contractors should be contractually required to submit to the employer adequate training documentation before start of their assignment. HSE supplement with occupational health and safety requirements is included in the standard contract with contractors.

Existing relevant documentation include:

- Recruitment Policy, dated 25 January 2019;
- SOPs for various roles and operations on the Project site;
- Job descriptions for specialists and workers;
- Data sheet: Number of Open Pit Staff; and
- Data sheet: Chaarat Employees, 25 February 2021.

20.5. Social and Community Management

The Project's area of socioeconomic influence stretches across Jalal-Abad and Talas Regions. This influence is strongest at the eight settlements within Chatkal District, including the village districts of Kanysh-Kiya (Aigyr-Jal, Kanysh-Kiya, Korgon-Say, Bashky-Terek, Chakmak-Su, total district population in January 2021: 10,912) and Chatkal (Jany-Bazar, Kurulush, Ak-Tash, total district population in January 2021: 6,389).

All eight settlements are the focus of the ongoing socioeconomic baseline analysis. Previously, Leshem Sheffer (2011) carried out a social baseline study based on social research conducted during Q3 2010 across Chatkal District. The aim of the research was to study the current social situation and opinions of district inhabitants regarding the Project's development.

The settlements closest to other nearby mining operations, in particular Chakmak-Su, Baskhy-Terek, and Korgon-Say, have received far greater private sector support than the smaller settlements located further away from the mines, such as Aigyr-Jal, Ak-Tash, Kurulush and Jany-Bazar.

The local population is predominantly ethnic Kyrgyz and practise moderate Islam. The level of educational attainment amongst members of local communities is low, with around 50% not graduating from high school. Monthly family incomes vary by settlement and by family, but the average household income in the six villages closest to the Project is KGS7,000 to KGS 8,000. Agriculture is the main source of income for local community members.

20.5.1. Stakeholder Dialogue and Grievance Mechanisms

Chaarat's main office in the Chatkal Valley is located in Kanysh-Kiya. The offices host stakeholder meetings and maintains a grievance register.

In recent stakeholder surveys, nearly all residents interviewed across the Chatkal District reported that they are optimistic about the economic future of the area as a result of local mining developments and the direct and indirect creation of related jobs. However, they report concerns relating to the environmental impacts of the mines in the region, in particular with respect to cyanide use. Locals report that they are already actively searching for accommodation in cities, so they have somewhere to move to in the event of catastrophic environmental damage in the Chatkal Valley. These concerns are at their highest at Chakmak-Suu, where the settlement is located near the tailings dam of another mine operated by China Gold (despite the absence of cyanide at that mine).

Relevant documentation relating to stakeholder dialogue and grievance mechanisms includes:

- Stakeholder Engagement Plan
- Grievance Procedure updated in 2018; and
- Community Development Policy developed in 2020.

20.5.2. Social Initiatives and Community Development

Chaarat operates in compliance both with the legislation of the Kyrgyz Republic and best international practices, including Performance Standards of the International Finance Corporation in order to ensure environmental and social sustainability. The Law of the Kyrgyz Republic on Subsoil Use specifies the national requirements pertaining to activities to promote social and economic development, which implies the mining companies to provide social package for the local communities.

The Company fully supports the projects that meet the strategic principles of socio-economic development of Chatkal district, adopted by the district administration and local authorities. Accordingly, Chaarat agrees the Memorandum on Cooperation (social package) jointly with Chatkal district administration provides funding in the amount of approximately USD164,705 on annual basis, which includes but is not limited to the following:

- Employment of the local residents;
- Capacity building and training; and
- Infrastructure facilities improvement.

In addition, Chaarat implements a voluntary programme for the development of school education, youth sports and strengthening the professional capacity of local communities. All community development activities, including social package, are paid by Chaarat corporate as part of cost of doing business in Kyrgyz Republic.

Company supports local suppliers of goods and services, creating opportunities for the development of small and medium-sized businesses, which additionally creates jobs and contributes to the positive economic development of the region. With the initiative of Chaarat a sewing workshop for sewing workwear was launched in 2018 and twelve local women been employed on permanent basis. Also, since last year, five local woodworking shops are engaged in core boxes manufacturing for Chaarat in five villages of Chatkal district.

There are two shops in Kanysh-Kiya and Jany-Bazar villages initially sponsored by Chaarat which sell products at prices that are lower than most shops in the valley. Shops sell mostly bulk products and non-perishables such as sugar, pasta, rice, flower and tea. Cigarettes and alcohol are not sold there. Other mining operators in the area do not have such shops.

Residents report that quality of life has improved over the past five years, in particular in Bashky-Terek and Chakmak-Suu, where social infrastructure has been improved through funding from the China Gold mine. In other settlements, such as Korgon-Say, residents report feeling worse off since the last operating mining company left in 2015.

20.6. Health and Safety

20.6.1. Health and Safety Management Arrangements

All new staff will be appropriately inducted to the site to ensure occupational health and safety of workers. The personal development and training of workers will be promoted by Chaarat and existing workers will undergo, where applicable, training relevant to their job description and work area. Such training and development may include:

- Mandatory general induction and general safety;
- General skills development (computer studies, language upgrading);
- Basic industrial skills training for any workers who do not yet meet the minimum skills requirement;
- Technical trades top-up training and job-specific/plant-specific training;
- Supervisory skills and leadership techniques; and

- Risk assessment training.

Chaarat conducts regular inspections and testing of all safety features and hazard control measures, focusing on engineering and personal protective features, work procedures, places of work, installations, equipment, and tools used.

The inspection should verify that issued PPE continues to provide adequate protection and is being worn as required. All instruments installed or used for monitoring and recording of working environment parameters should be regularly tested and calibrated, and the respective records maintained. Pre-operation equipment check-up is mandatory for Chaarat and contractors.

Chaarat should consider documenting compliance using an appropriate combination of portable and stationary sampling and monitoring instruments. Monitoring and analyses should be conducted according to internationally-recognized methods and standards.

Monitoring methodology, locations, frequencies, and parameters should be established individually for each project following a review of the hazards. Generally, monitoring should be performed during commissioning of facilities or equipment and at the end of the defect and liability period, and otherwise repeated according to the occupational health and safety (OHS) monitoring plan.

When extraordinary protective measures are required (for example, against hazardous materials), workers should be provided appropriate PPE and safe working procedures in addition to relevant health surveillance prior to first exposure, and at regular intervals thereafter.

20.6.2. Performance and Accident Records

Chaarat has established procedures and systems for reporting and recording OHS incidents, including near misses. Chaarat trains and encourages employees to report immediately to their immediate supervisor any situation they believe presents a danger to life or health, all occupational injuries and near misses, suspected cases of occupational disease, and dangerous occurrences and incidents.

All reported occupational accidents, occupational diseases, dangerous occurrences, and incidents together with near misses should be investigated with the assistance of a person knowledgeable/competent in occupational safety. The investigation establishes what happened, determines the root cause and identifies measures necessary to prevent a recurrence.

Chaarat currently has an occupational health and safety management plan for the construction and operational phases of the Project.

20.7. Mine Closure and Rehabilitation

Chaarat will develop decommissioning, reclamation, and closure plans once the Project design has been fully developed. A framework mine closure and rehabilitation plan to international standards has been developed as part of the ESIA process and should be continuously updated as the project progresses.

The scope of the framework Mine Closure and Rehabilitation Plan includes:

- A process of on-going planning and development of the closure and rehabilitation of the Project, particularly;
- The planning of timelines and costs;
- Consider the expected final landform and surface/sub-surface rehabilitation, including removal of plant and equipment and stabilisation and treatment of waste rock dumps;
- Provide risk assessment to help set priorities for preparatory work;
- Analyse different options as the plan is developed;
- Detail the management of how closure will be implemented;
- Describe the availability and quantity of skilled resources for the realisation of the plan;
- Proposals for post-closure aftercare and monitoring arrangements;
- Informing stakeholders of the expectations of the Kyrgyz Republic's legal and regulatory requirements, international best practice and compliance in this regard; and
- Establishing a preliminary estimate of the closure and rehabilitation costs.

20.7.1. HLF Closure Design Criteria

At the end of the life of mine, the heap facility will need to be remediated to:

- Achieve long-term stabilization of physical, chemical and ecological conditions;
- Provide a maintenance free facility as far as possible;
- Eliminate hazards to human health and the environment, and
- Achieve sustainable land and water use.

A combination of rinsing and chemical treatment will render the heap as chemically inert, as possible. Consequentially, any precipitation or snow melt that reports to the heap and seeps through it, will not become contaminated and may discharge directly into the environment without being subject to any treatment or other measures. The heap itself will be regraded to achieve a more natural appearance and ensure long-term structural stability. Finally, the heap will be covered with topsoil and revegetated. Table 20-5 summarises the closure design criteria.

Table 20-5 Closure Design Criteria

Parameter	Unit	Design Input Value	Reference/ Notes
Minimum Rinse Time	Weeks	44	Ausenco
Final Heap Slope	H:V	3:1	Ausenco
Topsoil Minimum Thickness	mm	300	Ausenco

Source: Chaarat, 2020

20.7.2. HLF Closure and Reclamation

Reclamation will be carried out to minimise potential impacts to the surrounding environment. Preliminary recommendations for closure and reclamation are summarized below:

- Grading, covering and revegetation of final heap slopes to provide adequate drainage and erosion protection from surface run-off. This may be carried out during operations as the final slope of the heap is developed;
- Rinsing and drain-down of the ore and cyanide destruction at the end of the HLF operations;
- Removal of the PLS Overflow and Emergency ponds, as required, and
- Decommissioning of the pregnant solution recovery system.

A preliminary closure and reclamation plan has been prepared in conjunction with the appropriate regulatory authorities.

20.7.3. CONCLUSIONS

Chaarat is currently in compliance with Kyrgyz Republic legislation on environmental and social aspects. Once operations at the site are underway, further work will be required to ensure that the Project continues to comply with state legislation and international best practice frameworks.

WAI completed a preliminary ESIA Q3 2017 the latest update on the ESIA was completed in Q3 2020 and is based upon the LogiProc 2019 BFS project description.

20.8. HYDROLOGY

20.8.1. DESIGN CRITERIA

20.8.1.1. PRECIPITATION

Rainfall and snow precipitation data are shown in Table 20-6 and Table 20-5.

For water balance purposes, the data in Table 20-6 are used to define dry, average, and wet years.

TABLE 20-6 MONTHLY PRECIPITATION

Month	Dry Year (25th Percentile) (mm)	Average Year (50th Percentile) (mm)	Wet Year (75th Percentile) (mm)
January	1	4	10
February	3	9	27
March	83	120	155
April	82	102	158
May	37	56	70
June	20	39	51
July	6	18	32
August	2	11	18
September	5	13	23
October	24	43	64
November	17	39	69
December	6	16	27
Total	**286**	**470**	**704**

Source: WAI (2018)

Table 20-7 2011 Precipitation from Chaarat Summer Camp Station

Month	Precipitation (mm)
January	2.0
February	41.2
March	242.8
April	91.6
May	115.6
June	288.4
July	53.6
August	16.0
September	15.6
October	66.8
November	238.2
December	32.0
Total	**1,204.0**

Source: Chaarat data

The Project summer camp climate station has over seven years of accumulated data. Annual precipitation totals for the whole years 2011 to 2016 include a minimum of 286.0 mm (2012), close to the 25th percentile dry year, and a maximum of 1,204 mm (2011), well in excess of the 75th percentile wet year (Table 20-6). The 2011 record was used in the water balance to represent an extreme wet year in the monthly site-wide water balances.

Note that three months in 2011 realised over 200 mm of precipitation. It is assumed that the precipitation totals in March and November were snow. The highest monthly total recorded between June 2010 and September 2017 was the month of June with 288.4 mm. Despite this, the maximum daily precipitation in 2011 was 40.4 mm, only slightly above the average of 35.1 mm from 2011-2016. The highest amount in those years was 50.2 mm in 2016, a year of only 330.4 mm of precipitation in total.

For extreme events, Table 20-7 defines the depth-duration-frequency precipitation values for a range of intensities and recurrence intervals.

Table 20-8 Depth-Duration-Frequency Precipitation Values

ARI (years)	AEP (%)	Precipitation Intensity (mm/h)								
		15 min	30 min	45 min	60 min	2 h	3 h	24 h	48 h	72 h
5	20	26.4	19.4	15.3	13.1	8.6	6.9	2.4	1.8	1.4
10	10	30.8	22.7	17.9	15.3	10.1	8.0	2.8	2.1	1.7
20	5	34.9	25.7	20.3	17.4	11.5	9.1	3.2	2.4	2.0
50	2	40.3	29.7	23.5	20.1	13.2	10.6	3.7	2.8	2.3
100	**1**	44.4	32.7	25.8	22.1	14.6	11.6	**4.1**	3.1	2.6
200	0.5	48.4	35.7	28.2	24.1	15.9	12.7	4.4	3.3	2.9
500	0.2	53.9	39.7	31.4	26.8	17.7	14.1	4.9	3.7	3.2
PMP	PMP	82.0	62.0	50.0	41.0	28.0	23.0	8.0	6.5	5.5

Notes: *Includes rainfall and snowmelt.*
Probable Maximum Precipitation (PMP) is equivalent to a 1-in-100,000-year average recurrence interval (ARI).
AEP – average exceedance probability
Source: *WAI (2017)*

The design criterion of a 1-in-100 year, 24-hour event is highlighted in Table 20-8 as a precipitation intensity of 4.1 mm/h. This represents a total 1-in-100 year, 24-hour precipitation of 98.4 mm. However, for the HLF, heap leach pad, PLS, PLS Overflow and Emergency ponds are designed to contain the 1-in-200 year, 24-hour precipitation of 105.6 mm and safely pass the 2/3 between the 1-in-1,000 year, 24-hour storm event. This is consistent with international design standards (CDA 2013) for medium fluid containing structures and is used to minimise the risk of ingress and overflows to the downstream environment as far as possible. The surface water run-off management structures, i.e. diversion channels and Attenuation Pond are design to safely pass up to the 1-in-200-year storm event.

It is noted that the highest daily precipitation recorded at summer camp of 50.2 mm is less than a five-year, 24-hour event.

For clarification, ARI does not mean that the event will not happen in 99 years out of 100; rather, as indicated by AEP; there is a 1% chance of this event occurring in any year.

Table 20-9 shows the monthly average potential evaporation data as measured at the mine site from 2010 to 2015.

TABLE 20-9 MONTHLY EVAPORATION

Month	Potential Evaporation (mm)
January	12
February	4
March	21
April	75
May	153
June	184
July	242
August	214
September	141
October	51
November	18
December	13
Total	**1,128**

Source: *WAI (2018)*

Table 20-10 shows the assigned run-off factors.

TABLE 20-10 RUN-OFF FACTORS

Area	Factor
Pit Walls and Floor	1.0
Process Area	1.0
WRD	0.3
HLF	1.0

Area Surrounding HLF	0.7
Ponds	1.0

Source: WAI 2020

For design purposes it was assumed that a run-off factor of 0.7 also applies to valley slopes. The factor of 0.3 applies to the WRD itself, but water balances may assume that much of the water that infiltrates will re-emerge down the valley either via the coarse tributary alluvium or as surface flow after re-emergence.

In the water balance it is assumed that snow accumulation extends from November to March, so there is no run-off until an assumed thaw in March and April, which includes all the sum of the precipitation in those months.

The HLF catchment in the dry valley is assumed to have a very high infiltration rate because of the nature of the rubbly Quaternary infill of essentially limestone blocks from the adjacent mountain slopes. This explains the name and the likely groundwater regime. In summer, run-off coefficients could be very low for general precipitation, although for a value of 0.7 has been assigned for conservative purposes. In any event, water infiltrating the ground will reach the controlling water table below the northern part of the dry valley and migrate eventually to the Kumbeltash Stream, either flowing in the alluvium to join the Sandalash River or springing out into the stream itself.

For design flood purposes, the worst case of frozen ground and zero infiltration was assumed, resulting in a run-off factor of 1.0.

20.8.1.2. Surface Water Flows

The Sandalash River is perennial, while most tributary valley streams are ephemeral, either because of snow accumulation in the winter or low summer rainfall.

Average annual flows in the Sandalash are 17.5 m^3/s according to NK Group, LLC (NK Group) (2016). Maximum flows are stated as 109 m^3/s for a 100-year return period (1% probability); 119 m^3/s for a 50-year return period (2% probability); and 109 m^3/s for a 25-year return period (4% probability). SRK (2010) reported gauged values from 2006-2010 ranging between 6.2 and 85 m^3/s.

For the Kumbeltash Stream, NK Group (2016) reported an annual average flow of 0.091 m^3/s (327.6 m^3/hr), which presumably includes periods of no flow, and maximum flows of 8.06 m^3/s (1-in-100), 6.77 m^3/s (1-in-50), and 5.32 m^3/s (1-in-25).

20.8.2. Mine and WRD Water Management

The upstream catchments of the open pit mine and associated satellite pits will need to be diverted away from the pits, as much as possible, to avoid excess inflows that would negatively impact on efficient operations and manage the discharge contact water to the environment. The following provides a brief description of the mine water management plan.

20.8.2.1. Diversion Ditches

Figure 20-1 shows the layout of mine water management infrastructure at the end of the mine life. A diversion ditch, shown in green, is designed to intercept run-off and divert it around the waste dumps and open pits. This ditch will be constructed during pre-production and will serve the project throughout the LoM.

20.8.2.2. Roadside Ditches

Roadside ditches, shown in blue, will collect run-off from haul roads, direct precipitation, and surface water not intercepted by the diversion ditch. Water collected in roadside ditches will flow by gravity to collection sumps where it will be subsequently transferred to pipelines. Ditches may or may not be lined depending on the nature of the surface material. Roadside ditches will change throughout the mine life to conform to road layout as it develops to support mining.

20.8.2.3. In-pit Sumps

In-pit sumps will be an integral part of the mine water management system. Groundwater or run-off presenting in the open pits which is not intercepted by roadside ditches will be directed to temporary in-pit sumps. In-pit sumps will keep the immediate working areas dry to permit mining to be conducted efficiently. They will be constructed on an as-needed basis, mined out, and re-established, as circumstances dictate. Water collected in in-pit sumps will be pumped into ditches or directly to collection sumps where it will be conveyed by pipeline to the settling pond.

20.8.2.4. Pipelines

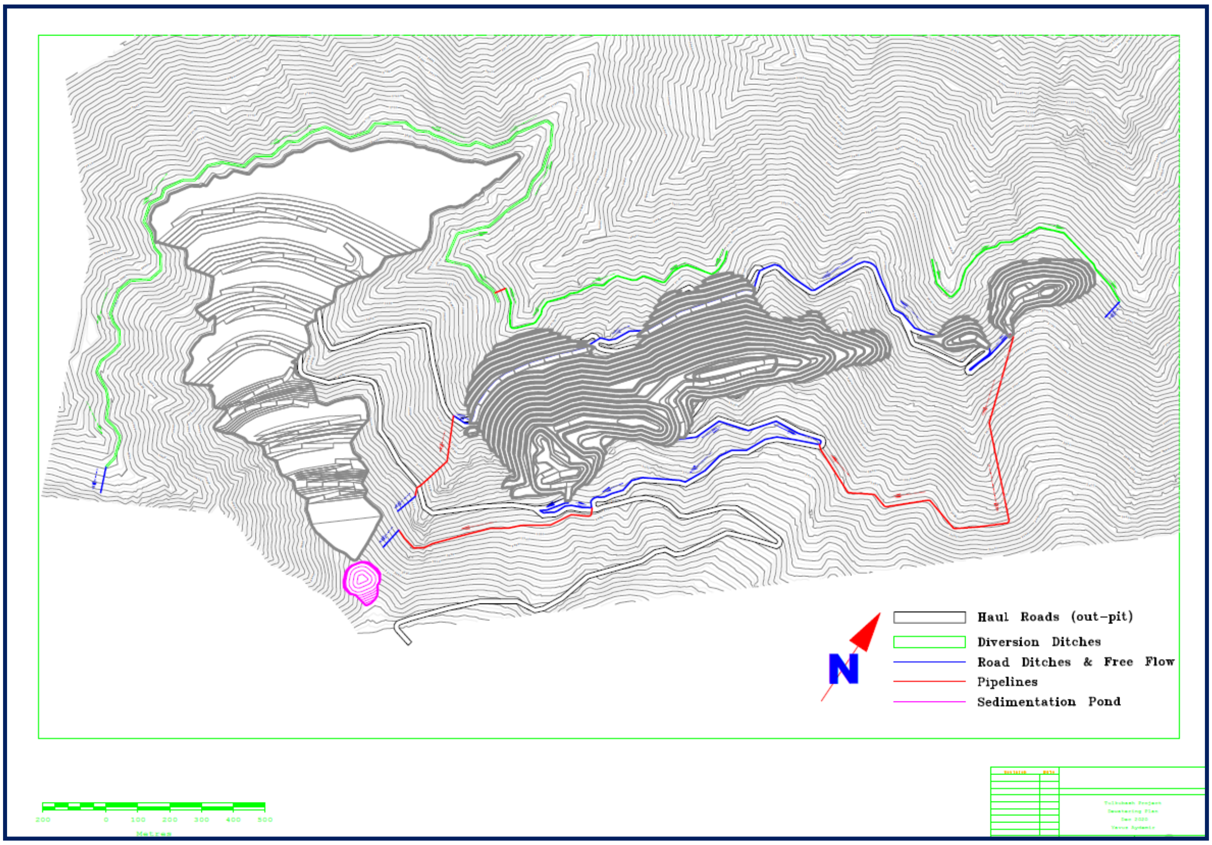
Pipelines, shown in red on Figure 20-1, will be used to transfer water from roadside ditches and in-pit sumps to the holding pond. Pipelines will be fed from collection sumps. Pipelines will make use of gravity flow wherever possible and pumping when necessary. Pipelines will be established as required by the mine development. Pipelines serving the Main Zone Pit will be established early in the mine life. Pipeline serving the Mid Zone Pits will be established towards the end of the mine life when those pits are mined.

20.8.2.5. Settling Pond

A settling pond will be constructed at the toe of the Irisai waste rock dump. Run-off from the waste dump and contact water from the mine area will be directed to the settling pond by direct flow or pipelines.

The base of the settling pond will be excavated in bedrock and may be grouted to improve containment. The downstream embankment of the pond will be a compacted, earth-fill dam. The settling pond will be designed with a spillway to allow the discharge of excess water. The capacity of the settling pond will be 25,000 m^3.

FIGURE 20-1 OPEN PIT AND WRD WATER MANAGEMENT INFRASTRUCTURE



Source: Chaarat

20.8.2.6. HLF WATER MANAGEMENT

External catchment water management applies to the water management for the HLF. Table 20-11 shows selected HLF design criteria with regard to water management.

TABLE 20-11 SELECTED HLF DESIGN CRITERIA

Sub-catchment	Unit	
Total Project Ore Capacity	Mt	25.88
Annual Ore Production	Mt	4.9
Daily Ore Production	t/d	13,500
Design Solution Flow Rate	m^3/hr	796
Nominal Flow Rate to PLS Pond	m^3/hr	643.2
Assumed Water in Ore Moisture (2.5%)	m^3/hr	337.5
Assumed Make-up Water Requirement	m^3/hr	33
HLF Footprint (for 25.88 Mt)	m^2	472,400
Design Precipitation (2/3 between the 1:1,000 and PMP)	m^3/hr	13.8

Source: Ausenco

The upstream catchments of the open pit mine and associated satellite pits will be diverted away from the pits to avoid inflows that would negatively impact mining operations, and to minimise the generation of contact water within the open pit.

For the HLF, surface water management measures include the following:

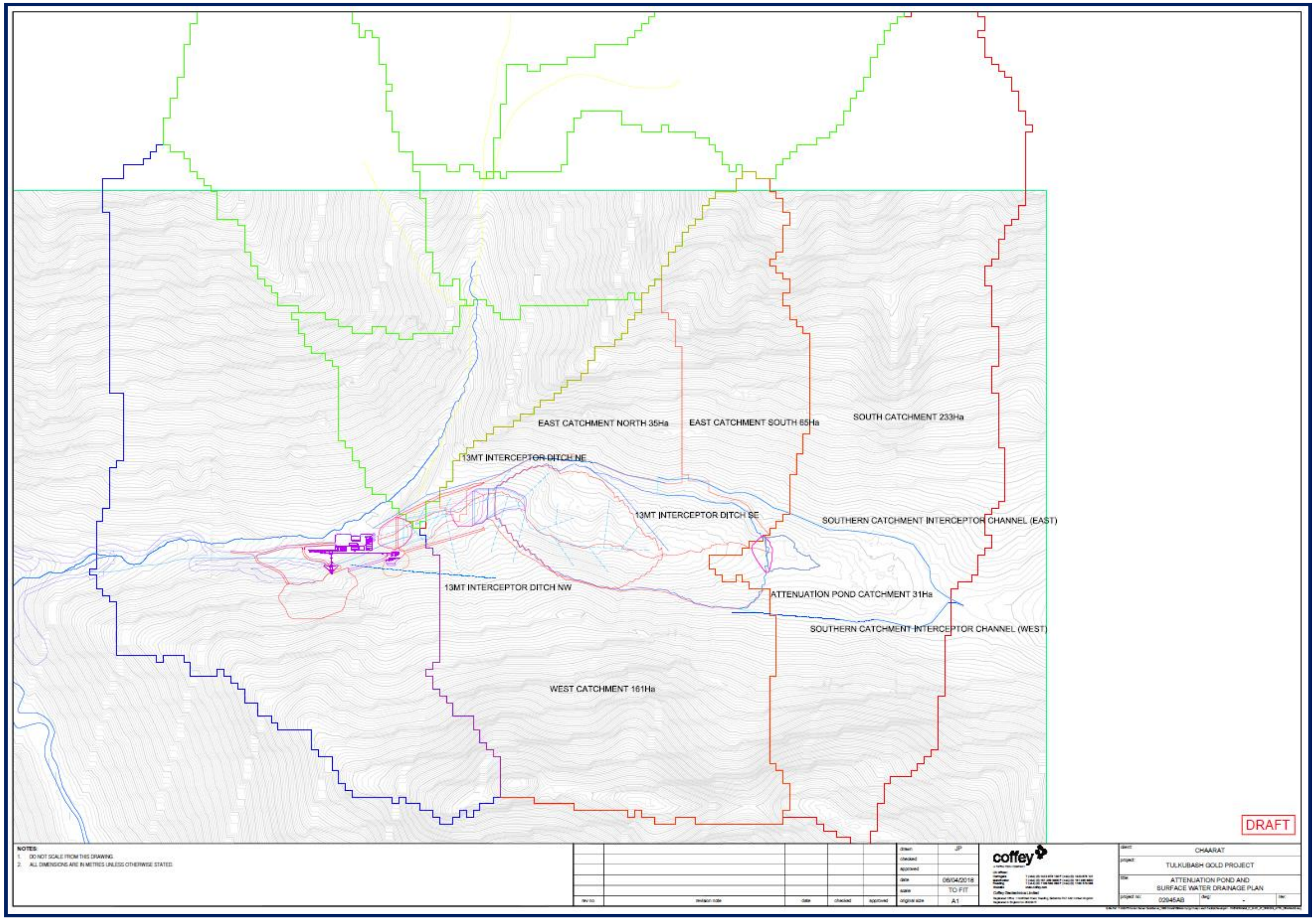
- The eastern collection drainage channel will collect surface water run-off from the east catchment slopes and divert it mainly to sediment pond 1 (and partly to the attenuation pond);
- The southern catchment channels will collect surface water run-off from the south catchment slopes mainly to the attenuation pond;
- The attenuation pond of 51,700 m^3 capacity at the south of the HLF will collect run-off from the catchment to the south and southwest of the HLF prior to discharge to settlement pond 2;
- Underliner drainage will collect and divert groundwater below the HLF; and
- A main collection pipe connecting the attenuation pond at the south of the HLF to settlement pond 2 at the north.

Catchments on the south side of the Sandalash River include those that surround other project infrastructure, such as the 360 Man Camp and the mine maintenance workshop.

For design flow purposes, 1-in-100, 24-hour flows were calculated for each of the sub-catchments to allow for the sizing of any diversion drains or road culverts to carry the design run-off.

The distribution of the catchments is shown in Figure 20-2.

FIGURE 20-2 ATTENUATION POND AND SURFACE WATER DRAINAGE PLAN



Source: Tetra Tech

The areas of the catchments referred to above and in Figure 20-2 are shown in Table 20-12.

Table 20-12 Catchment Areas

Area	Value (m^2)
HLF Footprint (25.88 Mt)	472,400
East Catchment (north)	350,000
East Catchment (south)	650,000
West Catchment	1,610,000
South Catchment	2,330,000
Attenuation Pond Catchment	2,390,000

Source: WAI 2018

These catchments are different to those assessed by WAI (2017) because of the increased area of HLF footprint and corresponding decrease in size of the external catchments. The WAI assessment considered several small catchments (1A-1C, 4A and 4B) totalling 519,000 m^2 and comprised the current east catchment (north) and part of the east catchment (south); a large catchment (5 – 3,843,000 m^2) covering the rest of east catchment (south) and the eastern part of the south catchment; and catchments on the west (2A-2C, 3A, 3B and 6) totalling 1,796,000 m^2 covering the current west catchment and part of the south catchment.

WAI (2017) evaluated the catchments as they were configured at the time, using the HEC-HMS unit hydrograph approach to assess the influence of intensity and critical duration on translation and attenuation in order to arrive at a design flows from the HLFs. Critical durations were between 30 minutes and 2 hour, i.e., not too short to reduce the volumes significantly, and not too long to spread out the arrival times too much from distant parts of the catchment. For catchments 1A-1C, 4A and 4B the total 1-in-200 flows were calculated as 6.6 m^3/s; for catchment 5 the 1-in-200 flow was 19.2 m^3/s; and for 2A-2C, 3A, 3B and 6 the total was 18.9 m^3/s.

The south catchment interceptor channels will intercept surface water run-off from the south catchment and east catchment (south), and direct flows over the slight watershed in the dry valley into that part of the valley to the south.

The design flow rates and velocities for the south catchment interceptor channels will be determined during detailed design.

The west catchment collector drains will intercept surface water run-off that infiltrates the scree on the western slopes of the valley and migrates below the HLF. The drains will comprise a series of perforated pipes installed in trenches below the HLF with a geotextile wrapped gravel surround. Subject to detailed design, a longitudinal collector drain may be constructed parallel to the HLF liner and outside the HLF footprint. Pipe sizing will be confirmed during detailed design.

The attenuation pond at the south of the HLF will collect and store run-off from the area not captured by the south catchment interceptor channels. The pond has been designed to provide a minimum storage capacity of 51,700 m^3 which has been calculated on the following basis:

- A catchment area of approximately 2,390,000 m^2;

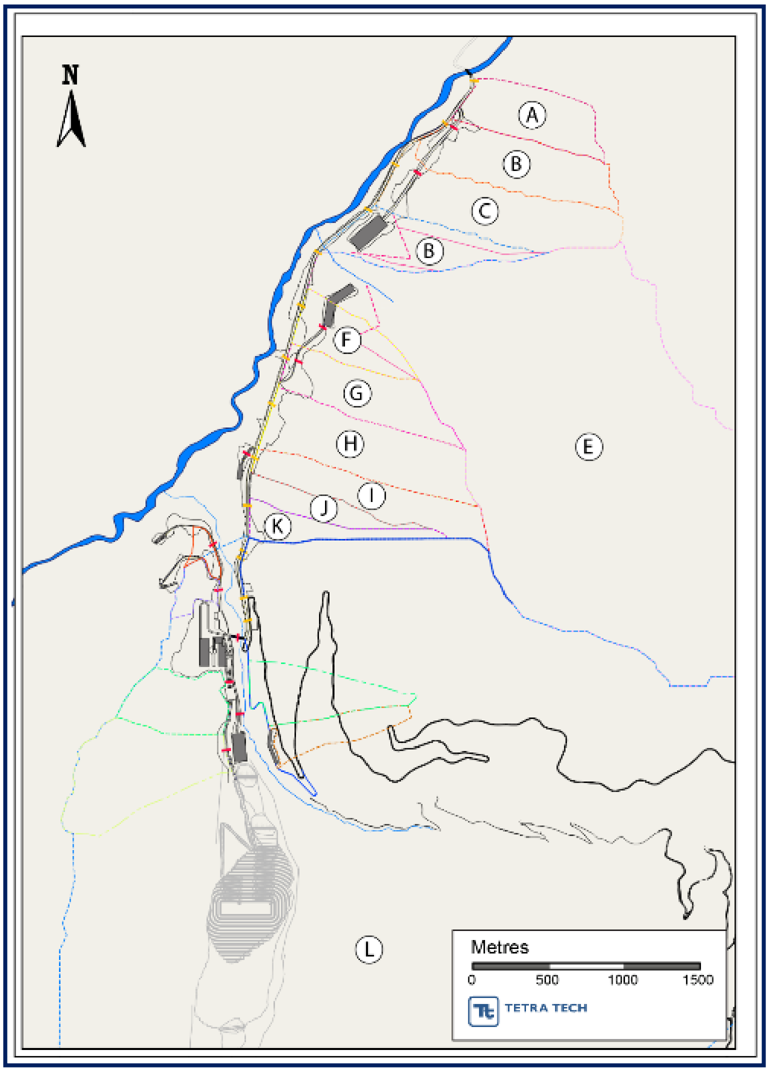
- A design rainfall event of 1-in-200-year, 24 hour, with rainfall intensity of 4.4 mm/h, which has been assessed as the critical design event (i.e., produces the largest volume of water to be stored);
- A surface water run-off coefficient of 0.7 (i.e., no infiltration); and
- An outflow rate of 1,400 ℓ/s over the duration of the storm event.

It is recommended that due to the relatively site monitoring, that continued monitoring of the climate takes place to ascertain if there is any variations in observational relationship between rainfall events and run-off in the dry valley to provide.

20.8.2.7. Other Infrastructure

Catchments on the south side of the Sandalash River (Figure 20-3) include those that surround other project infrastructure, such as the accommodation camp and the mine maintenance workshop.

FIGURE 20-3 SANDALASH RIVER SOUTH CATCHMENTS



Source: Tetra Tech

Table 20-13 shows the areas of these catchments.

TABLE 20-13 SANDALASH RIVER SOUTH CATCHMENT AREAS

Sub-catchment	Area (m^2)
A	261,784
B	364,101
C	449,637
D	265,281
E	5,261,580
F	187,477
G	285,571
H	432,199
I	291,818
J	127,431
K	137,131
L	15,160,847

Source: Google Earth

For design flow purposes, 1-in-100, 24-hour flows were calculated for each of the sub-catchments based on the intensity data in Table 20-7. For conservative purposes, it was assumed that the event (98.4 mm of rain) occurs when the ground is frozen; therefore, the run-off coefficient was assumed to be 1.0. These calculations allow for the sizing of any diversion drains or road culverts to carry the design run-off, shown in Table 20-14.

TABLE 20-14 SANDALASH RIVER SOUTH CATCHMENT DESIGN RUN-OFFS

Sub-catchment	Design Run-off (m^3/s)
A	0.30
B	0.41
C	0.51
D	0.30
E	5.99
F	0.21
G	0.33
H	0.49
I	0.33
J	0.15
K	0.16
L	17.27

Source: WAI 2018

The mine maintenance workshop is in catchment D, and the accommodation camp is divided between catchments E and F.

The ADR plant is situated on a spur between the Sandalash River and the Kumbeltash Stream at the lower end of the dry valley, and therefore receives little water except direct precipitation.

20.8.2.8. Water Supply

Two pumping stations will supply raw water to the site as described in Section 20.2.2.

20.9. Site-wide Water Balance

20.9.1. Introduction

The Project does not readily lend itself to a conventional site-wide water balance, due to a number of factors:

- The Project site is split into two areas by the Sandalash River, which will ultimately be the discharge point for the natural hydrological and hydrogeological systems on both sides. The open pits and WRD will be located on the north side of the Sandalash, while the remainder of the Project infrastructure will be located on the south side;
- Water supply is not a sensitive issue with the Sandalash River easily capable of providing Project water requirements, even at low flows;
- The open pits will not require significant dewatering, as mining only dips 50 m below the maximum projected groundwater level, 2,500 masl, before 2027;
- Site water management is dominated by the requirement for diversion of non-contact waters. Contact waters for the open pits and WRD are not expected to include any contamination, apart from suspended solids. Run-in and direct precipitation for the open pits and WRD will be collected and directed to a settling pond prior to discharge;
- The theoretical volumes in the HLF external catchment for diversion assume no infiltration, i.e., a run-off coefficient of 1.0. In the very high permeability Quaternary infill material of the dry valley this condition can only be realised at the end of winter if the ground is completely frozen, preventing any recharge to the ground. Under most circumstances there is likely to be a very low run-off coefficient resulting in little or no run-off; and
- The internal water balance of the HLF (leach pads, pregnant pond, emergency pond, etc.) will be carried out during detailed design for other than average annual conditions.

Accordingly, the site water balance is largely confined to the assessment of design run-off and implications for capacities of diversion drainage channels, as previously described in Section 20.8.

Two sets of water balance spreadsheets were produced: one for the north side of the Sandalash River and one for the south side of the Sandalash River.

Both balances contain basic data on monthly precipitation for average, dry, wet, and extreme wet years; and for the design precipitations of 1-in-100, 24-hour values for diversions, excepting the use of PMP for the HLF. It was assumed in the balances that snow precipitation accumulates from November through to March without run-off, and then thaws in March, adding to the specific precipitation for that month. Each spreadsheet also contains the areas of sub-catchments relevant to the activities being evaluated, and the run-off coefficients (factors) applied. For design floods, this is always assumed to be 1.0, which is the most conservative approach. Furthermore, the application of the 1-in-100, 24-hour events assumes that they arrive in the diversion channels instantaneously. This additional conservative assumption is offset by the possibility that shorter-duration intensities may be higher; however, in the absence of anticipated contaminants other than sediment, the only implication of short-term flows over the design flow is a temporary ingress of non-contact water into potentially contact environments.

For the north-side water balance, the output is essentially the anticipated flows on a monthly basis in the open pit and WRD diversion drains. For the south-side water balance, it is the theoretical flows in the diversion channels around the HLF footprint. Aside from the consideration of the PMP in the external catchments to the HLF, most precipitation events are likely to produce little or no run-off in the dry valley, but infiltrate into the ground and ultimately migrate to the Kumbeltash Stream or Sandalash River via the Quaternary infill groundwater system. The attenuation and dispersion mechanisms that apply to movement in groundwater will inevitably buffer the fluctuations in precipitation events and the impact of extreme events.

The balances are assumed to be relevant for construction, operation, and closure given that the principal elements are the drainage channels located with reference to the final operational configurations of the open pits and WRD, and with a 25.88 Mt HLF footprint.

20.9.2. Open Pits

Table 20-15 shows the monthly volumes diverted from, and reporting to, the open pits, for the four precipitation conditions: average, dry, wet, and extreme wet.

Figure 20-4, Figure 20-5, Figure 20-6, and Figure 20-7 show, respectively, open pit diversions and inflows; waste dump and stockpile catchment outflows; run-off diverted around waste dump; and total discharge to the Sandalash from the waste dump and the open pit diversions; all in m^3/hr.

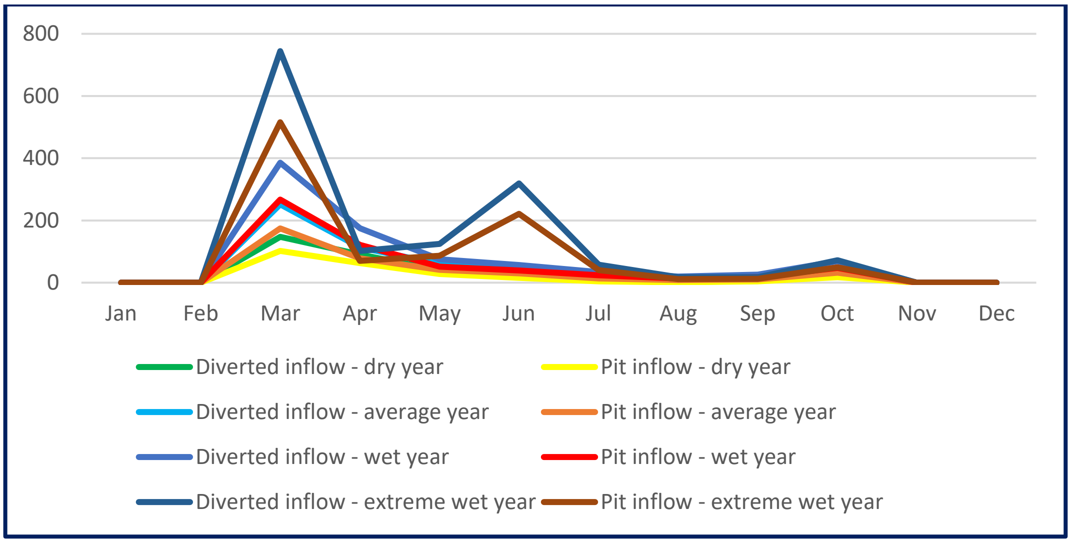
The run-in and direct precipitation is available for uses such as dust suppression.

TABLE 20-15 MONTHLY DIVERTED CATCHMENT RUN-OFF AND INPUTS TO OPEN PITS

		Month (m³)											
		Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Total Diverted Catchment	Average Year	0	0	187,248	81,274	44,621	31,075	14,342	8,765	10,358	34,262	0	0
	Dry Year	0	0	109,560	65,338	29,482	15,936	4,781	1,594	3,984	19,123	0	0
	Wet Year	0	0	286,848	125,894	55,776	40,637	25,498	14,342	18,326	50,995	0	0
	Extreme Wet Year	0	0	553,776	73,306	92,429	229,478	43,027	12,749	12,749	53,386	0	0
Run-in and Direct Precipitation	Average Year	0	0	129,720	56,304	30,912	21,528	9,936	6,072	7,176	23,736	0	0
	Dry Year	0	0	75,900	45,264	20,424	11,040	3,312	1,104	2,760	13,248	0	0
	Wet Year	0	0	198,720	87,216	38,640	28,152	17,664	9,936	12,696	35,328	0	0
	Extreme Wet Year	0	0	383,640	50,784	64,032	158,976	29,808	8,832	8,832	36,984	0	0

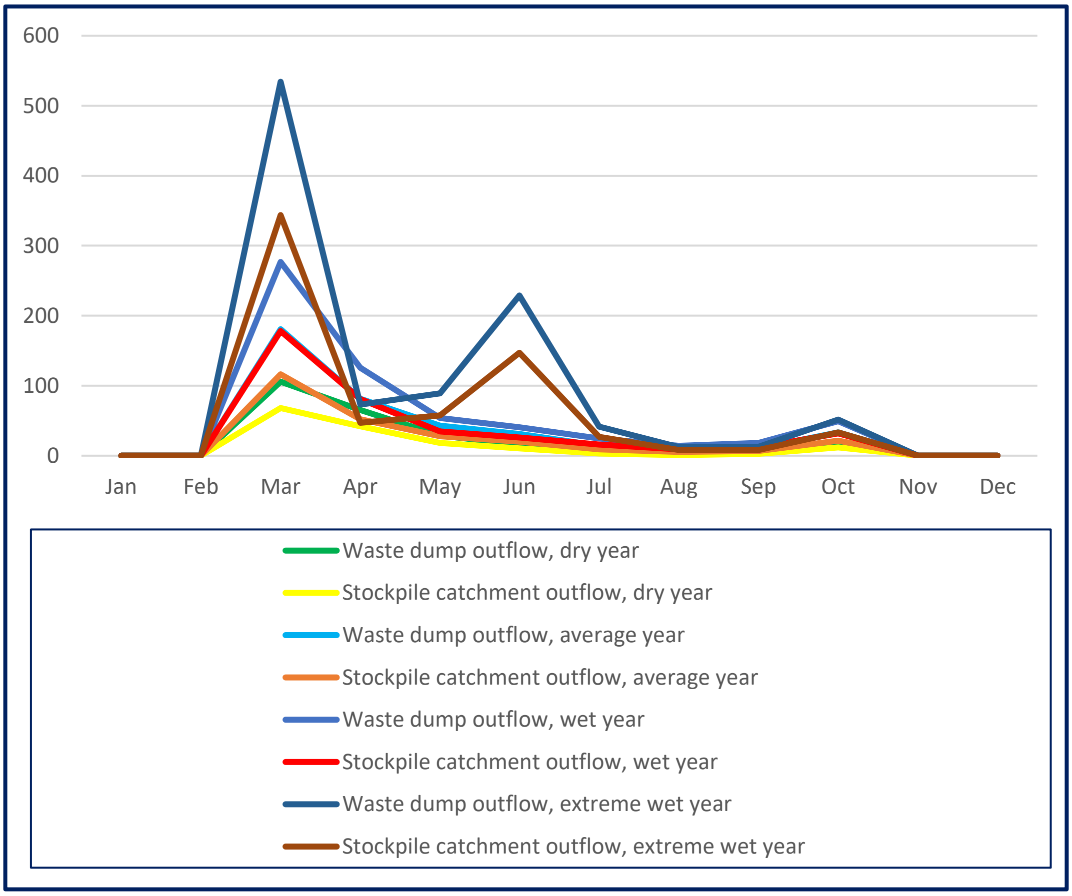
Source: WAI 2018

Figure 20-4 Open Pit Diversions and Inflows (m^3/hr)



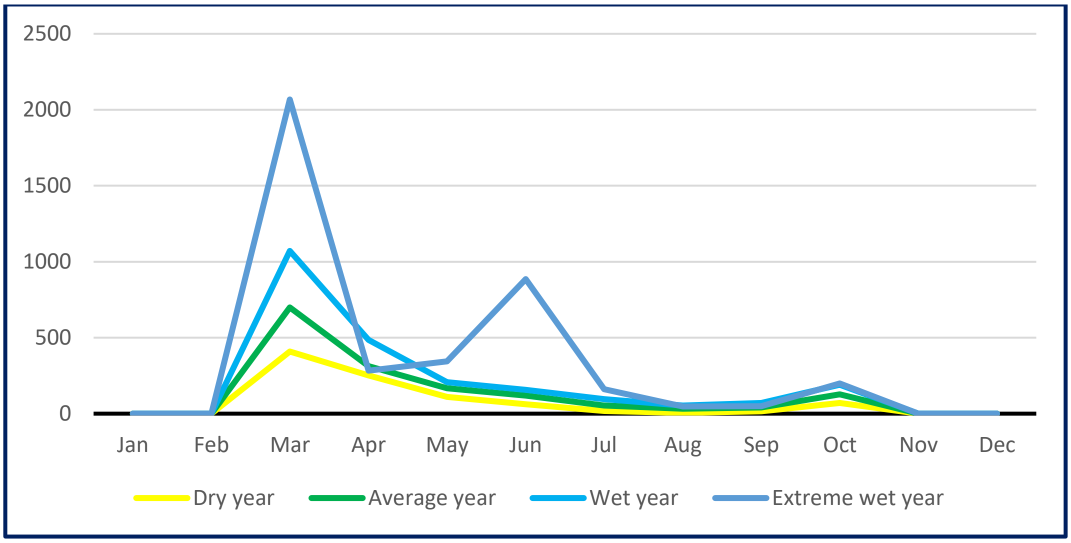
Source: WAI 2018

Figure 20-5 Waste Dump and Stockpile Catchment Outflows (m^3/hr)



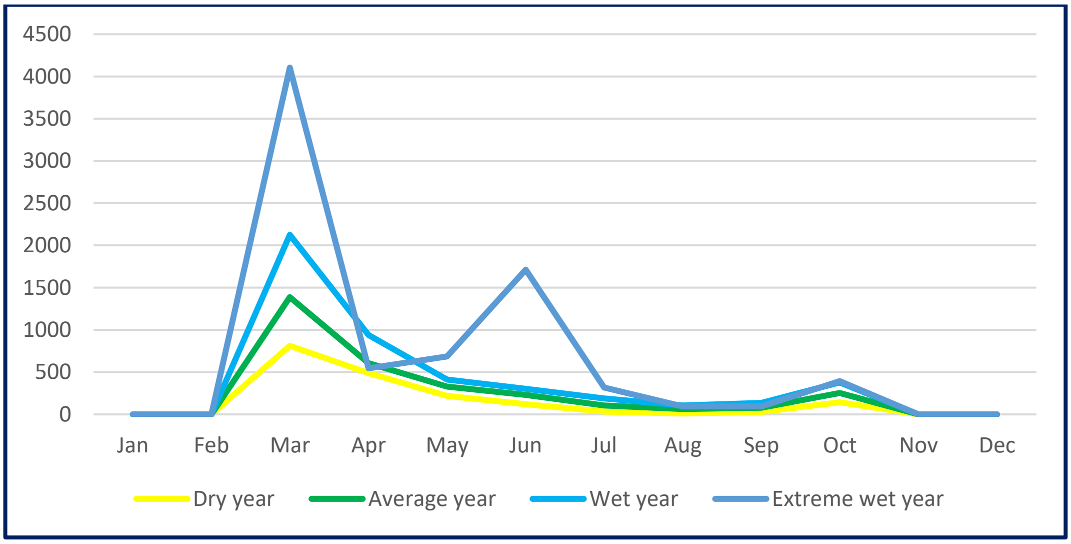
Source: WAI 2018

FIGURE 20-6 RUN-OFF DIVERTED AROUND WASTE DUMP (M^3/HR)



Source: WAI 2018

FIGURE 20-7 TOTAL DISCHARGE TO SANDALASH RIVER FROM WRD CATCHMENT AND OPEN PIT DIVERSIONS (M^3/HR)



Source: WAUI 2018

20.9.3. HLF CATCHMENT

Table 20-16 shows the monthly volumes derived from the HLF and its external catchments for the four precipitation conditions: average, dry, wet, and extreme wet. For the external catchments, a run-off factor of 0.4 (conservative) is assumed. Figure 20-8, Figure 20-9, Figure 20-10, and Figure 20-11 show precipitation flows to these catchments in m^3/hr for the four rainfall conditions

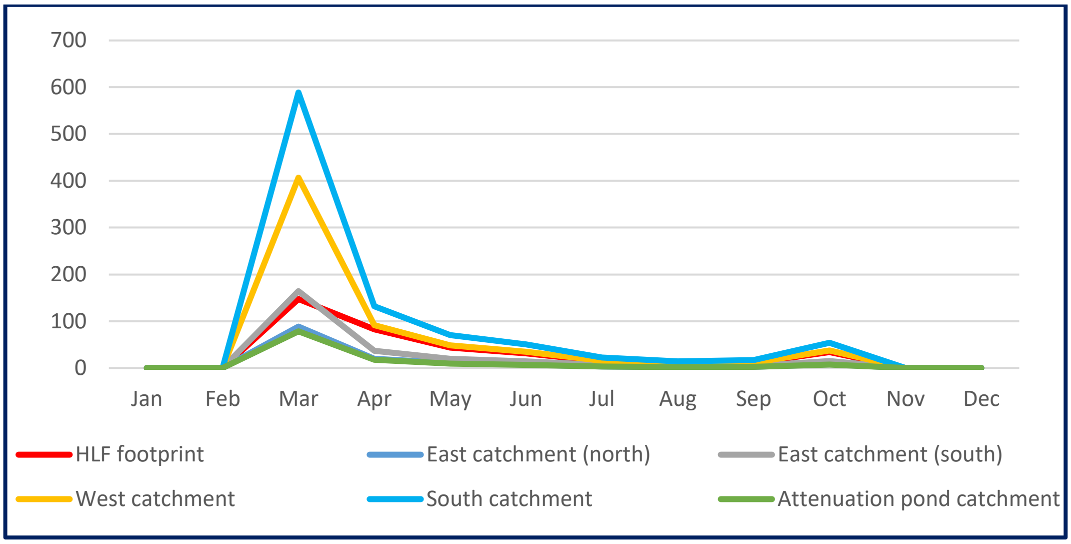
TABLE 20-16 MONTHLY RUN-OFF VOLUMES FOR HLF AND EXTERNAL CATCHMENTS

	Month (m^3)											
	Jan	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**
Average Year												
HLF Footprint	0	0	109,604	59,466	32,648	22,737	10,494	6,413	7,579	25,069	0	0
HLF East Catchment (north)	0	0	65,800	14,280	7,840	5,460	2,520	1,540	1,820	6,020	0	0
HLF East Catchment (south)	0	0	122,200	26,520	14,560	10,140	4,680	2,860	3,380	11,180	0	0
HLF West Catchment	0	0	302,680	65,688	36,064	25,116	11,592	7,084	8,372	27,692	0	0
HLF South Catchment	0	0	438,040	95,064	52,192	36,348	16,776	10,252	12,116	40,076	0	0
HLF Attenuation Pond Catchment	0	0	58,280	12,648	6,944	4,836	2,232	1,364	1,612	5,332	0	0
Dry Year												
HLF Footprint	0	0	64,130	47,806	21,571	11,660	3,498	1,166	2,915	13,992	0	0
HLF East Catchment (north)	0	0	38,500	11,480	5,180	2,800	840	280	700	3,360	0	0
HLF East Catchment (south)	0	0	71,500	21,320	9,620	5,200	1,560	520	1,300	6,240	0	0
HLF West Catchment	0	0	177,100	52,808	23,828	12,880	3,864	1,288	3,220	15,456	0	0
HLF South Catchment	0	0	256,300	76,424	34,484	18,640	5,592	1,864	4,660	22,368	0	0
HLF Attenuation Pond Catchment	0	0	34,100	10,168	4,588	2,480	744	248	620	2,976	0	0
Wet Year												
HLF Footprint	0	0	167,904	92,114	40,810	29,733	18,656	10,494	13,409	37,312	0	0
HLF East Catchment (north)	0	0	100,800	22,120	9,800	7,140	4,480	2,520	3,220	8,960	0	0
HLF East Catchment (south)	0	0	187,200	41,080	18,200	13,260	8,320	4,680	5,980	16,640	0	0
HLF West Catchment	0	0	463,680	101,752	45,080	32,844	20,608	11,592	14,812	41,216	0	0
HLF South Catchment	0	0	671,040	147,256	65,240	47,532	29,824	16,776	21,436	59,648	0	0
HLF Attenuation Pond Catchment	0	0	89,280	19,592	8,680	6,324	3,968	2,232	2,852	7,936	0	0
Extreme Wet Year												
HLF Footprint	0	0	324,148	53,636	67,628	167,904	31,482	9,328	9,328	39,061	0	0

	Month (m^3)											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
HLF East Catchment (north)	0	0	194,600	12,880	16,240	40,320	7,560	2,240	2,240	9,380	0	0
HLF East Catchment (south)	0	0	361,400	23,920	30,160	74,880	14,040	4,160	4,160	17,420	0	0
HLF West Catchment	0	0	895,160	59,248	74,704	185,472	34,776	10,304	10,304	43,148	0	0
HLF South Catchment	0	0	1,295,480	85,744	108,112	268,416	50,328	14,912	14,912	62,444	0	0
HLF Attenuation Pond Catchment	0	0	172,360	11,408	14,384	35712	6,696	1,984	1,984	8,308	0	0

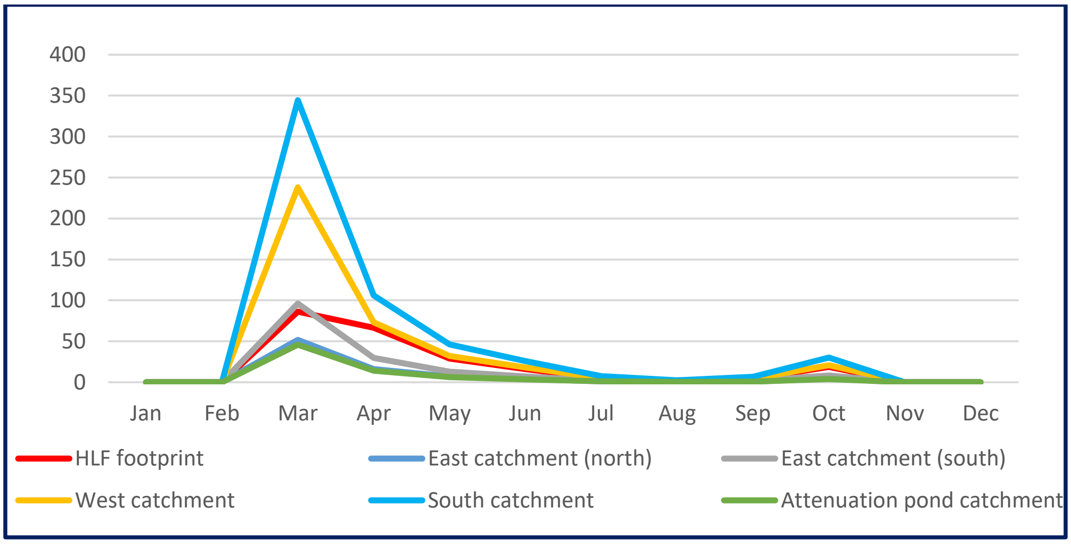
Source: WAI 2018

Figure 20-8 Average Year Precipitation Flow to HLF Catchment (m^3/hr)



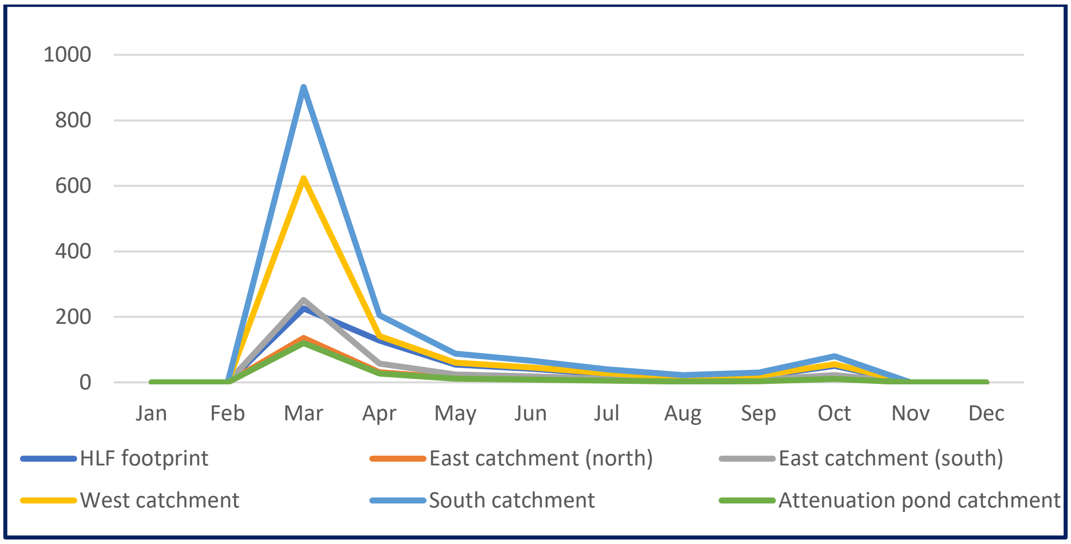
Source: WAI 2018

Figure 20-9 Dry Year Precipitation Flow to HLF Catchment (m^3/hr)



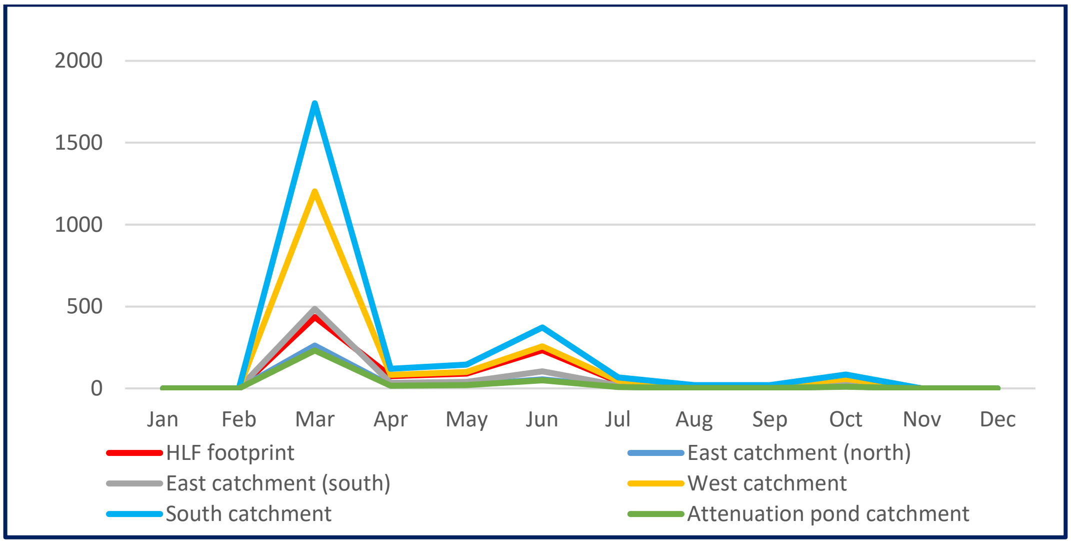
Source: WAI 2018

FIGURE 20-10 WET YEAR PRECIPITATION FLOW TO HLF CATCHMENT (M^3/HR)



Source: WAI 2018

FIGURE 20-11 EXTREME WET YEAR PRECIPITATION FLOW TO HLF CATCHMENT (M^3/HR)



Source: WAI 2018

20.10. HYDROGEOLOGY

General background information for the Tulkubash hydrogeology initially focused on the proposed pits and underground workings to the northeast of Tulkubash, around the winter camp and Adit 4 (Itasca and Atkinson 2010; SRK 2010); however, recent studies and investigations also include the area around the Tulkubash pit and Adit 2 (Tetra Tech 2014b).

Preliminary groundwater modelling was undertaken (Itasca/Atkinson 2010), but only based on site investigations and testing around Adit 4. Groundwater levels were extrapolated to

Tulkubash; however, it should be recognized that this can only be regarded as indirect evidence, although some measured levels have subsequently been made (Tetra Tech 2014a).

20.10.1. Geological Background

This section summarises aspects of the Property geology that are most likely to have some influence on the groundwater regime, particularly where heterogeneity and/or anisotropy might introduce and influence the hydraulic conductivity and groundwater movement.

The Sandalash Valley exposes a north-easterly (30º) trending sequence of Cambro-Ordovician sedimentary rocks that dip approximately 40º to the northwest (300º). This sedimentary sequence is cut by several strike faults and younger intrusions. The lower sequences of Cambro-Ordovician metasiltstones and argillite's have been termed the Karator and Chaarat Formations, which are unconformably overlain by a thick sequence of Devonian quartzites termed the Tulkubash Formation.

The mineralisation in the Tulkubash deposit is defined as deep epithermal to the approximately 600 m of vertical extent explored so far (Tetra Tech 2014b), and appears to be related to a series of diorite, monzonite, and rhyolite intrusions that have been intruded along major, regional-scale faults and thrusts along the Sandalash Valley.

Structurally, sinistral strike-slip faults, normal faults, rotated blocks, and multiple horst and graben structures dominate the area. In places, the currently active faults can be correlated to the orebodies. As mineralisation is pre-Cenozoic in age, the presently active faults supposedly reactivate pre-existing mineralised faults.

The dominant and presently active fault system, the Irisai Fault System, is parallel to the Sandalash River. The major faults of the Irisai Fault System can be traced along strike to the northeast to the Kara Buura Pass, where they terminate in the Talas-Ferghana Fault, and also several kilometers to the southwest. The Irisai -related faults dip towards the northwest or southeast, with medium to steep dips.

Tetra Tech (2014b) refers to the Sandalash Valley representing the fault-controlled hinge zone of an anticline, with the proposed mine area on the northwest dipping limb.

20.10.2. Groundwater Regime

20.10.2.1. Aquifers and Aquifer Properties

In the mine area, SRK (2010) reported that groundwater occurs and flows primarily in fractures and fault zones in the otherwise low permeability shales and argillites. SRK believes the ore zone itself is a zone of relatively high hydraulic conductivity, evidenced by the inflow in Adit 4 (near the winter camp, north of the Tulkubash site) coming primarily from the ore zone or drillholes.

During a field investigation in January 2010, only one relatively reliable value of hydraulic conductivity was obtained from a flow/shut-in recovery test in a sub-horizontal borehole (Itasca and Atkinson 2010). The borehole tested was situated inside Adit 4 in drill chamber 5 (DC5) and was designated BH4680 (known on site as BH12). DC5 is approximately 700 m west-southwest of the portal to Adit 4, and BH4680 extends from there 132 m to the south-southeast.

Itasca and Atkinson (2010) describes the testing as follows:

Two flow and shut-in/recovery tests were conducted in horizontal test hole BH 4680 (BH-12) in DC5, the first test after the corehole - which had surface casing to a length of 8 m - had been drilled to a length of 100 m and the second test after the corehole was at 130 m. Analysis of the resulting data indicates that the vertical hydraulic conductivity (since the horizontal corehole would have intersected primarily high-angle joints and fractures) is in the range of 9×10^{-4} to 5×10^{-3} m/d with a geometric mean value of 2×10^{-3} m/d. It should be noted that this value, which includes both the hydraulic conductivity of all joints/fractures encountered in the corehole and the lower hydraulic conductivity of the rock matrix, is the effective hydraulic conductivity of the so-called "equivalent porous media"

(In other words, a bulk average appropriate for modelling at most scales (apart from very localized.

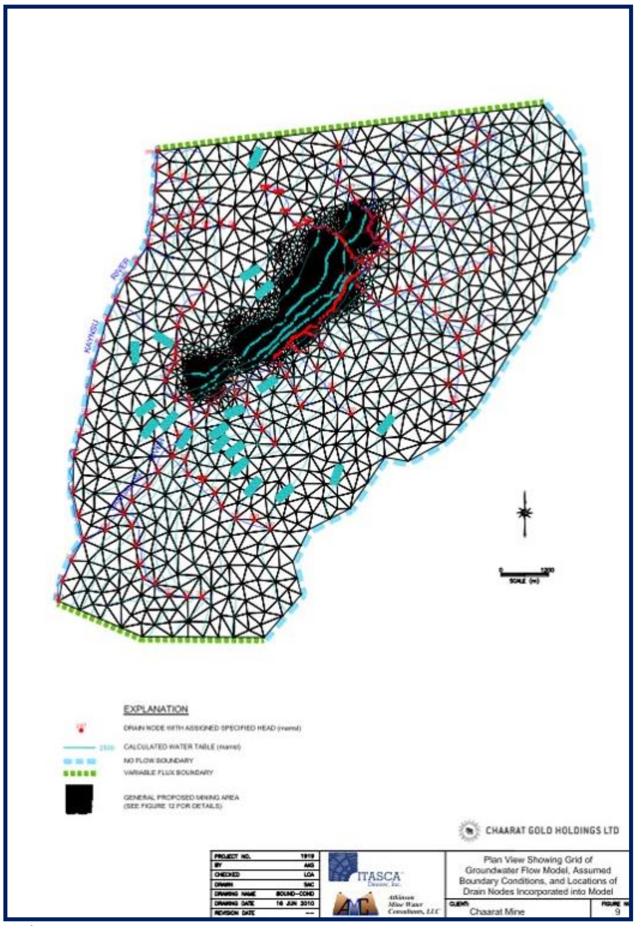
The value of 2×10^{-3} m/d is quite reasonable for rocks of this nature and was the initial or starting value used in calibrating the model.

The actual data are not provided in Itasca and Atkinson (2010), but a graphical analysis is shown. The duration of the test is not stated, although a test flow rate of 3.2 m^3/hr (0.89 ℓ/s) is given for the 8-100 m test and 3.1 m^3/hr (0.86 ℓ/s) for the 8-130 m test. It is assumed that the report was referring to adjacent boreholes BH4720 and BH4800 as the monitoring holes. The report approximates the hydraulic conductivity result to 1×10^{-3} to 3×10^{-3} m/d.

20.10.2.2. Groundwater Model

A preliminary 3D numerical groundwater flow model of the Property area was developed and described in Itasca and Atkinson (2010). The model focused on the proposed mine developments around Adit 4 and the Northwest pit, although the Tulkubash pit was included in the model domain as one of the open pits for which an estimate of dewatering inflow was a model objective. The model was constructed using Itasca and Atkinson's own finite-element code, MINEDW, derived from the United States Geological Survey's (USGSs) FEMFLOW. The finite-element grid used for the Chaarat model is shown in map view in Figure 20-12. The mesh is more finely discretised in the proposed mining areas where the horizontal dimensions of the elements are about 15 m to 20 m. The Tulkubash pit area is the furthest southwest of the several pits represented in Figure 20-12.

Figure 20-12 Element Grid of Preliminary Groundwater Flow Model



Source: Itasca and Atkinson (2010)

Vertically, the model grid has five regional layers in and near the vicinity of the confirmed ore zones, with additional layers around Adit 4. Principal boundary conditions adopted were fixed head boundaries representing the river system, the main mode of discharge from the local groundwater regime (either direct or via the Sandalash Valley alluvium), and the no-flow boundaries at the drainage divides. During calibration, a single fault was incorporated along the Sandalash River to facilitate discharge in the model.

Prior to the more recent hydrogeological investigations and testing, field data available for the preliminary model were very limited, and so possible ranges of hydraulic properties over the proposed Chaarat Property were primarily derived from calibration of the model using assumed recharge, water levels, discharge from Adit 4, and data from the hydraulic testing available at the time.

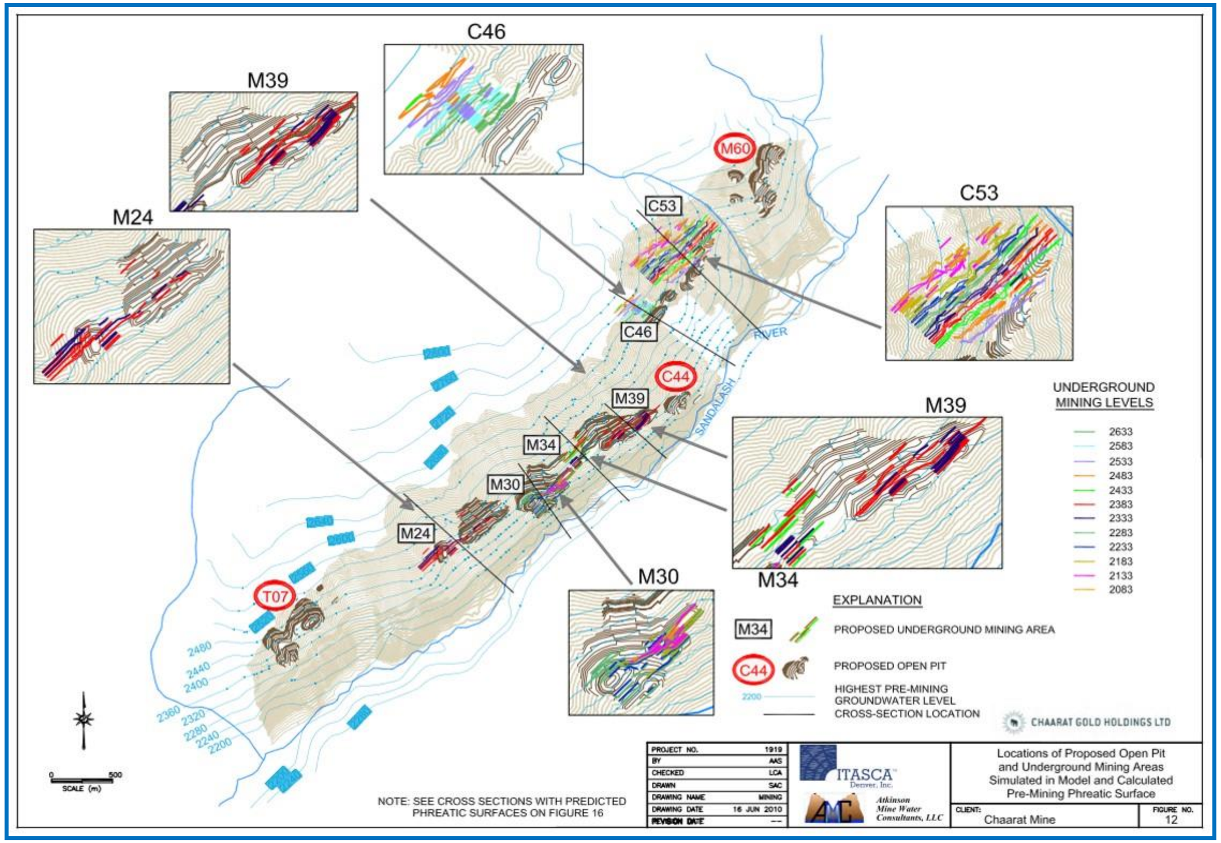
The Tulkubash and Sandalash formations were attributed with a hydraulic conductivity value of 3×10^{-2} m/d, decreasing with depth; and the Sandalash River alluvium 5^{-10} m/d. It was assumed that the hydraulic properties of the rocks in the proposed mining areas are homogenous and isotropic, with only some differentiation with depth. Hydraulic conductivities are likely to be higher around and beneath the rivers. Based on the inflows to Adit 4, the ore zone was assumed to have a relatively large hydraulic conductivity of 0.1 m/d. Dykes and the granodiorite were assumed in the model to be local barriers with hydraulic conductivity of 5 x

10^{-3} m/d. Specific yield for the Tulkubash and Sandalash formations was allocated a value of 0.005, with 0.01 in the ore zone, 0.005 in the granodiorite, and 0.1 in the alluvium.

The use of a groundwater model in this way (i.e. when specific data is sparse) is an acceptable approach to developing insights into the system being investigated and to inform the priorities and scope of subsequent investigations. Conventionally, the model would be revisited, updated, and improved as necessary following those investigations. To date however no further modelling has been carried out.

Preliminary groundwater modelling resulted in a calibrated groundwater levels map of the pre-mining water level (phreatic surface) as calculated during steady state simulations (Figure 20-13) The Tulkubash pit is represented as T07 on this figure, although the pit shell used is different from the current version, with the latter indicating a final pit bottom elevation of 2,540 masl compared with the 2,430 masl in the model.

FIGURE 20-13 PRE-MINING CALCULATED WATER TABLE FROM PRELIMINARY GROUNDWATER MODEL



Source: Itasca and Atkinson 2010

The nominal pre-mining groundwater elevation determined for the preliminary model was 2,440 masl, with a range between 2,500 masl and 2,340 masl across the Tulkubash pit footprint. The model calibration process included a five-year transient simulation to evaluate seasonally-variable recharge. Superimposed on this was a sensitivity analysis of variation in specific yield (for which there was, and still is not, any actual data). Across a range of reasonable values, the annual range of groundwater levels varied between 5.0 m for a specific yield of 0.02 to 14.6 m for a specific yield of 0.0005

The model predicted that only T07/Tulkubash of the open pits simulated would experience groundwater inflow for the open pit development considered at the time. Predicted inflows fluctuated seasonally between 140 and 330 m^3/d for the first four years of the mine plan proposed at the time, and declined to a range of 40 to 100 m^3/d in subsequent years. This estimate excluded direct precipitation and run-in to the pit.

Itasca and Atkinson (2010) acknowledge the inevitable limitations of the preliminary model, given the amount of data available to them at the time, and the necessary assumptions (albeit reasonable) that had to be made. Some of the predictions were reviewed by Tetra Tech (2014a) following acquisition of more data on inflows around Adit 4:

> *The model predicted that inflows to the underground mine developments would be in the range of 6,000 to approximately 7,000 m^3/d and cumulative inflows would likely not exceed 8,000 m^3/d under maximum development of the mine. Actual measured groundwater inflow into one section of Adit 4 that was constructed after the groundwater model was developed was 10,340 m^3/d, and actual inflow to another section of Adit 4 was 6,150 m^3/d. These substantial upward divergences from the flows that had been estimated by the model indicate that the model requires recalibration and may require significant revision in addition to the recalibration.*

The above comment is perfectly valid, and may equally affect the conclusions from the model regarding groundwater levels and potential inflows to the Tulkubash pit.

20.10.2.3. Groundwater Levels

Prior to the development of the Itasca and Atkinson groundwater model, there was no direct information on groundwater levels in the vicinity of the Tulkubash pit. The model derived pre-mining groundwater levels from its initial calibration (which was mainly aimed at conditions around Adit 4). This groundwater levels map is shown in Figure 20-13.

The proposed mining areas shown in Figure 20-13 appear to show pre-mining water elevations mostly in the range between 2,300 masl and 2,500 masl, with the levels at T07/Tulkubash between 2,340 masl and 2,500 masl. Neither the map nor the report (Itasca and Atkinson 2010) indicate what time of year this represents. Sensitivity analysis in the calibration process suggested an annual range between approximately 5 m and 15 m depending on assumed ranges in the (unknown) specific yield.

The currently planned Main Zone open pit is designed to reach an elevation of about 2350 m ASL or approximately 50 m below daylight at completion. Mining is planned to proceed 50 m below the maximum projected pre-mining water level of 2,500 masl in both 2025 and 2026. Mining to the final pit bottom at occurs in 2027.

The groundwater contours generated by the model indicate a pre-mining groundwater flow direction to the southeast and the Sandalash River, with a gradient of about 0.4, which is steep but consistent with the topography and low bulk permeability of the groundwater regime. It is also possible that structurally-controlled anisotropy in the system results in a higher hydraulic conductivity in the southwest-northeast direction compared with northwest-southeast, which would also contribute to a steeper gradient in the latter direction.

The elevations in the floor of the Sandalash Valley drop from approximately 2,400 masl to 2,100 masl along its length covered by the overall model domain, and from 2,280 masl to 2,180 masl alongside the mining areas shown in Figure 20-13.

Tetra Tech (2014a) refers to ongoing monitoring of six surface piezometers, of which three were extant at the time, and three underground piezometers in Adit 4. Tetra Tech summarise the data as follows:

> *Monitoring of water levels in piezometers at the site has shown that groundwater levels in the bedrock relatively high above the valley floor can rise sharply by a few meters to a few tens of meters in response to spring snowmelt and then decline gradually through the remainder of the year. This response is indicative of local recharge infiltrating rapidly through fractures that are hydraulically connected to the land surface. Water level changes were not analysed relative to site meteorological data to determine whether a similar relationship exists between precipitation events and bedrock groundwater levels.*

20.10.3. Tulkubash Dewatering

Initial groundwater levels in the footprint of the Tulkubash pit are expected to be in the range between 2,300 masl and 2,500 masl. This is corroborated by the measured level of 2,492 masl in borehole KP103-13.

When mine development reaches level 2,500 masl during the Year 3, groundwater inflows will be managed by a combination of pumps in the pit and drainage channels on the surface which will direct flow to a sedimentation pond at the base of the WRD the capacity is 25,000 m^3.

However, depressurisation is essential where required to ensure that the pit walls are not subject to enhanced stability issues resulting from saturation. This could be required above the regional water table, for example in local perched horizons or where the rocks are almost fully saturated due to the low rate of vertical groundwater recharge. In practise depressurisation can only be carried out during excavation of the pit, and at this time a programme of drilling of horizontal drains should be implemented where necessary. It is likely that this will need to concentrate on the northwest wall where pre-mining groundwater levels are highest. There should be a programme of installation of piezometers to monitor groundwater pressure conditions and the impact of the drains. Otherwise, visual observation of seepage areas during excavation will indicate where additional horizontal drains are necessary.

A sump and associated pumping plant may be needed in the pit to receive direct precipitation and run-in from the pit margins inside the peripheral drains. Drainage from any horizontal depressurisation drains should be collected and piped down to the sump (or to intermediate higher level sumps if appropriate). It should not be allowed to discharge in an uncontrolled manner as this may cause problems in benches below.

Other surface run-off should be intercepted and diverted round the pit perimeter to minimise run-in. Typically, these would be 50 m to 100 m from the pit perimeter to prevent leakage into the pit due to relaxation in the rocks closer to the margin.

20.10.4. DRY VALLEY

During 2018, in the dry valley where the HLF is located, KyrgyzGiiz GI installed several groundwater Bore Holes and measurements were taken that establish the general Quatemary infill water table. The elevation of the Sandalash Valley below the dry valley is approximately 2,150 masl, so it appears likely at this stage, with the likely high general permeability in the dry valley rubbly infill, that any groundwater recharge rapidly seeps to a water table controlled by the geometry of the base of the infill and the level of the Sandalash River, towards which groundwater will migrate.

Process water supply will be obtained from pumps installed close to the Process Plant adjacent to the Kumbeltash Stream.

21. Capital and Operating Cost Estimate

21.1. Capital Cost Estimate

21.1.1. Summary

The total estimated initial capital cost for the, construction, installation, and commissioning of all facilities and equipment is USD115.5 M with USD15.2 M deferred to later in the project cycle. A summary breakdown of the initial capital cost is provided in Table 21-1 and further capital cost details are provided in Appendix H.

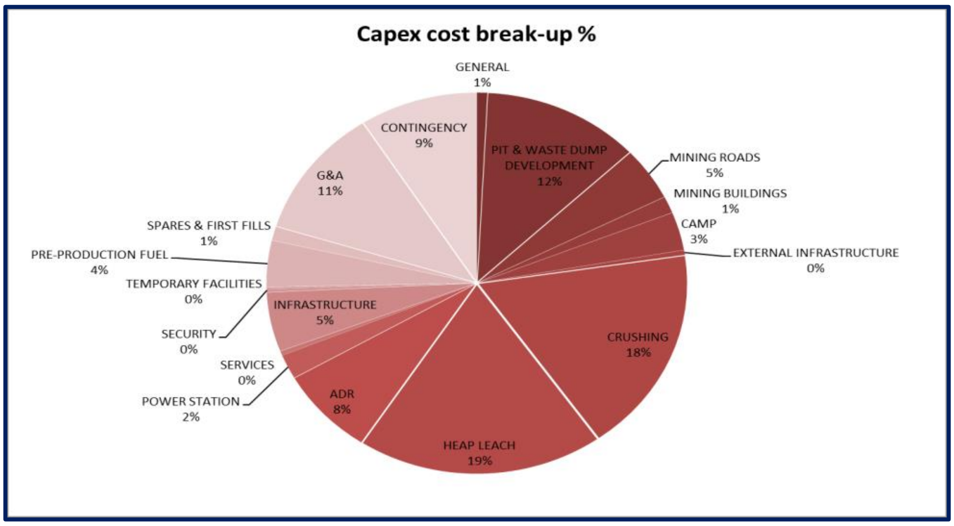
Table 21-1 Initial Capital Cost Summary

Area	Total (USD'000)
Mining	21,996
Infrastructure	4,236
Process Plant	59,807
Owners Cost	18, 913
Contingency @ 10%	10,496
Total Initial Capital Cost	**115,446**

This estimate includes direct field costs required to execute the project, plus indirect cost overheads, commercial requirements, and management. This estimate is based on the updated pricing for the major areas and escalation / inflation factors added to rates used based on the 2019 estimate to bring it in line with 2021 terms, Amounts in this capital cost estimate are expressed in United States Dollars (USD), unless otherwise noted.

Graph 21-1 illustrates an alternative view of the costs per area, showing the cost percentage splits required to establish the mine, where each area is a high-level representation of the commercial package required to execute the Project. Graph 21-1 excludes the contingency allowance.

GRAPH 21-1 INITIAL CAPITAL COSTS PER AREA (AS A PERCENTAGE)



21.1.2. BASIS OF ESTIMATE

This estimate is prepared in accordance with the AACE International cost estimate classification system. The estimate accuracy range is –10% to +10%. This 2021 BFS shows an accuracy improvement over the 2019 BFS (-10% to +15%) due to the noted increase in designs completed, and the actualised costs that have made the cost estimations more accurate. This equates to an AACE Class 3 estimation.

While some of the estimates are based on information from the previous BFS in Real 2019 Terms, the majority areas of the projects were revalidated by the main contractors Ausenco, YPT and Azmet. The equipment supplied not covered by the major contractors an 8% escalation was added to bring the equipment costs in line with 2021 rates.

21.1.2.1. RESPONSIBILITY MATRIX

LogiProc, as the lead consultant, produced the Project estimate. A team of engineers, and cost estimators developed the capital cost estimate with inputs from Chaarat. Table 21-2.

TABLE 21-2 ESTIMATE RESPONSIBILITY MATRIX

Area	Company
Mining (Open Pit)	Chaarat
Processing	Chaarat / LogiProc
Infrastructure	Chaarat / LogiProc
Site Facilities	Chaarat / LogiProc
Indirect Costs	Chaarat
Owner's Costs	Chaarat
Allowances	Chaarat / LogiProc

21.1.2.2. Estimate Approach

The Cost Breakdown Structure (CBS) for the capital estimate is based on the Work Breakdown Structure (WBS) indicated in Appendix I. The purpose of the CBS is to break the total project cost into a sum of smaller individual cost items.

The estimate is divided into Initial and Deferred capital costs. The Initial cost section of the estimate is divided into categories, each representing a specific geographical location. Each category is divided into areas representing different physical entities, be it a service, a building, a process, etc. Each area is divided by discipline containing quantities and unit costs for each activity or equipment required for any given area. Each discipline is broken down into individual cost items (activity or equipment).

For each individual cost item, the quantities, as well as the unit costs for the material, equipment, and installation are estimated. The capital estimate is the sum of all individual cost items.

21.1.2.3. Initial Costs

21.1.2.3.1. Categories

The Initial capital Cost estimate including pre-production operating expenses is divided into four different categories: Mining, Infrastructure, Process Plant, and Owners Costs:

1. **Mining** - The Mining category includes all pre-production cost items related to the mining site and activities, which include but are not limited to mobilisation of mining equipment, Pit and Waste dump development, Mining Roads, and Mining Buildings.
2. **Infrastructure** - The Infrastructure includes mainly the 360 Man Camp and the upgrade of the Kumbel pass to Tulkubash road.
   The 360 Man person accommodation camp capital estimate is based on a quote received from Çiftay and actual costs of partial camps from construction contractor. The accommodation camp will be a self-contained, multi-building facility that includes accommodation, mess, ablution, recreation and laundry services.
3. **Process Plant** - The Process Plant category includes all cost items related to the process plant site, which include but are not limited to all process equipment, Crushing, Heap Leach, ADR, Power Station, Services, Infrastructure (Process Buildings, Process Roads, and general), and Security.
4. **Owners Cost** - The Owners Cost category includes all cost items related to the temporary facilities, Pre-Production fuels, Spares & First fills, and G&A.

21.1.2.3.2. Areas

Areas were created in each category to break down the cost estimate into physical entities for buildings, processes, services, etc.

21.1.2.3.3. Disciplines

The disciplines are composed of earthworks, civil, structural, plate work, mechanical, piping, electrical & Instrumentation, transport, Projects services, pre-production, and consumables.

21.1.2.3.4. Activities / Equipment

Activities and/or equipment are assigned a number based on the project P&ID's which in turn are based on the WBS.

21.1.2.4. DEFERRED COSTS

21.1.2.4.1. CATEGORIES

Deferred costs cover Deferred Heap Leach, Mine Closure and Deferred Equipment.

21.1.2.5. CONTINGENCY

This allowance of 10% is included to cover uncertainties in both the Initial and Deferred Capital estimates. Such uncertainty could arise from interpretations related to VAT, Import Duties, escalation, and foreign exchange.

Estimated contingencies are allowances for undefined items incurred within the defined scope of work covered by the estimate, which cannot be explicitly foreseen or described at the time the estimate is completed due to the lack of complete accurate and detailed information.

The contingency allowance is an integral part of the estimate; it is not to be considered as a compensating factor for estimating inaccuracy, nor is it intended to cover such items as any potential labour disputes, currency fluctuations, escalation, force majeure, or other uncontrolled risk factors. It should be assumed that the contingency amount will be spent over the engineering and construction period.

The total initial capital contingency allowance is USD10.5 M, which is 10% of the initial capital cost estimate.

The total deferred capital contingency allowance is USD1.4 M, which is 10% of the deferred capital cost estimate.

21.1.2.6. COST INFORMATION

For each individual cost item, cost data was entered in various disciplines. Descriptions of the disciplines are found below.

21.1.2.7. EARTHWORKS

Earthworks were based on updated quantities supplied by LogiProc and Chaarat. Unit rates were applied, using the negotiated rates from Çiftay.

21.1.2.8. CIVILS

Concrete works were based on the completed designs done by contractors in 2020 and were priced using itemised bills of quantities by contractors who are acquainted with the region, including indicative costing for overheads associated with concrete works.

21.1.2.9. STRUCTURAL

Structural steel was mainly quantified by the selected vendors as part of their supply. Structures falling outside of the main process areas were quantified by LogiProc and Chaarat. The cost of these items outside the packages was estimated based on quantities multiplied with rates.

21.1.2.10. PLATEWORK

Platework, tanks and liners are based on the supplied information for the selected vendors. An estimate was made of the quantities for all platework, tanks, launders, pump boxes, and chutes, which fall outside of the main process packages. The cost of these items outside the packages was estimated based on quantities multiplied with rates.

21.1.2.11. MECHANICAL (EQUIPMENT)

For the major process sections, selected vendors were identified and approached to provide budget quotation, the documents supplied by the vendors included the scope of work and pricing schedules.

Chaarat has opted to work with the following selected vendors:

- Pamir Mining for mining and earthworks, including the Mine Workshop;
- YPT for the supply of the Crusher circuit;
- Azmet for the supply of the ADR, Gold Room and Reagents;
- Ausenco for the design of the Heap Leach;
- LogiProc for the non-major equipment covered under services;
- LogiProc material take-offs (MTO's) for the process area earthworks and roadworks; and
- Chaarat engineers for the MTO's for the haul road, mining buildings etc.

All equipment and material costs are included as Free Carrier (FCA) manufacturer plant Incoterms 2010; other costs, such as spares, taxes, duties, freight, and packaging are covered in the estimate.

21.1.2.12. PIPING

Piping within the major supply packages is included in the vendor costs. Piping falling outside of the main process areas constitutes a minor portion of the estimate. Previously compiled MTOs from the P&IDs and process plant model were utilised in the study.

21.1.2.13. ELECTRICAL, AND INSTRUMENTATION

The cost for electrical infrastructure and distribution consists of the following main elements:

- Power generation, for which quotations for containerised generation units were obtained;
- Switchrooms (facilities and switchgear), for which preliminary quotations were compared with costs for similar installations and provisions allowed for accordingly;
- Medium-voltage ring main power distribution system; and
- Low-voltage power distribution and lighting, and small power to the process and facilities areas.

21.1.2.14. TRANSPORT

Transport costs were considered individually for significant items, and further estimated with allowances, as well as generalised allowances for bulk material transport. Cost assessments from generalised rates priced by Move One Inc. (a specialist logistics company in the region) during the initial Feasibility Study were applied, and the cost estimates were compared with advice received and estimating sheets prepared by ICT, a logistics co-ordinator.

21.1.2.15. Construction and Installation

Construction and installation costs were estimated based on the quantities and pricing schedules supplied by the vendors. These contracts will include the supply and installation of concrete, steel, platework, piping, and electrical and control cabling and cable support, but includes installation only of equipment, which will be procured directly by, or on behalf of, Chaarat, and free-issued to the installers.

The cost of accommodation, fuel, electrical power, and general site requirements such as medical services, are priced in the general provision of such services and not as part of the construction and installation cost.

21.1.2.16. Project Currency and Foreign Exchange

As stated earlier, the capital cost estimate is expressed in USD. All major equipment cost and rates were obtained in USD currency. Any minor cost obtained in other currencies were converted to USD based utilising prevailing exchange rates during the study.

21.1.2.17. Duties and Taxes

A base estimate of the capital costs was first determined, exclusive of import duties and taxes, which were then separately assessed line by line in the capital estimate detail based on specialist advice.

Chaarat received tax advice from Baker Tilly Bishkek, LLC, that confirmed the only taxes applicable to the Project in the Kyrgyz Republic are VAT and import duties, neither of which are recoverable. Based on this advice, Chaarat reviewed the Project capital costs and, where applicable, applied the appropriate VAT and import duty rates. Baker Tilly Bishkek reviewed the Chaarat tax assumptions and approved the VAT and import duty application to broad categories of equipment and facilities.

21.1.2.18. Measurement System

The International System of Units measurement system is the standard measurements system used in this estimate.

21.1.3. Cost Change Methodology

The capital cost increases are described below in Table 21-3. The method used was to revalidate vendor packages, update quantities based on the design completed in 2020, actualised cost in 2019/2020, and the development of the additional roads based on the new pit development plan.

Table 21-3 Cost Change Narrative

Area	Description	Changes
M110	Mobilization of Ciftay / Equipment	• Actualised cost removed.

Area	Description	Changes
M200	Pit and Waste Dump Development	• According to the new Pit design the cost of pre-stripping is decreased.
M300	Mine Roads	• New quantities obtained and double accounting for vat was removed. • New designs of the (3) three pit roads were used instead of the single east pit road. • New design units and rates were used.
M400	Mining Buildings	• New Quantities and rates obtained for AN and Detonator storages. • Truckshop building scope increased based on Ciftay's recommendations for efficient continuation of maintenance work.
I300	External Infrastructure	• Revised cost was used based on supplied data
I100	Camp	• Sunk cost removed. Camp standard improved.
P100	Crushing	• New rates were obtained from the vendor. • Equipment costs increased by 4% due to inflation.
P200	Heap Leach	• Updated quantities and rates were obtained based on the completed design at IFC level. • Freight costs included to Bishkek site. • Custom Clearance costs added for pipes. • Transport cost added for the HLF piping. • Minor equipment costs increased by 8% due to inflation. • General construction cost increase by 5% due to inflation. • Fence was included in final estimate. • Previously in BFS there were 4 phases. In 2021 BFS Update this was reduced to 3 phases due to constructability concerns. Therefore, USD2.3 M of deferred cost shifted to initial CAPEX in BFS.
P300	ADR	• Main Circuit costs increase by 8,9% due to inflation. • New rates were obtained for steelwork, tanks and piping these costs increased by 21%. • Preferred Supplier Rates increased by 3.8% • Valve cost increased by 6.2% • Electrical & Control cost increased by 39.8% this is mainly due to the E-house which is included into the cost.
P400	Power Station	• General Increase of between 5% (construction cost) and 8% (on Equipment).
P500	Services	• General Increase of between 5% (construction cost) and 8% (on Equipment).
P600	Infrastructure	• New quantities were obtained based on updated calculations. • Rates increased. • Equipment costs increased by 8% due to inflation

Area	Description	Changes
		• Kumbeltash Stream Platform cost was introduced.
P700	Security	• General Increase of between 5% (construction cost) and 8% (on Equipment).
O100	Temporary Facilities	• None.
0200	Pre-Production Fuel	• New quantities obtained due to east pit road cancellation.
O300	Spares & First Fills	• General increase in equipment 8% due to inflation.
O400	G&A	• Revised according to new schedule and new target completion date in August 2023. Personnel wage and accommodation duration is 4 months less than BFS duration. • Other indirect costs were introduced.
O500	Mine closure	• None.
C5	Contingency	• General increase due to 10% allowance.

21.1.4. Initial Costs

21.1.4.1. Mining

The mining capital costs are shown in Table 21-4. The road works allowed for under the mining cost included Ore Haul road, pit roads, the camp access road and platform, and the roads to the ammonia nitrate and detonator access roads. The mine maintenance workshop will be supplied and maintained by Çiftay during the life of the project. This portion of the estimate also covers the pre-production cost of mining.

Table 21-4 Mining Capital Cost Estimate

Area	Cost (USD'000)
General	0,929
Pit & Waste Dump Development	14,180
Mining Roads	5,246
Mining Buildings	1,640
Total Mining Capital Cost	**21,996**

Mining costs in the pre-production period include Contract Mining costs and Owner Mining costs as described below.

The Contractors mining costs during the pre-production period are costs incurred by the contractor to conduct mining and mine development in the open pit area. They are governed by a mining contract. A description of these cost can be found in the Section 16.

The Owner Mining costs during the pre-production period are related to the cost of the Owner's mining team who provides contract oversight, contractor management, mine planning, and ore control functions. These costs can be broken into three components, labour, expenses, and lab costs. An explanation of these costs can be found in Section 19.

The Contractors mining costs during the pre-production period total USD12.6 M or USD1.70/t mined. Owner mining costs during pre-production total USD1.4 M or USD0.19/t mined.

21.1.4.2. Infrastructure

Infrastructure includes the upgrade of the Kumbel pass to Tulkubash road and the 360 Man shift Camp.

Table 21-5 Infrastructure Capital Cost Estimate

Area	Cost (USD'000)
Camp	3,761
External Infrastructure	0,475
Total Infrastructure Capital Cost	4,236

21.1.4.3. Process Plant

Chaarat approached selected vendors to supply the key process areas as vendor packages. Budget quotations were obtained for all of the major items based on the preliminary process specifications.

The following vendor design and supply companies were used in the BFS:

- YPT (Turkey) was selected for the design and manufacture of the crusher plant, lime plant and load out station complete with the required conveyors and buildings;
- Azmet (South Africa) was selected for design and manufacture of the ADR plant, Electrowining and gold room complete with the reagent make up and storage including the supply of all buildings; and
- Ausenco (Canada) was selected for the design of the heap leach Facility c/w all relevant ponds.

Table 21-6 provides a summary of the process capital cost estimate by system. Included in each area is the earthworks, civil, structural, plate work and electrical work required to supply and install.

Table 21-6 Process Capital Cost Estimate

Area	Cost (USD'000)
Crushing	20,266
Heap Leach Facility	21,909
ADR	8,652
Power Plant	2,409
Services	0,423

Area	Cost (USD'000)
Infrastructure	5,847
Security	0,302
Total Process Capital Cost	**59,807**

The Process infrastructure includes Process buildings, Process roads, Process control infrastructure, workshop tools, special safety equipment and the training of the operators and maintenance teams.

21.1.4.4. Owners Cost

Owners costs include temporary facilities, pre-production fuel, spares and first fills, and G&A. The estimate assumes no engineering, procurement and construction management (EPCM) contractor, with the Owner hiring personnel and consultants as required.

Table 21-7 shows the summary of the Owners costs.

Table 21-7 Owners Cost Estimate

Area	Cost (USD'000)
Temp. Facilities	0,190
Pre-Production Fuel	4,548
Spares & First Fills	1,351
G&A	12,824
Total Indirect Capital Costs	**18,913**

21.1.4.4.1. Temporary Facilities

The temporary facilities include the rental of the batch plant and the excavation of the borrow pits required during the construction phase.

21.1.4.4.2. Pre-Production Fuels

The fuel estimate includes the free issue of the construction fuel as per the contractual agreement between Chaarat and Çiftay. The estimate also includes the provision for the snow fall management and the camp fuel consumption during the construction phase of the project.

21.1.4.4.3. Spares & First Fills

The first fills estimates include provision for stocking the warehouse with initial reagent consumables and general maintenance spares.

21.1.4.4.4. G&A Cost

The Owner's cost provides for Owner-related activities associated with the project, and includes the allocation of head office costs for Chaarat, as well as EPCM functions. The estimate was prepared with a list of line items for various personnel, equipment, and general costs.

New mobile equipment costs are based on in-country supplier quotes and internal database pricing.

21.1.5. Deferred Cost

21.1.5.1. Deferred Heap Leach

The Heap Leach will be constructed in 3 phases during the mine life. Phase one will form part of the initial capital cost and phase 2 to 3 will from part of deferred costs.

Construction phases:

- Phase 1 – 6.48 Mt, constructed in year 2021-2023; cost of USD8,490 M.
- Phase 2 - 10.29 Mt, constructed in years 2024-2025; cost of USD4,346 M.
- Phase 3 - 9,11 Mt, constructed in years 2026-2027; cost of USD1,938 M.

21.1.5.2. Mine Closure

Once the mining operation has stopped and the remaining ore from the mine and ROM pad is processed and placed on the HLF for final leaching, rinsing, and closure, the machinery and personnel will be reassigned to complete the earthworks required for mine closure.

The activities required to perform the majority of the mine closure will use personnel and machinery already accounted for in the operating and capital costs. Earth moving and fabrication of physical barriers, making machinery safe, emptying vessels, and moving waste material to the WRD will be systematically performed as final production ramps down at the end of mine life.

An estimate of USD6,511,000 (including taxes) was developed for the labour and operating costs of the HLF during the flushing, drainage, and rehabilitation stages of the closure plan.

Much of the process plant and equipment will have a design life of up to 20 years. Due to the short LoM, it is assumed that the residual value of the processing plant and equipment will cover the costs of dismantle and removal from site, as well as rehabilitation of the mine, process, and infrastructure areas.

21.1.5.3. Deferred Equipment

The main equipment deferred in the process plant is located in the Crusher and ADR section. The third tertiary crusher has been deferred as the plant will start off by crushing the ore to a P_{80} of 12.5 mm, and in the ADR mercury removal equipment has been deferred.

21.1.6. Qualifications – Assumptions and Exclusions

21.1.6.1. Qualifications

In developing the capital cost estimate, LogiProc made the following assumptions:

- Materials for construction will be readily available, with local contractors and the client team knowledgeable and experienced to arrange transport in the region;
- Project execution will commence in the current economic climate where equipment and contractors are readily available to supply equipment and

perform the works and the market is competitive in favour of the purchasers; and

- The relative political stability in the region that allows relative ease of transport stays as at the time of preparing the estimate.

21.1.6.2. Exclusions

The following items were excluded from the capital cost estimate:

- Cost escalation during construction;
- Interest on loans and other financing costs during construction;
- Events that could generate extra costs if they occur:
    - Large-scale unexpected ground conditions;
    - Extraordinary climate events;
    - Labour disputes;
    - Schedule recovery or acceleration;
    - Force majeure; and
    - Currency fluctuations.
- Activities that will occur but are excluded:
    - Financing costs;
    - Taxes (except as supplied by Owner);
    - Cost outside LogiProc's battery limits;
    - Sunk costs; and
    - Research and exploration drilling costs.

The following items have been excluded from the capital cost estimate but are included in the financial model:

- Working capital and
- Operating costs. (Noting that Pre-production Opex costs are captured in Capex)

21.2. Operating Cost Estimate

21.2.1. Summary

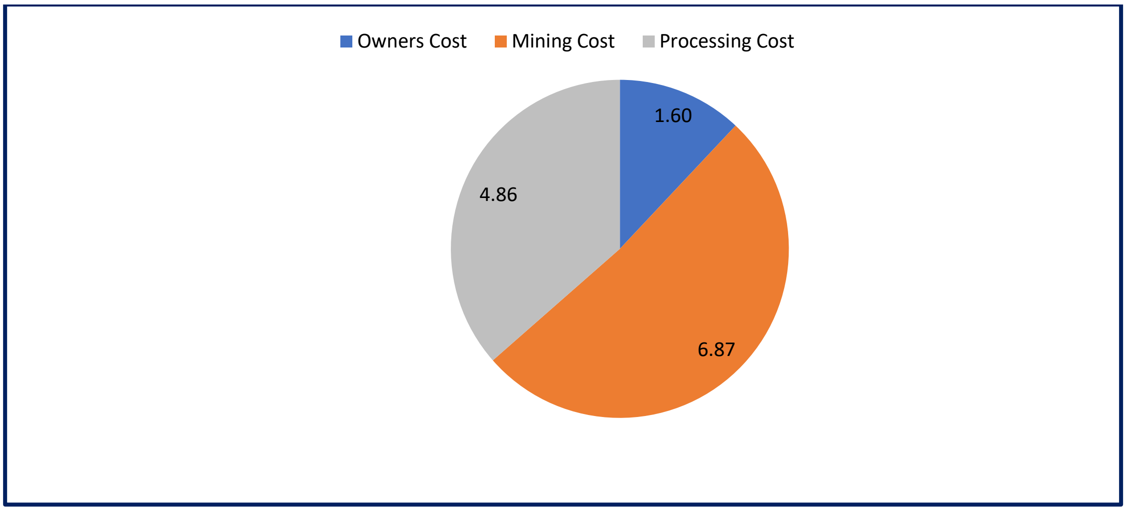
The total operating cost estimate over the LoM (for this section LoM excludes pre-production as the pre-production costs are capitalized) is shown in Table 21-8 and operating unit costs are shown in Graph 21-2.

Table 21-8 LoM Operating Cost Estimate

Area	Total (USD 000's)
Owners Cost	32,456
Mining Cost	139,301

Area	Total (USD 000's)
Processing Cost	98,579
Total LoM Operating Cost	**270,336**

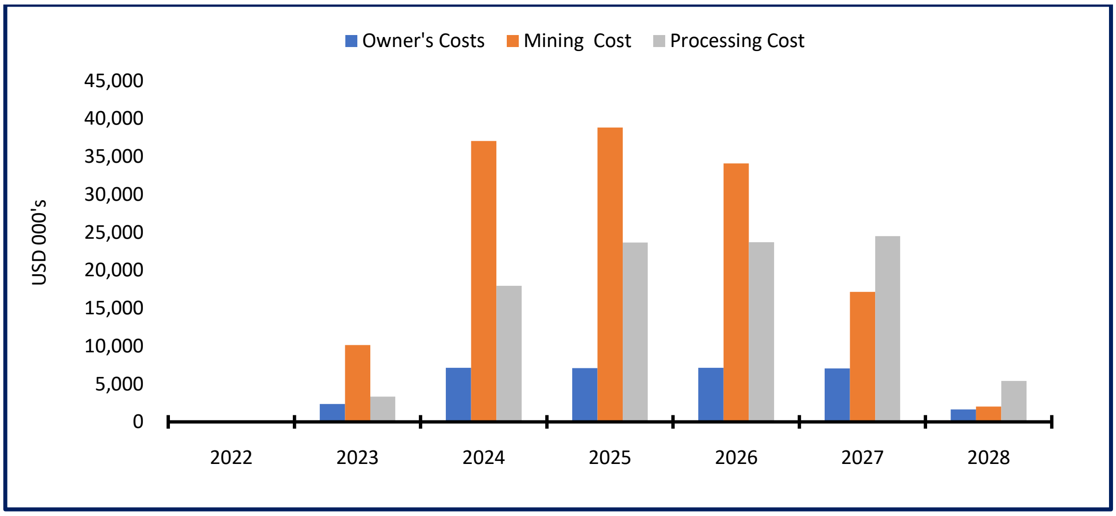
Graph 21-2 LoM Unit Operating Costs (USD/t ROM)



The mine cost comprises 52% of the total operating costs, processing and the owners cost make up 36% and 12% respectively.

Graph 21-3 illustrates the operating cost forecast over the LoM, Pre-production operating costs are shown as zero as they will be capitalised.

Graph 21-3 LoM Operating Cost Forecast



The total operating cost estimate over the LoM is shown in Table 21-9 and the unit operating costs are shown in Table 21-10.

Table 21-9 LoM Operating Cost Estimate

Area		Cost Including VAT (USD'000)
Contract Mining Cost		139,301
Owner Mining Costs		5,863
G&A Costs		26,592
Processing Costs		86,459
Stacking Cost		12,120
Refining Costs		3,782
Gold Transport Costs		319
Royalty Cost		73,583
Total LoM Operating Cost		**348,020**

Vat – 12 %

Table 21-10 Operating Cost Estimate per tonne (inclulding VAT)

Area		Cost Including VAT (USD/t ore)
Contract Mining Cost		6.87
Owner Mining Costs		0.29
G&A Costs		1.27
Processing Costs		4.23
Stacking Cost		0.59
Refining Costs		0.18
Gold Transport Costs		0.02
Royalty Cost		3.53
Total LoM Operating Cost		**16.98**

21.2.2. Owners Cost

The Owners cost comprises of the expenses related to an owners mining team and general and administration costs (G&A). The combined Owners cost over the steady state period is USD32.46 M.

An Owners mining team will oversee the contract mining operation and to manage grade control. In the first year of operation it is planned that the laboratory will fire assay all the mining grade control samples as well as assaying with hot cyanide to develop a basis and in subsequent years the laboratory costs will be reduced to 10% fire assays. The estimated LoM costs (including VAT) at full production are included in Table 21-11.

Table 21-11 Owners Mining Team Cost Estimate

Item	2023 (USD'000)	2024 (USD'000)	2025 (USD'000)	2026 (USD'000)	2027 (USD'000)	2028 (USD'000)	Total (USD'000)
Labour	232	696	696	696	696	116	**3,132**
Expenses	118	355	355	355	355	59	**1,598**
Lab	70	245	268	291	230	30	**1,134**
Total	420	1,296	1,319	1,342	1,281	205	**5,863**

The total Owners mining costs for the LoM, excluding the pre-production, amounts to USD5.86 M.

The general and administration costs are based on the labour costs provided by Chaarat. A detailed breakdown of the LoM G&A costs over the steady state period is shown in Table 21-12

Table 21-12 General and Administration Cost Estimate

Item	2023 (USD'000)	2024 (USD'000)	2025 (USD'000)	2026 (USD'000)	2027 (USD'000)	2028 (USD'000)	Total (USD'000)
Salaries	521	1,564	1,564	1,564	1,564	391	7,168
HR	6	19	19	19	19	5	85
Transport	78	233	233	233	233	58	1,067
Camp Catering	133	399	399	399	399	100	1,827
Finance & Admin	80	240	240	240	240	60	1,100
IT/Comms	55	166	86	86	86	22	501
Environment	30	91	91	91	91	23	417
Health & Safety	92	277	277	277	277	69	1,268
CR	87	260	260	260	260	65	1,192
Reclamation Fee	8	24	24	24	24	6	110
Chatkal Yard	6	19	19	19	19	5	88
Site Services	102	305	305	305	305	76	1,399
Power	603	1,808	1,808	1,808	1,808	452	8,285
Vehicles	66	198	198	198	198	50	908

Item	2023	2024	2025	2026	2027	2028	Total
	(USD'000)	(USD'000)	(USD'000)	(USD'000)	(USD'000)	(USD'000)	(USD'000)
Haul Road Maintenance (Fuel)	43	129	129	129	129	32	589
Kumbel Road Maintenance (Fuel)	43	129	129	129	129	32	589
TOTAL (USD)	**1,953**	**5,859**	**5,779**	**5,779**	**5,779**	**1,445**	**26,592**

21.2.3. Mining Cost

Mining costs are based on the production schedule, contracted mining rates and a projected fuel price of USD0.60/ℓ.

21.2.3.1. Contractor's Mining Costs

The Contractor's mining fee is driven by a unit rate based on the cumulative material mined and hauled. The rate increases for every 13.2 Mt mined. The contract mining cost also contains an overhaul component. The major portion of the mining cost estimate is based on the agreement with the mining contractor (Section 19). Contractor supervision is included in the contractor costs, but grade control and technical services are excluded as they are included in the owner mining cost.

The following provides further explanation of the contractor mining costs for the Project to be applied in the BFS. The terms of the mining contract have been agreed and finalised with Çiftay. These notes only explain the current structure and estimate of the contract mining costs.

The cost estimate is based on a variable rate structure with an initial unit cost of USD3.70/bcm. The rate increases every 5 M bcm or 13.2 Mt up to a total quantity of 25 M bcm or 66 Mt. After 25 M bcm is mined, the unit rate for mining is fixed at USD4.50/bcm. The staged increase in the unit mining rate can be seen in Appendix I. The cost of mining including fuel and other adjustments over the life-of-mine (LoM) is estimated to be approximately USD139.3 M.

While included in the mining cost, the contracted mining rate does not include the cost of fuel. Fuel will be free-issued to the contractor by the Owner. The assumed price of fuel is based on quotes received in 2019 Real Terms from local suppliers. The fuel cost was estimated separately based on projected consumption for drilling, blasting, loading, hauling, and all other mining unit operations. The overall cost of fuel over the LoM is estimated to be approximately USD16.2 M.

After 25 M bcm is mined, the cost of mining is adjusted based on the average annual haulage distance. If the weighted average annual haul for all material exceeds 5,000 m, a premium is paid on the tonnage hauled while if the average the manual haul is less than 5,000 m, a savings is experienced. Tonnage on average hauls up to 6,000 m or 1 km over the 5,000 m limit are subject to an additional charge. The surcharge increases from 6,000 – 7,000 m and again for all distances over 7,000 m.

The application of the overhaul surcharge starts in 2024 and lasts to the end of mining in 2028. The average cost of overhaul is expected to be USD0.88 M during the LoM.

The contractor rates used to calculate the mining cost are shown in Table 21-13. A density of 2.64 t/m³ was applied to determine the USD/t unit costs.

Table 21-13 Contractors Base Mining Fee (incl. VAT)

Material Mined	(USD/m³ mined)	(USD/t mined)
< 5 million cubic meters	4.14	1.57
5 to 10 million cubic meters	4.54	1.72
10 to 15 million cubic meters	4.76	1.80
15 to 20 million cubic meters	4.93	1.87
20 to 25 million cubic meters	5.04	1.91

The cash flowforecast of the Contractors mining cost for the LoM, excluding pre-production is presented in Table 21-14.

Table 21-14 Contractors Mining Cost over LoM (USD'000)

Item	2023	2024	2025	2026	2027	2028	Total
	(USD'000)	(USD'000)	(USD'000)	(USD'000)	(USD'000)	(USD'000)	(USD'000)
Contract Mining Fee	9,725	31,831	34,249	30,312	14,498	1,642	122,258
Fuel	902	4,690	4,546	3,819	1,968	242	16,166
Overhaul	(489)	542	54	(48)	675	143	878
Total	**10,138**	**37,064**	**38,849**	**34,084**	**17,140**	**2,027**	**139,301**

Overhaul is negative as this is a cost to the Contractor, not to the Owner.

The total contractors mining costs, for the LoM excluding the pre-production, amounts to USD139.3 M, or USD6.87/t ROM ore. This includes the base unit rate, fuel, overhaul and 12% VAT.

21.2.4. Processing Cost

The process operating cost includes:

- Labour (operations and maintenance);
- Reagents (plant);
- Consumables (plant);
- Maintenance spares;
- Power; and
- Light vehicles.

The total process operating cost for treating the oxide ore using the proposed flowsheet (Section 17) is estimated at USD4.23/t (including VAT). The stacking costs were presented separately and are estimated at USD0.59/t.

The process operating cost summary for the oxide and primary ore is presented in Table 21-15.

TABLE 21-15 PROCESS OPERATING COST SUMMARY

	Process Operating Cost (USD/t of ore processed)
	Including VAT
Consumables	0.53
Reagents	2.26
Power	1.00
Light Vehicles	0.02
Labour	0.25
Maintenance and Spares	0.18
Total	**4.23**

21.2.4.1. LABOUR

A summary of the process operational and maintenance labour costs is shown in Table 21-16 and Table 21-17, respectively. The labour costs were developed based on a schedule of labour costs (fully burdened employment costs to Chaarat) by labour category in the Kyrgyz Republic, which was provided by Chaarat.

TABLE 21-16 PROCESS LABOUR COST ESTIMATE

Position	No. of Staff	Employment Cost (USD/a)	Total Cost (USD/a)
General			
Process Manager	1	220,000	220,000
Metallurgical Superintendent	1	200,000	200,000
Metallurgist	2	19,050	38,100
Mill Clerk	2	5,475	10,950
Subtotal	6	-	469,050
Operations			
Shift Foremen	4	7,400	29,600
Front-end Loader Driver (ROM Pad)	Outsourced to Mining Contractor		
Crusher Operators	4	6,450	25,800
Crusher Day Labourers	4	3,600	14,400
Heap Leach Operators	4	6,450	25,800
Stacking Operators	4	6,450	25,800
Front-end Loader Driver (Truck Stacking)	Outsourced to Mining Contractor		
Piping Crew	8	6,450	51,600
Heap Dozer Operator	Outsourced to Mining Contractor		
Reagent Preparation	4	6,450	25,800
Heap Leach Day Labourers	4	3,600	14,400

Position	No. of Staff	Employment Cost (USD/a)	Total Cost (USD/a)
Subtotal	36	-	213,200
Recovery Plant			
CIC Operators	4	6,450	25,800
Elution and Electrowinning	2	6,450	12,900
Refinery Operators	2	6,450	12,900
Day Labourers	4	3,600	14,400
Subtotal	12	-	66,000
Metallurgical Laboratory and Quality Control			
Metallurgical Technicians	4	6,450	25,800
Sampling and Preparation	8	5,475	43,800
Subtotal	12	-	69,600
Total Process Labour Cost	**66**	**-**	**817,850**

TABLE 21-17 PROCESS MAINTENANCE LABOUR COST ESTIMATE

Position	No. of Staff	Employment Cost (USD/a)	Total Cost (USD/a)
Plant Maintenance			
Maintenance Superintendent	1	200,000	200,000
Maintenance shift foreman	4	7,400	29,600
Planner	2	19,050	38,100
Boilermaker	4	6,450	25,800
Fitter	4	6,450	25,800
Electrician	4	6,450	25,800
Instrument Technicians	4	6,450	25,800
Subtotal	**23**	**-**	**370,900**
Stores Keeper	2	4,500	9,000
Stores Assistant	2	3,600	7,200
Stores Labour	2	2,950	5,900
Subtotal	6	-	22,100
Total Process Maintenance Labour Cost	**29**	**-**	**393,000**

21.2.4.2. REAGENTS AND CONSUMABLES

Reagent consumptions were estimated based on the Feasibility Study testwork results, from experience on similar projects, or using industry standard assumptions. The annual reagent consumptions were calculated by multiplying the consumption per ton processed for a typical year at full production.

The unit cost of reagents and consumables were provided by Chaarat, based on the quotations provided by vendors on a free on board (FOB) basis, with the addition of transport costs for delivery to site, and the application of import duties and VAT where applicable. A summary of the reagent and consumable cost estimation is shown in Table 21-18 and Table 21-19 respectively, and the Chemical and Consumable price list is included in Appendix I.

Table 21-18 Consumables for Equipment

	Operating Hr per Year	Consumption (Excluding VAT and Duty) (USD/h)	Cost (Including VAT and Duty) (USD/a)	Cost (Including VAT and Duty) (USD/t ore)
Vibrating Grizzly	6119	3.66	26,205	0.01
Jaw Crusher	6119	54.48	390,061	0.08
Screens	6119	3.64	75,998	0.02
Cone Crusher	6119	62.40	1,302,061	0.26
Loader	4660	82.50	430,584	0.09
Piping/Drip Emitters (Fixed Allowance)	-	-	377,941	0.08
Total Consumables Cost	**-**	**-**	**2,602,849**	**0.53**

Table 21-19 Reagents

	Unit	Consumption Per Unit	Cost per Unit (USD) Including VAT	Cost including VAT and Duty (USD/a)	Cost including VAT and Duty (USD/t ore)
Leach, Adsorption and Detoxification					
Cyanide - Leaching	kg/t ore	0.60	3.05	9,003,600	1.83
Lime - pH modification	kg/t ore	0.50	0.33	811,800	0.17
Stripping and Goldroom					
Cyanide - Striping	kg/t carbon	8.50	3.05	94,419	0.02
Sodium Hydroxide	kg/t carbon	25.00	0.70	63,531	0.01
Nitric Acid	kg/t carbon	150.00	0.87	477,860	0.10
Activated Carbon	kg/t carbon	6.75	1.88	46,217	0.01
Fluxes	kg/oz Au & Ag	0.15	1.52	22,785	0.00
Diesel - Furnaces	ℓ/mo	20,000.00	0.60	144,000	0.03
Laboratory supplies	USD/a	-	441,580	441,580	0.09
Total Reagents Cost	**-**	**-**	**-**	**11,105,792**	**2.26**

21.2.4.3. Maintenance and Spares

Maintenance and spares were calculated assuming 5% of the cost of the mechanical equipment, which equates to USD886,867/a or USD0.18/t ore (including VAT).

21.2.4.4. Power

Annual power consumption was calculated in kilowatt hour by multiplying the absorbed power requirements for each area by the annual hour of operation. The annual cost of the power was then calculated by applying a unit power rate of USD0.193/kWh, assuming on-site power generation using diesel gensets. The cost of electrical power is based upon the price of fuel at USD0.60/ℓ including VAT. The summary of the power cost estimate is shown in Table 21-20.

Table 21-20 Process Power Cost Estimate

	Absorbed Power (kW)	Operating Hr per Year	Cost Including VAT (USD/a)	Cost Including VAT (USD/t ore)
Crushing and Material Handling	1,890	6119	2,227,848	0.45
Solution Handling	1,382	7,906	2,104,999	0.43
CIC Circuit	72	7,906	109,061	0.02
Acid Wash	10	1,457	2,919	0.00
Elution and Electrowinning	292	2,914	163,938	0.03
Carbon Regeneration	82	7,890	124,889	0.03
Filtration and Goldroom	23	2,914	13,025	0.00
Reagent and Utilities	38	7,906	57,882	0.01
Plant Lighting (2%)	-	-	96,091	0.02
Total Process Power Cost	**-**	**-**	**4,900,653**	**1.00**

22. Economic Analysis

The economic analysis has been accomplished through the construction of a Discounted Cash flow (DCF) model based on the planned production data as set out in the LoM plan, with due regard to appropriate financial model inputs and reasonable assumptions informed by Chaarat and the Competent Persons responsible for the Mineral Resource Estimate and Ore Reserve Report. This analysis is not a complete valuation of the project in terms of any of the international valuation codes. Its purpose is to assess the robustness of the project and confirm the economic viability of the ore reserves.

The DCF model is dependent on the accuracy of the inputs and assumptions underpinning the technical and economic inputs, which are linked to the completeness of the information available at the time of this analysis. The DCF model is a forward-looking exercise and all outputs are hence reliant on assumptions which may be subject to revision as more detailed information becomes available and as circumstances change.

A sensitivity analysis was conducted to analyse the influence to the main inputs (revenue/gold price, operating cost and capital cost) have on the Project merit measures (NPV, IRR, and payback periods).

22.1. Realisation (i.e. Refining and Transport) Costs and Royalties

The gold sales plan incurs a cost of USD0.26/g for refining and a transport cost that is split into a fixed fee of USD1400 per month and a variable fee of 0.03% of the value of the gold transported. Both of these costs exclude VAT.

The refining terms are as presented by Kyrgyzaltyn. The transport costs for gold dore are based upon a proposal from Interpost Ltd/Brinks Inc.

Taxation, including state royalty and a revenue tax as described in Table 22-1 is also prescribed.

Table 22-1 Prescribed Applicable Value Added Tax Rate

Royalty	Non-tax**	Fixed Royalty	Price per troy ounce in USD		Revenue Tax	Total*
			From	To		
5%	2%	7%	0	1 300	1%	8%
5%	2%	7%	1 301	1 400	3%	10%
5%	2%	7%	1 401	1 500	5%	12%
5%	2%	7%	1 501	1 600	7%	14%
5%	2%	7%	1 601	1 700	9%	16%
5%	2%	7%	1 701	1 800	11%	18%
5%	2%	7%	1 801	1 900	13%	20%
5%	2%	7%	1 901	2 000	14%	21%
5%	2%	7%	2 001	2 100	15%	22%
5%	2%	7%	2 101	2 200	16%	23%
5%	2%	7%	2 201	2 300	17%	24%

Royalty	Non-tax**	Fixed Royalty	Price per troy ounce in USD		Revenue Tax	Total*
			From	To		
5%	2%	7%	2 301	2 400	18%	**25%**
5%	2%	7%	2 401	2 500	19%	**26%**
5%	2%	7%	≥2,501		20%	**27%**

7% revenue tax

Accordingly, a royalty, aligned with the selected gold price(USD 1450/tr oz) of 12% is applied to gross revenue.

22.2. TAXATION

Tax calculations are included in the forecast cash flows available for distribution. Baker Tilley Bishkek confirmed that the key taxes applicable to the Project in the Kyrgyz Republic are VAT and import duties, neither of which are recoverable. Baker Tilley Bishkek also provided advice on the application of VAT and import duties on groups of goods and services, depending upon their source, material, type of equipment or service and purpose.

Logiproc considered the advice provided by Baker Tilley Bishkek and applied the applicable VAT and import duty rate to the capital and operating cost estimates.

This approach is considered reasonable for the BFS stage of the Project, but it is acknowledged that there is a degree of uncertainty in the analysis. These tax rates are according to the source and nature of the expenditure and it is understood that there is also a degree of interpretation of the tax rules in practise. In addition, the level of definition of the items in the BFS is not at a level that would allow the accurate application of the tax rules. Although preliminary vendors have been supplied and selected with the estimate based on quotes from these companies the final supplier of equipment, materials and services has also not yet been finalised. As such the country of origin could change, which would impact the VAT and import duty rate applicable.

Baker Tilley Bishkek have also highlighted that there is a potential for other direct taxes to be applied to the project, including; land tax on land rented or purchased, withholding on foreign companies and reverse VAT on non-resident companies.

22.3. PRODUCTION AND SALES

The life-of-project average material tonnages, grade and gold and silver production are shown in Table 22-2.

TABLE 22-2 FORECAST MINE PRODUCTION

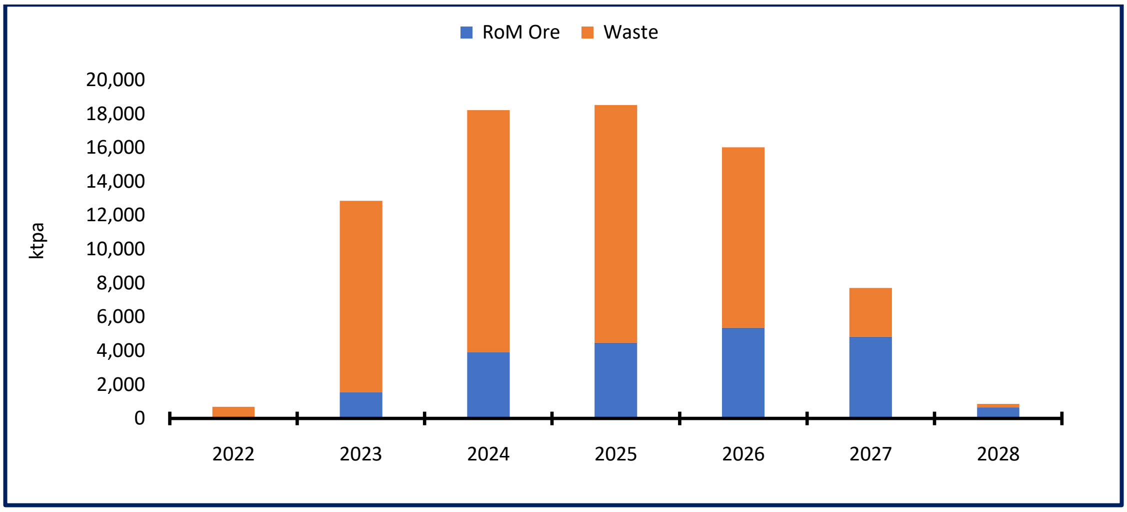
Description	Unit	Value
Ore Mined	kt	20,859.25
Waste Mined	kt	54,048.18
Recovered Gold	**('000 oz)**	**419.52**
Recovered Silver	**('000 oz)**	**536.14**

The revenue forecast has been based on the production forecast and a gold price of USD1,450/tr oz. This is based on a view of the likely gold price in real money terms over the

LoM. This price assumption is conservative when compared to the 24 March 2021 Spot Price of USD1,734.19/tr oz.

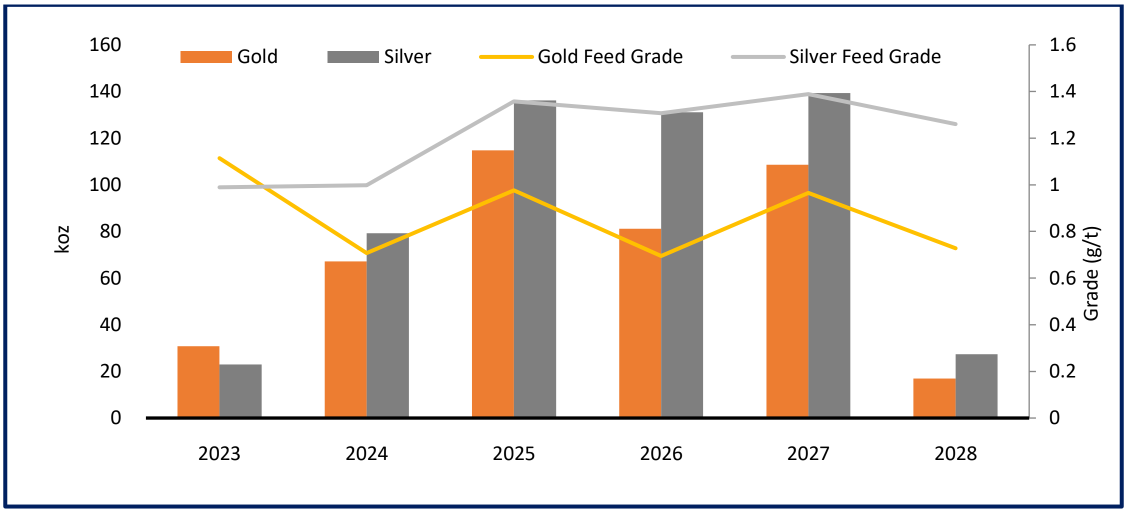
Although relatively insignificant, a silver price of USD17.50/tr oz was also included for the anticipated by-product. The forecast mine production over the LoM is illustrated in Figure 22-1.

FIGURE 22-1 ORE AND WASTE MINED



The forecast gold and silver production and average grades over the LoM are shown in Figure 22-2.

FIGURE 22-2 COMMODITY RECOVERIES AND ASSOCIATED GRADE



22.4. OPERATING, CAPITAL AND CLOSURE COSTS

The operating, capital and closure costs as described in this report have been used for the economic analysis. The average unit operating cost derived directly from the LoM is USD13.59/t ore (in real terms). The average capital expenditure over the LoM is USD6.21/t ore (in real terms), including a 10% contingency. This includes a provision for mine closure as estimated in (Table 22-3).

TABLE 22-3 CLOSURE COST ESTIMATES

Description	Total (USD 000's)
End mining, final leaching	2,295
Flushing, prod gold	1,368
Flushing, no gold, H_2O_2	813
Rehab, H_2O_2	1,559
Rehab only	0,476
Mine Closure Provision	**6,511**

The LoM provides for backfill volumes and costs (using contract mining rates) over the LoM, leaving a final void open in the reasonable expectation that this void will, in due course, be required to accommodate waste from future exploitation of the considerable Mineral Resources beyond the current open pit outline and LoM Plan.

22.5. WORKING CAPITAL

Movements in working capital (stores, debtors and creditors) have been provided for. This requirement has been estimated based on a current understanding of the relationship between Chaarat and the Kyrgystan Government.

The working capital, which assumes 15 days for debtors, 44 days for creditors and 4 days for inventory, will fluctuates from month to month. It will be recovered at the end of the LoM.

22.6. ECONOMIC MODEL AND DCF METHODOLOGY

Metric units are used throughout this economic analysis and unless otherwise stated, monetary values are stated in United States Dollars (USD).

Ungeared cash flows have been forecast and discounted back to a Net Present Value (NPV) using a range of real discount rates (0% to 10%). The pre-mining construction period is 13 months.

The financial years in the DCF model start on February 2021 and the cash flow continues up to 2028, to match the current LoM Plan and the Mineral Resources (20.9Mt). Each year's cash flow is deemed to have occurred in the middle of the period.

Calculations have been done in real February 2021 money terms. Real terms modelling permits meaningful comparison of like for like revenue and cost results in particular points in time over the LoM.

The analysis does not consider any balance sheet items and all prior unredeemed capital has been excluded as a sunk cost.

Detailed sensitivity analysis is presented to illustrate the valuation result variation in the event of variance from these assumptions. The economic model developed for the project is shown in Table 22-4.

TABLE 22-4 TULKULBASH PROJECT - BFS DISCOUNTED CASH FLOW (ANNUAL SUMMARY)

Description	Rem	Unit	Tot / Avg	2021	2022	2023	2024	2025	2026	2027	2028	2029
Production												
Waste Mined		kt	*54 048*	-	*658*	*11 307*	*14 292*	*14 060*	*10 653*	*2 860*	*217*	-
Ore Mined		kt	*20 859*	-	*38*	*1 560*	*3 933*	*4 469*	*5 371*	*4 844*	*644*	-
Total Mined		**kt**	**74 907**	**-**	**696**	**12 868**	**18 225**	**18 529**	**16 024**	**7 704**	**861**	**-**
Strip Ratio		*t : t*	*2.6*	-	*17.3*	*7.2*	*3.6*	*3.1*	*2.0*	*0.6*	*0.3*	-
Ore to Stockpile		kt	*20 859*	-	-	*1 138*	*3 893*	*4 920*	*4 920*	*4 920*	*1 068*	-
Gold Feed Grade		g/t Au	*0.85*	-	-	*1.11*	*0.71*	*0.98*	*0.69*	*0.96*	*0.73*	-
Silver Feed Grade		g/t Ag	*1.26*	-	-	*0.99*	*1.00*	*1.36*	*1.31*	*1.39*	*1.26*	-
Contained Gold		koz	*571*	-	-	*41*	*88*	*154*	*110*	*153*	*25*	-
Contained Silver		koz	*846*	-	-	*36*	*125*	*215*	*207*	*220*	*43*	-
Gold Recovery		%	73.46 *	-	-	75.55	75.85	74.39	73.90	71.17	67.86	-
Silver Recovery		*%*	63.40	-	-	63.40	63.40	63.40	63.40	63.40	63.40	-
Payable Gold		*koz*	*418*	-	-	*28*	*65*	*117*	*79*	*108*	*21*	-
Payable Silver		koz	*446*	-	-	*13*	*64*	*112*	*110*	*119*	*28*	-
Proportion of Gold		%	*44*	-	-	*67*	*51*	*51*	*42*	*48*	*43*	-
Proportion of Silver		%	*56*	-	-	*33*	*49*	*49*	*58*	*52*	*57*	-
Revenue												
Gold Revenue		USD 000's	*605 397*	-	-	*40 033*	*94 542*	*169 174*	*114 241*	*156 748*	*30 659*	-
Silver Revenue		USD 000's	*7 797*	-	-	*234*	*1 117*	*1 959*	*1 921*	*2 077*	*488*	-
Gross Revenue		**USD 000's**	**613 194**	**-**	**-**	**40 267**	**95 660**	**171 132**	**116 162**	**158 825**	**31 148**	**-**
Refining Cost		USD 000's	*3 782*	-	-	*250*	*591*	*1 057*	*714*	*979*	*192*	-
Transport Cost		USD 000's	*319*	-	-	*32*	*51*	*76*	*58*	*72*	*29*	-

Description	Rem	Unit	Tot / Avg	2021	2022	2023	2024	2025	2026	2027	2028	2029
Royalty and Tax	12% of Gross Rev.	USD 000's	*73 583*	-	-	*4 832*	*11 479*	*20 536*	*13 939*	*19 059*	*3 738*	-
Net Revenue		**USD 000's**	**535 510**	**-**	**-**	**35 153**	**83 539**	**149 463**	**101 452**	**138 715**	**27 189**	**-**
Capital Expenditure												
Mining		USD 000's	*21 996*	*2 824*	*6 645*	*12 527*	-	-	-	-	-	-
Infrastructure		USD 000's	*4 236*	*2 657*	*1 578*	-	-	-	-	-	-	-
Process Plant		USD 000's	*67 091*	*8 021*	*38 711*	*13 077*	*3 607*	*1 738*	*1 356*	*581*	-	-
Owners Cost		USD 000's	*25 424*	*2 740*	*7 353*	*8 820*	-	-	-	-	*6 511*	-
Contingency	*(10% of Capital Expenditure)*	*USD 000's*	*11 875*	*1 624*	*5 429*	*3 442*	*361*	*174*	*136*	*58*	*651*	-
Working Capital		*USD 000's*	-	-	-	*(9)*	*(1 569)*	*2 050*	*(1 077)*	*3 313*	*(2 017)*	*(691)*
Capital Expenditure	*incl. conting.*	**USD 000's**	**130 621**	**17 866**	**59 717**	**37 857**	**2 398**	**3 962**	**414**	**3 952**	**5 145**	**(691)**
Operating Cost												
Owner's Costs		USD 000's	*32 456*	-	-	*2 373*	*7 155*	*7 098*	*7 121*	*7 059*	*1 650*	-
Mining Cost		USD 000's	*139 301*	-	-	*10 138*	*37 064*	*38 849*	*34 084*	*17 140*	*2 027*	-
Processing Cost		USD 000's	*98 579*	-	-	*3 345*	*17 970*	*23 661*	*23 691*	*24 525*	*5 387*	-
Operating Cost		**USD 000's**	**270 336**	**-**	**-**	**15 856**	**62 189**	**69 607**	**64 895**	**48 724**	**9 064**	**-**
Operating & Capital Costs		**USD 000's**	**400 957**	**17 866**	**59 717**	**53 713**	**64 587**	**73 569**	**65 310**	**52 677**	**14 210**	**(691)**
Project Cash Flow After Tax		**USD 000's**	**134 554**	**(17 866)**	**(59 717)**	**(18 561)**	**18 952**	**75 894**	**36 142**	**86 038**	**12 980**	**691**
Cumulative		**USD 000's**	**134 554**	**(17 866)**	**(77 583)**	**(96 143)**	**(77 191)**	**(1 297)**	**34 845**	**120 883**	**133 862**	**134 554**

Note: Rounding off error from the block model to the production schedule

22.7. DISCOUNTED CASH FLOW(DCF) AND SUMMARY OF RESULTS

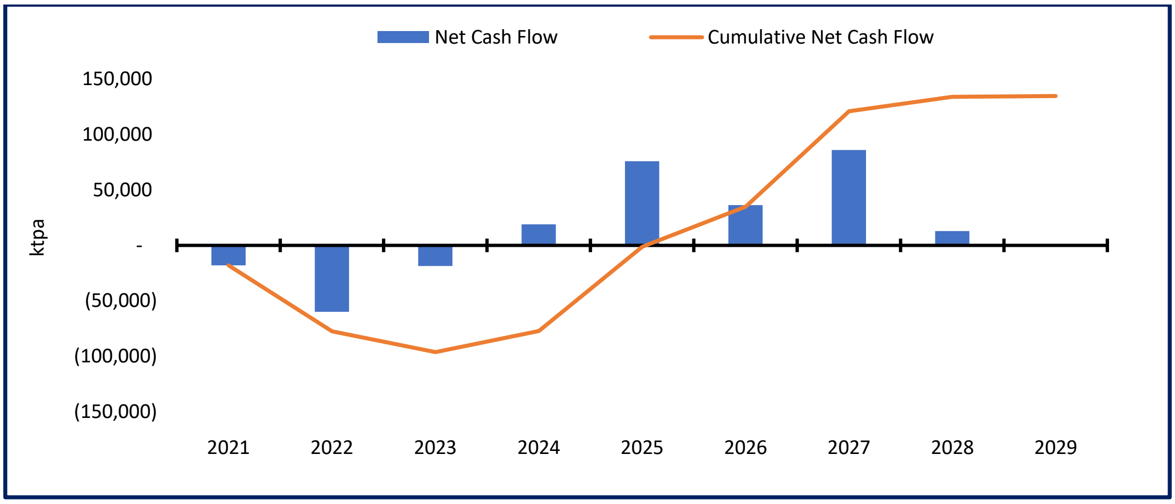
Operating costs for contract mining, owner mining, processing and G&A were deducted from the net revenue to derive the operating cash flows.

The initial capital, working capital, and closure costs were then also deducted from the operating cash flow to determine the net cash flow.

Initial capital expenditures include costs accumulated prior to the first production of gold.

The undiscounted net cash flow (NCF) and cumulative net cash flow (CNCF) that result from the Project's post tax production forecast, operating cost forecast and capital expenditure forecast are illustrated in Figure 22-3.

FIGURE 22-3 FORECAST CASH FLOWS



The MCNCF or the peak funding requirement rises to USD96.1M, in 2023. The financial model was established on a 100% equity basis, excluding debt financing and loan interest charges. A summary of the financial results for the DCF are presented in Table 22-5.

TABLE 22-5 SUMMARY OF DCF ANALYSIS RESULTS

Parameter	Units	Values
Gold Price (LoM Weighted Average)	USD/oz	1,450
Gold Sold	koz	418
Silver Sold	koz	446
Net Revenue	USD 000's	535,510
Operating Costs	USD 000's	270,336
Operating Profit	USD 000's	265,174
Capital	USD 000's	112,235
Closure and Reclamation	USD 000's	6,511
Contingency @ 10%	USD 000's	11,875
Total Capital Costs (incl. 10% cont.)	USD 000's	130,621
Net Cash Flow	USD 000's	134,554

Parameter	Units	Values
NPV at 0% discount rate	USD 000's	125
NPV at 5% discount rate	USD 000's	85
NPV at 10% discount rate	USD 000's	51
IRR	%	25%
Payback at 5% discount rate	years	~5
MCNCF at 5% discount rate	USD'000	(96 143)

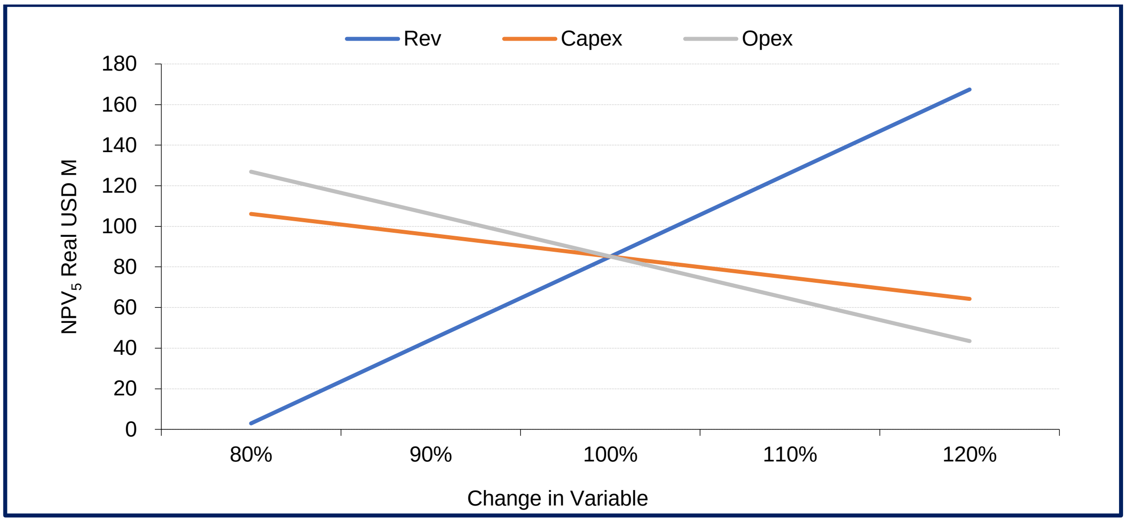
22.8. Sensitivity Analysis

Sound Mining investigated the sensitivity of NPV, IRR, and payback period to the key project variables. Using the post-tax base case as a reference, each of the key variables was changed between –15% and +15%, while maintaining the other variables constant. The following key variables were investigated:

- Revenue (gold price);
- Operating costs; and
- Capital costs.

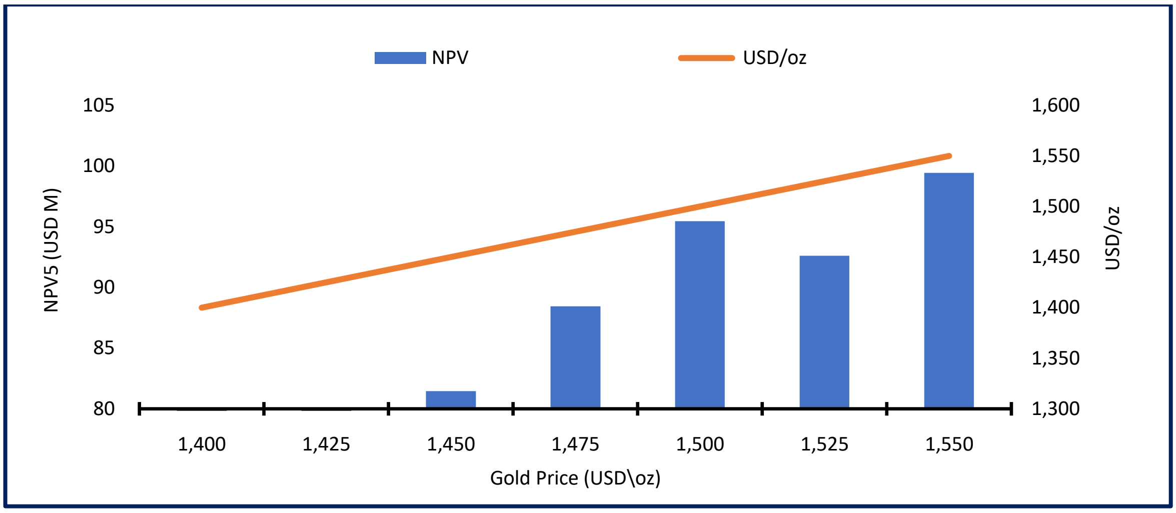
The Project's post-tax NPV, calculated at a 5% discount rate, is most sensitive to revenue followed by operating costs and capital costs, as shown in Figure 22-4.

Figure 22-4 NPV Sensitivities



Increased revenue is clearly the biggest driver of value but careful analysis of this factor needs to be undertaken since short term marginal increases in the gold price at various threshold limits can actually reduce overall revenue at steady production rates.

This is illustrated in a sensitivity of the value of the project to the gold price (Figure 22-5). The step change in the royalty calculation with an increase in the gold price leads to a stepped revenue sensitivity.



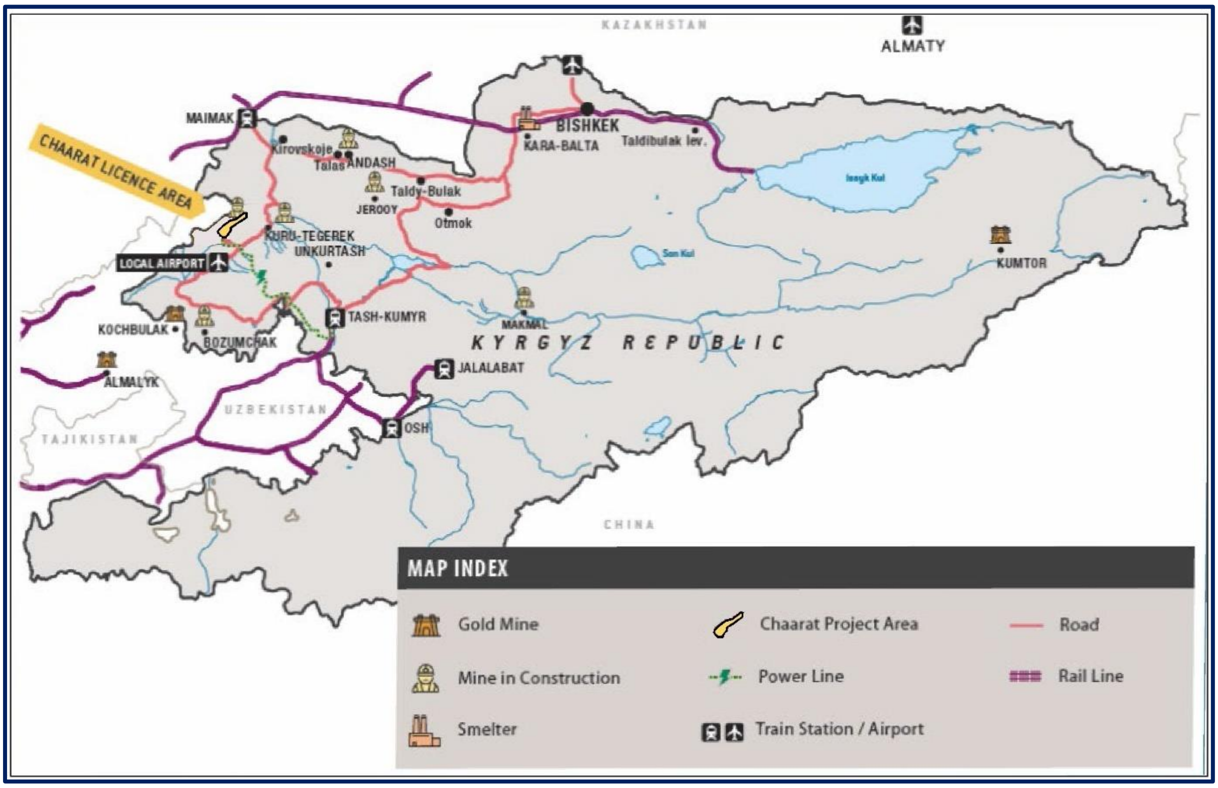
The consequence is a clear loss of value as the gold price moves beyond USD1,500/tr oz because the royalty then increases from 12% to 14%.

23. ADJACENT PROPERTIES

The Tulkubash deposit is located in the Middle Tyan-Shan Geological region of Kyrgyzstan, in which a small but increasing number of mines are now in production or under development. The Kyrgyzstan map of Gold mines is illustrated in Figure 23-1 below.

The nearest of these to the Tulkubash site is the Kuru-Tegerek deposit operated by Kichi-Charaat.

FIGURE 23-1 KYRGYZSTAN MINE MAP



24. Other Relevant Data and Information

24.1. Project Execution Plan

Chaarat has prepared a Project Execution Plan (PEP) to manage and control the delivery of the project in a safe and cost-effective manner.

The Project Execution Plan (PEP) establishes Chaarat's execution philosophy, defines the organization, methods and project management elements necessary to manage execution of the Tulkubash Project to accomplish effective engineering design, procurement, construction and commissioning of the mine, ore processing facilities, associated infrastructure and engagement of external services.

In formulating the PEP, priority was given to the use of technical and commercial resources that are familiar with working in Kyrgyzstan without compromising environmental considerations, social sensitivities, safety, quality, cost or schedule. The plan was compiled on the basis that an Integrated Project Management Team (IPMT) is providing a full scope of services from project approval through to final turnover.

The plan has been based on 5 basic phases: Project Initiation, Project Definition and Planning, Project Execution, Project Performance and Control, Project Closure.

The goals of the PEP are to: -

- Provide a brief description of the proposed approach to project execution;
- Anticipate and reduce or offset scheduling issues;
- Provide a description of the requirements for design, construction and commissioning using Integrated Project Management Team (IPMT) approach;
- Avoid or limit capital cost overruns;
- Anticipate issues related to construction during the design process;
- Provide a plan for overall project risk management;
- Outline how vendors and contractors will deliver quality products and services;
- Address delivery times for long-lead items on the critical path;
- Identify effective methods for forecasting, reporting, and controlling costs;
- Develop a detailed plan for project pre-commissioning, commissioning, and handover;
- Outline project safety management;
- Address QA/QC requirements and provide a brief project QA/QC plan; and
- Develop the following PEP-related documents:
    - Program schedule; and
    - Tulkubash Project operating manual.

24.1.1. Objectives and Expectations

Chaarat's strategy to unlock the long-term value of the Project may be summarised as follows:

- Stage 1 – Extend the Tulkubash heap leachable oxide resource base and develop low capital-intensive heap leach production;
- Stage 2 – Continue Tulkubash oxide exploration to expand the heap leachable oxide resource;
- Stage 3 – Complete a detailed Feasibility Study for the refractory Kyzyltash sulphide ore body; and
- Stage 4 – Develop in parallel a sulphide processing facility.

24.1.2. Project Constraints

24.1.2.1. Statutory and Community Requirements

Project execution will be subject to Kyrgyz regulatory frameworks, and the associated permit and Licence requirements. The application and approval process includes the submission of formal documents such as a TEO (feasibility study), Proekt (design report), OVOS (environmental impact assessment), in addition to the in-country 'adaptation and legalization' of foreign designs. In this context, applicable standards include GOST Russian Standards, Russian Construction Codes and Regulations (SNiP's) and the Electrical Installation Design Code (PUE).

24.1.2.2. Project and Site-Specific Constraints

The following constraints apply to the Project:

- All power will be self-generated;
- All consumables, personnel, and spares will be transported over mountainous terrain to site;
- Extreme winter conditions prevail for approximately three months of the year. Between November and April, it is too cold to perform some of the construction works such as earthwork compaction. The window for Heap Leach liner installation is even more limited to approximately May-September;
- Site access for large and heavy loads is limited. Transport of containers larger than 20 ft will not be possible until the Kumbel Pass to Site access road is upgraded; and
- There is limited logistic and infrastructure support in the region.

24.1.2.3. Technology

Challenging weather and logistical conditions apply the following technology constraints to the project:

- Operating mobile equipment in winter may require special provision for the heating of lubricants, fuel, and hydraulic oil and for winter specifications.
- Procurement and design will be informed by a requirement for:
  - Low to medium technology plant and equipment
  - Procurement of equipment from regional sources where possible.

- Ongoing support from critical equipment suppliers - Standardization of equipment and provision of 'rotable' equipment;
- Site repair facilities
- Specification of lower than benchmark availability targets.

Note that the adoption of a low to medium technology strategy is also consistent with Chaarat's goal to source personnel as far as possible from local areas.

24.1.2.4. Language and Culture

The Russian, Kyrgyz, Turkish, and English languages will be used on the Project. Table 24-1 provides a summary of the languages to be used in each phase by each stakeholder.

Table 24-1 Language Utilization

Description	IPMT	EP Contractors	Principal Engineer	Mining Contractor
Internal	Kyrgyz, Russian, English, Turkish	English, Turkish	English	Turkish, English
Reporting Execution	Russian, English	English	English	Russian, English
Operation	Kyrgyz, Russian, English, Turkish	English, Turkish	English	English, Turkish

The Project will require the availability of multilingual staff and translators during execution.

24.1.3. Organization and Responsibilities

24.1.3.1. Overview

As far as possible, Chaarat will employ Kyrgyz nationals, especially residents of Chatkal province, to construct and operate the Project. External Specialist skills will only be engaged where suitable skills are not available in-country and will be restricted to fixed contracts.

The Project will be managed from the Chaarat's headquarters in Bishkek.

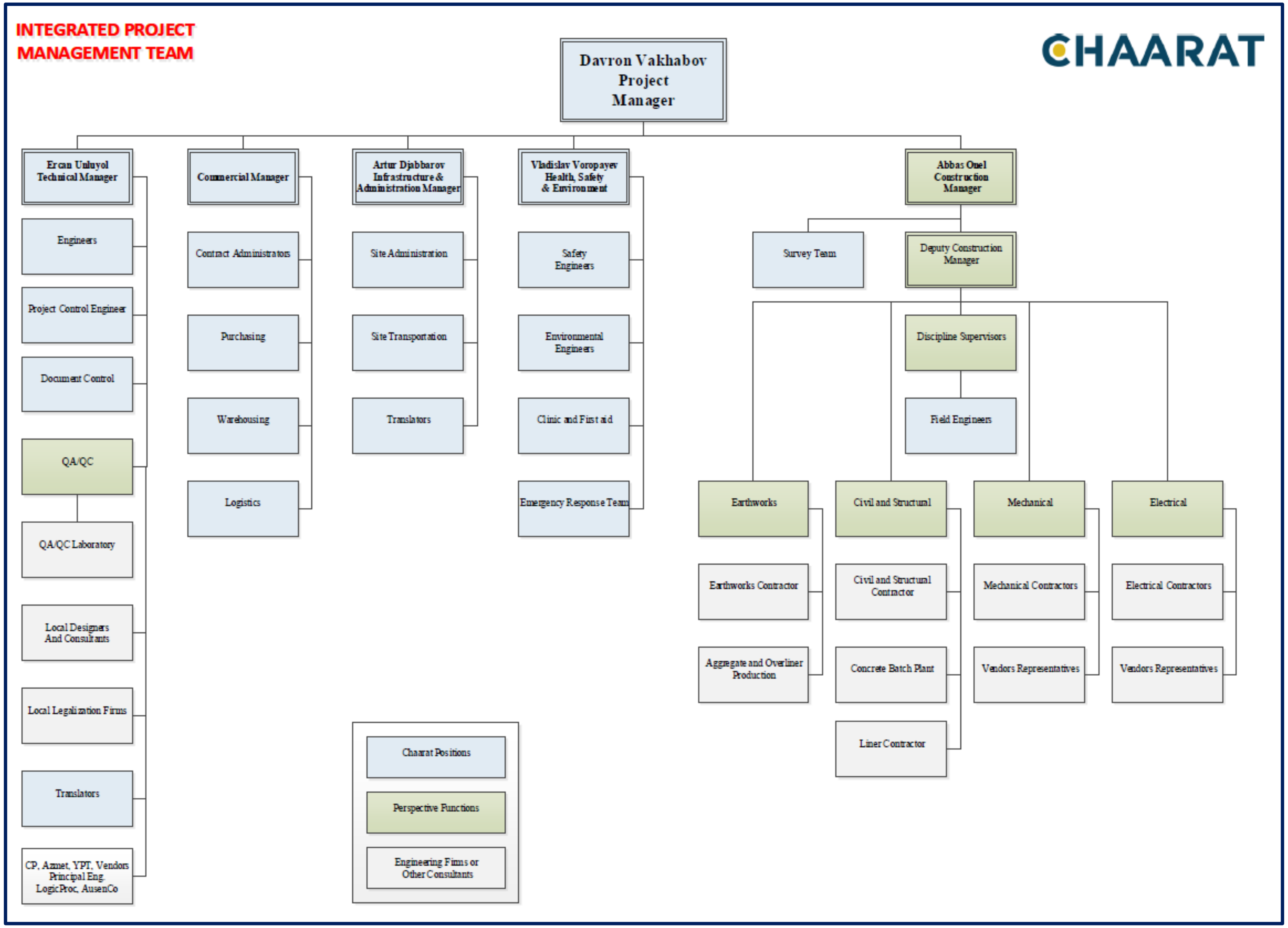
Three months before pre-commissioning of the Process Facilities, Chaarat will complete the recruitment of the Operations team to allow for a period of detailed training of the Operators.

Pre-commissioning will be managed by the IPMT with the support of the Operations team. Hot commissioning and plant start-up will be completed by the Operations team with the support of the Pre-commissioning team.

24.1.3.2. Project Organization Structure

Figure 24-1 shows the planned organizational structure for Chaarat.

FIGURE 24-1 PROJECT ORGANIZATIONAL STRUCTURE



24.1.3.3. ROLES AND RESPONSIBILITIES

Integrated Project Management Team (IPMT) is responsible for overall Project Management and coordination between responsible parties. LogiProc will act as a Principal Engineer and report to the IPMT Engineering Manager.

The main facilities in the Process area such as ADR, crushing circuit, power supply and fuel farm, and HLF will be designed by EP (Engineer and Procure) contractors who are specialists in their respective fields.

LogiProc will coordinate the overall engineering effort between the selected specialist EP (Engineer and Procure) contractors. LogiProc is also responsible for "filling the gaps" between the respective specialist engineers' scopes thereby integrating the EP contractors' final deliverables into a unified design.

The roles and responsibilities are summarized in Figure 24-2.

TABLE 24-2 ROLES AND RESPONSIBILITIES

Description	IPMT	LogiProc	Ausenco	Azmet	YPT	Çiftay	Ken-Too	Other Local Designers	Other Specialty Contractors
Project Management	A/R	-	-	-	-	-	-	-	-
Project Control	A/R	-	-	-	-	-	-	-	-
Contract Management and Administration	A/R	-	-	-	-	-	-	-	-
Quality Management	A/R	-	-	-	-	-	-	-	-
Commissioning and Handover	A	R	R	R	R	R	R	R	R
Basic Engineering	C/I	A/R	-	-	-	-	-	-	-
Detailed Design Mine	C/I	-	-	-	-	-	A/R	-	-
Detailed Engineering	C/I	A/R	A/R	A/R	A/R	A/R	A/R	A/R	A/R
In-Country Legalization of Designs	C/I	C	C	C	C	C	A/R	A/R	C
Construction of Shift Camp	A	-	-	-	-	R	C	-	-
Construction of Explosives and AN Storages	A	-	-	-	-	-	-	C	R
Earthworks	A	C	C	C	C	R	C	C	-
Construction of HLF	A	C	C	-	-	R	C		R
Construction of Crushing Circuit	A	C	-	-	R	-	C	-	-
Construction of ADR	A	C	-	R	-	-	C	-	R
Construction of Processing Infrastructure	A	C	-	-	-	-	C	-	R

Notes: R = responsible; A = accountable; C = consult; I = inform

24.1.4. Schedule

Chaarat has prepared a Project Master Schedule (PMS). The PMS details a 32-month period from the start of HLF earthworks in June 2021 through to the First Gold Pour in August 2023. Construction is planned to resume in June 2021, and continue through the winter season with minimal disruption. The schedule was developed in a logical sequence by detailing out those activities which were to take place in a consecutive fashion. Constraints were used only where appropriate. Furthermore, the PMS utilizes an in-house drafted WBS, which was established based upon the project construction sequence. It covers the following five Major Areas as shown in Table 24-3.

Table 24-3 Work Breakdown Structure for PMS

Major Area	Sub-Area Description
M1	Mining
I2	Infrastructure
P3	Process Plant
O4	Owner's Cost
C5	Contingency

These areas were then subdivided and developed further into Sub-Areas to integrate all engineering, procurement and constructions activities.

Chaarat's objective is to design and construct all components of the Project to the first gold pour in Q3 2023.

The execution strategy will have the following features:

- Fast-track approach with design, construction, and permitting in parallel;
- Single mining contractor;
- Single major earthworks contractor;
- Simple, modular where possible design, with off-site construction assembly and checking; and
- Oversight of specialist vendors and contractors during construction, installation and commissioning of equipment.

24.1.4.1. Key Schedule Parameters

The PEP is designed to meet key milestones set out by Chaarat and shown in Table 24-4.

Table 24-4 Tulkubash Gold Project Milestones and Dates

Milestone	Date
Pamir Remobilization	15th May 2021
Project Full Financing	1st June 2021
Resume of HLF Bulk Earthworks	17th June 2021
Approval to Proceed with ADR Equipment Manufacturing	3rd August 2021
Approval to Proceed with Crushing Equipment Manufacturing - YPT	1st September 2021

Milestone	Date
Camp Construction Complete - Phase 1	8th October 2021
Approval to Proceed with Crushing Equipment Manufacturing - Crushers	5th November 2021
Camp Construction Complete - Phase 2 Kitchen and Dining Hall	27th November 2021
Liner Order	30th December 2021
Camp Construction Complete - Phase 2 Remaining Buildings	30th January 2022
Start of Pit Road Construction	1st April 2022
Site Batch Plant Installation Completed	26th April 2022
Start of Pre-stripping	30th June 2022
Haul Road Construction Complete	13th September 2022
Power Generation Facility Commissioned	30th December 2022
First Ore Stacking to Heap Leach	18th May 2023
Irrigation Start	24th June 2023
First Gold Dore Poured	24th August 2023

The PEP is subject to site conditions, commercial and logistical constraints as set out in Table 24-5.

Table 24-5 Project Execution Plant Constraints

Constraints	Value
Limited Access due to Winter Conditions	December – April
Avalanche Risk Period	January – April
Ambient Conditions Suitable for Conventional Concrete Curing	May – October
Ambient Conditions Suitable for Installation of Lining Systems	May – October
Approximate Time between Start of Irrigation and First Pregnant Solution	30 days

Approximate durations of some of the key activities are provided in Table 24-6 below.

Table 24-6 High-level Activities and Durations

Activity	Durations (mo)	Dependency
Plant and infrastructure design completion	3	Project financing
Construction Permit	4	Design completed and legalized in-country.
Contractors Mobilization	3	Availability of accommodation facilities
Earthworks	10	Project financing
Process Plant Construction (ADR, Crushing Circuit, Power plant, Fuel Farm and infrastructure)	15	Detailed design by area. Subarea completion of earthworks. Procurement of long-lead items.
Mine Pre-strip	12	Mobilization of mining contractor.
Heap Leach Facility Construction	10	Mobilization of earthworks contractor. Procurement of long-lead materials. Supply of ore from mine for overliner.

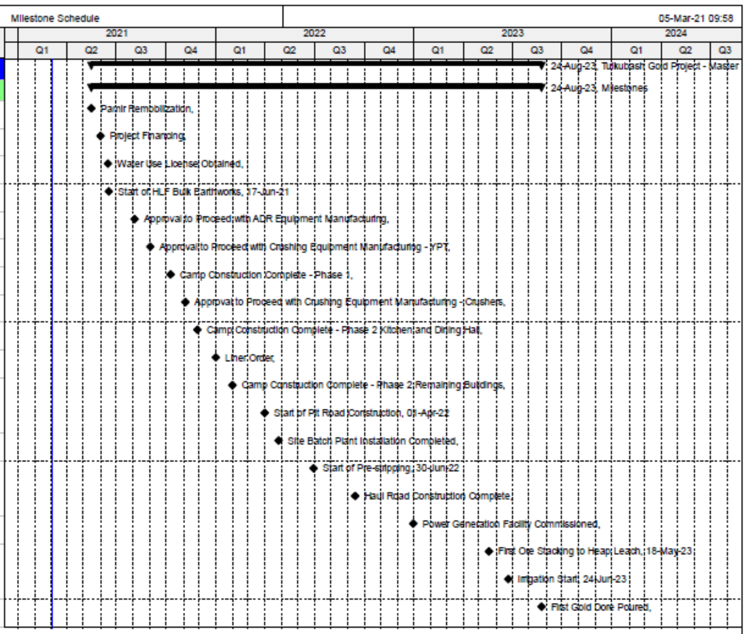
24.1.4.2. SUMMARY SCHEDULE

Key programme dates and activities are shown in Figure 24-2. The estimated project completion date is end of Q3 2023.

FIGURE 24-2 TULKUBASH PROJECT MILESTONE SCHEDULE

Tulkubash Gold Project - Master Schedule - Data Date 20210305

Activity ID	Activity Name	Activity % Complete	Start	Finish
Tulkubash Gold Project - Master Schedule - Data Date 20210305			15-May-21	24-Aug-23
Milestones			15-May-21	24-Aug-23
TULK14640	Pamir Remobilization	0%		15-May-21*
TULK10200	Project Financing	0%		01-Jun-21*
TULK13280	Water Use License Obtained	0%		14-Jun-21
TULK2040	Start of HLF Bulk Earthworks	0%	17-Jun-21	
TULK1930	Approval to Proceed with ADR Equipment Manufacturing	0%		03-Aug-21
TULK2080	Approval to Proceed with Crushing Equipment Manufacturing - YPT	0%		01-Sep-21
TULK14410	Camp Construction Complete - Phase 1	0%		08-Oct-21
TULK13270	Approval to Proceed with Crushing Equipment Manufacturing - Crushers	0%		05-Nov-21
TULK14600	Camp Construction Complete - Phase 2 Kitchen and Dining Hall	0%		27-Nov-21
TULK1950	Liner Order	0%		30-Dec-21
TULK2010	Camp Construction Complete - Phase 2 Remaining Buildings	0%		30-Jan-22
TULK14400	Start of Pit Road Construction	0%	01-Apr-22	
TULK13300	Site Batch Plant Installation Completed	0%		26-Apr-22
TULK2060	Start of Pre-stripping	0%	30-Jun-22	
TULK2090	Haul Road Construction Complete	0%		13-Sep-22
TULK14210	Power Generation Facility Commissioned	0%		30-Dec-22
TULK9400	First Ore Stacking to Heap Leach	0%	18-May-23	
TULK9390	Irrigation Start	0%	24-Jun-23	
TULK9000	First Gold Dore Poured	0%		24-Aug-23



CHAARAT | Actual Level of Effort | Remaining Work | Milestone | Page 1 of 1 | Date: 05-Mar-21 | Revision: Milestone Schedule | Checked: BT | Approved: EU
Actual Work | Critical Remaining Work | summary

24.1.5. PROJECT EXECUTION

24.1.5.1. STRATEGY

Construction will be undertaken by means of a number of contracts, including:

- The main site Contract, which is to develop the mine, complete major earthworks, construct and operate the camp, and construct and stack the HLF;
- EP Contracts to equipment designers and manufacturers – in particular, Heap Leach, the crushing circuit, ADR facilities, and the power station (incl fuel farm);
- Specialist geomembrane lining contractor;
- Structural, mechanical, platework, and piping;

- Electrical, instrumentation, and control; and
- Site services and catering.

The construction schedule will include all these activities.

Chaarat's IPMT will be responsible for the project and construction management using in-house resources, including some procurement. Chaarat will manage this process with input from the selected engineering companies, including LogiProc, Azmet, YPT, Ausenco, and Ken-Too. A flat organisation structure will favour the rapid decision-making required to 'fast-track' this project.

The IPMT will prepare a project procedures manual that will contain the administrative and related procedures required to execute the Project. The roles and authorities of the IPMT team members will also be contained in the procedure's manual.

24.1.5.2. INTERFACES

Chaarat is mindful of the internal and external interfaces arising in the course of Project execution. The project execution strategy and associated master documents will need to address the expectations and requirements of multiple internal and external stakeholders and participants. Key stakeholders include not only local and central Kyrgyz authorities, local communities, shareholders, contractors and the Chaarat Board of Directors, but also suppliers, and consultants. A reliable, bilingual (English/Russian) document management system will be prepared.

24.1.6. PROJECT CONTROLS

24.1.6.1. COST CONTROL

Project controls will be managed from the Bishkek office using 'Prism G2' software.

To control costs effectively, the following principles will be applied:

- Cost information, including budget, commitments, and expenditure will be updated on a regular basis;
- All changes to the budget will be approved by appropriate, authorized personnel and formally recorded;
- All information entered into the cost control system will be accurate and auditable; and
- Expenditure and actual payments must be reconciled at least monthly.

The Project Manager is the sole approver of budget reallocations, contingency use, and scope changes.

The Project cost controller is responsible for making changes to budget, commitments, expenditure, or contingency, or to add a cost element if scope changes necessitates such.

A cost element budget, in the control budget or contingency budget, may only be increased or decreased if the appropriate documentation has been completed and approved. Three types of notification and approval situations may be applicable, though these may be executed using the same form. The notification and approval situations are:

- Contingency use notice;

- Budget reallocation notice; and
- Project change notice.

At the start of a project a control budget is established against which project expenditure will be managed. This budget is the best estimate of the future cash flow of a project at the time of establishing the baseline and assumes a certain exchange rate for the base currency.

Currency escalation effects are not included in the control budget and the risk of these changes are managed external to the cost controller. It is the responsibility of the Financial cost controller to isolate and report foreign exchange effects. This will enable Chaarat to manage risk by:

- Taking out forward cover to the extent practicable;
- Making provision in a reserve fund; and
- If applicable, cover currency movements internally.

Currency escalation is excluded from the contingency budget and additional funds to cover this effect require a project change notice.

The baseline budget of the Project is based on a fixed exchange rate. Actual expenditure is subject to continuous change and comparing the expense to the baseline budget would not provide a fair reflection of the Project status.

24.1.6.2. SCHEDULE

The Project schedule will be managed using a Primavera P6 master programme with inputs from more detailed fabrication, transport, construction, installation and commissioning schedules from contractors, suppliers, and site management.

The master programme does not fully detail all construction and installation activities, as the Site Construction Team manages detailed programme for these. Two important outcomes of the Master Programme are:

- Prediction of the Project completion date in a substantiated manner; and
- Early warning of critical path slippage.

The Project Planner is an important role in the Project. The Project Planner will update the Master Schedule weekly and issue a report to the project manager with basic printouts and notes. A more comprehensive monthly report will be submitted by the project planner to the project manager.

The project planner communicates early signals of serious deviations to the schedule and progress lapses. The programme schedule will be formulated using the WBS and will be used by the cost controller to assist in early and long-term cash flow projections.

24.1.6.3. ENGINEERING PROJECT CONTROLS

All engineering deliverables shall be peer reviewed and approved by the Engineering Manager. The approved engineering drawings and specifications will be issued for construction and managed through the project document control system by the Technical Department. Any change to approved drawings or documents will require the following process:

- Description of the scope, cause or motivation, impact, cost and time impact of the change;
- If the change can be managed within the engineering budget, the Engineering Manager will manage the approval;
- If the change requires budget or schedule change, it will be escalated to the project manager; and
- An approved design change request will become the basis for a budget and schedule change request.

24.1.7. PROCUREMENT

A substantial proportion of parts and equipment required for the project will be sourced either via the EP Vendors or by the installation contractors themselves. Very high-cost individual items (such as the crushers) will be procured by Chaarat's Commercial Department, plus any items not covered by the EP Vendors and Contractors.

The standard approach for this Project will be selected tendering, whereby a list of suppliers will be invited to tender; suitable tender documentation will be issued, and the tenders returned will be analysed and compared.

Tenders for an item will be called for simultaneously and returns required before or on a specific date, time, and place, whether electronically or physically. Electronic (e-mailed) tenders will be acceptable. The personnel involved will be responsible for maintaining confidentiality and for ensuring that all tenders are taken into consideration.

The Commercial manager will maintain a procurement list, which will serve as a reference throughout procurement process for the Project.

The Commercial Manager assigns a contracts administrator to each contract or purchase order. When the Commercial Manager has not assigned a contract administrator, the Commercial Manager will by default be the contracts administrator. Contract administrators will manage the orderly execution and eventual closure of orders and contracts in a consistent manner.

Where the skills exist locally, priority will be placed on hiring local contractors and contractors from local small and medium-sized enterprises for subcontract work.

24.1.8. SITE MANAGEMENT

24.1.8.1. GENERAL

The Chaarat Construction Manager will have direct control of the Project site and will report directly to Project Manager.

24.1.8.2. HEALTH, SAFETY, SECURITY AND ENVIRONMENT

Health, safety, security and environment (HSSE) programmes and initiatives are essential to project success. A fully integrated programme has been prepared and will be implemented with the objective of achieving a "zero-harm" goal for employees, contractors and visitors working on the Project, as well as protecting the environment.

The Project will incorporate HSSE criteria in the design, constructability, and operability of each facility and major area with the requirement that all personnel must have completed site safety and environmental orientation. Tools will be implemented for performance tracking and accountability, including procedures for incident management.

Procedures will be developed with Construction contractors to cover:

- Pre-tender site safety audit checklists;
- Preventive maintenance procedures and inspections of tools and equipment;
- Documented plans for transporting heavy construction components and supplies;
- Pre-mobilization and kick-off meetings with IPMT and contractors;
- Hazard identification and risk assessment workshops;
- Project safety management plan audits; and
- Site safety and environmental orientation.

24.1.8.3. SITE MANAGEMENT PLAN

Contractors: will be responsible for:

- Accommodation and subsistence;
- Provision of construction labour;
- Provision of all construction equipment;
- Transportation of their workers to site;
- Site management of their works;
- QC programme in accordance with the construction technical specifications and the applicable codes and standards;
- Scheduling and reporting in accordance with the contract;
- Safety;
- Security for their tools and equipment;
- Supplying materials as required by contract;
- Meaningful project procedures; and
- Commissioning assistance.

The Construction Manager: will be responsible to plan, organize, and manage construction quality, safety, budget, and schedule objectives.

Site management responsibilities include, but are not limited to:

- Site work rules;
- Materials handling on site;
- Site offices;
- Site topographical survey;
- Site utilities for field offices;
- Concrete mix design – with assistance from engineering and procurement;
- Pre-Commissioning (cold commissioning).
- Communications system for construction;

- Document control on site – general project and construction;
- Constructability reviews – with support from engineering and procurement;
- Coordination of vendor representatives (erection support and commissioning);
- HSSE policy implementation;
- Site construction management;
- Warehouse and laydown area;
- Contract tendering – post tender meetings and recommendations;
- Contract administration;
- Earthworks and civil site supervision;
- Mechanical and piping site supervision;
- Structural site supervision;
- Electrical and instrumentation site supervision;
- Hot Commissioning support;
- On-site monitoring of construction equipment condition and safe operating capability;
- Survey and layout;
- Site QC;
- Cost reporting and controls – with engineering and procurement; and
- As-built drawings.

24.1.8.4. Camp

The 360 Man Camp will house Çiftay and Chaarat employees. Use will be made of the existing 'Advance camp' to accommodate the predicted peak number of 450 persons. The installation Contractors will be responsible for accommodating and feeding their own personnel.

24.1.8.5. Construction Water

During the construction phase, water will be required for the construction camp, dust suppression, and concrete production. As the project progresses, all construction water will be drawn from water filling station at the Sandalash river.

The potable water requirements will be satisfied from camp raw water wells that will be pumped to the water treatment facility in the shift camp facilities and then distributed for domestic usage.

24.1.8.6. Construction Power

During the construction phase, all power requirements will be met by utilizing 2 x 400 kVA generators. Diesel fuel will be transported from local temporary fuel storage of 84 tonnes of diesel fuel that will be refilled on a weekly basis.

24.1.8.7. Communications

During the construction phase, all communications including cell phones and internet will be provided by two Megacom towers powered by generators. The towers and generators are maintained by Chaarat's maintenance team.

Construction plots will be clearly identified, and construction works will be kept within these boundaries.

24.1.8.8. Construction Materials

The Civil contractor will be responsible for the operation of the batch plant and for supply of concrete. Aggregate will be sourced and crushed on site by Çiftay and stockpiled near the concrete batch plant pad located near the process plant area.

Non-acid generating (NAG) materials may be used for:

- Clean topping for substation;
- Engineered backfill for retaining walls;
- Site road dressing; and
- Piping, electric conduit, and cable trench protective backfill layers.

24.1.9. Project Completion

Chaarat is the project manager and Owner on the Project. There will be a clear delineation of the roles within Chaarat such that the construction team hands over to Chaarat's operational team at Project completion.

24.1.9.1. Mechanical Completion

Mechanical completion designates the point at which the contractor is considered to have completed his work such that the facility or system is ready for pre-commissioning. Mechanical completion represents 'substantial completion' at which time a complete list of deficiencies remaining is developed by the contractor and/or IPMT and used to measure the progress to final completion.

Mechanical completion of systems and facilities is a prelude to the pre-commissioning and hot commissioning of the overall plant

24.1.9.2. Commissioning and Start-up

Generally speaking, 'Pre-commissioning' will be managed by the Construction Commissioning team with the support of the Operations Team, whilst 'Hot Commissioning' and plant start-up will be managed by the Operations Team with the support of the Construction Commissioning Team.

- Pre-commissioning or subsystem level commissioning will be completed by the IPMT under the control of the Chaarat project manager to validate the correct and safe operation of subsystems. No material is used during the commissioning, but for liquid transfer and holding systems, water is often used in lieu of product. Cold commissioning of a subsystem can only commence if all components in the system have been issued with a mechanical completion certificate; and

- Hot commissioning follows the completion of all subsystem commissioning and will be completed by the Chaarat Operations team, with assistance from the construction team. Hot commissioning can only commence if all subsystems have been issued with a cold commissioning certificate.

The commissioning sequence will be developed at the start of the Project. Packages will be assembled to include all sign-off and test documentation, drawings and vendor information for each system that will be commissioned.

The various systems will be transferred to the Owner's operations team once they are completed and determined to be suitable for safe operation. The Owner's team will consist of plant operators and maintenance staff with the assistance of vendors, contractors, and pre-commissioning personnel as needed to trial the systems until they are finally accepted by the Owner's operations team. The transfer of systems will be formally documented and will include all mechanical/electrical testing documents and vendor's information.

24.1.9.3. Handover

Handover from the pre-commissioning team to the Chaarat operational team will take place at the end of cold commissioning. This step marks the handover of responsibilities and a change in reporting and accounting practice.

24.1.9.4. Close Out

During close out of the Project, the Chaarat's Technical Team will audit and update all project information including manuals and drawings. The IPMT will be disbanded, and outstanding obligations closed out.

24.2. Risks and Opportunities

Assessment of the more material risks was prepared during the Feasibility Study and identified areas of uncertainty with respect to the forecast revenue, capital and operating cost cashflows. The risk should be proactively monitored, and mitigation measures implemented during the execution of the Project. The risks are shown in Table 24-7 with their mitigation strategies.

Table 24-7 Material Negative and Positive Risks

Negative Risks (Threats)	Impact	Rating	Comments
Ore Body			
Copper (Cu) and Mercury (Hg) is not well enough defined.	Threat to health (Hg) Threat to recovery (Cu)	L	Space has been allowed for in the design for future installation of a Hg Retort. Multiple strategies and plant modifications are available for Cu.
Mining			
Planned Production not achieved due to: hauling constraints and relatively slow hauling (18kph)	A shortfall in revenue and project cash flow. Threat to revenue and cost.	M/H	The WRD will be located close to the main pit, to reduce Waste haulage turnaround time.
Non-Working Days - Only allowed for 10 days for lost days per annum.	More lost days will have a negative impact on estimated revenue. Threat to revenue and cost.	L	A management plan of emergency services, using trained personnel, will be implemented to deal with emergencies that include the impact of seasonal climatic events (i.e. avalanches and excessive water) and returning the mine to a workable condition.

Negative Risks (Threats)	Impact	Rating	Comments
Anfo vs Emulsion mix - Explosive specification requirement variation. Emulsion is more expensive. Blasting requirement may dictate higher consumption of Emulsion Mix.	Higher expenditure on Emulsion mix. Threat to cost.	L	Pamir Mining negotiated unit rates are fixed and include explosives.
Fuel Consumption, fuel price or travelling distances may differ from plan.	Higher fuel expenditure will have negative impact on cost. Threat to cost.	L	Cater for variances in detailed planning and implement proactive fuel price management strategies.
WRD valley requires additional drainage.	Higher costs.	L	Complete detailed engineering designs for the WRD early.
Unanticipated Gold Losses due to excessive water in haul trucks.	Threat to revenue.	L	Minimize fines and water content in haul trucks
Smaller slope failure - Major slope failures are not expected but smaller rock slope failures are a risk.	Failing of smaller slopes during pit development will cause slowdown in production. Threat to revenue	L	Regular monitoring and close management of benches
Mine Plan Flexibility - Insufficient flexibility in Mining Plan could affect the production rate.	Negatively impact revenue and add to costs.	H	Including smaller pits, earlier in the mine schedule, will increase the number of faces that can be mined. A stockpile will be made available.
Top Soil - Insufficient allowance in budget for laydown/storage of stripped top soil.	A threat of higher cost to provide laydown area for removed top soil.	L	Two sites have been identified to cater for 'storage' of vegetation and top soil. (As shown on the plot plan).
Metallurgical			
Delay in receiving sodium cyanide Licence for procurement.	This will delay the production of the gold and impact revenue	M	Dedicated person assigned to manage permitting process Application will be made early.
Mishandling of sodium cyanide during processing.	This can impact safety and health and consequently costs	L	Only let trained personnel handle sodium cyanide. Train medical personnel on site in sodium cyanide emergency care. Medical evacuation plan.
The approvals and permits for operation may be later than expected.	Late permits could delay the hot commissioning of the plant and impact revenue	M	Appoint a dedicated team in Bishkek to manage the design documentation and permit approval process.
Excessive fines	Could result in gold lock up.	M	Blast design to minimise fines. Mine planning to confirm delivery of potential fine material.
Starting the HLF in the winter, when it has no insulation or internal heat capacity, may result in liquor lock-up	Delayed gold production	M	Double stack at the start to provide insulation. Try and start irrigating as soon as possible. Current plan is for stacking to commence in summer. A decision has been take to not start start Leaching in the winter
Reagent Consumption - A risk of higher Reagent consumption exists based on type of material received.	A risk of higher reagent consumption will be a threat to production cost.	L	Detailed mine planning to separate undesirable material, which is addressed through the Ore Control process
Snow and rock fall at the ROM pad and plant area.	Injury to personnel or damage could add to cost	L	Protection measures (berms/catch basins) are planned and will be installed
Plant performance does not meet planned production	Threat to revenue	L	Close monitoring and management of plant performance to ensure plant operates according to design and planned production.
Legal/Logistics/Engineering/Environmental/Social/Economic			

Negative Risks (Threats)	Impact	Rating	Comments
Power (Fuel Storage, availability) - Limited fuel storage capacity will result in higher frequency of fuel procurement. Current storage allowance is two weeks.	Increased frequency of fuel procurement will result in higher procurement costs. Threat to cost	L	Fuel storage facility may need to be increased. Procurement frequency of fuel tanker shipment must be optimized to ensure minimum deliveries with assurance of continuance supply.
Access Road - Poor quality of access road is posing a risk to the safe and efficient travel and supply trains to site.	Repairs to road will require higher expenditure and impact the cost.	M	upgrade of road is in-progress and addressing the potential risks. Additional provision has been included in the owners operating cost for the maintenance of the roads.
Environmental Closure / Events - Currently the mine rehabilitation cost is not subject to a detailed estimate. The risk of spillage from ruptured HLF ponds must also be addressed.	Mitigation of the risk will result in higher expenditure. Threat to cost.	M	Mitigation steps and contracts to be closely managed to limit expenditure to what is necessary. Detailed study work to be initiated.
Avalanches & Rockfalls - The Tulkubash Project site is classified as a high Geohazard site.	Geohazard Management Plan could result in higher cost expenditure. Threat to cost.	M/H	Execution of plan must be closely managed to limit expenditure.
Security of Mining Tenure	Early termination or expiring of mining tenure is a threat to revenue.	L	Exploration and mining Licences are in place and to be kept up to date.
Legal Permitting - Explosives, fuel, and cyanide storage/handling.	Early termination or expiring of such legal permitting is a threat to revenue.	M	All legal permitting is in place. To be kept up to date.
Legalisation of Design - Legalisation and Adaptation of mine design can cause delay to detail design.	Threat to revenue	M	Close management of L&A process with Chaarat engineering department will mitigate the risk. As much L&A engineering requirements will be obtained pre detail design in order to ensure designs are aligned with Kyrgyz-standards before submission.
Social: Obstruction of access road due to social unrest.	Resolution of social unrest will result in higher expenditure of costs. Threat to cost	L	Social obligations with local community must be managed well.
Higher Tax liability than estimated in Feasibility Study	Additional capital and operating costs.	L	Include additional allowance for potential increased tax liability
Fluctuating Gold price in relation to state royalty equation	Threat to revenue	M	Gold sales need to be managed as far as possible
Higher Capital costs than estimated in Feasibility Study	Additional capital costs.	L	Firm quotes for capital costs would mitigate this risk.
Fluctuating commodity prices in relation to exchange rates	Threat to costs	L	Cater for variances in detailed planning and implement proactive commodity price management strategies.
Freight and logistics costs higher than estimated due to remoteness of the Project	Threat to costs	L	Procurement to monitor and manage these costs closely.
Increase in reagent usage as indicated by ALS-Stewart's latest testwork.	Increased operating cost	L	The ALS-Stewart testwork indicated reagent consumptions out of acceptable range of the other laboratories and is being discarded
Covid-19 or other pandemics	Suspension of Work	L	Site doctor, nurse and paramedics. Everyday quick health checks are being done. A procedure for Pandemics to be issued by HSE Manager. Certain medicines are stored at the site. In case of any person showing any symptoms, quarantine measures are applied.

TABLE 24-8 OPPORTUNITIES

Opportunity	Impact	Rating	Comments
Ore Body			
The Kyzyltash (sulphide) deposit remains untouched and will benefit from the general infrastructure provided for the Tulkubash Project.	Lower cut-off grade and lower to entry.	M	The sunk capital cost associated with the infrastructure development for the Project will improve the viability.
Additional exploration on strike likely to result in a larger mineral resource.	Opportunity for mine life extension.	H	Indications are that there is a larger reserve available within the Chaarat property. Chaarat's ongoing exploration drilling within the region will identify further potential.
Enhancing Inferred Resources to an indicated level of confidence through current ongoing infill exploration.	Opportunity for mine life extension and additional flexibility in future mine planning.	H	Additional information from exploration should be incorporated into the mine planning as soon as possible.
Mining			
Excessive dilution of 20% evident in planning could be much lower with proactive grade control.	Opportunity for cost saving.	H	In-pit identification of ore vs waste is likely to result in ROM ore with a higher than planned grade.
The nature of the mining contract.	Opportunity for cost savings Potential upside to the Project.	L	The identified mining contractor has worked in conditions similar to the Tulkubash environment. They have also had previous experience partnering with Chaarat.

25. Interpretations and Conclusions

25.1. Conclusions

This update of the 2019 BFS was necessary firstly to incorporate an enhanced mining plan that was possible due to an updated Mineral Resource estimate, and secondly to update the cost estimates.

The studies reported herein have confirmed that the orebody is amenable to a low-cost open pit mining and leaching operation that will deliver 418 koz Au over six years. RoM ore will be crushed to 80% passing 12.5 mm, stacked, leached and the pregnant solution passed through carbon columns to extract the gold. The final product will be a Doré bar of gold and silver with minor impurities.

The work involved reviews, checks, corrections and redesigns where necessary to ensure sufficient comfort in the materiality of the revenue and cost forecasts that have been used to assess the economic viability of the project.

25.2. Revenue related comments

The geological interpretations, block modelling and subsequent Mineral Resource estimate were reviewed with no errors or red flags encountered. Exploration drilling has defined an indicated resource of 789 koz of in-situ gold within approximately 2 km of a 6 km long strike, and the mineralisation is evidently continuous along strike.

Table 25-1 provides a comparison of the estimated Mineral Resource and the total depletion as scheduled from the open pit design.

Table 25-1 Comparison between In-situ Mineral Resource (07 November 2020) and Scheduled Depletion

Category	Quantity (Mt)	Content	
		(g/t)	(koz)
In-situ Mineral Resource			
Measured	-	-	-
Indicated	28.5	0.86	789
Total	**28.5**	**0.86**	**789**
Depleted Mineral Resource			
Measured	-	-	-
Indicated	20.9	0.85	571
Total (Measured and indicated)	**20.9**	**0.85**	**571**

Table 25-2 provides a comparison of the 2020 EOY Ore Reserve to the previously reported 2018 EOY Ore Reserve. This shows that the 2020 EOY Ore Reserves represent a 6% decrease in ore tonnage and a 7% decrease in grade compared to the 2018 EOY Ore Reserves. Overall, these changes result in a 13% decrease in contained ounces of gold.

The Inferred Resources within the pit limits, which are currently treated as waste, offer the potential to increase ore tonnage and contained ounces, along with decreasing the Strip Ratio (t:t) in the order of 5% to 10%.

TABLE 25-2 COMPARISION OF TULKUBASH ORE RESERVES AS AT 2018 EOY AND 2020 EOY

Parameter	Units	2018 EOY	2020 EOY	Variance
Ore	Mt	22.2	20.9	-6%
Grade (Au)	g/t	0.92	0.85	-7%
Metal (Au)	koz	658	571	-13%
Waste	Mt	58.6	54.1	-8%
Total	Mt	80.8	74.9	-7%
Strip Ratio	t:t	2.64	2.59	-2%
Recovery	%	68.9	73.6	7%
Recovered Au	koz	453	419	-7%

Source: Chaarat, 2021

The latest mine plan and associated production schedule are achievable and conservative with respect to the modifying factors that were applied for the Mineral Resource Estimate. However, the transporting of RoM ore to the heap leach pad could be problematic in the context of the local terrain and seasonal environmental changes. The area is remote and rugged with limited infrastructure, which leaves room for unforeseen difficulties and consequential expenses.

These challenges can nevertheless be managed. The Mining Contractor has extensive experience as a mining and civil engineering contractor in similar conditions and is well positioned to manage this type of mining operation. Both Chaarat and the Mining Contractor appreciate the importance of logistics and know that adequate access and a consistent power supply will be crucial to the operation, particularly in the context of an area that is affected by snowfalls for several months of the year.

The LoM recovery for gold and silver is estimated to be 73.6% and 63.4%, respectively. These have been based on results from metallurgical test work. The gold recoveries are variable and have been appropriately estimated using the geological block model for a more accurate assessment of recoverable metal over the LoM. However, it is noted that the performance of a full-scale plant may differ from the results anticipated from test work.

Chaarat are cognisant of the risks related to safety, health and the environment. These have been identified and management procedures and preventative measures are already being implemented.

25.3. COST RELATED COMMENTS

The Operating costs cover owner related expenses including general and administrative (G&A), the cost of mining, and a processing cost. The LoM operating cost estimate in real 2021 money terms are USD275.3 M or USD13.59/t ore (including VAT and import duties). Contingency has not been applied to the Operating Cost Estimate as it benefits from fixed and firm contract mining rates and, conservative estimates in the processing cost component.

The capital cost estimate for the Project was developed in real 2021 money terms and is within an accuracy range of –10% to +10%. The capital cost estimate amounts to USD130.7 M, including VAT, import duties and a 10% contingency. Normally a contingency of 15% would be applied to projects at an accuracy range of –10% to +10%. However, 21% of the initial capital estimate also benefits from fixed and firm contract mining rates.

While it was anticipated that the BFS would also address various gaps in the overall capital estimate, this was not necessarily achieved in all instances. Some 29% of the initial capital is still underpinned by information supplied by Chaarat. In addition, where insufficient information has been forthcoming, provisions have been made with particular reference to the estimated sum of USD2.8 M for a laboratory, utilities, sewage management, borrow pits, first fills, tools and equipment.

25.4. Comments on the Financials Assessment

This assessment was essential to demonstrate economic viability of the mineral resources depleted above to allow them to be presented as a JORC compliant Ore Reserve Report. The analysis also confirms that Chaarat will in all probability be able to deliver on the Tulkubash oxide mine with positive Project economics.

A pre-financing discounted cash flowmodel that included VAT and import duties, assumed a gold price of USD1,450/tr oz. It computed a real internal rate of return (IRR) of 25%, a NPV_5 of approximately USD81.5 M and an expected payback period of just over 5 years. The Project's value is most impacted by revenue followed by operating costs and capital costs.

The risks associated with the project are all manageable and provisions have been included in the budget where appropriate for the envisaged mitigation measures. These include, in particular, those related to gold price variations, the availability of the road from the Kumbel Pass to the Project site, congestion of internal haul roads, fuel consumption and/or price fluctuations, avalanches, logistics and local population expectations.

In conclusion, the updated Tulkubash Mineral Resource estimate has resulted in a new mine plan and a decreased Ore Reserve. However, the financial outlook of the project has improved due an improved mineral price, and gold recoveries. The success of this Project over the short term will unlock the significant longer-term potential of the Kyzyltash deposit.

26. RECOMMENDATIONS

The primary recommendation from this BFS is that Chaarat progresses the Project to the commissioning phase and eventually to steady state production.

However, in doing so, Chaarat should also:

- Consider ongoing opportunities to refine the production schedule with particular reference to the peak production periods;
- Introduce stringent proactive quality and grade control personnel into the mining operations early to ensure the quality of the RoM sent for processing.
- Continue to expedite the tender process to secure fixed and firm offers to continuously improve on the overall capital cost estimate;
- Maintain good relations with the relevant Kyrgyz authorities to facilitate a common understanding of their requirements and expectations so that timely approval of the necessary permits and approvals is possible. This will also help with the legalisation of the design;
- Establish a strategy around the sale of gold Doré to government to optimise the revenue stream. The strategy would need to consider the Kyrgyz government's priority purchase right and the existing royalty formula;
- Proactively monitor and manage the condition of the road over the Kumbel Pass, particularly during the winter season;
- Pay attention to haul road maintenance with particular emphasis on keeping them debris free and as dry as possible;
- Develop a strategy to optimally manage the seasonal variations in the price of fuel;
- Commission an independent logistics study to further firm up on the costs associated with freight and logistics;
- Commission additional studies on the geohazards to enable site specific mitigation measures to be crafted with respect to avalanches and rock falls. Proactively monitor the rock mass and groundwater with regard to slope failure. This should include drain holes and monitoring wells;
- Pay attention to the local community and proactively manage their expectations. An agreement on a social package would be beneficial in this regard; and
- Further develop the mitigation strategy to address impacts on the environment and, in so doing, improve on the current mine closure estimate. Improved data and information collection and monitoring may prove that less stringent measures are required to manage these potential impacts.
- Additional drilling to increase the ore reserve is warranted. There is 5 Mt of unused pad space in the HLF design that could be utilised with no additional Capex.

27. REFERENCES

- AMEC (2012). Environmental and Social Review and Gap Analysis. 27 Nov 2012
- Anon. (2004). Charaat Property Exploration Review. SRK Consulting. December 2004.
- Anon. (2012). Weather Station Report from the Agency on Hydrometeorology under the Ministry of Emergency Situations of the Kyrgyz Republic.10th August 2012.
- Cats, W., Verkholantsev. N., Savenkov A., Toktogaraev R. (2012). On the Results of Geological Exploration Works at the Chaaratskoj License Area for 2004-2012. Chaarat Internal Report. 128 pages.
- China Nonferrous Metal Industries Foreign Engineering and Construction Co Ltd and its affiliate China NERIN Engineering Co Ltd (2016). Chaarat Gold Mine: Chinese Feasibility Study Report (Executive Summary).
- Cole, A. and Seltmann, R. (2000). The Role of Granitoids during Variscan Orogenic Gold Mineralization in the Tien Shan and Ural Mountain Belts of Central Asia, in Bouchot, V. and Moritz, R., eds. Orogenic Gold Deposits in Europe. Editions BRMG. v. 297. p. 110-111.
- Davletbakov (2013). Background Environmental Assessment Report: Chaarat Deposit and Adjacent Area.
- Deustch, C.V., and Journel, A.G. (2017). GSLIB: A Geostatistical Software Library and User's Guide, Oxford University Press, New York, 340p., plus CD-ROM, 1997.
- Diner, J. (2017). Personal Communication.
- Fenne F. (2017). Personal communication, meeting, December 2017.
- Groves, D., Goldfarb, R., Gebre-Mariam, M., Hagemann, S., and Robert, F. (1998). Orogenic gold deposits: A proposed classification in the context of their crustal distribution and relationship to other gold deposit types, Ore Geology Reviews, Vol 13, p. 7-27.
- GSI (2014). October 2014 Resource Model Update for the Chaarat Gold Project, internal report to Chaarat Gold Holdings Ltd., October 19, 2014.
- GSI (2017). Tulkubash Checks, Partial Results as of August 28, 2017 Exploration Campaign, dated September 9, 2017.
- Gustavson Associates (2014). Chaarat Gold Project Resource Estimation, JORC Report", internal report to Chaarat Gold Holdings Ltd., June 23, 2014.
- Institute of Mine Seismology (2017). Probabilistic Seismic Hazard Analysis for a Site in Kyrgyzstan
- Isaaks, E.H., and Srivastava, R.M. (1989). An Introduction to Applied Geostatistics, Oxford University Press, 561p, 1989.
- Itasca and Atkinson (2016). Hydrogeologic Investigation and Predicted Groundwater Conditions to be Encountered during Proposed Mining of

Chaarat Gold Deposits in Western Kyrgyzstan – Input into Pre-Feasibility Study of Chaarat Project. Prepared for SRK Consulting. June 2010.

- Jukubiak, Z., 2017, Structural Study of the Tulkubash Mineralized Zone, Chaarat Gold Project, Chatkal District, Jalal Abad Province, Kyrgyzstan, Internal Report.
- Ken-Too (2015). Environmental Impact Assessment ("OVOS") Stage 1: Report on the exiting state of the environment at Chaarat.
- Kirchner, M. and Coetzee, L. (2011). General Mineralogical Characterization and Modified Gold Deportment Study on a Composite Gold Bearing Sample from the Tulkubash Project Area. SGS Mineralogical Report No: MIN 1210/208.
- Kramer, J., 2009, Structural Geology of the Chaarat Property, Internal Report.
- Leshem Sheffer Environmental Consultants Ltd (20122). Environmental and Social Baseline Study for Chaarat Gold Holdings Ltd.
- Letter from Kyrgyz Republic government regarding UNESCO Heritage Site borders, dated 2016
- MAIG (2020). The Tulkubash Gold Project Mineral Resource Estimate for Chaarat Zaav CJSC. Prepared for Chaarat Gold Holdings Ltd. November 2020.
- MCL (2018). Summary Report on Bottle Roll Cyanidation Tests – Tulkubash Drill Core Composites. MLI Job No.4227. 23rd March 2018.
- NK Group (2016). Engineering and Hydrological Survey at the Chaarat Deposit, Sandalash River Basin (Chatkal District, Jalal-Abad Oblast).
- Patel, S. (2014). Chaarat Geology and Exploration" Internal company report, Chaarat Gold Holdings Ltd.
- Porter, T. M. (2006). The Tien Shan Belt: Golden Heart of Central Asia, Issue 88, Published by the Mineral Deposits Division of GAC and the Geological Society of CIM.
- Roland, Y, Alexeiev, D., Kroner, A., Corsini, M., Loury, C., Monie, P. (2013), Late Palaeozoic to Mesozoic Kinematic History of the Talas-Ferghana Strike Slip Fault (Kyrgyz West Tianshan) as revealed by 40Ar/39Ar dating of syn-kenimatic White Mica, Jour. Asian Earth Sci 67/68, 76-92.
- Sehlotho, N. and Bryson, M. (2012). Flotation Flowsheet Development Testwork on the Tulkubash Gold Ores from Kyrgyzstan. Mintec External Report.
- SRK Consulting (2010). PreFeasibility Baseline Hydrogeology and Hydrology for Chaarat. Prepared for Chaarat Gold Holdings. Report Number 396943. August 2010.
- Tetra Tech (2014a). Hydrogeologic Testing Report. Technical Memorandum. Doc No. 311409-MEM-R0005-00. Prepared for Marcel DeGuire Chaarat Gold Holdings Ltd. January 2014.

- Tetra Tech (2014b). Open Pit Slope Feasibility Geotechnical Study – Chaarat Gold Project, Kyrgyzstan. Presented to Chaarat Gold Holdings Ltd. February 2014.
- WAI (2017a). Chaarat Gold Holdings Ltd. Tulkubash Heap Leach Feasibility Study Hydrology Report for the Heap Leach Facility. May 2017.
- WAI (2017b). Chaarat Gold Holdings Ltd. Tulkubash Gold Project, Kyrgyzstan Report on Metallurgical Testing Conducted as Part of a Feasibility Study. 24th August 2017.
- WAI (2018). Environmental and Social Impact Assessment in Support of the Chaarat Gold Feasibility Study (Phase 1 – Tulkubash) in Kyrgyzstan. Chapter 3 – Project Description.
- Wardell Armstrong (2020). Environmental and Social Impact Assessment In Support of the Chaarat Gold Feasibility Study, Tulkubash, Kyrgyzstan. Report Number MM1423. Prepared for Chaarat Gold Holdings Ltd. October 2020.
- Stewart Assay and Environmental Laboratories (2019): Results of Technological Testing of 22 composites samples in support of Charaat Gold 2021 BFS Update.

Report Sign-off Sheet

Richard Bewsey - *Study Author*
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Signature	*Date*

Peter Carter - *Mining Competent Person*
(Independent Third Party)

Signature	*Date*

Victor Usenko- *Geology Competent Person* (Independent Third Party)

Signature	*Date*

Study Content Sign-off Sheet

1. Metallurgy and Process:

Richard Bewsey
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Signature	*Date*

2. Geology and Mineral Resource:

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(Independent Third Party)

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3. Mining:

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