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Revised NI 43-101 Technical Report

Pre-Feasibility Study for the Rozino Gold Project, Bulgaria

PREPARED FOR: Velocity Minerals Ltd

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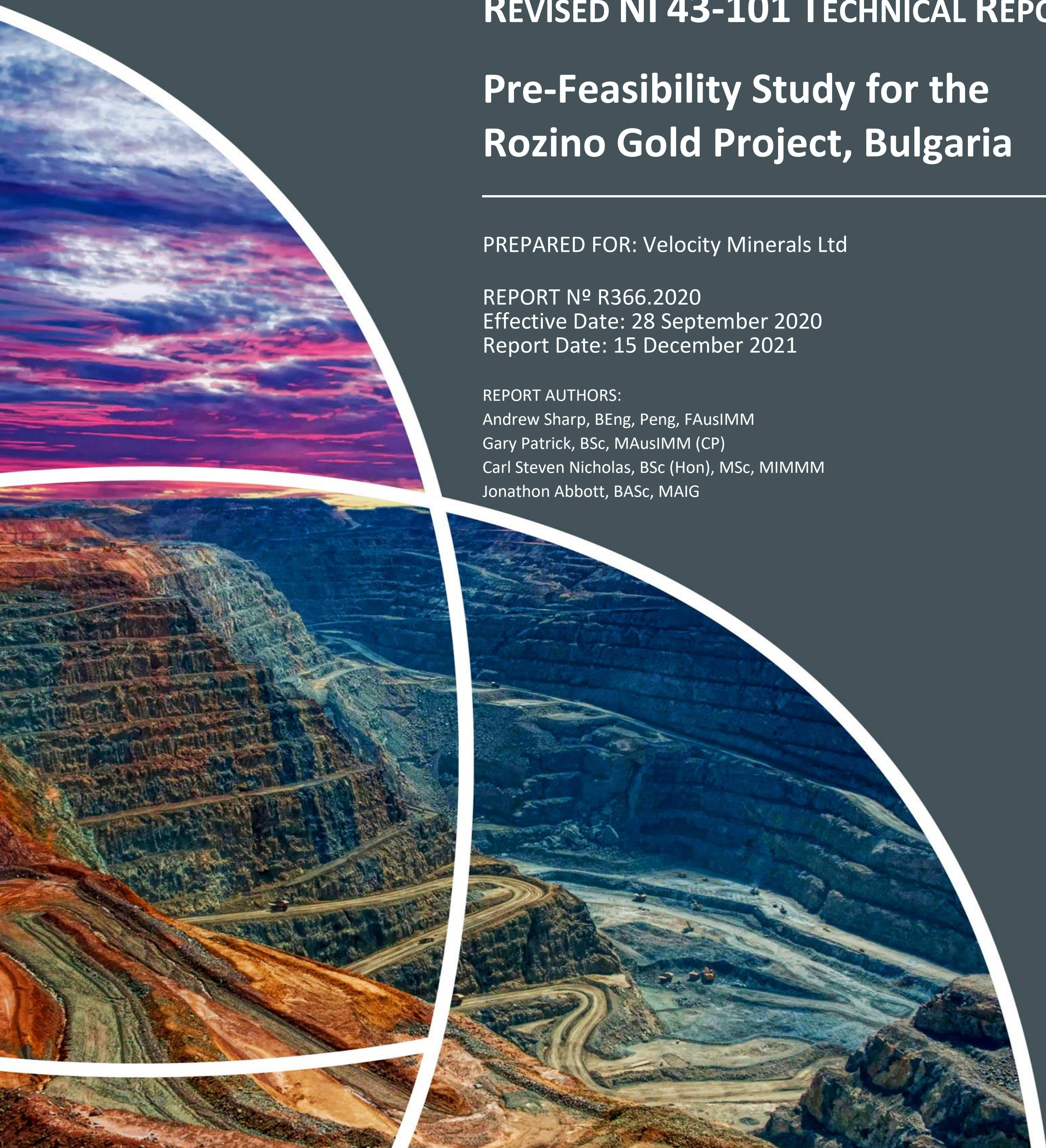

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1 Summary

1.1 Introduction

This Technical Report has been prepared by CSA Global Consultants Canada Ltd ("CSA Global") in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and is suitable for filing with Canadian Securities Commissions. This document has been prepared for Velocity Minerals Ltd ("Velocity") to disclose material information related to an updated Mineral Resource estimate and Pre-Feasibility Study (PFS) for the Rozino Gold Project ("Rozino" or the "Project"), located within the Tintyava Prospecting Licence ("Tintyava Property" or the "Property"). The effective date of the Technical Report is 28 September 2020. The Technical Report summarizes the results of the Pre-feasibility Study.

A number of companies contributed to the Pre-feasibility Study. Major contributors include MPR Geological Consultants Pty Ltd (for the Mineral Resource estimate), Mineesia Ltd (for environmental management and regulatory requirements), Golder Associates UK (for geotechnical, hydrological, hydrogeological, tailings management facility (TMF), raw water dam (RWD) and contact water dam (CWD) designs), and Halyard Inc (for processing and most of the infrastructure capital and plant processing costs). CSA Global were responsible for the Mineral Reserve estimate, mining, metallurgy, and overall study lead in relation to evaluating the Rozino Project.

The Mineral Resource estimate was disclosed in accordance with NI 43-101. All technical works have been undertaken under the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) report, "CIM Definition Standards for Mineral Resources and Mineral Reserves" prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.

The results of the mine plan and the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this section include, but are not limited to, statements with respect to the future price of gold, the estimation of Mineral Reserves and Mineral Resources, the realization of the Mineral Reserve estimate, the timing and amount of estimated future production, costs of production, capital expenditures, results of the permitting process, currency exchange rate fluctuations, requirements for additional capital, government regulation of mining operations and taxation, environmental risks, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

On 30 May 2017, Velocity (through its wholly owned subsidiary Kibela Minerals AD ("Kibela") and Gorubso entered into an option agreement. Pursuant to the terms of the option agreement, Velocity acquired the exclusive option (the "Option") to acquire a 70% interest in Tintyava Exploration and the Tintyava Property. The Option being exercisable through delivery of a PEA, within the meaning of NI 43-101. In February 2018 the Property was transferred from Gorubso to Tintyava Exploration EAD, the joint venture vehicle for Velocity and Gorubso.

On 31 October 2018 Velocity delivered a PEA Technical Report prepared under National Instrument 43- 101 of the Canadian Securities Administrators ("NI 43-101") to Gorubso. Following delivery of the PEA, Velocity was deemed to have earned an undivided 70% interest in Tintyava Exploration and the Tintyava Property. Rozino is located within the Tintyava prospecting licence.

On 1 March 2019, Velocity (through Kibela) entered into a shareholder's agreement with Gorubso regarding Tintyava Exploration and 70% of the shares of Tintyava Exploration were transferred to Kibela. The participating interests of Velocity and Gorubso in the JV are 70% and 30% respectively and both partners contribute pro rata to joint venture costs. If either Velocity or Gorubso does not contribute its portion of expenditures, then that party's interest in the JV will be diluted and if reduced to 10% or less, will convert to a 1% NSR royalty.

The financial model is reported at 100% attributable equity. The financial model does not include any consideration of funding or funding costs.

This report utilizes dollars of the United States of America (US$) as the base currency as the majority of capital and operating cost estimates are based in US$; these are denoted as $.

1.2 Property and Locality Description

The Rozino deposit is located within the Tintyava Property, which lies within the municipalities of Ivaylovgrad and Krumovgrad in southeast Bulgaria, about 350 km by road east-southeast of the capital, Sofia. The Tintyava Property, covered by Prospecting Licence 467, has an area of approximately 145 km^2. It is accessible year-round by sealed roads. Forestry roads and exploration tracks provide year-round access within the property.

The Project area's average annual temperature is around 12°C, ranging from around 2°C in January to 24°C in July. Maximum rainfall occurs during November and December, with rainfall of up to 100 mm per day. Snow cover is sporadic, lasting generally only for five to ten days per year. Exploitation activities will be able to be undertaken throughout the year.

The local terrain is characterized by low mountains and predominantly levelled hills and is cut by steep valleys. Elevations range from 110 masl to 740 masl and average about 360 masl. The Rozino Project site is bounded to the south by steep cliffs at Tashlaka. The terrain is dissected by the Byala reka (White river) and its tributaries. In the project area elevation ranges from about 300 to 450 masl in the north, reducing to approximately 300 masl in the south.

The project area is covered predominantly by indigenous and cultivated forests comprising European oak and black pine, and to a lesser extent grass pastures.

Small villages are dispersed widely throughout the licence area. The residents are involved primarily in subsistence farming, particularly livestock and the growing of tobacco. The other main land use within the licence area is state-controlled forestry. Rozino village is largely deserted, with only a few families remaining. There is a 20 kV power transmission line 2.5 km from the Project and, while this powerline supplies the villages with electrical power, additional power will be required for the development of the Project. All villages have access to fresh water through a network of reservoirs.

1.3 History

Geoengineering AD ("Geoengineering") initiated modern exploration within the Tintyava Property (the "Property") in the 1980's including diamond drilling in the Rozino area. Between 2001 and 2007 Hereward Ventures Bulgaria Ltd ("Hereward") explored the Tintyava area including diamond drilling in the Rozino area, some of which was undertaken in joint venture with Asia Gold Corp, ("Asia Gold").

The original Prospecting Licence (PL) containing the Rozino deposit was cancelled in 2013. In 2016 Gorubso-Kardzhali AD ("Gorubso") won a competitive tender for exploration rights to the Property. As part of an earn-in option agreement Velocity began exploration in July 2017 and in February 2018 the Property was formally transferred from Gorubso to Tintyava Exploration EAD, the joint venture vehicle for Velocity and Gorubso.

1.4 Geology and Deposit Type

Geology of the Rozino area comprises a series of discrete Palaeogene syn-tectonic pull-apart sedimentary basins within metamorphic basement.

Rozino is a low sulphidation epithermal ("LSE") gold deposit hosted within generally brecciated and conglomeratic Palaeogene sedimentary rocks as disseminations, replacement and vein mineralization. Alteration is characterized by a quartz, carbonate, chlorite, adularia, pyrite assemblage. The mineralogy consists mainly of pyrite with traces of base metals and rare arsenopyrite. Gold occurs at sulphide mineral boundaries and less commonly as free grains or encapsulated inclusions.

Velocity considers that untested parts of the Palaeogene basin within the Tintyava Property have potential to host LSE mineralization analogous to that observed at Rozino.

1.5 Exploration/Drilling and Sampling

This report reflects drilling and exploration sampling information available for the Property on the 28th of September 2020.

The author considers that quality control measures adopted for sampling and assaying of the exploration sampling and drilling of relevance to Mineral Resources and exploration have established that the sampling, and assaying is representative and free of any biases or other factors that may materially impact the reliability of the sampling and analytical results. The author considers that the sample preparation, security and analytical procedures adopted for the Tintyava exploration sampling and drilling provide an adequate basis for the current Mineral Resource estimates and exploration activities.

Exploration sampling of relevance to current exploration comprises soil, stream sediment and rock chip sampling undertaken by Velocity. Results of this sampling support Velocity's interpretation of the Property's geology and are, in the author's opinion, sufficiently suggestive of the potential for deeper mineralization to warrant further investigation including targeted exploration drilling.

The Rozino Mineral Resource estimates described in this report are based on Hereward Asia Gold and Velocity drilling information available on the 23rd of October 2019.The combined hole spacing for the deposit varies from around 50 by 50 metres and locally closer in central portions to around 100 by 100 metres in peripheral areas.

Velocity's drilling completed since October 2019, and not included in Mineral Resource estimation comprises 19 holes in the Rozino area and 19 exploration holes testing regional targets within the Tintyava Property as part of Velocity's current on-going exploration drilling program.

Post October 2019 Rozino area drilling includes 17 infill holes and 2 extensional holes. The infill holes targeted an area where the current resource model shows generally low gold grades in model blocks that are categorized as Inferred and outside the optimal pit constraining Mineral Resources which are not included in Mineral Resource estimates. These holes intersected mineralization of similar tenor to that shown by earlier drilling in peripheral portions of the Rozino deposit. The author recommends that Mineral Resource estimates are updated to include this drilling. The two extensional holes, which were drilled to the west of the current model estimates did not return significantly elevated gold grades.

Information available from Velocity's current and on-going exploration drilling totals 9 diamond holes at Rozino South, Kazak East and Kazak West areas. This drilling represents comparatively early stage exploration of areas that are being evaluated as part of Velocity's on-going exploration of the Tintyava Property.

1.6 Mineral Processing and Metallurgical Testing

The Rozino ore body comprises three ore types, Oxide (16%), Transitional (15%), and Sulphide (69%), each differentiated by the degree of weathering. Extensive metallurgical testwork was undertaken on a number of composites and variability samples considered representative of the deposit, ore types or certain constituent parts of it. It was concluded from the testwork that the optimal process for treating the Rozino mineralization is bulk flotation at a nominal grind size of 80% passing 75 µm (P_{80} 75 µm) to produce a gold-bearing concentrate. The concentrate requires further grinding to P_{80} 20 µm and then processing in a conventional CIL circuit for 48 hours. Subsequent desorption of the gold from the carbon, electrowinning and finally smelting will occur to produce a gold doré.

Depending on the ore type, some 35% to 60% of the gold in the ore is recoverable by gravity methods. The testwork fully investigated gravity recovery options and combinations with high intensity leaching of the gravity concentrate and flotation of the gravity tailings but failed to find a more economic solution than the simpler bulk flotation path.

The metallurgical testing covered a range of variability tests and concluded that the three ore types adequately cover the range of expected controls on recovery and plant processing characteristics (such as hardness and reagent consumption). It was also demonstrated that metallurgical recovery is a function of gold head grade; recovery increases with increasing head grade. The overall metallurgical recovery of gold for the Mineral Reserve is estimated to be 79.3% to doré, based on the proportions of ore types, head grades and the supporting Pre-feasibility Study testwork.

A comminution testwork programme was completed to characterize the crushing and grinding characteristics of the three ore types with respect to both conventional crushing/grinding and autogenous/semi-autogenous (AG/SAG) grinding. The ore was determined to range between "very easy" to "easy" for crushing, "medium" for milling and for wear rates between "non-abrasive" and "slightly abrasive".

Testing also covered reagent types and qualities required for the ore types as well as settling characteristics for dewatering.

1.7 Mineral Resource Estimation

Mineral Resources were estimated by Multiple Kriging (MIK) of two metre down-hole composited gold grades from diamond drilling by Hereward, Asia Gold and Velocity. Estimated resources include a variance adjustment to give estimates of recoverable resources for selective mining dimensions of 4 metres east by 6 metres north by 2.5 metres in elevation and are reported within a 150 metre deep optimal pit shell generated at a gold price of $1,500/oz. Estimates for mineralization tested by up to approximately 50 metre spaced drilling are assigned to the Indicated category, with estimates for broader and irregularly sampled mineralization classified as Inferred.

The optimization parameters generate a cut-off grade of 0.3 g/t Au, which is selected as the base case for Mineral Resource reporting Table 1-1 presents Mineral Resources estimated for Rozino at this cut-off grade. The figures in this table are rounded to reflect the precision of the estimate and include rounding errors. The Mineral Resource estimate was classified and reported in accordance with NI 43-101 and the classifications adopted by the CIM Council in May 2014 (CIM, 2014). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The Indicated Mineral Resources are inclusive of Mineral Reserves.

Table 1-1: Rozino Indicated and Inferred Mineral Resource estimates at 0.3 g/t cut-off

Rozino Indicated and Inferred Mineral Resource Estimates at 0.3 g/t cut-off			
Effective date of estimate: 15 April 2020			
	Tonnes (Mt)	**Grade (Au g/t)**	**Metal (Au koz)**
Indicated Mineral Resource Estimate	20.5	0.87	573
Inferred Mineral Resource Estimate	0.38	0.8	10

1.8 Mineral Reserve Estimate

The Rozino Gold Project supports an economic open pit mining operation. The Mineral Reserve estimate is based on the Indicated category of the Mineral Resource contained within the pit design. The Mineral Reserve estimate has considered all modifying factors appropriate to the Rozino Gold Project.

The Mineral Reserve is reported within a pit design that incorporates geotechnical parameters developed from drilling, geotechnical testing and technical analysis at a level of detail that supports a Pre-feasibility Study. The pit designs are the outcome of a valuation optimization process that is moderated to some extent by the obligation to minimize the project footprint. The reference point at which the Mineral Reserves are defined is where the ore is delivered to the Flotation Plant.

Table 1-2: Mineral Reserve for the Rozino Gold Project (effective date 30 August 2020)

Ore type	Reserve category	Tonnes (Mt)	Grade (Au g/t)	Contained metal (Au koz)	Metallurgical recovery (%)	Recoverable metal (Au koz)
Oxide	Probable	1.9	1.07	64	67.4	43
Transitional	Probable	1.8	1.15	68	70.7	48
Sulphide	Probable	8.1	1.27	332	83.3	277
Total	**Probable**	**11.8**	**1.22**	**464**	**79.3**	**368**

Notes:

- *The Mineral Reserve disclosed herein has been estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum "CIM Definition Standards for Mineral Resources and Mineral Reserves" (CIM, 2014).*
- *Mineral Reserves discard cut-off grade is 0.5 g/t gold.*
- *Mineral Reserves are based on a $1,500/oz gold price.*
- *Mineral Reserves account for mining dilution and ore loss.*
- *Probable Mineral Reserves are estimated from Indicated Mineral Resources.*
- *Sum of individual values may not equal due to rounding.*

No inferred Mineral Resources are included in the Mineral Reserves. Inferred Mineral Resources do not contribute to the financial performance of the project and are treated in the same way as waste.

Mining losses and mining dilution are incorporated in the MIK Mineral Resource estimate. CSA Global were able to determine that mineralization can be adequately modelled for its diluted, recoverable grade assuming a selective mining unit (SMU) of 4 x 6 x 2.5 m using the MIK methodology. CSA Global consider that the Mineral Resources can be effectively mined by open cut extraction using the selected mining equipment and qualifications relating to training, grade control practices, and drilling and blasting technique, without additional dilution and loss factors being applied.

There are no known likely mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the estimate. It is important to note that permitting for the Rozino Gold Project is not complete. Velocity has initiated the environmental and social impact assessment process, including the permitting procedures to meet Bulgarian regulations and gathering environmental data to improve the design of the Project.

1.9 Mining Method

The Rozino Gold Project comprises a single open pit utilizing standard mining equipment to selectively extract the ore over a nine-year period (inclusive of a two-year construction and pre-stripping period). The pit will be mined in two phases utilizing a waste rock dumping and backfill strategy that provides a compact footprint. The mining is at extraction rates of up to 22,000 tpd using 90 tonne class excavators and 55 tonne capacity haul trucks. All material will require blasting. The schedule delivers higher grade ore directly to the plant in the initial five-year period of the mine life at 5,000 tpd. Low-grade ore is stockpiled during this period and reclaimed for processing when the pit is exhausted. The resulting tailings from the low-grade ore are stored in the pit.

The mine plan has a pre-stripping period that is carefully balanced to produce waste rock for the construction of the water storage dam walls, the TMF embankment and ore stockpile pads. The pre-stripping period also delivers a small amount of ore that will enable the plant commissioning and start-up. Although the pre-stripping is over a two-year period, the first year entails mostly road building, clearing, grubbing and topsoil removal.

In-pit water management will primarily consist of runoff control and sumps. The dewatering infrastructure and equipment are sized to handle the estimated groundwater inflows and precipitation. In-pit water will be pumped to an external water diversion ditch that delivers the water flow to the CWD. Although pit water inflow rates are predicted to be low, two pit wall depressurization bores are included in the mine plan. The surface water handling plan will be based on diverting as much surface water as possible away from the open pits, collecting it by using ditches diverting ultimately to the CWD.

1.10 Recovery Methods

The optimal process route for treating the Rozino sulphide mineralization is flotation to produce a gold-bearing concentrate, followed by cyanidation of the concentrate in a conventional CIL circuit, elution to separate the precious metal complexes from the carbon, electrowinning of the precious metals to iron wool and finally refining to produce gold doré.

The flotation of the concentrate will occur at an on-site plant (the "Flotation Plant") designed to process 1.75 Mt of ore over the LOM at steady state operation. The further processing of the concentrate will occur at Gorubso's existing and operating carbon-in-leach (CIL) plant (the "Central Plant") located in Kardzhali, 85 km by road from Rozino. Gold doré will be produced at the Central Plant.

The Flotation Plant design is based on extensive metallurgical testing. Mass balance and ore characterization tests underlie the plant design specification. Vendor quotes cover more than 72% of all equipment and construction designs. Where vendor quotes were not available, CSA database reference information, vendor advice, and assumptions based on experience have been used.

Comminution at the Flotation Plant is by a three-stage crushing circuit (jaw crusher and two cone crushers) followed by a conventional ball mill. The crushing circuit design assumes a 75% availability. The availability of the milling and flotation circuit is assumed to be 92%. The ore will be ground to P_{80} 75 µm.

Reagent requirements and operating costs have been estimated for each ore type. Concentrate mass pull and grade will vary as a function of gold head grade and ore type processed. At an average mass pull of 3.8% the Flotation Plant will produce on average 67,000 tpa of concentrate with an average grade of 30 g/t gold.

Whilst the pit is operating, inert tailings from the Flotation Plant will be stored in the Rozino tailings management facility. After pit mining is complete, low-grade ore will be processed and the tailings will be stored in the depleted pit.

At the Central Plant the concentrate will be ground to P_{80} 20 µm prior to entering the leach circuit. The gold will be desorbed from the carbon, electroplated on steel wool and refined into gold doré bars. The concentrate leach recovery and reagent requirements have been estimated from testwork carried out on cleaner-circuit concentrate samples. The Central Plant is estimated to produce 368 koz of gold in doré over the project life. The doré is expected to contain up to 60% gold, 30% silver and 5% copper. Silver was not considered in the Mineral Resource estimate and is not included in any part of the financial analysis.

The Central Plant tailings will be detoxified by the currently operating INCO process and stored in the existing Central Plant tailings management facility.

1.11 Project Infrastructure

In terms of infrastructure, the Rozino deposit is a greenfield mining prospect and no infrastructure exists at the proposed mine site. The site is currently accessed from the main provincial road II-59 via a 12 km single-lane, paved road and closest to the Project, a 2 km unsealed dirt road. The access roads will be upgraded to accommodate mine traffic, especially concentrate haul trucks. Electrical power will be supplied by a dedicated 110 kV overhead powerline that will be constructed and connected to the Madzharovo substation 23 km to the north.

Site structures, including the enclosures for the comminution and flotation plant, will be appropriate for the relatively short mine life of about seven years. Other design and engineering criteria applied to infrastructure location was to minimize the distance and elevation differences between the pit, Flotation Plant and TMF. The compact project footprint reflects the constraints imposed by environmental considerations.

The Project water management plan is central to maintaining an appropriate environmental and operational performance for the Project. The principle adopted for site water management is to intercept and control contact water flowing within the operational areas to ensure that it stays within the catchment area located to the east of the mine operations. This contact water will report to the tailings management facility (TMF) and the contact water dam (CWD) located directly below the TMF. The water will then be pumped back to the water storage tanks located at the processing facility for use in the process plant and mining operation. The site water balance indicates that the Project will have a negative water balance. Water reuse will be maximized, but plant process make-up water will need to be sourced from external, local water sources. Water sourced from external sources will be pumped to the raw water dam (RWD) located below the CWD in the first instance. Water to augment the anticipated shortfall in supply from on-site sources and recycling will be sourced externally.

The location for the TMF was selected with the objectives of minimising the project footprint and being in close proximity to the Flotation Plant to reduce tailings and return water pumping costs. Thickened tailings disposal was selected as the most cost-effective solution compared with other options (such as paste or dry stack). A downstream raised TMF wall was selected as being the most appropriate for the seismicity of the region.

The TMF has been designed to store 8.6 Mt (6.125 Mm^3) of tailings delivered over a 6-year period. In the sixth and seventh year stockpiled low-grade ore will be processed and the tailings will be used to backfill the completed pit (approximately 2.6 Mt). The TMF is constructed with an initial starter wall approximately 37 m high and will be raised on an annual basis to a final height of 67 m.

The major infrastructure at the Rozino site is:

- ROM stockpile, tip, crusher circuit and conveyor system
- Ball milling circuit
- Flotation plant and related facilities for dewatering concentrates and tailings materials
- Concentrate weighbridge
- Mine maintenance work shops and warehouses
- Change rooms and administration complex
- Fuel storage and distribution
- Wash bay
- Security building
- Fire protection
- Industrial and effluent waste management
- Site roads
- Communications
- Standby generator.

1.12 Market Studies and Contracts

All doré will be sold through refineries based in Europe. The relatively small size of sales compared to market demand give no concern to impacts of sales on the metal price. No forward sales are considered. The Qualified Person has reviewed all market studies and the results support the assumptions of this Technical Report.

Velocity evaluated market price forecasts for gold using publications and opinion provided by UBS and Haywood Securities Inc.

The metal price selected for the financial analysis was $1,500 per oz. This price is 6.8% above the three-year rolling average of $1,404 per oz (data source: World Gold Council) as of 30 August 2020. The gold spot price on the same day was $1,957 per oz (data source: World Gold Council).

1.13 Environmental and Social

Velocity has initiated the environmental and social impact assessment process, including the permitting procedures to meet Bulgarian regulations and gathering environmental data to improve the design of the Project. Under the Bulgarian Environment Protection Act, the development of an economically viable mining reserve requires an Environmental Impact Assessment ("EIA") which complies with European Union (EU) environmental regulations. The prospecting licence agreement for the Tintyava Property was signed with the Minister of Energy and exploration activities were approved by the Ministry of Environment. All necessary permits to conduct the work proposed for the property are obtained and there are no known significant factors or risks that may affect access, title or the right or ability to perform work on the Property. There are currently no objections to the development of the Project.

The Rozino Project is located within the Eastern Rhodope mountains, which is an area of wide biodiversity, and therefore requires a compatibility assessment to comply both with Bulgarian Law and the European Union Natura 2000 Habitats Directive. An initial compatibility assessment was conducted and approved for the exploration program, with a second preliminary assessment completed for exploitation. The results of this preliminary assessment have informed the Project design, resulting in a reduced Project footprint. Additional measures include surface and groundwater studies, and trial blasts to further understand potential impacts arising from operations.

The EIA will include an assessment of the environmental and social impacts within the zone of influence of the planned development and compare these to existing conditions. Velocity has commenced baseline monitoring to characterize environmental conditions, including surface and groundwater quantity and quality, air quality (specifically airborne dust), acid drainage potential and ecology.

Preliminary hydrogeology and hydrology studies indicate that the Project will have a negative water balance, so any potential contaminated water will be contained and re-used in the plant process circuit. Geochemical results to date indicate a low potential for acid drainage, due to the significant neutralization potential of ore and the waste rock. Metal leach testwork indicate that the risk of harmful leachate is low. This, in conjunction with the negative water balance, implies that the risk of water contamination is low.

Social engagement activities have commenced and are ongoing. Local stakeholders are supportive of the Project and are employed in the Project where possible. There are few people living in proximity to the Project, and the Project footprint is designed to minimize encroachment onto privately-owned land.

The Project has the potential to impact a range of environmental and social aspects. Velocity is committed to managing the impacts of its operations in conformance with recognized international best practice. Mitigation measures will be developed through the EIA process to manage potential impacts and implemented for effective environmental and social development, operation and closure of the Project. An environmental management plan will be developed to ensure that appropriate control and monitoring measures are in place. It will be designed for review and updated throughout the life of the Project.

1.14 Capital and Operating Costs

The Rozino Project total capital expenditure is estimated at $94.8 M. Table 1-3 summarizes the main capital items.

Table 1-3: Total capital expenditure

Capital expenditure	$M
Rozino Gold Project Site Preparation	12.6
Mine Infrastructure	10.7
Flotation Plant and Mine Buildings	39.0
TMF incl waste overhaul	9.8
Central Plant Upgrades	1.1
Owner's Administration Costs	2.9
Indirect Costs	2.2
EPCM and Commissioning Costs	7.0
Contingency	9.6
Total Project CAPEX	**94.8**

All project costs incurred prior to the declaration of commercial production (24 months after commencement of construction) are considered pre-production capital costs that total $87.1 M. The remaining $7.8 M of capital expenditure (sustaining capital) will occur over the seven-year operating life. Approximately 95% of the sustaining capital is for TMF construction.

Operating costs were based on the development of equipment productivities, the Rozino local and regional operating environment and contractor quotations or supplier costs for machinery and services in Bulgaria.

Labour costs across all activities were estimated from a detailed labour survey and benchmarking exercise. An adjustment factor to allow for upward pressure in labour rates due to the integration of Bulgaria into the European Union commences at 7% in the first year of construction and reduces to 2% in the last year of production.

The mining operating costs includes the leasing of primary and ancillary mining equipment, drilling and blasting carried out by a contractor, loading, hauling of ore and waste, and ore rehandling.

Flotation Plant operating costs include all consumable items (balls for the ball mill, reagents, and chemicals) power, external services, and maintenance. A contingency of 7.5% is included.

Concentrate haulage will be provided by a contractor at a rate of $0.146 per wmt/km. The Central Plant costs include concentrate handling, cyanidation, carbon desorption, electro-winning and refining to produce gold-silver doré.

Mine closure and rehabilitation costs, as well as post-closure management for a period of 10 years were estimated. These costs are reflected as an environmental provision per processed tonne of ore over the operational life of the mine.

Administration costs were developed from first principles and based on Bulgarian labour, service and material input costs.

The average LOM mine operating cost is estimated to be $20.62/t of ore milled.

Table 1-4: LOM operating costs

Operating costs	$/tonne milled
Mining	8.43
Flotation Plant	7.04
Concentrate Haulage	0.53
Central Plant	2.35
Administration	1.93
Environmental Provision	0.33
All-In OPEX	**20.62**

1.15 Economic Analysis

The financial evaluation presents the determination of the key economic performance indicators for the Rozino Gold Project, including the Net Present Value (NPV), the payback period (time in years to redeem the initial capital investment), and the Internal Rate of Return (IRR) for the project. The discounted cash flow (DCF) model is reported at 100% attributable equity. Annual cash flow projections are estimated over the life of the mine based on the estimates of capital expenditures, production costs and sales revenues. The sales revenue is based on the production of a gold doré.

The financial estimate includes forward looking information for which a risk statement is included in the introduction above. Additional risk can come from actual results of changes in Project parameters as plans continue to be refined, possible variations in Mineral Reserves, grade or recovery rates, failure of plant, equipment or processes to operate as anticipated, accidents, labour disputes and other risks of the mining industry, and potential delays in obtaining additional governmental approvals.

The estimates of initial and sustaining capital expenditures and site production costs were developed specifically for this project and are presented in earlier sections of this report. Total initial capital totals $87.1 M and sustaining capital totals $7.8 M (see the report section Capital and Operating Costs for details and Table 22.4).

The Rozino Gold Project has cash costs of $699 per oz payable gold. The after-tax NPV, at assumed long term metal prices, using a 5% discount rate is $122.5 M and the internal rate of return is 27% (Table 1-6). Payback of the initial capital occurs 3.0 years after commercial production commences (Table 1-6). A total of 368 koz of gold is recovered over a seven-year processing life (Table 1-5) and nine-year total project life before rehabilitation.

The economic analysis and supporting financial information, including capital and operating costs, were developed in constant dollar terms. The economic analysis uses the Probable Mineral Reserves as described in the Mineral Reserve Estimate of this report. Cash flow forecasts on an annual basis using the Mineral Reserves for the base case metal price are included in the full discounted cash flow analysis and material schedule (Table 22-4). Sensitivity analysis charts are presented as *Figure* 1-1 and Figure 1-2. Key financial assumptions are presented in the tables below.

Table 1-5: Key project overview and metrics

Project Overview		Units	
Mining	Total ore production	Mt	11.8
	Total waste production	Mt	26.5
	Total mined	Mt	38.3
	Metal mined	gold koz	465
	Mine life	years	6.9

Project Overview		Units	
Processing	Steady state ROM production	Mtpa	1.750
	Years steady state production	years	5.0
	Average production rate	ktpd	4.7
	Average gold head grade	g/t	1.22
	Overall metallurgical recovery	%	79.3%
	Payable Au	LOM koz	368
		average koz pa	54

Table 1-6: Summary of economic results

Analysis Case	Summary of Economic Results	Units	Value
Pre-Tax	NPV @ 0%	$M	198.5
	NPV @ 5%	$M	137.0
	IRR	%	34.7%
	Payback (Project Start)	years	4.9
	Payback (Production Start)	years	2.9
Post-Tax	NPV @ 0%	$M	179.3
	NPV @ 5%	$M	122.5
	IRR	%	27.4%
	Payback (Project Start)	years	5.0
	Payback (Production Start)	years	3.0

A number of single parameter sensitivity impacts were analysed. Neither the gold cut-off grade, mine plan nor the processing plan were altered. The Rozino Gold Project's NPV is most sensitive to changes in metal price and factors such as head grade and metallurgical recovery. IRR is most sensitive to changes in capital expenditure.

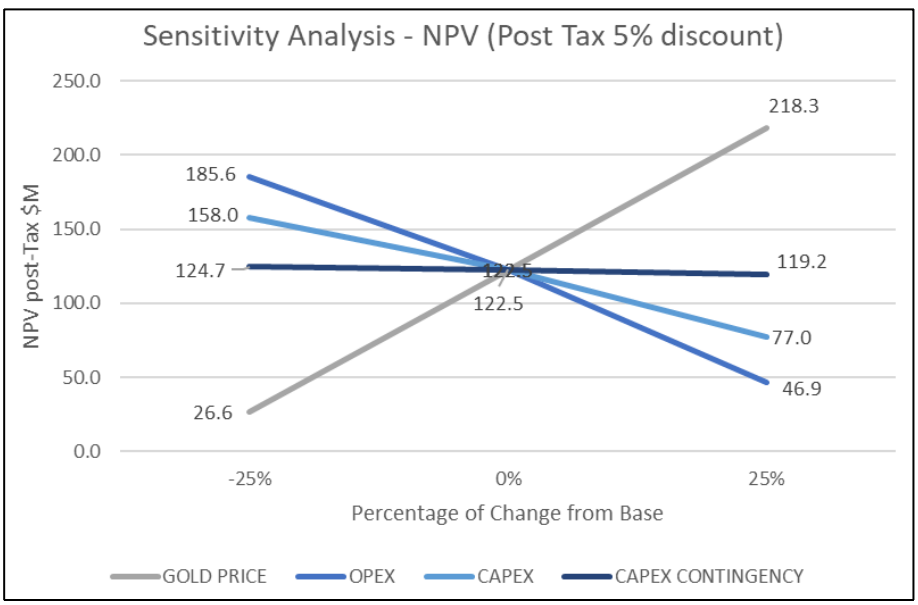


Figure 1-1: Standard financial sensitivities (NPV)

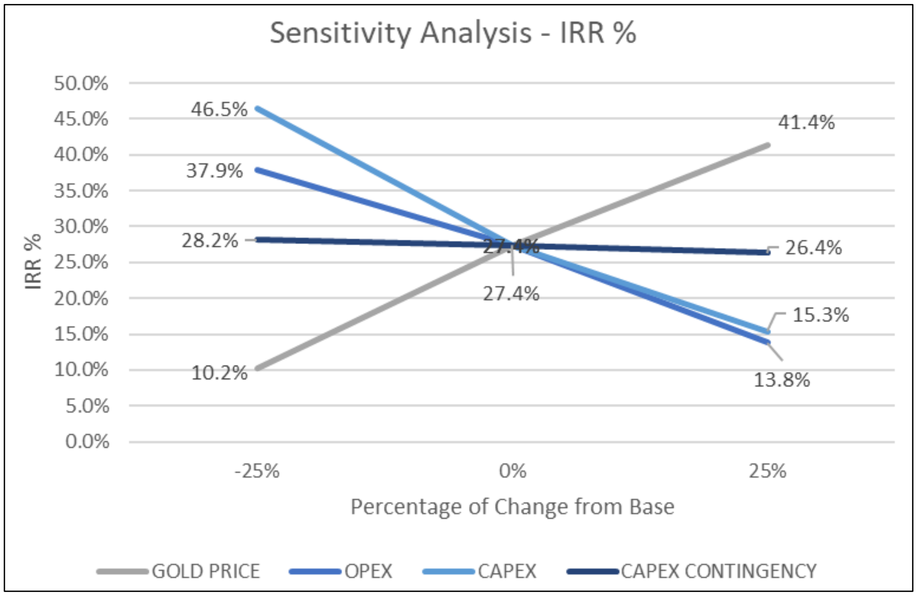


Figure 1-2: Standard financial sensitivities (IRR%)

1.16 Interpretations and Conclusions

An updated Mineral Resource estimate was prepared for the Rozino Gold Project with drilling and assay information incorporated to 15 October 2019. This Technical Report includes the first Mineral Reserve estimate for the Rozino Gold Project. The conversion of Mineral Resources to Mineral Reserves was made using industry recognized methods of determining operational costs, capital costs, mining rate and plant performance. Thus, it is considered to represent actual operational conditions of the proposed mining project. This report has been prepared with the latest information regarding environmental and closure requirements and has set out the type and extent of work required.

It is important to note that permitting for the Rozino Gold Project proposed mine is not complete. Velocity has initiated the environmental and social impact assessment process. Regulatory permitting procedures to meet Bulgarian regulations and gathering environmental data to improve the design of the Project have also been initiated. Under the Bulgarian Environment Protection Act, the development of an economically viable mining reserve requires an environmental impact assessment (EIA) which complies with European Union (EU) environmental regulations. The prospecting licence agreement for the Tintyava Property has been signed with the Minister of Energy and exploration activities have been approved by the Ministry of Environment. All necessary permits to conduct the work proposed for the property have been obtained and there are no known significant factors or risks that may affect access, title or the right or ability to perform work on the Property. There are currently no objections to the development of the Project.

The selected process path is backed by extensive metallurgical testwork. Adequate test-work data is available to provide operating parameters for flowsheet design and major equipment sizing within the contingency allowances normally associated with a Pre-feasibility Study.

The Rozino Gold Project has a comprehensive water management plan that integrates precipitation and surface flows, groundwater inflows, and water re-use with operational requirements, but particularly plant requirements.

The project includes an adequate provision for the rehabilitation and closure of the site in an environmentally sound and sustainable manner when mining operations cease. The project will provide temporary employment for between 300 to 500 construction workers and will generate up to 260 permanent jobs.

There are no known likely mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the valuation of the Project. However, the Project must not be considered to be without risk. Risks and opportunities to the mine plan not achieving the specified technical and financial parameters are detailed in the report.

1.17 Recommendations

1.17.1 Exploration and Mineral Resource Estimation

The author's recommendations for future exploration and resource definition programs are consistent with Velocity's work plan for 2020 and 2021, which targets expansion of the Indicated Mineral Resources at Rozino and definition of additional Inferred Mineral Resources in exploration areas, including mineralization intersected by exploration drilling.

Velocity's proposed work comprises two broad phases with a total budget of $1.60 million.

Phase One includes exploration drilling in the Rozino area and regional exploration within the sedimentary basins of the Ivaylovgrad Corridor, along with updating of the Rozino Mineral Resource estimates, and if supported by drilling results estimation of Inferred Mineral resources for the current exploration target areas. The exploration targets are at an early stage of evaluation and it is not certain that the proposed drilling will intersect mineralization, or lead to estimation of additional Inferred Mineral resources.

Phase Two includes additional exploration drilling of exploration targets. Contingent on positive Phase One results, Phase Two is also planned to include infill drilling within the volume of potential Inferred resources identified during Phase One.

1.17.2 Summary

All work has been completed to support the Rozino Gold Project Pre-feasibility Study. CSA Global believes that the technical outcomes and economic results of the Pre-feasibility Study support the statement of the Mineral Reserve estimate. Based on these conclusions CSA Global recommends that Velocity progress to a Feasibility Study. Prior to the Feasibility Study, additional work is recommended to enable the Feasibility Study to increase the accuracy of project planning and economic outcomes and reduce project technical risk.

The recommendations set out in the Technical Report cover the period up to the completion of the Feasibility Study and receipt of the EIA certificate. The cost of these studies and programs is estimated to be approximately $5.3 M. The exploration program set out in Section 26.1, at an estimated cost of $1.60 M, is included. Following EIA certification and a positive construction decision, the project would advance to detailed engineering and full project implementation.

Two phases are recommended, the first is longer lead preparatory works and studies to support the Feasibility Study. The second phase are the works of the Feasibility Study.

Table 1-7: Estimated cost of studies leading to the construction decision

Recommended activity/study	Estimated cost ($k)
Phase 1	
Drilling and geology	1,030
Total	**1,030**
Phase 1	
Regulatory Processes and Land Acquisition	350
Mining: Verification of Grade Control	150
Mining: Other Studies	75
Pit Geomechanical and Surface Geotechnical	250
Hydrogeology	50
Hydrology and water sourcing	150
Metallurgical and Process Optimization	375
Infrastructure	200
Environmental	75
Total	**1,675**
Phase 2	
Drilling and geology	570
Total	**570**
Phase 2	
Feasibility Study	2,000
Total	**2,000**
TOTAL	**5,275**

2 Introduction

2.1 Introduction

This Technical Report has been prepared by CSA Global Consultants Canada Ltd ("CSA Global") in accordance with National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and is suitable for filing with Canadian Securities Commissions. This document has been prepared for Velocity Minerals Ltd ("Velocity") to disclose material information related an updated Mineral Resource estimate and Pre-Feasibility Study (PFS) for the Rozino Gold Project ("Rozino" or the "Project"), located within the Tintyava Prospecting Licence ("Tintyava Property" or the "Property"). The effective date of the Technical Report is 28 September 2020. The Technical Report summarizes the results of the Pre-feasibility Study.

2.2 Scope of Work

Velocity is a Canadian-based exploration and development company focussed on an emerging gold district in southeast Bulgaria. Velocity shares are listed on the TSX Venture Exchange ("TSX:V") under the symbol VLC. A number of companies have contributed to the Technical Report. This report summarizes the work carried out by each of the companies and individuals listed below:

CSA Global

- Pit optimization and design.
- Report compilation.
- Reserve estimation and mine planning.
- Capital and operating cost estimation (mainly mining and administration).
- Economic evaluation.
- Interpretation of the results and recommendations.
- Metallurgical testwork supervision and evaluation.
- Dore metal sale costs.

Halyard Inc.

- Mineral processing.
- Site layout and facilities design.
- Capital and operating cost estimation (mainly plant and infrastructure).

MPR Geological Consultants Pty Ltd (MPR)

- Mineral Resource estimation.

Golder Associates (UK)

- Mine geotechnical testwork and evaluation.
- Site geotechnical testwork and evaluation.
- Surface and groundwater modelling.
- Geochemical testwork supervision, evaluation and modelling.
- Tailings management facility design.
- Water storage facilities design.

Mineesia Ltd

- Environmental management and regulatory requirements.

University of Mining and Geology (Sofia, Bulgaria)

- Access road design and construction.

Sharp Ideas in Mining Ltd

- ISRM drill-hole coding.

Nicolay Savov (Bulgaria)

- Power line design and specifications, routing, and regulatory requirements, capital and operating costs.

Proektirane I Analizi, PiA (Bulgaria)

- Water supply pumping and pipeline design and capita cost estimation.

Mary-An HR Agency (Bulgaria)

- Labour cost estimation.

Supporting contributions are recognized by staff from Gorubso and Velocity with particular mention to Plamen Nedyalkov, Stuart Mills, Daniel Marinov and Stefan Stamenov (consultant to Tintyava Exploration AD).

2.3 Qualified Person Responsibilities and Site Inspections

The Qualified Persons (QPs) preparing this report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in Velocity and neither are they insiders, associates or affiliates. The results of this report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Velocity and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101 and are members in good standing of appropriate professional institutions. The QPs are responsible for specific sections as follows:

Table 2-1: QP Responsibility Matrix

NI 43-101 Form F1 Item	QP Responsibility
Item 1: Summary	Individual QP responsibility for subsections in Section 1 associated to major sections. Contribution by all QPs to Sections 2 and 3. Final responsibility of Sections 2 and 3 with Andrew Sharp.
Item 2: Introduction	
Item 3: Reliance on Other Experts	
Item 4: Property Description and Location	Andrew Sharp
Item 5: Accessibility, Climate, Local Resources, Infrastructure and Physiography	
Item 6: History	Jonathon Abbott
Item 7: Geological Setting and Mineralization	
Item 8: Deposit Types	
Item 9: Exploration	
Item 10: Drilling	
Item 11: Sample Preparation, Analyses, and Security	
Item 12: Data Verification	
Item 13: Mineral Processing and Metallurgical Testing	Gary Patrick
Item 14: Mineral Resource Estimates	Jonathon Abbott
Item 15: Mineral Reserve Estimate	Andrew Sharp
Item 16: Mining Methods	
Item 17: Recovery Methods	Gary Patrick
Item 18: Project Infrastructure	Andrew Sharp
Item 19: Market Studies and Contracts	
Item 20: Environmental Studies, Permitting, and Social or Community Impact	Carl Nicholas
Item 21: Capital and Operating Costs	Andrew Sharp
Item 22: Economic Analysis	
Item 23: Adjacent Properties	
Item 24: Other Relevant Data and Information	
Item 25: Interpretation and Conclusions	25.1 Jonathon Abbott, else all other QPs
Item 26: Recommendations	26.1 Jonathon Abbott, else all other QPs
Item 27: References	Contribution by all QPs. Final responsibility with Andrew Sharp

QP visits to the Rozino Gold Project site took place as follows:

- Andrew Sharp visited the site on 30 July to 1 August 2019. The visit included review of the Project at Velocity's office in Ivaylovgrad, a review of core covering eight diamond holes, core storage, Rozino exploration site including driving to the potential dam site crest positions, both road accesses to the property, drive throughs of Rozino and Konitsi, the concentrate haulage route and the Gorubso processing facility. During the site visit in-depth discussions took place with Velocity and Gorubso main technical personnel, data and information was studied and reviewed, and information was exchanged.
- Mr. Jonathon Abbott, the Mineral Resources Qualified Person, visited Velocity's operations in the Ivaylovgrad area from 24 to 26 February 2018, including a field visit to the Rozino deposit on 25 February 2018, and inspecting original sample records and diamond drill core at Velocity's Ivaylovgrad offices on 24, 25 and 26 February 2018.
- Carl Nicholas, Qualified Person for environmental aspects visited the site 13 to 17 May 2019. The visit included visiting the Project offices, Rozino exploration site, surrounding infrastructure, core storage, and the Gorubso processing facility, including the Central Plant. During the site visit in-depth discussions took

place with Project personnel, field data and information was studied and reviewed, and relevant information was obtained.

- Mr. Gary Patrick, Qualified Person for Processing aspects, visited site 2 to 3 October 2018. The visit included review of the Project offices, Rozino exploration site, surrounding infrastructure, core storage, and the Gorubso processing facility, including the Central Plant. During the site visit in-depth discussions took place with the Project's main personnel, data and information was studied and reviewed, and information was exchanged.

Experts also completed site visits, including:

- Stuart Saich, working on behalf of Halyard for preliminary plant design aspects and metallurgical testwork reviews, visited Eurotest Control Laboratory (ETC) on 8 October 2019 and 15 October 2019, Gorubso facilities 9 October 2019, Rozino site 10 October, and Wardell Armstrong (WAI) facilities 14 October. The visit included review of the Project offices, Rozino exploration site, surrounding infrastructure, and the Gorubso processing facility, including the Central Plant and a review of testwork procedures and equipment at the metallurgical laboratories of WAI and ETC. During the site visit in-depth discussions took place with the Project's main personnel, data and information was studied and reviewed, and information was exchanged
- Xander Gwynn, Golder, visited the site on 23 - 26 April 2019. The primary purpose of the visit was the training of Velocity staff in geotechnical core logging and sampling. The visit included review of the Project offices in Ivaylovgrad, a review of core, core storage, and a visit to the Rozino exploration site. During the site visit in-depth discussions took place with Velocity technical personnel, data and information was studied and reviewed, and information was exchanged.
- Lawrence Miles, Golder, visited the site on 29 July to 2 August 2019. The primary purpose of the visit was the QA/QC of geotechnical core logging and sampling. The visit included review of the Project offices in Ivaylovgrad, a review of core, core storage, and visits to the Rozino exploration site and Eurotest laboratory in Sofia. During the site visit in-depth discussions took place with Velocity technical personnel, data and information was studied and reviewed, and information was exchanged.
- Grace Yungwirth, Golder, visited the site on 12 – 15 August 2019. The visit included a review and oversight of various hydrogeological test procedures and QA/QC, and the training of PiA personnel on testing and recording procedures. During the site visit in-depth discussions took place with Velocity technical personnel and various contractors; data and information was studied and reviewed, and information was exchanged.
- Hannah Redfern, Golder, visited the site on 12 – 17 August 2019. The visit included a review and oversight of various hydrogeological test procedures and QA/QC, and the training of PiA personnel on testing and recording procedures. During the site visit in-depth discussions took place with Velocity technical personnel and various contractors; data and information was studied and reviewed, and information was exchanged.
- Beth Ridpath, Golder, visited the site on 12 – 17 August 2019 and 1 – 6 September 2019. The visit included a review and oversight of planning for the site geotechnical investigation. Training of PiA personnel on SI procedures was provided. During the site visit in-depth discussions took place with Velocity technical personnel and various contractors; data and information was studied and reviewed, and information was exchanged.
- Angel Hodzhev, University of Mining and Geology, Sofia (MGU) had oversight of two technicians who inspected the site access road between the II-59 road and Rozino site on 8 July 2019. He did not visit the site.
- Nikolay Savov visited the site on 13 June 2019 and 11 October 2019, and also the Madzharovo substation and portions of the proposed powerline route. During the site visit in-depth discussions took place with Velocity technical personnel, data and information was studied and reviewed, and information was exchanged.

- Plamen Andreev (Proektirane I Analiza, Sofia – PiA) visited the site on 8 July 2020 and 22 July 2020 to undertake an inspection of external water supply sources (in the vicinity of the Rozino Project site) and potential pipeline routes.

2.4 Sources of Information

Sources of information include data and reports supplied by Velocity personnel as well as documents cited throughout the report and referenced in Section 27. Background Project information was sourced from the most recent revised Technical Report entitled "Preliminary Economic Assessment – Rozino Project, Tintyava Property, Bulgaria" prepared by CSA Global with an effective date of 20 September 2018.

3 Reliance on Other Experts

The report relies on other experts for the description of Tintyava Property tenure and ownership. These aspects are detailed and referenced in relevant sections of the report, and listed below:

- Section 4: The description of mineral tenure and Property ownership relies upon Toneva & Todorova (2021).
- Section 6**Error! Reference source not found.**: The description of mineral tenure and Property ownership relies upon Toneva & Todorova (2021).

The report authors are not qualified to comment on any legal considerations relating to the tenure status of the Tintyava Property and expresses no opinion as to the ownership status of the Tintyava Property. The report authors have not independently verified the legal status of Velocity's agreements with Gorubso-Kardzhali AD ("Gorubso").

No warranty or guarantee, be it express or implied, is made by the report authors with respect to the completeness or accuracy of the mineral tenure comprising the Tintyava Property.

The report relies on Manol Krivoshapkov (Bulgarian registered auditor #428) for the estimation of taxation, depreciation, amortization and Tintaya Exploration ED losses carried forward. This information was utilized in Section 22 and the information was supplied in a letter dated August 10, 2020.

4 Property Description and Location

4.1 Property Location and Description

The Rozino Project is located within the Tintyava Property, which is in the municipalities of Ivaylovgrad and Haskovo, southeast Bulgaria, about 350 km (by road) east-southeast of the capital, Sofia (Figure 4-1). To the east and south is the border with Greece and to the north and west are the municipalities of Lyubimets, Madzharovo and Krumovgrad. The Tintyava Property has an area of approximately 145 km^2 (14,500 hectares) and its centroid is located at longitude 41° 27' 28"N, latitude 25° 52' 22"E, approximately 25 km west-southwest of the border town of Ivaylovgrad and 10 km east of Krumovgrad town.

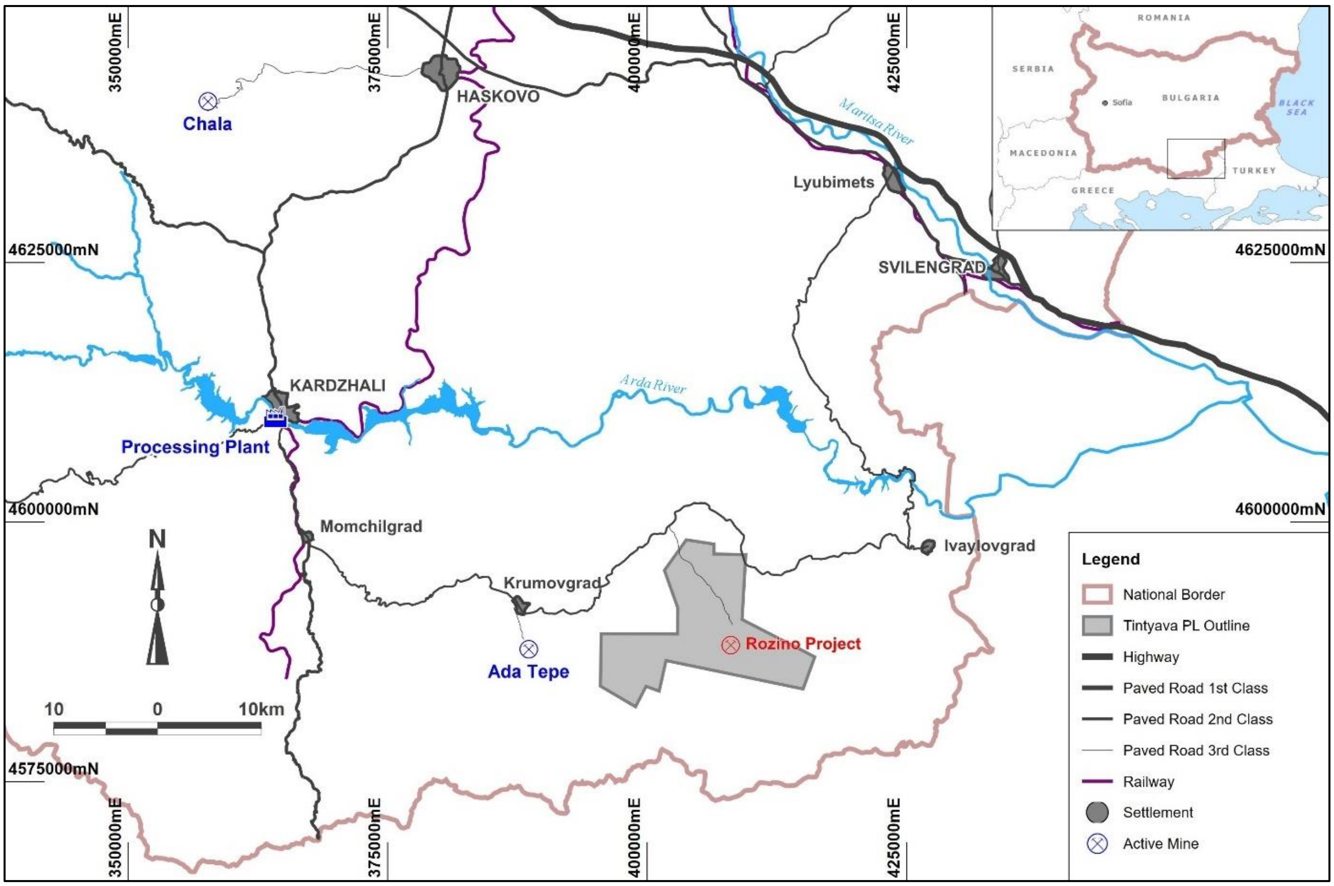


Figure 4-1: Location map of the Rozino Project, south eastern Bulgaria
Source: Velocity, 2020

A number of gold occurrences and prospects occur within the Tintyava Property, the most advanced of which is Rozino. The Rozino Project is located approximately two km south of the village of Rozino, 35 km (by road) east of the Ada Tepe gold mine (operated by Dundee Precious Metals Inc.), which is described in more detail in Section 23 (Adjacent Properties), and 85 km (by road) southeast of the city of Kardzhali. The existing CIL processing plant ("Central Plant"), to which the Rozino concentrate will be transported, is operated by Gorubso and located in Kardzhali.

4.2 Mineral Tenure

In 2016 Gorubso won a competitive tender to acquire the Tintyava Property. Gorubso and the Bulgarian Ministry of Energy and Waters entered into a prospecting and exploration licence agreement dated 2 May 2017, pursuant to which the Tintyava Property (prospecting licence number 467) was issued. The prospecting licence agreement gives the holder the exclusive right to explore for metal ores both on the surface and at depth within a certain parcel of land described by a set of coordinates. The Tintyava Property was subsequently transferred to Tintyava Exploration AD ("Tintyava Exploration"), a wholly-owned subsidiary of Gorubso.

On 30 May 2017, Velocity (through its wholly owned subsidiary Kibela Minerals AD ("Kibela")) and Gorubso entered into an option agreement. Pursuant to the terms of the option agreement, Velocity acquired the exclusive option (the "Option") to acquire a 70% interest in Tintyava Exploration and the Tintyava Property. The Option being exercisable through delivery of a PEA, within the meaning of NI 43-101.

On 4 September 2018, Velocity and Gorubso entered into an Exploration and Mining Alliance Agreement (the "Alliance"). The Alliance covers all existing and future Gorubso and Velocity projects (including the Tintyava Property) within an area of 10,400 km^2 covering the prospective Eastern Rhodope Gold Mining District in southeastern Bulgaria (the "Alliance Area"). Through the Alliance, Velocity has exclusive access to a CIL gold processing plant (the "Central Plant") and has negotiated options to earn a 70% interest in various near-surface gold projects located within the Alliance Area.

On 31 October 2018, Velocity delivered a PEA Technical Report prepared under National Instrument 43- 101 of the Canadian Securities Administrators ("NI 43-101") to Gorubso. Following delivery of the PEA, Velocity was deemed to have earned an undivided 70% interest in Tintyava Exploration and the Tintyava Property.

On 1 March 2019, Velocity (through Kibela) entered into a shareholder's agreement with Gorubso regarding Tintyava Exploration and 70% of the shares of Tintyava Exploration were transferred to Kibela. The participating interests of Velocity and Gorubso in the JV are 70% and 30% respectively. If either Velocity or Gorubso does not contribute its portion of expenditures, then that party's interest in the JV will be diluted and if reduced to 10% or less, will convert to a 1% NSR royalty.

4.3 Datum and Projection

The coordinate system used in this section and throughout this report is WGS 84 Zone 35N.

4.4 Prospecting Licence 467

The Tintyava Property, covered by Prospecting Licence 467, has an area of approximately 145 km^2. The corner pillar coordinates for the Tintyava Property are listed in Table 4-1 and a detailed map of the Tintyava Property is provided Figure 4-2.

Table 4-1: Corner coordinates for the Tintyava Property, Prospecting Licence 467

Point ID	X (UTM 35N)	Y (UTM 35N)
1	403790	4598274
2	406456	4597856
3	406470	4596911
4	409518	4596835
5	409161	4590598
6	416164	4586995
7	415036	4583906
8	401838	4586644

Point ID	X (UTM 35N)	Y (UTM 35N)
9	401180	4583698
10	399914	4584156
11	398896	4582194
12	395513	4584188
13	395598	4589222
14	402960	4589195
15	402968	4591723
16	401686	4595330

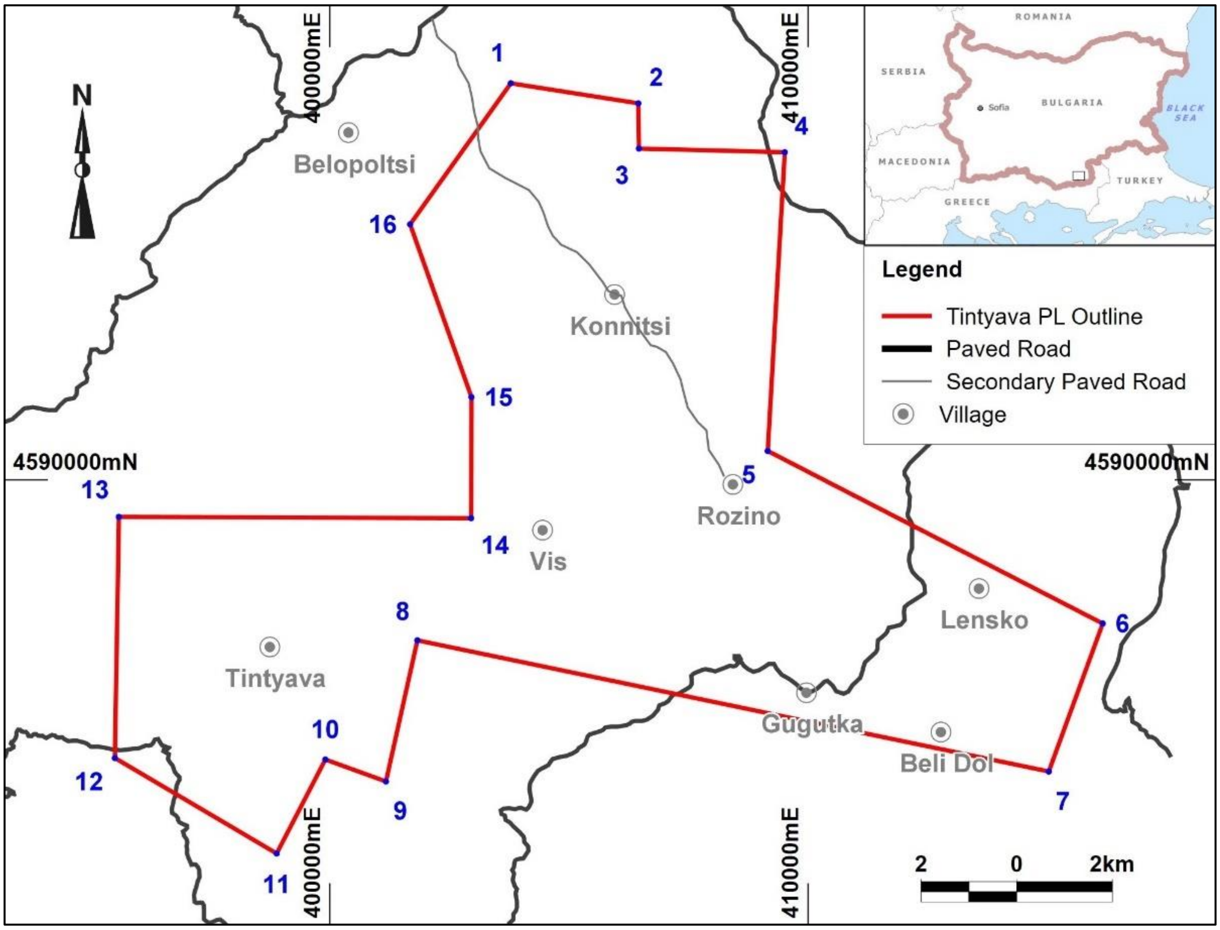


Figure 4-2: Map of Tintyava Property, south-eastern Bulgaria Showing corner labels referenced in Table 4-1
Source: Velocity, 2020

4.5 Royalties

A concession royalty ("Royalty") is payable to the Bulgarian state. The Royalty is determined at the time of the granting of a mining concession based upon projected profitability of the operation and in line with the mining plan submitted to the government. The Royalty is the price paid by the concessionaire to the concession grantor for the right to extract resources by mining. The Royalty rate is variable and based on the gross revenue generated by the sale of the metals contained in the ore. The Royalty rate is a function of profitability and is determined principally on annual pre-tax profit (but taking account of depreciation and impairment costs).

Article 4 of the Bulgarian Regulation on the Principles and Methodology for Determination of the Concession Royalties for Mining of Ores and Minerals under the Mineral Resources Act describes the method for determining of the Royalty amount. At a pre-tax profit to sales ratio equal to or less than 10%, the Royalty rate will be 0.8 % of the gross metal revenue. At a ratio equal to or greater than 50%, the Royalty rate will be 4% of the gross metal revenue. At intermediate levels of profitability, the Royalty rate will vary on a sliding scale between 0.8% and 4%, increasing by 0.08% for every 1% increase in the profitability ratio.

For the purposes of this Report, the Royalty payable to the Bulgarian state is estimated to be 2% of the net smelter return over the life of the mining operation.

4.6 Surface Rights

The Bulgarian State Forestry ("State Forestry") controls the majority of the surface rights within the Project area. Approximately 99% of the proposed mining infrastructure design is within State Forestry lands, with only a small proportion of the land is controlled by private landowners. On receipt of a mining concession, a contract will be required to transfer the surface rights to Tintyava Exploration for the term of the concession. In Bulgaria, if no agreement can be reached with the existing incumbent of the surface rights, the matter may be passed to the respective authorities to facilitate purchase.

4.7 Permitting

The Tintyava Property prospecting and exploration licence agreement (dated 2 May 2017) defined a three-year work program. The Ministry of Environment granted approval of the work program on July 3, 2017, which allowed exploration to commence.

Following completion and fulfillment of the three-year work program (31 July 2020), the Tintyava exploration licence agreement was extended for an additional two years. During the extension period Tintyava Exploration is committed to spend at least BGN 2.6M (approximately CAD$2.1M), which includes various exploration commitments, the completion of Prefeasibility and Feasibility Study reports, and the undertaking of an environmental assessment. The extension of the prospecting licence is valid until 31 July 2022, in advance of which an application to extend for an additional two years may be submitted to the Ministry of Energy.

Additional permits received from the local Ivaylovgrad municipality and State Forestry allow exploration drilling, other exploration activities and engineering work to be completed.

The JV holds all necessary permits to conduct proposed exploration, environmental studies and related activities on the Property. There are no known significant factors or risks that may affect access, title or the right or ability to perform work on the Property.

Prior to the development of the Rozino Project an Environmental Impact Assessment (EIA) will be prepared. Discussion of the EIA and other permitting requirements related to exploitation are presented in Section 20 of this report.

4.8 Environmental Liabilities

Other than ancient, small-scale workings, there has been no historical mining activity on the Tintyava Property and there are no known environmental liabilities on the Property.

An environmental bond, guaranteed by Gorubso via First Investment Bank, is held over the Tintyava Property. This bond guarantees the rehabilitation of any environmental damage due to exploration activities. Reclamation, such as rehabilitation of drill sites, is completed on an ongoing basis.

5 Accessibility, Climate, Local Resources, Infrastructure and Accessibility

5.1 Accessibility

The Tintyava Property is located approximately 350 km by road east-southeast of Sofia (Figure 5-1).

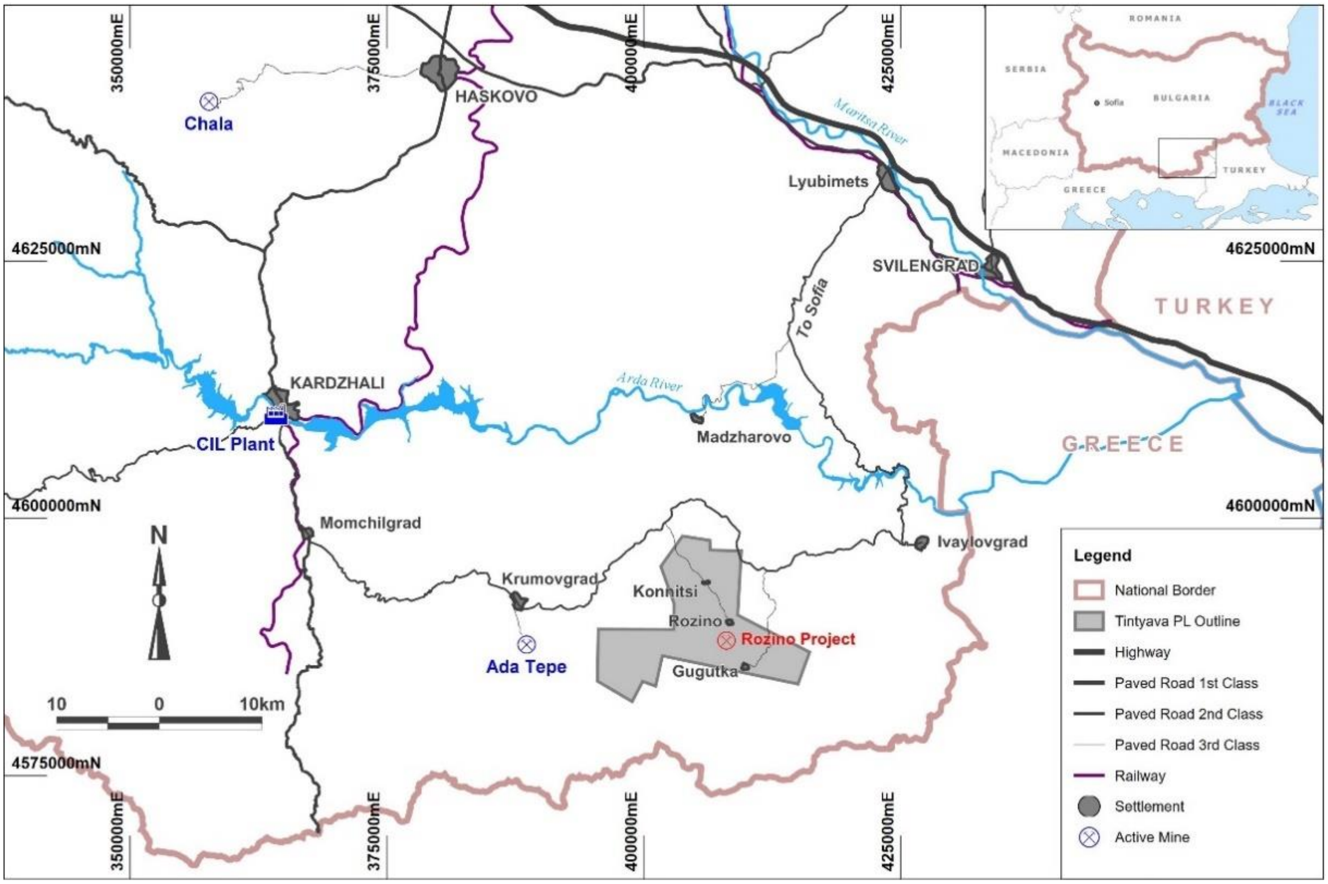


Figure 5-1: Regional location map
Source: Velocity, 2018

The main provincial road, designated II-59, is paved and passes through the towns of Kardzhali, Momchilgrad, Krumovgrad and Ivaylovgrad. The Rozino Project access road intersects the II-59 approximately 22 km east of Krumovgrad. The distance from the intersection with the II-59 and Rozino village is 12 km and passes through Konnitsi village about seven km's from the intersection. This is a paved, single lane municipal road that is in reasonable condition. The final two km's between Rozino village and the Rozino Project site is a gravel road. Exploration tracks and forestry roads currently provide year-round access within the property (Figure 5-2).

The Project site is also accessible from Gagutka village, approximately five km's to the southeast via a gravel road in poor condition. The distance between Ivaylovgrad and Gagutka is approximately 35 km by a paved road.

Personnel currently commute daily from Ivaylovgrad where Velocity has a project field office. The distance is approximately 50 km by paved road to Rozino village.

The site is accessible year-round except possibly during extreme storm events or short periods of snow cover.

Figure 5-2: View from approach road to the Rozino deposit (middle distance summit), view to south
Source: Velocity, 2018

5.2 Climate and Physiography

The Project area's average annual temperature is 12°C, ranging from about 2°C in January to 24°C in July. Maximum rainfall occurs during November and December, with rainfall of up to 100 mm per day. Snow is sporadic, lasting generally only five to ten days of the year. Exploration activities can be undertaken throughout the year.

The local terrain is characterized by low mountains and predominantly levelled hills and is cut by steep valleys. The Rozino Project site is bounded to the south by steep cliffs at Tashlaka. The terrain is dissected by the Byala reka (White river) and its tributaries. In the project area elevations range from about 300 to 480 masl in the north, reducing to approximately 300 masl in the south.

State controlled forestry is the main land use at Rozino. The licence area is mostly covered by indigenous and cultivated forest comprising European oak and black pine. Within the forest are scattered clearings of grassland which, in the past, may have been used for pasture.

5.3 Local Resources

Villages and hamlets with very small populations are dispersed throughout the licence area (Figure 5-1). Inhabitants are primarily involved in subsistence farming, particularly livestock and the growing of tobacco. The other main land use within the licence area is state-controlled forestry. The majority of the population lives in the towns of Ivaylovgrad and Krumovgrad. Rozino and Konnitsi are largely deserted with only a handful of families remaining.

A labour market study undertaken for Velocity indicates that the towns of Ivaylovgrad (population 3,800) and Krumovgrad (population 8,700), will become the main sources of unskilled or semi-skilled labour. Skilled and management personnel will be sourced from throughout Bulgaria where possible.

5.4 Infrastructure

The main provincial road, II-59, serves as the main access to the region. As described above, this is a paved road (2nd Class); it is in good condition and capable of carrying heavy commercial vehicles.

Local villages are electrified with a 22 kV supply, stepped down from a 110 kV main distribution line with an off-take at the Madzharovo substation approximately 22 km north of the Project site. Power to the Madzharovo substation is supplied by the 85 MW Studen Kladenets and the 217 MW Ivaylovgrad hydroelectric power stations.

All villages have access to fresh water through a network of reservoirs and pump stations.

Figure 5-3: Konitsi hamlet with powerline and sealed road, Rozino Project in distance – view to southwest
Source: CSA Global, 2019

6 History

The following summary of the Project's history is derived from the cited references, Hogg, 2017 and notes supplied by Velocity.

Modern exploration of the Tintyava Property commenced in the 1980's with work first being completed by Geoengineering who drilled 86 vertical diamond drill holes for 14,289 metres.

Hereward began exploration in 2001 and between 2004 and 2007 completed three drilling phases totalling 7,995 metres including 2,733 metres undertaken in joint venture with Asia Gold. Additional work completed during this period included surface mapping, trenching and metallurgical test-work.

In 2009, the original Prospecting Licence (PL) containing the Rozino deposit, which was of different extents to the current PL was due for expiry and Hereward in joint venture with Caracal Gold LLC, through a local company Cambridge Caracal Bulgaria EAD ("Caracal") submitted an application for Commercial Discovery in order to maintain their rights for the deposit. In order to minimize environmental permitting Caracal submitted a small underground mine design. The application was rejected by the Bulgarian government, who considered that an open pit mine design was required and despite extensive discussions between the parties, in 2013 the original PL was cancelled.

A new PL application was lodged in 2013 and a number of companies registered an interest in the Property, which triggered an automatic public tender procedure. In 2016 Gorubso-Kardzhali AD ("Gorubso") won a competitive tender for exploration rights to the Property and an exploration agreement was signed in April 2017.

As part of an earn-in option agreement Velocity began exploration in July 2017 and in February 2018 the Property was formally transferred from Gorubso to Tintyava, Exploration EAD a company set up to become the Joint Venture vehicle for Velocity and Gorubso.

There has been no historical production at the Property.

7 Geological Setting and Mineralization

7.1 Regional Geological Setting

The following summary of the Project's regional geological setting is derived from the cited references, Hogg (2017) and notes supplied by Velocity.

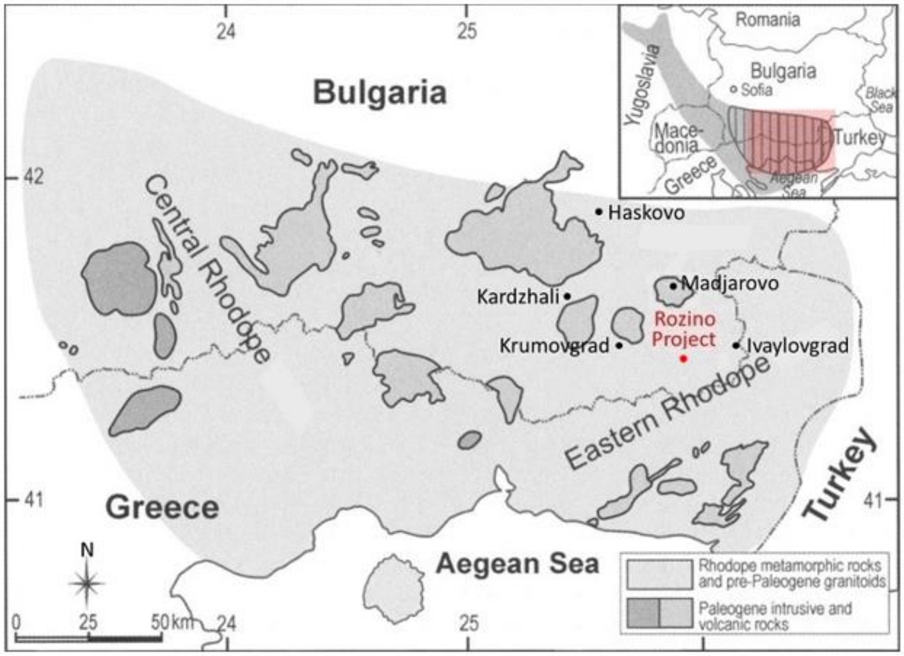
Tintyava lies within the Eastern Rhodope mineralization district of south-eastern Bulgaria which is located within an Eocene-Oligocene continental magmatic belt extending for about 500 km from Serbia and Macedonia to northwest Turkey. The eastern part of this belt is occupied by the Rhodope Massif, which comprises Precambrian to Mesozoic metamorphic rocks and Palaeogene magmatic rocks. Metamorphic rocks of the Rhodope basement comprise interlocking core complexes, such as the Kessebir and Biala Reka domes that are made up of two major tectonostratigraphic complexes; a gneiss-migmatite complex and a variegated complex (Figure 7-1).

The structurally lower gneiss-migmatite complex, which crops out in the core of the Kessebir metamorphic dome, is dominated by igneous protoliths. These include metagranites, migmatites and migmatised gneisses overlain by a series of pelitic gneisses, and rare amphibolites formed from Variscan or older continental basement. The overlaying variegated complex consists of a heterogeneous assemblage of pelitic schists, para-gneisses, amphibolites, marbles and ophiolite bodies with metamorphosed ophiolitic peridotites and amphibolitised eclogites intruded by gabbros, gabbronorites, plagiogranites and diorites. The variegated complex is intruded by volumetrically minor Upper Cretaceous plutonic bodies.

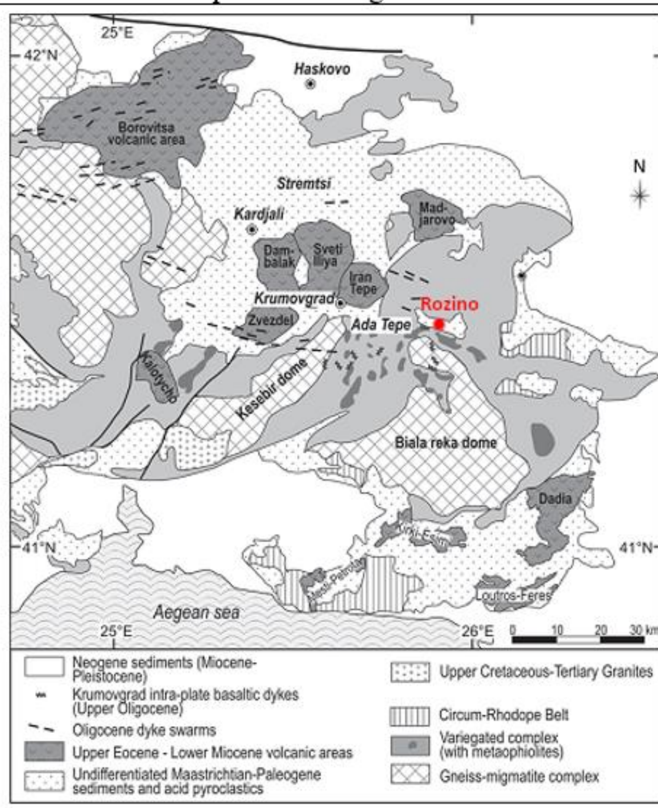
Palaeogene magmatic rocks of the Rhodope basement consist of calc-alkaline to shoshonitic intermediate, acid and subordinate basic volcanic rocks and their intrusive equivalents. The Palaeogene magmatism was accompanied by the formation of small copper-molybdenum porphyry deposits and abundant epithermal deposits (Mutafchiev and Skenderov, 2005).

Lava flows and domes of the ~35 Ma andesites of the Iran Tepe volcano are exposed northeast of Krumovgrad. This magmatic activity was followed by scarce latitic to rhyolitic dykes in the northern part of the Kessebir dome, and finally by intra-plate basaltic magmatism in the southern part of the dome (Marchev et al, 2004).

Rocks of the variegated complex are locally overlain by the Maastrichtian to Palaeocene age syn-detachment Shavarovo Formation, which is in turn overlain by Upper Eocene– Lower Oligocene coal-bearing-sandstone, syn-tectonic breccia conglomerates and marl-limestone formations.



Eocene volcanic complexes within the Palaeogene intrusive and volcanic belt, Rhodope Mountains. Inset shows the Palaeogene Macedonian-Rhodope-North-Aegean Volcanic Belt. After Marchev, 2004



Schematic geological map of the Eastern Rhodope showing metamorphic dome structures, major volcanic areas and dyke swarms. After Marchev et al, 2004

Figure 7-1: Regional geological setting

7.2 Tintyava Property Geological Setting

The following summaries of the Tintyava Property's geological setting is derived from Hogg (2017) and notes supplied by Velocity.

Geology of the Rozino area is interpreted to comprise a series of discrete Palaeogene syn-tectonic pull-apart sedimentary basins within metamorphic basement (Figure 7-2). The basements are controlled by northeast trending extensional faults and a major west – northwest trending dextral strike slip deep seated fault zone. This fault zone, which has been identified over more than 15 kilometres is locally known as the Byala Reka Shear Zone ("BRSZ") and shown as thick blue line in Figure 7-2. The basins lie within a zone designated at the Ivaylovgrad Corridor (Figure 7-5).

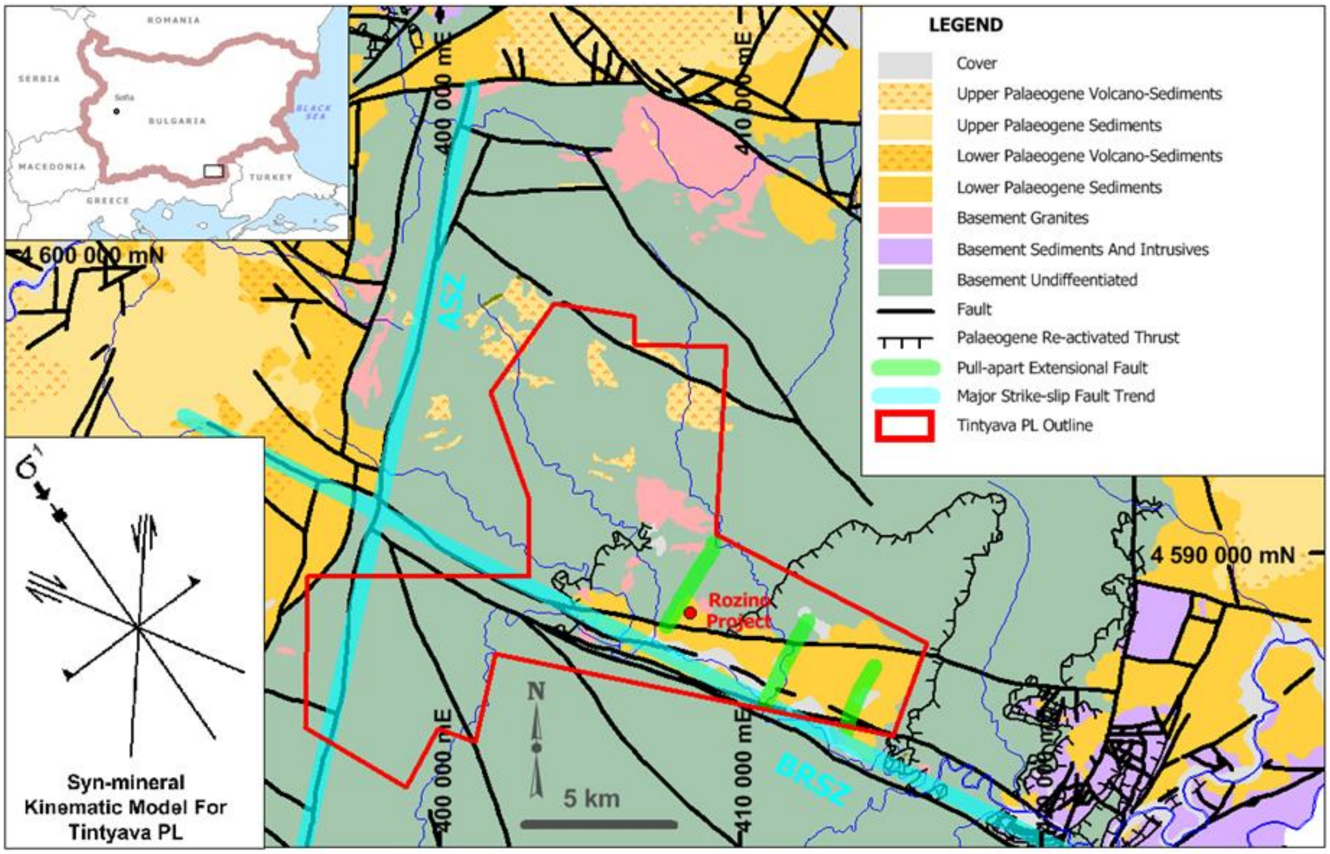


Figure 7-2: Tintyava Property geological setting
Source: After Goranov et al, 1995. Courtesy Velocity, September 2020

A major north northeast sinistral strike slip fault, locally known as the Avren Shear Zone ("ASZ") in the west of the Property hosts subparallel Palaeogene rhyolite dykes, suggesting that igneous activity was contemporaneous with basin development.

The schematic plan and section in Figure 7-3 and Figure 7-4 respectively represent Velocity's interpretation of the Tintyava Property's geology and show interpreted mineralization and conceptual target areas for future exploration. In this figure, mineralization intersected by drilling is schematically shown hatched in red and targets untested by drilling are shown as red outlines. The interpreted steep dips of the sediment package and basement contacts in the target areas are consistent with post mineralization tectonic disruption.

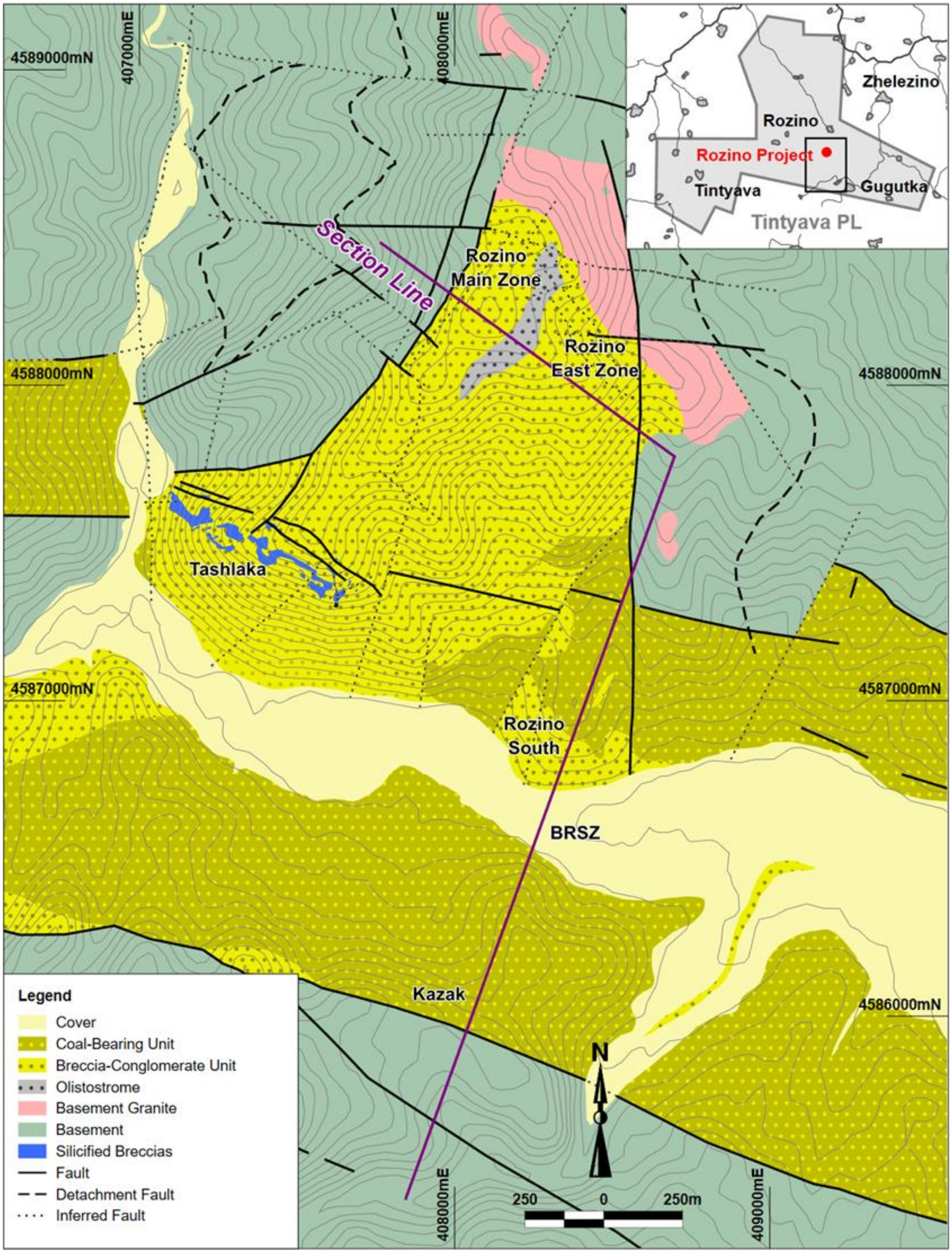


Figure 7-3: Tintyava Property schematic geology plan
Source: Courtesy Velocity, September 2020

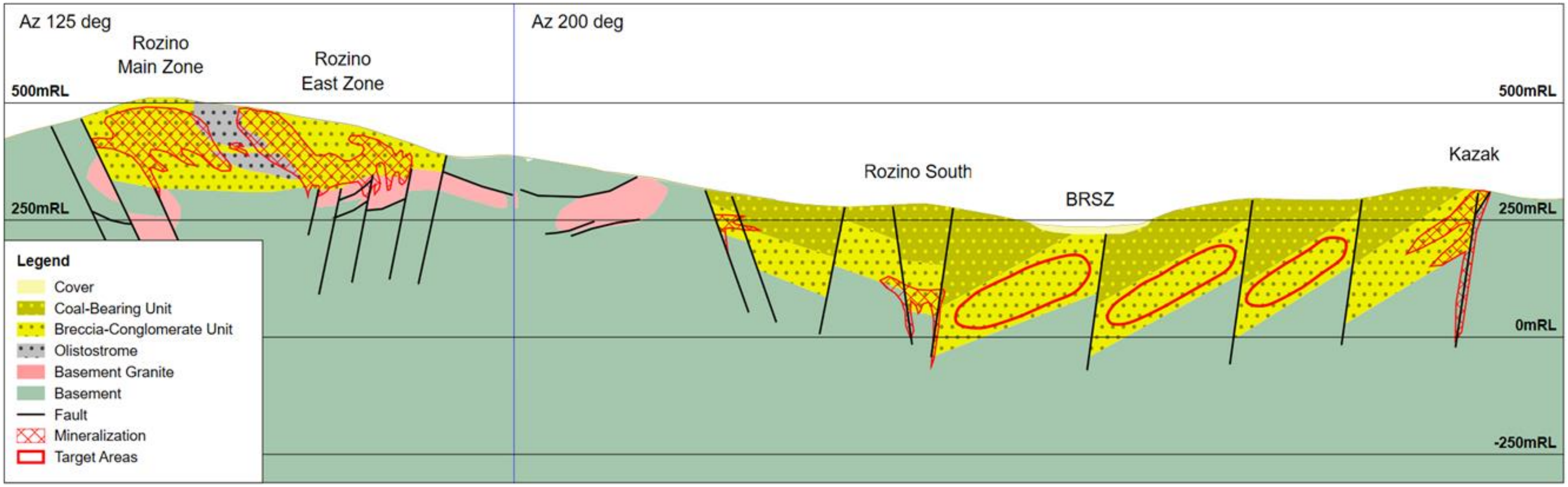


Figure 7-4: Tintyava Property schematic geology and mineralization section (section line shown in Figure 7-3)
Source: Velocity, 2020

Figure 7-5 shows geological domains interpreted for the Tintyava Property overlain on a topographic relief map with large drainages in blue. The discrete Palaeogene sedimentary basins within the Ivaylovgrad Corridor are in khaki and the larger Palaeogene rhyolite bodies are in pink. Small rhyolite dykes are present throughout the Property. The red lines show exploration targets

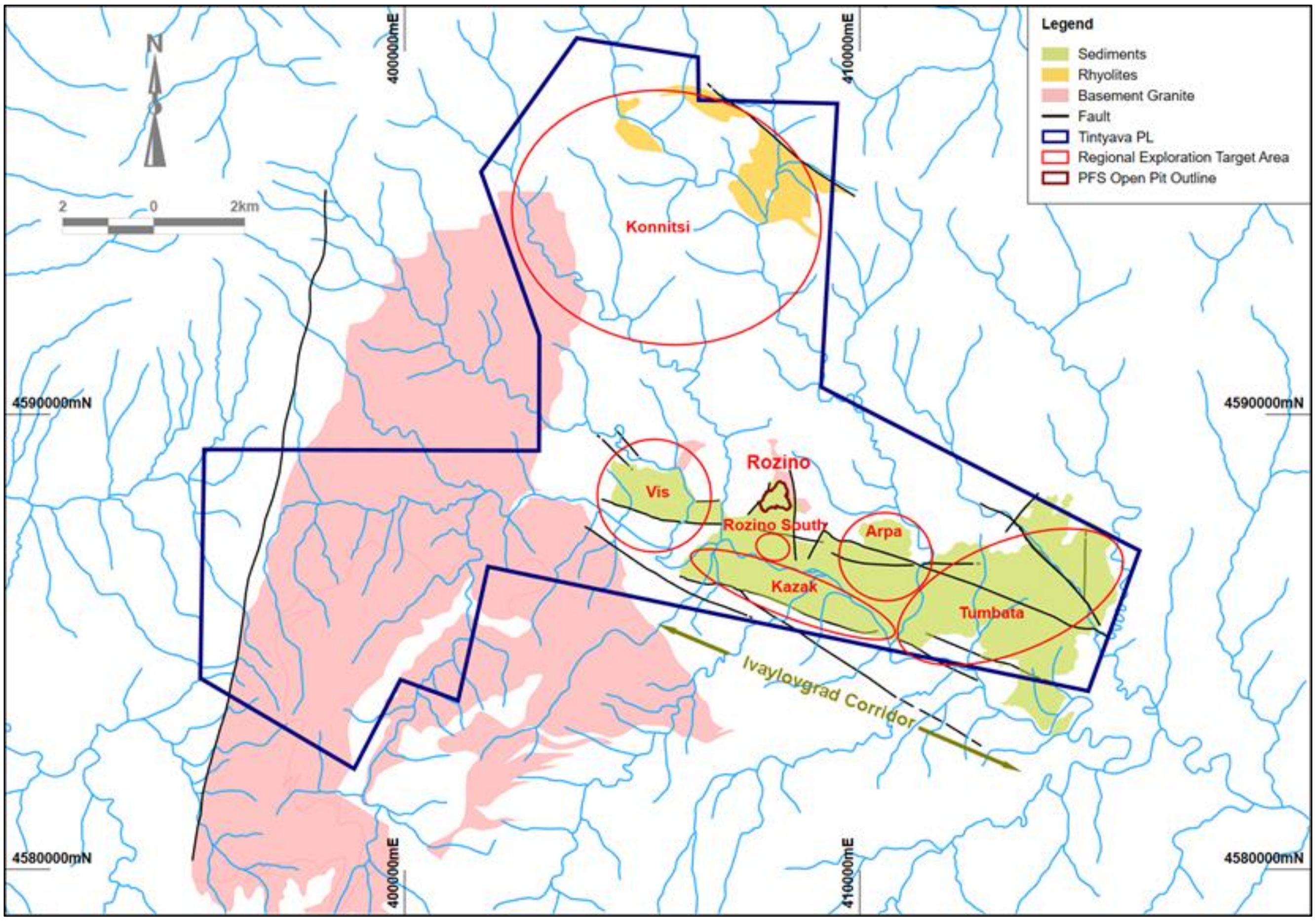


Figure 7-5: Schematic geological domains for the Tintyava Property
Source: Velocity, 2020

Velocity has identified telethermal alteration at Tashlaka, characterized by hot springs silicification and associated pathfinder geochemistry such as arsenic, antimony, barium, mercury and silver. Although no significant gold grades have been noted in this area, the identified alteration aids geological understanding of the Property, and exploration targeting. This alteration is interpreted to indicate that despite lying at a lower topographic level than the Rozino deposit, the alteration represents higher levels of the hydrothermal system. This is indicative of as yet undefined post mineralization extensional faulting between Tashlaka and the Rozino deposit.

7.3 Rozino Mineralization

The following summaries of the Rozino mineralization is derived from Hogg (2017) and notes supplied by Velocity.

Rozino is a low sulphidation epithermal gold deposit hosted within generally brecciated and conglomeratic Palaeogene sedimentary rocks as disseminations, replacement and vein mineralization. The mineralogy consists mainly of pyrite with traces of base metals and rare arsenopyrite. Gold is present at sulphide mineral boundaries and to a lesser degree as free grains or encapsulated inclusions. Gangue minerals consist of silica, iron carbonates and adularia. Alteration is characterized by a quartz, carbonate, chlorite, adularia, pyrite assemblages.

The dominant mineralization trend is northwest and parallel to the regional extensional fault regime with local mineralization development controlled by the intersection of steep structures sub-parallel to the bounding extensional faults, and gently dipping bedding. Mineralization is interpreted to have been emplaced around 100 to 150 metres below the palaeotopographic surface and related to shallow extensional faults representing reactivation of pre-existing steep structures.

Rozino mineralization controls include olistostrome bodies of metamorphic basement that effect the distribution of mineralization within the sedimentary basin Brecciated boundaries of the olistostrome bodies are generally more strongly mineralized than their massive cores which are generally poorly mineralized. This relationship is demonstrated by Figure 7-6, which shows a three dimensional wire-frame view of the olistostrome bodies interpreted by Velocity and Figure 14-4 which shows the olistostrome interpretation relative to the current resource model at 0.5 g/t cut off. The association between the olistostrome bodies and distribution of gold grades is demonstrated by Figure 7-7 which shows average mineralized domain composite grades for ten metre increments of their distance to the olistostrome boundary. These figures are based on a wire-frame representing the olistostrome interpretation provided by Velocity in September 2020.

Drilling has intersected mineralization over an area about 800 metres by 1,000 metres to a vertical depth of around 195 metres. The mineralization is interpreted to be completely oxidized to average depth of about 11 metres, with fresh rock occurring at an average depth of about 22 metres.

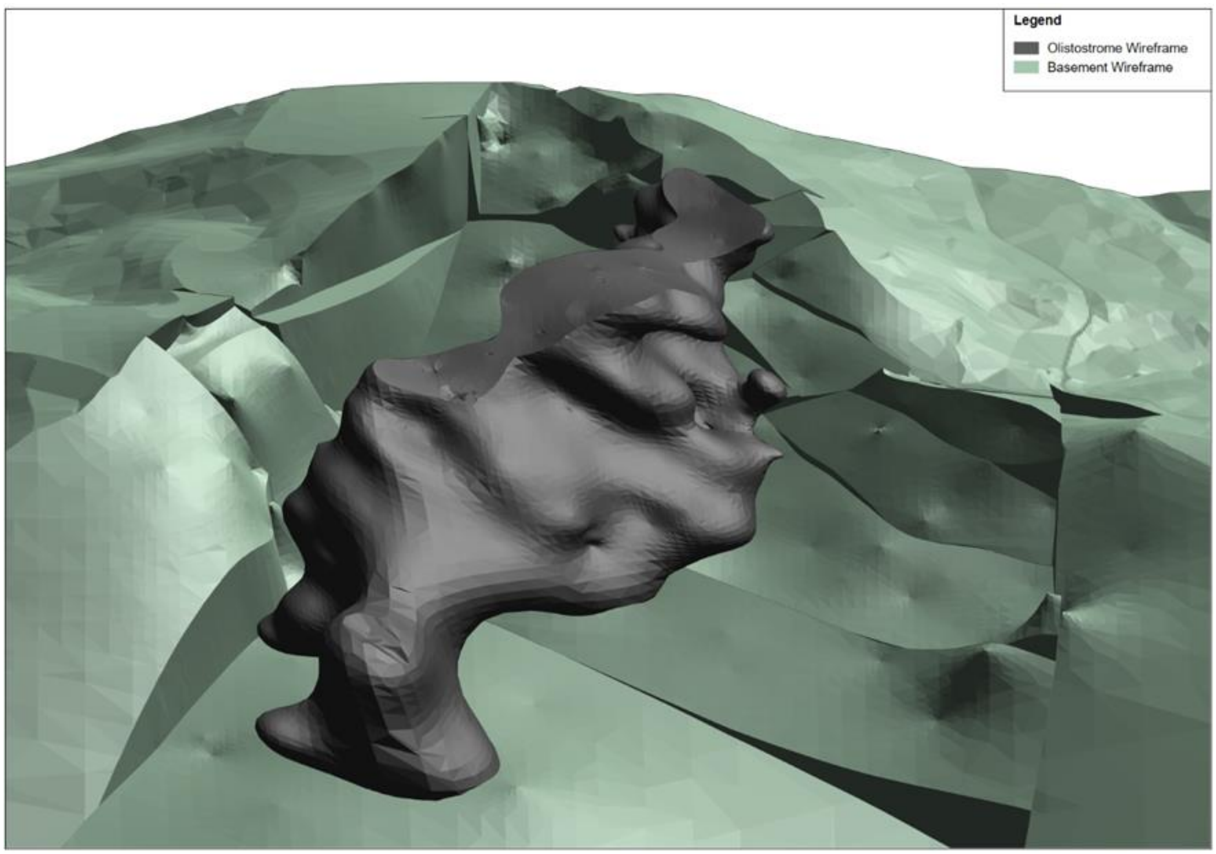


Figure 7-6: Rozino sedimentary basin and olistrome
Source: Figure supplied by Velocity, 2020

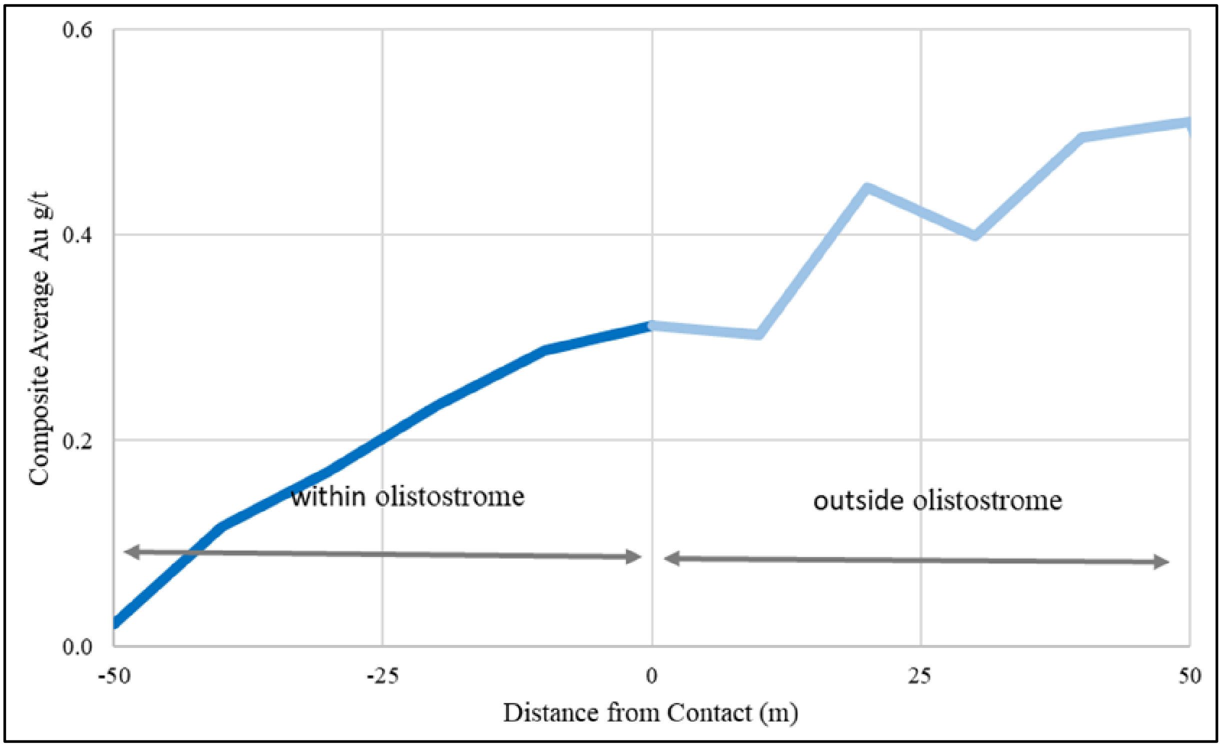


Figure 7-7: Average composite grade versus distance from olistostrome boundary

8 Deposit Types

Rozino is a LSE gold deposit hosted within breccia and conglomerate Palaeogene sediments as disseminations, replacement and vein mineralization. The dominant mineralization strike is northwest, parallel to the regional extensional fault regime, with local mineralization development controlled by the intersection of steep structures sub-parallel to the bounding extensional faults, and gently dipping bedding.

Velocity's exploration focussed on the dominant northwest trend of veins within the Palaeogene sediments. The veins appear to be controlled by steep structures interpreted to extend into the basement.

Drilling to date has not intersected significant gold mineralization within the basement. However, drilling has intersected crustiform gold and base-metal bearing, northwest striking LSE veins within the basement. These veins are generally narrow and rarely wider than 10 centimetres. The LSE hydrothermal fluids were confined within the impermeable basement and these non-reactive fluid pathways have very narrow alteration selvedges that are difficult to detect by drilling.

Upon reaching the basal unconformity the hydrothermal fluids would have de-pressurized and throttled boiling is the interpreted mechanism for gold deposition. The poorly consolidated breccia conglomerate sediments are interpreted to have been wet, further neutralizing the hydrothermal fluid, creating disseminated gold haloes peripheral to the boiling zones. Where hydrothermal pathways intersect coarse sandstones, stockwork quartz carbonate veins are developed at the expense of disseminated mineralization (Figure 8-1).

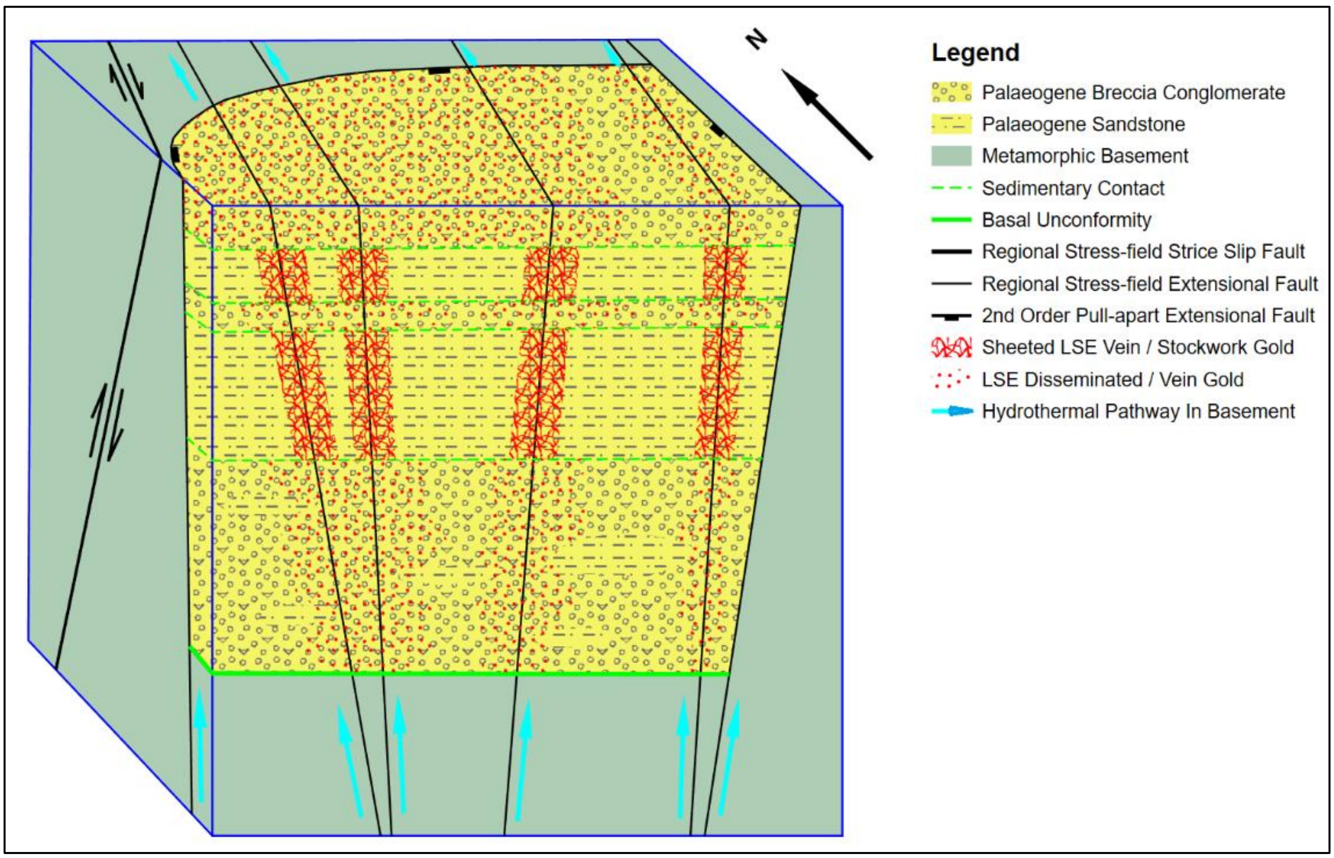


Figure 8-1: Rozino mineralization schematic model
Source: After Hogg 2017. Courtesy Velocity, September 2020

Velocity consider that all untested parts of the Palaeogene basins in the Tintyava Property have the potential to host epigenetic, LSE mineralization analogous to that at Rozino, making the basin sediments to the south of Rozino targets for further exploration.

Velocity's exploration of Tintyava Property has included soil sampling of areas within the BRSZ where similar rocks to those hosting the Rozino deposit outcrop, including the Rozino South and Kazak areas. For large areas of the Property, the prospective rock units are overlain by younger sedimentary rocks, recent alluvial cover or both. Velocity considers that for such areas, any potential mineralization is likely to be geochemically blind to soil sampling, and that deeper drilling is an appropriate exploration approach.

The schematic plan and section Figure 7-4 and Figure 7-5 represent Velocity's interpretation of the Tintyava Property's geology and shows interpreted mineralization and conceptual target areas for future exploration. In this figure, mineralization intersected by drilling is schematically shown hatched in red and targets untested by drilling are shown as red outlines. The interpreted steep dips of the sediment package and basement contacts in the target areas are consistent with post mineralization tectonic disruption.

The target areas shown in Figure 7-5 include the following:

- Rozino South, where the exploration target includes breccia conglomerates and underlying basement.
- The BRSZ exploration target, which includes breccia conglomerate beneath recent transported alluvial cover and the formational coal-bearing unit.
- The Kazak exploration target, which is hosted in steeply dipping sediments and the basement and mineralization has been sheared, indicating post-mineralization tectonism.

Velocity has identified telethermal alteration at Tashlaka, characterized by hot springs silicification and associated pathfinder geochemistry including arsenic, antimony, barium, mercury and silver. Although no significant gold grades have been noted in this area, the identified alteration aids geological understanding of the Property, and exploration target. This alteration is interpreted to indicate that despite lying at a lower topographic level than the Rozino deposit, the alteration represents higher levels of the hydrothermal system. This is indicative of as yet undefined post mineralization extensional faulting between Tashlaka and the Rozino Deposit.

9 Exploration

9.1 Work Completed

9.1.1 Introduction

This report includes exploration sampling available for the Tintyava Property on the 28th of September 2020.

Modern exploration of the Tintyava Property commenced in the 1980's by Geoengineering. Velocity's current exploration activities and planning do not include the results from previous tenement owners. Historic exploration sampling by Geoengineering, Hereward and Asia Gold is not relevant to current exploration or Mineral Resources and is not detailed in this report.

Table 9-1 summarizes key exploration activities undertaken within the Property and Figure 9-1 shows the sampling locations relative to the extents of the Tintyava PL and surface expression of the mineralized domain.

For Table 9-1 sampling within 100 metres of the mineralized domain interpreted for resource modelling, or within the domain, is designated as being within the Rozino area. All other sampling is classified as Tintyava regional exploration. One Hereward trench spanning the nominated boundary is assigned to the Rozino area reflecting the trench origin with trench metres subdivided by area.

Table 9-1: Key surface exploration activities

Phase	Type	Unit	Rozino area	Tintyava regional	Total
Hereward	Trenching	number	60	16	76
		metres	2,906	1,184	4,090
Velocity	Trenching	number	6	9	15
		metres	576	372	948
	Stream sed.	number	-	66	66
	Soil samples	number	164	3,025	3,189
	Rock chip	number	94	12	106

Surface trenches intersected mineralization within the Rozino area, and supported drill planning, with geological mapping improving understanding of mineralization controls. Within the Rozino area, the other exploration sampling has been effectively superseded by drilling, and is not relevant to current Mineral Resource estimates or exploration. Current exploration and planning includes regional data from outside the Rozino area.

9.1.2 Velocity Exploration Activities

Velocity's exploration activities since 2017 include initial work in the Rozino area, and regional exploration within the Ivaylovgrad Corridor. This work included surface mapping, soil sampling, rock chip sampling, stream sediment samples, surface rock sampling and trenching. Figure 9-1 shows exploration sampling relative to the extents of the Tintyava PL and surface expression of the Rozino mineralized domain interpreted for Mineral Resource estimation.

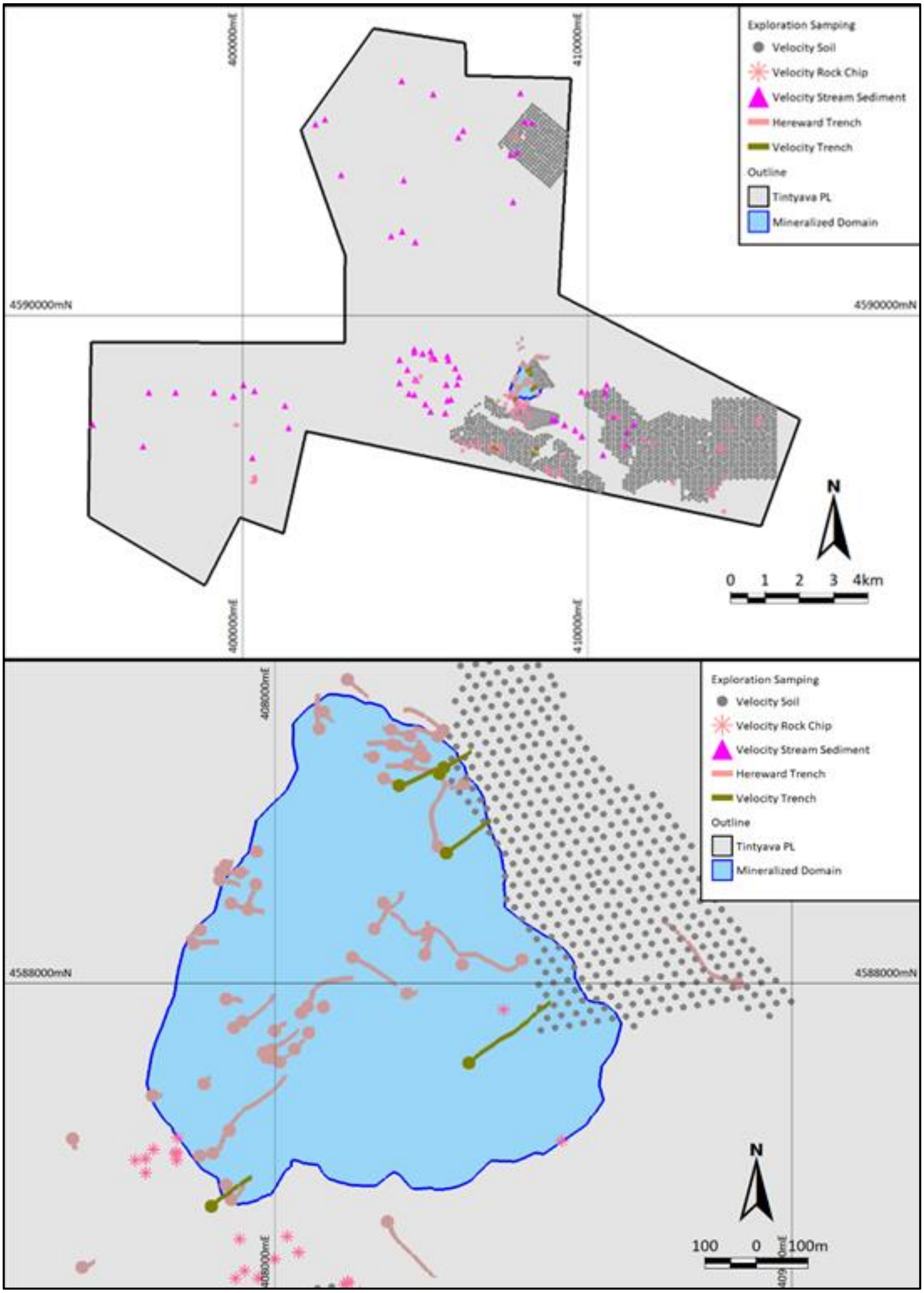


Figure 9-1: Exploration sampling relative to PL boundary and mineralized domain
Source: Prepared by MPR in September 2020 from information provided by Velocity

9.1.2.1 Trenching

Trenches were dug by hand to an average width of approximately 0.7 metres and generally penetrated 10 to 20 centimetres of bedrock below surface cover. Trench depth was limited to approximately two metres due to safety concerns. Trench sampling procedures were based on drill hole sampling procedures with continuous one metre channel samples collected from trench bases in halved 96 mm diameter plastic pipe, controlling the volume of the sample material.

Spacing of trenches is highly variably, ranging from locally less than 20 metres to around 300 metres in the Rozino area, with very broad spacing in peripheral areas.

9.1.2.2 Stream Sediment Sampling

Velocity's stream sediment protocol was as follows:

- Sample sites were chosen from straight stream lengths avoiding areas with contamination from neighbouring streams or infrastructure.
- For each site, sub-samples of gravel-clay were collected from different 8 to 10 locations over 25 to 50 metres stream intervals and sieved to produce 1.5 to 3Kg of <1 mm sample from around 12 to 15 kg of material.
- Monitoring of sample reliability included submission of duplicates and coarse blanks.

The stream sediment samples are generally irregularly and broadly spaced, with 66 samples distributed throughout the 145 square kilometre Property (Figure 9-1).

9.1.2.3 Soil Sampling

Velocity's soil sampling procedures included the following:

- Sample sites were located within 10% of the nominal grid spacing from the proposed grid location, ensuring the site was uncontaminated and the soils in situ. Sample locations were recorded by hand-held GPS.
- Samples were collected from the "B" horizon (62%), mixed "B" and "C" horizons (33%), or rarely the "A" or "C" horizons, at an average depth of 22 cm.
- Samples were generally (90%) sieved at the site with 0.5 to 1.5 Kg of sub 1 mm material collected for assay. Rare damp samples (10%) were collected without sieving.
- Monitoring of sample reliability included routine submission of duplicates and coarse blanks.

Spacing of soil samples varied by area (Table 9-2). Regional exploration targets inducing the Kazak, Arpa, Tumbata and Konnitsi areas were sampled at 100 by 100 metres spacing for a combined 18.78 square kilometres. Sampling in the Rozino and Rozino South areas at 25 by 25 metre spacing covered a combined 0.87 square kilometres.

Table 9-2: Velocity soil sampling by area

Nominal spacing	Area	Number samples	Area (km^2)
25 by 25 m	Rozino	428	0.27
	Rozino South	926	0.60
	Subtotal	**1,354**	**0.87**
100 by 100 m	Arpa	217	2.25
	Kazak	372	3.84
	Konnitsi	289	2.91
	Tumbata	928	9.48
	Vis	29	0.29
	Subtotal	**1,835**	**18.78**
TOTAL		**3,189**	**19.65**

9.1.2.4 Rock Chip Sampling

Velocity's rock chip sampling includes grab samples intended to identify potentially anomalous material and continuous chip samples which aim to generally represent the grade and tenor of available outcrop. Monitoring of sample reliability included submission of coarse blanks in sample batches.

9.1.3 Sample Representivity

Velocity's monitoring of sample reliability for the exploration sampling indicate that the sampling is free from material sample biases and sufficiently representative of the sampled material, including the sieved portions of soil and stream sediment samples for intended exploration purposes. It is unknown whether the sieved portions soil and stream sediment samples submitted for analysis are representative of the in-situ material. This does not reduce the samples suitability for the intended exploration purposes.

9.2 Summary of Exploration Activities and Results for Key Areas

9.2.1 Rozino South

Velocity's exploration activities in Rozino South comprised the following:

- Reconnaissance mapping undertaken as part of regional mapping of the Tintyava Property,
- Collection of 926 soil samples at 25 by 25 metre spacing over an area of 0.60 square kilometres.
- Collection of 32 rock chip samples

The soil sampling identified arsenic and ppb level gold anomalies in the southern part of the area (Figure 9-2). Mapping and analysis of rock chip samples did not indicate surface alteration, or significantly elevated gold grades at ppm level. However, Velocity considered the soil anomalies sufficiently suggestive of potential deeper mineralization to warrant further investigation, and they were tested by exploration diamond drilling as described in Section 10.7 which is ongoing.

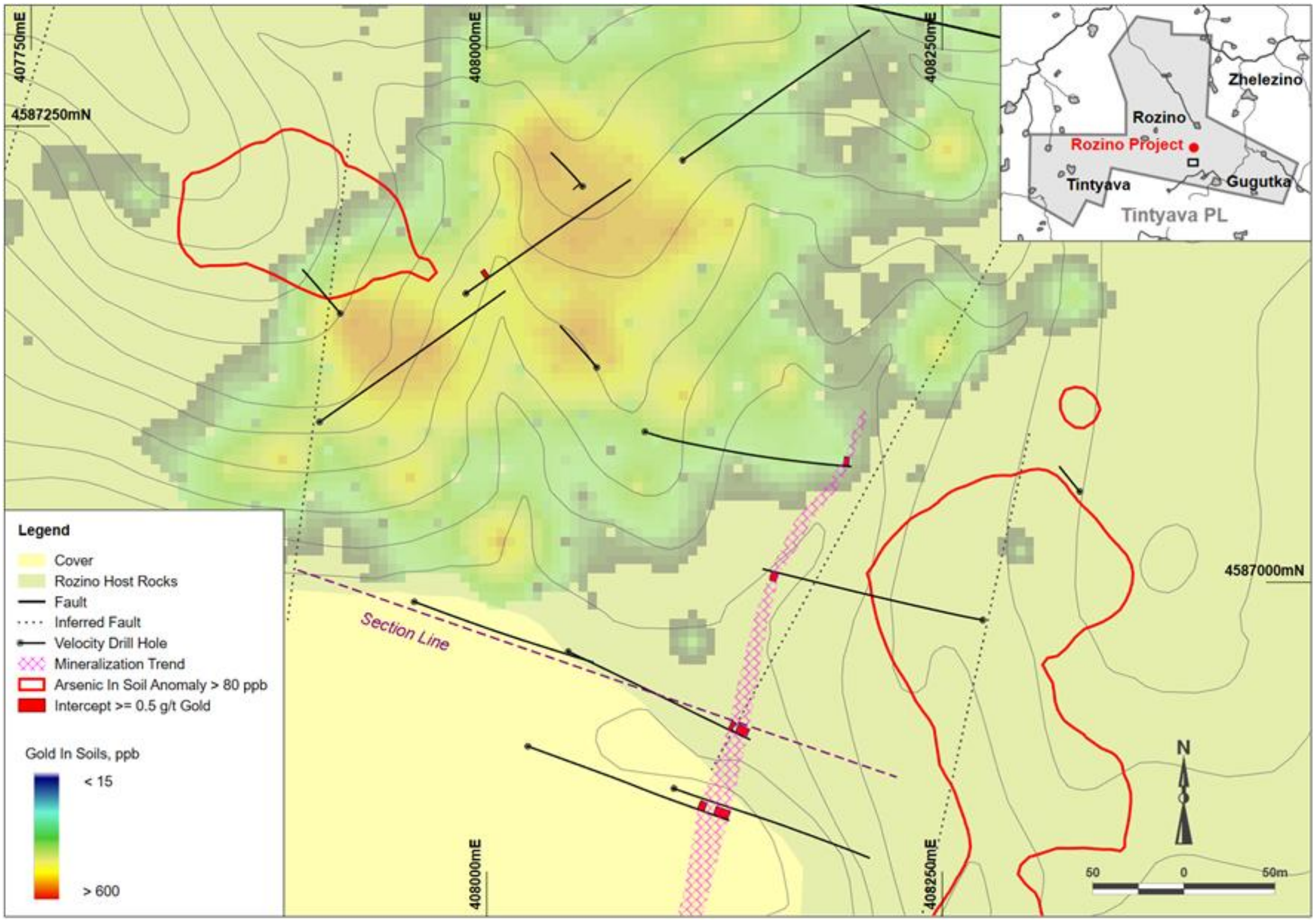


Figure 9-2: *Rozino South soil anomaly, geology and drillhole traces*
Source: Velocity, 2020

9.2.2 Kazak East and West

Velocity's exploration of the combined Kazak area included collection of 372 soil samples at 100 metre by 100 metre spacing over an area of around 3.8 square kilometres. This sampling showed two zones of anomalous ppb level gold grades which are interpreted to cluster around a regional NNW trending fault structure that limits the BRSZ at Kazak West and Kazak East (Figure 9-3). Additional exploration within the area of the Kazak West soil anomaly included collection of rock-chip samples from silicified breccia-conglomerates along with nine trenches totalling 372 metres.

On the basis of the combined results of the exploration sampling, Velocity consider the Kazak East and Kazak West areas to have sufficient potential to host significant gold mineralization to warrant reconnaissance exploration diamond drilling as described in Sections 10 and 26.

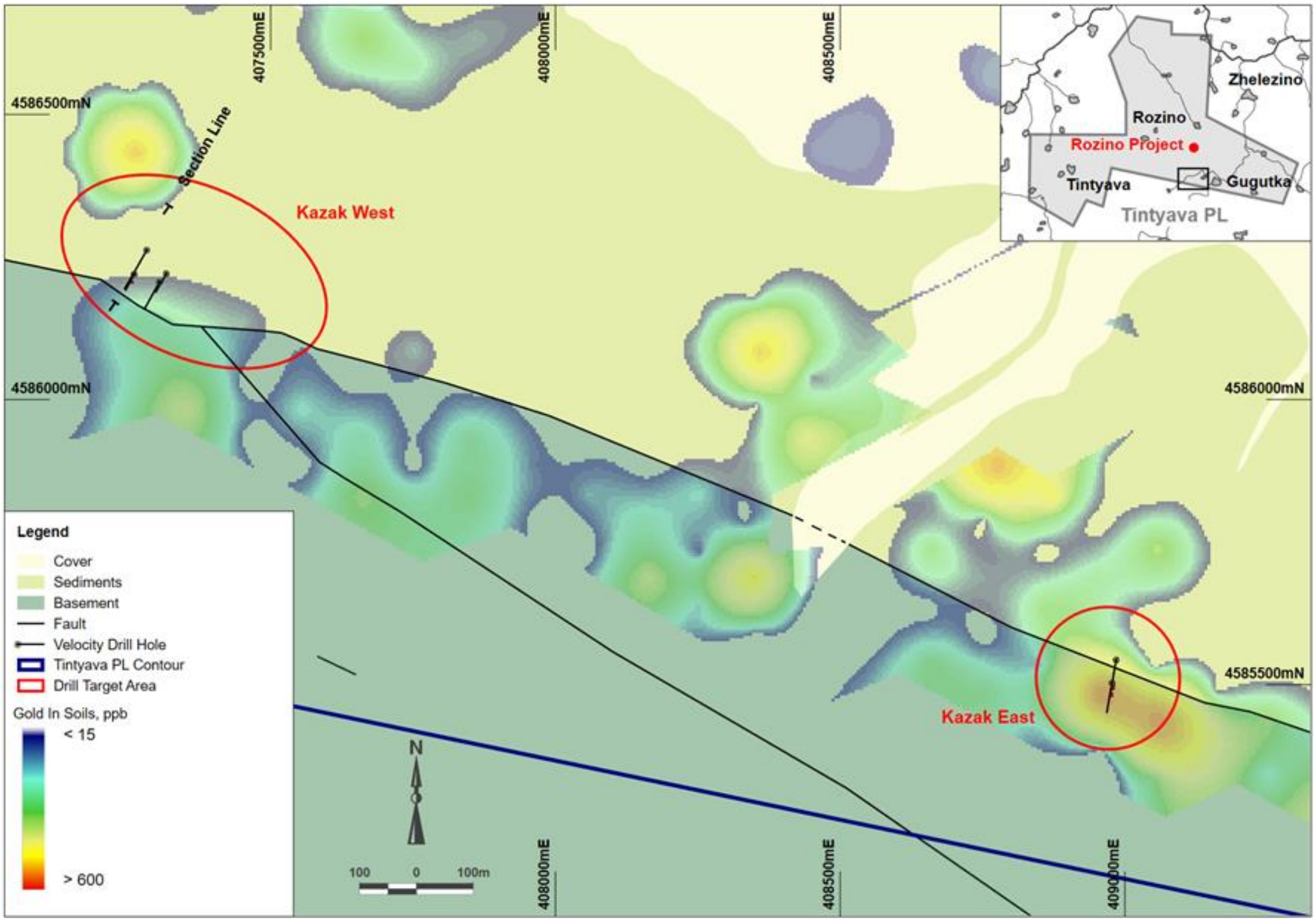


Figure 9-3: Kazak soil anomalies, geology and drillhole traces
Source: Velocity, 2020

9.2.3 Tumbata

Velocity's exploration sampling of the Tumbata area includes collection of 928 soil samples at 100 metre by 100 metre spacing over an area of around 9.48 square kilometres, and 14 rock chip samples.

The soil sampling showed two zones of anomalous ppb level gold grades within areas of mapped Palaeogene breccia-conglomerates (Figure 9-4) which are interpreted to be analogous to the host rocks of the Rozino deposit. analysis of rock chip samples did not indicate surface alteration, or significantly elevated gold grades at ppm level. However, Velocity considered the soil anomalies sufficiently suggestive of potential deeper mineralization to warrant further investigation, and exploration drilling is proposed for this area (Section 26).

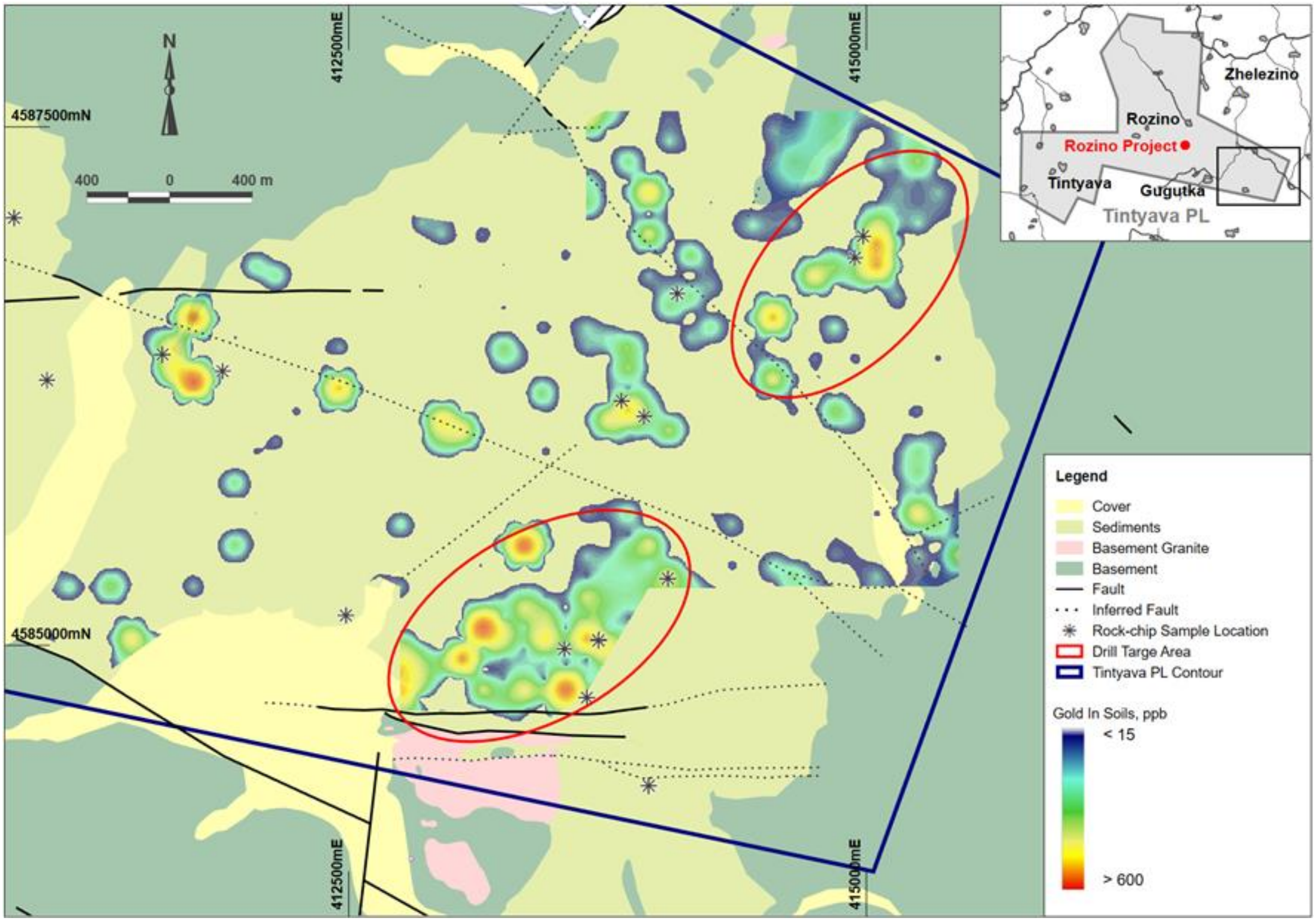


Figure 9-4: Tumbata soil anomalies and geology
Source: Velocity, 2020

10 Drilling

10.1 Drilling Completed

At the date of this report, Velocity's exploration drilling of the Tintyava Property is on-going. This report reflects drilling information available on the 28th of September 2020. This drilling includes 349 diamond holes for 49,643 metres of drilling completed by Velocity, along with Hereward and Asia Gold during the mid-2000's and vertical holes completed by Geoengineering in the 1980's.

Rozino Mineral Resource estimates described in this report are based on information available for Hereward Asia Gold and Velocity drilling on the 23rd of October 2019. The drilling information informing the estimates (the "Estimation Dataset") comprises data from angled diamond holes drilled by Hereward, Asia Gold and Velocity. Velocity's diamond drilling provides 82% of the mineralized domain composites informing the estimates with Asia Gold and Hereward drilling contributing 3% and 16% respectively.

This report subdivides the Tintyava drilling as outlined below. Subdivision of Velocity's drilling into pre and post October 2019 phases reflects the dataset used for resource estimation.

- **1980's Geoengineering drilling:** Few details of sampling and assaying are available for this drilling, and little information is available to demonstrate the reliability of data from these holes. These holes are excluded from the estimation dataset. They are not relevant to Mineral Resource estimates or exploration and are not detailed in this report.
- **2004 to 2007 Hereward and Asia Gold drilling:** Information from this drilling is included in Mineral Resource estimates and provides around 18% of mineralized domain composites informing the current estimates.
- **April 2017 to October Velocity drilling**: This drilling, which comprises 170 diamond core holes in the Rozino area provides 82% of the mineralized domain composites informing the current estimates.
- **November 2019 to September 2020 Rozino area drilling:** This drilling comprises 17 infill holes targeting mineralization classified as Inferred in the current resource block model and 2 drill holes outside the modelled mineralization.
- **November 2019 to September 2020 exploration drilling:** This drilling comprises 19 holes drilled by Velocity targeting exploration targets within the Tintyava Property.

Table 10-1 summarizes the available drilling by phase and location. Drill holes within the mineralized domain interpreted for the current estimate, or within 100 metres of the domain boundary, are designated as being within the Rozino area. The subtotal of 2004 to October 2019 drilling shown in this table represents information available for Mineral Resource estimation.

Figure 10-1 shows drill hole traces by phase relative to the extents of the Tintyava PL and mineralized domain. Figure 10-2 shows traces for the estimation dataset drilling relative to the resource model coloured by category and the crest of optimal pit shell used to constrain Mineral Resource estimates. Model estimates outside the optimal pit do not have demonstrated reasonable prospects for eventual economic extraction and are not included in Mineral Resource estimates.

Table 10-1: Drilling database by phase and area

		Rozino area		Tintyava regional		Total	
		Holes	**Metres**	**Holes**	**Metres**	**Holes**	**Metres**
Geoengineering		76	12,441	10	1,849	86	14,289
Asia Gold		6	740	9	1,993	15	2,733
Hereward		28	3,794	12	1,468	40	5,262
Velocity	Apr 2017 to Sep 2018	56	9,055	-	-	56	9,055
	Oct 2018 to Oct 2019	114	12,733	-	-	114	12,733
	Subtotal to Oct 2019	**170**	**21,787**	**-**	**-**	**170**	**21,787**
	Nov 2019 to Sep 2020	19	2,722	19	2,850	38	5,572
	Subtotal Velocity	**189**	**24,509**	**19**	**2,850**	**208**	**27,359**
Subtotal 2004 to Oct 2019		**204**	**26,321**	**21**	**3,461**	**225**	**29,782**
Subtotal 2004 to Sep 2020		**223**	**29,043**	**40**	**6,311**	**263**	**35,353**
TOTAL DRILLING		**299**	**41,484**	**50**	**8,159**	**349**	**49,643**

Velocity's drilling tests most of the modelled Rozino mineralization on an approximately 50 by 50 metre pattern with drill holes inclined to the northeast at about 50˚ along northeast-southwest (055˚) trending traverses.

Hereward and Asia Gold drilling generally tests central portions of the modelled mineralization on an approximately 50 by 50 metre pattern with holes generally inclined to the northwest at around 55˚. These holes are generally aligned sub-parallel with mineralization trends and define mineralized zones less robustly than Velocity's drilling which intersects mineralization trends at a greater angle providing a more reliable basis for resource estimation.

The combined hole spacing for Rozino varies from around 50 by 50 metres and locally closer in central portions of the deposit, to around 100 by 100 metres in peripheral areas. Exploration drilling outside the Rozino area is generally very broadly and irregularly spaced.

Quality control measures adopted for Velocity's diamond drilling have established that this sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling. Although reliability of Hereward and Asia Gold data has not been established with the same degree of rigour, available information indicates that these data are representative and free of any material biases.

The author considers that the available information has established that, for the combined drilling dataset, excluding Geoengineering drilling which is not relevant to resource estimation or exploration, the sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling.

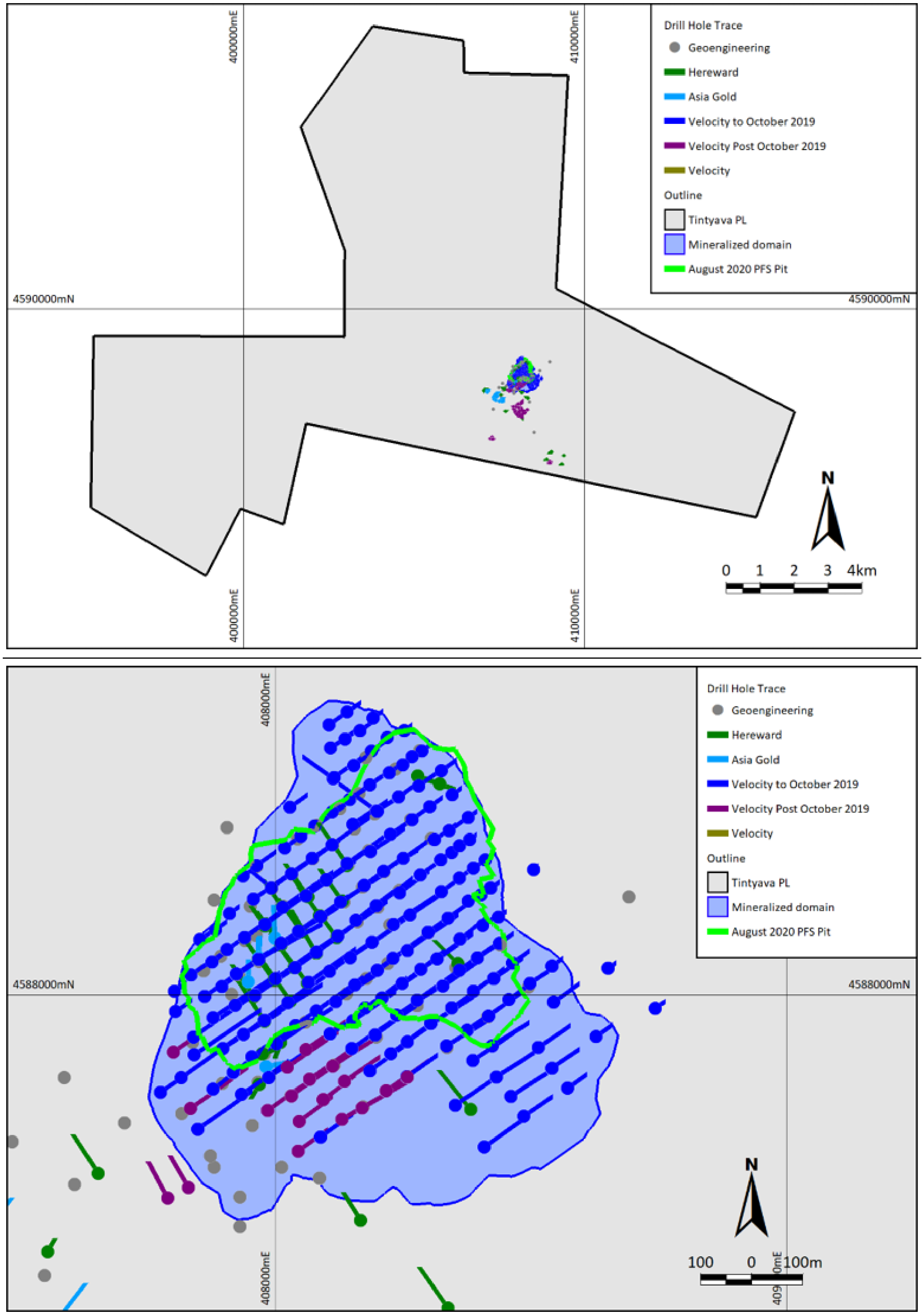


Figure 10-1: Drillhole traces, PL boundary and mineralized domain

Source: Prepared by MPR in September 2020 from information provided by Velocity

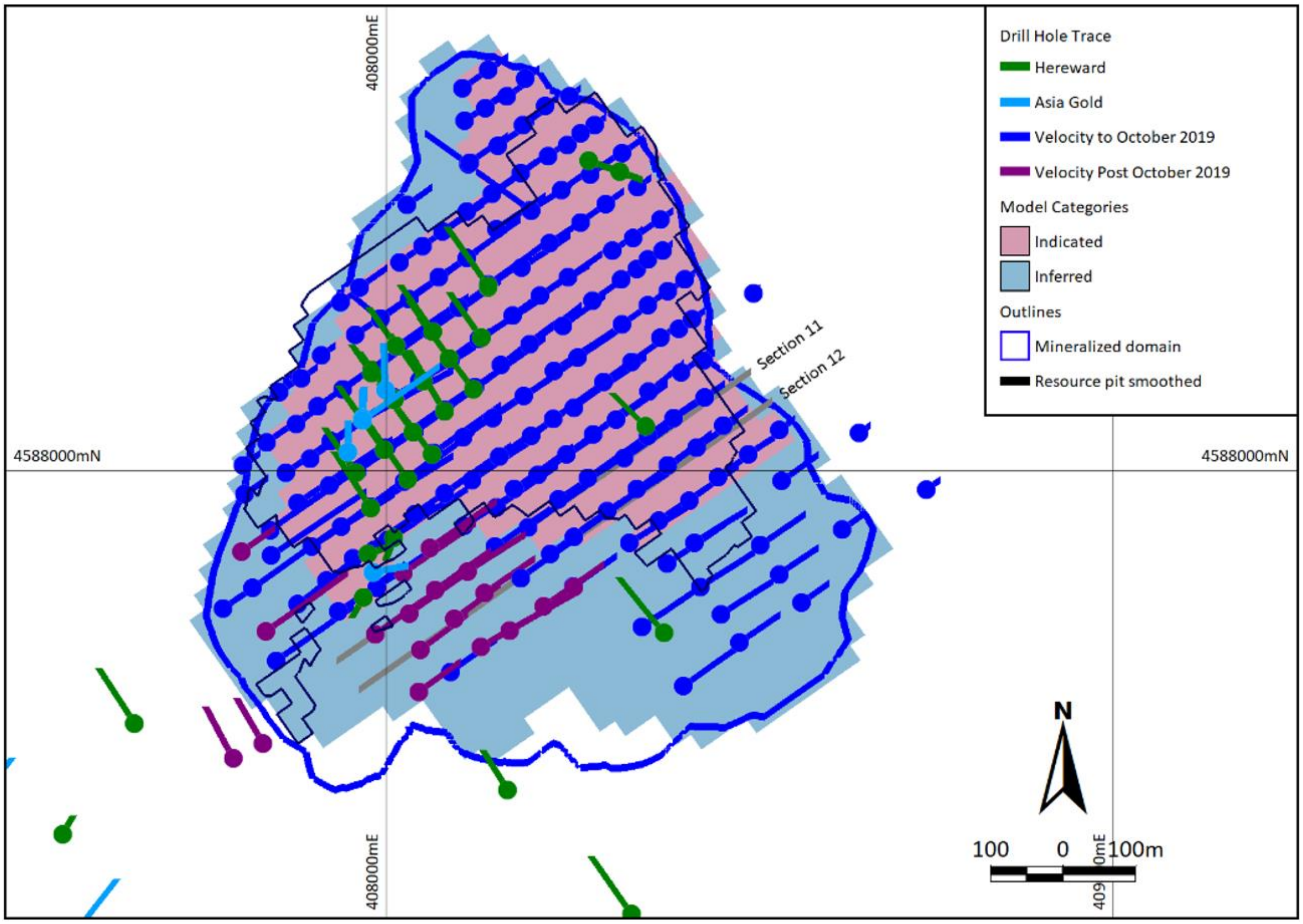


Figure 10-2: Drillhole traces, model categories and resource pit shell

Source: Prepared by MPR in September 2020 from information provided by Velocity

10.2 Hereward and Asia Gold Drilling

10.2.1 Summary

Diamond holes drilled by Asia Gold and Hereward and contribute around 3% and 16% of the mineralized domain composites informing the current resource model respectively. Information from this drilling phase within the broader Tintyava project help inform Velocity's exploration and planning of on-going exploration drilling.

No information, such as core recoveries is available to directly indicate the representivity of samples from Asia Gold and Hereward drilling. Assay results for duplicate core samples collected from a Hereward drill hole by Velocity and paired comparisons between gold grades from the combined Hereward and Asia Gold dataset and Velocity drilling support the general reliability of the drilling, sampling and assaying for the Hereward and Asia Gold drilling and demonstrate that the assaying is representative and free of any biases or other factors that may materially impact the reliability of the analytical results.

The author considers that the sample preparation, security and analytical procedures adopted for the Tintyava exploration sampling and drilling provide an adequate basis for the current Mineral Resource estimates and exploration activities.

10.2.2 Drilling and Sampling Procedures

The following summary of drilling and sampling for Hereward and Asia Gold's drilling is derived from the cited references, Hogg, 2017 and notes supplied by Velocity.

Velocity have located collars for most of Hereward and Asia Gold's Rozino area drill holes and accurately surveyed their locations with a Differential Geographic Positioning System (DGPS) instrument consistently with their surveying of Velocity's holes.

Velocity geologists report that Hereward and Asia Gold's holes were surveyed, probably with a Reflex tool, but this information is not available and in the current database these holes are assumed to run straight at designed orientations. Due to the relatively wide drill hole spacing, comparatively shallow depths and comparatively broad mineralized zones, the lack of comprehensive down-hole surveys for these holes is of little concern for the current estimates.

The author considers that hole paths of Hereward and Asia Gold's drill holes have been located with sufficient accuracy for the current Mineral Resource estimates.

Drill core from Hereward and Asia Gold's drilling was carefully arranged in core boxes and halved with diamond saw perpendicular to the dominant geological fabric. Half core samples were collected in uniquely numbered plastic bags together with a sample number tag and stored in a secure facility prior to transportation to an accredited commercial laboratory for sample preparation and analysis.

For the 2001 to 2005 drilling, the sample batches were transported by company personnel to Eurotest Control AD in Sofia, for sample preparation and analysis. For the 2006 drilling, sample batches were transported by company to personnel to Sofia airport and shipped by a reputable courier service to ACME Analytical Laboratories in Vancouver, Canada for sample preparation and analysis.

10.2.3 Paired Sample Comparison to Velocity Drilling

No information, such as core recoveries are available to directly indicate the representivity of drill samples from Hereward and Asia Gold drilling. To provide some indication of the reliability of these data, two metre down-hole composited gold grades from the combined dataset of Hereward and Asia Gold drilling were compared with the nearest composite from Velocity drilling.

The paired comparison used a maximum separation of ten metres in plan and five metres vertical and yielded 356 pairs with an average separation distance of 6.6 metres. This selection criteria, which in the author's experience is comparatively broad for such comparisons was selected to give sufficient pairs for meaningful analysis. As evidenced by core duplicate assays described in Section 11, gold grades of the Rozino mineralization are highly variable, and it is reasonable to expect considerable scatter between individual paired composite grades identified by these criteria.

As expected, the paired comparison shows substantial scatter for individual pairs with mean grades impacted by three pairs with outlier high grades from Velocity's drilling of between 17 and 36 g/t. As shown by Table 10-2 excluding the three outlier pairs and composites with gold grades of less than 0.1 g/t mean grades for the paired data are very similar. This comparison supports the general reliability of samples from Hereward and Asia Gold drilling.

Table 10-2: Paired composites from Velocity and Hereward-Asia Gold drilling

	Full set (Au g/t)		0.1 to 12.0 g/t (Au g/t)	
	Hereward/Asia Gold	Velocity	Hereward/Asia Gold	Velocity
Number	356		214	
Mean	0.51	0.70	0.73	0.73
Mean difference		36%		0%
Minimum	0.00	0.01	0.10	0.10
1st Quartile	0.09	0.09	0.20	0.19
Median	0.22	0.19	0.37	0.35
3rd Quartile	0.47	0.44	0.70	0.59
Maximum	11.2	36.0	11.2	11.1

10.3 Velocity Drilling Procedures

10.3.1 Drilling and Sampling Procedures

Drilling and sampling procedures for Velocity's Tintyava drilling included only minor differences between phases. All on-site core handling and sampling was supervised by Velocity geologists using protocols established by Velocity which are consistent with the author's experience of good quality, industry standard techniques.

All of Velocity's drilling was undertaken by GEOPS Balkan Drilling Services Ltd using track mounted diamond coring rigs (Figure 10-3). The drilling utilized PQ and HQ wireline triple tube core barrels (122.6 and 96 mm hole diameter respectively) with generally three metre drill runs and shorter runs where necessary to maximize core recovery. For the 2017 drilling and two holes from the post October 2019 drilling core was orientated where possible using a DeviCore BBT orientation tool. For all other drilling core was not oriented.

Figure 10-3: Velocity diamond drilling – drill site detail
Source: Velocity, 2020

Routine core handling procedures comprised the following:

- Core was placed directly in wooden core boxes at drill sites and transported to Velocity's core storage facility in Ivaylovgrad by Velocity personnel at the end of every day shift.
- All drill core was photographed and geotechnically logged including core recovery.
- For oriented core, the orientation marking was checked and the core line marked and fabrics measured.
- Routine geological logging employed industry standard methods with rock type, alteration, veining, tectonic structures, bedding and sulphides recorded on standard log sheets. Logged data was later typed into pre-configured logging software which validates during data entry and subsequently imported into Velocity's master Geobank database.
- Sample intervals were assigned and marked by Velocity geologists, with a nominal length of one metre honouring geological contacts with a minimum length of 0.5 metres.
- Core was generally halved for sampling with a diamond saw and half-core samples collected by Velocity geologists and sealed in heavy duty plastic bags.
- The samples were weighed, packed and sealed in plastic barrels which were transported to ALS Minerals laboratory in Romania by an individual directly employed by Velocity.

10.3.2 Collar and Downhole Surveying

Velocity geologists supervised the positioning of drilling rigs at designed hole locations set out by using a Trimble R2 GNSS DGPS instrument, with rigs aligned to design azimuths by compass.

Upon completion of the drilling of each hole, a cement marker, inscribed with the drill hole name, was placed at the collar, and the collar surveyed by DGPS to determine collar coordinates to a minimum vertical resolution of +/- 0.40 metres.

With the exception of a single, shallow hole for which no samples were assayed, all Velocity drill holes were down-hole surveyed using a DeviShot magnetic wireless multishot tool. The 56 holes drilled from July 2017 to March 2018 (RDD-001 to RDD-056), were surveyed at an average spacing of around 17 metres, with most holes having an initial shallow survey at around 10 metres depth providing an indication of collar orientation. Holes drilled after November 2018 (RDD-057 to RDD-202 and KDD-001 to KDD-006) were generally down-hole surveyed at intervals of around 28 metres. In contrast to the earlier Velocity drilling, down-hole surveying of these holes did not generally include initial surveys at shallow depths.

The author considers that hole paths of Velocity's drilling have been located with sufficient accuracy for the estimated Mineral Resources and exploration purposes.

10.4 Velocity 2017 to October 2019 drilling

Velocity's 2017 to October 2019 diamond drilling provides 82% of the mineralized domain composites informing Mineral Resource Estimates which are described in Section 14 of this report. Individual drill hole results for this drilling are not detailed in this report.

Information available to demonstrate the representivity of core samples from Velocity's 2017 to 2019 diamond drilling included in the estimation dataset includes core recovery measurements.

Core recovery measurements were supplied for 99.98% of this drilling as recovered lengths for core runs which range from 0.1 to 3.2 metres in length and are dominated by intervals of around three metres. These data were composited to three metre intervals to provide a consistent basis for analysis. Table 10-3 summarizes core recoveries for the three metre composites by modelling domain.

Table 10-3: Velocity pre-October 2019 diamond core recovery by domain

Oxidation domain	Background		Mineralized		Total	
	Number	Average recovery	Number	Average recovery	Number	Average recovery
Oxide	20	92.3%	703	93.9%	723	93.8%
Transition	48	95.2%	781	94.8%	829	94.8%
Fresh	1,064	98.7%	4,707	98.6%	5,771	98.6%
Total	**1,132**	**98.4%**	**6,191**	**97.6%**	**7,323**	**97.7%**

Fresh rock core recoveries average 98.6% with only approximately 2% of composites showing recoveries of less than 90%. These recoveries are consistent with the author's experience of high quality diamond drilling. Although lower than for fresh rock, average core recoveries for oxidized and transitional intervals are within the range shown by the author's experience of good quality diamond drilling.

Quality control measures adopted for Velocity's Rozino Velocity's 2017 to 2019 diamond drilling included in the estimation dataset have established that this sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling.

10.5 Velocity November 2019 to September 2020 Drilling

10.5.1 Introduction

This report reflects drilling information available for the Property on the 28th of September 2020. Relative to the October 2019 drilling dataset used for resource estimation, this database includes information for an additional 38 holes for 5,572 metres, comprising 19 holes in the Rozino area and 19 holes testing regional targets within the Tintyava Property. For this subdivision, drill holes within the mineralized domain interpreted for the current estimate, or within 100 metres of the domain boundary are designated as being in the Rozino area.

As shown in Figure 10-2 the 19 Rozino area holes comprise infill drilling within the current resource model extents and extensional drilling outside the modelled mineralization.

The infill drilling comprises 17 holes that targeted potential extensions of strongly developed mineralization to the north and east of the drilled area. In this area, the current model estimates from broadly spaced drilling available in October 2019 show generally comparatively low gold grades and are classified as Inferred. The estimates fall outside the pit shell constraining resources and are not included in Mineral Resource estimates.

The extensional drilling includes two diamond holes to the west the interpreted mineralized domain which targeted potential extensions to the Rozino mineralization.

The 19 exploration holes comprise 13 holes for 2,360 metres in the Rozino South area, 2 holes for 183 metres at Kazak East and 4 holes for 306 metres at Kazak West. This drilling represents comparatively early stage exploration of areas that are being evaluated as part of Velocity's on-going exploration of the Tintyava Property.

10.5.2 Sample Reliability

Information supplied for the current review includes core recovery measurements for 99.97% of Velocity's post October 2019 diamond drilling. These measurements comprise recovered lengths per core run which range from 0.1 to 3.1 metres in length and are dominated by intervals of about three metres. The recovery measurements were composited to three metre intervals to provide a consistent basis for analysis. Table 10-4 summarizes core recoveries for the three metre composites by oxidation domain, assigned on the basis of geological logging, or surrounding drilling for one Rozino area hole for which no oxidation logging is available.

Table 10-4: Velocity Post October 2019 Diamond Core Recovery

Oxidation domain	Rozino area		Tintyava regional		Total	
	Number	Average recovery	Number	Average recovery	Number	Average recovery
Oxide	69	97.5%	68	97.2%	137	97.4%
Transition	24	90.6%	52	98.1%	76	95.7%
Fresh	862	99.2%	795	98.9%	1,657	99.0%
Total	**955**	**98.8%**	**915**	**98.7%**	**1,870**	**98.8%**

Fresh rock core recoveries average 99.0% with only approximately 1% of composites showing recoveries of less than 90%. These recoveries are consistent with the author's experience of high quality diamond drilling. Although lower than for fresh rock, average core recoveries for oxidized and transitional intervals are within the range shown by the author's experience of good quality diamond drilling.

Quality control measures adopted for Velocity's November 2019 to September 2019 diamond drilling have established that this sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling.

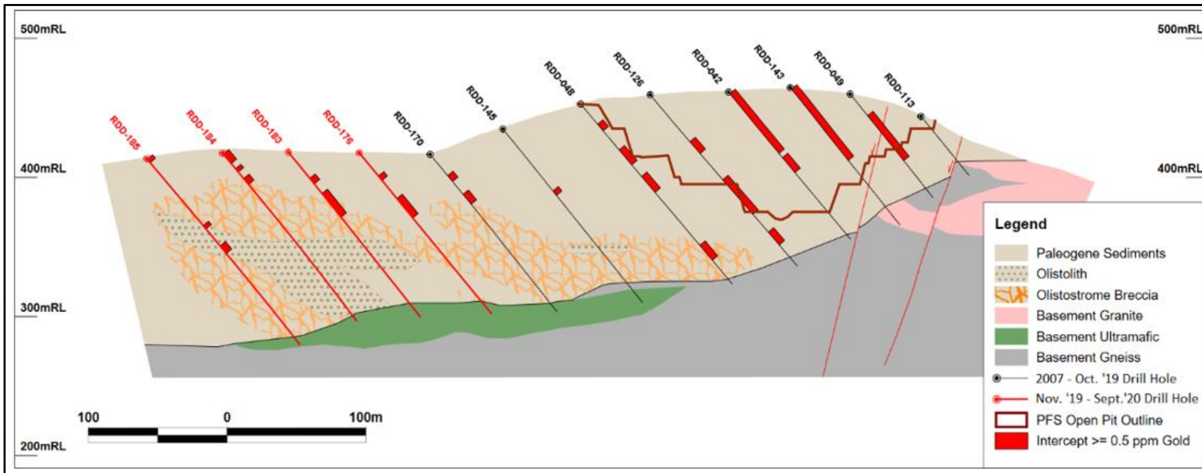
10.6 Rozino Area Drilling Results

Table 10-5 and the example cross sections in Figure 10-4 show significant intercepts from the November 2019 to September 2020 Rozino area drilling. These intercepts were calculated by Velocity using a 0.2 g/t gold trigger value, minimum 0.5 g/t composite grade and maximum of 3 metres of consecutive material grading less than 0.3 g/t. The north-easterly inclined drill holes are to perpendicular to the interpreted gently dipping mineralization trends and true widths are interpreted to average around 96% of down-hole intercept thicknesses.

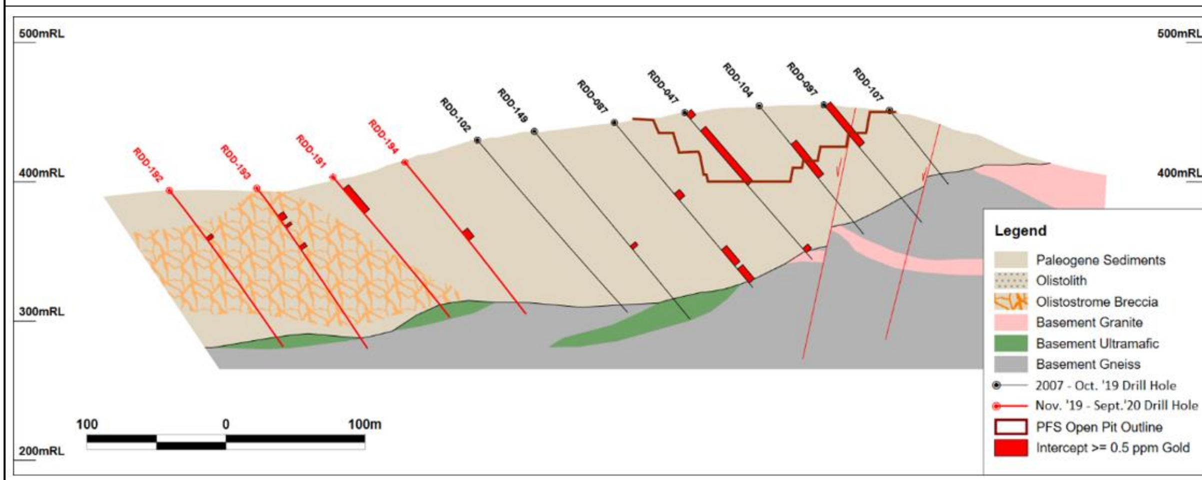
Table 10-5: Significant intercepts from post-October 2019 Rozino area drilling

Drillhole	Collar location			Depth (m)	Orientation	Downhole (m)		Au g/t (uncut)
	mE	mN	mRL		Azimuth/Dip	Interval	Length	
RDD-171	408,258	4,587,839	413	30	000/-90	Not assayed		-
RDD-176	408,111	4,587,860	418	150	055/-50	21.7-25.7	4.0	1.12
						42.7-60.7	18.0	0.56
						Incl. 52.7-56.7	4.0	1.81
RDD-179	408,023	4,587,858	439	188	055/-50	13-22	9.0	1.07
RDD-180	408,060	4,587,893	441	177	055/-50	6.2-9.2	3.0	0.53
						33.4-38.4	5.0	0.58
RDD-183	408,067	4,587,835	418	152	055/-50	24.4-29.4	5.0	1.16
						38.4-60.2	21.8	0.88
						Incl. 47.2-53.2	6.0	2.21
RDD-184	408,031	4,587,803	417	155	055/-50	0.8-9.7	8.9	1.13
						14.6-17.6	3.0	0.61
						23.8-29.3	5.5	0.91
RDD-185	407,985	4,587,774	413	174	055/-50	0.5-3.7	3.2	1.21
						63.1-66.1	3.0	1.72
						82.1-89.1	7.0	0.58
RDD-189	407,801	4,587,888	468	92	055/-50	No significant intercept		-
RDD-190	407,834	4,587,778	450	221	055/-50	110.1-113.1	3.0	1.05
						169.1-172.1	3.0	1.10

Drillhole	Collar location			Depth (m)	Orientation	Downhole (m)		Au g/t (uncut)
	mE	mN	mRL		Azimuth/Dip	Interval	Length	
RDD-191	408,135	4,587,831	403	131	055/-50	11.3-34.5	23.2	0.95
						Incl. 24.5-34.5	10.0	1.78
RDD-192	408,046	4,587,753	394	139	055/-50	43.1-46	2.9	0.66
RDD-193	408,093	4,587,796	395	140	055/-50	24.8-29.8	5.0	0.64
						33.8-35.8	2.0	0.55
						51.8-54.8	3.0	3.99
RDD-194	408,184	4,587,852	414	140	055/-50	65-72.6	7.6	0.56
RDD-195	408,216	4,587,812	416	165	055/-50	No significant intercept		-
RDD-196	408,170	4,587,779	404	173	060/-53	No significant intercept		-
RDD-197	408,130	4,587,757	384	125	060/-53	No significant intercept		-
RDD-198	408,044	4,587,695	371	129	055/-55	3-6	3.0	1.83
RDD-201	407,789	4,587,603	287	126	330/-50	No significant intercept		--
RDD-202	407,830	4,587,623	287	115	330/-50	No significant intercept		-



Section 11. Looking northwest



Section 12. Looking northwest

Figure 10-4: Rozino area drilling example cross sections (section lines shown in Figure 10-2)
Source: Velocity, 2020

The 17 Rozino area diamond holes drilled from November 2019 to September 2020 intersected mineralization of similar tenor to that shown by earlier drilling in peripheral portions of the Rozino mineralization, where gold mineralization is generally less well developed than shown by drilling in central portions of the deposit. It anticipated that these holes will be included in future resource estimates.

The two holes drilled to the west of the modelled mineralization (RDD-201 and RD-202) did not return any significant intercepts.

10.7 Rozino South Exploration Drilling Results

Figure 9-2 shows the locations of Velocity's Rozino South exploration drilling which targeted the soil anomaly described in Section 9.

Samples from these holes returned elevated gold grades at, or below the contact between un-mineralized sediment and basement, at down-hole depths of commonly greater than 200 metres. Velocity interpret this mineralization as being related to thin hydrothermal breccia zones associated with epithermal quartz-pyrite veinlets similar in character to those found at the Rozino deposit.

The mineralization is interpreted as north northeast striking linear feature, which is intersected by drilling over an extent of 200 metres and remains open in both directions.

Table 10-6 and the example cross section in show significant intercepts from the November 2019 to September 2020 Rozino South drilling. These intercepts were calculated by Velocity using a 0.2 g/t gold trigger value, minimum 0.5 g/t composite grade and maximum of 3 metres of consecutive material grading less than 0.3 g/t. True widths are interpreted to average around half of the down-hole intercept thicknesses.

Table 10-6: Rozino South significant intercepts from post-October 2019 drilling

Drillhole	Collar location			Depth (m)	Orientation	Downhole (m)		Au g/t (uncut)
	mE	mN	mRL		Azimuth/Dip	Interval	Length	
RDD-172	408,053	4,587,217	296	141	320/-80	10.2-12.2	2.0	0.58
RDD-173	408,061	4,587,118	271	162	320/-80	No significant intercept		-
RDD-174	407,920	4,587,147	303	158	320/-80	No significant intercept		-
RDD-175	408,326	4,587,049	280	102	320/-80	No significant intercept		-
RDD-177	408,273	4,586,979	265	219	280/-55	203-209.15	6.2	1.18
						Incl. 208-209.15	1.2	5.23
RDD-178	407,989	4,587,158	278	176	055/-50	21-25	4.0	0.92
RDD-181	407,908	4,587,088	284	197	055/-50	No significant intercept		-
RDD-182	408,108	4,587,231	289	198	055/-50	No significant intercept		-
RDD-186	408,087	4,587,082	270	209	110/-55	201.2-206.1	4.9	0.63
RDD-187	408,045	4,586,962	233	204	115/-55	178.4-182.4	4.0	0.52
						186.4-198.8	12.4	0.83
RDD-188	407,960	4,586,989	233	188	110/-55	No significant intercept		-
RDD-199	408,023	4,586,910	235	207	110/-55	177.2-180.9	3.7	0.62
						190.6-193.6	3.0	2.11
RDD-200	408,103	4,586,887	230	200	110/-45	No significant intercept		-

Figure 9-2 shows the locations of Velocity's Rozino South exploration drilling which targeted the soil anomaly described in Section 9.

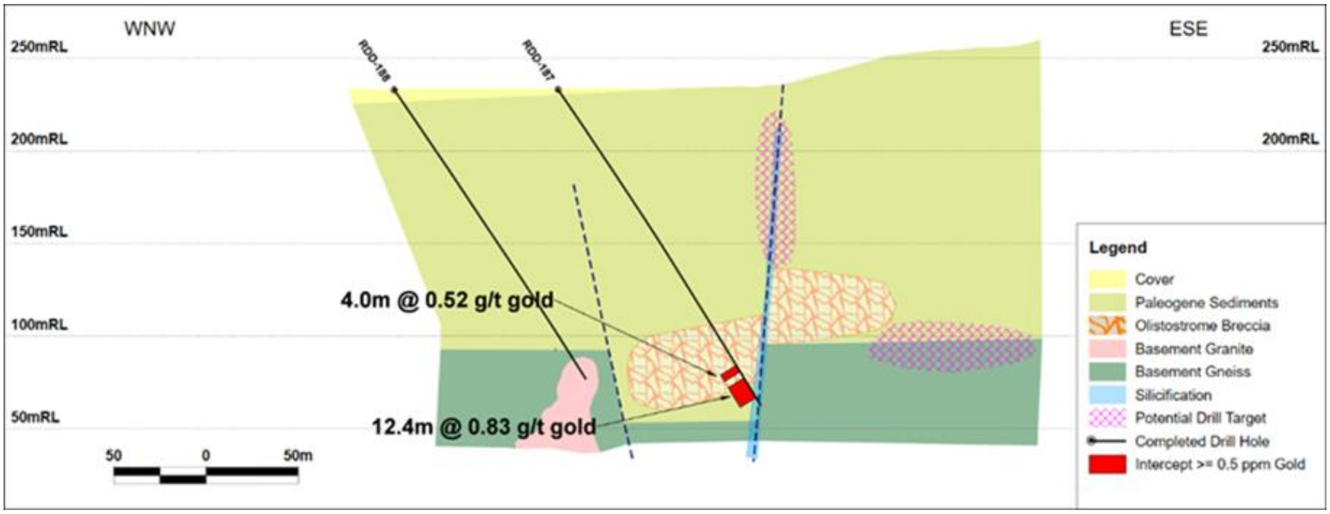


Figure 10-5: Typical cross section, Rozino South (section line shown in Figure 9-2)
Source: Velocity, 2020

10.8 Kazak East and West Exploration Drilling Results

Velocity's exploration drilling of the Kazak East and Kazak West areas (Figure 9-3) is on-going and too little information is available for substantive interpretation of the results. Table 10-7 shows significant intercepts available for drilling in this area and Figure 10-6 shows an example cross section for Kazak West. These intercepts were calculated by Velocity using a 0.2 g/t gold trigger value, minimum 0.5 g/t composite grade and maximum of 3 metres of consecutive material grading less than 0.3 g/t.

Velocity's drilling of the Kazak East area comprises two diamond holes for 183 metres of drilling, which intersected mineralization within intensely pyrite altered gneissic basement rocks.

Assessment and interpretation of the Kazak East and Kazak West areas is at an early stage, and the mineralization is not yet well understood, including the mineralization orientation. The association between down-hole intercept lengths in Table 10-7 and true mineralization widths is unknown.

Table 10-7: Significant intercepts from post-October 2019 Kazak drilling

Drillhole	Collar location			Depth (m)	Orientation	Downhole (m)		Au g/t (uncut)
	mE	mN	mRL		Azimuth/Dip	Interval	Length	
Kazak East								
KDD-004	408,978	4,585,502	290	83.1	190/-50	27.1-36.9	9.8	1.01
KDD-005	408,986	4,585,542	283	100.1	190/-50	72.2-85.2	13	1.20
						Incl. 73.2-78.2	5.0	2.24
Kazak West								
KDD-001	407,305	4,586,206	334	79.9	210/-45	No significant intercept		-
KDD-002	407,317	4,586,220	326	54.4	210/-45	No significant intercept		-
KDD-003	407,261	4,586,219	323	46.6	210/-45	17.5-22.2	4.7	0.57
KDD-006	407,285	4,586,259	310	125.5	210/-50	No significant intercept		-

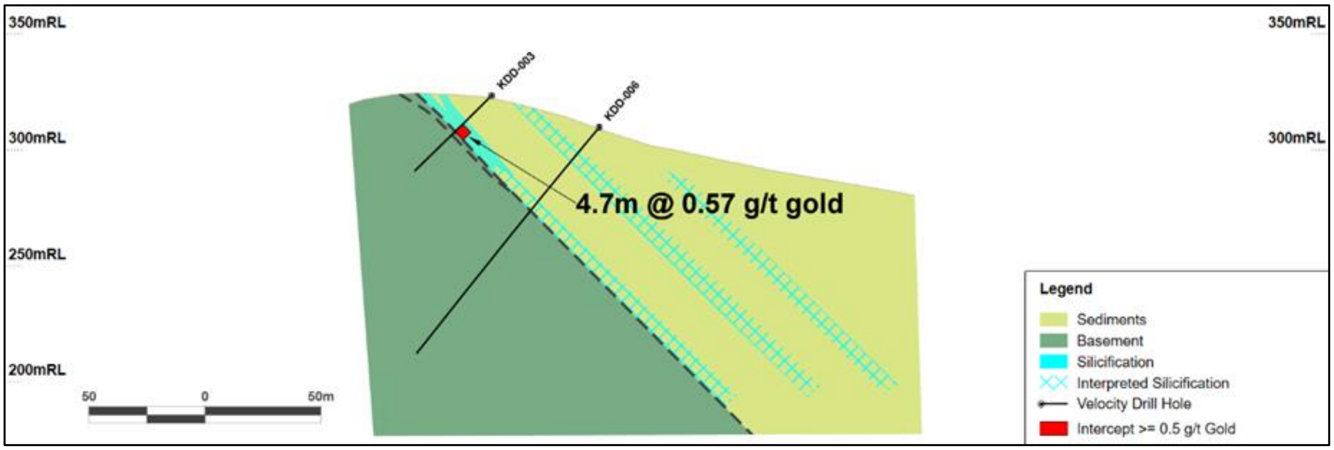


Figure 10-6: Typical cross section, Kazak West (cross-section line shown in Figure 9-3)
Source: Velocity, 2020

11 Sample Preparation, Analyses and Security

11.1 Summary

This report reflects sampling information available for the Property on the 28th of September 2020, including exploration sampling and drilling completed by Hereward and Asia Gold during the mid-2000's and Velocity since 2017. Mineral Resource estimates are based on information available for Hereward Asia Gold and Velocity drilling on the 23rd of October 2019.

Sampling and assaying of Geoengineering drilling are of little relevance to the Mineral Resource estimates or current exploration and are not detailed in this report.

All sample preparation and gold assaying of primary samples from the exploration and drilling was undertaken by independent commercial laboratories. Analyses undertaken by company personnel are limited to density measurements undertaken by Hereward and Velocity employees respectively.

Hereward and Asia Gold's diamond core was sampled and analysed by industry standard methods. The core was generally halved for analysis with a diamond saw with generally one metre intervals, samples analysed for gold analysis by fire assay by commercial laboratories.

Hereward and Asia Gold's monitoring of sampling and assay reliability for drill samples included duplicates and blanks for both data sets and certified reference standards for Asia Gold's drilling. These data are not available for the current review. Assay results for duplicate core samples collected from a Hereward drill hole by Velocity and paired comparisons between gold grades from the combined Hereward and Asia Gold dataset and Velocity drilling support the general reliability of the drilling, sampling and assaying for Hereward and Asia Gold drilling.

Exploration sampling included trench sampling by Hereward, and soil, stream sediment, rock chip and trench sampling by Velocity. This sampling employed industry standard methods, with analysis by reputable independent laboratories. Velocity's monitoring of sampling and assaying reliability for the exploration sampling included coarse blanks for all sampling types and duplicates for stream sediments, soil samples and trenches.

The author considers that quality control measures adopted for sampling and assaying of Velocity's exploration sampling have established that the sampling, and assaying is representative and free of any biases or other factors that may materially impact the reliability of the sampling and analytical results.

Diamond core from Velocity's drill holes was halved with a diamond saw and sampled over generally one metre down-hole intervals. The samples were submitted to ALS in Romania for analysis by thirty-gram fire assay. Reliability of sampling and assaying for these data has been established by duplicates, blanks and certified reference standards, and inter-laboratory repeat assaying.

Summaries of Quality Assurance and Quality Control (QAQC)information for Velocity's drilling in this chapter are subdivided by phase. Information for drilling completed prior to October 2019, which is of direct relevance to Mineral Resources is separated from data from later drilling which does not directly affect Mineral Resource estimates.

The author considers that quality control measures adopted for sampling and assaying of Velocity's drilling have established that the sampling, and assaying is representative and free of any biases or other factors that may materially impact the reliability of the sampling and analytical results. Although the reliability of Hereward and Asia Gold data has not been established with the same degree of rigour, the available information indicates that these data are representative and free of any material biases.

The author considers that the sample preparation, security and analytical procedures adopted for the Tintyava exploration sampling and drilling provide an adequate basis for the current Mineral Resource estimates and exploration activities.

11.2 Hereward and Asia Gold Drilling

Diamond holes drilled by Asia Gold and Hereward and contribute around 3% and 16% of the mineralized domain composites informing the current resource estimates respectively. Drilling from this phase also contributes to the exploration drilling available for the Tintyava project, and adds to the dataset informing Velocity's exploration of the project.

The following description of sample preparation and analyses for this drilling is derived from Hogg, 2017 and notes supplied by Velocity.

- Drill core was carefully arranged in core boxes and halved with diamond saw perpendicular to the dominant geological fabric.
- Half core samples were collected in uniquely numbered plastic bags together with a sample number tag. The sample bag was tightly sealed and placed into 20-sample batches that included one field duplicate and one blank. Asia Gold's samples also included reference standards.
- Each batch was sealed with a security tag and stored in a secure facility prior to transportation to an accredited commercial laboratory for preparation and analysis.
- For the 2001 to 2005 drilling, samples were transported by company personnel to Eurotest Control AD in Sofia an ISO 9001 certified laboratory, for preparation and analysis. At Eurotest Control AD, the samples were crushed to 90% passing 2mm with a 400g split pulverized to 85% passing 75 microns. Gold assaying was by aqua regia digest with AAS determination, with samples assaying at greater than 2.5 g/t also analysed by fire assay.
- For the 2006 drilling, sample were transported by company to personnel to Sofia airport and shipped by a reputable courier service to ACME Analytical Laboratories (ACME) in Vancouver, Canada an ISO 9001 and ISO 17025: 1999 certified laboratory for preparation and analysis. At ACME, the samples were crushed to 90% passing 2mm and a 1Kg split pulverized to 95% passing 75 microns. Preparation equipment was flushed with barren material after processing each sample. Samples of pulverized material were analysed for gold by 50 gram fire assay.
- A chain of custody for the samples was maintained at all times.

Hereward and Asia Gold's QAQC monitoring of sampling and assay reliability included duplicates and blanks for both data sets and certified reference standards for Asia Gold's drilling. These data are not available for the current review. Velocity's investigations of the reliability of Hereward's diamond drilling data included collection of 122 duplicate half core duplicate samples from one drill hole. These samples were submitted to ALS for analysis, including fire assaying for gold consistently with primary Velocity samples.

Table 11-1 and Figure 11-1 demonstrate that although, as expected for duplicate core samples, there is considerable scatter for individual pairs, average gold grades reported by ALS are reasonably consistent with the original sample assays. Reasons for the comparatively minor difference in average grades are uncertain. It may simply reflect the relatively small dataset and does not significantly affect confidence in the combined estimation dataset.

As discussed in Section 10, paired comparison of two metre down-hole composited gold grades from the combined dataset of Hereward and Asia Gold drilling with composites from Velocity drilling showed similar average grades. This comparison supports the general reliability of drilling, sampling and assaying for the Hereward and Asia Gold drilling.

Table 11-1: Duplicate check assays for Hereward diamond core

	Full Range		0.05 to 5.0 g/t	
	Original (Au g/t)	**Duplicate ALS (Au g/t)**	**Original (Au g/t)**	**Duplicate ALS (Au g/t)**
Number	122		97	
Mean	**0.59**	**0.51**	**0.62**	**0.55**
Mean difference		**-7%**		**-7%**
Minimum	0.83	0.68	0.48	0.41
1st Quartile	1.55	1.60	1.12	1.16
Median	0.05	0.02	0.05	0.05
3rd Quartile	0.10	0.07	0.12	0.13
Maximum	0.26	0.20	0.39	0.34
Correl. Coef.	0.59		0.59	

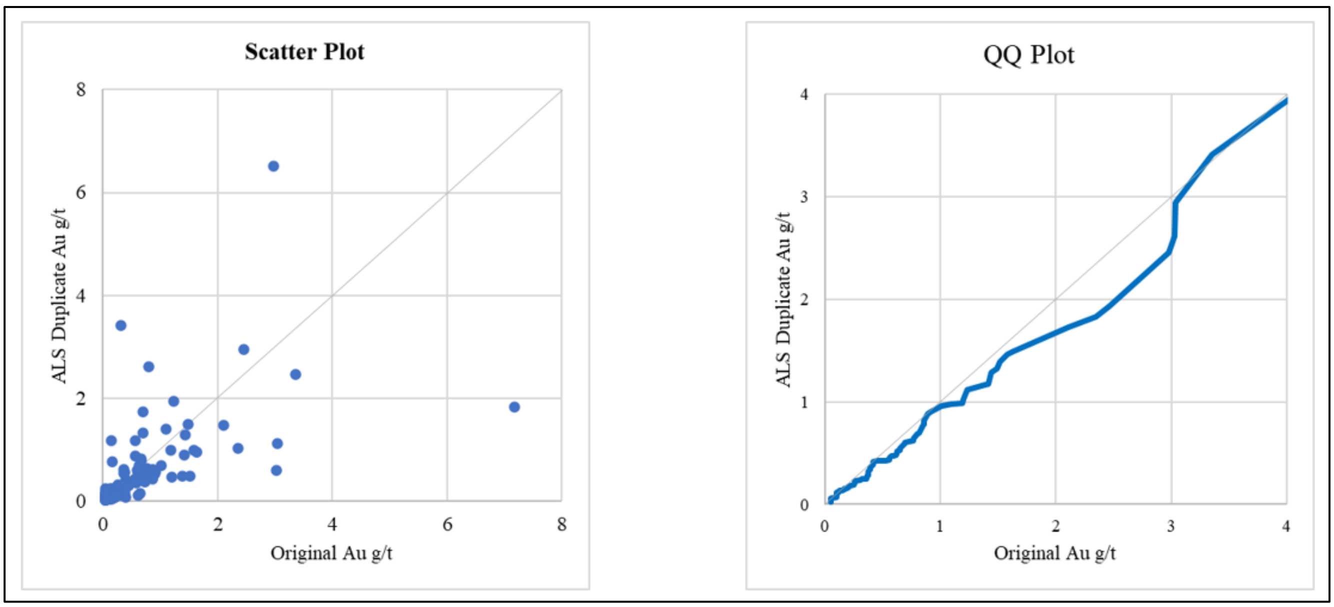


Figure 11-1: Duplicate check assays for Hereward diamond core

11.3 Velocity Exploration and Drilling

11.3.1 Sampling Procedures and Sample Security

Collection and on-site handling of samples from Velocity's exploration sampling and drilling was supervised by Velocity geologists and employed protocols established by Velocity which are consistent with the author's experience of good quality, industry standard techniques. The sample handling and security procedures were consistent for all sampling phases and types.

Exploration sampling procedures comprised the following:

- For each stream sediment sampling site, 8 to 10 sub-samples collected from different sites over a 25 to 50 metre interval of the stream were sieved to produce 1.5 to 3 Kg of <1 mm sample from around 12 to 15 kg of material.
- Soil samples were collected from an average depth of 22 cm and generally sieved at the site giving 0.5 to 1.5 Kg of sub 1 mm material. Rare damp samples were collected without sieving.
- Rock chip sampling includes grab samples and continuous chip samples.

- Trench samples represent continuous one metre channel samples collected from trench bases in in halved 96 mm diameter plastic pipe, controlling the volume of the sample material.
- Monitoring of the reliability and accuracy of exploration sampling included submission of coarse blanks and field duplicates for all sample types and duplicates for soil, stream sediment and trench samples.

Diamond drill core sampling procedures comprised the following:

- Core was placed directly in wooden core boxes at the drill site and transported to Velocity's core yard in Ivaylovgrad by Velocity personnel at the end of every day shift.
- Sample intervals were assigned and marked by Velocity geologists, with a nominal length of one metre honouring geological contacts with a minimum length of 0.5 metres.
- Core was generally halved for sampling with a diamond saw and half-core samples collected by Velocity geologists and sealed in labelled plastic bags along with a pre-printed sample tag with sample number and barcode.

Routine storage and dispatch procedures for all sampling types comprised the following:

- The samples inclusive of duplicates, blanks and standards were weighed, and packed in polywoven bags which were sealed in plastic drums for delivery to the ALS Minerals laboratory in Romania by a Velocity employee. The drums were sealed with a metal clip ring and plastic seal tag to detect tampering.
- Sample submission forms were included with each assay batch and an electronic copy emailed to ALS. Upon receipt by ALS, the sealed drums are checked for tampering and samples reconciled with sample submission forms.

The upper set of photographs in Figure 11-2 shows the general lay-out of storage for drill core and returned coarse rejects and sample pulps at Velocity's storage facility in Ivaylovgrad. The lower set of photographs in this figure demonstrate sample packaging for dispatch to ALS.

Prior to delivery to ALS, all sample collection and transportation were undertaken or supervised by Velocity personnel. No other personnel were permitted unsupervised access to samples before delivery to ALS. A chain of custody was maintained at all times, with records taken during sampling, sample dispatch, laboratory arrival and return of coarse rejects and pulps to Velocity's Ivaylovgrad storage facility.

(a) Logging and core storage area, (b) Coarse reject storage in sealed plastic drums, (c) Pulp storage area

(A) Individual sample bags with sample tags (B) Polywoven bag labelled with sample sequence, (C) Plastic drum with sample and batch sequence and plastic tag seal

Photographs courtesy Velocity

Figure 11-2: Velocity's core storage facility and sample packaging

11.3.2 Sample Preparation and Analysis

All primary assaying of Velocity's exploration and drill samples was undertaken by ALS Minerals laboratory in Romania. ALS is independent of Velocity and provided analytical services on a standard commercial basis. The laboratory is certified to ISO 17025.

Upon receipt by ALS, each sample batch was checked for consistency with the sample submission form and entered into the ALS LIMS system

Stream sediment and soil samples were dried at less than 60oC and sieved to minus 180 microns. The samples were analysed for suite of elements including gold by aqua regia digest with ICP-MS determination.

After oven drying rock chip and trench samples were jaw crushed to 70% passing 2 millimetres. A one-kilogram sub-sample of crushed material collected by rotary splitting was pulverized to 85% passing 75 microns in a ring and puck pulverizer. Thirty-gram riffle split sub-samples of pulverized sample were analysed for gold by fire assay with lead collection, solvent extraction and AAS finish. For samples with initial assays reporting over 10 g/t assay values were derived from gravimetric finish fire-assay of a second 30-gram sub-sample. Samples from outside the Rozino area were also assayed for a suite of elements by aqua regia digest with ICP-AES determination.

For diamond core drill samples, sample preparation comprised oven drying and jaw crushing of entire, generally 3 to 3.5-kilogram samples to 70% passing 2 millimetres. A one-kilogram sub-sample of crushed material collected by rotary splitting was pulverized to 85% passing 75 microns in a ring and puck pulverizer. Thirty-gram riffle split sub-samples of pulverized sample were analysed for gold by fire assay with lead collection, solvent extraction and AAS finish. For samples with initial assays reporting over 10 g/t assay values were derived from gravimetric finish fire-assay of a second 30-gram sub-sample.

11.3.3 Sampling and Assay Reliability for Exploration Sampling

Velocity's monitoring of the reliability and accuracy of exploration sampling included submission of coarse blanks and field duplicates for all sample types and duplicates for soil, stream sediment and trench samples.

The coarse blank samples of un-mineralized marble collected from well outside the mineralized area were blind to the assay laboratory test for contamination during sample preparation, and provide a check of sample misallocation by field staff, the laboratory and during database compilation.

Table 11-2 summarizes coarse blank gold assay results for the exploration sampling. For preparation of this table samples assaying at below the detection limits of 1 ppb for soil sampling and stream sediment samples and 0.005 g/t for rock chips and trench samples were assigned values of half the relevant detection limit. Table 11-2 demonstrates that each set of coarse blank assays show very low gold grades with no indication of significant contamination or sample misallocation.

Table 11-2: Blank assay results for Velocity exploration sampling

	No. per primary	Assay results				Prop'n > detection
		Number	Minimum	Average	Maximum	
Soils Au ppb	34	98	0.50	0.63	3.00	18%
Stream Sed Au ppb	33	2	0.50	0.50	0.50	0%
Rock Chips Au g/t	53	2	0.0025	0.0025	0.0025	0%
Trench Au g/t	26	32	0.0029	0.003	0.007	13%

Two duplicates are available for the 66 primary stream sediment samples. With a maximum gold assay grade of 2 ppb this small set of samples provides little information about repeatability of stream sediment sampling and is not detailed in this report.

Table 11-3and Figure 11-3 summarize gold assays for field duplicate soil and trench samples.

Field duplicate assays for soil samples show considerable scatter, with one extremely variable pair (Location 16/5, 19 ppb versus 1,480 ppb) which may be indicative of sample misallocation. The general variability is consistent with the generally very low gold grades, and does not affect confidence in the reliability of the soil sampling for exploration purposes.

Table 11-3 and Figure 11-3 demonstrate that although there is some scatter for individual pairs, the trench duplicate assay results generally correlate reasonably well with original results demonstrating the adequacy of field sub-sampling procedures for this sampling.

Table 11-3: Field duplicate assays for Velocity exploration sampling

SOIL SAMPLES (Au ppb)				
Au ppb	**Full range**		**< 1000 ppb**	
	Original	**Duplicate**	**Original**	**Duplicate**
Number	71		70	
Mean	44.1	30.4	23.6	30.6
Mean difference		-31%		30%
Minimum	0.5	0.5	0.5	0.5
1st quartile	2.0	2.0	2.0	2.0
Median	5.0	4.0	5.0	4.0
3rd quartile	17.0	9.5	17.0	8.8
Maximum	1480	682	524	682
Correl. Coef.	0.29		0.83	
TRENCH SAMPLES (Au g/t)				
Au ppb	**Full range**		**0.1 to 7.0 g/t**	
	Original	**Duplicate**	**Original**	**Duplicate**
Number	919		320	
Mean	0.22	0.23	0.50	0.51
Mean difference		6%		1%
Minimum	0.0025	0.0025	0.11	0.10
1st quartile	0.02	0.02	0.18	0.18
Median	0.06	0.06	0.28	0.29
3rd quartile	0.19	0.19	0.54	0.53
Maximum	6.05	9.73	4.94	5.12
Correl. Coef.	0.86		0.76	

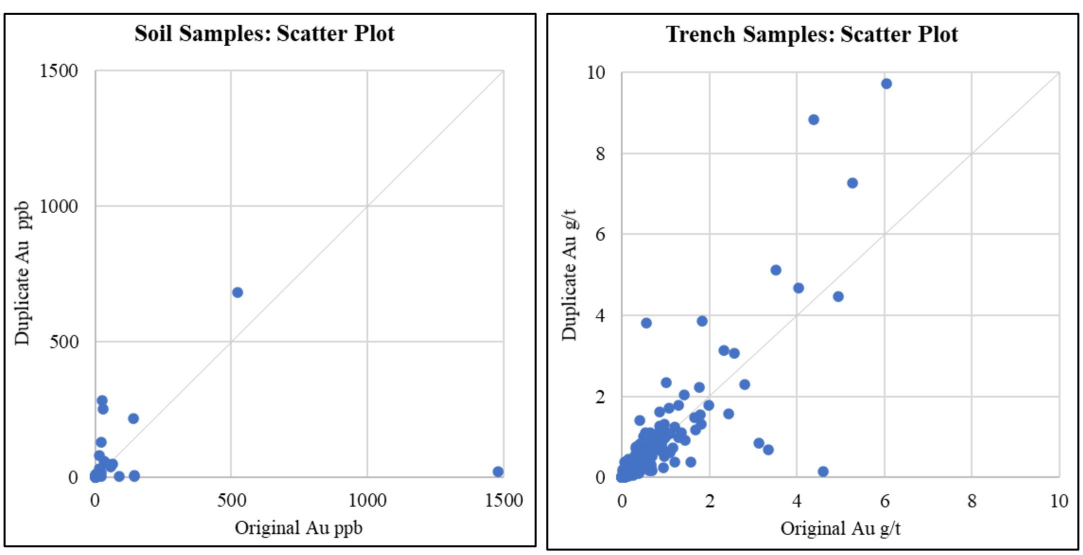


Figure 11-3: Field duplicate assays for Velocity exploration sampling
Source: MPR, 2020

The author considers that the sample preparation, security and analytical procedures adopted for Velocity's Tintyava exploration sampling an adequate basis for current exploration activities.

11.3.4 Sampling and Assay Reliability for 2017 to October 2019 Drilling

Velocity's monitoring of the reliability and accuracy of sampling and assaying for the 2017 to October 2019 drilling is consistent with the author's experience of good quality, industry standard techniques. These protocols include routine submission of field duplicates, coarse blanks and certified reference standards along with submission of selected pulp samples for independent repeat assaying by a second laboratory.

As outlined below, QAQC information available for Velocity's 2017 to October 2019 drilling supports the general reliability of sampling and assaying for this drilling.

11.3.4.1 Field Duplicates

Duplicate core samples were collected for the 2017 to October 2019 drilling at an average frequency of around one duplicate per 28 primary samples. For the duplicated core intervals, both the original and duplicate sample represent quarter core samples collected by quartering the core with a diamond saw.

Table 11-4 and Figure 11-4 demonstrate that although there is some scatter for individual pairs, duplicate assay results generally correlate reasonably well with original results demonstrating the adequacy of field sub-sampling procedures. Reasons for the small difference in mean grades for higher-grade pairs are uncertain. However, it appears to represent an artefact of a small number of samples and is not significant at the current level of project assessment.

Table 11-4: Velocity 2017 to October 2019 field duplicate results

	Full Range		**0.1 to 7.0 g/t**	
	Original (Au g/t)	**Duplicate (Au g/t)**	**Original (Au g/t)**	**Duplicate (Au g/t)**
Number	694		302	
Mean	**0.25**	**0.28**	**0.47**	**0.49**
Mean difference		**9%**		**4%**
Minimum	0.003	0.003	0.10	0.10
1st Quartile	0.03	0.03	0.17	0.18
Median	0.09	0.09	0.26	0.27
3rd Quartile	0.24	0.23	0.50	0.52
Maximum	6.05	9.73	4.94	5.12
Correl. Coef.	0.89		0.82	

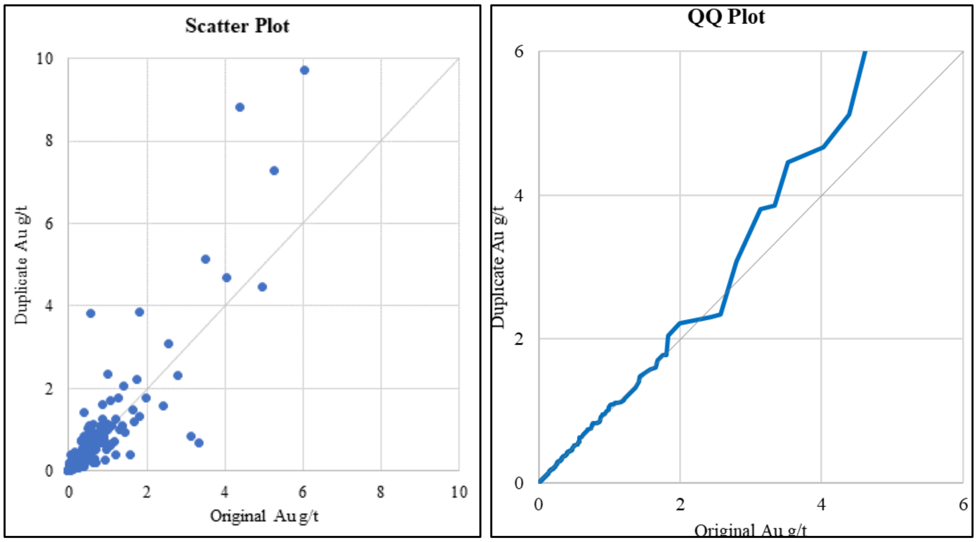


Figure 11-4: Velocity 2017 to October 2019 field duplicate results

11.3.4.2 Coarse Blanks

Velocity routinely included samples of un-mineralized marble collected from well outside the mineralized area in assay batches. These coarse blanks, which were blind to the assay laboratory were submitted at an average frequency of around one blank per 24 primary samples for the 2017 to October 2019 drilling. The blanks test for contamination during sample preparation, and provide a check of sample misallocation by field staff, the laboratory and during database compilation.

Table 11-5summarizes coarse blank gold assay results for the 2017 to October 2019 drilling with samples assaying at below the detection limit of 0.005 g/t assigned values of half the detection limit. This table demonstrates that, with a maximum grade of 0.010 g/t coarse blank assays show very low gold grades relative to typical Rozino mineralization with no indication of significant contamination or sample misallocation.

Table 11-5: Velocity 2017 to October 2019 coarse blank assays

Assay date	Number of blanks	Assay (Au g/t)			Proportion > detection
		Minimum	Average	Maximum	
2017	240	0.0025	0.003	0.007	9%
2018	130	0.0025	0.003	0.007	7%
2019	414	0.0025	0.003	0.010	11%
Total	**784**	**0.0025**	**0.003**	**0.010**	**10%**

11.3.4.3 Reference Standards

Samples of certified reference standards prepared by Geostats Pty Ltd, Perth, Western Australia were routinely inserted in sample batches for the 2017 to October 2019 drilling at an average rate of around 1 standard per 26 primary samples.

As shown by Table 11-6, although there is some variability for individual samples, average assay results for standards generally reasonably reflect expected values, with no evidence of material biases.

Table 11-6: Velocity 2017 to October 2019 reference standards assays

Standard	Expected (Au g/t)	Number of assays	Assays (Au g/t)			Average vs Expected
			Minimum	Average	Maximum	
G300-7	1.00	47	0.95	1.00	1.05	0%
G303-5	16.11	9	15.4	15.6	15.8	-3%
G303-8	0.26	31	0.23	0.25	0.26	-5%
G310-4	0.43	92	0.40	0.42	0.46	-2%
G312-1	0.88	113	0.76	0.88	1.02	0%
G314-10	0.38	14	0.35	0.38	0.43	-1%
G315-2	0.98	119	0.91	0.99	1.07	1%
G318-1	1.05	19	0.99	1.02	1.06	-3%
G398-2	0.50	60	0.47	0.50	0.53	-1%
G901-8	47.24	7	45.0	46.2	47.4	-2%
G903-9	11.26	11	10.5	11.2	11.6	-1%
G905-6	5.96	17	5.77	6.03	6.34	1%
G910-8	0.63	91	0.59	0.62	0.67	-1%
G914-6	3.21	105	3.05	3.26	3.60	2%
G914-7	9.81	7	9.61	9.86	9.99	1%
Total	**2.06**	**742**	**0.23**	**2.05**	**47.4**	**0%**

11.3.4.4 Inter-Laboratory Repeat Assays

Velocity's monitoring of ALS assaying of samples from the 2017 to October 2019 drilling included submitting 1,024 pulp samples to SGS in Chelopech Bulgaria for check assaying. These samples, which are from eight diamond holes represent around 5% of primary drill samples from this drilling phase.

The Chelopech laboratory has been audited and the Management System certified to be in accordance with ISO 9001:2015 by Bureau Veritas Holding SAS.

Samples submitted to SGS included 35 samples of reference standards, for which average assay results reasonably matched expected values (Table 11-7) confirming the general reliability of the SGS results.

Table 11-7: SGS Reference standards assays

Standard	Expected (Au g/t)	Number of assays	Assays (Au g/t)			Average vs Expected
			Minimum	Average	Maximum	
G312-1	0.88	7	0.80	0.91	1.10	3%
G314-10	0.38	18	0.36	0.38	0.40	0%
G315-2	0.98	7	0.97	0.99	1.02	1%
G914-6	3.21	3	3.16	3.18	3.20	-1%
Total	**0.84**	**35**	**0.36**	**0.85**	**3.20**	**1%**

Table 11-8 and Figure 11-5 demonstrate that although SGS repeat assays generally correlate reasonably well with the ALS assays, they show slightly lower average gold grades. Reasons for this trend, which is not supported by ALS assays of reference standards are uncertain. It does not significantly impact confidence in estimated Mineral Resources.

Table 11-8: Inter-laboratory repeat results

	Full range		0.1 to 9.0 g/t	
	ALS (Au g/t)	SGS (Au g/t)	ALS (Au g/t)	SGS (Au g/t)
Number	1,024		701	
Mean	**0.55**	**0.52**	**0.66**	**0.63**
Mean difference		**-5%**		**-4%**
Minimum	0.003	0.005	0.10	0.10
1st Quartile	0.09	0.09	0.21	0.20
Median	0.23	0.22	0.37	0.36
3rd Quartile	0.51	0.50	0.70	0.68
Maximum	14.1	14.2	8.22	7.90
Correl. Coef.	0.99		0.98	

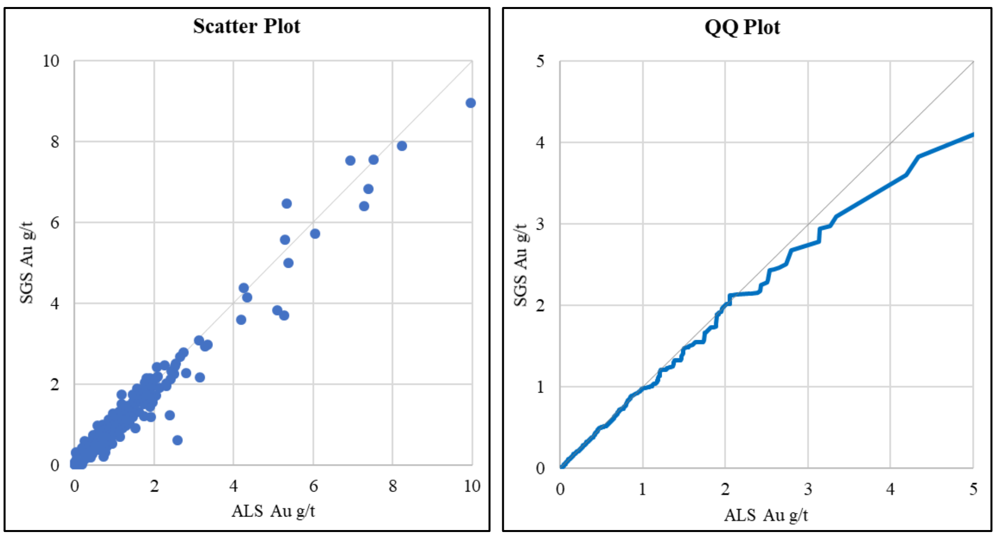


Figure 11-5: Inter-laboratory repeat results

11.3.5 Sampling and Assay Reliability for November 2019 to September 2020 Drilling

Velocity's monitoring of the reliability and accuracy of sampling and assaying for the November 2019 to September 2020 drilling, which includes routine submission of field duplicates, coarse blanks and samples of certified reference standards is consistent with the author's experience of good quality, industry standard techniques.

The following QAQC summaries combine information from Rozino area drilling and exploration reflecting the consistency of sampling and assay methods for these programs.

As outlined below the QAQC information support the general reliability of sampling and assaying for this drilling with sufficient confidence for the current exploration activities. The author recommends that prior to inclusion of data from this drilling phase in Mineral Resource estimates, Velocity undertake additional confirmation of the reliability of assaying, including inter-laboratory repeat assaying.

11.3.5.1 Field Duplicates

Field duplicate samples available for the Velocity's November 2019 to September 2020 drilling represent an average frequency of around one duplicate per 33 primary samples. For the duplicated core intervals, both the original and duplicate sample represent quarter core samples collected by quartering the core with a diamond saw.

The November 2019 to September 2020 drilling intersected proportionally fewer elevated gold grades that Velocity's earlier drilling which commonly targeted central portions of the Rozino mineralization. With just 15 intervals returning primary assays of greater than 0.1 g/t, gold grades for duplicates from the November 2019 to September 2020 drilling are commonly too low to provide a substantive indication of the reliability of field sampling. As shown in Table 11-9, and Figure 11-6, the small set of mineralized duplicates show comparable repeatability to that demonstrated by earlier phases of Velocity's drilling thus demonstrating the adequacy of field sub-sampling procedures.

Table 11-9: Velocity November 2019 to September 2020 field duplicate results

	Full Range		0.1 to 7.0 g/t	
	Original (Au g/t)	**Duplicate (Au g/t)**	**Original (Au g/t)**	**Duplicate (Au g/t)**
Number	149		13	
Mean	**0.08**	**0.05**	**0.34**	**0.29**
Mean difference		**-43%**		**-14%**
Minimum	0.003	0.003	0.12	0.10
1st Quartile	0.01	0.01	0.14	0.12
Median	0.02	0.02	0.18	0.13
3rd Quartile	0.04	0.04	0.26	0.20
Maximum	4.59	1.54	1.78	1.54
Correl. Coef.	0.44		0.98	

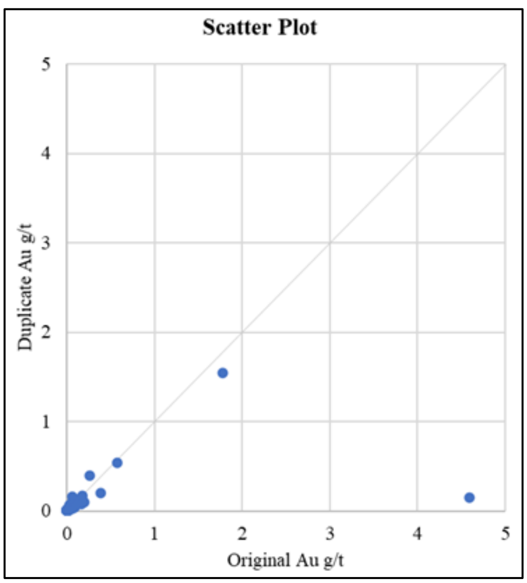


Figure 11-6: Velocity November 2019 to September 2020 field duplicate results

11.3.5.2 Coarse blanks

Coarse blank assay results available for Velocity's November 2019 to September 2020 drilling represent an average frequency of around one duplicate per 31 primary samples. The coarse blanks, comprising un-mineralized marble collected from well outside the mineralized area are blind to the assay laboratory.

For preparation of the summary of coarse blank assays in Table 11-10, samples assaying at below the detection limit of 0.005 g/t were assigned values of half the detection limit. This table demonstrates that, with a maximum grade of 0.025 g/t, coarse blank assays show very low gold grades relative to typical Tintyava mineralization with no indication of significant contamination or sample misallocation.

Table 11-10: Velocity November 2019 to September 2020 coarse blank assays

Number of blanks	Assay (Au g/t)			Proportion > detection
	Minimum	Average	Maximum	
167	0.003	0.003	0.025	19%

11.3.5.3 Reference standards

Samples of certified reference standards prepared by Geostats Pty Ltd, Perth, Western Australia were routinely inserted in assay batches of samples from Velocity's November 2019 to September 2020. The available results represent an average frequency of around one standard per 31 primary samples.

As shown by Table 11-11, although there is some variability for individual samples, average assay results for standards generally reasonably reflect expected values, with no evidence of material biases.

Table 11-11: Velocity November 2019 to September 2020 coarse blank assays standards assays

Standard	Expected (Au g/t)	Number of assays	Assays (Au g/t)			Average vs Expected
			Minimum	Average	Maximum	
G300-7	1.00	5	1.00	1.01	1.03	0%
G303-8	0.26	23	0.23	0.26	0.38	-3%
G312-1	0.88	42	0.79	0.89	1.06	-5%
G314-10	0.38	15	0.35	0.37	0.40	-2%
G315-2	0.98	16	0.94	0.98	1.01	0%
G318-1	1.05	9	1.01	1.03	1.08	-1%
G398-2	0.50	30	0.38	0.49	0.53	1%
G911-5	0.20	6	0.19	0.20	0.21	-2%
G914-6	3.21	12	3.08	3.82	9.82	-3%
G914-7	9.81	5	9.39	9.82	10.2	-1%
Total	**1.12**	**163**	**0.19**	**1.16**	**10.2**	**-2%**

11.4 Bulk Density Measurements

Bulk density measurements available for the Tintyava Property total 252 immersion measurements of diamond core, including 70 and 182 measurements of core from Hereward's and Velocity's drilling respectively.

Details of the methodology for Hereward's density measurements are not available. Velocity's density measurements were performed by Velocity field staff on wax coated core samples averaging 0.18 metres in length.

Average densities reported by ALS for 18 repeat immersion measurements closely match Velocity's average measurements for these intervals supporting the reliability of Velocity's density measurements (Table 11-12).

Table 11-12: ALS repeat density measurements

Oxidation	Number of measurements	Average density (t/m^3)		
		Velocity	ALS	Difference
Oxide	4	2.30	2.31	1%
Transition	3	2.44	2.41	-1%
Fresh	11	2.53	2.55	1%
Total	**18**	**2.46**	**2.48**	**1%**

Table 11-13 summarises the primary density measurements coded by the mineralization and oxidation wire-frames used for resource modelling. This table excludes two measurements from a Hereward hole around 300 metres outside the Rozino area.

Table 11-13: Density measurements

Phase	Zone	Oxidation	Number of measurements	Density (t/m^3)		
				Minimum	Average	Maximum
Background	Hereward	Fresh	6	2.47	2.58	2.72
	Velocity	Fresh	9	2.39	2.51	2.59
	Total	**Fresh**	**15**	**2.39**	**2.54**	**2.72**
Mineralized domain	Hereward	Oxide	-	-	-	-
		Transition	5	2.44	2.51	2.56
		Fresh	57	2.30	2.59	2.74
	Velocity	Oxide	26	2.12	2.35	2.67
		Transition	31	2.01	2.38	2.66
		Fresh	116	2.03	2.53	2.84
	Combined	Oxide	26	2.12	2.35	2.67
		Transition	36	2.01	2.40	2.66
		Fresh	173	2.03	2.55	2.84

Table 11-13 demonstrates that Velocity's and Hereward's measurements show comparable average densities for fresh mineralization. Although these samples test different mineralization, and the comparison is not definitive it provides some confidence in the general reliability of Hereward measurements. Too few measurements are available for oxide and transitional material for comparable comparisons.

The author considers that the available measurements have established average mineralization densities with sufficient accuracy for the current Mineral Resource estimates.

12 Data Verification

Verification checks undertaken by the author to confirm the validity of the database compiled for the current study include the following:

Checking for internal consistency between, and within database tables.

- Spot check comparisons between database entries and original field sampling sheets.
- Comparison of assay entries with laboratory source files.
- Comparison of assay values between nearby holes.
- Comparisons between assay results from different sampling phases.

These checks were undertaken using the working database compiled by the author and check both the validity of Velocity's master database and potential data-transfer errors in compilation of the working database.

The consistency checks showed no significant inconsistencies.

While visiting the Velocity's field office in Ivaylovgrad, the author compared sample identifiers and down-hole intervals shown by original field sampling sheets to database entries for 6,328 intervals from 20 Velocity holes. These checks showed no significant inconsistencies.

As summarized in in Table 12-1, for all routine down-hole assay intervals from Velocity's drilling, and 93% of Velocity's exploration samples, the author compared database assay entries with laboratory source files supplied by Velocity. These checks showed no inconsistencies.

Table 12-1: Database vs Laboratory source file checks for Velocity exploration and drilling

Group	Type	Number of assays	Checked	
			Number	Proportion
Exploration	Stream sediment	66	55	83%
	Soil	3,189	2,900	91%
	Rock chip	106	106	100%
	Trench	846	846	100%
	Subtotal	**4,207**	**3,907**	**93%**
Diamond drilling		25,060	25,060	100%
TOTAL		**29,267**	**28,967**	**99%**

With no original records available, Hereward and Asia Gold data cannot be verified with the same degree of rigour as Velocity's data. However, the author's checks including review of duplicate assays and nearest-neighbour comparisons indicate that these data are generally reliable.

The author considers that the Tintyava exploration and drilling data has been sufficiently verified to form the basis of the current Mineral Resource estimates and exploration activities, and that the database is adequate for the Mineral Resource estimates and exploration activities.

13 Mineral Processing and Metallurgical Testing

This section covers the extensive testing program conducted for the Rozino ore types. It also includes summaries of historical testwork that were used to develop the process flowsheet and its design.

13.1 Summary

The starting point for the design of the Flotation Plant was historical testwork undertaken by Eurotest Control (ETC) for the 2018 PEA. In addition to this historical work a comprehensive testwork program was established and completed as part of the PFS project phase.

13.1.1 Ore Description

Rozino is a low sulphidation epithermal (LSE) gold deposit, hosted predominantly by Palaeogene breccia and conglomerate sedimentary rocks.

Mineralization includes disseminations, replacement, and veins with pyrite (with rare traces of base metals) and arsenopyrite. Gold is present at sulphide mineral boundaries and to a lesser degree as free grains or encapsulated inclusions.

13.1.2 Sampling and Representivity

The samples for testwork originated from contiguous drill core samples. Several intervals were composited to form "master composites" for ore zone testwork. The intervals were selected from drill core to generate representative samples of the entire body of mineralization or specific sub-regions and included the expected range of grades the Flotation Plant would experience.

13.1.3 Ore Types

The main ore types that identified in this study are:

- Oxide
- Transitional
- Sulphide.

These ore types are identified in the PFS metallurgical testwork as having sufficient differences in recovery, process reagents and other qualities (leading to cost differences) such that they merit separate identification.

The ore types are based on the degree of oxidation of the rock, where Oxide is the most weathered rock, Sulphide has little to no weathering, and Transitional is partly weathered. The Mineral Resource model identifies each block with an ore type. The ore types in the Mineral Resource model were developed by geologists working for Velocity Minerals.

The drill core photographic database was logged visually to describe for all intervals the ISRM weathering code. The ISRM method added additional rigour to the process of identification of oxidation state. The metallurgical composites were then assigned ISRM average weathering codes. As is explained in the following sections, the metallurgical testwork demonstrated high correlation of the ISRM coding, sulphur content and metallurgical performance. Sulphur content increases between Oxide to Sulphide ore, as does gold recovery to the bulk concentrate. However, sulphur content was not measured throughout the assay database and so is not included in the Mineral Resource estimate. The ISRM weathering code of the composites was related back to the Mineral

Resource ore types (Oxide, Transitional and Sulphide) and so ties together the relationship between metallurgical performance and ore type.

The ISRM weathering code system is provided here for reference as Table 13-1.

Table 13-1: A visual scale for logging of weathering in core photos modified after ISRM (1981)

Term	Description	Code
Fresh	No discoloration. Rings when struck	0
Fresh Jointed	Discoloration is limited to surface of, and 1–2 cm from fractures. Discoloration less than 10% of rock. Rings when struck.	1
Slightly Weathered	Discoloration extends out from joints into the rock and affects less than 40% of rock – OR there is Very Weak pervasive iron staining overall.	2
Moderately Weathered	Discoloration extends from joints and fractures, generally throughout the rock, the rock is intact not friable. Textures are preserved. Discoloration affects 40–100% of the rock.	3
Highly Weathered	Discoloration throughout, the material is friable but hard cores and occasional fresh rock cores may remain, textures are preserved.	4
Extremely Weathered	Decomposed, discolored resembles a soil, but rock textures may be preserved.	5

Using the drillhole database coded for ISRM weathering:

- The Mineral Resource Oxide ore type is on average "moderately" weathered
- The Transitional ore type is on average between "moderately" and "slightly" weathered
- The Fresh ore type is one average between "fresh" and "fresh jointed".

There can be local or patchy variations of weathering within the ore type zones; the ore type zones are not completely uniform.

13.1.4 Testwork

Testwork on the Rozino ore types was conducted for comminution and metallurgical properties. Table 13-2 outlines the different testwork categories

Table 13-2: Rozino PFS testwork program

Comminution testwork	Metallurgical investigations
• CWi - impact crushing index • Ai – abrasion index • SMC tests • BWi - ball mill grindability (closed screen size of 106 µm) • RWi - rod mill grindability index	• Diagnostic leach tests • CIL tests (fresh feed) • GRG testwork • Bulk gravity testwork • High intensity leach tests (gravity concentrate) • Flotation testwork • CIL of reground products (flotation concentrate) • Settling tests on flotation products

The major findings of the testwork programs undertaken to support the PFS flowsheet and design are detailed in Table 13-3.

Table 13-3: Major findings from PFS testwork program

Testwork findings	Conclusions
The ore is of medium hardness and is suitable to both conventional 3-stage crush and SAG-ball mill circuits: • Axb of 95.4 (oxide) to 50.7 (Transitional) • BWi of 11.23 kWh/t (Oxide) to 12.49 kWh/t (Transitional).	A 3-stage crush- ball mill circuit configuration was selected as most optimal for the different ore types. The orebody is fairly homogeneous with respect to comminution properties. Oxide ore exhibits the softest properties.
Gravity recovery (theoretical) varies by ore type: • Oxide at 45.2% • Transitional at 47.2% • Sulphide at 62.5%.	The ore contains a moderate level of gravity recoverable gold, but gravity alone is not sufficient to obtain a high overall recovery.
A gravity recovery product has an initial gold recovery of 95% in an intensive leaching circuit with an ultimate recovery of 97% after regrinding and CIL.	The gravity gold leaches reasonably well. However, the slow leach kinetics and fine liberation size required renders it not particularly amenable to treatment using a batch leach reactor.
The grinding circuit product (P_{80} 75 µm) responds well to flotation, a high recovery percentage in the mid-90s is obtained during processing of the sulphide ore, with fairly low mass pulls of 4-5% by weight.	The flotation circuit includes a rougher-scavenger circuit and a single cleaning stage to reduce the concentrate mass pull and increase the gold grade treated by the CIL circuit.
Flotation concentrate responds favorably to cyanide carbon in leach (CIL), with recoveries in the high 90s for Oxide/Transitional concentrate, approximately 82% for the sulphide concentrate; after the concentrate is subjected to an optimum regrind size P_{80} 20 µm.	Leach kinetics are fairly slow, and a reasonably fine regrind size (P_{80} 20 µm) is required to ensure high gold extraction. Cyanide consumption rates are low.
Copper is recovered to the flotation concentrate and is readily soluble by cyanide.	A cold cyanide strip is required in the elution cycle to remove copper.

The recovery expectations were extracted from testwork and subjected to a series of de-rating factors, which account for a discount of approximately 0.5% from laboratory testwork conditions. The results show that gold recovery is dependent on the degree of ore oxidation and the predicted overall weighted recovery of the combined processes is set at 79.3%. This includes a CIL solution loss of 0.5%.

In this report particle size is described as, for example, "P_{80} 20 µm" meaning 80% of the particles are less than 20 µm.

13.2 Historical Metallurgical Testwork

The most relevant historical testwork was carried out by ETC in 2018 as part of the scoping carried out to support flowsheet development and metallurgical recoveries in the 2018 PEA.

In January 2018 a metallurgical composite sample was prepared for detailed flotation and cyanidation tests. This consisted of 1,621 samples of coarse rejects from 124 mineralized intervals in 36 exploration diamond drill holes. The final composite sample weight was 130 kg.

The drill holes from which the metallurgical samples were collected are distributed across the entire deposit covering the preliminary block model (Figure 13-1) and represent the main Sulphide and Transitional mineralization. The length of each sample was one metre, but a small number varied by +/-20% due to a core loss or lithological boundaries.

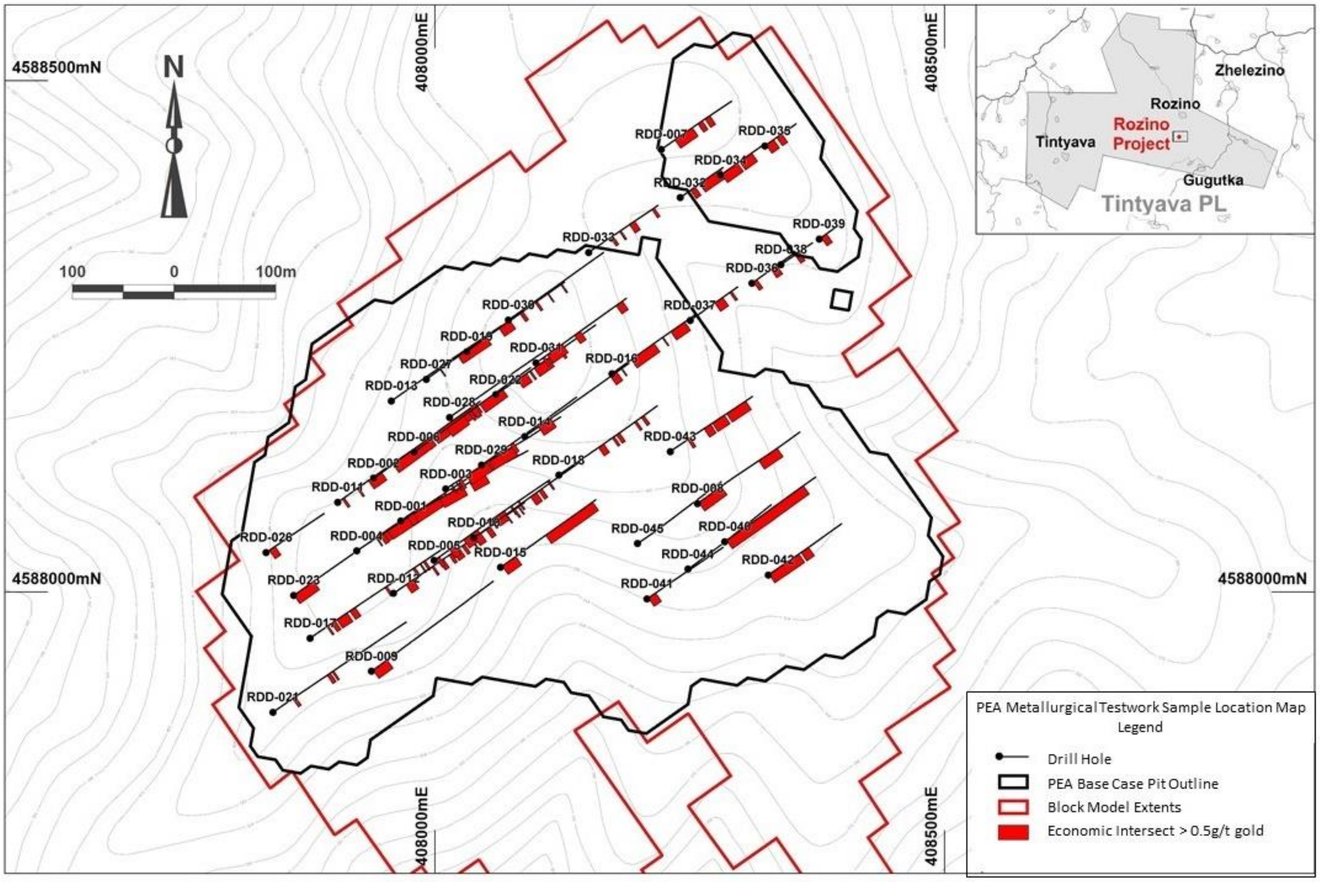


Figure 13-1: Plan view of PEA exploration drillholes
Source: Velocity, 2020

The material used for the composite samples was -1.7 mm coarse reject material returned from the assay laboratory after completing exploration assays. Each sample was split five times using a Jones riffle splitter. The initial sample parameters are shown in Table 13-4.

Table 13-4: Exploration drillhole sample intervals, January 2018

Parameter	Range		Average
	From	To	
Sample interval depth	2.0 m	144.7 m	
Sample interval length			1 m
Initial sample weight: - Half core - Quarter core	 0.62 kg 0.14 kg	 6.50 kg 2.18 kg	 2.54 kg 0.71 kg
5th single split weight	10 g	200 g	80 g
Sample grades	0.009 g/t	146 g/t	1.24 g/t
Sample interval grades	0.05 g/t	2.99 g/t	1.05 g/t

13.2.1 PEA Sample Preparation

The received sample had a total weight of 130 kg and a grain size of P_{80} 1.7 mm. The Australian Standard; AS 3988-91 "Copper, Lead, Zinc, Gold and Silver Ores - Guidance for the Preparation of Gold Determination Tests", was used to prepare the resource global composite sample.

13.2.2 PEA Head Assays

The head assay of the 2018 PEA Master Composite is shown in Table 13-5.

Table 13-5: 2018 PEA Master Composite head assays

Element	Units	Assay
Au	g/t	1.28
S	%	0.42
C	%	1.20

The results of an ICP analysis carried out on the 2018 PEA Master Composite sample are shown in Table 13-6.

Table 13-6 PEA 2018 Master Composite head sample ICP analysis

Element	Units	Content	Element	Units	Content
Ag	g/t	2.0	Mn	g/t	1,111
Al	%	1.37	Mo	g/t	5.0
As	g/t	125	Na	g/t	377
B	g/t	<1	Ni	g/t	49
Ba	g/t	62	P	g/t	397
Be	g/t	<1	Pb	g/t	28
Bi	g/t	<5	Sb	g/t	<5
Ca	%	2.58	Sn	g/t	<2
Cd	g/t	<1	Sr	g/t	47
Co	g/t	11	Te	g/t	<2
Cr	g/t	54	Ti	g/t	97
Cu	g/t	48	Tl	g/t	<2
Fe	%	2.79	V	g/t	18
Ga	g/t	<1	W	g/t	<20
K	g/t	3,014	Y	g/t	12
La	g/t	18	Zn	g/t	74
Li	g/t	23	Zr	g/t	1.3
Mg	g/t	9,375			

Data from the chemical analyses on the 2018 PEA Master Composite characterize it as gold bearing (1.28 g/t Au), with other valuable components (silver 2 g/t; copper 48 g/t; lead 28 g/t and zinc 74 g/t). The potential penalty elements in the ore are arsenic at 125 g/t and chromium 54 g/t. Antimony at <5 g/t and cadmium at <1 g/t are negligible and are not expected to affect the quality of the final products.

13.2.2.1 X-Ray Diffraction of 2018 Master Composite

Radiographic studies of the 2018 Master Composite sample were carried out using an automated X-ray diffractometric system (D 500 SIEMENS) with computer control, copper-monochromatic radiation at 40 kV and 30 mA and inlet aperture blend.

The results of the XRD mineralogical analysis are shown in Table 13-7.

Table 13-7: PEA 2018 Master Composite XRD analysis

Mineral	Content (wt.%)
Quartz (SiO_2)	61
Clinochlore {$[Mg,Al,Fe]_6[Si,Al]_4O_{10}(OH)_8$}$_{11}$	11
Orthoclase ($KAlSi_3O_8$)	11
Muscovite {$KAl_2[AlSi_3O_{10}](OH)_2$}	7
Dolomite {$CaMg(CO_3)_2$}	6
Calcite ($CaCO_3$)	3
Pyrite (FeS_2)	1

The X-ray analysis measures the sulphur content as 0.42% and the iron content as 2.79%. The iron is present in oxide form and in the sulphide mineral pyrite; the sample had an average pyrite content of approximately 1%.

The predominant gangue mineral is quartz, assaying at 61%. The other main gangue components are represented as clinochlore, orthoclase, muscovite, and dolomite minerals. The organic carbon content of the sample was determined to be 1.2%.

13.2.3 Mineralogy of 2018 PEA Master Composite

The mineralogical analysis carried out on the gold bearing samples indicated that the predominant minerals are as follows:

- Primary minerals: pyrite, arsenopyrite, chalcopyrite, sphalerite, magnetite
- Vein minerals: quartz, calcite
- Supergene minerals: hematite and/or limonite, bornite.

The primary sulphide mineral is pyrite. The remaining minerals occur as single grains, with chalcopyrite more common. Pyrite precipitates as idiomorphic and hypidiomorphic grains most commonly as aggregates, but more rarely with chalcopyrite and sphalerite. The supergene changes have mostly affected pyrite which in some places occurs as pseudomorphs altered completely to hematite or limonite.

Free gold was not visually observed in the sample. From the vein minerals, only quartz and calcite were observed. Hematite and limonite are also variably prevalent. Several bornite grains were identified.

13.2.4 PEA Comminution Test

13.2.4.1 Bond Work Index Test

A Bond Ball Mill Work Index determination was carried out at the University of Sofia on the 2018 Master Composite sample. The results are shown in Table 13-8.

Table 13-8: Master Composite Bond Work Index test results

Feed particle size (F_{80} mm)	Product particle size (P_{80} mm)	Product specific mass (g/rev)	Energy consumption (kWh/g)	Bond Work Index (kWh/t)
1.8	0.095	1.87	$5.7.10^{-3}$	14.52

Based on the Bond Ball Mill Work Index test results, the material is considered to be moderate to hard.

13.2.5 Diagnostic Leach Tests

Diagnostic leach tests were carried out on the 2018 Master Composite sample at two different grind sizes:

- P_{80} 75 µm (80% passing 75 µm)
- P_{90} 75 µm.

The results of the P_{80} 75 µm grind size test are shown in Table 13-9.

Table 13-9: 2018 Master Composite diagnostic leach test results – P_{80} 75 µm (Test 1)

Stage	Au g/t	Gold phase	Au %	Cum %Au
Hg amalgamation	0.459	Free	58.61	58.61
Primary cyanidation	0.142	Cyanide soluble	28.58	87.19
Secondary cyanidation	0.138	Locked in Iron Oxide minerals	1.82	89.01
After chemical destruction	0.071	Locked in Sulphide minerals	7.81	96.82
		Associated/locked in silicates	3.17	100
Feed	1.109		100	

The results in Table 13-9: indicate that gold recovery by gravity could be up to 59%. The predicted cyanide recoverable gold content by adopting conventional cyanidation and CIL is 87.2%.

The analysis suggests that 7.81% of the gold in the sample is locked in pyrite and not be recoverable by cyanidation. Finely dispersed gold mineralization associated with silicates (3.17%) will also not be recoverable by cyanidation. Results of the phase analysis carried out at a grind size of P_{90} 75 µm is shown in Table 13-10.

Table 13-10: 2018 Master Composite diagnostic leach test results – P_{90} 75 µm (Test 2)

Stage	Au g/t	Gold Phase	Au %	Cum %Au
Hg Amalgamation	0.543	Free	52.16	52.16
Primary cyanidation	0.139	Cyanide soluble	35.59	87.75
Secondary cyanidation	0.144	Locked in Iron Oxide minerals	1.03	88.78
After chemical destruction	0.080	Locked in Sulphide minerals	7.72	96.50
		Associated/locked in silicates	3.49	100
Feed	1.135		100	

Results in Table 13-10 indicate that gold recovery by gravity could be up to 52%, while the predicted cyanide recoverable gold content by adopting conventional CIL is 87.8%. The analysis suggests that 7.72% of the gold in the sample is locked in pyrite, and that the finely dispersed mineralization associated with silicates (3.49%) is also not recoverable by cyanidation.

The results confirm the limitations of gold recovery by gravity with maximum theoretical recoveries at 52 and 59%, and that gold recovery using conventional cyanide leaching and CIL is at 88 to 89%. Based on the phase analysis, optimal gold recovery by CIL would be achieved at the finer grind size of P_{90} 75 µm.

13.2.6 2018 PEA Master Composite Whole Ore Cyanidation Leach Tests

Whole ore cyanidation leach tests were carried out to determine the potential gold extraction and leach kinetics. Mechanically stirred agitation leach tests at grind sizes of P_{60} 75 µm and P_{80} 75 µm were undertaken. Tests were carried out at a cyanide concentration of one g/l for 48 hours. Results of the whole ore cyanidation leach tests are summarized in Table 13-11.

Table 13-11: 2018 whole ore cyanidation leach test results

Grind size (µm)	Recovery (%)	Reagent consumption (kg/t)	
		Lime	Cyanide
P_{60} 75 µm	69.8	0.37	2.14
P_{80} 75 µm	85.9	0.37	2.98

The gold leach kinetic curves are shown in Figure 13-2.

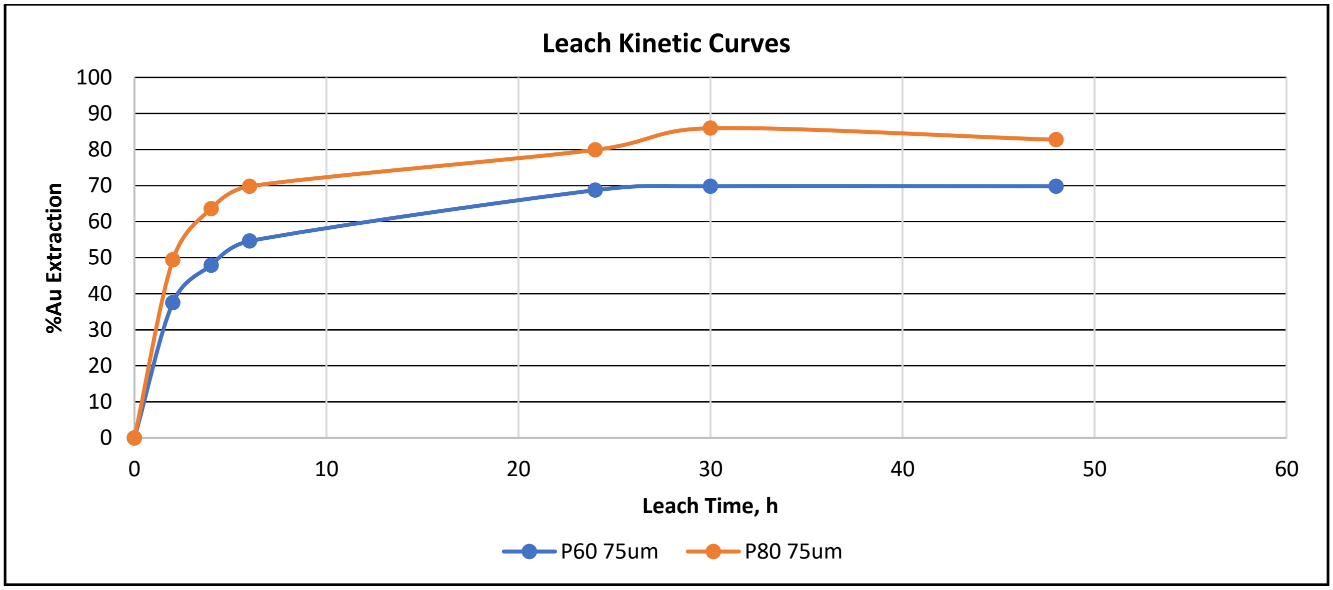


Figure 13-2: 2018 Gold leach kinetic curves

The whole ore cyanidation leach results indicate that a higher gold extraction was achieved at the finer grind size P_{80} 75 µm. Leach kinetics are rapid; 86% of the gold is extracted after 30 hours of leaching. The leach curves do indicate some minor signs of "preg-robbing" due to the presence of organic carbon in the ore.

The gold extraction obtained from mechanically agitated leach tests corresponds well with the results of the phase analysis, where grinding to P_{80} 75 µm resulted in a combined gold recovery by cyanide leaching of 87.2%.

13.2.7 2018 PEA Flotation Tests

13.2.7.1 PEA Batch Rougher Tests

Various batch rougher flotation tests were carried out to determine the effect of:

- Grind size
- Collector type
- pH.

The results of the scoping rougher test carried out at a grind size P_{80} 75 µm, natural pH, and with the addition of PAX as collector, and MIBC as frother are shown in Table 13-12.

Table 13-12: PEA Rougher Test #1 results

Product	Mass Pull		Content		Recovery, %	
	g	%	Au g/t	S %	Au	S
>1 mm	29	1.5	1.76	0.10	1.8	0.3
Rougher Concentrate	110	5.5	23.3	9.2	89.2	95.6
Middling 1	153	7.6	0.58	0.18	3.1	2.6
Tailing	1,708	85.4	0.10	0.01	6.0	1.6
Feed	**2,000**	**100**	**1.44**	**0.53**	**100**	**100**

Table 13-12 shows that 89.2% of the gold is recovered as a gold-bearing pyrite concentrate assaying 23.3 g/t Au.

13.2.7.2 PEA Locked Cycle Tests

Locked cycle tests (LCTs) were carried out at the optimum grind size P_{80} 75 µm, with and without the addition of the promoter Aerofloat 242 (A242) added as a gold collector. The circuit configuration adopted for the LCTs included a rougher/scavenger flotation stage, with single-stage cleaning of the rougher concentrate. The scavenger concentrate was recycled back to the head of the rougher circuit. The results of the LCTs are shown in Table 13-13 and Table 13-14.

Table 13-13: LCT #1 results

Cycle	Product	Yield		Content		Recovery, %	
		g	%	Au g/t	S %	Au	S
1	Concentrate	92.97	4.65	23.80	10.78		
	Tailing			0.11	0.01		
	Feed						
2	Concentrate	79.88	3.99	23.90	13.72	89.39	98.28
	Tailing			0.12	0.01		
	Feed		100.00	1.07	0.56		
3	Concentrate	71.03	3.55	33.30	15.88	90.41	98.32
	Tailing			0.13	0.01		
	Feed		100.00	1.31	0.57		
4	Concentrate	54.89	2.74	39.50	20.48	90.06	98.30
	Tailing			0.12	0.01		
	Feed		100.00	1.20	0.57		
5	Concentrate	69.12	3.46	33.30	17.31	91.27	98.41
	Tailing			0.11	0.01		
	Feed		100.00	1.26	0.61		

Table 13-14: LCT #2 results

Cycle	Product	Yield		Content		Recovery, %	
		g	%	Au g/t	S %	Au	S
1	Concentrate	35.36	1.78	59.1	30.08		
	Tailing			0.114	0.01		
	Feed						
2	Concentrate	87.04	4,35	22.4	12.46	91.15	98.27
	Tailing			0.099	0.01		
	Feed		100,00	1.07	0.55		
3	Concentrate	72.75	3.64	30.2	17.27	91.27	98.49
	Tailing			0.109	0.01		
	Feed		100.00	1.20	0.64		
4	Concentrate	81.81	4.09	27.8	13.33	90.12	98.27
	Tailing			0.13	0.01		
	Feed		100.00	1.26	0.55		
5	Concentrate	113.11	5.66	20.4	10.77	92.79	98.47
	Tailing			0.095	0.01		
	Feed		100.00	1.24	0.62		

The addition of A242 resulted in a more stable froth and better flotation conditions. Balancing the streams in closed cycle demonstrated that the grade-recovery relationships for both circuits are very similar, with the average of the last three cycles resulting in 90.6% gold recovery at a final concentrate grade of 35.4 g/t Au using the A242 collector, and 91.4% gold recovery at a final concentrate grade of 26.1 g/t Au using potassium iso-butyl xanthate (KIBK).

The results of the final concentrate analysis are shown in Table 13-15.

Table 13-15: 2018 final concentrate analysis

Element	Units	Assay
Ag	g/t	41
As	g/t	1,993
Ba	g/t	20
Cr	g/t	143
Cu	%	0.13
Mo	g/t	26
Pb	%	0.044
Zn	%	0.057

The final concentrate analysis indicates that an arsenic content of 0.2% can be expected which may attract penalties if the concentrate were sold to a concentrate trader. The X-ray diffraction results for the final concentrate are shown in Table 13-16. These results indicate that the cleaner concentrate is gold-bearing (41 g/t) with pyrite as the dominant sulphide mineral and elevated levels of copper (0.3%) and arsenic (0.2%).

Table 13-16: 2018 XRD results – final concentrate mineralogy

Mineral	Content, %Wt.
Quartz (SiO_2)	37
Clinochlore {$[Mg,Al,Fe]_6[Si,Al]_4O_{10}(OH)_8\}_{11}$	10
Microcline ($KAlSi_3O_8$)	6
Muscovite {$KAl_2[AlSi_3O_{10}](OH)_2$}	8
Dolomite {$CaMg(CO_3)_2$}	3
Calcite ($CaCO_3$)	2
Pyrite (FeS_2)	32
Siderite ($FeCO_3$)	1

13.2.8 PEA Flotation Concentrate Cyanidation Leach Tests

Cyanidation leach tests were carried out on the cleaner flotation concentrate to determine the ultimate gold extraction, and the reagent consumptions. Cyanidation leach tests were carried out with and without regrinding of the cleaner concentrate. Leach tests were carried out at a cyanide concentration of 2 g/l, and a leach residence time of 48 hours. Sodium hydroxide was used in place of lime.

The results of the concentrate cyanidation leach tests are shown in Table 13-17.

Table 13-17: 2018 PEA concentrate leach test results – final concentrate

Grind size	Recovery (%)	Reagent consumption (kg/t)	
		Caustic	Cyanide
87.95% passing 75 µm	87.5	5	3.1
80% passing 20 µm	88.1	5	3.1

Regrinding of the flotation concentrate increased the gold extraction by only 0.6%. The content of gold in the tails is relatively high at 3.3 g/t. These gold losses, based on the phase analysis, are probably due to finely dispersed gold in pyrite which is not leachable by cyanidation.

13.2.9 PEA Gravity Tests

Gravity tests were carried out using a laboratory-sized Knelson KC-MD3 concentrator unit. Two gravity tests were carried out on the Master Composite, the first with no grinding (particle size P_{80} 1.7 mm), and on a sample reground to P_{80} 75 µm using a sample weight of 10 kg.

The results of the two batch gravity tests are shown in Table 13-18and Table 13-19.

Table 13-18: PEA gravity test results – no grinding

Product	Mass		Grade		Recovery	
	g	%	Au g/t	S %	Au %	S %
>1 mm	2,090	20.9	1.27	0.29	19.6	15.3
Pan concentrate	7	0.07	201	39.8	10.3	7.0
Residue	93	0.93	10.2	6.45	7.0	15.2
Final tails	7,810	78.1	1.10	0.32	63.1	62.5
Feed	**10,000**	**100**	**1.36**	**0.39**	**100**	**100**

Table 13-19: PEA gravity test results – regrinding

Product	Mass		Grade		Recovery	
	g	%	Au g/t	S %	Au %	S %
>1 mm	52	0.52	12.2	0.13	5.00	0.13
Pan concentrate	12	0.12	479	40.0	44.3	8.9
Residue	95	0.95	11.8	3.67	8.93	6.7
Final tails	9,841	98.4	0.54	0.45	41.7	84.3
Feed	**10,000**	**100**	**1.26**	**0.53**	**100**	**100**

The results show that at the finer grind size (P_{80} 75 µm), the gravity recoverable gold (GRG) component increases about four-fold over the unground sample, from 10% to 44% gold recovered, with the pan concentrate assaying 480 g/t Au. It is evident that a portion of the gold in the feed is recovered with pyrite into the gravity concentrate.

13.2.10 PEA Gravity-Flotation Test

A gravity-flotation flowsheet was tested on the 2018 Master Composite ore sample. In this flowsheet the ore sample was subjected to gravity concentration, followed by flotation of the gravity tailings.

The results of the gravity and flotation test on the gravity tailings are shown in Table 13-20and Table 13-21.

Table 13-20: PEA gravity test results (P_{80} 75 µm)

Product	Mass		Content		Recovery, %	
	g	%	Au g/t	S %	Au	S
>1.7 mm	51.7	0.52	12.23	0.13	5.0	0.1
Pan Concentrate	11.7	0.12	479	40.0	44.3	9.5
Residue	95.3	0.95	11.85	3.67	8.9	7.1
Tailings	9,841	98.4	0.54	0.42	41.7	83.2
Feed	**10,000**	**100**	**1.26**	**0.49**	**100**	**100.**

Table 13-21: PEA Flotation Test Results (Gravity Tailings)

Product	Mass		Content		Recovery, %	
	g	%	Au g/t	S %	Au	S
Concentrate	16.4	0.82	37.5	27.1	53.7	55.5
Mid. Product 2	53.3	2.67	1.47	4.9	6.8	29.2
Mid. Product 1	31.2	1.56	2.79	3.3	7.6	13.0
Final Tailings	1,899	94.9	0.19	0.01	31.8	2.4
From Gravity Test №2	**2,000**	**100**	**0.57**	**0.40**	**100**	**100**

The combined gravity-flotation recovery of 85% was lower than that achieved by flotation alone, and therefore no gravity stage is justified in the final gold recovery circuit as defined by this testwork undertaken on the 2018 PEA Master Composite. However, the testwork is not exhaustive; additional testwork may yield a better understanding of the relative merits of gravity-flotation vs flotation recovery.

13.3 PFS Sample Origin

The PFS testwork program was undertaken on a number of Master Composite samples generated in 2019 and 2020. One of the composites chosen in the 2019 campaign originally as Transitional was later determined as being Oxide. This composite is described, for clarity, as Oxide-2.

13.3.1 2019 Master Composites

13.3.1.1 Phase 1

A total of 58 intervals of quartered drill core material, weighing 802 kg, were submitted to WAI in September 2019 for testing. The material submitted represented the three main material types, and two different mineralization types. A summary of the samples submitted is given in Table 13-22.

Table 13-22: Summary of samples submitted

Sample ID	Number of intervals	Mass (kg)
Oxide Master Composite	12	190
Transitional Master Composite[1]	15	177
Sulphide Master Composite	19	266
Vein Zone Composite	5	70
Olistostrome Composite	7	99
Total	**58**	**802**

Note 1: Transitional master composite was later designated Oxide 2.

PFS - PHASE 1

Samples Sent to WAI	kg	PFS Report name
Oxide Master Composite	189.8	Oxide 1
Transitional Master Composite1	177.2	Oxide 2
Sulphide Master Composite	266.0	Sulphide 1
Vein Zone Composite	70.3	
Olistostrome Composite	98.7	

Inter-laboratory calibration sample

Samples sent to ETC		kg
REM001	Low grade	30
REM002	High grade	30
REM003	Vein zone	30
REM004	Olistostrome	30
REM005	Material Blend	30
REM006	Sulphide-1	30

In November to December 2019, reviews of the metallurgical results resulted in some required changes to the testwork program.

1. An intense investigation of the oxide-recovery performance, which required the application of a more stringent logging method to be applied to the oxidation. The application of the ISRM logging method found a strong ISRM code to recovery performance. Back analysis of the Transitional Master Composite found that the composite was in reality very close to average oxide logging. A new Transitional master composite would be required.

2. The variability testing demonstrated that a grade-recovery relationship existed. The grade of the Sulphide Master Composite was lower than the Reserve average and so this triggered the requirement of a second Sulphide Master Composite with a grade closer to the Mineral Reserve average.

3. The variability testing combined with the grade and oxide relationships demonstrated that no other tested variability control existed and no further testing was required on Olistostrome, Vein or Material Blends.

4. The inter-laboratory calibration test demonstrated equivalent testing procedures between ETC and WAI

PFS - PHASE 2

Samples Sent to WAI	kg	PFS Report name
Transition 1 - February 2020	119	Transition 1
Transition 2 - May 2020	25	Transition 2

Samples Sent	kg	PFS Report name
REM022	228	Sulphide 2

Figure 13-3: PFS phase description

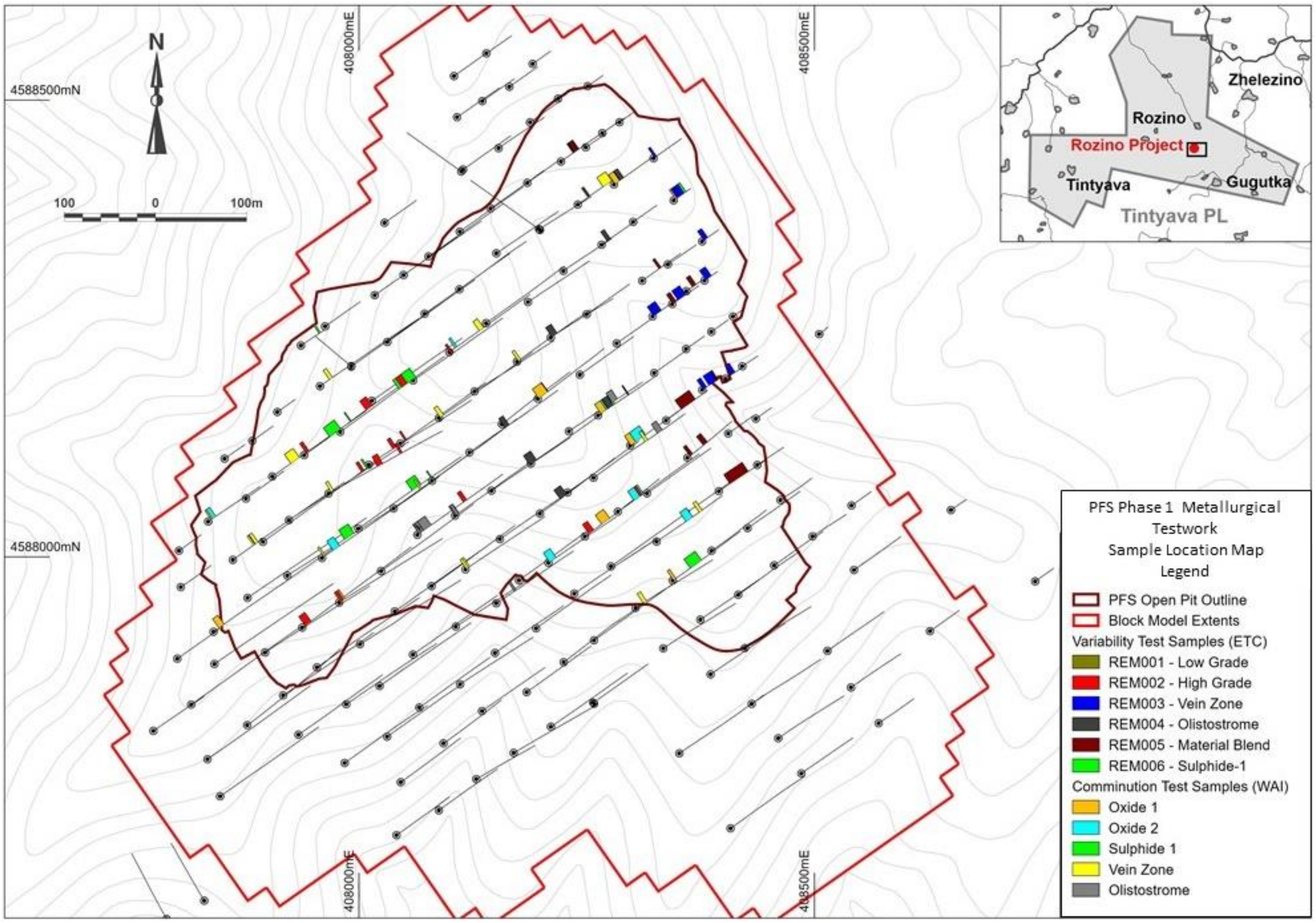


Figure 13-4: Location of Phase 1 metallurgical testwork samples
Source: Velocity 2020

Upon receipt, each sample was weighed, photographed and logged into the laboratory sample tracking system. Photographs of the as-received samples are shown below in Figure 13-5 to Figure 13-9.

Figure 13-5: Oxide Master Composite, RDD044, 36.3-53.9 m interval
Source: WAI, 2019

Figure 13-6: Oxide Master Composite, RDD044, 36.3-53.9 m interval
Source: WAI, 2019

Figure 13-7: Sulphide Master Composite, RDD005, 113.9-122.4 m interval
Source: WAI, 2019

Figure 13-8: Vein Zone Composite, RDD032 52.1-61.9 m interval
Source: WAI, 2019

Figure 13-9: Olistostrome Composite, RDD043 87.6-98.6 m interval
Source: WAI, 2019

13.3.1.2 Phase 2

Two samples of Transitional material identified as "Transition 1" and "Transition 2" were submitted to WAI in February and May 2020 for testing. The samples were prepared from reserve assay rejects. A summary of the samples received is given in Table 13-23. Figure 13-10 shows the collection location of Phase 2 metallurgical testwork samples.

Table 13-23: Summary of samples received

Sample ID	Mass (kg)
Transition 1	119.0
Transition 2	25.0

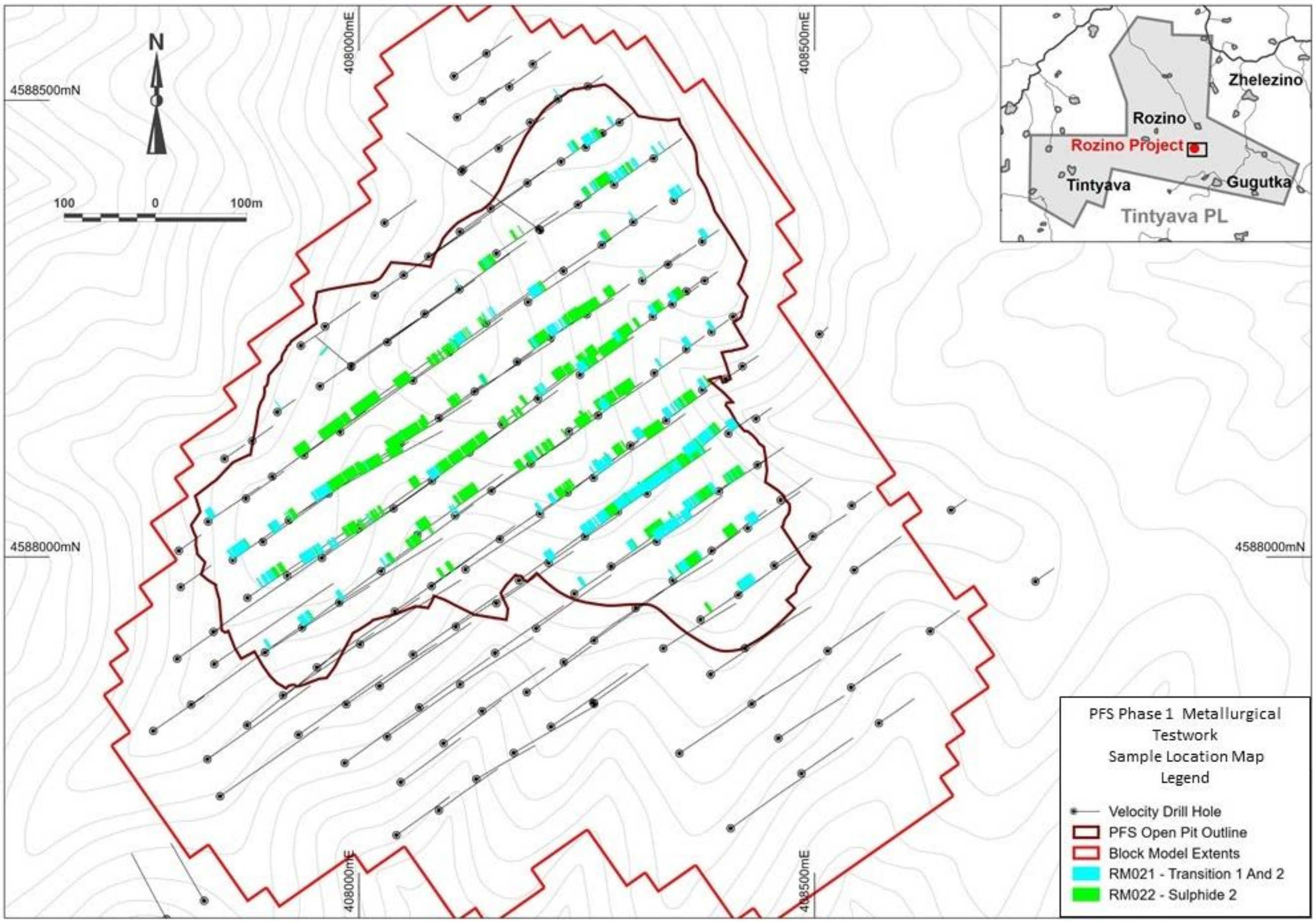


Figure 13-10: Location of Phase 2 metallurgical testwork samples
Source: Velocity, 2020

Upon receipt, each sample was weighed, photographed and logged into the laboratory sample tracking system. Photographs of the as-received samples are shown below in Figure 13-11 and Figure 13-12.

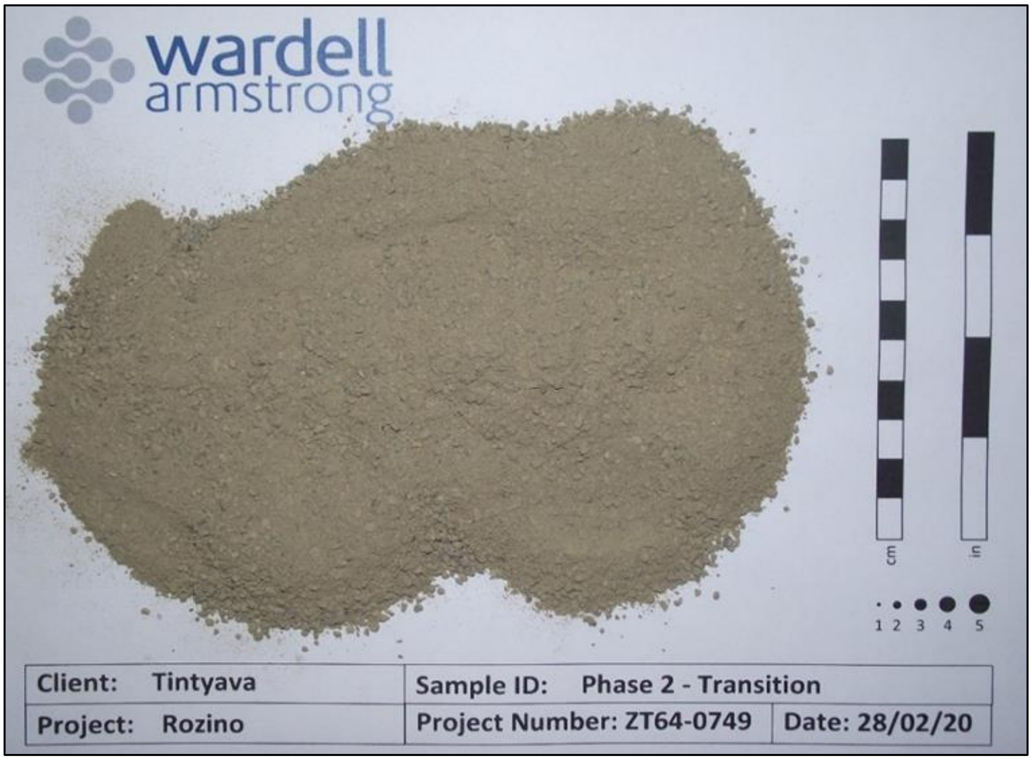


Figure 13-11: Transition 1 sample
Source: WAI, 2020

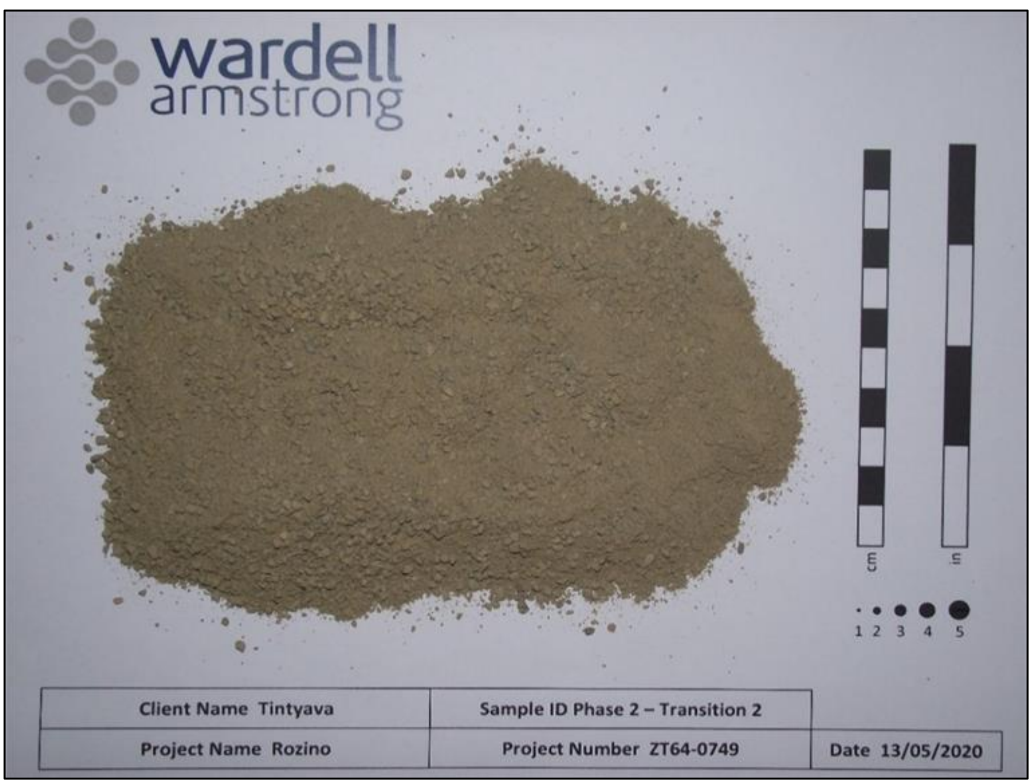


Figure 13-12: Transition 2 sample
Source: WAI, 2020

Variability testwork was undertaken by Eurotest Control (ETC) in two separate phases. Samples represented the following variability material types:

- Low-grade ore
  - The sample was selected to target ore around the cut-off grade. The sample selected measured 0.30 g/t gold, slightly below the cut-off grade, but met the goal.
- High-grade ore
  - The grade of the ore was targeted to be 3-4 times higher than the mean. The composite grade after selection was 7.6 g/t gold, which met the goal.
- Vein zone
  - This sample was selected to represent what was potentially a different ore type based on early geological logging of the Project.
- Olistostrome LOM grade
  - The Olistostrome material is slightly softer rock than the sediment and was considered to be potentially different in processing.
- Material blend
  - A blend of samples from the eastern area of the Mineral Resource in order to test if there were any negative impacts to mixing ore types.

ETC also tested a subsample from the Wardell S-1 composite (for inter-laboratory verification) which was designated as REM006: Sulphide-1.

Phase 1

Six samples consisting of drill core from different drill cross-sections, with a weight of 30 kg each and a grain size of P_{80} 1.7 mm, were received by ETC. The variability samples were identified as shown in Table 13-24.

Table 13-24: Phase 1 variability sample description ETC

Sample ID	Sample description
REM001	Low grade
REM002	High grade
REM003	Vein zone
REM004	Olistostrome
REM005	Material Blend
REM006	Sulphide-1

Phase 2

A review of the Phase 1 metallurgical results indicated a grade-recovery relationship but that the grade of the Sulphide 1 composite was such that it would not provide an accurate prediction of metallurgical performance for the Mineral Reserve. A second sulphide composite with a grade closer to the likely Mineral Reserve was collected. The sulphide sample was designated as REM022 (or also referred to as the Sulphide-2 composite).

A composite sample, comprising 977 intervals from different drill holes that intersected the sulphide zone of the ore body, with a weight of 228 kg and a particle size of P_{80} 1.7 mm, was provided to ECT for testing.

13.4 Head Ore Analysis

Chemical assays were performed on sub-samples representative of head ore grade of the three main composite samples submitted for metallurgical testing (Oxide 1/Oxide 2/Sulphide 1).

Samples were analysed for gold and silver using the "screened metallics" protocol, shown in Table 13-25 and Table 13-26, along with separate assays for a range of base metals, sulphur and carbon as summarized in Table 13-27.

Table 13-25: Screened metallics assay results – gold

Sample	+100 µm		-100 µm		Au distribution (%)		Calculated head assay Au (g/t)
	Wt. (%)	Au (g/t)	Wt. (%)	Au (g/t)	+100 µm	-100 µm	
Oxide 1	7.6	1.27	92.4	1.02	9.24	90.76	1.04
Oxide 2	9.4	1.46	90.6	1.03	12.84	87.16	1.07
Sulphide 1	5.5	1.16	94.5	0.84	7.50	92.50	0.86

Table 13-26: Screened metallics assay results – silver

Sample	+100 µm		-100 µm		Ag Distribution (%)		Calculated head assay Ag (g/t)
	Wt. (%)	Ag (g/t)	Wt. (%)	Ag (g/t)	+100 µm	-100 µm	
Oxide 1	7.6	<5	92.4	<5	7.55	92.45	<5
Oxide 2	9.4	5	90.6	6	7.97	92.03	6
Sulphide 1	5.5	6	94.5	<5	6.58	93.42	<5

The results showed gold ranging from 0.86 g/t for the Sulphide 1 sample to 1.07 g/t for the Oxide 2 sample. Silver grades were <5 g/t for the Oxide 1 and Sulphide 1 samples and 6 g/t for the Oxide 2 sample.

The proportion of coarse (+100 µm) gold present in the samples was between 7.5 and 12.8%. The proportion of coarse (+100 µm) silver present in the samples was between 6.58 to 7.97%.

Table 13-27: Head assays (Master Composites)

Element	Units	Assay		
		Oxide 1	Oxide 2	Sulphide 1
Au	g/t	1.04	1.07	0.86
Ag	g/t	<5	6	<5
Cu	%	0.005	0.004	0.004
Zn	%	0.007	0.007	0.008
$S_{(TOT)}$	%	0.017	0.036	0.860
$S_{(SUL)}$	%	0.003	0.024	0.840
$C_{(TOT)}$	%	0.41	0.35	1.71
$C_{(ORG)}$	%	0.13	0.14	0.13

The gold grades range from 0.86 g/t Au for the Sulphide 1 sample to 1.07 g/t Au for the Oxide 2 sample.

Each sample was also submitted for ICP multi-element analysis for a range of trace elements. The results are given in Table 13-28.

Table 13-28: ICP results (Master Composites)

Element	Units	Oxide 1	Oxide 2	Sulphide 1	Element	Units	Oxide 1	Oxide 2	Sulphide 2
Ag	g/t	2.86	3.57	1.35	Na	%	0.14	0.07	0.18
Al	%	5.79	4.40	5.47	Nb	ppm	11.7	8.5	11.7
As	ppm	83.0	97.5	423	Ni	ppm	65.5	49.0	52.5
Ba	ppm	340	320	330	P	ppm	460	400	480
Be	ppm	1.83	1.43	1.59	Pb	ppm	23.0	27.2	31.8
Bi	ppm	0.21	0.20	0.29	Rb	ppm	174.5	159.5	160.0
Ca	%	0.93	0.66	3.85	Re	ppm	0.002	0.002	0.002
Cd	ppm	0.10	0.13	0.12	S	%	0.02	0.04	0.84
Ce	ppm	48.0	42.3	48.2	Sb	ppm	3.27	3.72	4.35
Co	ppm	16.6	11.2	14.4	Sc	ppm	14.3	10.2	12.5
Cr	ppm	274	257	249	Se	ppm	1	<1	<1
Cs	ppm	5.51	4.95	4.70	Sn	ppm	2.0	1.8	2.0
Cu	ppm	45.3	40.7	43.7	Sr	ppm	58.9	47.9	89.3
Fe	%	3.36	2.41	3.16	Ta	ppm	0.79	0.59	0.77
Ga	ppm	14.90	12.00	14.40	Te	ppm	0.08	0.05	0.06
Ge	ppm	0.19	0.22	0.22	Th	ppm	7.94	7.53	8.46
Hf	ppm	0.3	0.3	0.5	Ti	%	0.342	0.224	0.314
Hg	ppm	0.058	0.077	0.032	Tl	ppm	1.56	1.54	1.34
In	ppm	0.048	0.042	0.047	U	ppm	1.5	1.8	1.8
K	%	3.40	3.01	3.31	V	ppm	105	76	97
La	ppm	23.3	21.1	23.9	W	ppm	3.8	3.3	3.2
Li	ppm	40.1	32.9	35.4	Y	ppm	20.7	15.8	18.7
Mg	%	0.76	0.37	1.02	Zn	ppm	68.0	64.0	79.0
Mn	ppm	1,370	769	1,150	Zr	ppm	10.7	10.4	19.4
Mo	ppm	15.95	23.20	18.95					

Detailed head assays were conducted on each of the Phase 2 Transitional master composite samples for a range of elements. Samples were analysed for gold and silver using a screened metallics protocol, as summarized in Table 13-29 and Table 13-30.

Table 13-29: Screened metallics assay results – gold

Sample	+100 µm		-100 µm		Au distribution (%)		Calculated head assay Au (g/t)
	Wt. (%)	Au (g/t)	Wt. (%)	Au (g/t)	+100 µm	-100 µm	
Transitional 1	10.0	1.80	90.0	1.02	16.44	83.56	1.10
Transitional 2	3.7	1.76	96.3	1.10	5.85	94.15	1.12

Table 13-30: Screened metallics assay results – silver

Sample	+100 µm		-100 µm		Ag distribution (%)		Calculated head assay, Ag (g/t)
	Wt. (%)	Ag (g/t)	Wt. (%)	Ag (g/t)	+100 µm	-100 µm	
Transitional 1	10.0	<5	90.0	<5	10.03	89.97	<5
Transitional 2	3.7	<5	96.3	<5	3.74	96.26	<5

The results showed gold grade in both samples to be similar at 1.10 g/t in the Transitional 1 sample and 1.12 g/t in the Transitional 2 sample. The silver grade in both samples was below the detection limit of 5 g/t. With respect to the distribution of gold, the results indicated that both samples contain low proportions of coarse gold with

16.4% contained within the +100 µm fraction in the Transitional 1 sample compared with just 5.8% in the Transitional 2 sample. This difference is, however, most likely attributable to the higher mass of sample in the Transition 1 +100 µm fraction (10.0% vs. 3.7%).

A separate analysis for copper, zinc, sulphur and carbon is shown in Table 13-31.

Table 13-31: Head assay results (Transitional Master Composites)

Element	Units	Assay	
		Transitional 1	Transitional 2
Au	g/t	1.10	1.12
Ag	g/t	<5	<5
Cu	%	0.004	0.004
Zn	%	0.007	0.007
$S_{(TOT)}$	%	0.130	0.086
$S_{(SUL)}$	%	0.110	0.073
$C_{(TOT)}$	%	0.36	0.39
$C_{(ORG)}$	%	0.07	0.07

The head assay results showed the concentration of base metals in both samples to be ≤0.007%. The total sulphur (S(TOT)) grades varied from 0.086% in the Transitional 2 sample to 0.13% in the Transitional 1 sample. The proportion of the total sulphur present as sulphide sulphur remained consistent at around 85% for both samples.

By comparison, the amount of carbon present in both samples was consistent with total carbon (C(TOT)) levels of between 0.36 to 0.39%. The organic carbon (C/ORG)) content of both samples is 0.07%.

The head assays for the various composites tested by ETC are reported in Table 13-32.

Table 13-32: Head assays (Variability Composites)

Sample	Au g/t	Ag g/t	As g/t	Cu g/t	Zn g/t	% TS	% S^{2-}	% TC	%OrgC
REM001	0.298	<1.0	65	39	62	0.083	0.97	0.32	0.35
REM002	7.633	9.5	136	66	171	0.059	0.72	0.46	0.66
REM003	0.742	1.9	75	31	54	0.049	1.33	0.12	0.17
REM004	1.081	2.4	96	54	86	0.220	1.95	0.29	0.30
REM005	0.784	1.5	64	29	58	0.068	1.18	0.23	0.24
REM006	0.845	2.0	390	43	70	0.130	1.67	0.70	0.83

13.5 Comminution Tests

The primary objective of the comminution testwork programme was to characterize the crushing and grinding characteristics of the ore with respect to both conventional crushing/grinding and autogenous/semi-autogenous (AG/SAG) grinding.

A total of five samples were submitted to WAI. Each sample was subjected to the following suite of tests:

- Bond Low Energy Impact Testing
- SAG Mill Comminution (SMC) Testing
- Bond Abrasion Index Testing
- Bond Rod Mill Work Index Testing
- Bond Ball Mill Work Index Testing.

A summary of the comminution test results is shown in Table 13-33.

Table 13-33: Summary of comminution test results

Comminution test		Units	Sample description				
			Oxide 1	Oxide 2	Sulphide 1	Vein Zone	Olistostrome
Specific gravity		g/cc	2.65	2.59	2.61	2.58	2.59
Bond Crushing Work Index		kWh/t	4.67	8.65	8.92	4.11	5.05
SMC test	A x b	-	95.4	50.7	62.0	60.1	77.6
	SCSE	kWh/t	6.98	8.74	8.10	8.17	7.44
Bond Abrasion Index		-	0.27	0.33	0.20	0.26	0.09
Bond Rod Mill Work Index		kWh/t	10.45	11.80	11.41	12.02	12.34
Bond Ball Mill Work Index		kWh/t	11.23	12.49	11.29	11.02	11.01

The results of the testing showed that, in terms of crushability, the samples ranged from “very easy” to “easy”. The Bond Crushing Work Index values ranged from 4.11 kWh/t for the Vein Zone Composite to 8.92 kWh/t for the Sulphide Master Composite.

SMC testing produced A x b values ranging from 50.7 for the Oxide 2 sample to 95.4 for the Oxide 1 sample whilst ta values ranged from 0.52 to 1.01 for the Oxide 1 and Oxide 2 samples, respectively. The SAG mill specific energy (SCSE) values ranged from 6.98 kWh/t for the Oxide 1 sample to 8.74 kWh/t for the Oxide 2 composite and were in the lower 37% of results within the JKTech database.

Bond Abrasion testing showed the samples to have Abrasion Index values ranging from 0.09 for the Olistostrome composite to 0.33 for the Oxide 2 composite. On this basis the Olistostrome composite was classified as being “non-abrasive”, the remaining four samples where characterized as being “slightly abrasive”.

Bond Rod Mill Work Index values ranged from 10.45 kWh/t for the Oxide 1 composite to 12.34 kWh/t for the Olistostrome composite with values ranging from 11.01 kWh/t for the Olistostrome Composite to 12.49 kWh/t for the Oxide 2 composite. All Bond Rod and Bond Ball values were classified as “medium” using standard classification criteria.

The milling indices reflect a tighter range for ball mill operation than for the SAG mill, indicating that a ball mill may be operationally simpler on variable plant feed mixes.

13.6 Gravity Tests

As part of WAI Phase 1 testing, both the extended gravity gold recovery test (E-GRG) and bulk gravity concentration tests were carried out.

E-GRG tests were performed using a standard methodology developed by Laplante et Assoc (1995). This consisted of subjecting a 20 kg sample of each ore type to gravity concentration using a Knelson KC-MD3 gravity concentrator at three different grind sizes.

For the bulk gravity tests, each sample was subjected to two stages of gravity testing at target grind sizes of P_{80} 212 µm and 75 µm respectively. Once completed, the concentrates from each stage were combined and submitted for leach testing whilst a separate sub-sample of the gravity tailings was subjected to leaching. A summary of both sets of gravity results are shown in the Table 13-34.

Table 13-34: Gravity testwork results

Composite sample	Test type			
	E-GRG		Bulk Gravity	
	GRG value (%)	Grade, Au (g/t)	Grade, Au (g/t)	Recovery, Au (%)
Oxide 1	45.2	35.0	55.8	37.9
Oxide 2	47.2	46.3	87.1	49.4
Sulphide 1	62.5	40.5	66.0	61.5

The results of the E-GRG tests indicated GRG levels ranging from 45.2% for the Oxide 1 sample to 62.5% for the Sulphide 1 sample. Gold recoveries from the bulk gravity tests ranged from 37.9% (concentrate grade 55.8 g/t Au) for the Oxide 1 sample to 61.5% (concentrate grade 66.0 g/t Au) for the Sulphide 1 sample.

As part of the Phase 2 program, bulk gravity tests were undertaken to both confirm the amount of gold that could be recovered from each of the Transitional samples using a gravity process and to generate material for subsequent flotation and leach testing.

Different test methodologies were applied to each of the samples with the Transitional 1 sample subjected to testing using a rougher-cleaner flotation methodology at a grind size of P_{80} 212 µm whilst the Transitional 2 sample was subjected to two sequential stages of gravity concentration at grind sizes of P_{80} 212 µm and 75 µm respectively.

Summaries of the respective test methodologies applied to each of the samples are shown below in Figure 13-13 and Figure 13-14.

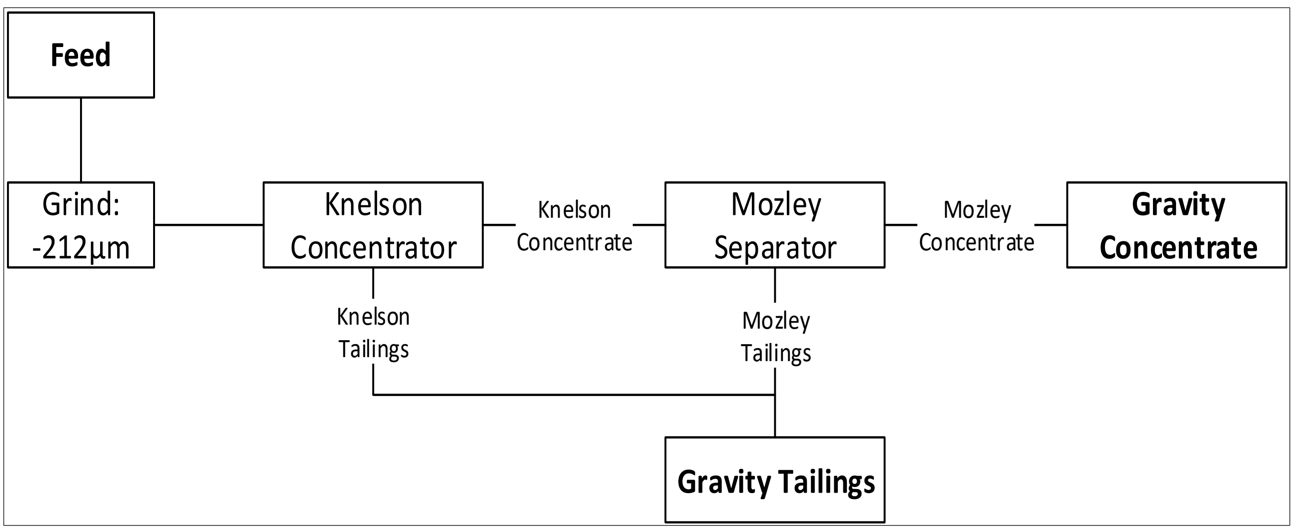


Figure 13-13: Transitional 1 gravity test methodology

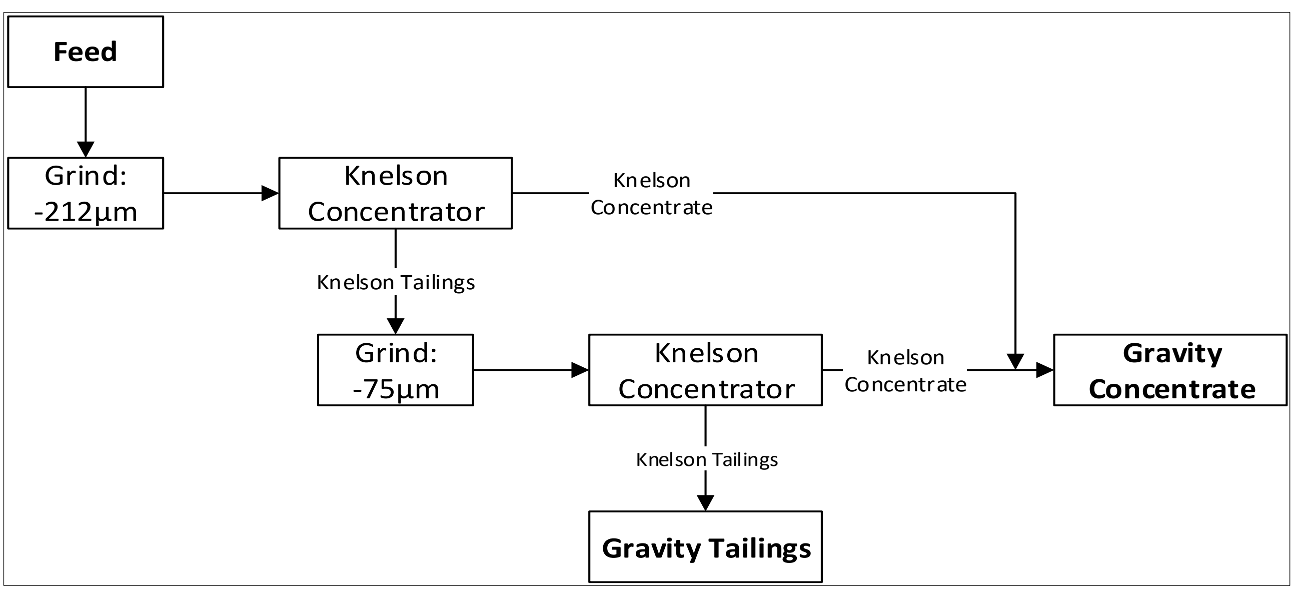


Figure 13-14: Transitional 2 gravity test methodology

A summary of the results obtained from this testwork is given in Table 13-35.

Table 13-35: Bulk gravity results

Sample	Product	Mass (%)	Assay (g/t)			Distribution (%)		
			Au	Ag	Cu	Au	Ag	Cu
Transitional 1	Concentrate	0.1	145.13	144.49	909.24	18.04	5.15	2.31
	Tailings	99.9	0.81	3.25	47.00	81.96	94.85	97.69
	Feed	100.0	0.98	3.42	48.05	100.00	100.00	100.00
Transitional 2	Concentrate	0.8	38.56	70.46	6,814.56	41.11	17.01	42.54
	Tailings	99.2	0.45	2.80	75.00	58.89	82.99	57.56
	Feed	100.0	0.76	3.35	129.46	100.00	100.00	100.00

Gravity testing of the Transitional 1 material achieved a gold recovery of 18.0% to a concentrate grading at 145.1 g/t Au. The Transitional 2 material achieved a recovery of 41.1% to a concentrate grade of 38.6 g/t Au using the regrind procedure.

These results clearly demonstrate that the cleaning of the gravity concentrate is beneficial with respect to concentrate grade, whilst the adoption of two stages of gravity separation is beneficial with respect to gold recovery compared to the Phase I testwork.

Two-stage bulk gravity tests were also carried out by ETC on a 20 kg sample representing each of the variability composite samples:

- Stage 1 – material ground to a P_{80} 200 µm
- Stage 2 – material ground to a P_{80} 75 µm.

The results of the two-stage bulk gravity tests are summarized in Table 13-36.

Table 13-36: Two-stage bulk gravity test results

Composite ID	Head grade (g/t Au)	Gravity conc. total (Calc.) g/t Au	Gravity tail grade (g/t Au)	Gravity conc. P_{80} 200 µm	Gravity conc. P_{80} 75 µm	Gravity conc. mass pull (% Wt.)	Gravity conc. % Au recovery	Gravity tail % Au recovery
REM001	0.298	12	0.19	89.5	94.4	0.92	36.2	63.8
REM002	7.633	616	1.92	94.4	91.8	0.93	75.1	24.9
REM003	0.742	55	0.27	85.5	88.2	0.87	63.8	36.2
REM004	1.081	84	0.36	86.6	85.5	0.86	66.7	33.3
REM005	0.784	55	0.30	89.2	89.6	0.89	62.6	37.4
REM006	0.845	53	0.32	94.2	92.9	0.94	59.1	40.9

Gravity recoverable gold values ranged from 36.2% for the low-grade composite (REM001) to 75.1% for the high-grade composite (REM002). The gravity concentrate ranged from 12 g/t Au for the low-grade composite to 616 g/t Au for the high-grade composite. The remaining samples, REM003 to REM006, show little variability.

The variability composites demonstrate a gravity recoverable gold component between 36 and 75%, although the Knelson concentrate grade is quite low at about 50 g/t Au, which indicates that the gold is associated predominantly with sulphides, which is known to be primarily pyrite.

13.7 Flotation Tests

13.7.1 Master Composites

13.7.1.1 Phase 1

A series of flotation tests were undertaken to investigate the recovery of gold by means of froth flotation. Testing was undertaken on samples of both fresh feed and gravity tailings at the optimum grind size P_{80} of 75 µm.

Samples were subjected to flotation using a rougher-scavenger methodology as summarized in Figure 13-15 and used in the earlier testwork programs.

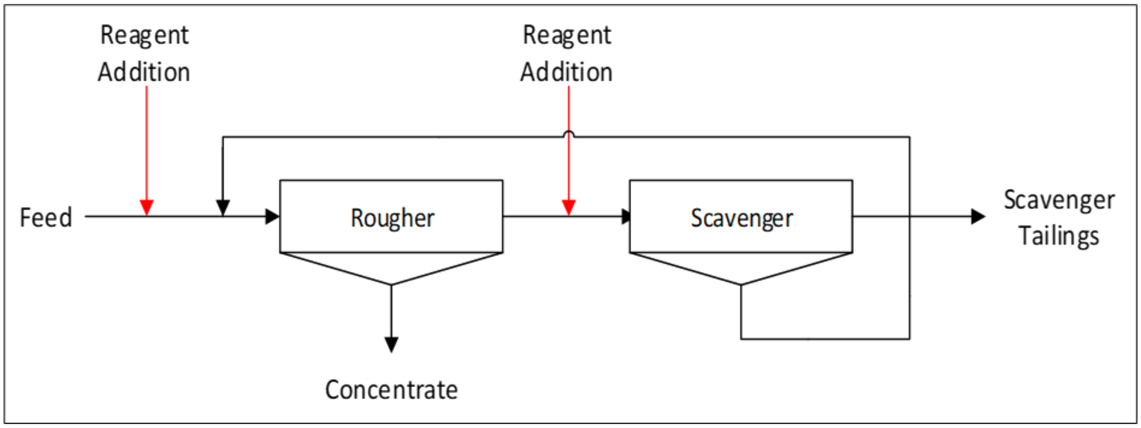


Figure 13-15: Rougher-scavenger circuit configuration

A summary of the Locked Cycle Test (LCT) results conducted on the fresh feed samples are given in Table 13-37. The LCTs were carried out over 6 cycles to reach steady state conditions.

Table 13-37: LCT results (Fresh Feed)

Sample	Test	Product	Mass (%)	Assay		Recovery (%)	
				Au (g/t)	$S_{(TOT)}$ (%)	Au	$S_{(TOT)}$
Oxide 1	LCT1	Rougher Concentrate	1.9	29.47	1.08	65.95	32.74
		Scavenger Tailings	98.1	0.29	0.04	34.05	67.26
		Feed	100.0	0.84	0.06	100.00	100.00
	LCT2	Rougher Concentrate	2.2	27.52	0.98	58.38	31.87
		Scavenger Tailings	97.8	0.45	0.05	41.62	68.13
		Feed	100.0	1.05	0.07	100.00	100.00
Oxide 2	LCT1	Rougher Concentrate	1.6	49.29	2.34	67.22	57.10
		Scavenger Tailings	98.4	0.39	0.03	32.78	42.90
		Feed	100.0	1.17	0.07	100.00	100.00
	LCT2	Rougher Concentrate	1.7	46.49	2.47	69.99	41.36
		Scavenger Tailings	98.3	0.33	0.06	30.01	58.64
		Feed	100.0	1.10	0.10	100.00	100.00
Sulphide 1	LCT1	Rougher Concentrate	4.0	22.42	20.86	94.32	94.58
		Scavenger Tailings	96.0	0.06	0.05	5.68	5.42
		Feed	100.0	0.95	0.89	100.00	100.00
	LCT2	Rougher Concentrate	3.6	25.68	23.26	91.18	93.14
		Scavenger Tailings	96.4	0.09	0.06	8.82	6.86
		Feed	100.0	1.02	0.91	100.00	100.00

The results showed a maximum gold recovery of 66.0% to a concentrate with a grade of 29.5 g/t Au for the Oxide 1 composite, 70.0% to a concentrate with a grade of 46.5 g/t Au for the Oxide 2 composite, and 94.3% to a concentrate with a grade of 22.4 g/t Au for the Sulphide 1 composite.

The results of the LCTs performed on the gravity tailings are summarized in Table 13-38.

Table 13-38: LCT results (Gravity Tailings)

Test	Product	Mass (%)	Assay		Recovery (%)	
			Au (g/t)	$S_{(TOT)}$ (%)	Au	$S_{(TOT)}$
Oxide 1	Rougher Concentrate	2.3	9.10	0.70	41.14	32.34
	Scavenger Tailings	97.7	0.29	0.03	52.86	67.66
	Feed	100.0	0.49	0.05	100.00	100.00
Oxide 2	Rougher Concentrate	2.2	11.40	1.11	50.53	45.32
	Scavenger Tailings	97.8	0.25	0.03	49.47	54.68
	Feed	100.0	0.49	0.05	100.00	100.00
Sulphide 1	Rougher Concentrate	3.5	7.73	13.65	80.32	80.33
	Scavenger Tailings	96.5	0.07	0.12	19.68	19.67
	Feed	100.0	0.33	0.59	100.00	100.00

The results showed gold recoveries of 41.1% at a concentrate grade of 9.1 g/t Au from the Oxide 1 gravity tailings, 50.5% at a grade of 11.4 g/t Au for the Oxide 2 gravity tailings, and 80.3% to a grade of 7.7 g/t Au for the Sulphide 1 gravity tailings.

A detailed chemical analysis was undertaken on the final concentrate produced during the first locked cycle test (LCT1) conducted on each of the ore types. The purpose of the analysis was to independently confirm the grade of gold within each sample and to identify the presence of any penalty or deleterious elements. The specific gravity (SG) of each concentrate was also determined.

The results obtained are summarized in Table 13-39.

Table 13-39: Detailed concentrate analysis results

Element	Units	Assay		
		Oxide 1	Oxide 2	Sulphide 1
SG	g/cm³	2.76	2.79	3.36
Au	g/t	30.3	43.4	24.9
Ag	g/t	52.8	61.0	27.0
Al	%	8.12	7.17	4.14
As	ppm	331	580	6,450
Ba	ppm	390	380	260
Be	ppm	2.8	2.8	1.7
Bi	ppm	0.7	0.9	4.1
Ca	%	0.96	0.86	1.88
Cd	ppm	0.5	1.0	1.2
Ce	ppm	129.5	133.0	59.4
Co	ppm	51	57	85
Cr	ppm	2,370	900	320
Cs	ppm	7.6	8.2	4.2
Cu	ppm	1,220	2,780	808
Fe	%	7.03	7.45	20.40

Element	Units	Assay		
		Oxide 1	Oxide 2	Sulphide 1
Mo	ppm	71.9	34.9	38.8
Na	%	0.13	0.11	0.12
Nb	ppm	18	12	11
Ni	ppm	1,120	506	303
P	ppm	1,100	1,100	600
Pb	ppm	138	239	523
Rb	ppm	171	157	95
Re	ppm	<0.02	<0.02	<0.02
S	%	1.14	2.31	>10.0
Sb	ppm	7.1	12.5	51.5
Sc	ppm	20	16	10
Se	ppm	<10	<10	10
Sn	ppm	7	7	3
Sr	ppm	57	54	50
Ta	ppm	1.3	0.8	0.8
Te	ppm	<0.5	0.5	1.4

Element	Units	Assay			Element	Units	Assay		
		Oxide 1	Oxide 2	Sulphide 1			Oxide 1	Oxide 2	Sulphide 1
Ga	ppm	22.0	19.6	13.6	Th	ppm	18	19	10
Ge	ppm	<0.5	<0.5	<0.5	Ti	%	0.57	0.39	0.36
Hf	ppm	1	1	1	Tl	ppm	2.0	2.7	1.8
Hg	ppm	1	2	<1	U	ppm	3	4	3
In	ppm	0.11	0.11	0.19	V	ppm	166	140	98
K	%	3.42	3.11	2.23	W	ppm	10	8	4
La	ppm	63	63	27	Y	ppm	46	42	21
Li	ppm	51	48	26	Zn	ppm	250	380	420
Mg	%	1.04	0.66	0.69	Zr	ppm	21	26	28
Mn	ppm	4,300	3,120	780					

The copper and arsenic quantities in the concentrate may cause the need for additional downstream processing by either removing the metals before producing a doré, or for their removal or metal stabilization in concentrate tailings. Mercury, cadmium, and antimony values were extremely low.

13.7.1.2 Phase 2

Flotation testing was also undertaken on the Transitional composites to investigate the recovery of gold by means of froth flotation, with testing undertaken on fresh feed and gravity tailings. The flotation tests were carried out at a grind size P_{80} 75 µm, adopting the rougher-scavenger processing circuit shown in Figure 13-15.

The results of the LCTs are summarized in Table 13-40.

Table 13-40: Locked cycle flotation test results

Sample	Test	Method	Product	Mass (%)	Assay		Recovery (%)	
					Au (g/t)	$S_{(TOT)}$ (%)	Au (g/t)	$S_{(TOT)}$ (%)
Transitional 1 (Fresh Feed)	LCT1	Ro/Scav	Rougher Concentrate	4.1	21.83	2.19	75.4	79.25
			Scavenger Tailings	95.9	0.30	0.02	24.6	20.75
			Feed	100	1.17	0.11	100	100
Transitional 1 (Gravity Tailings)	LCT1	Ro/Scav	Rougher Concentrate	1.9	23.30	2.81	53.2	50.0
			Scavenger Tailings	98.1	0.39	0.05	46.8	50.0
			Feed	100	0.81	0.10	100	100
	LCT2	Ro-Cl /Scav	Cleaner Concentrate	0.3	141.62	14.96	62.1	53.5
			Scavenger Tailings	99.7	0.28	0.04	37.9	46.5
			Feed	100	0.73	0.09	100	100
	LCT3	Ro/Scav	Rougher Concentrate	2.6	21.27	1.94	66.7	72.6
			Scavenger Tailings	97.4	0.28	0.02	33.3	27.8
			Feed	100	0.82	0.07	100	100
Transitional 2 (Gravity Tailings)	LCT1	Ro/Scav	Rougher Concentrate	2.7	6.27	1.48	34.5	68.7
			Scavenger Tailings	97.3	0.33	0.02	65.5	31.3
			Feed	100	0.49	0.06	100	100

Testing of the Transitional 1 fresh feed material achieved a gold recovery of 75.4% to a concentrate grade of 21.8 g/t Au whilst testing of the Transitional 1 gravity tailings material achieved a maximum gold recovery of 66.7% at a grade of 21.3 g/t Au (LCT3).

By comparison, gold recovery from the Transitional 2 gravity tailings sample was considerably lower at just 34.5% to a concentrate with a grade of 6.3 g/t Au.

13.7.2 Variability Composites

A series of scoping flotation tests were carried out on the gravity tailings stream from the bulk gravity test. The rougher-scavenger flotation tests were performed in open circuit adopting the standard reagent suite, which included PAX as a collector and MIBC as a frother (see WAI Report No. MM1385 for specific details of test conditions).

13.7.2.1 Rougher Tests

The main objective of these tests was to determine the initial metallurgical performance for the different variability samples tested. The results of the scoping rougher-scavenger flotation tests are summarized in Table 13-41. The results obtained indicate that gold recovery from the gravity tailings stream ranges from 61% for sample REM003 to 84% for samples REM002 and REM006. It should be noted that these relatively low recoveries are related to the testing of gravity tailings material.

13.7.2.2 Open Cycle Cleaner Tests

The objective of the open cycle cleaner tests was to obtain a low concentrate mass pull, and a high final concentrate gold grade. Flotation optimization tests were performed testing alternate reagents, including PAX and MIBC, in addition to A404 as a promoter (recommended for flotation of gold ores), as well as copper sulphate as an activator, and sodium hydrogen sulphide (NaHS) as a sulphidising agent for the flotation of oxidized gold ore types.

The results of the open cycle cleaner tests are summarized in Table 13-42.

Table 13-41: Rougher-scavenger scoping test results

Composite ID	Head grade (g/t Au)		Head grade (% TS)		Combined rougher-scavenger concentrate						Tailings grade	
					Mass pull		Grade		Recovery, %			
	Calc.	Assay	Calc.	Assay	g	%	Au, g/t	TS, %	Au	TS	Au, g/t	TS, %
REM001	0.186	0.192	0.330	0.29	181.90	9.10	1.36	3.13	66.31	85.78	0.07	0.05
REM002	2.110	1.920	0.470	0.43	120.37	6.02	29.29	7.31	83.52	93.60	0.37	0.03
REM003	0.298	0.271	0.140	0.18	117.83	5.89	3.10	1.89	61.18	81.96	0.12	0.03
REM004	0.356	0.230	0.363	0.23	144.69	7.23	3.60	2.78	73.18	88.92	0.10	0.03
REM005	0.318	0.296	0.270	0.25	149.83	7.49	2.78	2.25	65.63	62.32	0.12	0.11
REM006	0.400	0.410	0.550	0.52	141.62	7.08	4.77	7.06	83.86	91.49	0.12	0.11

Table 13-42: Open cycle cleaner test results

Composite ID	Head grade				Cleaner concentrate						Tailings grade	
	g/t Au		%TS		Mass Pull		Grade		Recovery, %			
	Calc.	Assay	Calc.	Assay	g	%	Au g/t	% TS	Au	TS	Au g/t	% TS
REM001	0.18	0.19	0.35	0.29	10.6	0.53	12.05	33.27	35.91	50.60	0.07	0.05
REM002	2.17		0.44		11.8	0.59	244.10	37.85	66.26	50.23	0.40	0.04
REM003	0.35		0.18		9.6	0.48	32.90	14.93	44.29	38.73	0.14	0.08
REM004	-	-	-	-	-	-	-	-	-	-	-	-
REM005	-	-	-	-	-	-	-	-	-	-	-	-
REM006	0.38		0.59		19.0	0.95	24.40	36.63	61.70	58.72	0.06	0.05

13.7.2.3 Locked Cycle Tests

Based on the results of the flotation optimization tests, which determined the optimum reagent type, dosage rate, flotation times, and the flotation circuit (with, or without cleaner flotation stage), the optimum test conditions for carrying out locked cycle flotation tests were determined.

The tests were performed according to the flowsheet shown in Figure 13-16. The locked cycle tests were carried out using 6 cycles, with the intermediate concentrate from each cycle sequentially recycled.

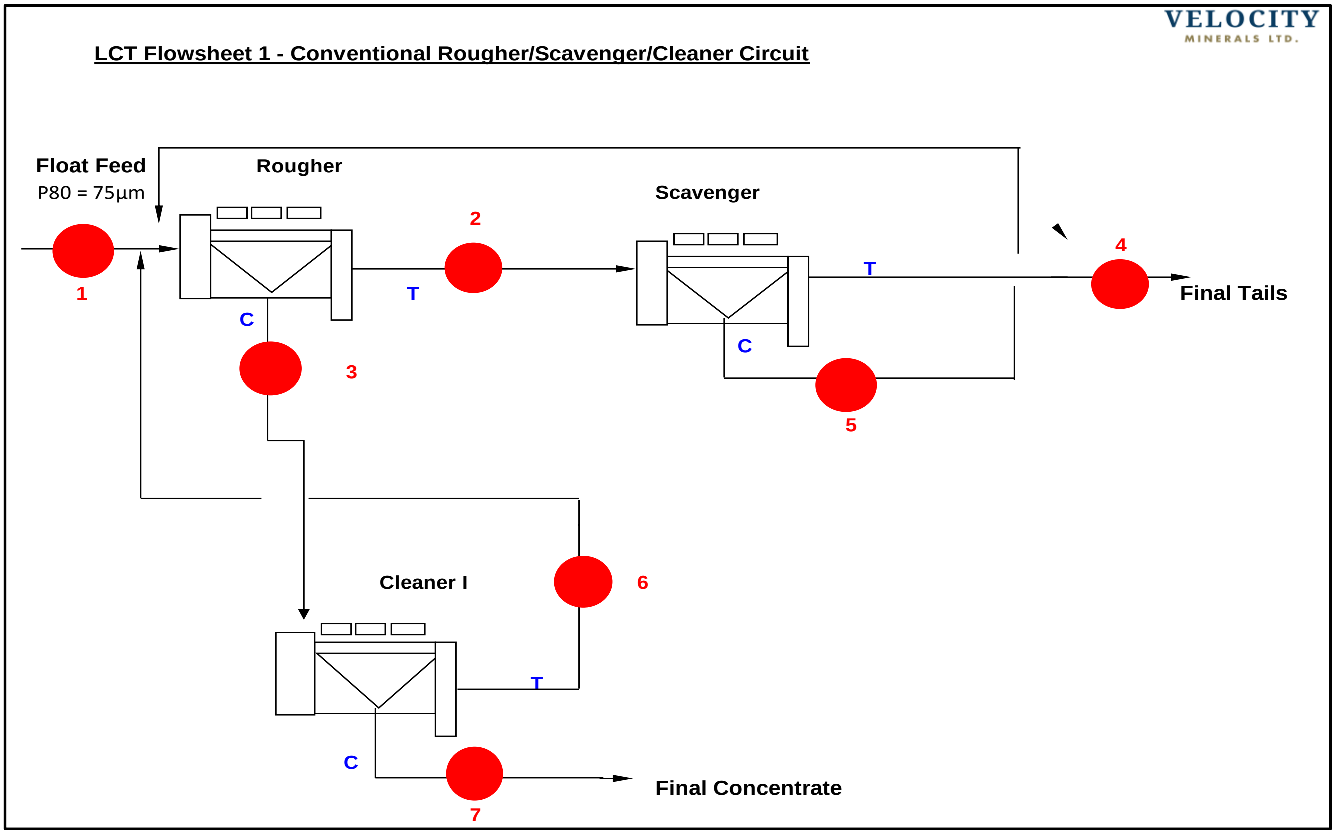


Figure 13-16: Locked cycle flowsheet (C = Concentrate, T = Tails)

The results of the LCTs carried out on the gravity tailings stream are detailed in Table 13-43.

Table 13-43: Locked cycle test results

Sample ID	Cycle	Product	Mass (%)	Assay		Recovery (%)	
				Au (g/t)	$S_{(TOT)}$ (%)	Au	$S_{(TOT)}$
REM001	5+6	Final Concentrate	3.1	4.09	9.92	62.6	88.5
		Scavenger Tailings	96.9	0.077	0.04	37.4	11.5
		Feed	100	0.20	0.34	100	100
REM002	5+6	Final Concentrate	2.9	57.20	12.70	79.6	90.0
		Scavenger Tailings	97.1	0.431	0.04	20.4	10.0
		Feed	100	2.054	0.40	100	100
REM003	5+6	Final Concentrate	2.30	10.99	5.68	65.0	82.3
		Scavenger Tailings	97.7	0.138	0.03	35.0	17.7
		Feed	100	0.385	0.16	100	100

Sample ID	Cycle	Product	Mass (%)	Assay		Recovery (%)	
				Au (g/t)	$S_{(TOT)}$ (%)	Au	$S_{(TOT)}$
REM004	5+6	Final Concentrate	4.0	6.38	4.73	69.5	87.2
		Scavenger Tailings	96.0	0.117	0.03	30.5	12.8
		Feed	100	0.369	0.22	100	100
REM005	5+6	Final Concentrate	3.1	7.65	6.30	67.5	88.0
		Scavenger Tailings	96.9	0.118	0.03	32.5	12.0
		Feed	100	0.350	0.22	100	100
REM006	5+6	Final Concentrate	1.8	18.13	31.02	85.4	94.3
		Scavenger Tailings	98.2	0.057	0.03	14.6	5.7
		Feed	100.0	0.383	0.59	100.0	100

13.7.3 Cleaner Tests

Locked cycle tests were carried out on the head sample (S-2 Composite) with and without a cleaning stage. The results of the float tests are shown in Table 13-44 and Table 13-45.

Table 13-44: Test no. F10 (with cleaner) on head sample (S-2 Composite)

Head sample (Calc.)		Flotation concentrate		Flotation tailings		Flotation concentrate mass pull (%)	Recovery (%)	
Au g/t	% TS	Au g/t	% TS	Au, g/t	S, %		Au	TS
1.18	0.617	27.30	12.3	0.044	0.098	4.18	96.4	84.6

Table 13-45: Test no. F11 (without cleaner) on head sample (S-2 Composite)

Head sample (Calc.)		Flotation concentrate		Flotation tailings		Flotation concentrate mass pull (%)	Recovery (%)	
Au, g/t	%TS	Au g/t	%TS	Au g/t	%TS		Au	TS
1.274	0.624	17.00	7.37	0.041	0.100	7.28	97.02	85.88

The results obtained indicate that high gold recoveries, in the order of 97%, may be obtained to produce a concentrate with a grade of 27.3 g/t Au with a cleaning stage, and 17.0 g/t Au without a cleaning stage. Concentrate mass pulls for the tests with (F10) and without (F11) a cleaning stage were 4.2% and 7.3% by weight respectively.

13.7.4 Gravity-Flotation Results

The results of the combined gravity-flotation tests for the variability (REM001 to REM005) and Sulphide-1 Composite samples are summarized in Table 13-46.

Table 13-46: Gravity-flotation results for the Variability (REM001-005) and Composite (Sulphide-1) samples

Sample ID	Material type	Mass pull (%)	Head grade		Gravity + Cleaner conc. (Calc.)	
			Au, g/t	S, %	Au, g/t	S, %
REM001	Low Grade material	3.99	0.298	0.35	5.85	-
REM002	High Grade material	3.76	7.633	0.66	195.34	-
REM003	Vein Zone	3.13	0.742	0.17	23.08	-
REM004	Olistostrome LOM grade	4.84	1.081	0.30	20.13	-
REM005	Material blend	3.95	0.784	0.24	18.33	-
REM006	Sulphide-1	2.72	0.845	0.83	30.50	-

Sample	Material type	Tails	Final tailings		Total Recovery	
		Mass pull (%)	Au, g/t	S, %	Au, %	S, %
REM001	Low Grade material	96.01	0.08	0.04	76.11	95.85
REM002	High Grade material	96.24	0.43	0.04	94.92	92.49
REM003	Vein Zone	96.87	0.14	0.03	87.32	88.69
REM004	Olistostrome LOM grade	95.16	0.12	0.03	89.86	91.48
REM005	Material blend	96.05	0.12	0.03	87.83	92.46
REM006	Sulphide-1	97.28	0.057	0.03	94.01	96.62

13.8 Leach Tests

13.8.1 Phase 1

As part of the Phase 1 testwork program conducted by WAI, cyanide leach testing was undertaken on samples of gravity concentrate, gravity tailings and flotation concentrate (both fresh feed and gravity tailings) to investigate and optimize gold leaching using a conventional CIL process.

13.8.1.1 Gravity Products

The gravity concentrates produced during the bulk gravity testing were subjected to two stages of cyanide leach testing to establish the amount of gold and silver that could be recovered from each of the samples. The first stage of leaching was conducted on the gravity concentrate as-is (with no prior regrinding), whilst the second stage of leaching was conducted following the regrinding of the stage 1 leach residue to a particle size of P_{80} 20 µm.

A summary of the results achieved through leaching of the gravity products is given in Table 13-47.

Table 13-47: Gravity product leach recoveries

Composite sample	Product	Au Recovery (%)		
		Stage 1	Stage 2	Total
Oxide 1	Concentrate	93.0	5.4	**98.4**
	Tailings	85.7	-	**85.7**
Oxide 2	Concentrate	90.1	8.8	**98.9**
	Tailings	88.8	-	**88.8**
Sulphide 1	Concentrate	90.4	3.6	**94.0**
	Tailings	57.2	-	**57.2**

The results showed overall gold recoveries from the gravity concentrate to range from 94.0% for the Sulphide 1 sample to 98.9% for the Oxide 2 sample. Gold recoveries from the gravity tailings ranged from 57.2% for the Sulphide 1 sample to 88.8% for the Oxide 2 sample.

When combined with the preceding gravity concentration stage, the total amount of gold that could be recovered to solution through leaching of the gravity concentrate and tailings ranged from 79.8% for the Sulphide 1 sample to 93.8% for the Oxide 2 sample, as summarized in Table 13-48.

Table 13-48: Combined gravity-leach gold recoveries

Composite sample	Product	Au stage recovery (%)		Overall recovery (% Au)
		Gravity	Leach	
Oxide 1	Concentrate	37.9	98.4	37.4
	Tailings	62.1	85.7	53.2
	Total	-	-	**90.5**
Oxide 2	Concentrate	49.4	98.9	48.9
	Tailings	50.6	88.8	44.9
	Total	-	-	**93.8**
Sulphide 1	Concentrate	61.5	94.0	57.8
	Tailings	38.5	57.2	22.0
	Total	-	-	**79.8**

13.8.1.2 Flotation Concentrate (From Bulk Flotation of Fresh Feed)

The flotation concentrates produced during bulk flotation of the fresh feed were subjected to a series of cyanide leach tests to establish the amount of gold and silver that could be recovered from each of the samples.

The Oxide 1 and Oxide 2 flotation concentrates were each subjected to two leach tests. The Sulphide 1 flotation concentrate was subjected to four cyanide leach tests.

A summary of the gold recoveries that will reflect actual performance achieved through cyanide leaching of flotation concentrate produced by the flotation of fresh feed material is shown in Table 13-49. Some individual results did give higher recoveries, but at levels of cyanide consumption that indicate costs higher than the revenue from the recovered metal.

Table 13-49: Fresh feed flotation concentrate leach recoveries

Composite sample	Recovery (% Au)
Oxide 1	98.1
Oxide 2	98.1
Sulphide 1	81.6

The data showed leach stage gold recoveries to range from 81.6% for the Sulphide sample to 98.1% for the Oxide 1 and Oxide 2 samples.

The combined gold recoveries achievable through flotation of the head sample followed by cyanide leaching of the flotation concentrate are shown in Table 13-50.

Table 13-50: Combined flotation-leach gold recoveries

Composite sample	Product	Au stage recovery (%)		Overall recovery (% Au)
		Flotation	Leach	
Oxide	Concentrate	65.9	98.1	64.6
	Tailings	34.1	-	-
	Total	-	-	**64.6**
Transitional	Concentrate	72.8	98.1	71.4
	Tailings	27.2	-	-
	Total	-	-	**71.4**
Sulphide	Concentrate	92.2	81.6	75.3
	Tailings	7.8	-	-
	Total	-	-	**75.3**

The results showed overall gold recoveries of 64.6%, 71.4% and 75.3% for the Oxide, Transitional and Sulphide samples respectively.

13.8.1.3 Flotation Concentrate (Gravity Tailings)

A single cyanide leach test was conducted using a two kg sub-sample of the bulk gravity tailings to determine the gold and silver recoveries that could be achieved from this product. Leaching was performed without any further regrinding.

The results showed leach stage gold recoveries to range from 68.3% for the Sulphide sample to 84.6% for the Transitional sample.

Overall gold recoveries based on the combined gravity-flotation-leach methodology are summarized in Table 13-51.

Table 13-51: Combined gravity-flotation-leach gold recoveries

Sample	Product	Au stage recovery (%)			Overall recovery (% Au)
		Gravity	Flotation	Leach	
Oxide	Concentrate	37.9	-	98.4	37.4
	Tailings	62.1	42.1	84.3	22.1
	Total	-	-	-	**59.4**
Transitional	Concentrate	49.4	-	98.9	48.9
	Tailings	50.6	50.5	84.6	21.6
	Total	-	-	-	**70.5**
Sulphide	Concentrate	61.5	-	94.0	57.8
	Tailings	38.5	80.3	68.3	21.1
	Total	-	-	-	**78.9**

The results showed overall gold recoveries of 59.4%, 70.5% and 78.9% for the Oxide, Transitional and Sulphide samples respectively.

13.8.1.4 Phase 2

Cyanide leach testing was conducted on both gravity and flotation concentrates generated during the flotation testing carried out on the Transitional composites in order to investigate the amount of gold that could be leached into solution.

13.8.1.5 Cyanide Leaching of Gravity Concentrate

The gravity concentrates produced during the bulk gravity testing were subjected to cyanide leaching to determine the amount of gold and silver that could be recovered from each of the samples. The cyanidation leach tests were carried out as-is (no regrinding) for a leach duration of 48 hours.

A summary of the gold recoveries achieved through leaching of the gravity concentrates is given in Table 13-52.

Table 13-52: Combined gravity-leach gold recoveries

Sample	Product	Au stage recovery (%)		Overall recovery (% Au)
		Gravity	Leach	
Transitional 1	Concentrate	18.0	87.6	15.8
	Tailings	-	-	-
	Total	-	-	-
Transitional 2	Concentrate	41.1	87.5	36.0
	Tailings	-	-	-
	Total	-	-	-

The results showed leach stage recoveries of 87.6% and 87.5% for the Transitional 1 and Transitional 2 gravity concentrates respectively. Overall gold recoveries, considering the preceding gravity stages, ranged from 15.8% (Transitional 1) to 36.0% (Transitional 2).

13.8.1.6 Cyanide Leaching of Flotation Concentrate from Fresh Feed

A summary of the gold recovery values achieved through leaching of the Transitional 1 fresh feed flotation concentrate is shown in Table 13-53.

Table 13-53: Combined flotation-leach gold recoveries

Sample	Au stage recovery (%)		Overall recovery (% Au)
	Flotation	Leach	
Transitional 1	75.4	93.2	70.2

The result showed a leach stage recovery of 93.2% for gold after 72 hours of leaching. When recalculated, considering gold recovery during the preceding flotation stage, this equated to an overall gold recovery of 70.2%.

13.8.1.7 Cyanide Leaching of Flotation Concentrate from Gravity Tailings

The flotation concentrate generated during locked cycle testing of the Transitional 1 fresh feed was subjected to cyanide leaching to establish the amount of gold and silver that could be recovered.

The sample was subjected to two stages of leaching with the flotation concentrate reground to a P_{80} of 20 µm prior to leaching.

The gold recoveries achieved through separate cyanide leaching of flotation concentrates derived from gravity tailings are summarized in Table 13-54.

Table 13-54: Combined gravity-flotation-leach gold recoveries

Sample	Product	Au stage recovery (%)			Overall recovery (% Au)
		Gravity	Flotation	Leach	
Transitional 1	Concentrate	18.0	-	87.6	15.8
	Tailings	82.0	66.7	96.3	52.7
	Total	-	-	-	**68.4**
Transitional 2	Concentrate	41.1	-	87.5	36.0
	Tailings	58.9	81.4	81.4	16.5
	Total	-	-	-	**52.5**

The results showed leach stage recoveries of 96.3% and 81.4% for the Transitional 1 and Transitional 2 gravity tailings flotation concentrates respectively which, when recalculated considering the preceding gravity and flotation stages, equated to overall gold recoveries of 68.4% for the Transitional 1 sample and 52.5% for the Transitional 2 sample.

As the two samples were not subjected to the same program of testing, a direct comparison of the results cannot be undertaken. However, based on the data that is available, the results would appear to demonstrate that higher gold recoveries can be achieved using a flotation-leach methodology as opposed to a gravity-flotation-leach methodology.

Based on the results achieved for the Transitional 1 sample at 70.2%, gold recovery was 1.8% higher using the simpler flotation-leach methodology than using the more complicated gravity-flotation-leach methodology (which was at highest, 68.4%).

Testing to determine the extent of gold extraction using high intensity/concentration cyanide leach (HIL) and CIL techniques were also carried out by ETC.

13.8.2 Gravity Concentrate Leach

High intensity cyanidation leaching (HIL) of the gravity concentrates was carried out at 2% cyanide solution strength (NaCN), and a leaching time of 48 hours. The results of the high intensity cyanidation leach tests are shown in Table 13-55.

Table 13-55: High intensity cyanidation leach test results

Sample ID	Sample description	Grade, Au g/t		Recovery % (Calc. by tail)
		Gravity conc.	Tail residue	
REM001	Low grade ore	11.71	2.87	75.50
REM002	High grade ore	615.66	50.44	91.81
REM003	Vein zone	54.49	11.39	79.10
REM004	Olistostrome LOM grade	83.80	9.93	88.15
REM005	Material blend	54.87	10.30	81.23
REM006	Sulphide S-1	53.36	2.91	94.55

The results obtained by HIL of gravity concentrates REM001 to REM005 showed that the gold extraction ranged from 76 to 92%. Tailings gold grades of 2.9 g/t Au were measured for REM001 with a feed grade of 11.7 g/t Au, and a tailings grade of 50.4 g/t Au for REM002 with a feed grade of 615.7 g/t Au.

The sample REM006 (Sulphide 1 composite) was reground to P_{80} 20 µm ahead of leaching. Results showed a 94.6% gold recovery and a gold grade in the leach residue of 2.9 g/t from a feed grade of 53.4 g/t Au. The results indicate that additional gold could be extracted by regrinding the concentrate prior to the CIL process.

13.8.3 Flotation Concentrate Leach

Cyanidation leaching of the flotation concentrates was carried out at 1 g/l NaCN, and for a leach time of 48 hours. Hydrogen peroxide was added to increase the leach kinetics. Results of the cyanidation leach tests (CIL) carried out on flotation concentrates are shown in Table 13-56.

Table 13-56: CIL test results

Sample ID	Sample description	Gold recovery (%)				
		Calc. by tail	3 hours	6 hours	24 hours	48 hours
REM001	Low grade ore	59.14	43.53	38.88	58.39	57.34
REM002	High grade ore	89.31	45.74	37.84	90.92	86.78
REM003	Vein zone	90.22	36.98	41.15	90.71	87.62
REM004	Olistostrome LOM grade	77.65	57.64	57.16	78.87	75.64
REM005	Material blend	85.81	67.20	71.95	85.88	84.72
REM006	Sulphide 1	68.75	13.82	13.64	66.00	67.29

The results indicate that significant cyanide consumption was reported in the first three and six hours of the leach cycle gold recoveries were commensurately low during this time.

As a result, the cyanide concentration in the solution was increased from one g/l to three g/l NaCN for testing REM003 and REM005 at six hours, and to six g/l NaCN for the remaining samples where the highest consumption was reported. After this the concentration was maintained at levels >1,000 ppm free NaCN.

Gold extraction ranged from of 59 to 90% for the individual samples which were ground to P_{80} 20 µm. The sulphide sample (REM006) reported a low gold recovery of 69% (based on tailings calculation) and a correspondingly high gold grade in the leach residue of 6.2 g/t Au. A minor degree of preg-robbing was also indicated by this set of tests for the samples REM001 to REM005.

13.9 Combined Gravity-Flotation Leach Recovery

The results of the overall gold recovered from the gravity-flotation process followed by cyanidation of the respective gravity and flotation concentrates, are summarized in Table 13-57.

Table 13-57: Overall combined gold recovery

Sample ID	Sample description	Gravity conc. + leach (% Au recovery)	Flotation conc. + leach (% Au recovery)	Overall recovery (% Au)
REM001	Low grade ore	27.30	23.63	50.9
REM002	High grade ore	68.93	17.72	86.6
REM003	Vein zone	50.46	21.22	71.7
REM004	Olistostrome LOM grade	58.80	17.97	76.8
REM005	Material blend	50.83	21.66	72.5
REM006	Sulphide zone	55.85	26.43	82.3

The highest combined gold recovery was achieved for the REM002 high-grade sample with 86.6% recovery. An overall gold recovery of 82.3% was achieved for REM006, the Sulphide 1 sample.

13.10 Dewatering Tests

Dewatering testing, in the form of static settling tests, were undertaken on samples of both the fresh feed flotation concentrate leach residues and the flotation tailings in order to determine the respective settling characteristics and provide an estimation of the likely thickener unit area requirements.

The settling tests were conducted according to the standard Talmage and Fitch method, in cylinders, with the density of the original pulp adjusted to 20% solids.

An initial program of flocculant screening was undertaken using the flotation tailings, after which both the flotation concentrate leach residue and the tailings were subjected to dosage optimization and subsequent thickener area testing using the optimum flocculant.

Based on flocculant screening tests which had been performed on the flotation tailings, all settling testwork was conducted using the BASF Magnafloc 155 flocculant.

A summary of the settling results for the concentrate and tailings material are shown in Table 13-58 and Table 13-59.

Table 13-58: Flotation concentrate leach residue thickener unit area determination test results

Composite ID	Flocculant dosage (g/t)
	60
Oxide 1	3.53
Oxide 2	3.63
Sulphide 1	2.80

Table 13-59: Flotation tailings thickener unit area determination test

Composite ID	Flocculant dosage (g/t)	
	10	20
Oxide 1	1.11	0.95
Oxide 2	0.99	0.81
Sulphide 1	1.00	1.09

The calculated thickener unit area requirements for the flotation concentrate leach residue material ranged from 2.80 m^2/t/h for the Sulphide 1 sample to 3.63 m^2/t/h for the Oxide 2 sample. Whilst thickener unit area requirements for the flotation tailings ranged from 0.99 m^2/t/h for the Oxide 2 sample to 1.11 m^2/t/h for the Oxide 1 sample at a flocculant dosage of 10 g/t.

At the higher flocculant dosage of 20 g/t, the thickener unit area requirements for the flotation tailings ranged from 0.81 to 1.09 m^2/t/h for the Oxide 2 and Sulphide 1 samples respectively.

13.11 Metallurgical Implications

13.11.1 Preferred Process Option

Extensive testwork carried out on the main Rozino ore types evaluated the following process options:

- Option 1: Gravity-Intensive Leach-CIL (GCIL);
- Option 2: Gravity-Intensive Leach-Flotation-CIL (GFIL); and
- Option 3: Flotation-CIL (FCIL).

Option 1 proved to be the most viable for treating the Oxide and Transitional ore zones. However, due to environmental restrictions at the Rozino site regarding the use of sodium cyanide, this is not a viable process option and it was not considered any further.

Gravity tests showed a moderate GRG gold recovery component. However, the combined recovery obtained from GFIL compared to FCIL did not justify the inclusion of a gravity-intensive leach-direct EW circuit.

The optimal flowsheet selected for the PFS base case was that of FCIL. The main differences to the flowsheet developed for the PEA are the inclusion of:

- A single stage of cleaning in the flotation circuit; and
- Concentrate regrind mill

The recommended flowsheet for treating all material types is shown in the Block Flow Diagram Figure 13-17.

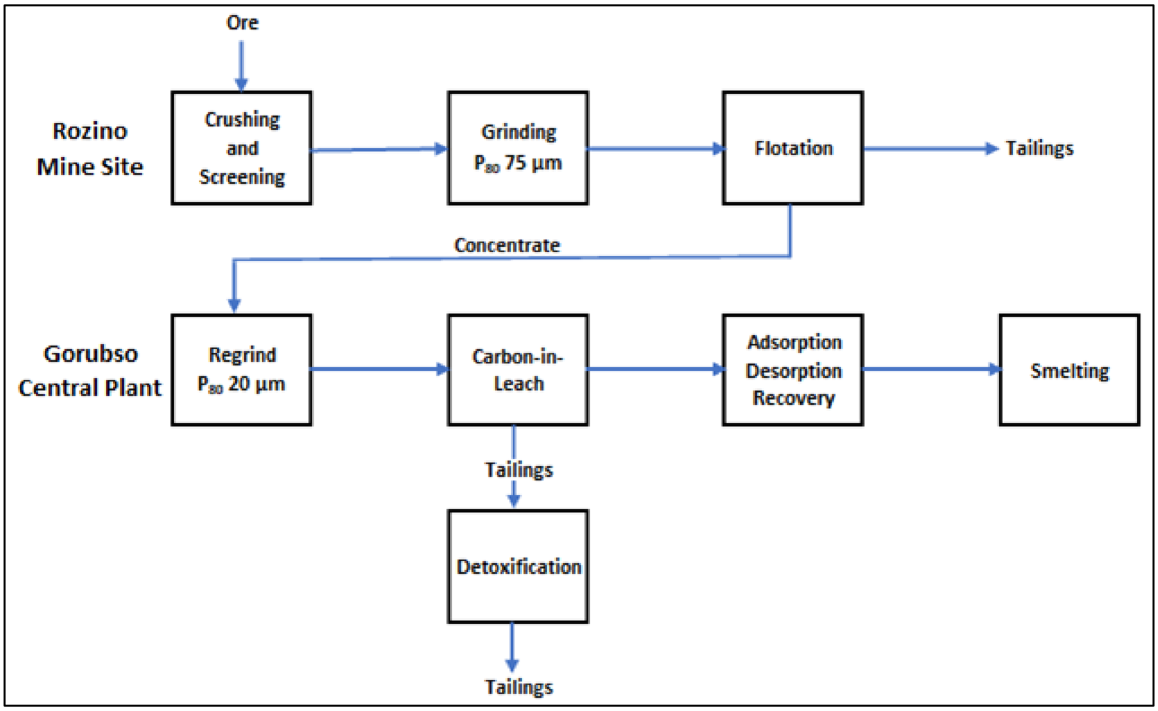


Figure 13-17: Recommended process flowsheet

13.11.2 Recovery Predictions

Overall results obtained from the testing of the different material types, evaluating the alternate process flowsheet options, are summarized in Table 13-60 to Table 13-62.

Table 13-60: GFIL vs FCIL (Oxide Composites)

Circuit configuration	Unit stage	Flotation circuit	Head grade		Gravity[1]	HIL	GIL	Float feed (%)	Flotation			CIL	Float-CIL	Tails grade		GFIL	FCIL	CIL derate	GFIL to doré	FCIL to doré
	Composite ID		Au g/t	% TS	% Au recovery				% Wt.	% Au	Au g/t	% Au recovery		Au g/t	% TS	% Au recovery				
					Gravity-HIL				Flotation-CIL											
GFIL	Ox-1	Rough-Scav	1.04	0.017	37.9	98.4	37.4	62.1	2.3	42.1	9.1	84.3	22.1	0.29	0.03	59.4		0.5	58.9	
	Ox-2		1.07	0.036	49.4	98.9	48.9	50.6	2.2	50.5	11.4	84.6	21.6	0.25	0.03	70.5		0.5	70.0	
	Average		1.06	0.027	43.7	98.7	43.1	56.3	2.2	46.3	10.3	84.5	22.0	0.27	0.03	65.1		0.5	64.6	
FCIL	Ox-1	Rough-Scav	1.04	0.017	-	-	0.0	100.0	2.2	65.9	18.8	97.5	-	0.22	0.02		64.2	0.5		63.7
	Ox-2		1.07	0.036	-	-	0.0	100.0	2.1	72.8	38.0	97.7	-	0.30	0.01		71.1	0.5		70.6
	Average		1.06	0.027	-	-	0.0	100.0	2.1	69.3	28.4	97.6	-	0.26	0.02		67.6	0.5		67.1

[1]*Gravity recovery not derated.*

Table 13-61: GFIL vs FCIL (Transitional Composites)

Circuit configuration	Unit stage	Flotation circuit	Head grade		Gravity[1,2]	HIL	GIL	Float feed (%)	Flotation			CIL	Float-CIL	Tails grade		GFIL	FCIL	CIL derate	GFIL to doré	FCIL to doré
	Composite ID		Au g/t	% TS	% Au recovery				% Wt.	% Au	Au g/t	% Au recovery		Au g/t	% TS	% Au recovery				
					Gravity-HIL				Flotation-CIL											
GFIL	T-1	Rough-Scav	1.10	0.13	26.8	92.0	24.7	73.2	2.6	66.7	21.3	96.3	47.0	0.28	0.02	71.7		0.5	71.2	
	T-2		1.12	0.09	41.1	87.5	36.0	58.9	2.7	34.5	6.3	80.8	16.4	0.33	0.02	52.4		0.5	51.9	
FCIL	T-1		1.10	0.13	-	-	0.0	100.0	4.1	75.4	21.8	93.2	70.2	0.30	0.02		70.2	0.5		69.7

[1]*Knelson-Mozley - no derating required.* [2]*Gravity recovery not derated.*

Table 13-62: GFIL vs FCIL (Sulphide Composites)

Circuit configuration	Unit stage	Flotation circuit	Head grade		Gravity[1]	HIL	GIL	Float feed (%)	Float			CIL	Float-CIL	Tails grade		GFIL	FCIL	CIL derate	GFIL doré	FCIL doré
	Composite		Au g/t	% TS	% Au recovery				% Wt.	% Au	Au g/t	% Au recovery		Au g/t	% TS	% Au recovery				
WAI Testwork																				
					Gravity-HIL			**Flotation-CIL**												
GFIL	Sulphide 1	Rough-Scav	0.86	0.86	61.5	94.0	57.8	38.5	3.5	80.3	7.7	68.3	21.1	0.07	0.12	78.9		0.5	78.4	
FCIL	Sulphide 1	Rough-Scav	0.86	0.86	-	-	-	100.0	3.6	91.2	25.7	74.4	67.8	0.09	0.06		67.8	0.5		67.3
ETC Testwork																				
GFIL	Sulphide 1	Rgh-Scav-Clnr	0.86	0.86	59.1	94.6	55.9	40.9	1.8	85.4	18.1	68.8	24.0	0.06	0.03	79.9		0.5	79.4	
	Sulphide 1	Rgh-Scav-Clnr			59.1	94.6	55.9	40.9	1.8	85.4	18.1	75.4	26.3	0.06	0.03	82.2		0.5	81.7	
	Sulphide 2	Rgh-Scav	1.24	0.56	56.6	95.8	54.2	43.4	6.8	93.5	7.2	72.9	29.6	0.04	0.12	83.8		0.5	83.3	
	Sulphide 2	Rgh-Scav-Clnr	1.24	0.56	65.1	95.8	62.3	34.9	2.0	93.8	23.1	69.6	22.8	0.03	0.00	85.1		0.5	84.6	
FCIL	Sulphide 2	Rgh-Scav	1.27	0.624	-	-	-	100.0	7.3	97.0	17.0	86.1	83.6	0.04	0.10		83.6	0.5		83.1
	Sulphide 2	Rgh-Scav-Clnr	1.18	0.617	-	-	-	100.0	4.2	96.4	27.3	86.1	83.1	0.04	0.10		83.1	0.5		82.6

A summary of the final recovery performance achieved adopting the GFIL and FCIL flowsheet for treating the different material types is presented in Table 13-63.

Table 13-63: Overall summary results, GFIL vs FCIL

Zone	Composite ID	Flotation circuit configuration	GFIL			FCIL			Δ % Au recovery
			Recovery	CIL[1] derate	Doré	Recovery	CIL[1] derate	Doré	
			% Au						
Oxide	Avg. Ox	Rough-Scav	65.1	0.5	64.6	67.6	0.5[1]	67.1	2.5
Transitional	Transitional 1	Rough-Scav	71.7	0.5	71.2	70.2	0.5	69.7	-1.4
Sulphide	Sulphide 2	Rough-Scav	83.8	0.5	83.3	83.6	0.5	83.1	-0.2
		Rough-Scav-Clnr	85.1	0.5	84.6	83.1	0.5	82.6	-2.1

[1]A 0.5% CIL deration factor is assumed to consider solution gold losses in the CIL tail.

Table 13-63 indicates that the recovery differential between GFIL and FCIL process options does not warrant the installation of a gravity circuit at the Rozino concentrator.

Results of testing at ETC, with and without a cleaner flotation stage, demonstrated the benefit of including a cleaner stage in the flotation circuit. Inclusion of a cleaner circuit resulted in a lower concentrate mass pull, and therefore a higher gold concentrate grade

Comparative results of GFIL and FCIL tests with a cleaner stage included for the Sulphide ore are shown in Table 13-64 and Table 13-65.

Table 13-64: GFIL test results

Zone	Composite ID	Flotation circuit configuration	Resources (% distn.)	% Au recovery (doré)
Oxide	Avg Ox	Rough-Scav	13.7	64.6
Transitional	Transition 1	Rough-Scav	14.7	71.2
Sulphide	Sulphide 2	Rough-Scav-Cleaner	71.6	84.6
Total			**100.0**	**79.9**

Table 13-65: FCIL test results

Zone	Composite ID	Flotation circuit configuration	Resources (% distn.)	% Au recovery (doré)
Oxide	Avg Ox	Rough-Scav	13.7	67.1
Transitional	Transitional 1	Rough-Scav	14.7	69.7
Sulphide	Sulphide 2	Rough-Scav-Cleaner	71.6	82.6
Total			**100.0**	**78.6**

Based directly on the testwork results for the FCIL process, the overall combined recovery to doré is predicted to be 78.6%. However, it was noted during the testwork process that there is a positive grade-recovery relationship, and a summation of the Mineral Reserve estimate indicates a grade 5% higher than the grade of the composites used in the FCIL testing. Therefore, when applying the testwork results to the Reserve, a mild increase to recovery may be anticipated. This is supported by testwork in this program.

The author is reliant on the financial modelling expert Andrew Sharp in development of the grade-recovery adjustment and has reviewed his work for inclusion to this report. Based upon that work, the overall recovery for the FCIL process is predicted to be 79.3%.

The PEA utilized a composite feed grade of 1.24 g/t Au, which is virtually identical to the current expected Reserve feed grade of 1.22 g/t Au. There is little overall variation in the results of the PFS and PEA testwork results based on an overall blend of material types. However, there is a significant variation in recovery based on material type (Oxide, Transitional and Sulphide) where recovery is lower as oxidation increases. There is a second impact on recovery where recovery reduces with lower head grades. A recovery model was developed that relates both oxidation effects and head grade and is discussed in Section 13.11.3.

These combined effects can alter the expected quarterly recovery as compared to the LOM average by ranges of -15% to +5%.

13.11.3 Gold Recovery Model

Grade-recovery models for each material type were developed by Andrew Sharp and were documented in a CSA Global memorandum "Rozino Project Prefeasibility FCIL Recovery Formula" dated 29 June 2020.

Sulphur content is demonstrated as being the most accurate proxy for gold recovery. However, sulphur content was not assayed consistently for the drill-hole core and is not estimated in the geological block model. Therefore, sulphur is not an acceptable or robust proxy for gold recovery predictions.

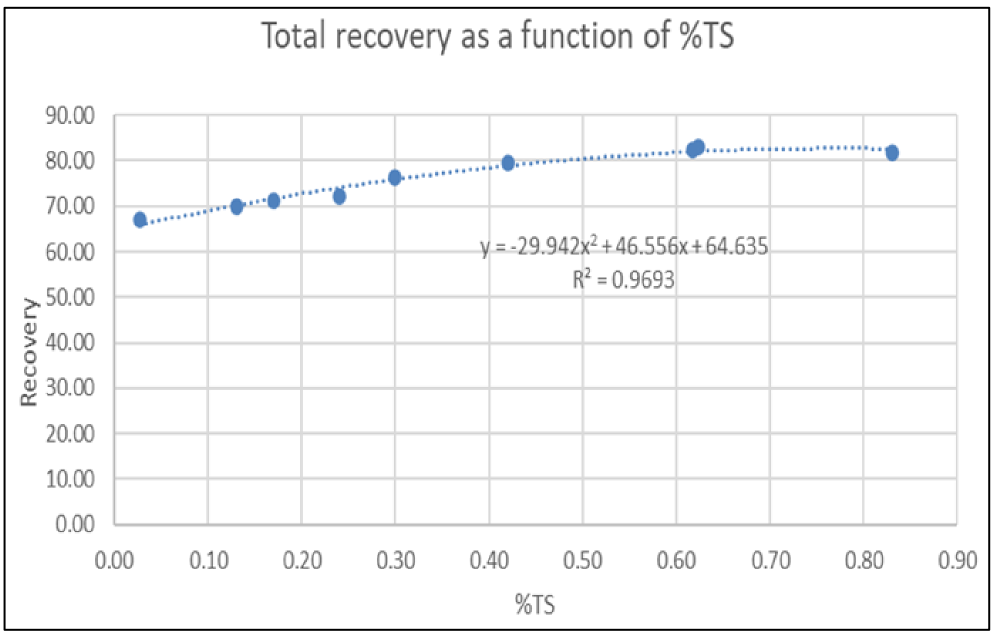


Figure 13-18: Total sulphur vs gold recovery

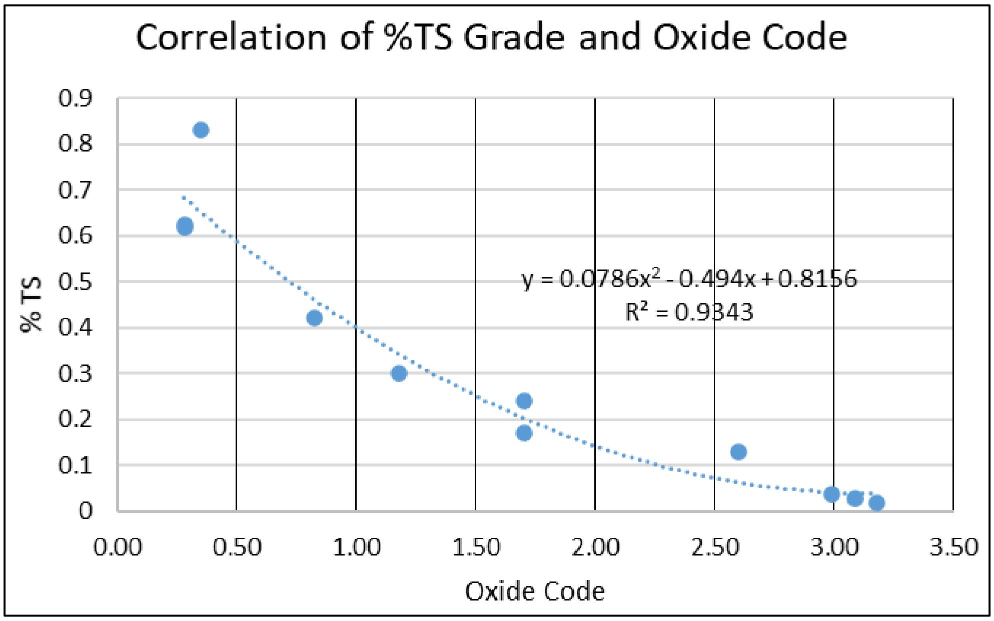


Figure 13-19: ISRM Oxidation code vs % total sulphur

The ISRM weathering code (referred to here as the oxidation code) has been measured for all drill holes and is well correlated to total sulphur content as well as recovery (albeit marginally less well than total sulphur content).

The recovery model was generated from two relationships:

- Oxidation state
- Head grade.

The final formulas are synthesized from the raw relationships to functions that can be applied to the three oxidation zones as defined in the Rozino geological model supplied by Tintyava Exploration AD. The recovery model includes a 0.5% CIL loss.

The generalized recovery formula developed for oxidation state (Ox is the oxidation code ranging from 0 to 5 – fresh to completely oxidized) and grade is:

$$Recovery = (95.08-6.225*Ox) * ((Au-0.12)/Au)$$

Where Ox is < 0.5, Ox is made equal to 0.5.

However, since the geological model has static oxidation codes for Sulphide, Transitional and Oxide zones, the general formula can be simplified for each ore type:

- Sulphide:

  $Recovery = 91.97 * ((Au-0.12)/Au)$ where Au below 0.12 $Recovery = 0.0$

  - Test* - using S-2 Rgh-Scav-Clnr with Au = 1.18, the total recovery is 82.61, equal to the test result.
- Transitional:

  $Recovery = 78.90 * ((Au-0.12)/Au)$ where Au below 0.12 $Recovery = 0.0$

  - Test* - using T-1 Rgh-Scav with Au = 1.10 Ox = 2.6, the total recovery is 70.29, 0.5% higher than the test result.

- Oxide:

$$Recovery = 75.91 * ((Au-0.12)/Au) \qquad where\ Au\ below\ 0.12\ Recovery = 0.0$$

  - Test* - using Ox1+Ox2 Rgh-Scav with Au = 1.01, the total recovery is 66.89 which is 0.3% lower than the actual test results obtained.

The formula may be further modified to give exact results to the tests for Oxide and Transitional ores but this would move away from a general formula that best fits all information utilized. The accuracy of the general formula for each individual test is +/- 3%.

The total orebody recovery will depend on the cut-off applied because it changes the head grade) and the volume of each ore type mined. The grade-recovery formulae can also be applied to mine production to give recoveries variable to the mix of ore type and head grade at any time in the mine life.

Based on the results of the variability testing conducted in Phase 1, ore type mixing does not appear to detrimentally affect the recovery results. From this observation it is assumed that reagent consumption will be proportional to the tonnage of the ore types in the feed and that the recovery formula apply equally to mixed feeds (using each formula on the tonnage of each ore type fed in the mix). This assumption requires further testing in the Feasibility Study stage.

14 Mineral Resource Estimates

14.1 Introduction

The author estimated recoverable resources for Rozino by MIK with block support correction, a method that has been demonstrated to provide reliable estimates of resources recoverable by open pit mining for a wide range of mineralization styles.

The estimates are based on diamond drilling data supplied by Velocity in October 2019. Triangulated surfaces representing the base of oxidation and the top of fresh rock interpreted by Velocity from drill hole logging and supplied to MPR in August 2020 were used for density assignment

The estimates are reported within a 150 metre deep optimal pit shell generated at a gold price of $ 1,500/oz and are constrained below a topographic wire-frame produced from DGPS surveys and supplied by Velocity in October 2019

The Mineral Resource estimates have been classified and reported in accordance with NI 43-101 and the classifications adopted by CIM Council in May 2014 (CIM, 2014). Estimates for mineralization tested by generally consistently 50 by 50 metre and closer spaced drilling are classified as Indicated, with estimates for more broadly sampled zones assigned to the Inferred category.

The Qualified Person responsible for the Mineral Resources is Jonathon Abbott who is a full time employee of MPR Geological Consultants Pty Ltd and a member of the Australian Institute of Geoscientists.

14.2 Resource Dataset

The current estimates are based on two metre down-hole composited gold assay grades from diamond holes drilled by Hereward, Asia Gold and Velocity included in a drilling database supplied by Velocity on the 23rd of October 2019, which total 204 holes for 26,321 metres in the Rozino area.

Un-assayed intervals from Velocity's drilling were assessed on a case by case basis with reference to nearby holes and geological logging. They were classified as either barren and assigned a gold grade of zero g/t, or potentially mineralized and excluded from the estimation dataset. Composites with assigned grades of zero represent around 0.6% of mineralized domain composites and uncertainty over the reliability of this approach does not significantly impact general confidence in estimated resources.

In November 2019, Velocity submitted 140 samples for intervals from 8 pre-August 2019 holes for which assay results were not available in the estimation dataset for assay. These additional assays include two metre composites with gold grades of up to 2.0 g/t which were assigned a grade of zero in the composite estimation dataset.

The author recommends that, all un-assayed mineralized domain drill hole intervals, for which suitable sample material is available are assayed consistently with the approach adopted for assaying of Velocity's drilling to date. The author recommends that resource estimates are updated to include such assays and the November 2019 assays described above as appropriate.

The composite dataset includes 11,254 composites with gold grades ranging from 0.00 to 154.1 g/t and averaging 0.41 g/t. Velocity's diamond drilling provides 82% of the mineralized domain composites informing the current estimates with Asia Gold and Hereward drilling contributing 3% and 16%, respectively.

14.3 Mineralization Interpretation and Domaining

Resource modelling includes a mineralized domain interpreted from two metre downhole composited gold grades and geological logging from diamond drilling. The mineralized domain captures intervals of greater than 0.1 g/t with the lower boundary generally reflecting the contact between variably mineralized sedimentary rocks and generally un-mineralized basement.

The lower mineralized domain contact was digitized on cross-sections aligned with Velocity's drilling traverses with snapping to drill hole traces where appropriate, and wireframed into a three-dimensional surface which was closed to create a solid wireframe representing the mineralized domain.

Triangulated surfaces representing the base of oxidation and the top of fresh rock interpreted by Velocity from drill hole logging and supplied to MPR in August 2020 were used for density assignment. Within the resource area the depth to the base of complete oxidation averages around 11 metres with fresh rock occurring at an average depth of around 22 metres.

The mineralized domain covers an area around 800 by 1,000 metres and extends to a maximum depth of around 195 metres, with the basal contact averaging around 85 metres below surface.

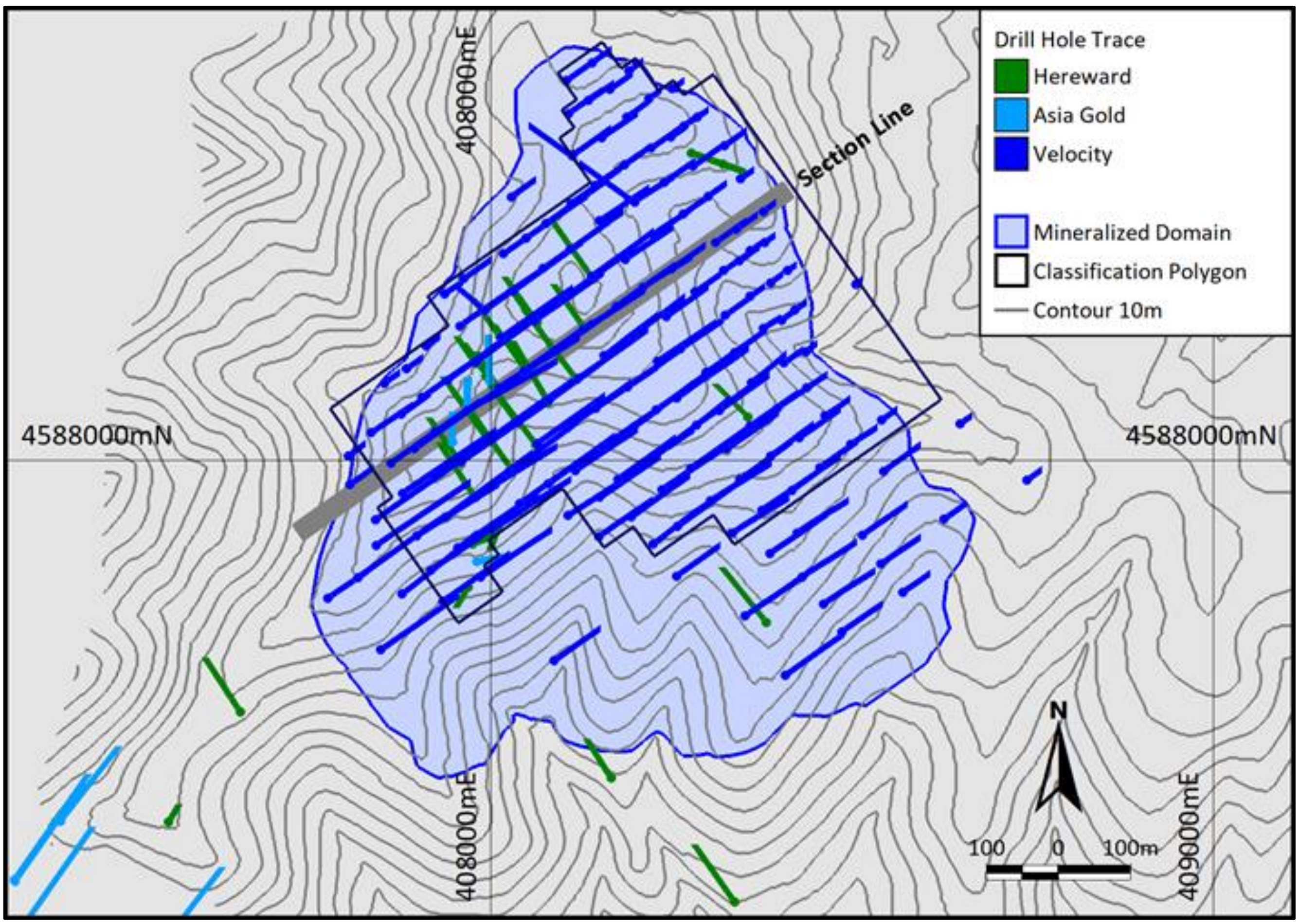


Figure 14-1 shows the surface expression of the mineralized domain and drillhole traces coloured by drilling phase Figure 14-2 presents an example cross section of the modelling domains relative to drillhole traces coloured by composited gold grade. These figures include only the estimation dataset drilling.

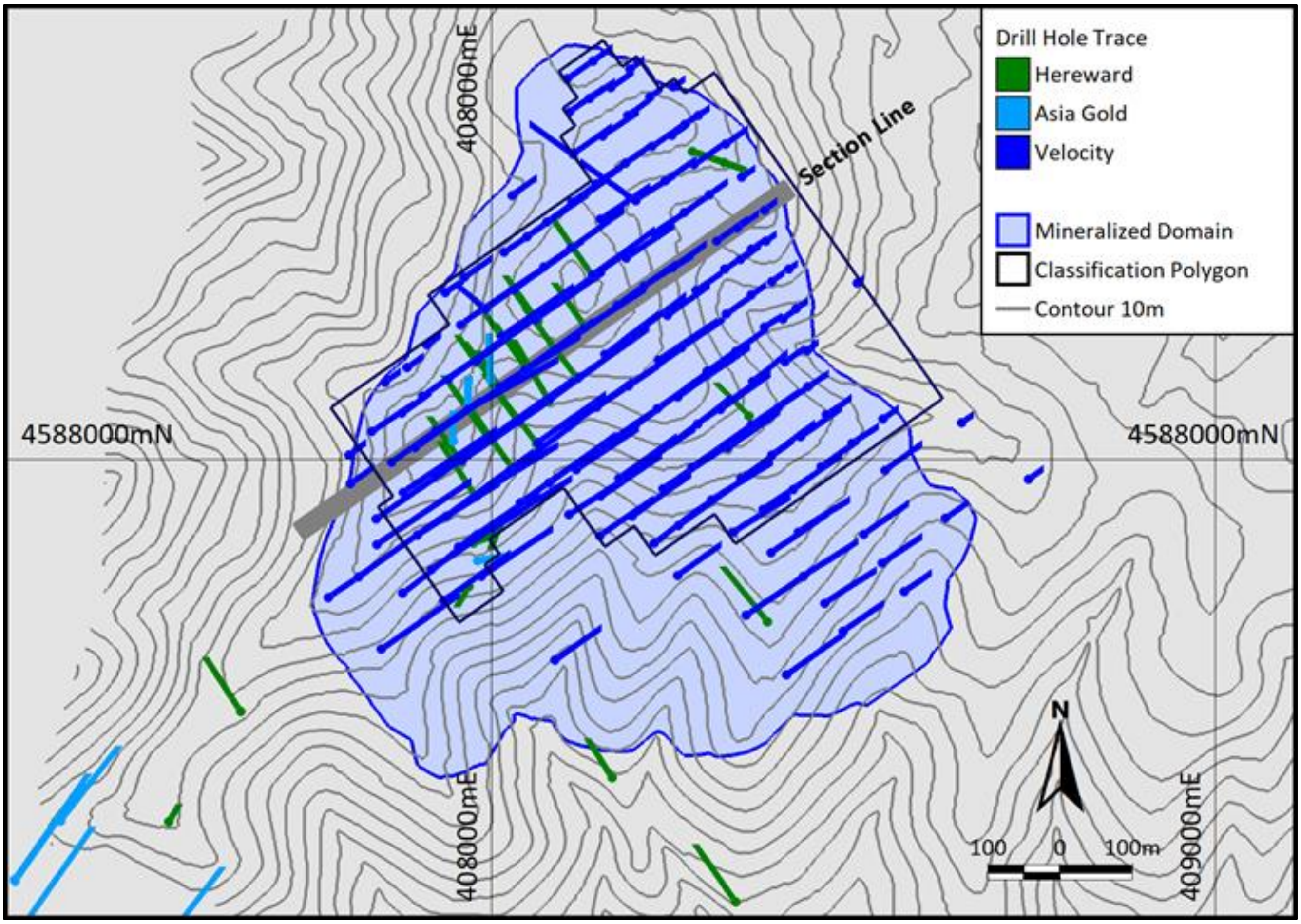


Figure 14-1: *Drillhole traces and surface expression of mineralized domain*

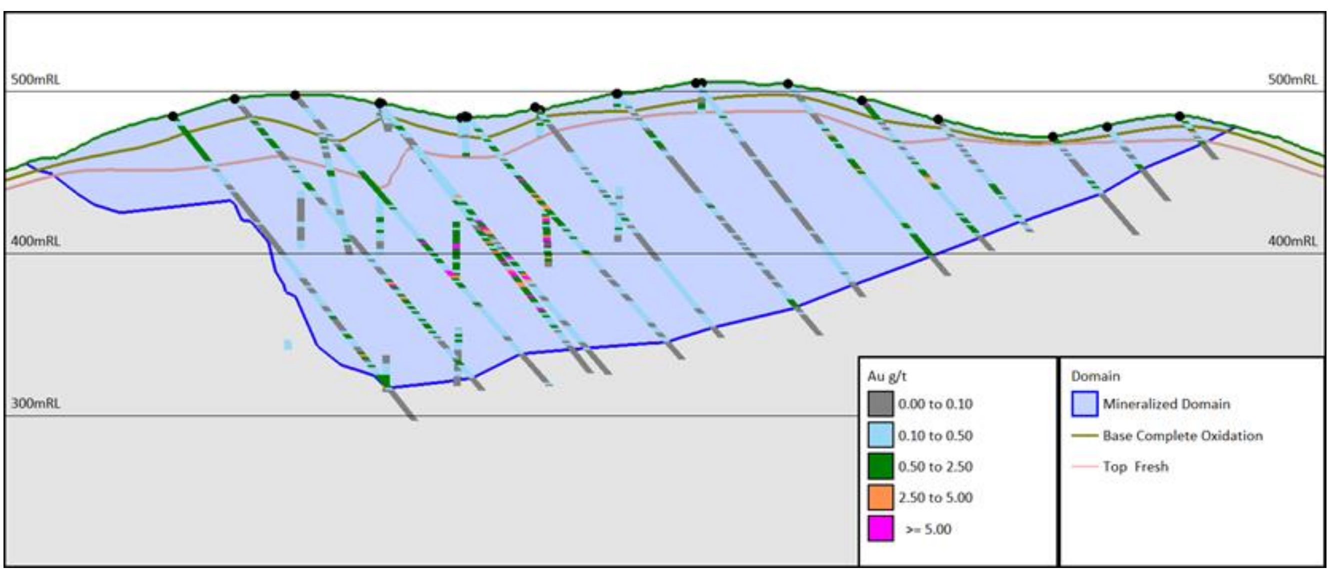


Figure 14-2: *Cross section of mineralized domain and drill traces, view to northwest (section line shown in Figure 14-1)*

14.4 Estimation Parameters

14.4.1 Indicator Thresholds and Bin Mean Grades

Table 14-1 presents the indicator thresholds and bin average grades for the mineralized domain estimation dataset.

Indicator class grades used for the MIK modelling were generally determined from the mean gold grade of each indicator class. The impact of a small number of outlier composites was reduced by cutting six outlier composites with gold grades of greater than 60 g/t to 60 g/t for determination of the mean grade for the highest indicator class giving a value of 11.485 g/t. In the author's experience this approach is appropriate for MIK modelling of highly variable mineralization such as Rozino.

Table 14-1: Mineralized domain indicator thresholds and class grades

Percentile	Threshold (Au g/t)	Mean (Au g/t)
10%	0.027	0.017
20%	0.050	0.038
30%	0.070	0.057
40%	0.102	0.086
50%	0.144	0.123
60%	0.200	0.170
70%	0.283	0.239
75%	0.344	0.313
80%	0.423	0.380
85%	0.542	0.477
90%	0.755	0.634
95%	1.210	0.944
97%	1.706	1.427
99%	3.565	2.351
100%	154.100	12.875

14.4.2 Variogram Models

Indicator variograms were modelled for each indicator threshold from the mineralized domain composites (Table 14-2). For determination of variance adjustment factors a variogram was modelled for composite gold grades. The spatial continuity observed in the variograms is consistent with geological interpretation and trends shown by composite gold grades.

Table 14-2: Variogram models

Rotation relative to model axes (y-15)							
Percentile	Nugget	First Structure (Exponential)		Second Structure (Spherical)		Third Structure (Spherical)	
		Sill	Range (x,y,z)	Sill	Range (x,y,z)	Sill	Range (x,y,z)
10%	0.17	0.40	64,40,8	0.14	120,40,130	0.29	235,120,155
20%	0.16	0.36	90,35,8	0.14	120,40,215	0.34	270,150,280
30%	0.16	0.36	82,17,8	0.14	92,40,160	0.34	230,140,275
40%	0.17	0.38	64,18,8	0.14	92,41,160	0.31	270,180,275
50%	0.18	0.39	44,18,8	0.14	84,45,150	0.29	240,160,185
60%	0.19	0.39	44,27,7.5	0.14	84,45,89	0.28	185,160,150
70%	0.20	0.39	37,46,7	0.14	56,52,89	0.27	170,130,105

75%	0.21	0.39	36,45,6.5	0.14	52,52,59	0.26	160,115,95
80%	0.22	0.40	34,41,6.5	0.14	41,47,59	0.24	145,100,80
85%	0.24	0.41	37,24,6	0.14	40,40,49	0.21	140,78,70
90%	0.26	0.46	29,17,6	0.14	38,36,40	0.14	140,68,65
95%	0.30	0.49	25,15,5.5	0.13	34,30,30	0.08	95,60,55
97%	0.34	0.50	25,15,4.5	0.13	30,30,25	0.03	90,55,42
99%	0.40	0.51	21,14,4.5	0.07	29,29,12	0.02	60,40,30
Au g/t	0.30	0.60	28,19,8	0.06	60,23,80	0.04	200,89,140

14.4.3 Block Model Dimensions

The block model framework used for MIK modelling covers the full extents of the composite dataset and mineralized domain. The model is aligned with the 055 trending Velocity drilling traverses and includes panels with dimensions of 25 metres by 25 metres by 5 metres vertical. The plan-view panel dimensions reflect drill hole spacing in more closely drilled portions of the deposit.

14.4.4 Search Criteria

The five progressively more relaxed search criteria used for MIK estimation are presented in Table 14-3. The search ellipsoids were aligned with dominant domain orientation.

Table 14-3: Search criteria

Search	Radii (m) (x,y,z)	Minimum data	Minimum octants	Maximum data
1	60,60,8	16	4	48
2	75,75,10	16	4	48
3	75,75,10	8	2	48
4	100,100,15	8	2	48
5	150,150,22.5	8	2	48

14.4.5 Variance Adjustment

Estimated resources include a variance adjustment to give estimates of recoverable resources for selective mining unit (SMU) dimensions of 4 metres east by 6 metres north by 2.5 metres in elevation with high quality Grade Control sampling on a 5 by 8 by 1.25 metre pattern. These selectivity parameters were specified by Velocity and reflect their view of likely potential mining scenarios.

The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14-4 were estimates from the gold grade variogram model in Table 14-2 and the specified mining selectivity parameters.

Table 14-4: Variance adjustment factors

Block/panel	Information effect	Total adjustment
0.288	0.746	0.215

14.5 Bulk Density Assignment

Densities of 2.35, 2.40 and 2.55 t/bcm for oxide, transitional and fresh material respectively. These values are based on the average of the available density measurements (Table 11-13).

14.6 Classification of the Estimates

Available information does not define Rozino mineralization with sufficient confidence for estimation of Measured Resources. The current estimates are classified as Indicated and Inferred, primarily on the basis of estimation search pass and a plan view polygon defining the limits of 50 by 50 metre and closer drilling as shown in Figure 14-1.

Panels informed by search pass 1 and 2 within the classification polygon were initially classified as Indicated, with all other estimates classified as Inferred. Comparatively few panels initially classified as Inferred within the volume of Indicated panels were re-classified as Indicated. These re-classified panels are generally near-surface and not informed by search pass 1 and 2 due to the octant requirements for these search passes.

The classification approach gives a consistent distribution of categories and classifies estimates for mineralization tested by up to approximately 50 metre spaced drilling as Indicated, with estimates for broader and irregularly sampled mineralization classified as Inferred.

14.7 Model Reviews

Model reviews included comparison of estimated block grades with informing composites. These checks comprised inspection of sectional plots of the model and drill data and review of swath plots and showed no significant issues.

Figure 14-3 shows a representative plan and cross-section of the Rozino resource model. These plots show model panels scaled by the estimated proportion above 0.5 g/t cut off and coloured by the estimated gold grade above this cut off relative to the resource domains and drill holes traces coloured by two metre composited gold grades. Resource classification is shown by the hatch style for each panel. Figure 14-4 shows the resource model blocks scaled at 0.5 g/t cut off relative to a three dimensional representation of Velocity's olistostrome interpretation.

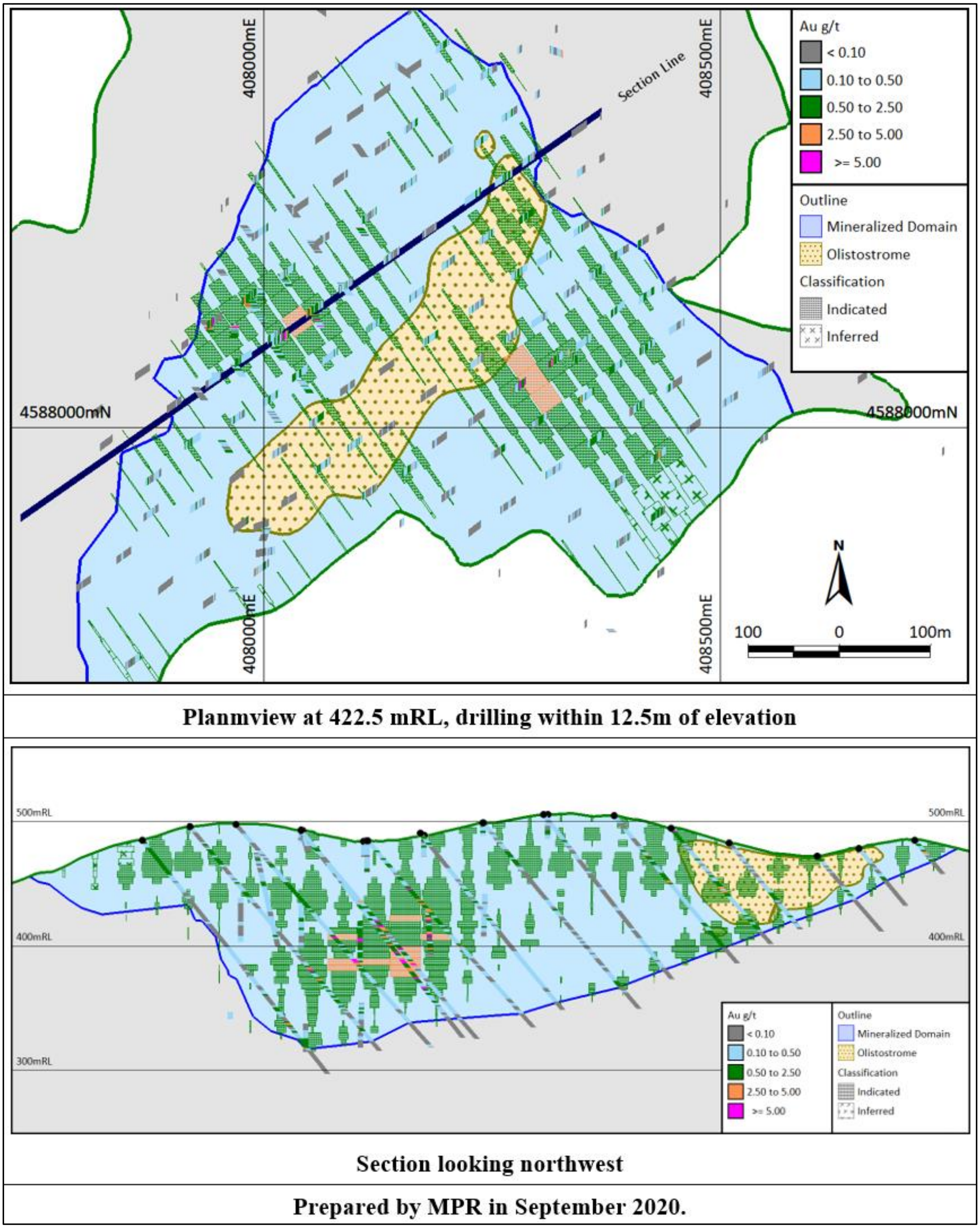


Figure 14-3: Example of block model estimates at 0.5 g/t cut-off

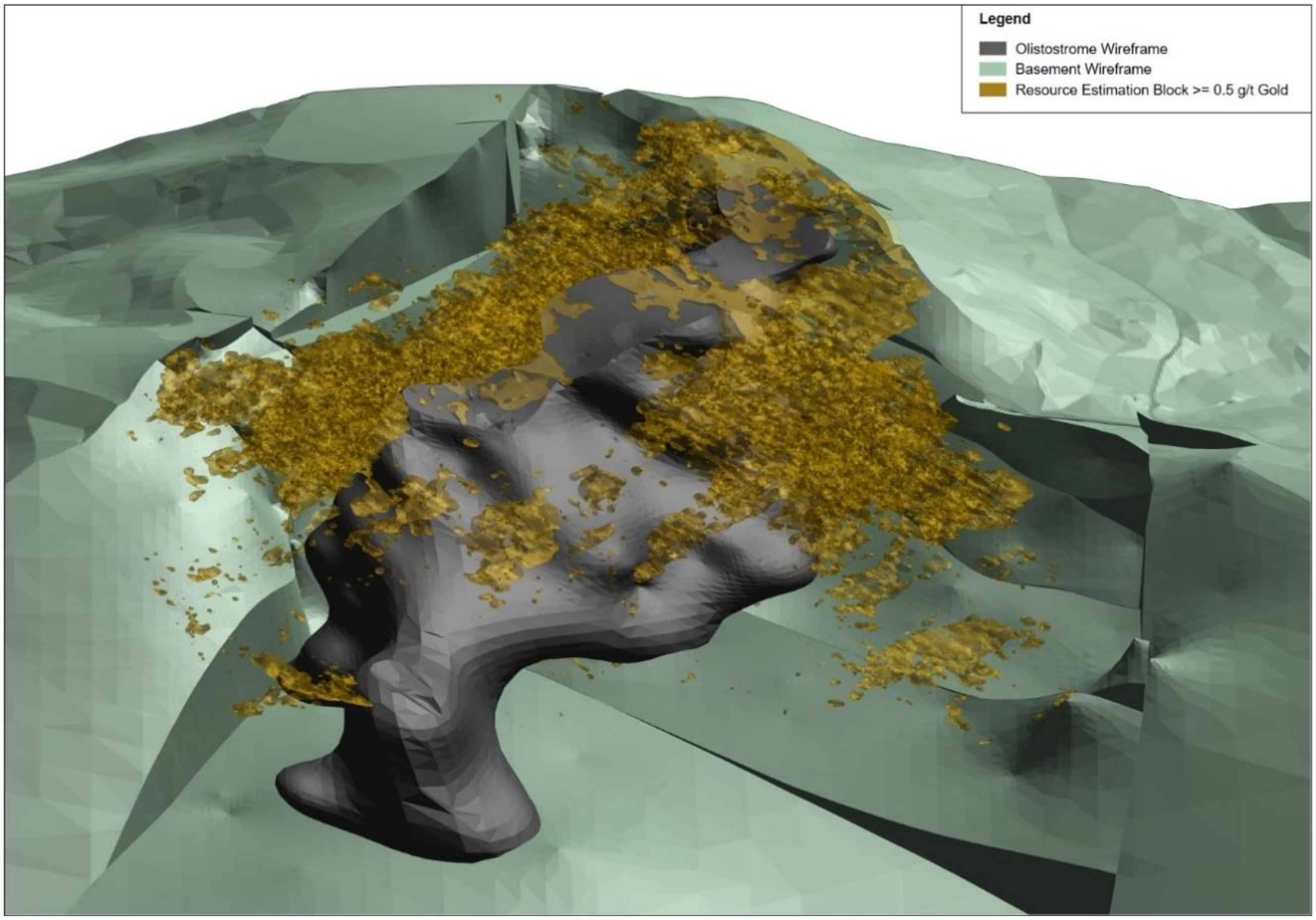


Figure 14-4: Rozino sedimentary basin, olistostrome and model blocks
Source: Velocity, 2020

It should be noted that when viewing the vertical section and plan through the resource model there are situations where the model blocks appear to be un-correlated to mineralized domain wire-frame and mineralized intercepts in the neighbouring drill holes. This is occurring because of the way the resource model has been presented. The model blocks plotted are only those that contain an estimated resource above 0.5 g/t gold cut-off and the proportion above cut off has been used to scale the dimensions of the model block for presentation purposes. The scaling occurs about the model block centroid co-ordinate and therefore introduces the apparent miss-match between data and the resource model blocks.

The swath plots in Figure 14-5 compare average estimated mineralized domain panel grades and average mineralized domain composite grades by model axes. The average composite grades include an upper cut of 25 g/t which represents the 99.9th percentile of mineralized domain composites and reduces the impact of a small number of outlier composite gold grades of up to 154.1 g/t.

The plots in Figure 14-5 show that although, as expected, average block grades are smoothed compared to average composite grades they generally closely follow the trends shown by composite mean grades with the exception of areas of variably spaced or limited sampling. Minor local deviations between the model and composite trends seen on the plots are influenced due to the following features:

- The data used in the estimation of the MIK panel grades are coming from a greater volume than the vertical slices being compared which are consistent with model panel dimensions.
- Areas of variable spacing, with drilling preferentially clustered in higher, or lower grade mineralization causes apparent inconsistencies between average composite and model grades as presented in the swath plots.

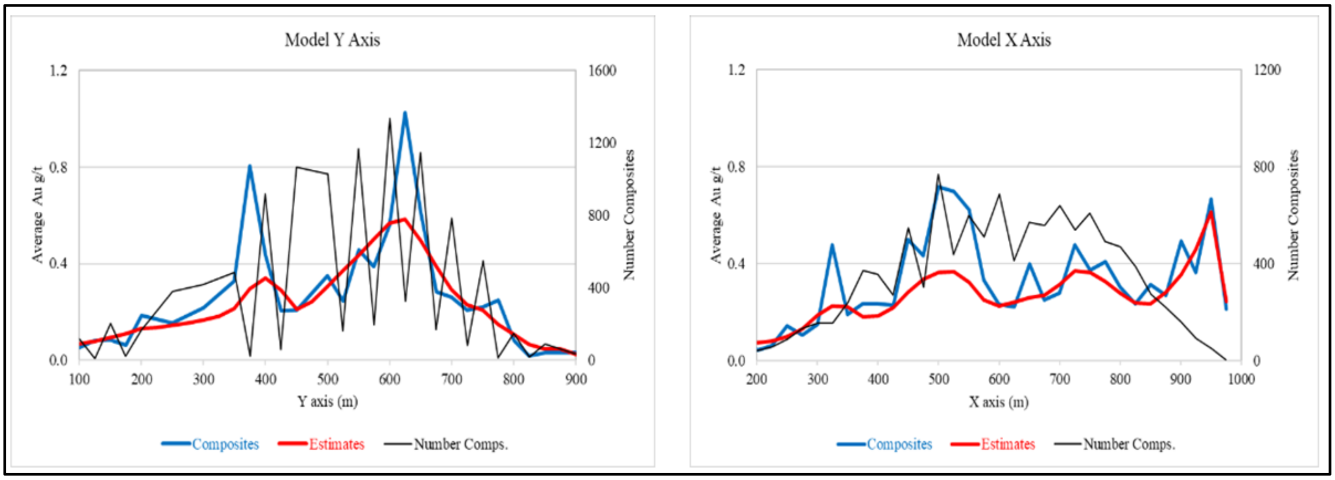


Figure 14-5: Average panel grades vs composite grades

14.8 Mineral Resource Estimate

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. To provide estimates with reasonable prospects for eventual economic extraction, Mineral Resources are reported within an optimized pit shell generated with the parameters shown in Table 14-5. These cost and revenue parameters were specified by Velocity. They generate a cut-off grade of 0.3 g/t Au, which is selected as the base case for Mineral Resource reporting.

Table 14-5: Parameters used to generate pit shell to constrain Mineral Resource estimates

Parameter		Value
Gold price		US$1,500 per troy ounce
Cost per tonne of material mined		US$2.59 per tonne
Cost per tonne of material milled, excluding mining		US$11.74 per tonne
Metallurgical recovery		79.3%
Refining charge		$1.44 per ounce
Average pit wall angles	Wall azimuth 030° to 150°	36°
	Wall azimuth 150° to 030°	40°

The optimal pit shell generated for constraining Mineral Resources has dimensions of around 770 metres by 660 metres, with a maximum depth of around 150 metres.

Table 14-6 shows the Indicated and Inferred Mineral Resource estimates for Rozino for a range of gold cut off grades. The author's evaluations demonstrate that all estimates resulting from each of the cut-off grade scenarios presented in this table meet the test of reasonable prospect of economic extraction. The figures in Table 14-6 are rounded to reflect the precision of the estimates and include rounding errors. The Indicated Mineral Resources are inclusive of Mineral Reserves.

Table 14-6: Rozino Indicated and Inferred Mineral Resource estimates

Effective date of estimates: 15 April 2020			
Cut-off (Au g/t)	**Tonnes (Mt)**	**Grade (Au g/t)**	**Metal (Au koz)**
Indicated Mineral Resource estimate			
0.2	27.2	0.72	630
0.3	**20.5**	**0.87**	**573**
0.4	15.5	1.04	518
0.5	12.0	1.22	471
0.6	9.42	1.40	424
Inferred Mineral Resource estimate			
0.2	0.49	0.7	11
0.3	**0.38**	**0.8**	**10**
0.4	0.29	0.9	8
0.5	0.23	1.0	7
0.6	0.17	1.2	7

15 Mineral Reserve Estimate

15.1 Introduction

The Mineral Reserves for the Rozino Gold Project have been reported using the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines, (2019). The Mineral Reserves are part of the Mineral Resources as stated in Section 14. The CIM Definition Standards 2014 define a Mineral Reserve as:

"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified."

The Rozino Gold Project has a single entity of mineralization that is well suited to open pit mining, predominantly because of the mineralization being near or at surface. The mineralization requires selective mining to ensure best value delivery.

15.2 Pit Optimization

The Rozino Mineral Resource block model was used to produce a mining block model that was then optimized to maximize cash flow using the Lerche-Grossman algorithm within context of a mining strategy.

15.2.1 Mineral Resource Model Conversion

The Mineral Resource block model is rotated 30˚ to grid north and has 25 x 25 x 5 m MIK panels. The pit is, however, relatively small in relation to the size of the panels and requires the application of complex slope geometry in the pit optimization phase that is not well represented by the wide and flat MIK panel dimensions. Consequently, as a compromise, the 25 x 25 x 5 m panels were split into 25 equal- sized 5 x 5 x 5 m rotated sub-blocks. The Mineral Resource codes for each sub-block (densities, mineral proportions, and grades) are identical to the panel codes. These smaller blocks provided improved accuracy and definition to wall angle estimation for the pit optimization process.

Verification to ensure that the mining block model was loaded identically to the original Mineral Resource model was undertaken. Implicit in the sub-blocked model is that some panels may only be partially recovered. It is an assumption of the applied method that a partially mined panel will deliver equally proportioned results to the whole panel. This is not entirely consistent to the theory of MIK as the recoverable proportion of the block assumes that the entire panel is selected for ideal accuracy. However, it should be noted that the volume of mineralization (ore) attributed to partially recovered panels is less than 5% of the Mineral Resource within the pit design. The impact of an average variation in results as a function of the sub-block method adopted is not considered significant with respect to the Mineral Reserve estimate. The impacts of modifying factors are described in the following sections.

15.2.2 Mining Strategy

The pit optimization process must be considered within the context of the overall mining strategy. The PEA (CSA Global, 2018) established a plant processing rate of 1.75 Mtpa and that a cut-off above the incremental break-even cut-off (IBECO) was helpful to the cash flow (in the PEA, the elevated operational cut-off was 0.7 g/t and it's IBECO was 0.35 g/t). The PFS Mineral Resource Indicated category tonnage was similar to that used in the PEA at the same cut-off, although with a reduction in estimated gold grade. Based on that information, it was concluded that the PEA strategy of using the elevated cut-off as the principal means to derive best project value remained valid. However, some minor modifications were made with new PFS information.

The base case PEA optimization was found to report about 9.47 Mt of mineral above a 0.7 g/t discard cut-off. The pit shell also included over 2.0 Mt of mineral below the 0.7 g/t cut-off but above the IBECO. This material was not utilized in the PEA and was discarded.

In the PFS the cut-off concept altered slightly in that the operational mine-life cut-off grade is raised to 0.8 g/t Au (high-grade ore). Material between 0.5 and 0.8 g/t Au (low-grade ore) is to be stockpiled and processed after the open pit extraction is complete. CSA Global have estimated that this strategy adds around $22M to the PFS pre-tax base case as compared to the PEA strategy. The tailings management facility (TMF) is dimensioned to contain the entire +0.8 g/t fraction mined and processed. Tailings of the low-grade material (0.5 to 0.8 g/t) will be stored within the mined-out pit. An additional implication of this is that the TMF is a fixed size and as such is no longer a factor in the IBECO (i.e. mining an additional tonne of low-grade mineralization does not increase the size of the TMF).

The capital cost to store tailings in the pit is estimated by CSA Global to be $0.2M and the establishment cost is independent of the tonnage stored (it is the same amount for 2.0 Mt as it is for 4.0 Mt). Therefore, the capital cost is not included in the IBECO calculation. It is estimated that the pit could store up to 5.0 Mt of tailings material, which is almost twice the capacity of the currently identified low grade material (2.7 Mt).

The pit optimization process relies on a waste strategy that minimizes the environmental footprint and decreases the waste mining cost. This strategy sees the mining of the eastern side of the deposit first as Phase 1 (located closest to the external waste dump location and also named Pit01). Once Pit01 is complete, the western pit (named Phase 2 or Pit02) commences, filling Phase 1 and gaining a waste cost reduction along with a reduced site footprint (not only from reducing the waste footprint, but also the TMF footprint by moving it further up the valley.

It was estimated that the PFS waste strategy adds $0.4 M to haulage costs compared to the PEA (CSA Global, 2018) as the PFS waste rock dump is higher. However, it decreases the clearing, topsoil removal and final rehabilitation cost by $0.6 M due to the smaller footprint. The net impact of the PFS waste strategy is a mild reduction in project cost of $0.2 M, but a significant 26 ha reduction in project footprint area.

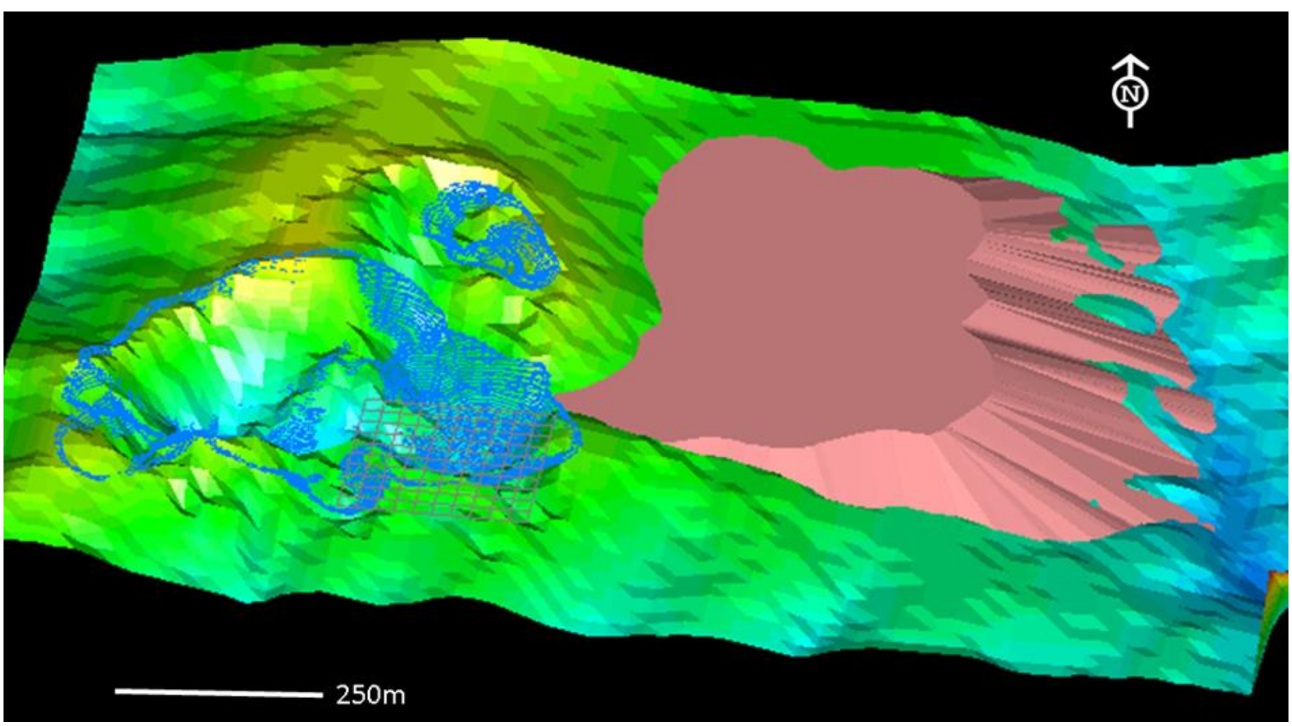


Figure 15-1: Isometric view to north (PEA waste dump in pink, PEA pit as blue strings)

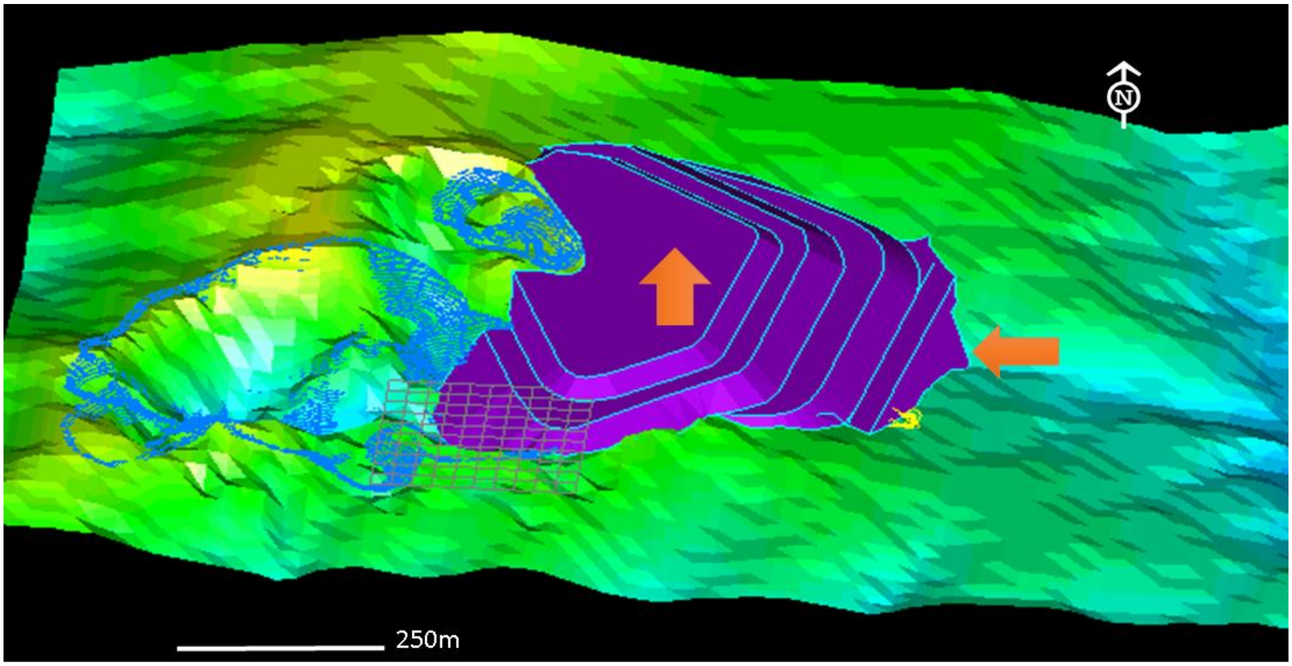


Figure 15-2: Isometric view to north (magenta – initial PFS waste dump design, blue strings – the PEA Pit)
Note: Arrows show the most significant PFS waste dump changes in growth compared to the PEA.

The combination of plant throughput, the size of the Mineral Resource and the number of mining benches means that the Mineral Resource can be mined only in two phases, a third phase such as an early high grade phase or a later cut-back is not possible. There is a "natural" division within the mineralization extent that enables the design of two pit phases with separated ramp access – east and west (Phase 1 and Phase 2 respectively). Phase 2 does contain higher grade than Phase 1.

Mining the west pit first strategy (thus generating an improved cash flow earlier in the mine life) would increase the project footprint by around 25 ha due to the reduced ability to backfill and the TMF facility would have to be moved downstream to allow for a larger waste rock dump footprint. Both of these would be detrimental from an environmental and regulatory point of view. Operating costs, initial clearing capital and closure costs would also increase. CSA Global estimates that the IRR of a west-pit-first strategy is no higher than the east-pit-first strategy – the additional waste and stripping costs are balanced by the earlier improved cash flow due to the improved grade profile. The adopted order of mining the pits is first Phase 1 (east) and then Phase 2 (west). This order of pit phasing is considered the best choice in terms of environmental impact, and with no difference in return on investment than the alternative strategy was thus adopted for the PFS.

15.2.3 Ore Confidence Category

The pit optimization process utilized only Indicated Mineral Resources to generate cash flow. No inferred Mineral Resources are included in the Mineral Reserves. Inferred Mineral Resources do not contribute to the financial performance of the project and are treated in the same way as waste.

15.2.4 Pit Optimization Procedure

15.2.4.1 Step 1

An initial series of pit shells were created by varying the gold price factor to the base price in steps of 5% and optimizing discounted cash flow using the LG algorithm. A discount rate of 5% and a 1.75 Mtpa mineral mining limit was applied. The pit wall angles were set at the inter-ramp angle (IRA). The pit shell which was used to guide pit design was based on incremental pit shell, incremental cash flow and incremental strip ratio. The final selection was the 90% price ($1,305 per oz) pit shell as material added with larger shells shows a clear change in economic performance and is adding waste for much higher risk on return (risk being measured as margin over incremental volume mined). The shells were examined for the possibility of phasing alternatives (discussed in Section 15.2.1), but the east-then-west mining strategy dominated. This option was therefore selected for design guidance. The initial cost schedule used the PEA (CSA Global, 2019) as guidance.

15.2.4.2 Step 2

After optimization, the pit and complementary waste designs were completed using the 90% price pit-shell and waste dump positions (giving access limitations) as constraints. The pit design was used to modify the IRA angles to overall slope angles (OSA) angles. This set of design parameters was used to modify the mining cost estimate.

15.2.4.3 Step 3

Optimization as per Step 1 was repeated using the new OSA angles and costs.

15.2.4.4 Step 4

The pit design was modified using the new 90% price shell.

15.2.4.5 Step 5

Optimizations were reported. OSA modifications for this repetition were minor and the cost model did not require updating. The costs described in this report and the mine-support document are based on this final run.

15.2.4.6 Step 6

Final designs for the pits and waste dumps were produced. A final design to the final optimization shell reconciliation was completed. This reconciliation demonstrated excellent correlation (see below).

15.2.5 Optimization Input Costs

The optimization input costs were estimated from first principles; costs were derived directly from Bulgarian suppliers or considered applicable to Bulgaria. The initial cost estimation relied on the PEA data, but these costs were replaced during the optimization iterations with costs continually fine-tuned to revised pit designs and waste dump configurations. There are marginal differences to the final cost estimates due to refinement in precise waste dump to mine bench pairing and small changes in dump and pit designs (the cost development process is identical to that in the mine cost section). At the end of the optimization-design-scheduling process a reconciliation of the optimization costs and the final schedule was completed. This demonstrated that although there were some differences in the final schedule compared to the optimization schedule, the total cost difference was minimal (within 2% for all parameters).

15.2.6 Drill and Blast

Drill and blast costs were determined on the basis of the mineral reserve with a minimum cut-off grade of 0.5 g/t, and ore type.

Where the panel ore proportion was greater than 50%, the ore drill and blast costing applied:

Oxide $/t	**Transitional $/t**	**Sulphide $/t**
0.09+0.43	0.09+0.73	0.09+1.07

Where the ore proportion fell below 50%, a mixed waste and ore cost applied:

Oxide $/t	**Transitional $/t**	**Sulphide $/t**
0.09+0.43*2P+0.44*(1-2P)	0.09+0.43*2P+0.44*(1-2P)	0.09+0.43*2P+0.44*(1-2P)

The costs were loaded to the mining block model and visually verified.

The cost of secondary ore breakage by an excavator-mounted rock hammer stationed at the ore loading bin is included in the ancillary equipment cost.

15.2.7 Load and Haul

Load and haul costs for ore and waste were applied separately and varied as a function of elevation above and below the pit lip datum (430 masl elevation). The costs are based on the use of the Volvo FMX 10X4 55 t haul trucks which are the preferred and selected truck for the mining operation.

Table 15-1: Load and haul costs applied to pit optimization

Mining costs (load and haul)	Units	FMX 10x4 50t
Reference elevation	masl	430
Single lane road width	m	9
Dual lane road width	m	12
Reference load and haul	$/t mined	0.88
Oxide – Waste	$/t mined	0.93
Trans – Waste	$/t mined	0.90
Fresh – Waste	$/t mined	0.88
Oxide – Ore	$/t mined	0.96
Trans – Ore	$/t mined	0.94
Fresh – Ore	$/t mined	0.90
Overhaul unit costs		
Oxide – Waste – vertical overhaul positive	$/t/Vm	0.0028
Trans – Waste – vertical overhaul positive	$/t/Vm	0.0026
Fresh – Waste – vertical overhaul positive	$/t/Vm	0.0024
Oxide – Ore – vertical overhaul positive	$/t/Vm	0.0038
Trans – Ore – vertical overhaul positive	$/t/Vm	0.0034
Fresh – Ore – vertical overhaul positive	$/t/Vm	0.0034

Load and haul costs were loaded to the mining model and visually verified.

15.2.8 Mine Support

Mine support costs were estimated at $0.35 per tonne and applied to ore and waste.

15.2.9 Grade Control

Grade control was estimated as $0.60 per tonne ore and applied to ore only.

15.2.10 High-Grade (HG) Ore Rehandle

The PFS is based on $0.64 per tonne rehandled or $0.19 per tonne ore for loader only (CAT988 or equivalent) rehandle of 30% of the material at a peak rate of 425 t/hr (75 m tramming distance) and an average feed rate of 275 t/hr. This assumes that campaigning of the different ore types can be efficiently scheduled with 70% of ore assumed to be direct tipped to the crusher. The crusher is designed with a two truck-load bin capacity (~100 tonnes).

HG ore is defined as any ore with an NSR of $11 per tonne or greater, which equates to a cut-off of approximately 0.8 g/t. HG ore was defined as any material greater than 0.8 g/t based on the panel grade above the 0.5 g/t cut-off. The 0.8 g/t cut-off panel grade and proportion were not utilized.

15.2.11 Low-Grade (LG) Ore Rehandle

The schedule strategy is that all LG material is stockpiled. The distance between the LG stockpiles and the crusher will mean that all LG ore on the stockpile will be loaded into a truck and direct tipped. In this process the scheduling was completed at the maximum capacity of the crusher of 475 tpoh. The LG ore rehandle cost of $0.54 per tonne was applied to all material between the LG cut-off of 0.5 g/t and the HG cut-off of approximately 0.8 g/t. A small amount of support equipment is assigned to road maintenance and rehandle excavation support to this phase of the operation.

15.2.12 Mine Administration

Mine administration was estimated at $0.12 per tonne mined using costs from the mine operating period (excluding mine start-up) and applied to ore and waste.

15.2.13 Mine Dewatering

The cost applied to the optimization was $0.04 per tonne for any material below the 430 masl elevation. The final financial model refined this approach and derived costs based on rainfall accumulations and groundwater inflows per period and the depth of mining in relation to the pit exit level. The final pumping cost was determined to be less than that applied in the optimization. However, the discrepancy is not material to the decisions made in this report.

15.2.14 General Administration

A general administration cost of $2.18 per tonne of high grade and $1.57 per tonne of low grade were applied.

The general administration cost was developed by the direct application of personnel required for the various functions using labour rates calculated in the Rozino labour cost memorandum as previously detailed.

Four basic activities were included:

- Administration (all fiscal operations and overheads, liability and asset insurance, operations manager, external road maintenance, community and public relations)
- Safety and security
- Environmental and permitting
- Human resources.

Note that costs such as meals are not included in the general administration costs but were allocated per person as a labour overhead. Labour costs constitute 30% of the general administration cost.

Administration costs include:

- Insurance to cover the Rozino Project site. The insurance does not cover concentrate haulage, concentrate processing, doré transport and mine equipment (included in leasing charge). The insurance estimate is based on a review of other similar projects and is estimated at $50,000 per month.
- External road maintenance is estimated at $11,000 per month based on a low capital amount indicating that some upgrades will be staged throughout the operating life alongside general road maintenance.
- Other fees are estimated for community and public relations, auditing, phone charges, computers and software, light vehicle leasing and maintenance, travel and town office rent.

Environmental section costs include:

- External service fees to cover on-going permitting, water sampling, studies, and site refuse disposal at $22,000 per month
- Operational materials associated to ongoing environmental activities at $13,000 per month

- Vehicle lease and maintenance.

Health and Safety Section costs include:

- Safety incentive program at $30,000 per annum
- General safety supplies not covered in the operations areas at $50,000 per annum
- Vehicle lease and maintenance.

Human Resources Section costs include:

- Legal fees, personnel events, recruitment, severance and relocation expenses at $204,000 per annum
- Busing expenses at $30,000 per annum
- Cleaning materials and services at $66,000 per annum.

Table 15-2: Scheduled administration costs

Administration	**Units**	**Rozino Project – PFS Mine Administration Cost**									
Description	year	-2	-1	1	2	3	4	5	6	7	**Total**
No. of days	days	365	365	365	365	365	366	365	365	329	
Total administration costs											
Administration	$k	668	969	1850	2039	2039	2044	1994	1629	1425	**13,019**
Safety and security	$k	197	286	545	600	600	602	587	480	420	**4,319**
Environmental and permitting	$k	171	247	471	519	519	521	508	415	363	**3,734**
Human Resources	$k	136	197	376	414	414	415	405	331	290	**2,978**
Total	**$k**	**1,172**	**1,699**	**3,243**	**3,572**	**3,572**	**3,582**	**3,494**	**2,856**	**2,497**	**25,688**
Tonnage milled											
Total	kt	0	0	1,488	1,750	1,750	1,750	1,750	1,750	1,586	**11,824**
Per tonne milled											
Administration	$/unit	0.00	0.00	1.24	1.16	1.16	1.17	1.14	0.93	0.90	**1.10**
Safety and security	$/unit	0.00	0.00	0.37	0.34	0.34	0.34	0.34	0.27	0.26	**0.37**
Environmental and permitting	$/unit	0.00	0.00	0.32	0.30	0.30	0.30	0.29	0.24	0.23	**0.32**
Human resources	$/unit	0.00	0.00	0.25	0.24	0.24	0.24	0.23	0.19	0.18	**0.25**
Total	**$/unit**	**0.00**	**0.00**	**2.18**	**2.04**	**2.04**	**2.05**	**2.00**	**1.63**	**1.57**	**2.17**

The site administration costs were benchmarked against a set of peer operations (small open pit operations with a similar plant and labour market). The costs are marginally lower based on the Rozino Project having slightly lower labour cost environment than the average peer. The variance is within the margin of error for this type of comparison.

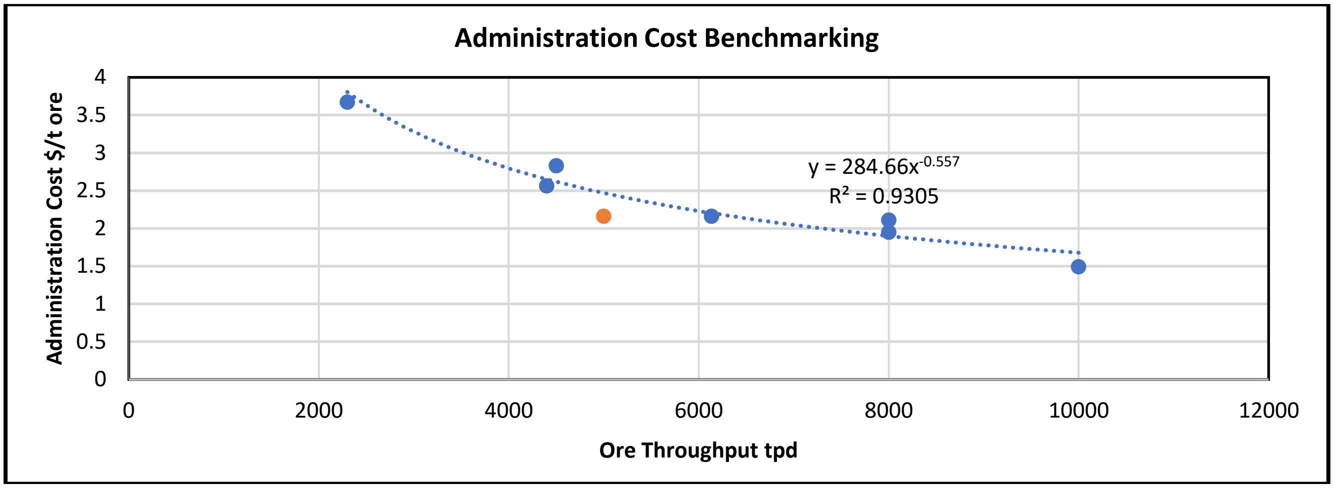


Figure 15-3: Administration cost: peer benchmarking

15.2.15 Ore Processing Operating Cost

The ore processing operating cost is a combination of the Flotation Plant cost, the cost to haul the concentrate to the Central Plant and the Central Plant treatment cost. This cost was estimated as being variable by oxidation type and is estimated in Table 15-3.

Table 15-3: Optimization ore processing costs and mass pull

	Unit	Oxide	Transitional	Sulphide
Flotation	$/tonne	7.60	7.60	6.79
Flotation Mass pull to CIL	%	2.14%	4.06%	4.18%
Concentrate Processing	$/tonne concentrate	62.16	62.16	57.68
Concentrate Haulage	$/tonne concentrate	13.92	13.92	13.92
Concentrate Treatment + Haulage	$/tonne	1.63	3.09	2.99
Total	**$/tonne**	**9.23**	**10.69**	**9.78**

Concentrate haulage costs were obtained by a single contractor quote at BGN0.25/wmt/km for an 85 km haul (verified with regional road haulage rates). Concentrate moisture content of 12% is assumed and storage capacity at the Rozino site is minimal. The concentrate haulage cost equates to $13.92/dmt.

Concentrate treatment costs of approximately $62.16/t were based on negotiations with Gorubso. However, indications from Gorubso are that the Sulphide ore may have a treatment cost of $57.68/t due to a lower cyanide consumption rate than the oxidized ores. The final schedule assumed a mixed supply of concentrate types as Gorubso will process a mixed-feed concentrate. This variance is not significant to the optimization results; it adds $0.19/t to the sulphide ore costs or $2.25M to the total operating cost of $243.9M (~0.9%). Based on the metallurgical testwork it is expected that the final processing cost will be a mix of the lower and higher cost based on the degree of oxidation, as applied in the optimization process.

15.2.16 Sustaining Cost

The Flotation Plant has a short life, and its estimated operating cost is considered to cover all sustaining requirements. The TMF construction costs are not linked to incremental tonnages and so they are not considered sustaining. The closure rehabilitation costs are only mildly related to incremental tonnages and were therefore not considered.

15.2.17 Metallurgical Recovery

Metallurgical recovery formulae were developed by CSA Global from the PFS metallurgical testwork. Recovery is a function of oxidation state and gold head grade. The formulae in the optimization applied to the gold grade above the 0.5 g/t cut-off in Table 15-4.

Table 15-4: Gold metallurgical recovery formulae

Oxide	75.91*(Au-0.12)/Au
Transitional	78.9*(Au-0.12)/Au
Sulphide	91.97*(Au-0.12)/Au

The formulae were input directly into the mining block model. As the formulae are not linear, accumulations must be completed on a block-by-block basis for acceptable accuracy. The recovery formulae include the impact of CIL losses of 0.5%.

15.2.18 Metal Costs

CSA Global estimated metal costs based on a doré that would measure no more than 60% gold. The doré purity was based on the following table.**Error! Not a valid link.**No direct tests to produce a doré bar have been completed. The CIL smelting loss for copper assumes that cold stripping will be undertaken to reduce the copper content to no greater than 5%. During the optimization, the doré purity was estimated as 60% Au based on early testwork results. Doré sale costs were based on a gold purity of no more than 60% and no more than 5% copper.

Gold metal sales costs were estimated at $8.97 per oz payable gold (assuming a 60% doré gold purity). The gold payable proportion was estimated at 99.8%.

Although it is likely that the doré will contain some silver, silver content was not estimated in the Mineral Resource estimation and so no value was attributed to silver in any part of the optimization or financial assessment. The cost to transport the doré inclusive of the silver, copper and other contaminants is covered by the $8.97 per oz payable gold summary cost. Transport and insurance costs assume a refinery located in Switzerland, but this is not an established contract. Detailed cost estimations will be required in the FS on the choice of refinery; Bulgarian alternatives will feature strongly in the evaluation.

15.2.19 Royalty

A Bulgarian state royalty (concession royalty) is determined at the time of granting a mining licence. The royalty is based on the NSR but varies annually as a function of profitability in line with the mining plan submitted to the government. Royalties are generally between 0.8 and 2.5% NSR.

The royalty was assumed to be an average of 2% over the life of the mine and calculated on the NSR.

15.2.20 Metal Price

The optimization used a base gold price of $1,450 per oz. This price is lower than the $1,500 per oz applied to the final financial model. However, this does not alter the choice of the design shell which was selected at $1,305 per oz. The final pit shell selection has other considerations such as the waste strategy and additional waste costs that should apply to larger shells (hence reducing the likelihood of selecting them through increasing cost and associated risk – additional explanation is provided below).

15.2.21 Constraints

No area or geographical constraints surrounding the Rozino deposit were identified and that required the optimization to be constrained by using area exclusions or "heavy" blocks. Environmentally sensitive areas, private land ownership, and water catchment areas were considered during the conceptual placement of infrastructure (tailings management facility, water storage dams, plant and waste rock dump).

15.2.22 Operational Discard Cut-off

The input parameters for the calculation of the IBECO are set out in Table 15-5. The calculated IBECO cut-offs for the Oxide, Transitional and Sulphide material types are 0.47, 0.51 and 0.43 g/t respectively. The underlying Mineral Resource model has cut-off values developed at 0.6, 0.5 and 0.4 g/t (amongst others at higher and lower values). Consequently, an IBECO of 0.5 g/t was selected for the operational cut-off. At $1,500 per oz there is a marginal increase in profitability if the operational cut-off were reduced to 0.4 g/t and there was no change to the ore-waste cost balance. If the cut-off were to be decreased in a future reserve estimation, a redesign of the WRD, TMF and low-grade stockpile configurations would be required to make a refined estimate of the cost balance change and justify the change in cut-off. It is estimated based on current knowledge that a change to a 0.4 g/t cut-off is not material to the PFS stated economics.

The formula used to calculate IBECO for each ore type is:

$$IBECO = TC/Rec/Pay/((Price\text{-}Sell)*(1\text{-}Roy)/31.1035)$$

Where:

- TC = Total cost $/t = SustCap + Env + LowAdmin + CIL + Float + LGreh + OreInc
- SustCap is the applicable sustaining capital cost in $/t
- Env is the applicable environmental cost in $/t
- LowAdmin is the low-grade ore administration cost at time of processing in $/t
- CIL is the cost to process the concentrate in ore feed terms in $/t
- Float is the cost to run the Flotation Plant on this ore type in $/t
- LGreh is the cost to rehandle the low-grade ore in $/t

OreInc is the incremental ore mining cost at the time of mining in $/t and measures only the additional costs applicable to ore mining over waste mining.

$$Example\ TC = 0.00 + 0.00 + 1.57 + 1.63 + 7.6 + 0.56 + 0.81 = \$12.17/t$$

and

$$IBECO = 12.17 / 0.561 / 0.998 / ((1450 – 8.97) * (1\text{-}0.02) / 31.1035)$$

$$IBECO = 0.47\ g/t$$

Operational Discard Cut-off applied = 0.50 g/t

Table 15-5: Input parameters for the calculation of IBECO

Parameter	Units	All ore types		
Metal price	$ per oz Au	$1450		
Doré selling costs	$ per oz Au	8.97		
NSR royalty	% of NSR	2.0%		
Dore payability	% of Au in doré	99.8%		
		Oxide	**Transitional**	**Sulphide**
Ore incremental mining cost	$/tonne	0.81	1.05	1.03
Stockpile rehandle cost	$/tonne	0.56	0.56	0.56
Flotation	$/tonne	7.60	7.60	6.79

Flotation mass pull to CIL	%	2.14%	4.06%	4.18%
Concentrate CIL processing	$/tonne concentrate	62.16	62.16	57.68
Concentrate haulage	$/tonne concentrate	13.92	13.92	13.92
CIL + concentrate haulage	$/tonne	1.63	3.09	2.99
Low grade admin	$/tonne	1.57	1.57	1.57
Environmental provision	$/tonne	0.00	0.00	0.00
Sustaining capital	$/tonne	0.00	0.00	0.00
Total	**$/tonne**	**12.17**	**13.87**	**12.94**
Oxidation	units	3.08	2.60	0.35
Recovery at cut-off	%	56.7%	60.2%	66.0%
IBECO	g/t Au	0.47	0.51	0.43
Cut-off	g/t Au	0.5	0.5	0.5

15.2.23 Mining Recovery and Dilution

CSA Global consider that at the level of a PFS no additional mining dilution or mining loss to that contained in the MIK estimation of resources is considered necessary for the statement of a Probable Reserve. The Probable Reserve for the Rozino Project is based on the December 2019 MIK Mineral Resource Estimate undertaken by MPR.

CSA Global were able to determine through written communications and discussions with Neil Schofield (FSSI), and other written correspondence with MPR that the mineralization can be adequately modelled for its diluted recoverable grade properties assuming an SMU of 4 x 6 x 2.5 m using the MIK methodology. CSA Global consider that the mineral defined by the Mineral Resource estimate can be adequately mined by open cut extraction using drill and blast on 5 m benches, utilizing 90 t excavators with a two m wide bucket to mine two 2.5m flitches, loading to 55 t capacity trucks. In order for the assumption of zero additional mining dilution or mining loss to be applied, the cost estimate includes provision to assure the mining crew (technical and production) are well trained, that excellent grade control practices are applied and mining technical support (especially for blast design, monitoring and displacement detection) is constantly applied.

CSA Global have considered the mechanical properties of the ore and waste material in relation to excavation mechanics, that is the blasting, excavation and loading of the ore into trucks. The data reviewed was from that collected and reported by Golder (2019, 2020) and from review of the site road cuttings and drill core by the QP. In this regard, the Rozino orebody is considered geomechanically relatively simple; there are no significant amounts of overly weak material that may cause undue sliding of excavation faces into or out of the excavation face, water ingress is low (no movement of mineral due to water flow) and the material is expected to be mechanically sound when wet (no puddling or water disassociation of materials or bogging-down of machinery). Conversely, there are no observed advantages to excavation mechanics such as a highly visible mineral boundaries or distinctive ore-to-waste differentiation that may augment grade and tonnage recoveries.

Mining loss and dilution may occur not as a function of the design of the SMU unit and Resource Estimation, but as a result of the mining methodology. Consequently, some dilution (especially that associated to human error) cannot be encapsulated through the MIK methodology directly. Applying appropriate operational management systems additional dilution (to the MIK estimate) and mining losses may be kept to a minimum and considered insignificant in relation to the other estimation risks. To support the application of no further mining loss and dilution factors it is a requirement that all appropriate methods will be applied in studies and cost estimation to ensure that these sources are minimized. These activities include but are not limited to:

- Completion of a grade control SMU study during the Feasibility Study. It is important that the study be accompanied by confirmatory grade control drilling results.

- Maintaining a well-trained mining production and technical crew.
- Design, monitor, and evaluate mine blasts to minimize ore heave, and physically monitor and adjust the location of all mineral boundaries.
- Ore excavation activity will always be monitored and directed by a skilled geological technician.
- Mining of the ore from the optimal direction to minimize dilution due to adverse ore dips.
- To enable optimal short-term mine planning grade control drilling should be at least 3 months in advance of mining.
- Regular and frequent ore reconciliations with constant feed-back to planning, production, and monitoring.
- The mining schedule rate must be at a level that allows adequate grade control and operational management to minimize dilution and optimize scheduling.
- Adequate pit lighting and supervision during night shift will be required.

15.2.24 Optimization Input Wall Angles

The Golder report entitled "Rozino Pre-feasibility Study Geotechnical – Stage 4 – Interpretive Analysis and Slope Design – December 2019" presented stability analysis and slope design criteria that were a product of the analysis of geotechnical data collected for the Rozino Gold Project. The data collected included oriented drill core, logging of existing drill core and UCS and tensile tests on core were completed. The analysis considered the presence of modelled faults, rock density, rock type, rock mass joint characteristics and orientations, the effect of pit blasting (pre-split blasting was applied), seismic influence and the hydrological regime.

The design of the berms also considers 2D analysis of rock falls. The geotechnical report utilized the PEA pit design. Design analysis included kinematic analysis and overall slope stability.

Golder concluded that "the analyses for each pit domain provide for similar or slightly more aggressive inter-ramp angles than those derived for the PEA". The introduction of robust rock mass and structural data has allowed for optimization of the geotechnical pit slope angles. The geotechnical slope design is based on geotechnical core rock mass logging, laboratory testing data, and the large- and small-scale structural models. The input parameters to the slope design process have been sourced from structural and rock mass core logging, laboratory tests and data from similar projects and rock types. The kinematic analysis combined with the limit equilibrium modelling indicate that the pit slope angles derived will satisfy the acceptability criteria. The Factor of Safety and Probability of Failure of the overall slope models cut through the geological and weathering wireframes satisfy the acceptability criteria.

The results of the geotechnical analysis developed a set of berm, batter and inter-ramp angle recommendations for 6 geotechnical zones that were aligned to the PEA pit design.

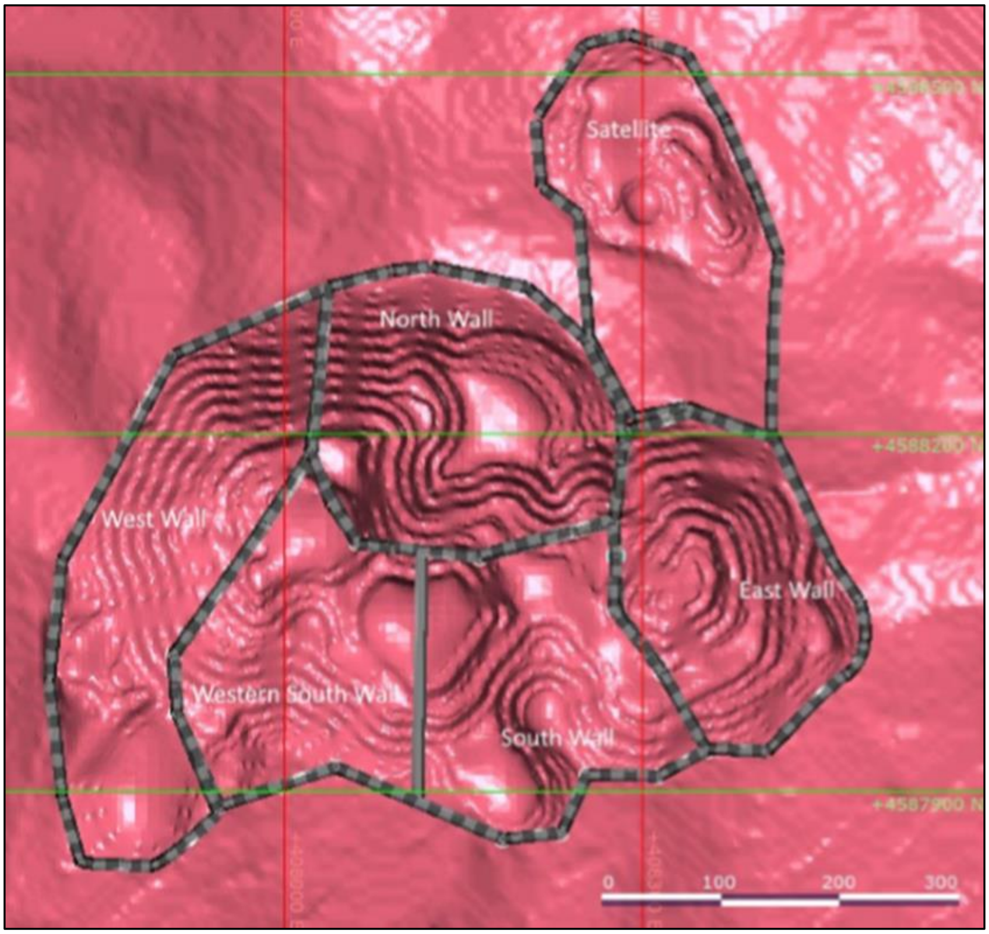


Figure 15-4: Plan of Rozino PEA pit with pit-wall domains indicated WGS 84 Zone 35N
Source: Golder, 2019

Those geotechnical zones were further subdivided by wall strike orientation, rock type (sediment, olistostrome and basement) and weathering (weathered (oxidized), partially weathered (transitional) or un-weathered (sulphide)). This developed 87 possible design sectors. Additional notes were supplied on the presence of faults.

CSA Global reviewed the 87 sectors and found that they could be simplified to 20 sectors as some combinations of rock type, weathering and wall angle were of little significance in terms of pit wall area. Some wall sector simplifications arose from the PFS optimization results.

The engineering block model contains block codes that combined oxidation (OXCOD) and Rock Type (LITH) to a single field RCOD as per Table 15-6.

Table 15-6: Relationship of lithology and oxidation in the Mineral Reserve block model

Combined Oxide-Lithology (RCOD)		Lithology LITH		Oxidation OXCOD	
1	Oxide Sediment	1	Sediment	1	Oxide
2	Transitional Sediment	1	Sediment	2	Transitional
3	Sulphide Sediment	1	Sediment	3	Sulphide
4	Oxide Olistostrome	2	Olistostrome	1	Oxide
5	Transitional Olistostrome	2	Olistostrome	2	Transitional
6	Sulphide Olistostrome	2	Olistostrome	3	Sulphide
7	Oxide Basement	3	Basement	1	Oxide
8	Transitional Basement	3	Basement	2	Transitional

9	Sulphide Basement	3	Basement	3	Sulphide

The CSA Global pit optimization and design sectors, OSAs, IRAs, BFAs, and bench widths utilized for pit optimization and design are summarized in Table 15-7.

Table 15-7: Sectors used in pit optimization and design

SECTR	IRA (°)	OSA (°)	Batter (°)	Berm (m)
1	36.2	36.2	70	5
2	40.4	40.4	80	5
3	34.1	34.1	80	6.5
4	44.3	44.3	80	8.5
5	47.2	47.2	80	7.5
6	48.8	48.8	80	7
7	32.4	32.4	80	7
8	45.7	45.7	80	8
10	45	45	80	8
11	48.1	48.1	70	8
12	52.2	52.2	80	9
13	53.4	43.2	80	8.5
14	52.2	42.2	80	9
15	58.5	48.5	80	6.5
16	47.1	47.1	70	8.5
17	58.5	58.5	80	6.5
18	53.4	53.4	80	8.5
19	47.7	37.7	80	11
20	52.2	42.2	80	9
21	52.2	52.2	80	9

Sectors 14 to 16 and 19 to 21 include provision for pit ramps. The OSA reductions were developed over a series of designs and optimizations.

Because rock type and oxidation state as well as pit sector control wall angles, the sectors do not remain static with depth. Figure 15-5 shows the sectors for bench 445.

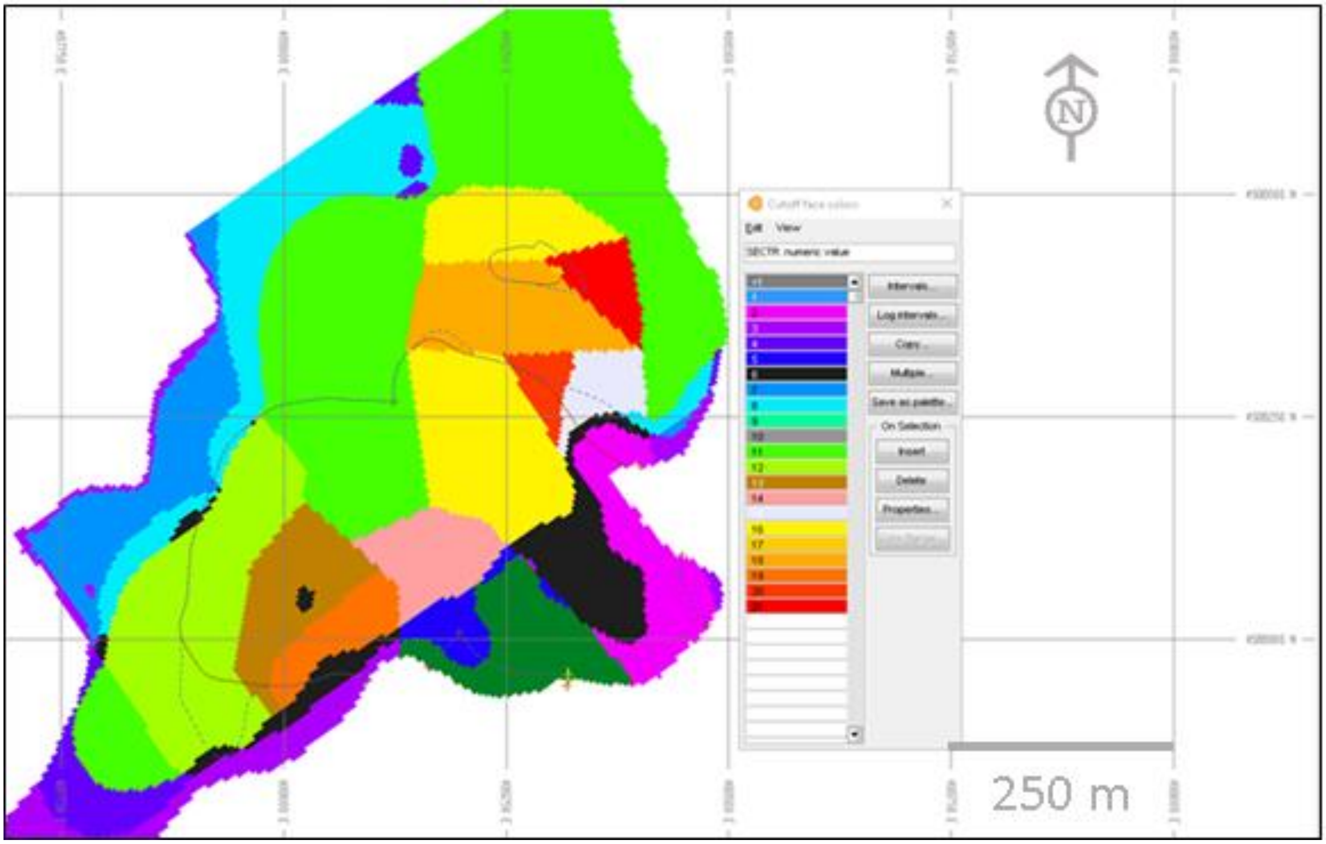


Figure 15-5: Bench 445 showing wall design sectors and the pit design WGS 84 Zone 35N

The QP considers that the interpretation of the geotechnical parameters was adequately performed. As part of the iterative nature of the design process, the PFS pit design will be subject to further geotechnical sector revisions during the FS.

15.2.25 Optimization Results

Lerche-Grossman optimizations maximizing the discounted net cash flow were completed at 5% gold price increments above and below the base case gold price of $1,450 per oz. A cash flow discount rate of 5% and a 60 m per annum maximum advance rate was applied.

Figure 15-5, Figure 15-7, and Figure 15-7 show undiscounted cash flow, incremental strip ratio and incremental cash cost respectively per recoverable ounce of gold as well as shell mineral and waste tonnage.

The rapid rate of pit advance in a shallow pit results in little significant difference between discounted and undiscounted cash flow. All figures use the base gold price of $1,450 per oz to determine the revenue within the respective shells.

The incremental cash flow above the $1,305 per oz pit shell is minimal due to the addition of ore at a strip ratio of greater than 4:1 and reducing quantities of ore in each subsequent increment.

The same 0.5 g/t IBECO was used for all gold prices. However, the optimization algorithm automatically rejects mineral if there is insufficient grade to pay for processing at lower metal prices. Ideally, higher cut-off indicators would be used for lower ranges of prices (and vice-versa for higher metal prices), but this is less practical in the Lerche-Grossman process. The indicator cut-off refinement would deliver slightly more value to the optimization shells, however it has been estimated that this is outweighed by increased waste haulage costs that would also have to accompany these larger shells. The simpler case of not altering either the waste cost or the indicator was taken for time efficiency as the most appropriate shell for design work had both the appropriate indicator and waste costs. The 0.5 Au g/t cut-off indicator is most appropriate over the $1,100 to $1,500 per oz price range. Above $1500 per oz the 0.4 g/t indicator would be more appropriate and below $1,100 the 0.6 g/t indicator. Given that the base price is within the most appropriate range for 0.5 g/t and the mining strategy being limiting on a final shell selection, only the 0.5 g/t indicator was utilized.

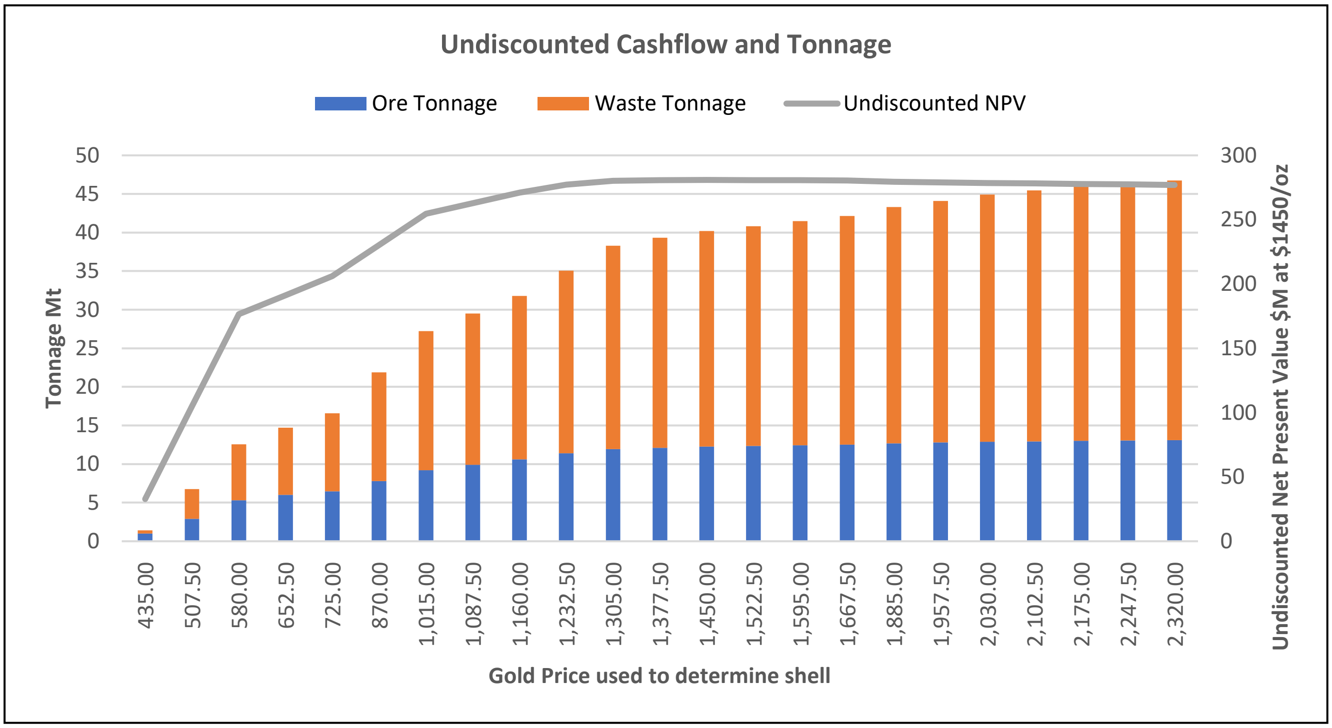


Figure 15-6: Pit shells and un-discounted net cash flow

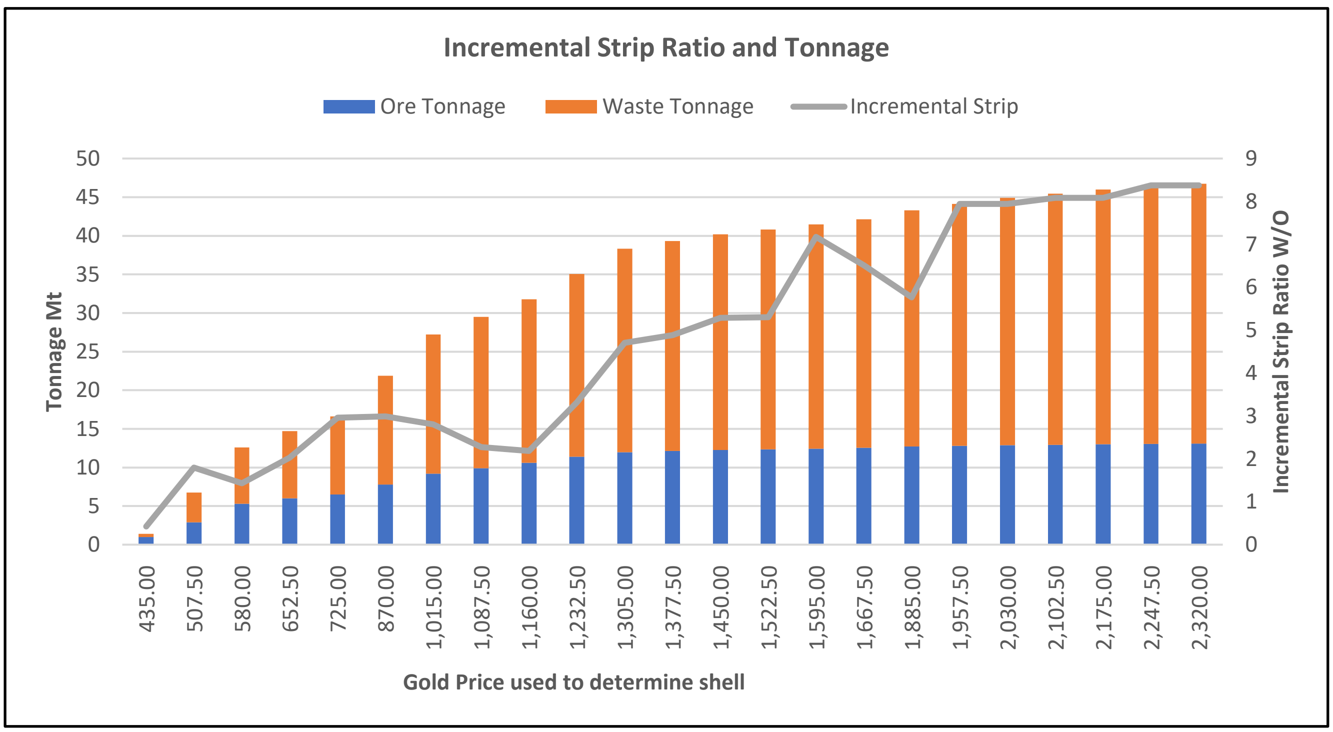


Figure 15-7: Pit shells and incremental strip ratio

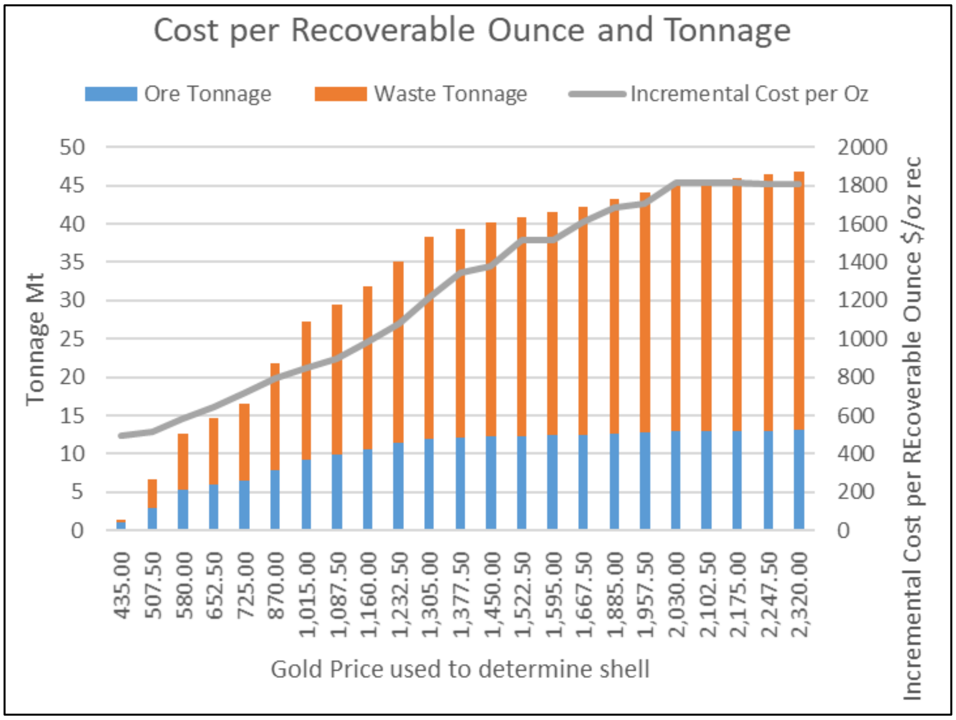


Figure 15-8: Pit shells and incremental cost per recovered ounce

The optimal pit shell is at a gold price of $1,305 per oz, but its selection is also based on the entire mining strategy, including minimising the project footprint and waste rock and tailings disposal.

Table 15-8: Parameters generated by the $1,305 per oz pit shell

Parameters Within the $1,305 Pit Shell (Mineral is 100% Indicated)			
Mineral	Tonnage	Mt	11.9
	Grade	Au g/t	1.22
	Contained gold	koz	470
	Gold recovery	%	79.3
	Recovered gold	koz	373
Non-mineral	Tonnage	Mt	26.4
	Strip ratio	(W/O)	2.21
Total	**Tonnage**	**Mt**	**38.3**

Table 15-9: Costs ($/t mine, processed and $/oz recovered gold) for the $1,305 pit shell

Item	$M	$/t mined	$/t processed	$/oz Au recovered
Mining Cost	103.4	2.70	8.66	277
Processing Cost	117.6		9.84	315
Administration Cost	24.4		2.04	65
Refining Cost	3.3		0.28	9
Royalty Cost	10.7		0.90	29
Total Cost	**259.5**		**21.71**	**695**
Revenue	541.1		45.28	1,450

15.3 Pit Design

The selected economic pit shell ($1,305 per oz) was used as the basis for the pit design. The pit design was completed in the Minesight software using the pit wall angle sector parameter (SECTR) to control changes to batter angle, berm width and IRA. The OSA was estimated by a process of adding the pit ramps in a series of design and pit optimizations aimed to minimize the impact of the input ramps to net cashflow. The ramp designs aimed to incorporate the fewest switchbacks and best operator visibility; these last safety considerations sometimes work counter to the calculation of net cashflow reduction. The pit optimization shell selection benefitted from three iterations of design and optimization to ensure integrity of the optimization and design process. The design ultimately selected for the PFS contained approximately 1% less ore but the same total tonnage as the optimized pit shell. The total net cash flow of the design was 2% less than the optimized pit shell.

The pit design comprised two phases: Phase 1 (Pit01) and Phase 2 (Pit02).

All ramps are 12 m in width and designed to suit the Volvo 55t FMX trucks. The ramp width is a key cost saving in comparison to the 17 m width required for similar capacity trucks such as the Cat772.

The Phase 1 pit includes a one-way ramp of 9 m width in the bottom four benches and for access of the small northeast sub-pit for its final five benches. All pit bases have ramp access.

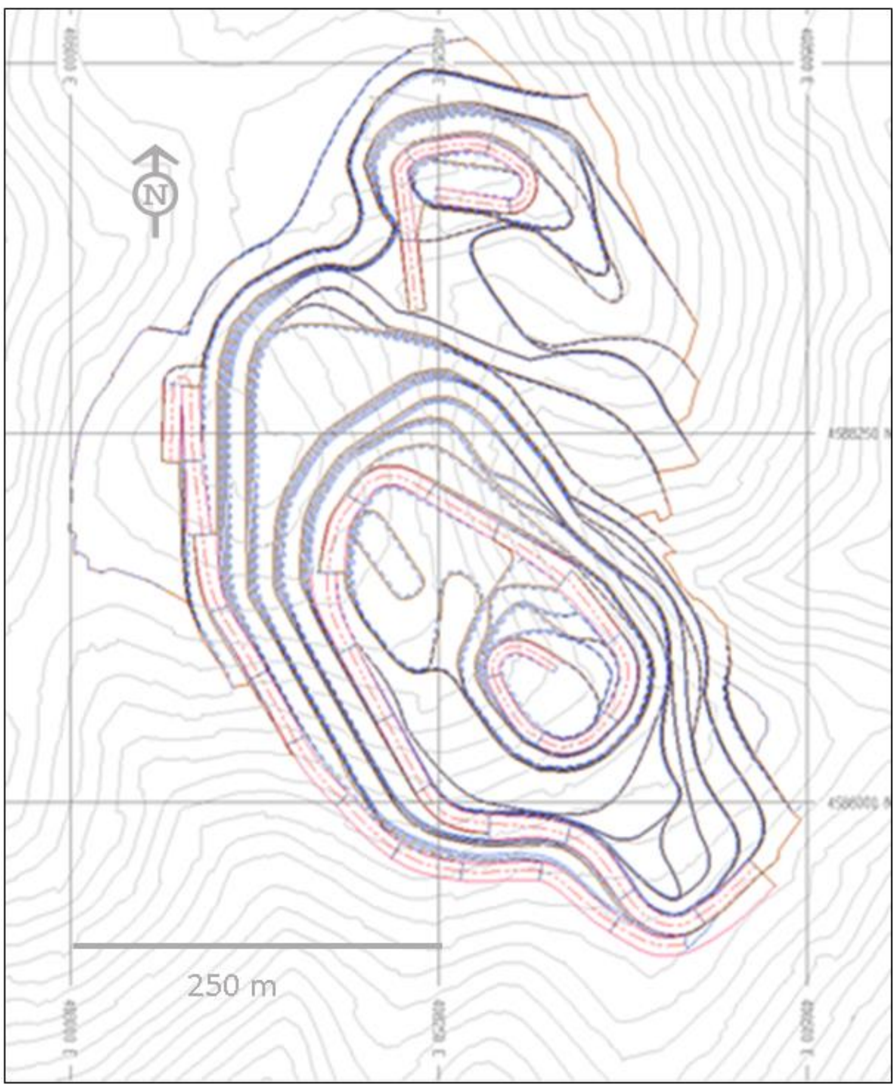


Figure 15-9: Phase 1 design – WGS 84 Zone 35N

The minimum pit base width is 20 m, but most are more open than that as the underlying Mineral Resource model has a 25 x 25 m parent cell and this dominates the optimization basal width. In addition to this, the pit bases tend to be wide and open as there is a shallow plunge to the multiple mineralization zones in the Mineral Resource block model from the north-east downwards towards the south-west.

Designed ramps and pit walls were manually smoothed during the design process to give walls that are more practical to attain. Ramps were also carefully designed to improve driving conditions.

The Phase 2 pit contains two switchbacks. The highest switchback was constructed with a 10 m internal radius in order to reduce truck impact and improve traffic visibility. The lower switchback is in the final four benches and so will have minimal volume.

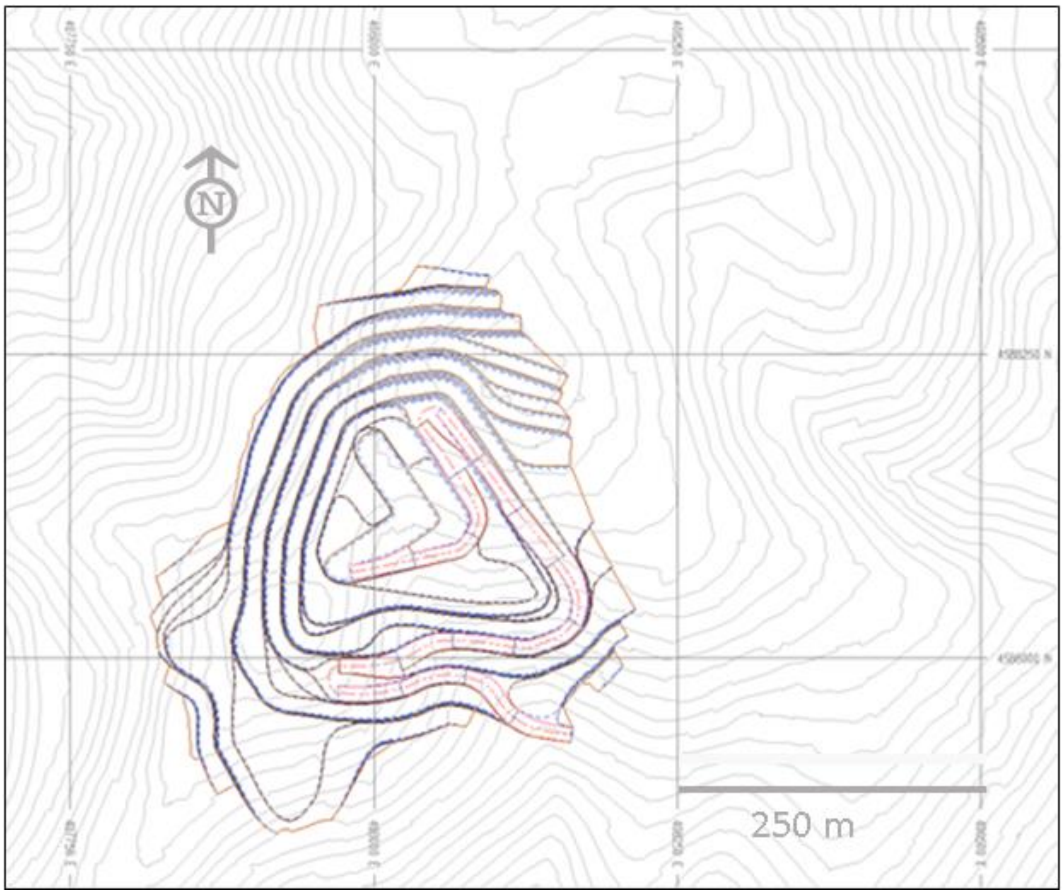


Figure 15-10: Phase 2 design WGS 84 Zone 35N

All catch-berms have ramp accesses designed but are not meant for vehicular access.

Catch-berms are designed every four benches (each 20 m vertically).

No major material catch-berm was designed as most walls les than 100 m in height.

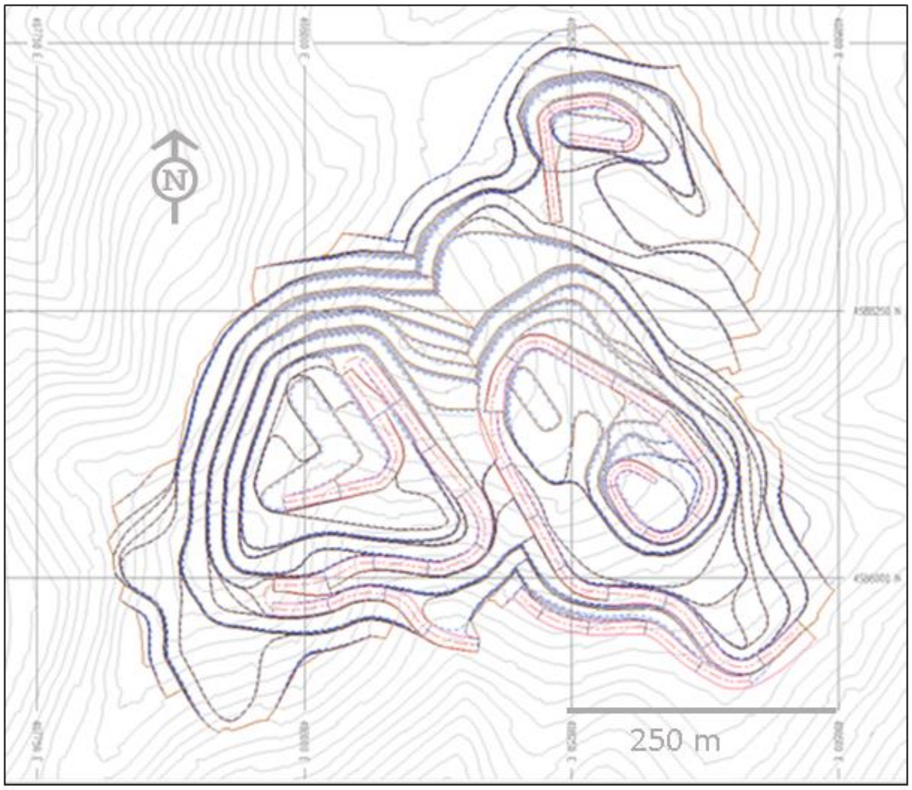


Figure 15-11: The combined Phase 1 and 2 pits WGS 84 Zone 35N

The pit ramp structures in the designs include consideration of the waste dumping strategy. The Phase 1 ramps below 435 masl elevation are to be buried under Phase 2 waste and so they are not useable for the lower Phase 2 advancement.

Table 15-10: Parameters within the pit design

Parameters within the pit design (Mineral is 100% Indicated)			
Mineral	Tonnage	Mt	11.8
	Grade	Au g/t	1.22
	Contained gold	koz	464
	Gold recovery	%	79.3
	Recovered gold	koz	368
Non-mineral	Tonnage	Mt	26.5
	Strip Ratio	(W/O)	2.21
Total	**Tonnage**	**Mt**	**38.3**

Table 15-11: Costs and revenue within the pit design

Item	$M	$/tonne mined	$/tonne processed	$/oz rec
Mining Cost	103.4	2.70	8.75	281
Processing Cost	117.6		9.84	316
Administration Cost	24.4		2.04	66
Refining Cost	3.3		0.28	9
Royalty Cost	10.7		0.90	29
Total Cost	**257.8**		**21.81**	**700**
Revenue	**534.0**		**45.18**	**1450**

Table 15-12: Ore type and rock type breakdown in Mineral Reserve

Reserve summary	Ore (Mt)	Waste (Mt)	Total (Mt)	Gold			
				Grade (g/t)	Contained (koz)	Recovery (%)	Recovered (koz)
Oxide	**1.9**	**3.9**	**5.7**	**1.07**	**64**	**67.4**	**43**
Sediment	1.7	3.4	5.1	1.07	58	67.4	39
Olistostrome	0.2	0.3	0.5	1.06	6	67.3	4
Basement	0.0	0.1	0.1	1.07	0	67.4	0
Transitional	**1.8**	**3.8**	**5.7**	**1.15**	**68**	**70.7**	**48**
Sediment	1.7	3.2	4.9	1.17	63	70.8	44
Olistostrome	0.2	0.5	0.6	1.02	5	69.6	3
Basement	0.0	0.1	0.1	0.80	1	67.1	1
Sulphide	**8.1**	**18.8**	**26.9**	**1.27**	**332**	**83.3**	**277**
Sediment	6.2	12.6	18.8	1.30	259	83.5	216
Olistostrome	1.4	5.3	6.8	1.07	49	81.6	40
Basement	0.5	0.8	1.3	1.56	25	84.9	21
Total	**11.8**	**26.5**	**38.3**	**1.22**	**465**	**79.3**	**368**
Sediment	9.5	19.3	28.8	1.24	379	78.9	299
Olistostrome	1.7	6.1	7.9	1.06	60	79.2	47
Basement	0.5	1.1	1.6	1.51	26	84.3	22

Table 15-13: Detailed cost breakdown for the Mineral Reserve with rock type and ore type in $M

Reserve summary	Total cost	Revenue	Rev-cost	Load and haul		Pumps	Drill and blast		Support	Mine G&A	LG rehandle	Grade control	HG rehandle	Mill	Admin	Refining	Royalty
		$1,450/oz		Ore	Waste	All	Ore	Waste	All	All	LG ore	Ore	HG ore	Ore	Ore	Ore	Ore
	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m	$m
Oxide	**36.2**	**62.4**	**26.2**	**2.0**	**4.1**	**0.0**	**1.0**	**2.2**	**2.0**	**0.7**	**0.4**	**1.1**	**0.4**	**17.1**	**3.6**	**0.4**	**1.2**
Sediment	32.6	56.5	23.9	1.8	3.6	0.0	0.9	2.0	1.8	0.6	0.4	1.0	0.3	15.4	3.2	0.3	1.1
Olistostrome	3.3	5.8	2.5	0.2	0.3	0.0	0.1	0.2	0.2	0.1	0.0	0.1	0.0	1.6	0.3	0.0	0.1
Basement	0.3	0.1	-0.2	0.0	0.1	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Transitional	**39.7**	**70.0**	**30.4**	**1.9**	**3.8**	**0.0**	**1.5**	**2.7**	**2.0**	**0.7**	**0.4**	**1.1**	**0.4**	**19.8**	**3.6**	**0.4**	**1.4**
Sediment	35.4	64.3	28.9	1.7	3.2	0.0	1.4	2.3	1.7	0.6	0.4	1.0	0.3	17.8	3.2	0.4	1.3
Olistostrome	3.6	5.0	1.4	0.2	0.5	0.0	0.1	0.3	0.2	0.1	0.0	0.1	0.0	1.6	0.3	0.0	0.1
Basement	0.7	0.7	0.0	0.0	0.1	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0
Sulphide	**181.9**	**401.5**	**219.6**	**8.1**	**17.6**	**0.5**	**9.5**	**19.7**	**9.4**	**3.2**	**0.6**	**4.9**	**1.5**	**79.4**	**17.0**	**2.5**	**8.0**
Sediment	135.0	313.2	178.2	6.1	11.8	0.4	7.3	13.3	6.6	2.3	0.4	3.7	1.2	60.7	13.1	1.9	6.2
Olistostrome	36.3	57.5	21.2	1.4	5.0	0.1	1.6	5.5	2.4	0.8	0.2	0.8	0.3	13.8	2.9	0.4	1.1
Basement	10.6	30.8	20.2	0.5	0.8	0.0	0.6	0.9	0.5	0.2	0.0	0.3	0.1	4.9	1.1	0.2	0.6
TOTAL	**257.9**	**534.0**	**276.1**	**12.0**	**25.4**	**0.5**	**12.1**	**24.6**	**13.4**	**4.6**	**1.4**	**7.1**	**2.2**	**116.3**	**24.1**	**3.3**	**10.6**
Sediment	202.9	434.0	231.1	9.7	18.6	0.4	9.6	17.6	10.1	3.5	1.2	5.7	1.8	94.0	19.5	2.7	8.6
Olistostrome	43.3	68.4	25.1	1.8	5.8	0.1	1.9	6.0	2.8	0.9	0.3	1.0	0.3	17.1	3.5	0.4	1.4
Basement	11.7	31.6	20.0	0.5	1.0	0.0	0.6	1.0	0.6	0.2	0.0	0.3	0.1	5.2	1.1	0.2	0.6

Table 15-14: Detailed cost breakdown for the Mineral Reserve with rock type and ore type in $/t

Mineral Reserve summary	$/t ore	$/t ore	$/t ore	Load and haul		Pumps	Drill and Blast		Support	Mine G&A	LG rehandle	Grade control	HG rehandle	Mill	Admin	Refining	Royalty
				Ore	Waste	All	Ore	Waste	All	All	LG ore	Ore	HG ore	Ore	Ore	Ore	Ore
				$/t ore	$/t waste	$/t	$/t ore	$/t waste	$/t	$/t	$/t ore	$/t ore	$/t ore	$/t ore	$/t ore	$/t ore	$/t ore
Oxide	**19.56**	**33.70**	**14.14**	**1.10**	**1.05**	**0.00**	**0.56**	**0.57**	**0.35**	**0.12**	**0.23**	**0.60**	**0.19**	**9.23**	**1.92**	**0.21**	**0.66**
Sediment	19.46	33.75	14.29	1.10	1.05	0.00	0.56	0.57	0.35	0.12	0.23	0.60	0.19	9.23	1.92	0.21	0.66
Olistostrome	19.02	33.23	14.21	1.14	1.06	0.00	0.56	0.57	0.35	0.12	0.25	0.60	0.19	9.24	1.90	0.21	0.66
Basement	108.33	33.73	-74.61	1.37	1.02	0.00	0.68	0.57	0.35	0.12	0.21	0.60	0.19	9.33	1.93	0.25	0.81
Transitional	**21.47**	**37.90**	**16.43**	**1.03**	**0.99**	**0.00**	**0.84**	**0.71**	**0.35**	**0.12**	**0.22**	**0.60**	**0.19**	**10.70**	**1.92**	**0.23**	**0.75**
Sediment	21.21	38.55	17.34	1.02	0.99	0.00	0.84	0.71	0.35	0.12	0.22	0.60	0.19	10.70	1.93	0.24	0.76
Olistostrome	23.81	33.21	9.40	1.08	1.00	0.00	0.83	0.70	0.35	0.12	0.27	0.60	0.19	10.71	1.88	0.21	0.66
Basement	24.59	25.05	0.46	1.07	0.98	0.00	0.82	0.70	0.35	0.12	0.40	0.60	0.19	10.67	1.72	0.15	0.50
Sulphide	**22.40**	**49.44**	**27.04**	**0.99**	**0.94**	**0.02**	**1.17**	**1.05**	**0.35**	**0.12**	**0.07**	**0.60**	**0.19**	**9.78**	**2.10**	**0.31**	**0.98**
Sediment	21.75	50.46	28.72	0.99	0.94	0.02	1.17	1.06	0.35	0.12	0.07	0.60	0.19	9.78	2.11	0.31	1.00
Olistostrome	25.68	40.64	14.97	0.99	0.93	0.02	1.16	1.03	0.35	0.12	0.12	0.60	0.19	9.78	2.04	0.25	0.81
Basement	21.23	61.57	40.34	1.02	0.95	0.03	1.18	1.08	0.35	0.12	0.02	0.60	0.19	9.78	2.16	0.38	1.22
TOTAL	**21.81**	**45.17**	**23.36**	**1.01**	**0.96**	**0.01**	**1.02**	**0.93**	**0.35**	**0.12**	**0.12**	**0.60**	**0.19**	**9.84**	**2.04**	**0.28**	**0.89**
Sediment	21.25	45.45	24.20	1.01	0.96	0.01	1.01	0.91	0.35	0.12	0.12	0.60	0.19	9.84	2.04	0.28	0.90
Olistostrome	24.84	39.25	14.41	1.01	0.94	0.02	1.07	0.98	0.35	0.12	0.14	0.60	0.19	9.80	2.02	0.24	0.78
Basement	21.91	59.42	37.51	1.02	0.96	0.02	1.15	0.98	0.35	0.12	0.04	0.60	0.19	9.82	2.13	0.37	1.18

15.4 Mineral Reserves

The Rozino Gold Project supports an economic open pit mining operation. The Mineral Reserve estimate is based on the Indicated category of the Mineral Resource contained within the pit design. The Mineral Reserve estimate has considered all modifying factors appropriate to the Rozino Gold Project.

The reference point at which the Mineral Reserves are defined is where the ore is delivered to the Flotation Plant.

Table 15-15: Mineral Reserve for the Rozino Gold Project (effective Date 30 August 2020).

Ore type	Reserve category	Tonnes (Mt)	Gold grade (g/t)	Contained metal (koz gold)	Metallurgical recovery (%)	Recoverable metal (koz gold)
Oxide	Probable	1.9	1.07	64	67.4	43
Transitional	Probable	1.8	1.15	68	70.7	48
Sulphide	Probable	8.1	1.27	332	83.3	277
Total	**Probable**	**11.8**	**1.22**	**464**	**79.3**	**368**

Notes:
- *The Mineral Reserve disclosed herein has been estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum "CIM Definition Standards for Mineral Resources and Mineral Reserves" (CIM, 2014).*
- *Mineral Reserves discard cut-off grade is 0.5 g/t gold.*
- *Mineral Reserves are based on a $1,500 per oz gold price.*
- *Mineral Reserves account for mining dilution and ore loss.*
- *Probable Mineral Reserves are estimated from Indicated Mineral Resources.*
- *Sum of individual values may not equal due to rounding.*

No inferred Mineral Resources are included in the Mineral Reserves. Inferred Mineral Resources do not contribute to the financial performance of the project and are treated in the same way as waste.

Mining losses and mining dilution are incorporated in the MIK Mineral Resource estimate. CSA Global were able to determine that mineralization can be adequately modelled for its diluted, recoverable grade assuming a selective mining unit (SMU) of 4 x 6 x 2.5 m using the MIK methodology. CSA Global consider that the Mineral Resources can be effectively mined by open cut extraction using the selected mining equipment and qualifications relating to training, grade control practices, and drilling and blasting technique, without additional dilution and loss factors being applied.

There is no known likely value of the following factors of mining metallurgical, infrastructure, permitting or other relevant factor that could materially affect the estimate. It is important to note that permitting for the Rozino Gold Project is not complete. Velocity has initiated the environmental and social impact assessment process, including the permitting procedures to meet Bulgarian regulations and is gathering environmental data to improve the design of the Project. Under the Bulgarian Environment Protection Act, the development of an economically viable mining reserve requires an Environmental Impact Assessment EIA which complies with European Union environmental regulations. The prospecting licence agreement for the Tintyava Property has been signed with the Minister of Energy and exploration activities have been approved by the Ministry of Environment. All necessary permits to conduct the work proposed for the property have been obtained and there are no known significant factors or risks that may affect access, title or the right or ability to perform work on the Property. There are currently no objections to the development of the Project.

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Areas of uncertainty that may materially impact mineral reserve estimation include but are not limited to:

- Commodity price and exchange rate assumptions

- Mining recoverability assumptions
- Capital and operating cost estimates
- Geotechnical slope designs for pit walls.

16 Mining Methods

16.1 Introduction

The Rozino Gold Project comprises a single open pit utilizing standard mining equipment to selectively extract the ore over a nine year period (inclusive of a two year pre-stripping period). The pit will be mined in two phases utilizing a waste rock dumping and backfill strategy that provides a compact footprint. The mining is at extraction rates of up to 22,000 tpd using 90 tonne class excavators and 55 tonne capacity haul trucks. All material will require blasting. The schedule delivers higher grade ore directly to the plant in the initial five year period of the mine life at 5,000 tpd. Low-grade ore is stockpiled during this period and reclaimed for processing when the pit is exhausted. The resulting tailings from the low-grade ore are stored in the pit.

The mine plan has a pre-stripping period that is carefully balanced to produce waste rock for the construction of the water storage dam walls, the TMF embankment and ore stockpile pads. The pre-stripping period also delivers a small amount of ore that will enable the plant commissioning and start-up. Although the pre-stripping is over a two-year period, the first year entails mostly road building, clearing, grubbing and topsoil removal.

16.2 Mineral Reserve

Section 15 details the Mineral Reserve estimation, pit optimization, pit design, mining dilution, ore loss and pit geotechnical information.

16.2.1 Material Physical Properties

The material physical properties and assumptions used in the estimation of mining equipment requirements and the mine plan are summarized in Table 16-1. These material properties apply equally to ore and waste.

Table 16-1: Material Physical Properties

Parameters	Units	Value
Dry bulk density (ore and waste):		
- Oxide	t/m^3	2.31
- Transitional	t/m^3	2.41
- Sulphide	t/m^3	2.58
Moisture content	%	5%
Swell factor	%	50%
Compaction factor	%	10%
Ore mining loss	%	0%
Ore dilution	%	0%

Note: It is considered that no further dilution or mining loss are applicable in consideration of the equipment and selective mining method being applied to the MIK resource estimate. Please refer to deeper discussions in the Mineral Resource and Mineral Reserve sections.

16.2.2 Site Water Management

Site water management is central to maintaining an appropriate environmental and operational performance for the Project. The principle adopted for site water management is to intercept and control contact water flowing within the operational areas and direct it to the contact water dam (CWD) for re-use. The site water balance indicates that the Project will have a negative water balance. Water reuse will be maximized, but plant process make-up water will need to be sourced from external, local water sources. This water will be pumped to a raw water dam (RWD) directly below the contact water dam, and then pumped to water storage tanks at the processing facility.

Golder Associates (UK) ("Golder") were engaged by Velocity to advise on surface and groundwater monitoring requirements and to develop a surface water and groundwater model, generate a site-wide water balance, and design the water storage dams and the TMF to support the PEA and NI43-101 Technical Report.

16.2.3 Hydrogeology

Golder undertook a hydrogeological study to develop a conceptual site model; make preliminary assessments of inflows, pore pressures and dewatering design in relation to the open pit development and provide inputs to the site water balance; assess the potential seepages in relation to the waste and water storage facilities; and assess the viability of water supply from wells adjacent o the pit (in the Palaeogene rocks) and qualitatively assess the groundwater quality.

Based on the hydrogeological conceptual site model developed from available information and site investigation, Rozino is located in an area dominated by underlying hydrogeological units with moderate to low permeability. Bulk hydraulic conductivities for the various rock types within the pit area vary from 1X10-8 to 2X10-8 m/s for basement gneiss and 2X10-8 to 5X10-8 for the sedimentary and olistostrome rocks.

Groundwater depth and flow direction is generally controlled by topography with groundwater flowing predominantly to the southeast. Depth to groundwater across the site ranges from 20 to 30 m below ground level in the open pit area. Estimated groundwater inflows to the pit will occur at up to 300,000 m^3/a over mine life.

16.2.4 Pit Dewatering and Depressurization

Pit wall depressurization requirements are generally determined based on geotechnical objectives to achieve target pore water pressures within the pit walls. Depressurization requirements are usually informed based on output from groundwater models of geotechnically defined critical sections of the pit wall, often associated with identified geotechnical weaknesses.

To de-pressurize the pit southwestern wall, two electric submersible pumps are planned for installation within dewatering boreholes once the Phase 2 (west pit) is developed below the 435 masl elevation. The pumps will discharge into an external pit network of drainage channels that will direct all mine water to the contact water dam. The exact location of the wells and even their need will be assessed with actual data from monitoring of the pit water in-flows and detailed geotechnical observations prior to the pits reaching the 435 masl.

16.2.5 Pit Inflows

The two sources of water inflows into the proposed pits are:

- Groundwater inflows – once the pits extend below the local water table groundwater inflows will occur through the bulk rock-mass and any permeable structures intercepted by the pits.
- Surface water (direct precipitation and runoff) inflows – incident rainfall which falls within the pit footprints (and does not infiltrate into the ground) and any surrounding surface catchments which are not prevented from draining to the pits.

Both groundwater and surface water pit inflows are dependent on the dimensions of the pit as it develops.

The surface water handling plan will be based on diverting as much surface water as possible away from the open pits, using ditches and sumps and directing it to the contact water management dam by gravity. Within the pit area the water will be directed to the external system of ditches and sumps (when the pits are above 435 masl) or to internal sumps and then pumped to the same external network. As the pit deepens, intermediate sumps and pumps will be required.

As the mine pit progresses, dedicated high-lift pumps will eventually be required. Pontoon-mounted pumps will be used to draw from a Phase 1 pit bottom sump. This will ensure the pumps are not submerged when sump water levels rise rapidly in the event of to a rainfall event. Pumping infrastructure will maintain pace with mining as the pit expands and deepens. The key operational requirements will be to minimize water flows into the pit using perimeter bunds, drains and fill by:

- Providing pit pumping capacity for foreseeable extreme events
- Maintaining pit wall drainage
- Providing permanent and temporary sumps capable of handling the expected water inflows
- Groundwater Inflows.

The permeability of the rocks in the Rozino Project are generally low and therefore pit inflows from the general rock mass are correspondingly low. Fault and fracture zones which intersect the pit give an elevated permeability in some locations and act as preferential groundwater flow paths and may result in localized zones of high groundwater inflows. The duration of enhanced inflows via any faults or fractures intercepted will be dependent on the hydraulic continuity of these zones and may also be seasonal, with increased flows in the winter (wet season).

Hydrological modelling by Golder has anticipated that the pit groundwater inflow is 415 m^3/day (~5 l/s) for the Phase 2 pit (west pit) and 80 m^3/day (~1 l/s) for the Phase 1 pit (east pit), when each are at maximum depth.

The pit dewatering requirements are aligned with the production schedule and assumes that no pumping of groundwater will be required until the bench mining has progressed below the 435 masl elevation as any groundwater inflows will drain into the surface cut-off channels and ditches, which will ultimately be directed to the contact water system.

16.2.6 Surface Water

Similar to groundwater inflows, rainfall runoff will drain naturally into diversion ditches until the bench mining has progressed below the 435 masl elevation. Water accumulation from rainfall and rainfall runoff is adjusted on a quarterly basis for average seasonal rainfall and based on the area of pit exposed in each phase.

Once mining reaches the 435 masl elevation in each of the phases, groundwater and rainfall volumes are calculated on a per period (quarterly) basis and the pumping requirements determined.

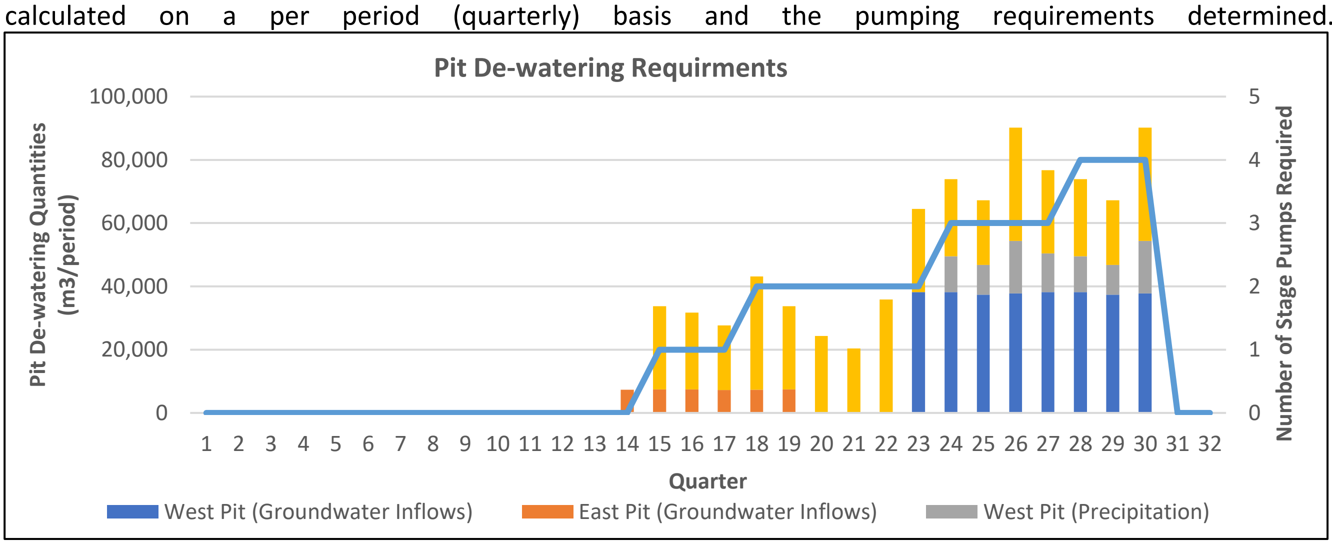


Figure 16-1 illustrates the expected dewatering quantities per period (groundwater inflow and precipitation) once each pit phase has mined below the 435 masl elevation (point at which natural drainage will no longer occur). On Figure 16-1, the x-axis commences with Period 1 being the start of construction.

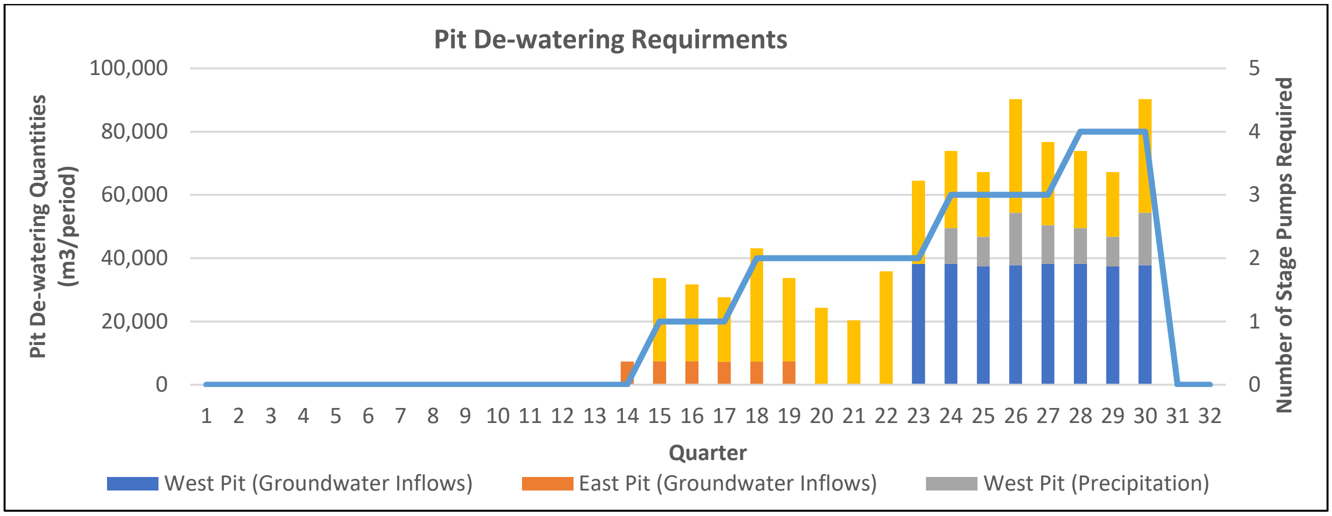


Figure 16-1: Estimated pit de-watering requirements per quarter

The pit dewatering systems will be driven by a combination of groundwater and surface water inflows and the water management design considers both elements as groundwater inflows will continue even during extended dry periods and may be locally significant in areas of enhanced permeability.

Pit dewatering will be achieved principally by utilising an in-pit sump dewatering system capturing all groundwater and surface water pit inflows. These inflows will gravity drain to an in-pit sump(s) at the base of the pit and will be subsequently pumped out of the pit to a sediment treatment system. Additional dewatering is planned with the installation of two electric submersible pumps within dewatering bores once the Phase 2 (west pit) mining reached the 435 masl elevation. The aim of these pumps is to de-pressurize the southwestern pit wall

The Phase 2, west pit has the largest water inflows at 415 m^3/day once the pit is mined below the 435 masl elevation and 80 m^3/day for the Phase 1 pit (east pit) and each at maximum depth. This volume is not large in the context managing of surface water inflows to a pit and the 24-hour 1-in-100 year return period event volume could be dewatered in approximately three days using industry-standard dewatering pumps.

16.3 Waste Dump Design

The location of the waste rock dump (WRD) external to the pit is proposed to be within the valley catchment area that contains the tailings management facility (TMF) and the water storage dams. The initial WRD is in close proximity to the pit exit located to the east of the pit. The WRD will be constructed in 20-30 m layers commencing on the western slopes of the Uren dere valley. Additionally, waste will be stored within the Phase 1 pit when it is complete. This is an important strategy for both the reduction of waste haulage costs and the minimization of the affected land area. Pit waste rock will also be utilized to construct the TMF embankment wall and the water storage dam walls. Pit waste rock will also be used to construct the high-grade ore stockpile base.

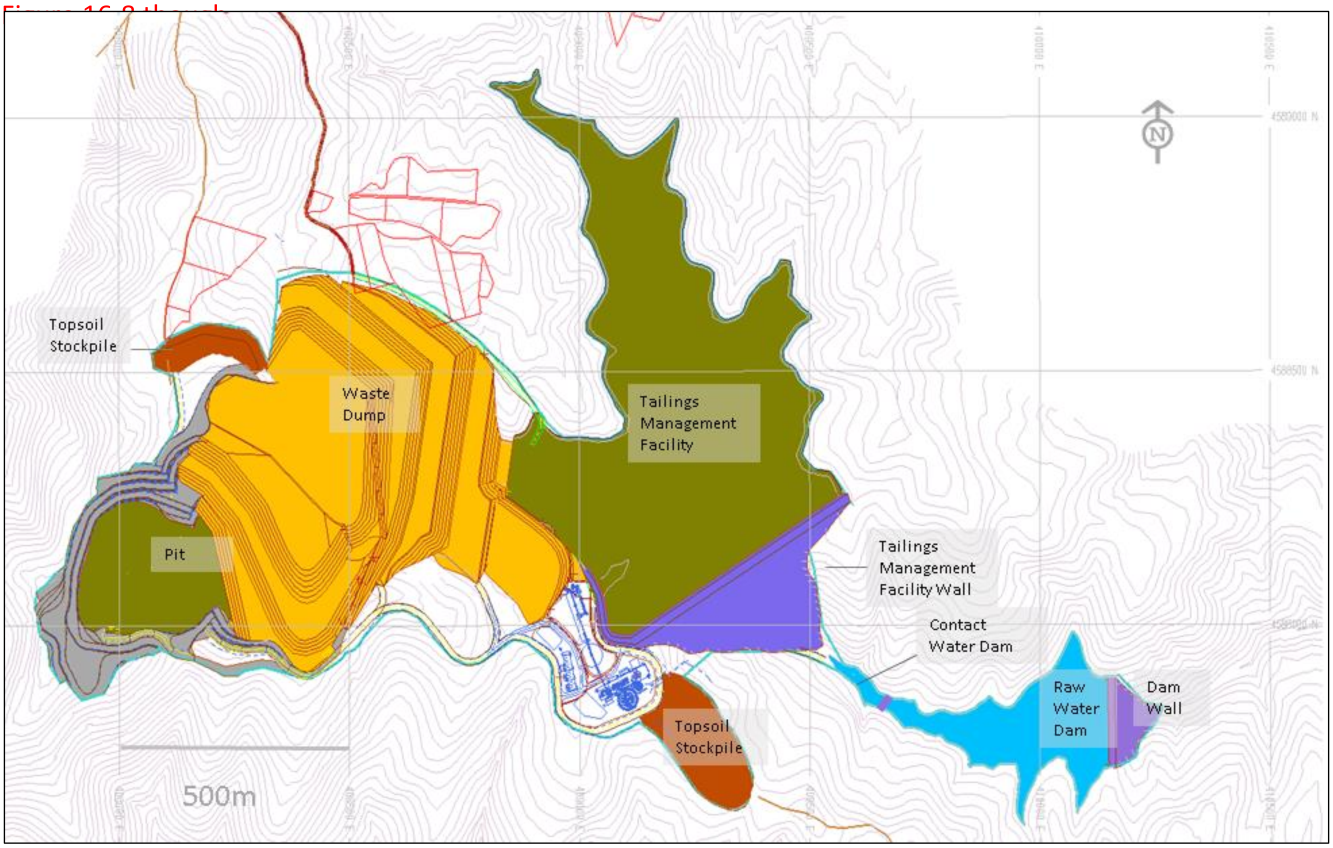


Figure 16-16 show the progressive development of the WRD over the life of the operation.

Low-grade ore will be stockpiled on the eastern side of the WRD at the same overall slope profile of 20°. Figure 16-2 shows a typical cross-section basis of design. Sections 16.7.1 to 16.7.9 illustrate the progression of the waste dump and low-grade stockpiles with a set of end-of-year graphical depictions.

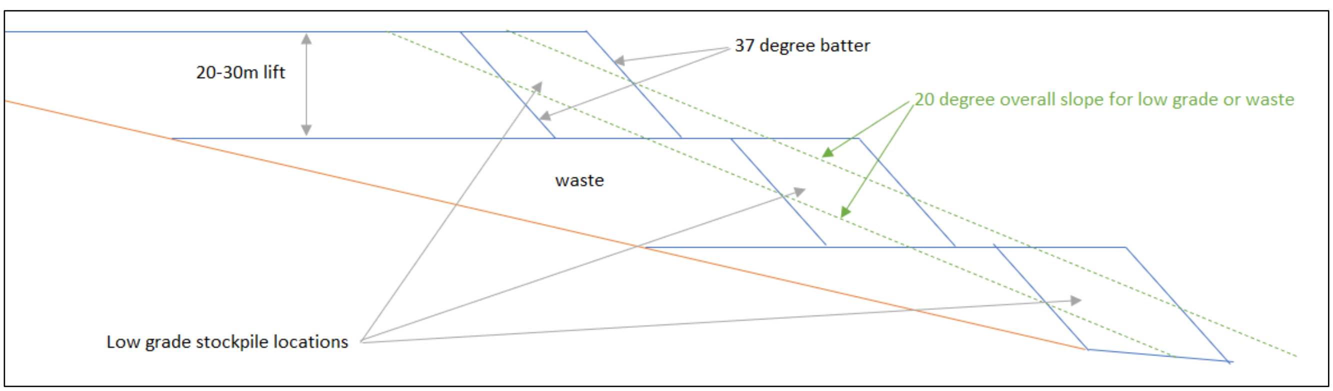


Figure 16-2: Schematic cross section of typical WRD and low-grade stockpile construction

Acid-base accounting testwork under the supervision of Mineesia Ltd indicates that the waste rock is neither acid generating nor is it anticipated that metal leaching will occur. Both the ore and the waste rock have low sulphide contents and relatively high neutralizing potential, resulting in net neutralization ratios of > 2. Metal leaching testwork results suggest that any leachate from ore and waste rock is not considered harmful to receiving waters and will therefore meet the criteria for disposal as inert waste. Consequently, no special considerations relating to acid rock drainage (ARD) controls have been implemented in the design of the waste dumps. It should be noted that this assessment is preliminary; geochemical characteristics of the rock may change in the long term as neutralization compounds and site-specific conditions change.

16.4 Expected Life of Mine

The life of the Rozino Project is nine years, including two years of construction activity. The two-year construction period includes limited ore mining. The final two years of mining is focussed on the reclaiming and processing of the low-grade ore stockpile. The operating life-of-mine (LOM) is seven years.

16.5 Mining Methods

Mining will be by conventional open pit drill and blast followed by load and haul. The mining cycle commences with a grade control process that will utilize reverse circulation (RC) drilling on a 5 x 8 m pattern over 16 m of depth. The results of the grade control evaluation will be used to define ore block geometries leading to ore / waste definition, appropriate blasting patterns and mine sequencing. Drilling and blasting will be performed on 5 m benches. The ore and waste, although blasted on 5 m benches, will be extracted on 2.5 m flitches. The equipment selected is of a type that will enable selective mining of the ore. For this, the ore bucket of the excavator is half the width of the SMU and the truck size allows for three full truck- loads per SMU. The excavator and truck type are matched for swing reach, height, and optimal passes per truck load.

The sequencing of grade control, blasting, water control and excavation will require expert planning and operations co-ordination. It will be imperative that the mining team are focussed to grade delivery to the plant whilst maintaining all other mining metrics.

The compact nature of the operation means that an automated truck dispatch system will not be required. Management of hauling from source to destination will be via a centralized control room equipped with computer equipment, telephone network and CB Radio TX/RX system.

16.5.1 Grade Control

The grade control cost estimate and planning application is based on RC drilling on a 5 m x 8 m pattern and taking samples at 1 m intervals from a 5 ¼" drillhole. The 5 m x 8 m pattern is based on benchmark data for similar sized SMUs. Drillholes are planned to be 16 m deep (to give a one m overlap to the adjacent lower pattern and to avoid taking samples in the broken ore at the bench floor). Grade control activities will ensure that each 5 m mining bench has some areas in grade control drilling whilst others are available for mining. The PFS assumes that ore and waste are intermingled but that the grade control drilling volume can be limited to a 1 : 1 waste : ore ratio, i.e. there will be some areas of the pit (dominated by waste) that will have no grade control drilling.

Because RC drilling is a specialized activity, a contractor will provide the service. A separate maintenance and warehouse area is provided. Approximately 45,000 to 55,000 m of RC drilling will be required per annum, thus requiring a single rig operating a on single shift, 5 days per week to maintain a schedule to support planning and scheduling. Each drillhole will be logged geologically, and samples will be sent off-site for assay. Sampling and interpretation of results will be conducted by suitably trained Velocity samplers and geologists. The geologist will also complete other services, including geotechnical evaluations and geological face mapping. This information will aid pit wall construction, geometallurgical forecasting and mine planning.

16.5.2 Drill and Blast

Rock fragmentation will be accomplished through drilling and blasting. There may be small amounts of free-dig or ripable material at the surface in the Oxide material, but this was not assumed in any planning considerations. Oxide material constitutes less than 15% of the total material. Based on core logging reviews and inspection of rock outcrops and road cuttings only small amounts of the Oxide will not require blasting.

For drill and blast operations it is envisaged that a contractor supplies drilling and blasthole loading equipment, light vehicles, consumables, and management oversight. Explosives will be supplied on a just-in-time basis by the contractor. No Rozino employee will be required to hold a blasting licence, nor will there be an explosive storage or explosive magazine on site. Drill and blast designs will be completed by the contractor using grade control information and under the supervision of company engineering planners.

Drill and blast design parameters (powder factor, burden and spacing) will be mostly controlled by the degree of weathering and the requirement for finer particle sizes in ore. Six drill patterns were defined in the PFS mine planning as a function of weathering and ore sizing.

The industry-accepted Kuzram blast fragmentation prediction model was informed by local geotechnical data (obtained by Golder, 2020) to design the blast patterns. The specifics of the blast pattern design are presented in Table 16-2.

Table 16-2: Proposed blast pattern design for various material types

Blasting parameters	Units	Ore			Waste		
		Oxide	Transitional	Sulphide	Oxide	Transitional	Sulphide
Bench height	m	5	5	5	5	5	5
Hole diameter	mm	76.2	76.2	76.2	76.2	76.2	76.2
Powder factor	kg/bcm	0.23	0.39	0.47	0.23	0.23	0.39
Powder factor	kg/t	0.10	0.16	0.18	0.10	0.10	0.15
Spacing	m	4.00	3.10	2.80	4.00	4.00	3.10
Burden	m	2.80	2.80	2.50	2.80	2.80	2.80
Stemming length	m	1.9	1.9	1.9	1.9	1.9	1.9
Sub-Drill	m	0.42	0.42	0.38	0.42	0.42	0.42
Explosive per hole	kg	12.8	16.8	16.6	12.8	12.8	16.8
Explosive	type	ANFO	E3000	E3000	ANFO	ANFO	E3000
P_{80} Particle size	mm	152	445	628	152	631	744
Percent passing 0.5 m	%	100%	84%	71%	100%	71%	65%

Ore and waste will have complex geometric intermingling that will result in the need to blast some ore and waste together. CSA Global estimate that the ore pattern will be applied to equal amounts of ore and waste. This requires review when more detailed ore definition data from grade control is available. Figure 16-3 and Figure 16-4 below illustrate the estimated particle size distributions for the various material types.

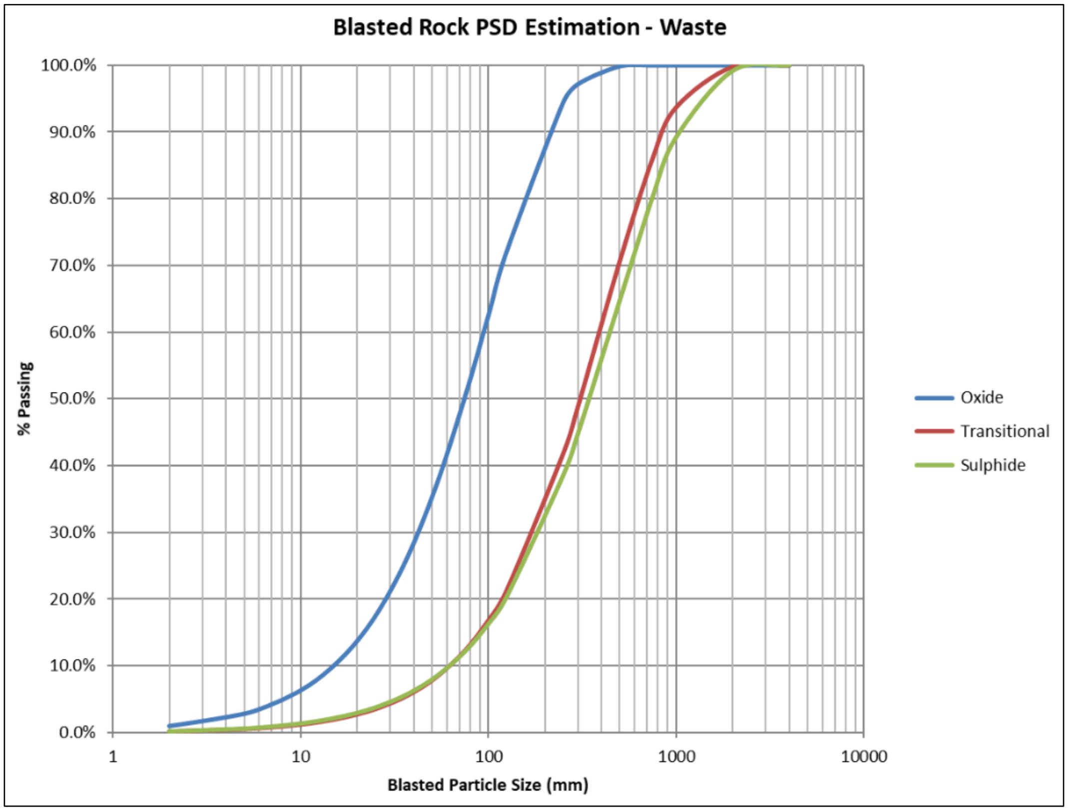


Figure 16-3: Rozino blasted rock PSD for ore

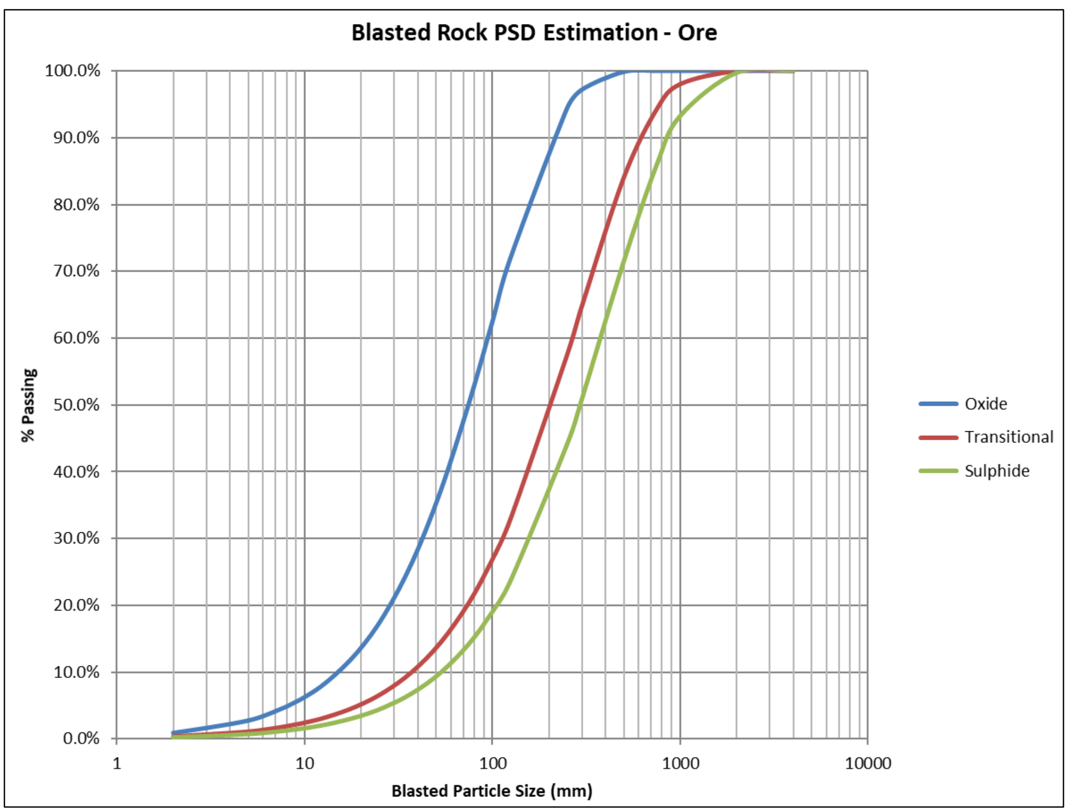


Figure 16-4: Rozino blasted rock PSD for waste

It is estimated that around 15% of the Sulphide ore and 7% of the Transitional ore may require secondary fragmentation to allow feed to the jaw crusher. This figure will be reduced by breakage induced during loading and hauling that is estimated to enable a manageably low proportion requiring secondary rock breaking at the plant tip.

Pre-split blasting requirements were estimated on the basis of the final pit wall perimeter and pit volumes as designed in the PFS. There will be a requirement of 40,309 linear metres (in 5 m benches) of pre-split blasting for all material categories (15.4 Mm^3) equating to 380.5 bcm per linear pre-split metre. Pre-split of the final pit walls will be achieved through the drilling of 76 mm holes with a spacing of one m and a 50% decoupling ratio using emulsion or emulsion-packaged explosives. It is recommended the pre-splits are fired in sequence with a 10 to 15 m buffer blast at the final wall position.

Three top-hammer blasthole drill rigs capable of drilling 76 mm holes for main production blasting were selected. The drill rig will cover a range of drillholes that will allow other configurations of blasting patterns to be applied if required. To allow for pre-split coverage the carousel will allow a drilling depth of 31 m to be achieved. The planned annual productive hours for the drill rigs were estimated at approximately 6,178 hr/yr. At peak production three drill rigs will be required.

The drill and blast operation will be supported by a stemming loader.

16.5.3 Ore and Waste Loading

Loading of ore and waste into a 55 tonne class modified rock-body Volvo FMX diesel haul truck will be performed using a 90 tonne class hydraulic excavator. The CAT 390 hydraulic excavator was selected for this service. The excavator's instantaneous production rate of approximately 672 LCM/hr (loose cubic metres), bucket capacity of 5.7 LCM, and six passes per truck load, requires an annual productivity of approximately 5,146 hr/yr. At peak production three excavators will be required. The bucket size in relation to the truck and the SMU are important considerations in the selective mining requirements of the operation.

16.5.4 Ore and Waste Hauling

The haul truck productivity is based on "queuing theory" that discounts truck and excavator system productivity for hauling time, loading time, and the number of trucks in the system. The purpose of the "queuing theory" is to determine the lowest unit operating cost for a truck and excavator system but constrained by system productivity and optimal truck numbers. At a bowl capacity of 33 LCM and a 95% fill factor, planned annual productive hours of approximately 5,065 hr/yr are achieved with a truck fleet of mostly 12 units. There are times where higher and lower truck numbers are required, mostly when in-pit dumping is available (lower numbers) or the need for waste overhaul to the TMF (higher numbers). The scheduling assumes that one or two additional trucks can be leased for short periods. However, detailed and short-term scheduling may reduce the need for more than 12 trucks.

The compact nature of the mine haulage road network does not necessitate an automated truck dispatch system. Management of hauling from source to destination will be via a centralized control room equipped with computer equipment, telephone network and CB Radio TX/RX system. The control room operator will receive a daily updated routing plan to be executed by directing haul trucks from the relevant material source to destination. The control room operator shall keep a tally of each journey for each haul truck and will coordinate re-fueling, maintenance and breakdown repair services as and when the need arises. Communication between control room operator and equipment operator will be via CB radio with each type and piece of equipment designation a call-sign/equipment number. Data recorded by the control room operator will be communicated to the technical services department daily for analysis, management reporting and feedback into a continual improvement programme.

16.5.5 Ancillary Equipment

Dumping and bench clear-up activities will be undertaken by a D8 dozer and a Cat 980 front-end loader operating day shift only, but on an as-needed basis outside of dayshift as well. The loader's main duty will be for ore rehandling at the ROM stockpile area.

A water bowser will have variable utilization during the year, from almost none during the wet winter months to two shifts per day in the dry summer months.

A fuel truck will service all equipment not able to drive to the plant area for refueling.

A Cat 150 grader will maintain all roads on a dayshift or as-needed basis.

Minor ancillary equipment includes:

- A Cat 325 excavator mounted with a hammer for rock breakage and pit wall cleaning duties
- two 65 seat buses for personnel transport
- One 80 t low loader
- Four lighting plants
- Six light vehicles for mine operations and maintenance (other vehicles are covered in administration and mine administration)
- One tyre handler
- Two submersible sump pumps
- Three high-head stage pumps
- One compactor for road construction and maintenance.

16.5.6 Mine Management

Mine management will cover all duties required for operational and technical leadership of the mine. The manning will include a full-time senior mining engineer, mining engineer, surveyor, and a geologist, and their respective assistants. They will utilize a range of computer programs to conduct technical evaluations and planning as required.

16.6 Pit Production Schedule

Production schedules and haul route information was derived from the mine planning work undertaken as part of the PFS study. The overall goal of the schedule is to deliver 1.75 Mtpa of ore to the Flotation Plant. Based on results of the metallurgical testing, ore type proportions in plant feed have no detrimental effect on recovery performance and also do not affect throughput rate. However, this detail requires more detailed analysis and testing the FS.

The Flotation Plant feed schedule is based on a processing nameplate capacity of 1.75 Mtpa with wet commissioning commencing at the start of Year 1 and ramp-up to full production completed within the first year (65% Q1, 85% Q2, 95% Q3, 99% Q4). Figure 16-5 illustrates the process feed tonnages and expected gold feed into the Flotation Plant respectively (DF denotes "direct feed from pit" and SP denotes "stockpile subtractions"). Note the quarterly head grade profile is between 1.2 and 1.4 g/t Au over the mining life; this reduces when the deeper portion of Phase 2 pit is reached, and the low-grade ore stockpiled is reclaimed at the end of mine life.

The schedule also integrates the requirements for pre-production waste rock (for construction), mining two pit phases to manage the waste dumping and backfill strategy, managing equipment fleet variations, ensuring the various stockpiles have adequate space, and using a high-grade ore first/low-grade ore last strategy to improve investor return.

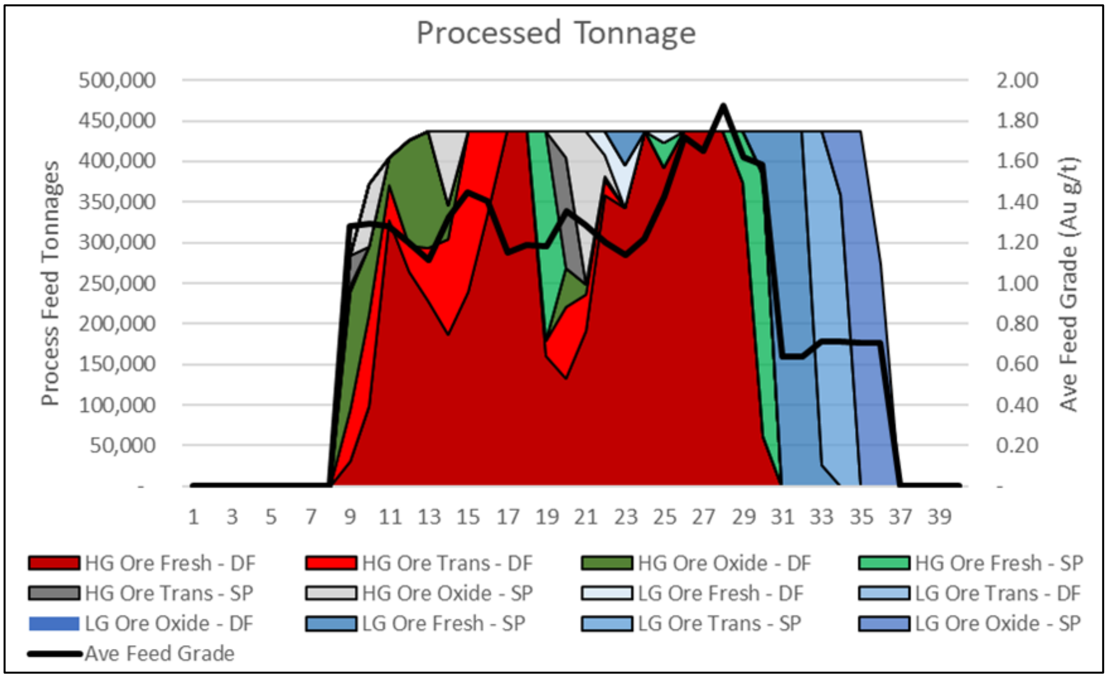


Figure 16-5: Process feed schedule per material type and source per quarter

Figure 16-6 illustrates the planned LOM production profile per quarter for the open pit operations from the commencement of construction. Both pit Phases 1 and 2 are mined during the construction period to maximize waste extraction. Subsequently and at the start of the operations, the Phase 1 pit operates for about two years and continues building a low-grade ore stockpile. The Phase 2 pit then commences but production throughput is reliant on consuming high-grade ore stockpiled during Phase 1 due to a lack of ore in the first few benches. Once Phase 2 is complete, low-grade ore is rehandled into the Flotation Plant.

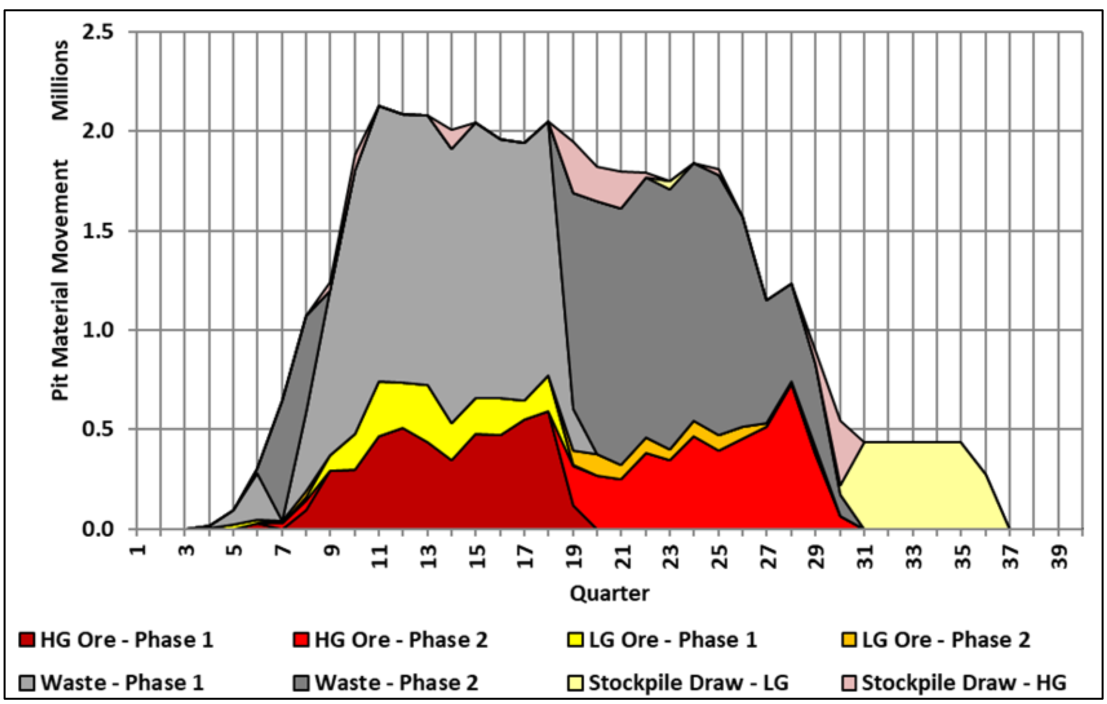


Figure 16-6: Rozino LOM production schedule

The mining schedule assumes that the operation works 24/7, 365 days in a year, less 15 days for unscheduled delays. Mining is scheduled for three eight-hour shifts per day with a rolling weekly fourth shift rest roster. Some members of the mining crew members are allocated to day shift only. Table 16-3details the overall available time for operations (hours per annum – hrpa), and hence equipment utilization.

Table 16-3: Equipment availability and utilization

Equipment Utilization	Units	Excavator	Haul truck	Drill and blast	Major ancillary
Equipment Non-Operating Time	min per shift	95	95	55	80
Un-planned Maintenance and Repairs	min per shift	20	20	0	20
Safety Meetings	min per shift	15	15	10	10
Meal and Anti-fatigue Breaks	min per shift	30	30	30	30
Shift Change	min per shift	30	30	15	20
Equipment Non-Operating Time	hr per annum	1,741	1,741	1,004	1,460
Operating Delays	min per shift	65	65	50	65
Warm up/idle down	min per shift	15	15	15	15
Refuelling/Minor Repairs	min per shift	30	30	20	30
Clean up	min per shift	20	20	15	20
Operating Delays	hr per annum	1,186	1,186	913	1,186
Total Annual Work Time Available	hr per annum	8,736	8,736	8,736	8,736
Total Time Available for Production	hr per annum	5,809	5,809	6,820	6,090
Mechanical Availability Note 1	%	88.2%	86.8%	90.4%	85.5%
Total Annual Time for Operations	**hr per annum**	**5,125**	**5,045**	**6,165**	**5,204**

Note 1 Mechanical Availability is based on weighted average of expected mechanical availability for the specific equipment's lifecycle.

Equipment requirements over the life of the Project are summarized in Table 16-4 and depicted graphically in

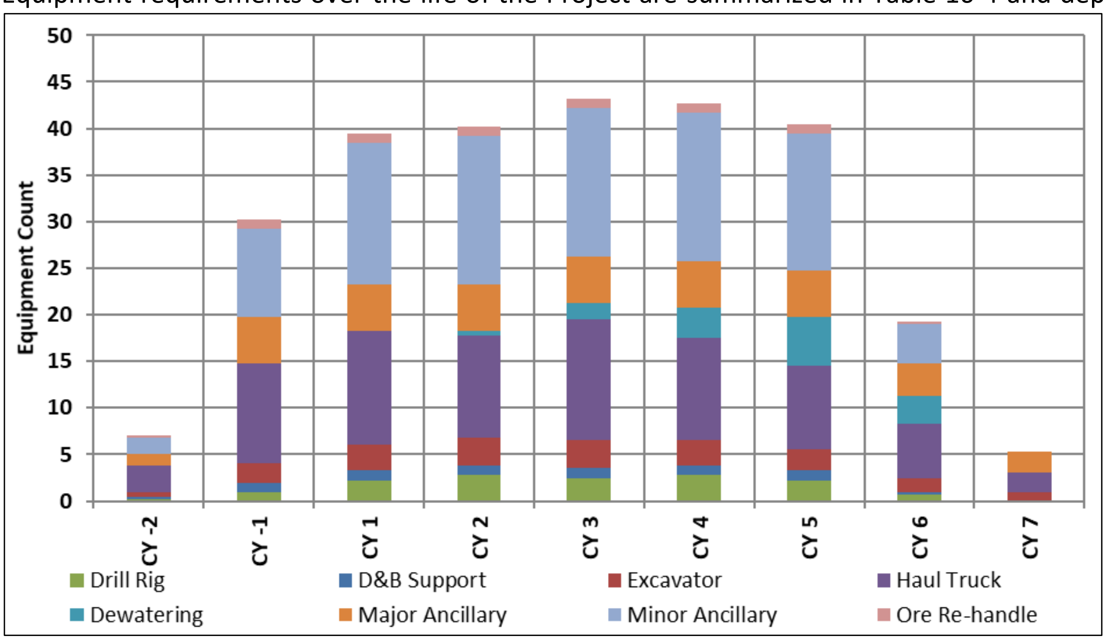


Figure 16-7.

Table 16-4: Rozino LOM equipment requirements

Equipment type	Peak	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Drill rig	3	0	1	2	3	3	3	2	1	0
Drill and blast support	1	0	1	1	1	1	1	1	0	0

Excavator	3	1	2	3	3	3	3	2	2	1
Haul truck	13	3	11	12	11	13	11	9	6	2
Dewatering	5	0	0	0	1	2	3	5	3	0
Major ancillary	5	1	5	5	5	5	5	5	4	2
Minor ancillary	16	2	10	15	16	16	16	15	4	0
Ore rehandle	1	0	1	1	1	1	1	1	0	0
Road maintenance	0	0	0	0	0	0	0	0	0	0
Machine count	**47**	**7**	**30**	**40**	**40**	**43**	**43**	**41**	**19**	**5**

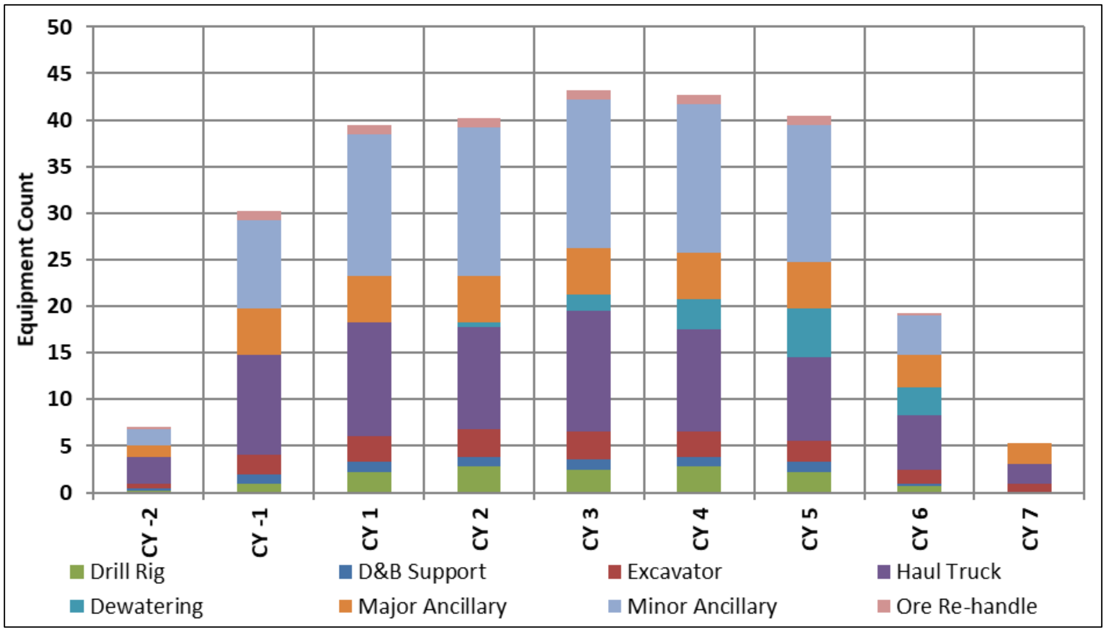
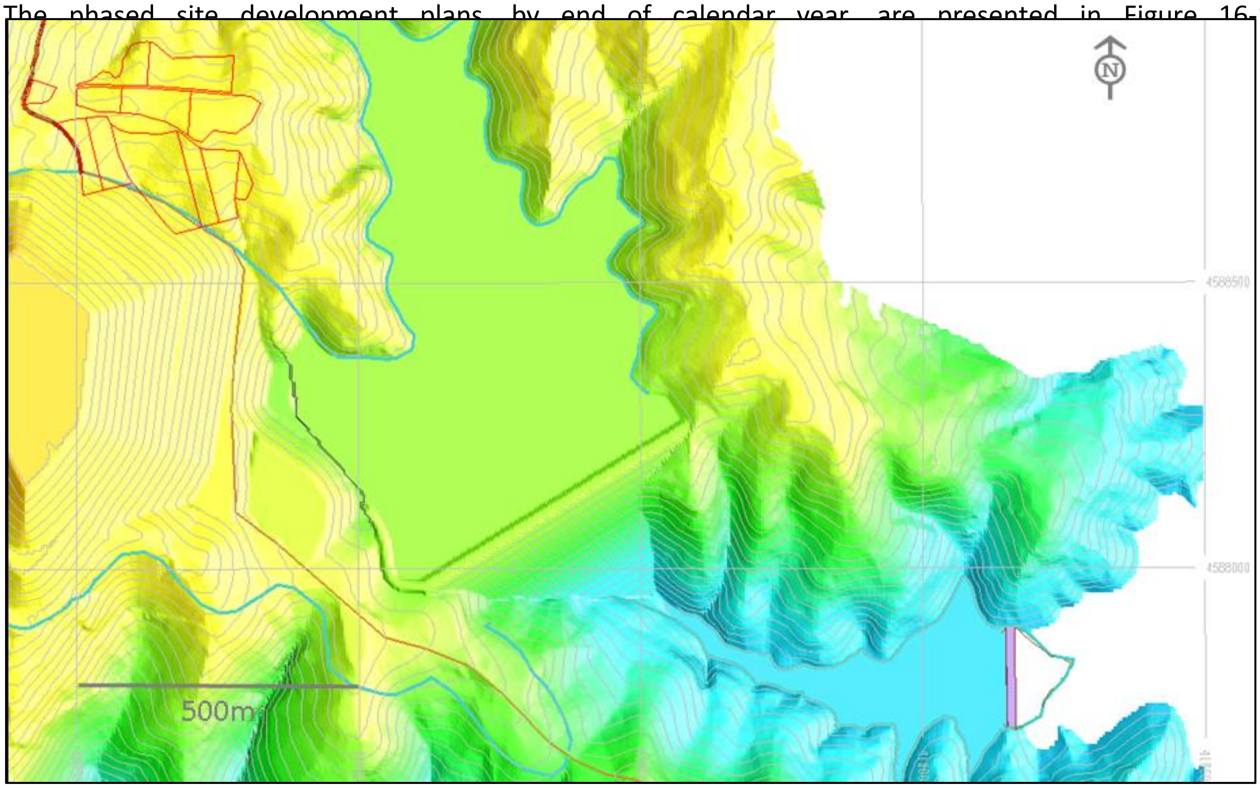


Figure 16-7: Rozino LOM required mining equipment (CY refers to Year)

The phased site development plans, by end of calendar year, are presented in Figure 16-10 to Figure 16-18. These plans start at the end of Year -2 and progress through to the final landform in Year 8.



16.6.1 Year -2 Activities

The first construction year is scheduled to include:

- Development of all roads and road upgrades (including the access road)
- Clearing, grubbing and topsoil removal for 45% of the project area
- Earthworks completion for the plant and commencement of foundation works
- Completion of half of the power line construction
- Installation of the diesel power generator for construction (later to be the operational back-up unit)
- Construction of the mine workshops
- Establishment of the mobile crushing plant (for TMF and RWD under-liner materials)
- Mining of the first bench in the pit to access waste rock
- Construction of the RWD.

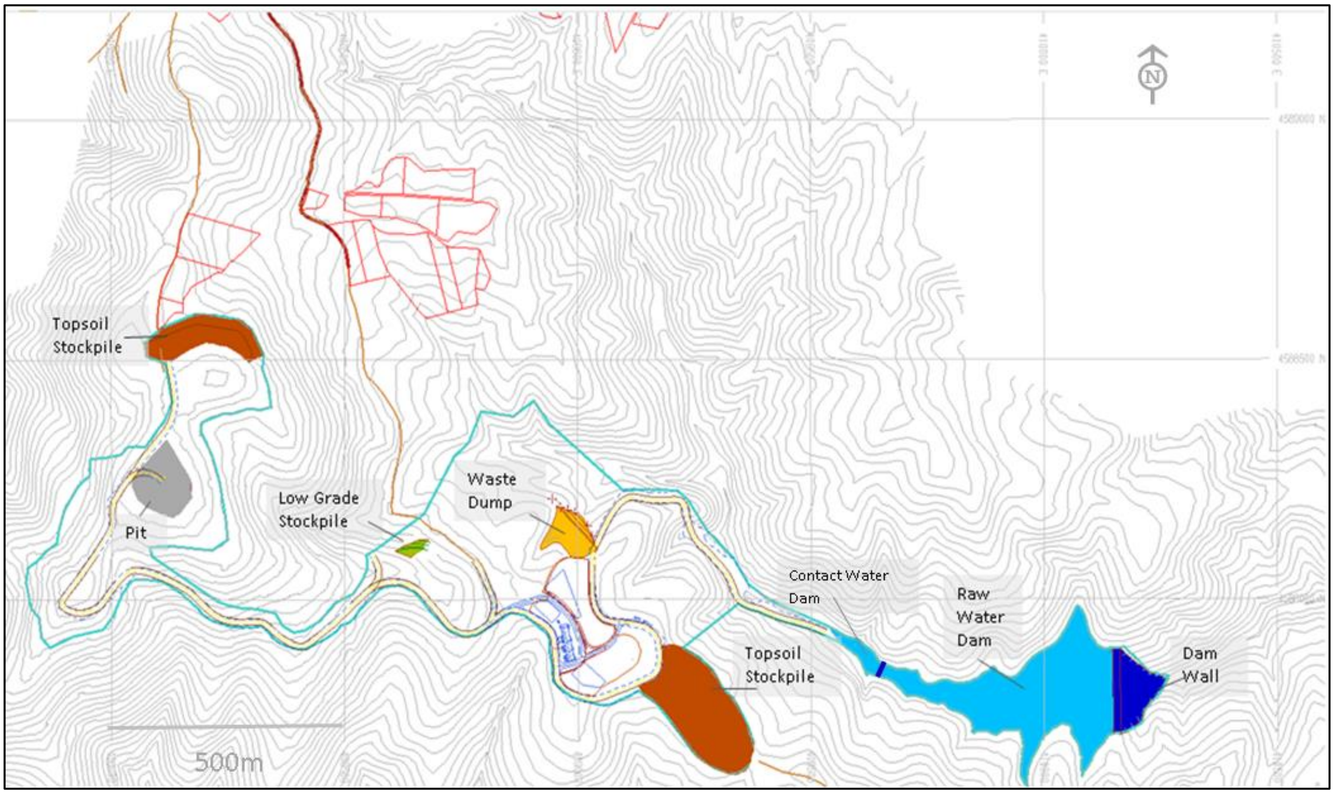


Figure 16-8: End of calendar Year -2 Rozino site development

16.6.2 Year -1 Activities

The second year of activities is scheduled to include:

- Clearing, grubbing and topsoil removal for 45% of the project area (now at about 90%)
- Completion of plant and office construction
- Completion of power line construction
- Construction of the TMF to allow commencement of ore treatment
- Mining in the pit, targeting primarily waste rock.

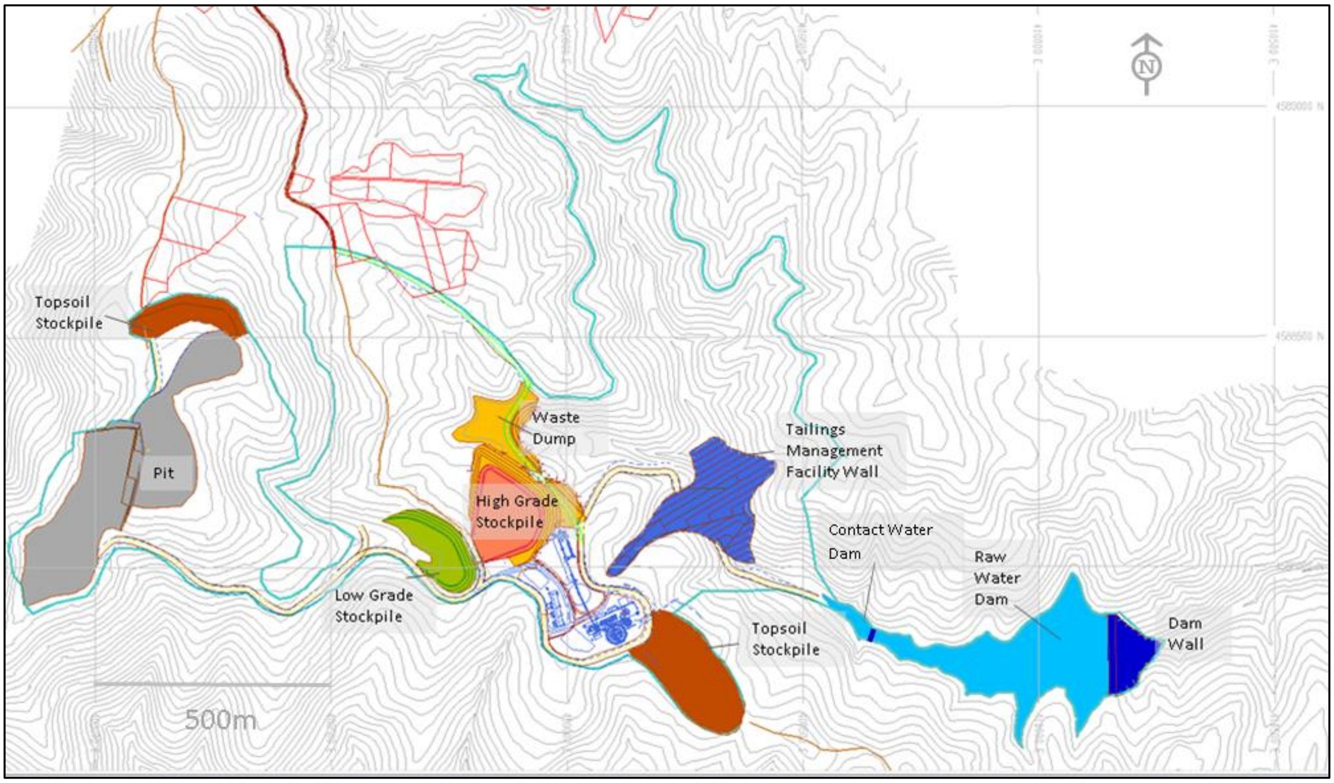


Figure 16-9: Figure end of Year -1 Rozino site development

16.6.3 Year 1 Activities

The first year of operational activities is scheduled to include:

- Clearing, grubbing and topsoil removal for 5% of the project area (now at 95%)
- Commissioning of the plant and ramp-up to full and steady state production throughput
- Construction of the TMF to allow two years of tailings capacity and the crushing of all fine material for the under-liner (using the mobile crushing plant which leaves site at the end of Year 1)
- Mining of the Phase 1 pit.

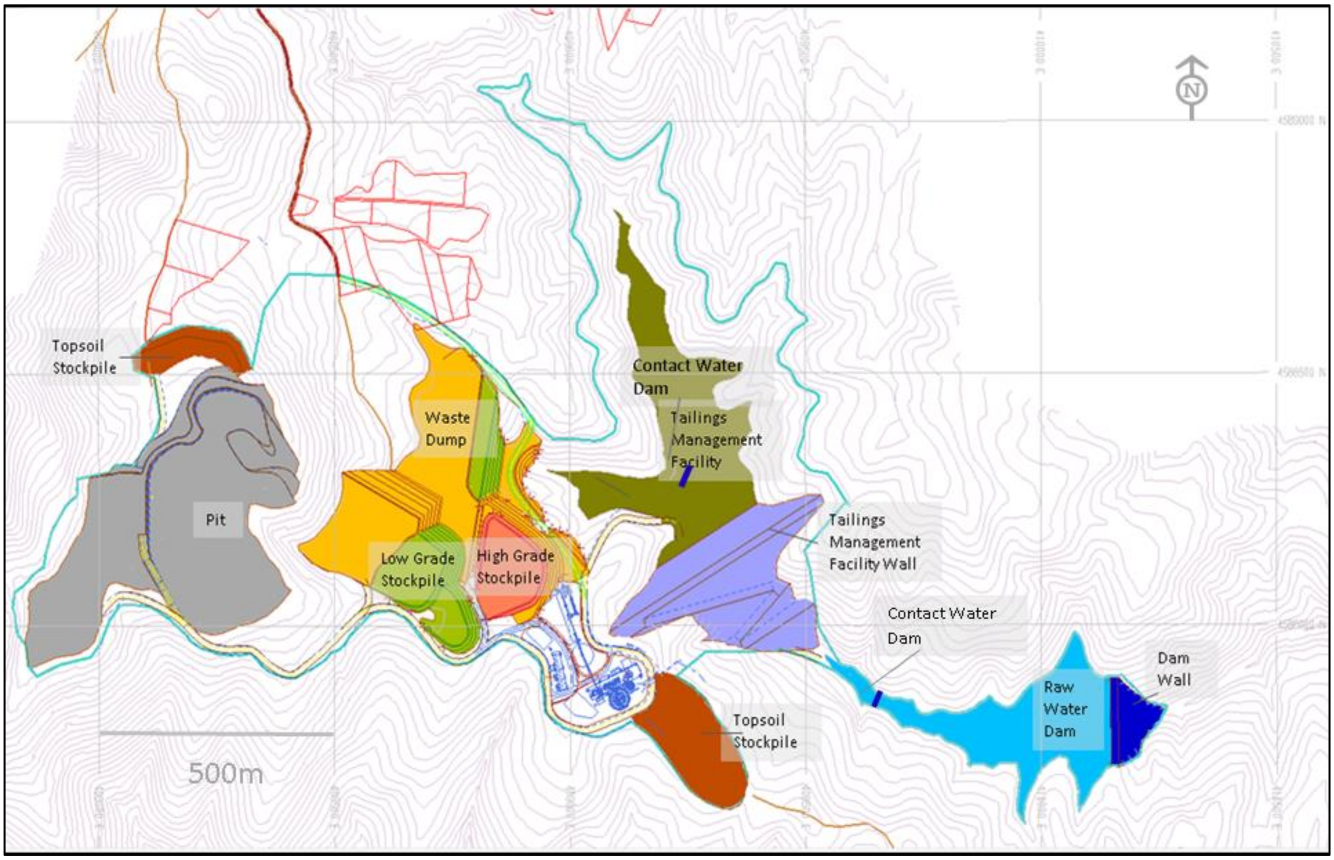


Figure 16-10: End of Year 1 Rozino site development

16.6.4 Year 2 Activities

The second year of operation is scheduled to include:

- Clearing, grubbing and topsoil removal for 5% of the project area (now at 100%)
- Mining the Phase 1 pit
- Commencement of some in-pit backfill with waste rock.

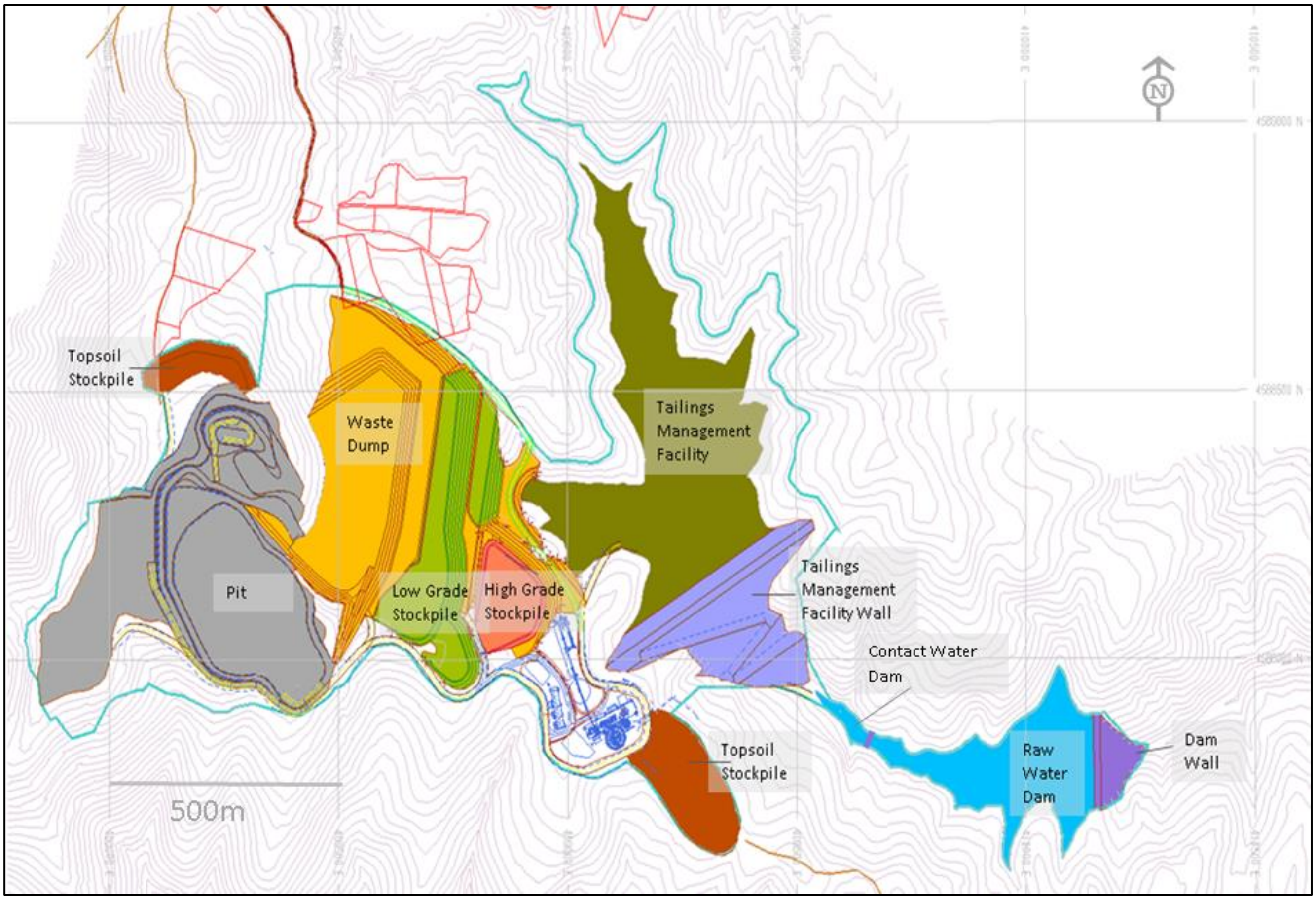


Figure 16-11: End of Year 2 Rozino site development

16.6.5 Year 3 Activities

The third year of operation is scheduled to include:

- Completing Phase 1 pit and the start of Phase 2 mining
- Use of high-grade ore stockpiled to overcome strip ratio issues at the start of Phase 2
- Continuation of some in-pit waste rock backfill
- Development of the WRD to its maximum elevation
- Lifting and downstream construction of the TMF retaining wall.

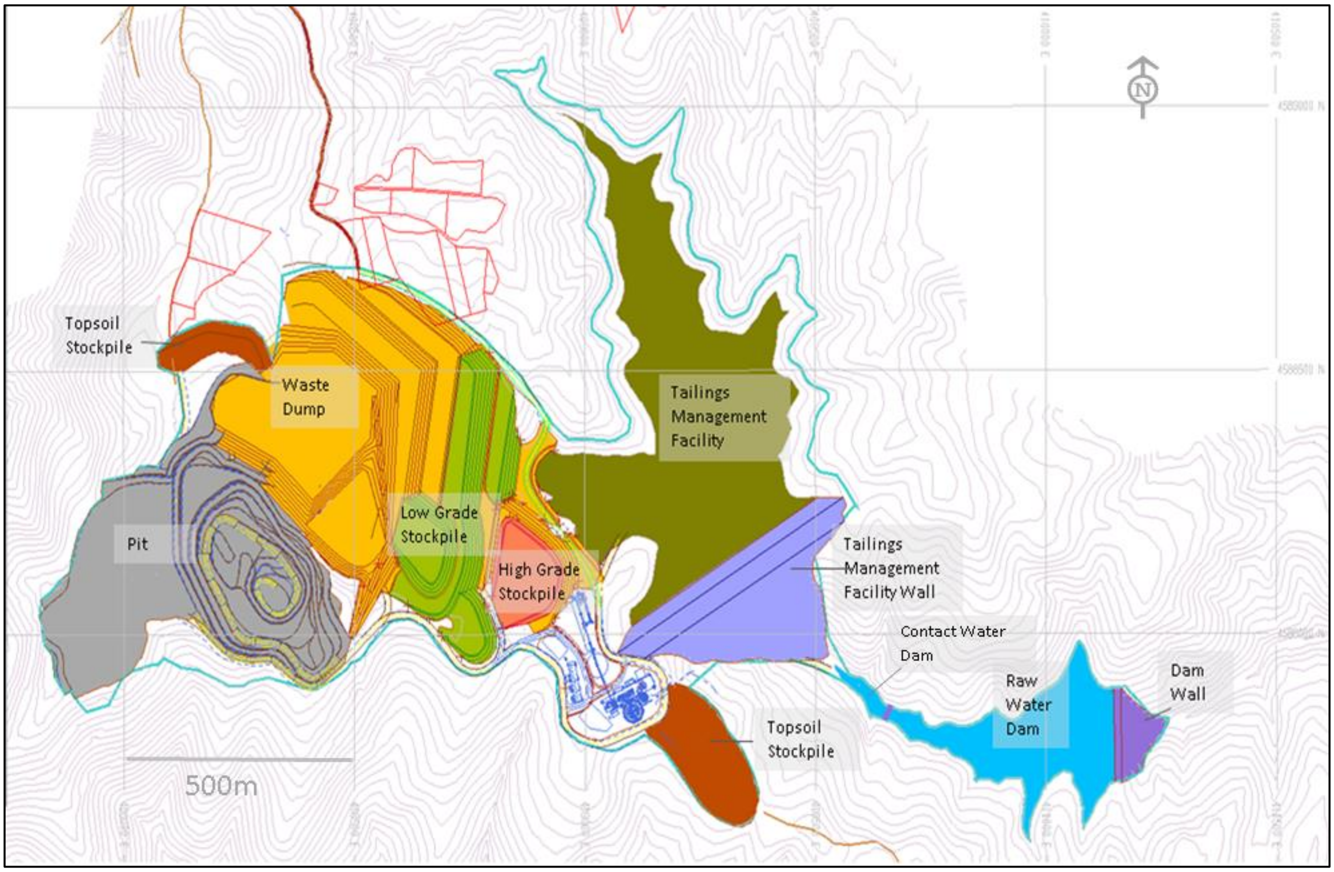


Figure 16-12: End of Year 3 Rozino site development

16.6.6 Year 4 Activities

The fourth year of operation is scheduled to include:

- Mining Phase 2
- Continued progress in pit backfill placement
- Completion of the TMF retention wall to 100% capacity.

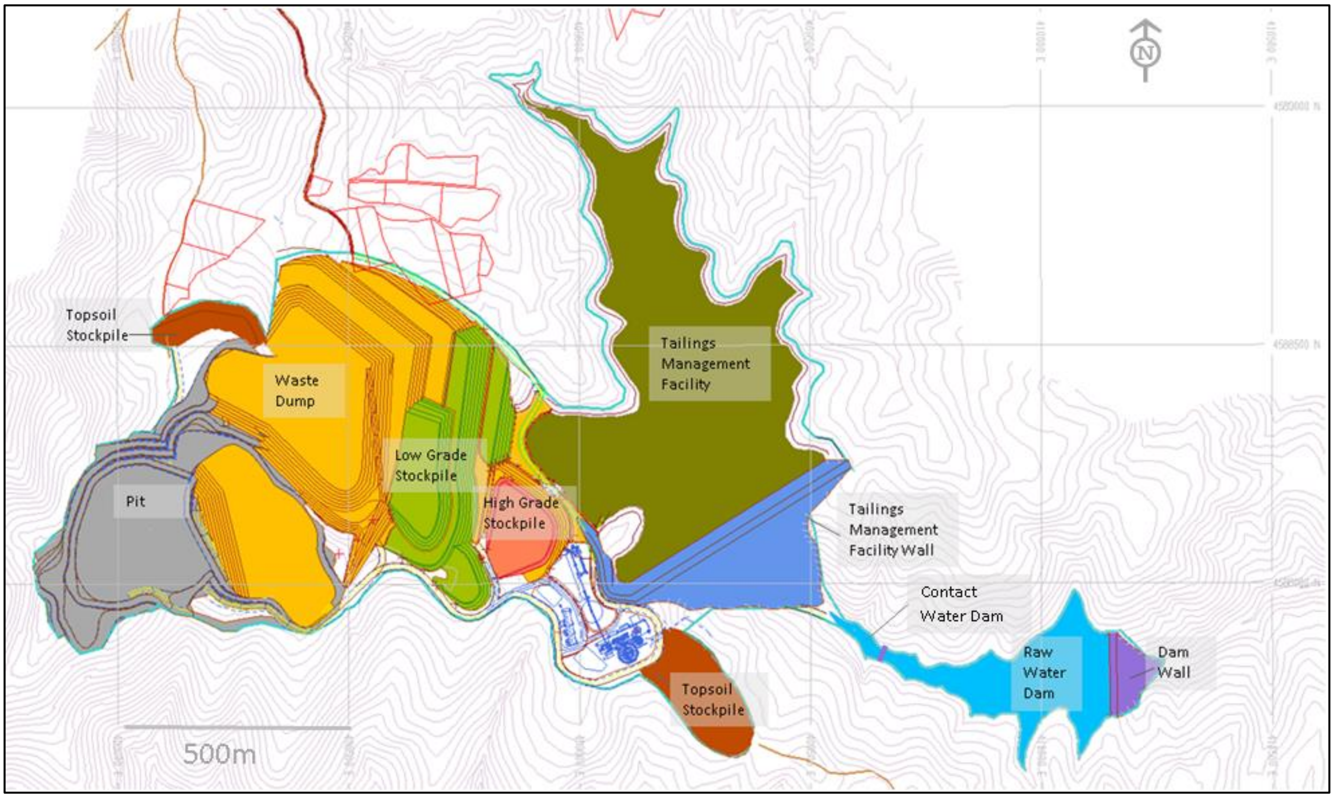


Figure 16-13: End of Year 4 Rozino site development

16.6.7 Year 5 Activities

The fifth year of operation is scheduled to include:

- Mining the Phase 2 pit to within 3 months of completion
- Continuation of in-pit backfill.

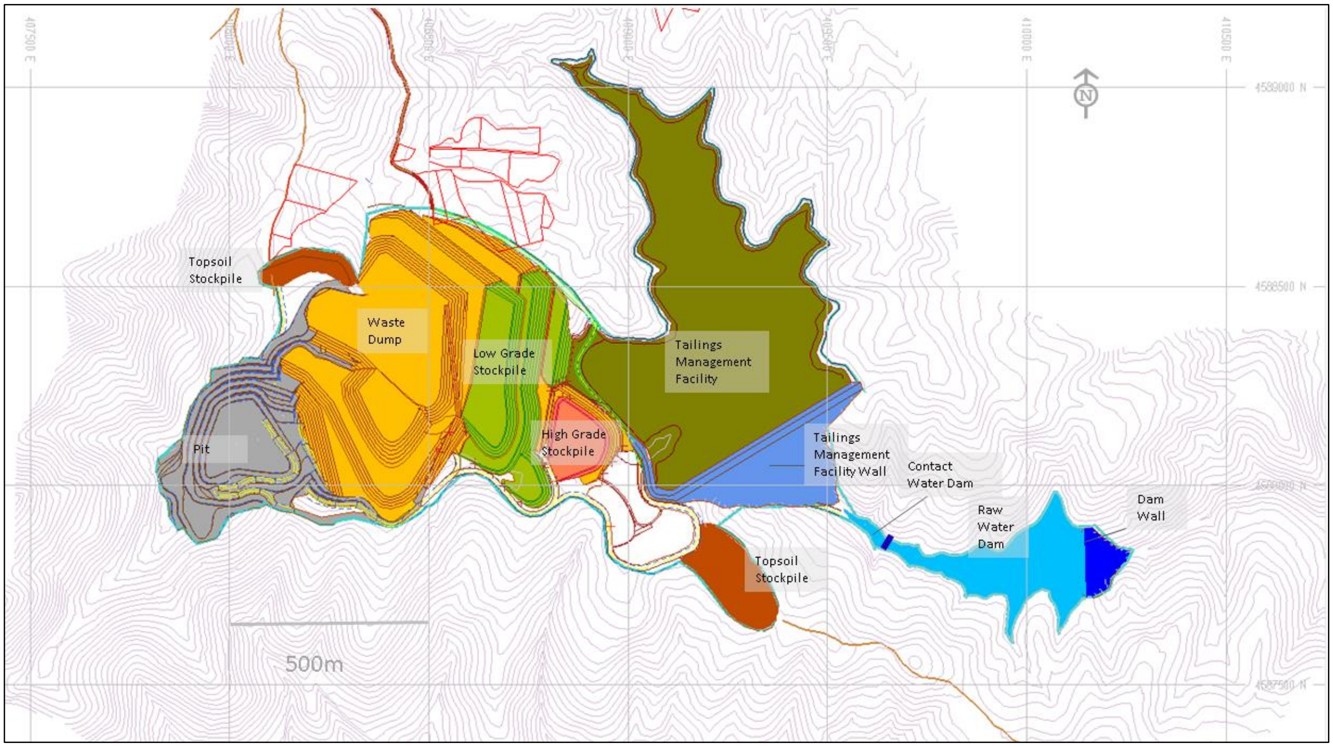


Figure 16-14: End of Year 5 Rozino site development

16.6.8 Year 6 Activities

The sixth year of operation is scheduled to include:

- Completion of Phase 2 pit mining in Q1
- Adapting the pit for tailings disposal
- Low-grade ore rehandle to the plant
- Completion of use of the TMF for tailings disposal
- Commencing use of the pit of tailings disposal, installation of relevant pumps and pipes.

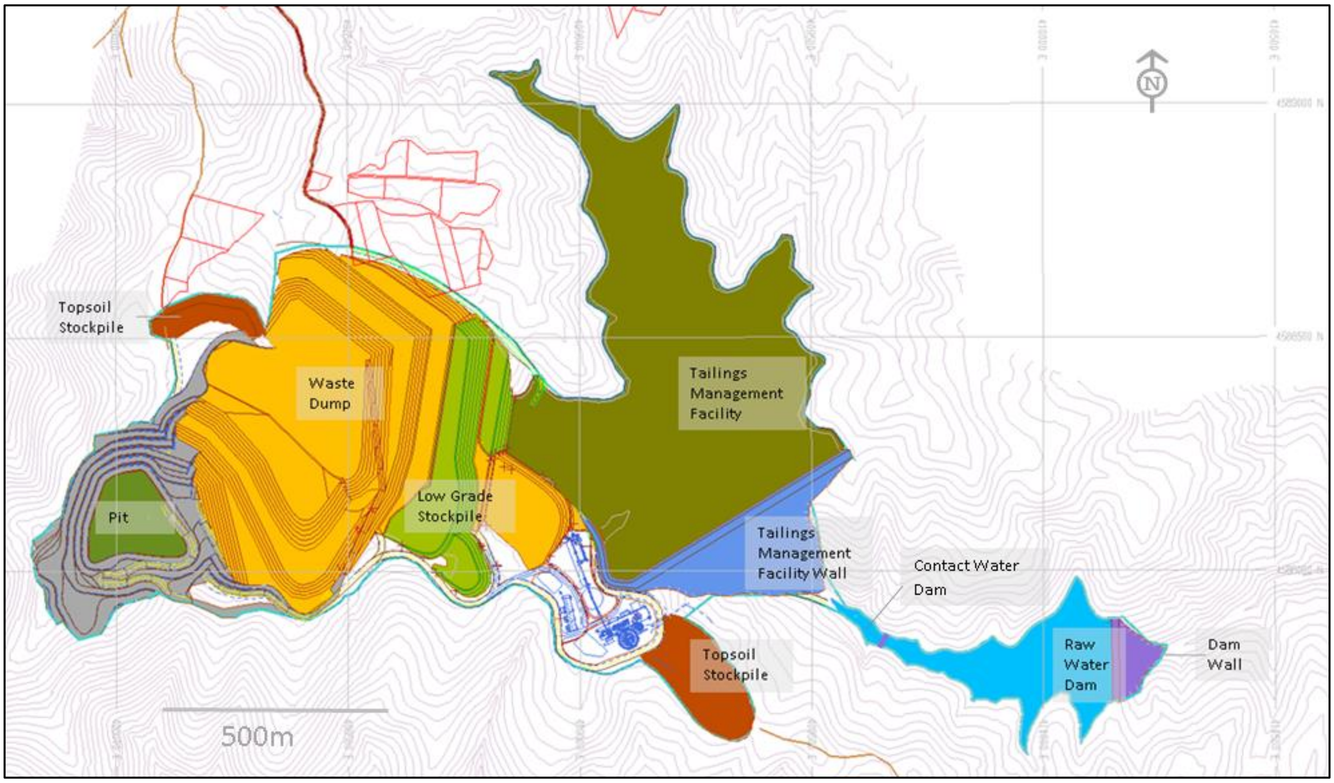


Figure 16-15: End of Year 6 Rozino site development

16.6.9 Year 7 Activities

The seventh year of operation is scheduled to include:

- Mining the low-grade ore stockpile to completion
- Commencing rehabilitation activities.

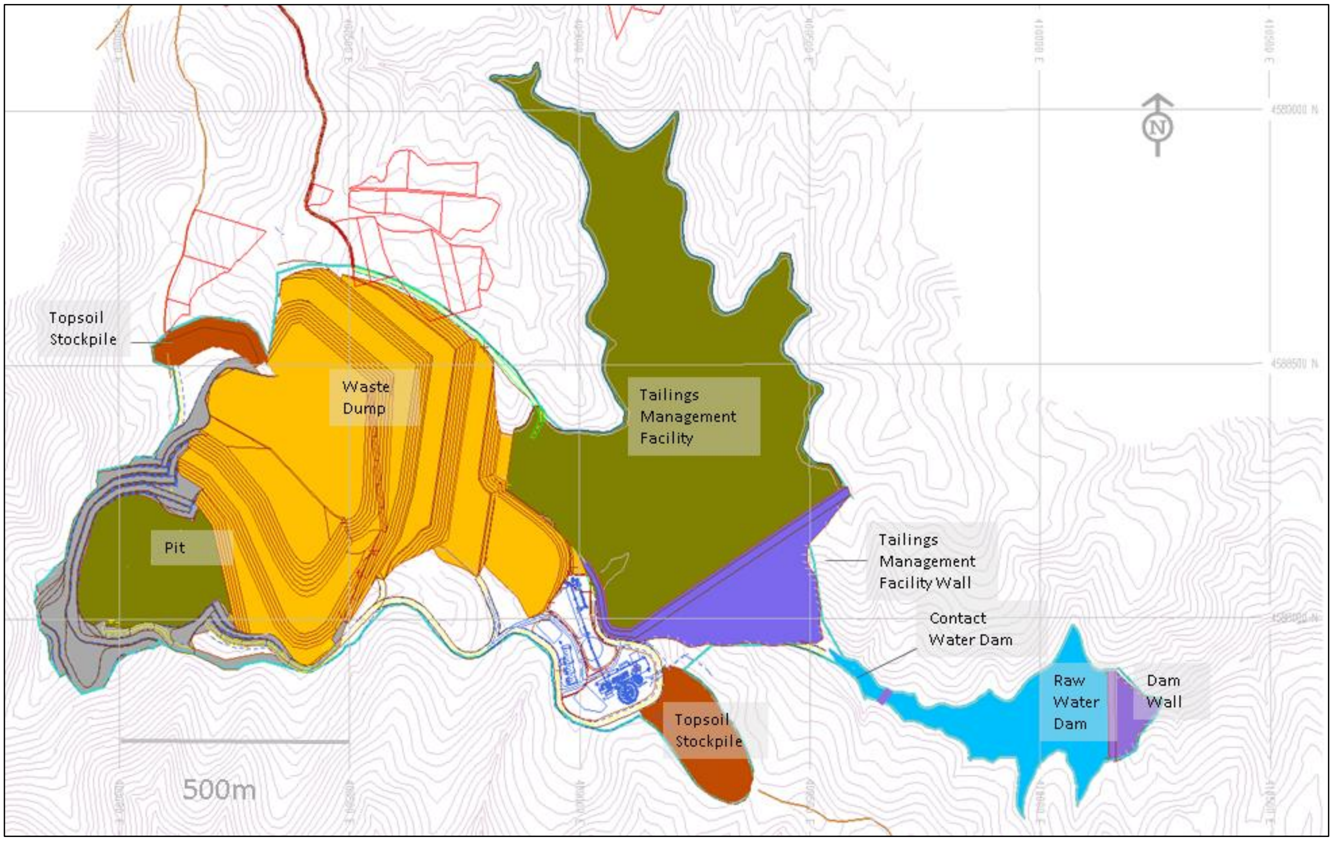


Figure 16-16: End of Year 7 Rozino site development

16.6.10 Mine Closure Year 8+ Activities

The Project site is rehabilitated.

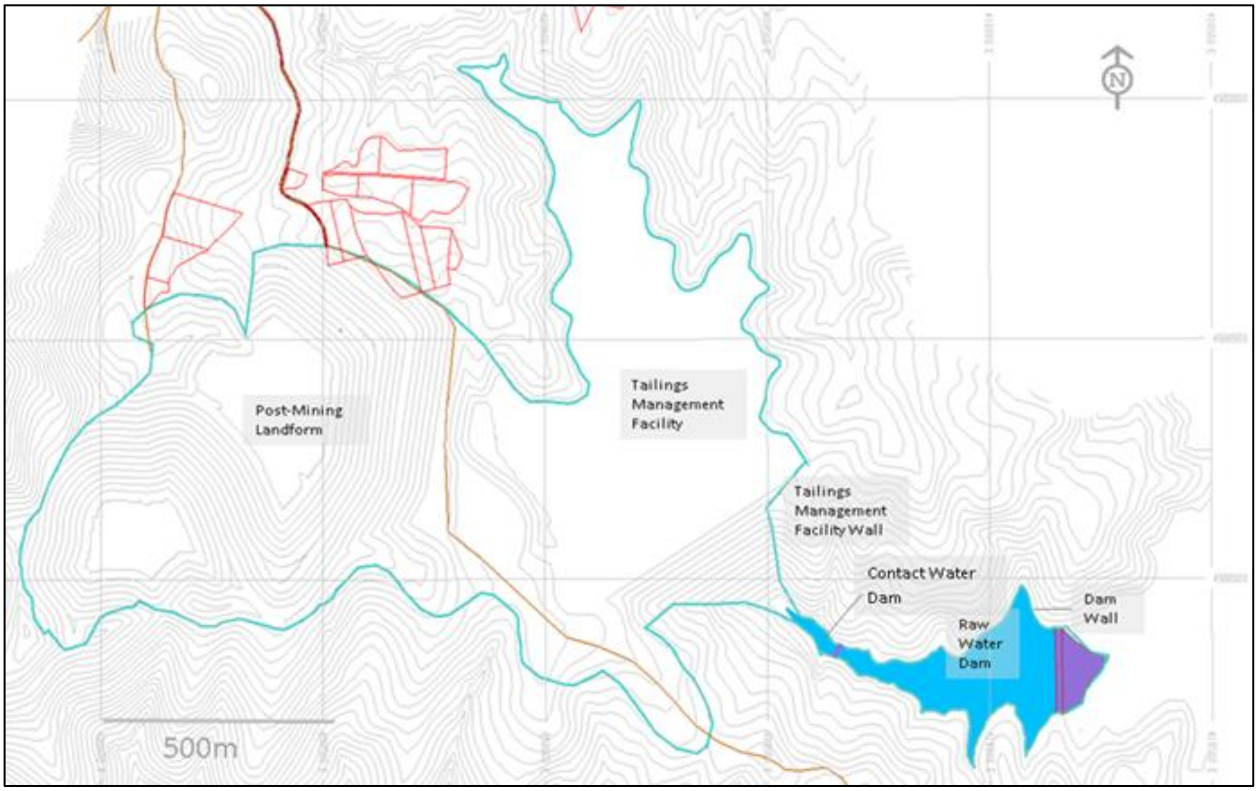


Figure 16-17: End of calendar Year +8 Rozino site development

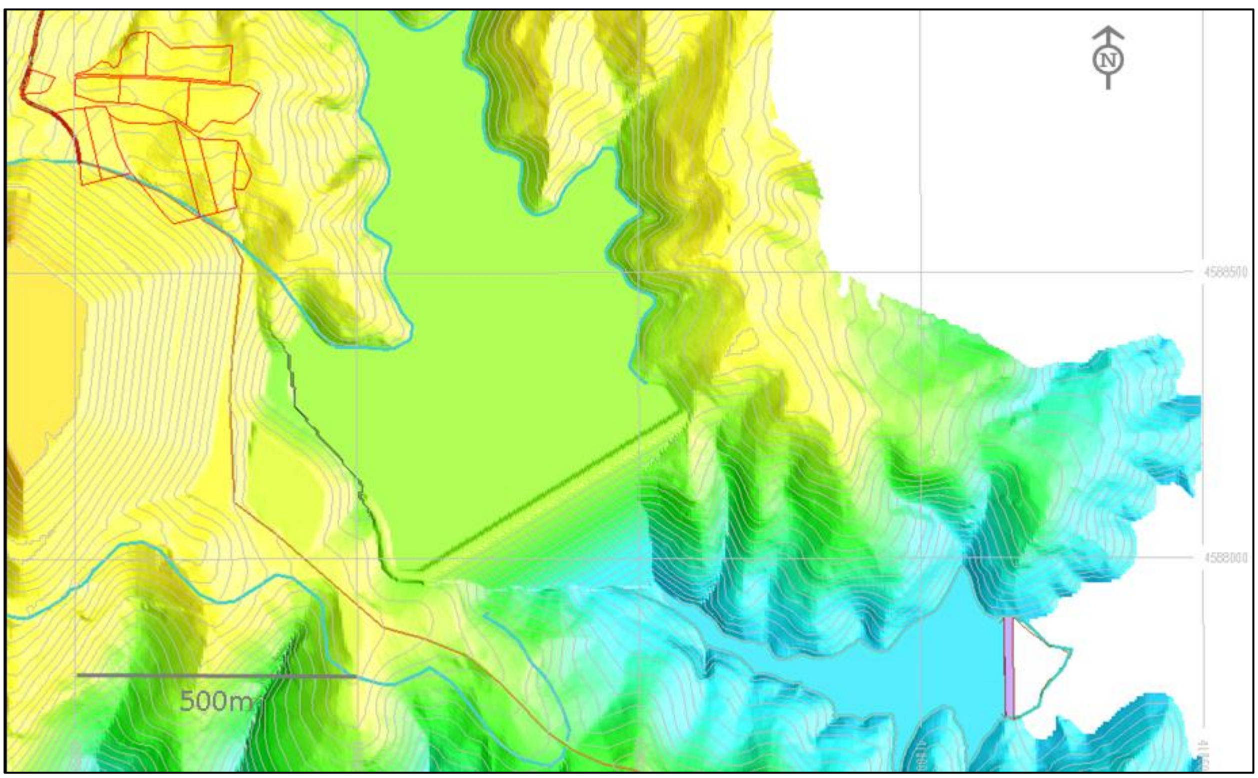


Figure 16-18: Post-mine closure showing the rendered surface

17 Recovery Methods

The Flotation Plant, including comminution, is designed to process 1.75 Mtpa of ore following the initial ramp-up phase. The optimal process derived for the Rozino sulphide mineralization is flotation to produce a gold-bearing sulphide (pyrite) concentrate at the Rozino site. This will be followed by the dewatering and transportation of the concentrate to the Central Plant (operated by Gorubso), where it will be reground to a P_{80} 20 µm and processed by conventional carbon-in-Leach (CIL) to produce a gold-silver doré.

The ROM ore is delivered as a mix of the three ore types but the proportions will differ depending on relative abundance at any one time in the mining schedule. The ROM ore will pass through a three-stage crushing circuit and a ball mill to produce a flotation feed with a target grind size P_{80} 75 µm. A gold-bearing concentrate will be produced by conventional bulk sulphide flotation, followed by a single stage of flotation cleaning. The concentrate mass pull, depending on the proportion of ore types in the feed, is expected to range between 2.14% (for oxide) and 4.18% (for sulphide) by weight, with an average of 3.8%. The filtered concentrate is expected to have a moisture content ranging between 10 and 12%. Flotation tailings will be thickened to recover water before it is pumped to the Rozino TMF for the initial 5.25 years of mine life, and then to the Phase 2 pit (after termination of mining) during the last 1.75 years.

On average, 67,000 t of concentrate, with gold grades averaging between 15 g/t and 40 g/t, will be produced annually. The thickened and filtered concentrate will be loaded into trucks and transported 85 km on paved roads to Gorubso's existing and operating Central Plant.

The concentrate will be offloaded at the Central Plant in a fenced compound with up to two weeks of storage capacity. A front end loader will move the concentrate to a feed bin which will then discharge to a slurry mixing tank. The slurry will then be ground to P_{80} 20µm in a closed circuit using a stirred mill and a cyclone classification system. The fine ground product from the cyclone overflow will be thickened and pH conditioned before reporting to the existing counter-current flow CIL circuit.

Excess copper will be removed from the loaded carbon by applying a cold cyanide solution wash prior to precious metal desorption. The gold and silver will be desorbed from the loaded carbon by a hot pressurized acid wash. The metals will be recovered by electrowinning onto steel wool. The steel wool cathodes and metal sludge will be collected in trays, calcined and smelted into doré. Leach tailings from the Central Plant will be detoxified and pumped to the existing Gorubso TMF.

It is estimated that a total of 368 koz of saleable gold in doré will be produced over the mine life.

17.1 Design Basis

This section provides design factors and site information used in the design and sizing/selection of equipment for the Flotation Plant and associated facilities.

17.1.1 Design Factors

The adopted equipment design factors (EDFs) used are listed in Table 17-1.

Table 17-1: Equipment Design Factors (EDF)

Equipment Types	EDF
Feeders	1.15 on solids rates
Jaw / cone crusher	1.15 on solids rates
Screens	1.15 on solids flows
Centrifugal pumps	1.15 to 1.25 on flow

Equipment Types	EDF
Compressors	1.15 on flow
Agitators	1.00
Tanks and hoppers	1.00
Thickeners	1.30 on area
Filters	1.25 on area
Hydrocyclones	1.10 on flow
Ball mill	1.10 on solids rate
Flotation cells	2.5x on float time

17.2 Process Design Criteria

Based on the extensive testwork programs undertaken by Wardell Armstrong and Eurotest Control, the base case flowsheet for the treating of Rozino ore was developed for the PFS.

Detailed process design criteria (PDC), a mass balance (MB) and a mechanical equipment list (MEL) were prepared to support the basis of plant design. Based on the MEL, the equipment datasheets were prepared and sent to suppliers for quotation and preparation of the capital cost estimate.

Plant general arrangement layouts were also prepared to assist with preparing the Bill of Quantities, which were subsequently used as the basis for the capital cost estimate.

17.2.1 Rozino Plant Production Criteria

The Rozino plant was designed to process 1.75 Mtpa of ore over the life-of-mine following the ramp-up phase. As the ore is essentially a pyrite concentrate containing gold, sulphur feed grades are expected to largely dictate concentrate production rates. However, for the design of the downstream Central Plant equipment sizing, sulphide concentrate mass pull of 4.5% by weight was used.

Assuming a plant availability of 92%, the operating regime for the concentrator was set at 8,059 h/a, which is typical for a plant of this level of complexity and size. This sets the nominal throughput at 217 dry t/h.

For the Mineral Reserve, the average expected overall gold recovery for Oxide material is 67.4%, Transitional 70.7% and Sulphide 83.3%, for an average of 79.3% to final doré. Over the life of the Project it is estimated that gold recovery will vary from 65 to 85% on an annual basis depending on the relative proportions of oxidized ore in the plant feed, and the head grade.

Key production criteria for the concentrator are provided in Table 17.2.

Table 17-2: Rozino plant production criteria

Criteria	Units	Value
Annual processing capacity	tpa	1,750,000
Operating time		
Crushing	hpa	6,500
Concentrator	hpa	8,059
Nominal process rate		
Crushing	t/h	278
Concentrator	t/h	217
Head grade – design		
Gold	g/t Au	1.22
Sulphur	% TS	0.42

Criteria	Units	Value
Concentrate mass pull		
Nominal	% Wt	3.8
Design	% Wt	4.5
Gold recovery to concentrate	%	91.4
Concentrate grade	**g/t Au**	
Nominal	g/t Au	26.1
Design	g/t Au	30.0
Concentrate moisture content	%	10-12

17.2.2 Central Plant CIL Production Criteria

Based on the nominal concentrate mass pull of 3.8% by weight, the Central Plant treatment facility is designed to process on average 65,000 tpa of concentrate.

Key production criteria for the Central Plant are provided in Table 17.3.

Table 17-3: Central Plant CIL production criteria

Criteria	Units	Value
Annual processing capacity	tpa	65,000
Operating time - CIL	hpa	8,059
Nominal processing rate		
- CIL	t/h	8.1
- CIL	tpd	390
Design gold head grade	g/t Au	30
CIL residence time		
- Nominal	h	36
- Design	h	48
Design loaded carbon grade	g/t Au	3,500
CIL recovery	%	87.6
Solution losses	%	0.5

17.3 Process Flowsheet and Plant Description

17.3.1 Process Flowsheet

Figure 17-1 provides a schematic representation of the proposed flowsheet for the Rozino Plant.

The Rozino plant consists of the following process unit stages:

- Crushing, sizing and conveying
- Fine ore storage (covered stockpile)
- Grinding (ball mill)
- Flotation
- Tailings thickening
- Concentrate thickening and filtration
- Tailings disposal
- Services: water, air, and reagent systems.

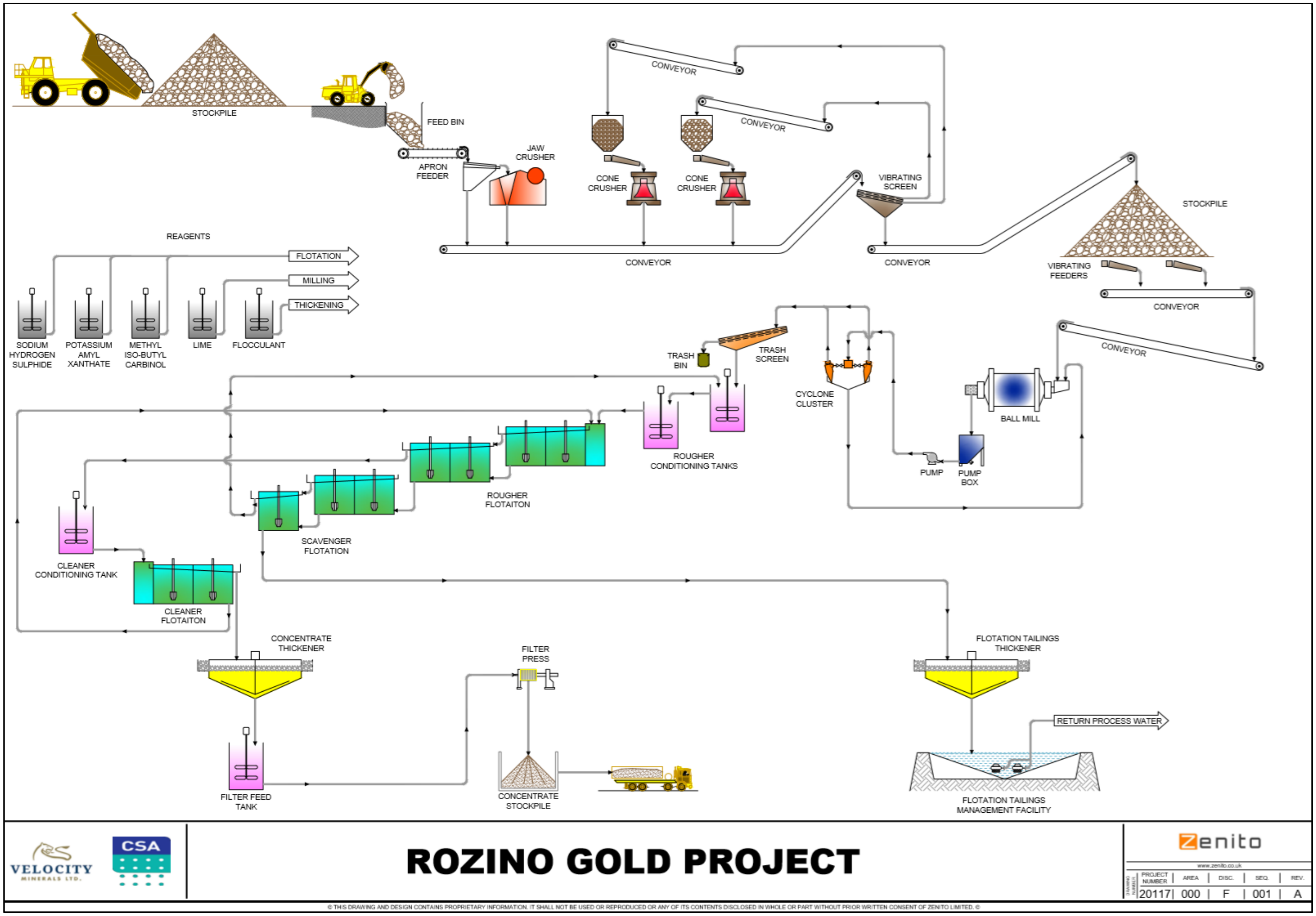


Figure 17-1: Rozino flotation plant schematic

Source: Zenito, 2020

17.4 Process Description

17.4.1 Flotation Plant

The full plant layout is shown in Figure 17-2 and is described in detail in the following sections.

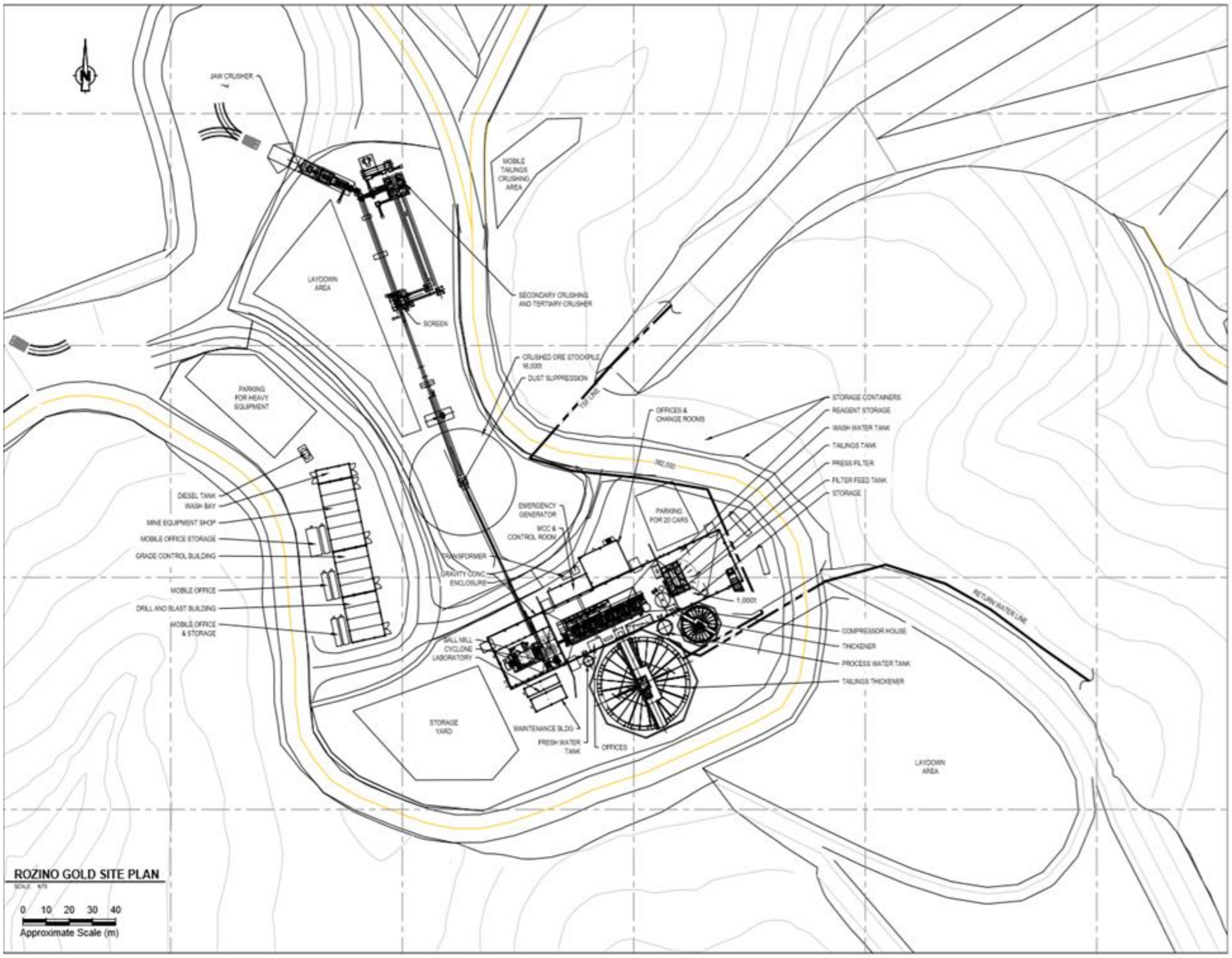


Figure 17-2: Flotation Plant site layout
Source: Halyard, 2020

17.4.1.1 Area 100 – Crushing

Selection Basis

The low crushing work index (4.7-8.92 kWh/t) and moderate Bond ball mill work index (11.2-12.5 kWh/t) for the various ore types favours the use of a multi-stage crushing and ball milling circuit. This type of comminution is common with these types of ores. An apron feeder was selected as the preferred primary feeder as it can handle the impact of large rocks falling into the ROM bin. A static grizzly was selected to scalp off any fines from the feed to the jaw crusher.

Description

The crushing circuit is designed to produce a crushed ore of P_{80} 12 mm.

The crushing circuit consists of three crushing stages; a primary jaw crusher, followed by secondary and tertiary cone crushers. ROM ore is either direct-tipped from the pit or fed via a Cat 966 loader from the stockpile to a 100 t wear plate lined coarse ore bin. A 700 mm x 700 mm screen will prevent oversize entering to the jaw crusher. A fixed mounted 15 kW hydraulic hammer will break oversize ore. The ROM bin apron feeder (1300 mm x 6400 mm 22 kW) will supply the primary jaw crusher (42" x 48" 160 kW). The crushed rock reports to the screen via a 32" X 45 m conveyor. A magnet will remove steel scrap to a metal scrap bin. Separate dust collectors will aid water sprays for dust suppression and maintain air quality within each of the crusher buildings. A 5t hoist is provided to aid maintenance in the primary crusher building.

Primary crushed material is screened using a double-deck 8' x 24' 45 kW screen; the undersize reports to the fine-ore stockpile and the screen oversize products report to the secondary cone crusher and tertiary cone crushers respectively. The secondary and tertiary crushers operate in a closed loop with the double-deck screen until all material is fed to the fine-ore stockpile via the lower deck of the sizing screen. Two belt weightometers are provided. Both cone crushers include lube and cooling units. The secondary crusher is a Metso Nordberg HP600 315 kW unit, and the tertiary is a HP800 500 kW unit.

Assuming a 16-hour operation and an average equipment availability of 75%, a 5,000 tpd throughput is may be achieved by the installation of HP600 and HP800 for secondary and tertiary cone crusher duties respectively.

17.4.1.2 Area 110 – Stockpile

Selection Basis

A live fine ore stockpile was selected over a silo primarily because of the capital cost of the latter if it were fabricated from steel. The alternative of using a concrete silo was not examined for the PFS design but could be evaluated in the FS.

A live capacity of 24 hours (5,000 t) was selected on the basis of a re-fit of the jaw crusher taking 12 to 14 hours. In the event of an extended maintenance shut-down a bulldozer can be used to feed ore from the stockpile.

Description

The fine-ore stockpile area includes a 24-hour live volume (5,000 t) and crushed-ore stockpile with a surge capacity (dead) of 15,000 tonnes.

The crushed ore will discharge onto the fine-ore stockpile via a 36" stacker conveyor with a nominal throughput of 278 t/h. The crushed material is reclaimed from the base of the stockpile using two belt feeders with variable frequency drives for fine tuning flow rates, in a duty/standby arrangement. The ore discharges onto the mill feed conveyor at a nominal throughput rate of 217 t/h.

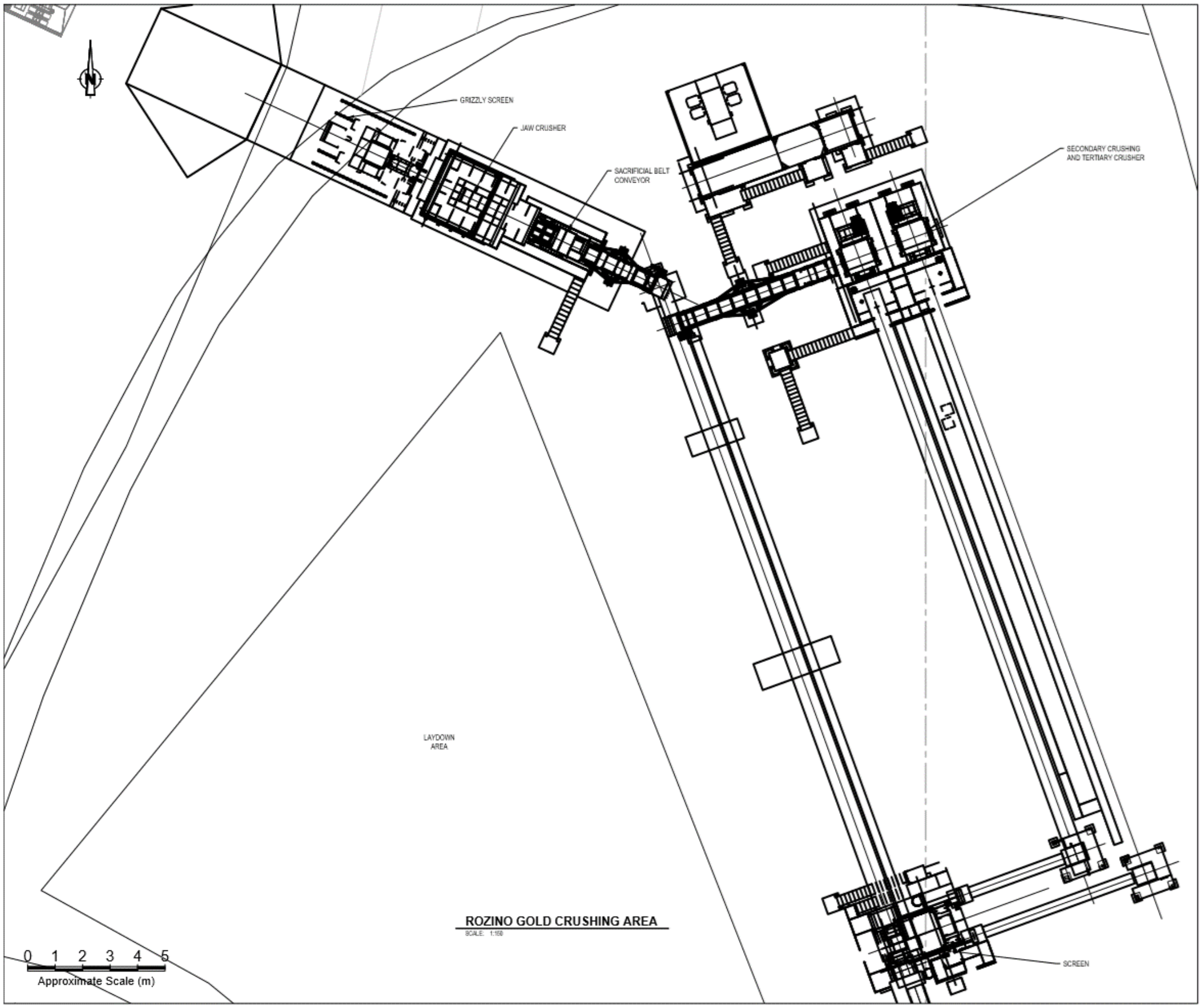


Figure 17-3: Plan view of Flotation Plant crushing facility
Source: Halyard, 2020

17.4.1.3 Area 200 – Grinding

Selection Basis

A conventional three-stage crushing circuit followed by a ball mill was selected as the preferred grinding circuit for the Rozino plant. This arrangement demonstrated lower capital and operating costs compared to the alternative primary crushing than a SAG-Ball comminution circuit evaluated.

Description

The grinding area consists of a ball mill operating in a closed circuit with a cyclone classification cluster. The grinding circuit is designed to produce a P_{80} 75 µm.

The ball mill dimensions are 4.88 m diameter x 7.62 m length (16’ diameter x 25’ length) mill with a 3,500 kW motor. The ball mill installation includes trunnion and gear box lubrication, cooler systems and control system. The ball mill discharges into a pump-box which feeds the cyclone pack via a cyclone feed pump, in a duty/standby arrangement. The cyclone pack consists of 14 cyclones, of which only 12 are operational at any time. The cyclone overflow gravitates to the flotation area, whilst the underflow is re-circulated back to the head of the ball mill.

Metallurgical testing indicated that gold recovery by gravity is within a percent of the selected bulk flotation process. The structural design of the plant allows for inclusion a gravity circuit (shown in Figure 17-4 to the right of the ball mill in grey) should future testing prove that gravity recovery is viable.

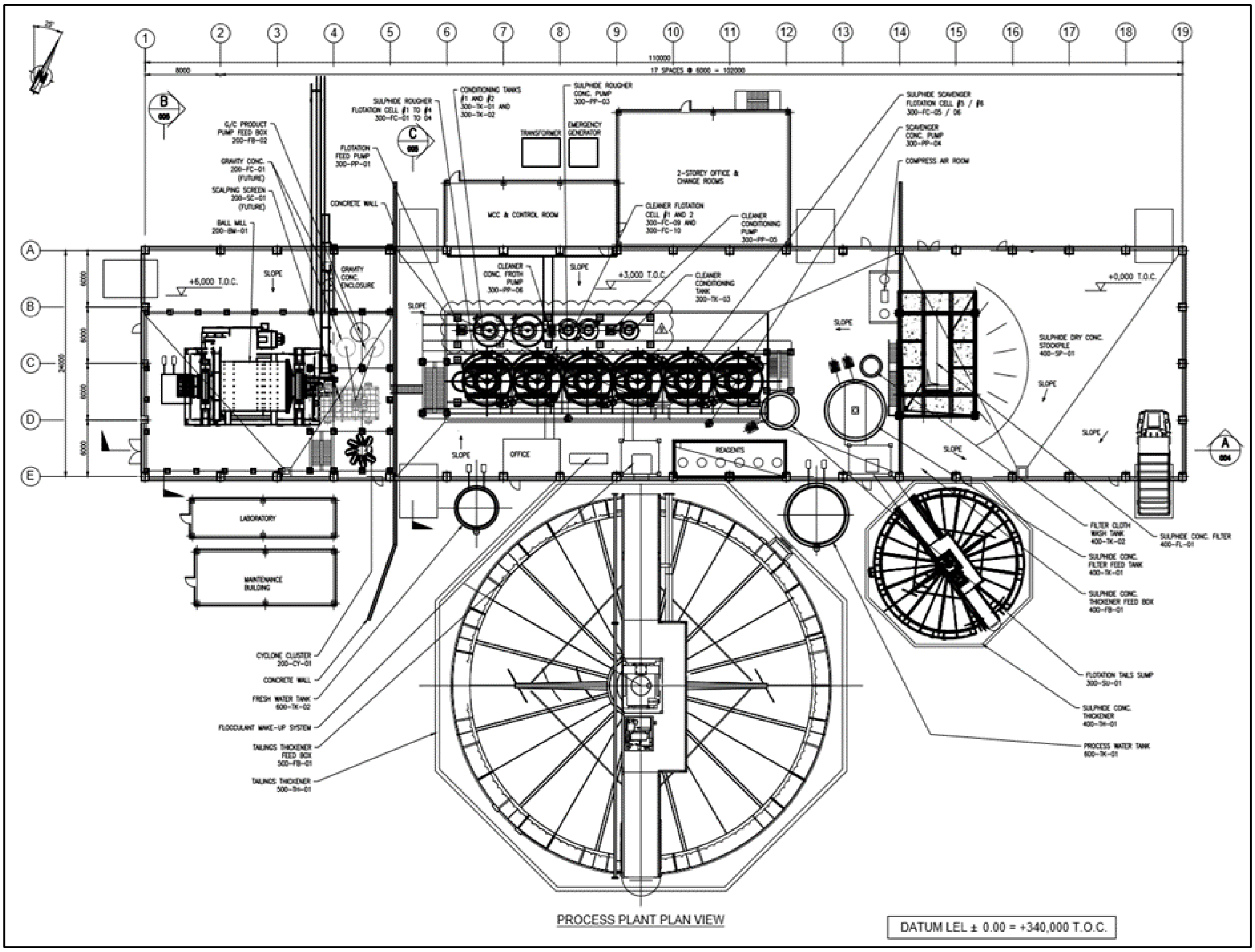


Figure 17-4: Plant layout plan for grinding to tailings and concentrate production
Source: Halyard, 2020

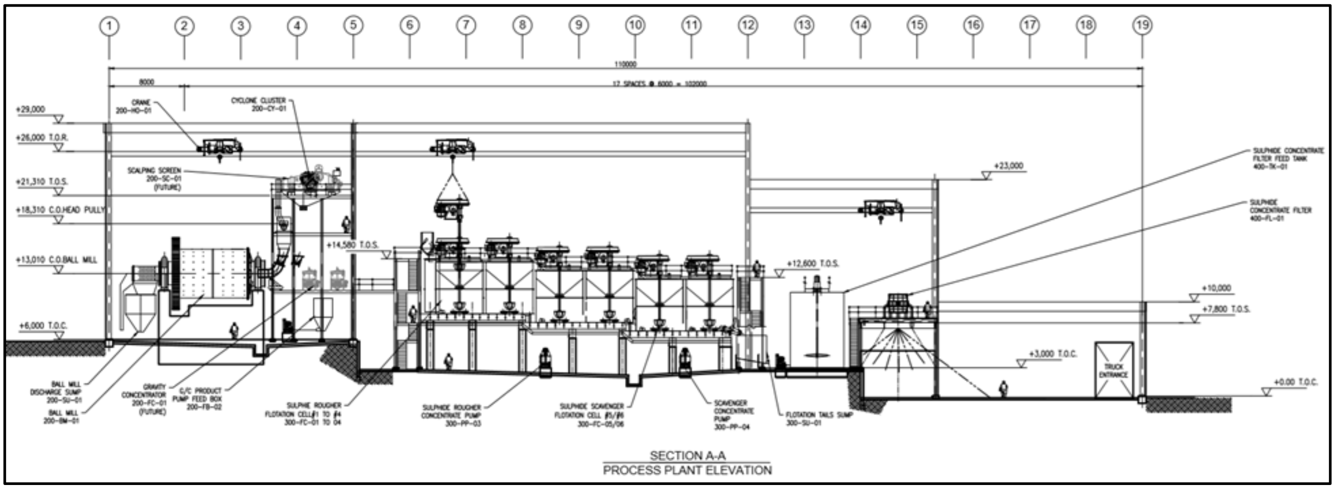


Figure 17-5: Grinding and concentrate building long section
Source: Halyard, 2020

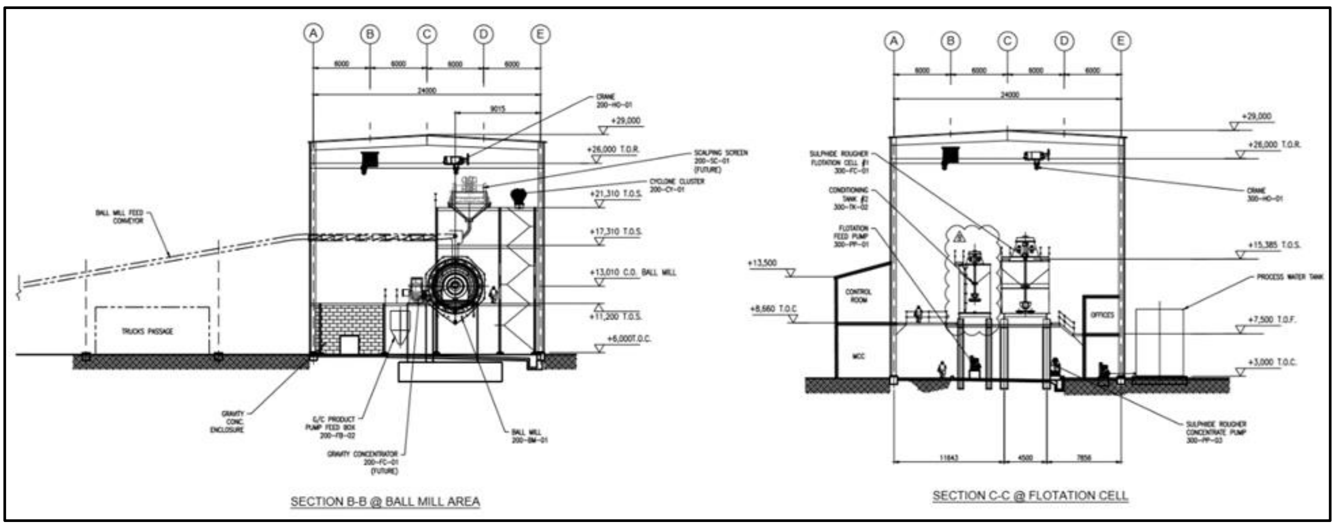


Figure 17-6: End elevation views of the grinding and concentrate building
Source: Halyard, 2020

17.4.1.4 Area 300 – Flotation

Selection Basis

Conventional tank cells were selected for the flotation circuit. Conventional tanks are more cost effective than staged flotation reactors (SFRs) from the perspective of capital and operating cost.

The sizing and selection of flotation cells were based upon laboratory float times, an industry scale-up factor of 2.5, and an assumed froth factor of 0.85.

Four rougher cells will be installed to prevent 'short circuiting'.

Description

The flotation circuit consists of two conditioning tanks with agitators, four rougher flotation cells, two scavenger cells and two cleaner flotation cells.

The cyclone overflow gravitates to the trash screen to remove any oversize and debris entering the flotation circuit. The screen undersize discharges into the first conditioning tank (50 m^3) where the slurry will be mixed with copper sulphate added as an activator. The slurry flows to the second conditioning tank (50 m^3) where the collector reagent PAX is added. An automatic flotation feed sampler is provided after the trash screen.

The slurry first reports to the rougher flotation circuit which consists of four 75 m^3 tank cells (via a flotation feed pump and a 10 m^3 rougher flotation feed box). The rougher concentrate is pumped to the cleaner conditioning tank. The rougher tailings reports to the scavenger flotation circuit which consists of two 75 m^3 tank cells. All agitator motors are sized at 90 kW. An automatic flotation tails sampler is included.

The scavenger concentrate is recycled back to the rougher circuit feed box. The scavenger tailings are pumped to the flotation tailings thickener for water recovery prior to discharge to the tailings management facility.

The cleaner flotation circuit consists of a conditioning tank (10 m^3), and two 10 m^3 flotation tank cells. The cleaner tailings are recycled back to the head of the rougher flotation circuit. The final concentrate reports to the concentrate thickener via the cleaner concentrate froth pump. The cleaner agitator motors are sized at 15 kW.

17.4.1.5 Area 400 – Concentrate Thickening and Filtering

Selection Basis

Static settling test results were available for the sizing of the concentrate thickener. No filtration tests were carried out to size and select the concentrate pressure filter unit. In lieu of test data, database values for similar concentrate types and particle size were used for sizing purposes. Vendors provided some information on moisture content in concentrates, which assisted with establishing the design moisture content of 10–12% w/w.

Description

The concentrate thickening and filtration circuit consists of a 16 m diameter concentrate thickener and a vertical plate and frame pressure filter.

The thickener will receive concentrate at a rate of 11.3 t/h and a feed density of 23% solids. The thickener overflow will recycle back to the process water tank and be re-used in the grinding and flotation areas. Thickener underflow (at 50 to 55% solids) will be discharged to the 200 m^3 filter feed via two 50 mm by 40 mm underflow pumps, in a duty/standby arrangement. The filter feed tank will be equipped with an agitator to prevent the settling out of the solids. The concentrate thickener will be fitted with a recycle arrangement in the event that the desired underflow density is not achieved. The concentrate filter feed storage tank is designed to have a concentrate slurry capacity equivalent to 12 hours production.

Slurry from the concentrate filter feed tank will be pumped to the concentrate filter via two 75 mm by 50 mm filter feed pumps, in a duty/standby arrangement. The concentrate filter will be a vertical plate and frame pressure filter designed to operate on a 75% utilization basis, and produce 11.3 tonnes of concentrate per day, at a moisture content of 10 to 12%.

The filter press has a dedicated compressor and air receiver.

The concentrate falls into a concrete bay for load out by a front end loader. The loader will discharge to 20 or 30t concentrate trucks operated by a haulage contractor.

The trucks will require GPS tracking, sampling, the concentrate marked with lime as a security measure (Figure 17-7), tarped and security locked and tagged (Figure 17-8). As each vehicle leaves the concentrate shed, all surfaces and tyres will be washed to prevent environmental and or road contamination and concentrate loss. The floor of the concentrate area and washdown apron is sloped to a collection point and sump for recollection into the thickener.

Each truck will be weighed prior to departure and on arrival at the Central Plant. The trucks will be monitored by GPS tracking and photographic records of the security measures will be examined for each truck by the security team.

Figure 17-7: Example concentrate lime marking

Figure 17-8: Example of concentrate truck security tagging and sealing

17.4.1.6 Area 500 – Tailings Area

Selection Basis

The tailings thickener was selected based on static settling tests undertaken by WAI.

Description

The tailings circuit consists of a 40 m diameter thickener, and a conventional paddock fenced tailings dam.

The 40 m diameter thickener will receive tailings at 224 t/h and a solids density of 35% by weight. Thickener overflow solution will be recycled back to the 400 m^3 process water tank, while the thickener underflow (at 55% solids concentration) will be pumped to the TMF using two 200 mm by 150 mm underflow pumps, in a duty/standby arrangement. In the event that the underflow density is not at the desired solids content, there is the option to recirculate the underflow.

Water from the TMF will be reclaimed and pumped back to the process water tank via two barge-mounted 150 mm reclaim water pumps.

17.4.1.7 Area 600 – Water Distribution

Water for the plant comprises two categories; process water and fresh water.

Process Water

Process water is stored in a 400 m^3 tank and is sourced primarily from the tailings thickener overflow and the reclaim water from the tailings dam. The process water tank is also supplemented with water from the contact water dam (CWD - mine site contact water) and the raw water dam (WRD). Process water is supplied to the plant via two 200 mm by 150 mm pumps, in a duty/standby arrangement.

Process water within the plant will be used in grinding, flotation, thickening, filtration, tailings disposal and flocculant make-up.

Fresh Water

Fresh water for the plant is stored in a 150 m^3 tank and is sourced primarily from the RWD (imported fresh or non-contact water).

Fresh water is used for the following:

- Screens and cloth wash:
  - The screens within the crushing area and the cloth wash for the concentrate filter. Fresh water is supplied by two 100 mm by 75 mm pumps, in a duty/standby arrangement.
- Gland service water:
  - Centrifugal pumps within the plant requiring gland water are supplied by two 40 mm by 25 mm gland water pumps, in a duty/standby arrangement.
- Reagent make-up:
  - Mixing of reagents.
- Fire water:
  - Fire water is provided to the fire stations in the plant by two 40 mm by 25 mm fire water pumps in a duty/standby arrangement. The fire water pumps have back-up power through a diesel back-up generator in case of an emergency.

17.4.1.8 Area 701 – Reagent Distribution

The reagent distribution area includes the make-up of the following reagents:

- Frother: Methyl iso-Butyl Carbinol (MIBC):
  - The MIBC make-up system consists of a storage tank with an agitator and two distribution pumps, in a duty/standby arrangement. These pumps deliver the MIBC to the flotation area.
- Collector: Potassium Amyl Xanthate (PAX):
  - The PAX make-up system consists of a one m^3 mixing tank, a two m^3 storage tank with an agitator and two circulation pumps (in a duty/standby arrangement) that deliver the PAX solution to the flotation area.
- Activator: Copper Sulphate ($CuSO_4.5H_2O$):
  - The copper sulphate make-up system consists of a one m^3 mixing tank, a 2 m^3 storage tank with an agitator and two circulation pumps (in a duty/standby arrangement) that deliver the copper sulphate solution to the flotation area.
- Sulphidizer: Sodium Hydrogen Sulphide (NaHS):
  - The NaHS make up system consists of a one m^3 mixing tank, a two m^3 storage tank with an agitator, and two circulation pumps (in a duty/standby arrangement) that deliver the NaHS solution to the flotation area.
- Collector: Aerofloat 404 (A404):
  - The collector reagent A404 is delivered as a concentrated liquid and will be dosed to the second conditioning tank from 200 l drums using peristaltic pumps.

17.4.1.9 Area 702 – Flocculant Distribution

The flocculant distribution system includes a flocculant powder hopper, screw feeder, a 2 m^3 mixing tank with agitator, a five m^3 holding tank with agitator, and two dosing pumps. This setup allows for the proper wetting and aging of the flocculant.

The two dosing pumps will deliver the mixed flocculant to both the concentrate and tailings thickeners. Prior to being added to the thickener, each line will have an inline mixer for further dilution of the flocculant to 0.02% w/v.

17.4.1.10 Area 750 – Air Supply

Compressed air within the plant is supplied by the following equipment:

- Air compressor - 75HP
- Filter press air compressor – estimated 60HP
- Air drier - 350 cfm, 100 psig
- Instrument air receiver - 500 US Gallon capacity
- Plant air receiver - 500 US Gallon capacity
- Filter press air receiver – estimated 400 US Gallon capacity

In addition to the compressed air, the air supply area will also include a blower that will deliver forced air to the flotation cells. The blower is sized to provide 92 m^3/min at 51 kPa.

17.4.2 Central Plant Facility

Gorubso-Khardjali AD own the Central Plant which is fully operational but requires some modification to achieve the recoveries as specified in this Technical Report. Gorubso are fully responsible for the handling and processing of the concentrate after delivery and through to production of gold doré. The process is described in the following sections for full process transparency.

The Central Plant uses conventional carbon-in-leach (CIL) technology for the recovery of gold. A process flow diagram is included as Figure 17-9. The plant is operated by Gorubso which has a well-trained workforce. Some modifications to the circuit will be required to accommodate the Rozino concentrate but only shows that part of the Central Plant required to process the Rozino concentrate. The figure includes all required modifications. The greater Central Plant also includes crushing and ball milling circuits

The Central Plant has a nominal throughput capacity of 480 tpd or 172,800 tpa depending on ore type and gold grade, and can process on average 165,000 tpa. The concentrate will be campaigned through the Central Plant for periods in the order of 4 weeks at a time when steady state production is achieved. This will allow plant operating parameters to be fine-tuned, allow the plant to run at optimal throughput capacities and efficiencies, and facilitate concentrate handling and tracking, gold recovery reconciliation and accounting. Additionally, disruptions due to changes in material feed from third-party operations will be minimized.

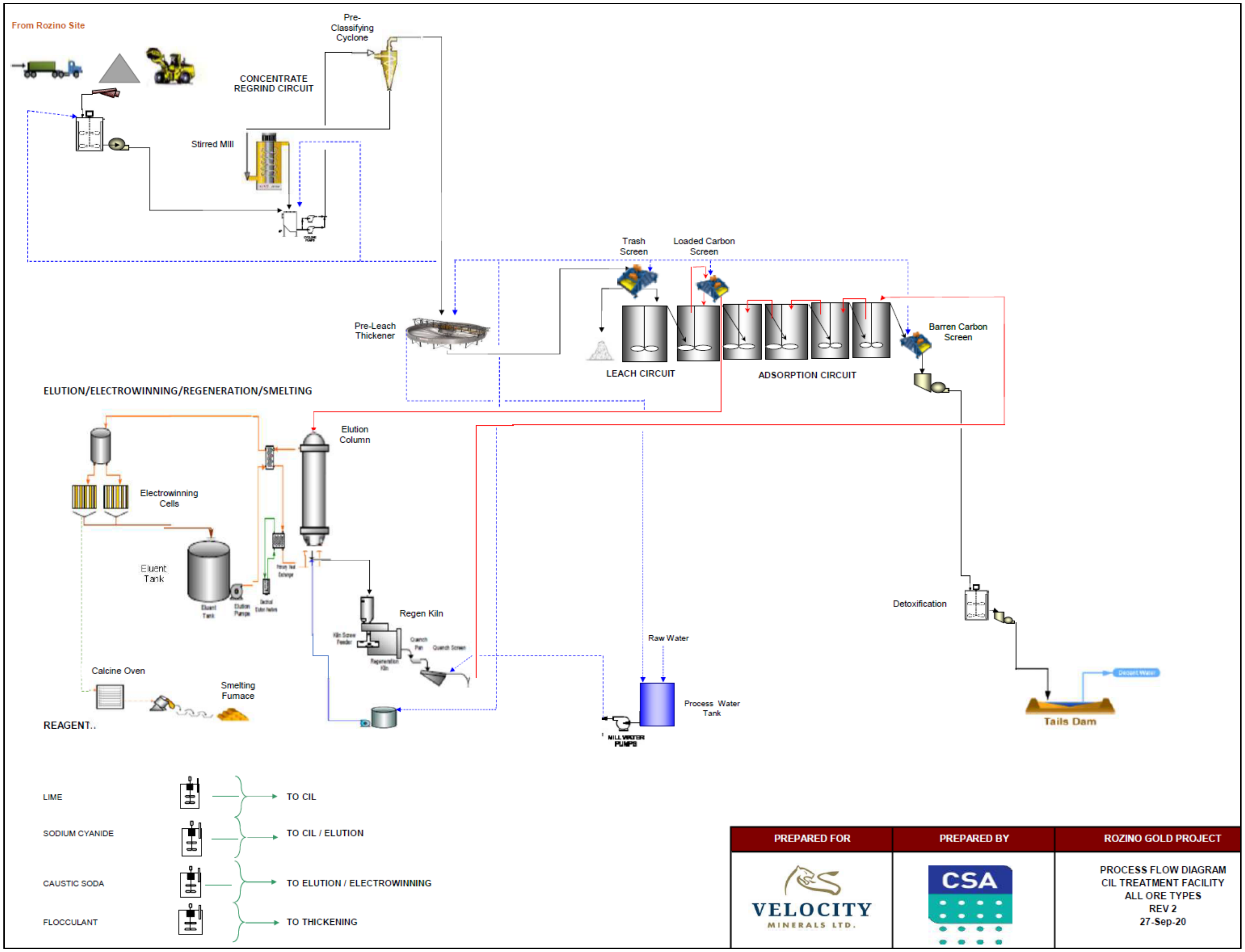


Figure 17-9: Process flow diagram central plant

17.4.2.1 Area 800 – Feed and Comminution System

The flotation concentrate from the Flotation Plant will be transported by road and stockpiled for processing at the Central Plant. Trucks will offload the concentrate by dumping it into a secure area near the feed hopper. A front end loader will move the material from the stockpile into the hopper at a controlled rate. From the hopper, the concentrate will be fed into a re-pulping tank equipped with an agitator, via a hopper, belt feeder and a primary feed conveyor. The re-pulping tank which has a $10m^3$ capacity and agitator slurries the concentrate, after which it reports to the cyclone pack via a pulping tank pump, cyclone feed tank and centrifugal cyclone feed pump. The underflow of the cyclone reports to a HIG 125 kW stirred mill which grinds the concentrate to a p_{80} of 20µm.

The overflow of the cyclone reports to the existing feed thickener.

17.4.2.2 Area 900 – Carbon-in-Leach and Absorption, Desorption and Refining

The CIL facility has the capacity to treat an average of 65,000 tpa of sulphide concentrate produced by the Flotation Plant.

The facility utilizes conventional CIL technology and consists of the following stages:

- Pre-leach thickening;
- Leaching and carbon-in-leach (CIL) adsorption of gold onto activated carbon;
- Recovery of loaded carbon, elution, and electrowinning of gold and silver from the pregnant eluate;
- Regeneration of activated carbon;
- Calcining and smelting into doré;
- Cyanide destruction;
- Tailings thickening and disposal to the TMF.

The leach circuit comprises a pre-leach thickener and a CIL leach and carbon adsorption circuit comprising a single leach tank, followed by five stages of leaching and adsorption.

The lamellae-type thickener increases the pulp density of the slurry prior to leaching to approximately 45 to 50% (w/w) solids.

Flocculant is pumped to the thickener feed-box and feed-well (multi-point dosing) to assist in settling of the solids. Dilution water is added to the flocculant before addition to the thickener feed. A bed-level measuring device monitors the thickener bed depth. Flocculant dose rate is controlled according to the bed depth.

Thickened underflow is pumped by one of two thickener underflow pumps (duty and standby) which are controlled to maintain a constant thickener-bed pressure. Clarified thickener overflow gravitates to a hopper and is pumped to the process water transfer tank and thence to the process water tank.

Thickener underflow reports to the leach tank. An automatic cross flow slurry sampler will be provided to collect a sample for metallurgical accounting purposes. An in-line density meter and flow meter provides mass flow indication to the leach tank.

Lime is initially added to the leach feed pulp to provide protective alkalinity, followed by dosing with sodium cyanide solution to leach the gold and silver. A free sodium cyanide level of one g/l will be maintained throughout the leach train via the staged addition of cyanide. Mixing of the slurry is carried out by agitators fitted with dual axial flow impellers with a hollow shaft for air addition. Air is also sparged into the leach tanks and first adsorption tanks to maintain a dissolved oxygen concentration of 6-8ppm.

The leach adsorption circuit comprises of a single 260 m^3 leach tank and five 260 m^3 adsorption tanks. The leach time is 12 hours at a design slurry density of 45% to 50% solids. Each adsorption tank is fitted with a vertical mechanically wiped inter-stage screen to retain the carbon in the tank. The design carbon concentration in the adsorption tanks will be 25 to 30 g/l. Air-lifts are used for intermittent, counter-current carbon transfer. A recessed impeller pump is used to recover loaded carbon from the first adsorption tank. The loaded carbon pump discharges onto a 1.2 m wide by 1.8 m long loaded carbon screen for dewatering. Water sprays located above the screen assist with washing the slurry off the carbon. Clean loaded carbon gravitates from the loaded carbon screen into the loaded carbon hopper.

Barren carbon from the elution circuit or the regeneration kiln, or fresh conditioned carbon, is returned to the adsorption circuit CIL tank 5 or 6 after dewatering on a sieve bend.

Leach slurry from the last adsorption tank discharges onto the carbon safety screen to recover any fine or misplaced carbon from the tailings slurry. The carbon safety screen is a 1.2 m wide by 1.8 m long horizontal vibrating screen fitted with 0.65 mm aperture screen panels. The safety screen underflow reports to the tailings pump-box. Two CIL tailings pumps are installed (one duty, one standby) and pump the CIL tailings to the cyanide detoxification circuit prior to being pumped to the Gorubso TMF.

There is a sump pump installed in the leach area to return any spillage back to the leach circuit.

Elution and Gold Recovery

Elution of the loaded carbon is carried out utilising a one tonne capacity pressure Zadra circuit with a high-pressure column for the elution process.

Loaded carbon will first be acid washed to remove salts from the carbon prior to elution. The carbon will be washed with a solution containing 3% hydrochloric acid before being rinsed with raw water. Concentrated hydrochloric acid (32%) will be injected into the water line feeding the bottom of the column and mixed with raw water to attain a diluted strength of 3% acid. An acid soak time may be specified in the elution control system if required.

After acid washing, the carbon will be rinsed for two hours using raw water and will then be allowed to drain. Acid solution and rinse solution will be neutralized in a tank with caustic solution to slightly alkaline condition and will then be pumped to the pre-leach thickener.

The elution process is conducted with a solution of 0.2% cyanide and 2% caustic soda at 120˚ and at a pressure 300 kPa in the elution column. Fresh water from the potable water plant will be used to make up the eluate solution. A diesel-fired oil heater and two heat exchangers are used for elution solution heating. Eluate leaving the elution column reports to a flash vessel where it will boil and lose temperature to below boiling prior to reporting to the single electro-winning cell to recover the gold and silver from solution. Barren eluate from the electro-winning cell reports to the eluate tank and will be reheated to temperature and recycled through the column. A typical elution cycle is approximately 12 hours.

The gold and silver eluted from the carbon will be recovered onto steel wool cathodes in the electrowinning cells. The steel wool cathodes and metal sludge is collected in trays and is then calcined and smelted into doré bars. Periodically, cathodes containing precious metal are manually removed from the electrowinning cells and transferred to a calcining oven. The cathodes are calcined (overnight) at 800 ˚C to oxidize the remaining steel wool.

Barren eluate will be pumped to the leach tank splitter box.

The calcined cathode material is then mixed with fluxes and smelted in a diesel fired tilting furnace to produce doré ingots. Slag is returned to the ball mill feed. Exhaust gases from the electrowinning cells, drying oven, are discharged to atmosphere. A bag house will be installed to collect dust and fumes from the smelting furnace.

Provision has been made in the design for a cold cyanide solution wash of the carbon to remove copper (in addition to an acid wash) prior to the elution step. This is done to enable the elution of carbon from an external source.

Eluted barren carbon is transferred hydraulically to either the regeneration kiln for reactivation, or to the dewatering screen located above the final CIL tank and then back into the CIL circuit.

Eluted carbon is reactivated utilising a diesel fired horizontal rotating kiln operating at 750 ˚C. The regeneration kiln capacity will be 50 kg of carbon per hour, or about 20 hours to regenerate a batch of carbon. Regenerated carbon from the kiln reports to a quench tank and will then be transferred hydraulically to the dewatering screen above the last CIL tank.

There is provision to add new carbon to the circuit via the regeneration kiln quench tank system.

The gold room includes security-controlled access and electronic surveillance equipment, a vault and associated equipment for handling and weighing the gold and silver doré bars.

Sump pumps located in the elution area and gold room, pump clean up water back to the mill or to the leach feed.

Detoxification Circuit and Tailings Disposal

Prior to disposal of the tailings, there is a cyanide destruction stage to reduce the WAD cyanide levels to permissible levels (<50 ppm) in the TMF. Detoxification is carried out using the INCO process by the addition of sodium meta-bisulphite (to produce sulphur dioxide), air, and copper sulphate (as an activator).

The discharge from the detoxification circuit is then pumped to the TMF.

Reagents

The following reagents are used at the Gorubso plant:

- Quicklime
- Sodium cyanide
- Hydrochloric acid
- Sodium hydroxide
- Flocculant
- Activated carbon
- Gold Room Fluxes.

<u>Quicklime</u>

Quicklime slurry will be supplied from local suppliers and will be distributed via a ring main to the mill feed and leach circuit. Lime slurry dosing is carried out by automated solenoid valves to attain a set point pH. The pH is monitored at various points in the leach circuit via the control system.

Provision will be made to add lime slurry to the following three points in the circuit:

- Stirred mill feed
- Leach tank 1
- CIL tank 2.

A sump pump is located in the reagent mixing areas to permit spillage return to the process.

Sodium Cyanide

Cyanide is supplied in one tonne bulk bags and manually mixed with raw or clean water to a 20% w/v solution in an agitated mixing tank. One bag per mix is used. A dust hood and exhaust fan is provided for personnel protection.

Cyanide solution is transferred to a storage tank for distribution and use. Cyanide solution is distributed via a pumped ring main to the leach circuit, the eluate tank, and to the cold cyanide wash tank. Two distribution pumps (one duty, one standby) have been installed. Cyanide solution dosing to the leach circuit is carried out using needle valves to attain a set flow rate and thus concentration. Rotameters are installed in the main dosing point lines to measure the cyanide solution flow. Cyanide strength in the circuit will be monitored by periodic titration of the solution in the slurry. A titration hut is located on the leach tank floor for this purpose.

Cyanide is added to the following two points in the circuit from the ring main:

- Leach tank 1
- CIL tank 1.

Hydrochloric Acid

Hydrochloric acid is supplied in 200 l or 1,000 l intermediate bulk containers (IBC) and dosed neat via a dedicated dosing pump to the elution column.

A dedicated sump pump will return spillage back to the leach tailings hopper.

Sodium Hydroxide (Caustic Soda)

Caustic soda is supplied in bags and manually mixed with raw/clean water to a 20% w/v solution in an agitated tank. A dust hood and exhaust fan will be provided for personnel protection. Caustic soda solution is dosed via dedicated pumps to the eluate tank. Caustic solution is added to the cyanide mixing tank prior to mixing utilising the eluate tank dosing pump.

Flocculant

Flocculant is supplied in bags and automatically mixed with raw/clean water to a 0.25% w/v solution in a dedicated flocculant mixing plant. The flocculant will be loaded manually into the hopper of the mixing plant.

Flocculant is dosed to the pre-leach and tailings thickeners via dedicated dosing pumps (one duty, one standby).

Activated Carbon

Activated carbon is supplied in one tonne bulk bags and added to the CIL circuit. New carbon is pre-conditioned to remove any carbon fines.

Diesel

Diesel is supplied from bulk tanker and pumped into a bunded storage tank. Diesel will be distributed to the elution circuit and regeneration kiln via two dedicated distribution pumps (one duty, one standby).

Gold Room Smelting Fluxes

The smelting flux constituents are:

- Borax
- Soda ash
- Nitre
- Silica flour.

The reagents are delivered to the Central Plant and stored in a warehouse prior to use.

Water Services and Reticulation

Raw Water

Raw water is sourced from a borehole. Duty and standby pumps draw from the base of the raw water tank to feed the:

- Reagent mixing area
- Gold room requirements
- Gland water supply
- Fire water tank.

Clean water from the clean water system will be recycled back to the raw water tank.

Process Water

Process water is recovered from the process via the pre-leach thickener and the tailings thickener, as well as return water from the TMF. The water is stored in the process water tank. Duty and standby pumps will draw water from the base of the process water tank to feed process water to the following:

- Re-pulping area
- Fine milling and classification circuit
- Leach circuit
- Clean-up hose points.

Fire Water

A separate fire water tank receives feed water from the raw water system. Reticulation to the distribution points is via a dedicated fire water pump with a back-up diesel powered pump. Line pressure in the fire water system is maintained by an electric jockey pump.

Fire hydrants and hose reels are placed throughout the process plant, fuel storage and plant offices at intervals that ensure coverage in areas where flammable materials are present.

Potable Water

Potable water is supplied from the local water supply.

Pumps draw water from the potable water tank to service the following:

- Administration complex
- Plant offices
- Control room
- Site laboratory
- Ablution facilities
- Safety showers
- Elution circuit.

Gland Water

Gland water pumps (duty and standby) draw water from the raw water tank and supply the following slurry pumps with gland seal water:

- Cyclone feed pumps
- Leach feed pumps
- CIL tails pumps
- Thickener underflow pumps.

17.5 Consumable Material Requirements

All material requirements for the treatment of the Rozino ore through the Flotation Plant and the Central Plant are estimated values based metallurgical testing, the mine plan and other relevant data developed for the PFS.

17.5.1 Flotation Plant

17.5.1.1 Power

The expected power consumption at Rozino is dependent on the proportion of ore types processed as they have different comminution properties. Oxide and Transitional ore types are expected to consume 20.9 kWhr/t of power and the Sulphide ore type 22.6 kWhr/t. This translates to 36.6 MWhr per annum for Oxide or Transitional ore and 39.5 MWhr for Sulphide ore with the life of mine average being 39.0 MWhr per annum.

17.5.1.2 Water

The expected plant water consumption (imported make-up water) is expected to be approximately 730 kl per annum. Losses are predominately via the tailings. This consumption is made up from on-site water inflows by precipitation and groundwater inflows. The net import of fresh water from external sources for the entire operation will be between 125,000 m^3 and 310,000 m^3 per annum depending on the amount of precipitation (see Section 18).

17.5.1.3 Reagents

Reagent consumption is dependent on ore type. The reagent consumption for the respective ores is shown in Table 17-4 and Table 17-5. The ore types in the mine plan assume that no separation of the ores is required based on the performance of mixed ore types in the variability testing for recovery performance. The reagent consumptions shown are for singular ore types on an annual basis but the actual consumption is predicted to be proportional to the mix of ore types in the feed. Further testing to confirm and refine chemical consumptions on mixed ore types is recommended in the Feasibility Study.

Table 17-4: Flotation Plant Oxide/Transitional Ore annual reagent consumption

Reagent	Total consumed (tonnes/year)	Form of supply
Frother MIBC	27	Liquid
PAX (potassium amyl xanthate)	149	Powder
Collector A404	26	Liquid
Flocculant	53	Dry flakes
NaSH	875	Flakes
Copper Sulphate	525	Solids in bags

Table 17-5: Flotation Plant Sulphide Ore annual reagent consumption

Reagent	Total consumed (tonnes/year)	Form of supply
Frother MIBC	44	Liquid
PAX (potassium amyl xanthate)	61	Powder
Flocculant	53	Dry flakes

17.5.1.4 Wear Steel (balls and liners)

The major wear-steel component consumption is shown in Table 17-6. The consumption rates assume a full year of processing the respective ore types although the ore types will be delivered as a mixed material so actual consumptions will vary between these extremes. The ball wear estimates are based on modelling from the PFS abrasion and ore hardness data derived from the PFS testwork. The liner wear rates are estimated as a function of ball wear rates.

Table 17-6: Flotation Plant wear steel consumption

Area	Item	Steel consumption tonne/year	
		Oxide	Sulphide
Primary Crusher	Liners	10	11
Secondary Cone Crusher	Liner	11	13
Tertiary Cone Crusher	Liners	12	13
Ball Mill	Balls	1,544	1,715
Ball Mill	Liners	47	53
Chutes	Liners	3	4

17.5.2 Central Plant

Gorubso are the operators of the Central Plant and will be fully responsible for the processing of the concentrate. However, the recovery characteristics predicted in this report are based on certain reagent consumptions and therefore they are described here for full transparency.

17.5.2.1 Power

The expected power consumption based on the installed and designed plant motor capacity and duties for the concentrate processed (for an average 65,000 tpa), is estimated to be 2,720 MWh /a.

17.5.2.2 Water

The expected net water consumption is expected to be 84,000 m^3 per annum

17.5.2.3 Reagents

The expected reagent consumption is based on the PFS metallurgical testwork and current plant data. An average annual throughput rate of 65,000 t at an average gold grade of 30 g/t is anticipated. Consumptions are shown in Table 17-7.

Table 17-7: Central Plant annual reagent consumption

Reagent	Total consumed (tonnes/year)	Form of supply
Copper Sulphate	24	Bagged powder
Sodium Hydroxide	6	Liquid
Quicklime	236	Bagged powder
Sodium Cyanide	776	Dry flakes
Flocculant	3	Bags of solids
Gold Room: Borax	1	Bagged solids
Gold Room: Silica	1	Bagged solids
Gold Room: Nitre	2	Bagged solids
Gold Room: Soda Ash	2	Bagged solids
Gold Room: Sulphuric Acid	29	Liquid
Gold Room: Anti-scalant	2	Bagged solids

17.5.2.4 Regrind Media

The major wear component consumption at the Central Plant will be regrind media, with an expected consumption rate of 39 tpa based on PFS metallurgical testwork information and a concentrate throughput rate averaging 65,000 tpa.

18 Project Infrastructure

18.1 Introduction

In terms of infrastructure, the Rozino deposit is a greenfield mining prospect and no infrastructure exists at the proposed mine site. The site is currently accessed from the main provincial road II-59 via a 12 km single-lane, paved road and closest to the Project, a 2 km unsealed dirt road. Power will be supplied by a dedicated 110 kV overhead power line that will be constructed and connected to the Madzharovo substation 23 km to the north. Water to augment the anticipated shortfall in supply from on-site sources and recycling will be sourced externally. The village of Rozino, now largely abandoned, is located one km to the north of the project site.

Site structures, including the enclosures for the comminution and flotation plant, will be appropriate for the relatively short mine life of about seven years. Other design and engineering criteria applied to infrastructure location was to minimize the distance and elevation differences between the pit, Flotation Plant and TMF. The compact project footprint reflects the constraints imposed by environmental considerations.

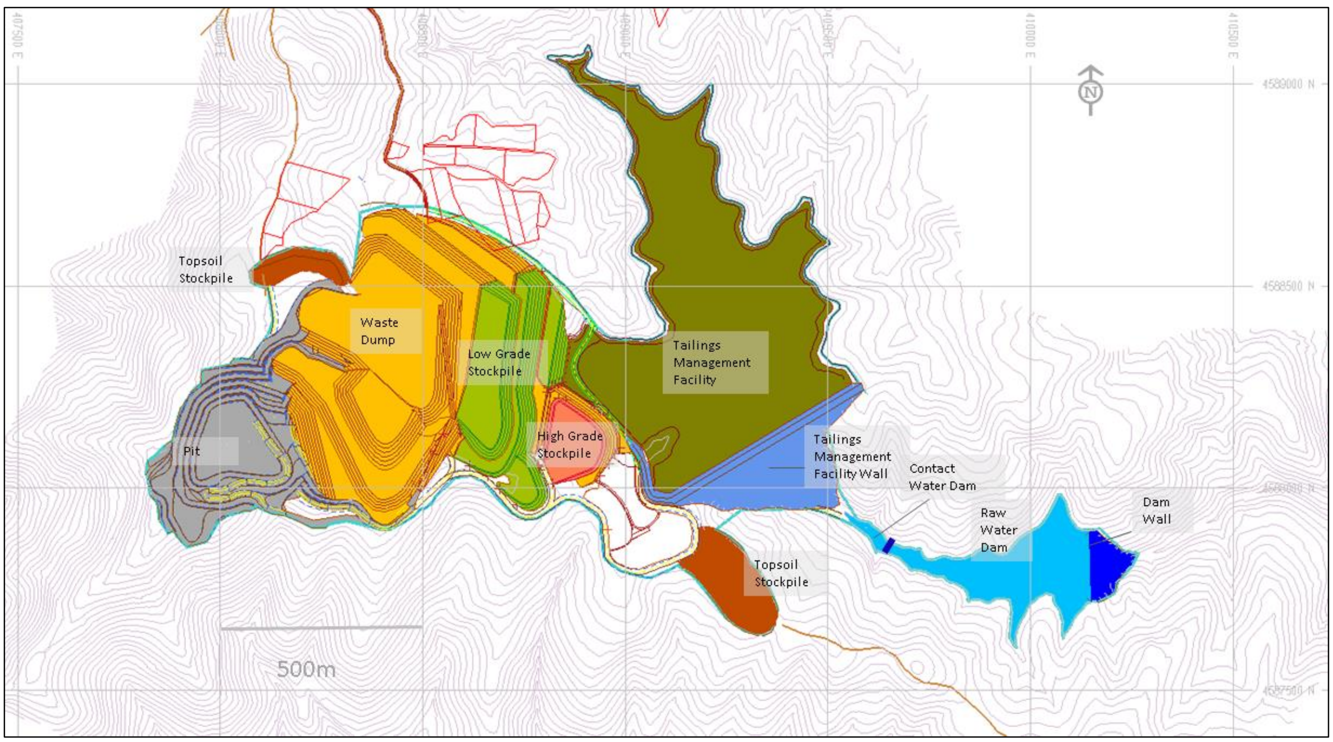


Figure 18-1: Proposed placement of key infrastructure at Rozino at the end of Year 5

18.2 Access Road

The primary road accessing the project site from the main provincial road, the II-59 linking Krumovgrad and Ivaylovgrad, is a 12 km paved municipal road that will require upgrading to accept anticipated mine traffic during construction and operations. A capital cost evaluation was completed by MGU (University of Mining and Geology, Sofia) for Velocity that recommended the thickness of the asphalt surface be increased (to approximately 0.30 m), improvements to drainage and culverts, and the installation of additional signage. Additionally, and because of the road width of 4 m, a number of pull-outs will be constructed and an improved road shoulder will be constructed along the entire length of the road. The existing 2 km road linking the project site to the municipal paved road is a gravel road in poor repair. This will remain a gravel road but will be upgraded to a standard that will accommodate typical mine traffic. The location of the Flotation Plant and site facilities allows for the safe separation of heavy mining equipment and normal site vehicles.

Allowance for maintenance of the 14 km access road over the life od the operation was included in the operating costs.

Figure 18-2: Existing sealed access road between II-59 and the Rozino Project site
Source: Velocity, 2020

18.3 Power Supply

Electrical power to the Rozino site will be supplied from the Madzharovo substation by a 23 km 110 kV overhead high voltage transmission line. Power to the Madzharovo substation is supplied by the 85 MW Studen Kladenets and the 217 MW Ivaylovgrad hydroelectric power stations (Figure 18-3). A substation will be located at the plant site to facilitate power distribution to various areas across the site, but mainly to the plant. The transmission line is designed with a peak load capacity of 10 MW. Average consumption for the Rozino site is estimated at 4.5 MW.

A preliminary technical, regulatory, and financial evaluation of the construction of the 110 kV power line and associated infrastructure was undertaken by Nikolay Savov, a Bulgarian qualified professional in this field. Discussions by him with ESO ED, the owner and operator of the substation, indicate that the Madzharovo substation has sufficient spare capacity to supply Rozino. Indications are that electricity supply is stable and reliable, with a greater than 95% supply reliability. Added reliability is the fact that the Madzharovo substation is supplied by two hydroelectric facilities.

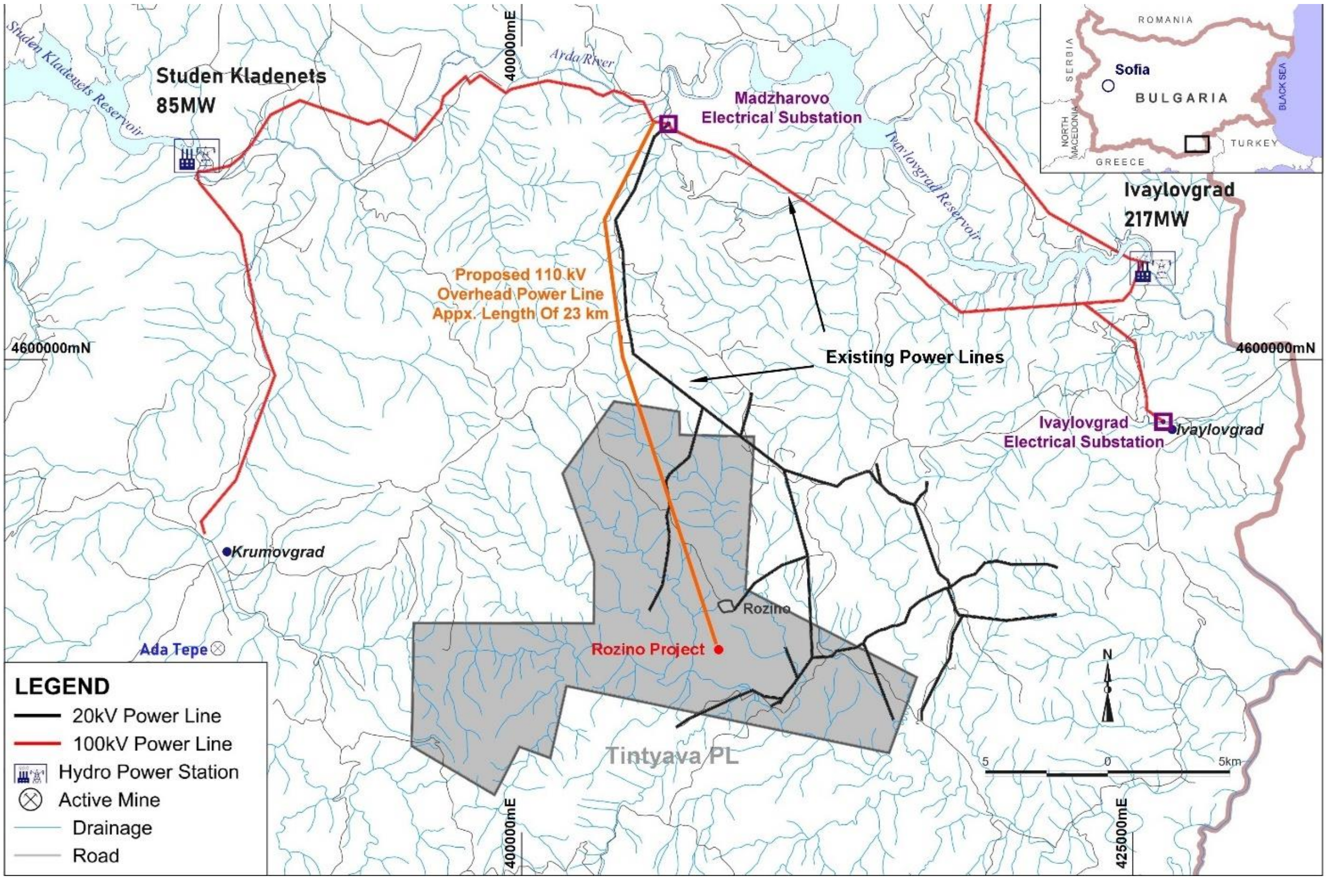


Figure 18-3: Local power distribution network
Source: Velocity, 2020

An evaluation of the various options for supplying electrical power to the site was undertaken in order to ensure that the optimal supply option for a project with a seven-year life was selected. These included:

- Powerline vs LNG/diesel gensets. Despite the 23 km length of the powerline, and seven-year mine life, the powerline option provided a significantly more economical solution than gensets. Furthermore, transportation of fuel for gensets adds an element of risk with respect fuel supply certainty and public safety due to increased road traffic.
- Sourcing power from the following substations was considered:
    - Ivaylovgrad substation, 23 km from the Rozino Project site
    - Madzharovo substation, 23 km from the Rozino Project site
    - Ivaylovgrad Dam substation, 30 km from the Rozino Project site.

Terrain was an important consideration in deciding the route and ultimately the connection to the Madzharovo substation proved the most favourable.

- ESO ED advised that an existing 110 kV powerline would not be available to the project due to existing user agreements and government regulations.
- A further consideration was whether to construct a 20 kV powerline or a 110 kV powerline. Despite the lower construction cost of the 20 kV powerline, power costs were higher due to a lower energy efficiency. Over the life of the mine, the 110 kV powerline proved to be the most economical option. This will be further evaluated during the Feasibility Study.

The planning, design, construction, and integration of the electrical infrastructure required to supply Rozino will be in conjunction with and managed by ESO ED or EVN. The overhead powerline will have the following design:

- Galvanized steel pylons with concrete footings,
- Transmission lines of aluminium-steel with a cross-sectional area of 185 mm^2,
- Pylons spaced at approximately 250 m,
- Easement total width of 20 m,
- A 10 MVA substation situated at the Rozino plant.

Figure 18-4 shows a 110 kV pylon of the type envisaged for the Rozino project, as well as views of the Madzharovo substation.

Figure 18-4: Pylon to support a 110 kV powerline and the Madzharovo substation
Source: Velocity, 2020

It is anticipated that the powerline will be routed through predominantly state-owned land which will ensure a more efficient easement process and a shorter regulatory timeframe. The procedure for the investigation, coordination, permitting, construction, and commissioning of the powerline will take approximately 24 months.

18.4 Water Management

The Project water management plan is central to maintaining an appropriate environmental and operational performance for the Project. The principle adopted for site water management is to intercept and control contact water flowing within the operational areas to ensure that it stays within the catchment area located to the east of the mine operations. This contact water will report to the tailings management facility (TMF) and the contact water dam (CWD) located directly below the TMF. The water will then be pumped back to the water storage tanks located at the processing facility for use in the process plant and mining operation. Water sourced from external sources will be pumped to the raw water dam (RWD) located below the CWD in the first instance.

Velocity appointed Golder Associates (UK) to undertake water modelling, design the necessary water management and storage facilities, and develop a water balance model. The proposed CWD will have a capacity of about 25,000 m^3 and that of the RWD approximately 350,000 m^3 (approximately 75% of the annual plant consumption at a usage rate of 83.1 t/h or 486,000 m^3/a). A hydrological study indicated that up to 310,000 m^3/a of surface runoff and groundwater inflows may be expected. Considering this, it is anticipated the Project will have a negative water balance on an annual basis and will require additional sources of make-up water to supplement the groundwater and surface runoff quantities. While water reuse will be maximized, water balance models for drier than average, average, and wetter than average rainfall scenarios indicate that between 125 Ml and 310 Ml per annum will need to be imported from external sources.

It is planned that make-up water will be sourced from the Arpa dere either directly from the river of by a well field located in the valley alluvial sediments (see below).

The Project water management plan has been developed to ensure minimum impact on the surrounding community users. All surface water runoff within the processing facilities and mining areas will be collected in channels (if required) or allowed to gravitate to the TMF or CWD. The groundwater and surface water reporting to the open pit will be collected in a sump and pumped into a channel that will direct water to the TMF.

Discharge of water from the water storage facilities into the environment is not expected as there is a negative water balance.

18.4.1 Surface Water Management

18.4.1.1 Surface Water Management – Site Infrastructure

Most of the proposed Project infrastructure is located within a single catchment.

The mine infrastructure is located in the upper reaches of the Uren dere, a creek with intermittent flow and only during the wet season. The pit is located on, or very near the catchment divide, and will have only minor surface water drainage and diversion requirements to prevent rainfall runoff draining into the pits. The location of the pit may also present the opportunity to gravity drain rainfall runoff along the upper benches within the pit, to discharge laterally outside the pit perimeter which would result in reduced dewatering requirements within the pit and commensurate capital and operating costs.

The location of the Flotation Plant site directly adjacent to the WRD and TMF will require contact-water trenches on the southern and western side of the pad that will drain into the CWD. Contact water cut-off trenches from the pit and the western side of the WRD will drain into the TMF until such time as the elevation of the tailings embankment exceeds that of the plant site pad elevation. At this point, at about the end of Year 4, the contact water will drain into the CWD below the TMF.

The northern portion of the WRD is not contained within the pit (as backfill) and a cut-off drain to intersect and divert rainfall (non-contact water) runoff into the Uren dere will be required. This cut-off trench will direct water directly into the creek.

A cut-off trench on the northeastern side of the TMF and CWD will direct non-contact water to either the RWD or into the creek below the RWD.

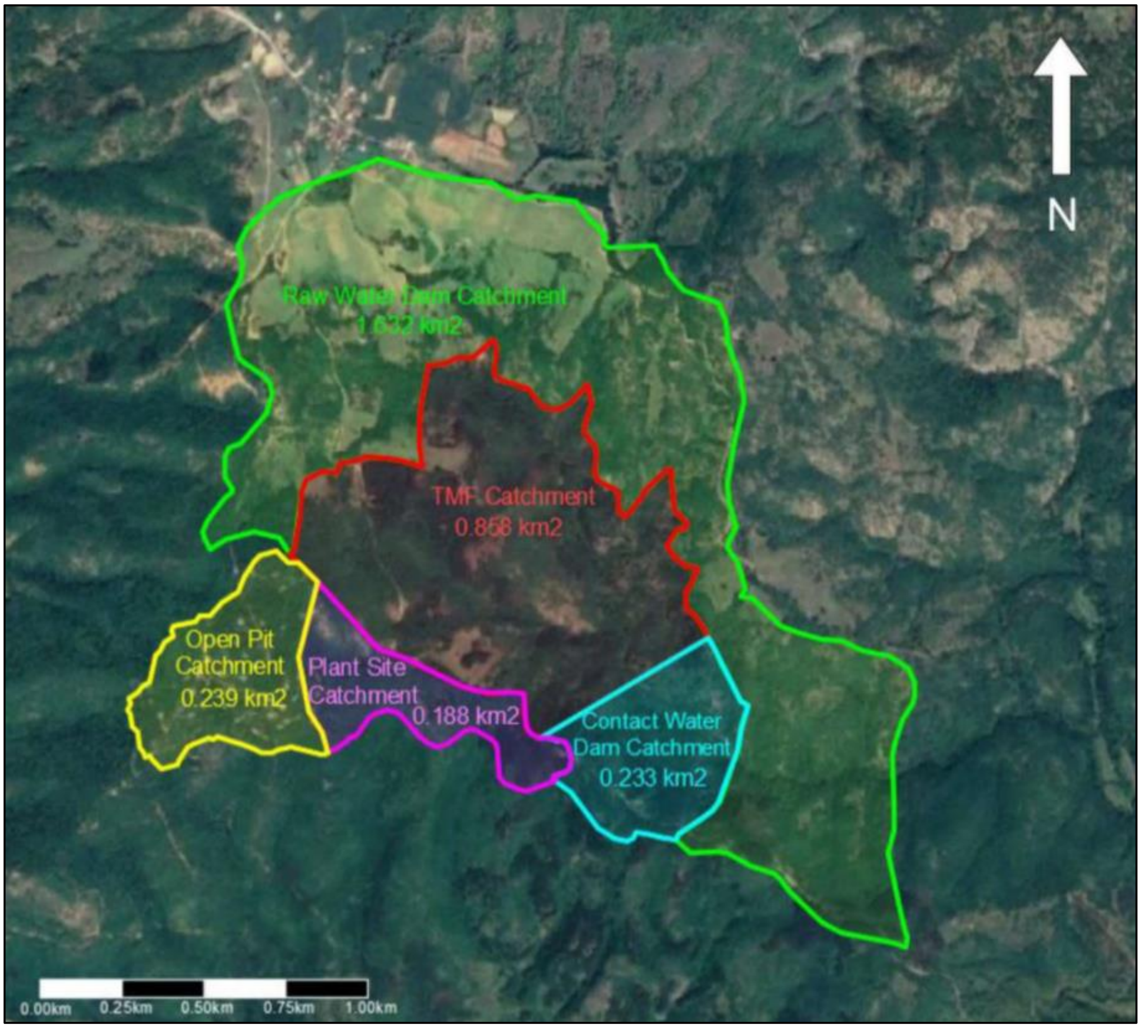


Figure 18-5: Catchment with Project infrastructure
Source: Golder, 2020

18.4.2 Water Supply

Water to compensate for the anticipated approximately 125 000 to 310 000 m^3 per annum supply deficit for the Rozino Project will be pumped from the Arpa dere (creek) near the village of Gugutka (approximately 1.7 km east of the Rozino Project site) via a 1.15 km pipeline to the raw water dam. A study carried out by PiA, a Bulgarian engineering consultancy, provides for the pumping of water directly from the Arpa dere during the wet months of the year (January through June inclusive). Adequate water storage on site will allow for an uninterrupted supply of water to the Flotation Plant and other site facilities through the remaining months. Water will also be sourced from a well located in an existing pumping station adjacent to the Arpa dere riverbed.

Abstraction from the Arpa dere will be close to the confluence with the Uren dere and adjacent to the existing Rozino village pump station mentioned above. This pump station was designed to pump water from a spring (draining into a pond) to the now abandoned Rozino village. The flow rate from this spring, depending on the season, varies between 6 and 11 l/s throughout the year. It is estimated that the flow rate required to support Rozino village with its reduced population is in the order of 0.34 l/s. Water for the Rozino Project will be abstracted from this source throughout the year, as well as the river.

It is intended to place the Rozino Project pump station adjacent to the existing pump station and use the existing 20 kV power supply. An anchored and horizontal pipe laid in the riverbed will drain into a pond and sump arrangement. The pump station will contain three pumps (two duty, one standby) designed to deliver water to the raw water dam at a rate of up to approximately 100 l/s over a distance of 1.15 km. The hydraulic head between the pump station and the pipe discharge point is 61 m. The design includes all electrical equipment (transformer, controls, flow metres, lightning protection) housed in containers laid on concrete. The pipeline will be buried to a depth of approximately one metre and will be routed adjacent to the Uren dere creek. The pipeline will contain air, non-return and safety valves.

Permitting and permissions for the abstraction of water from the Arpa dere will commence during the EIA process. Alternative water abstraction sources have been identified, including the alluvial sediments in the Byala Reyka valleys via a series of wells.

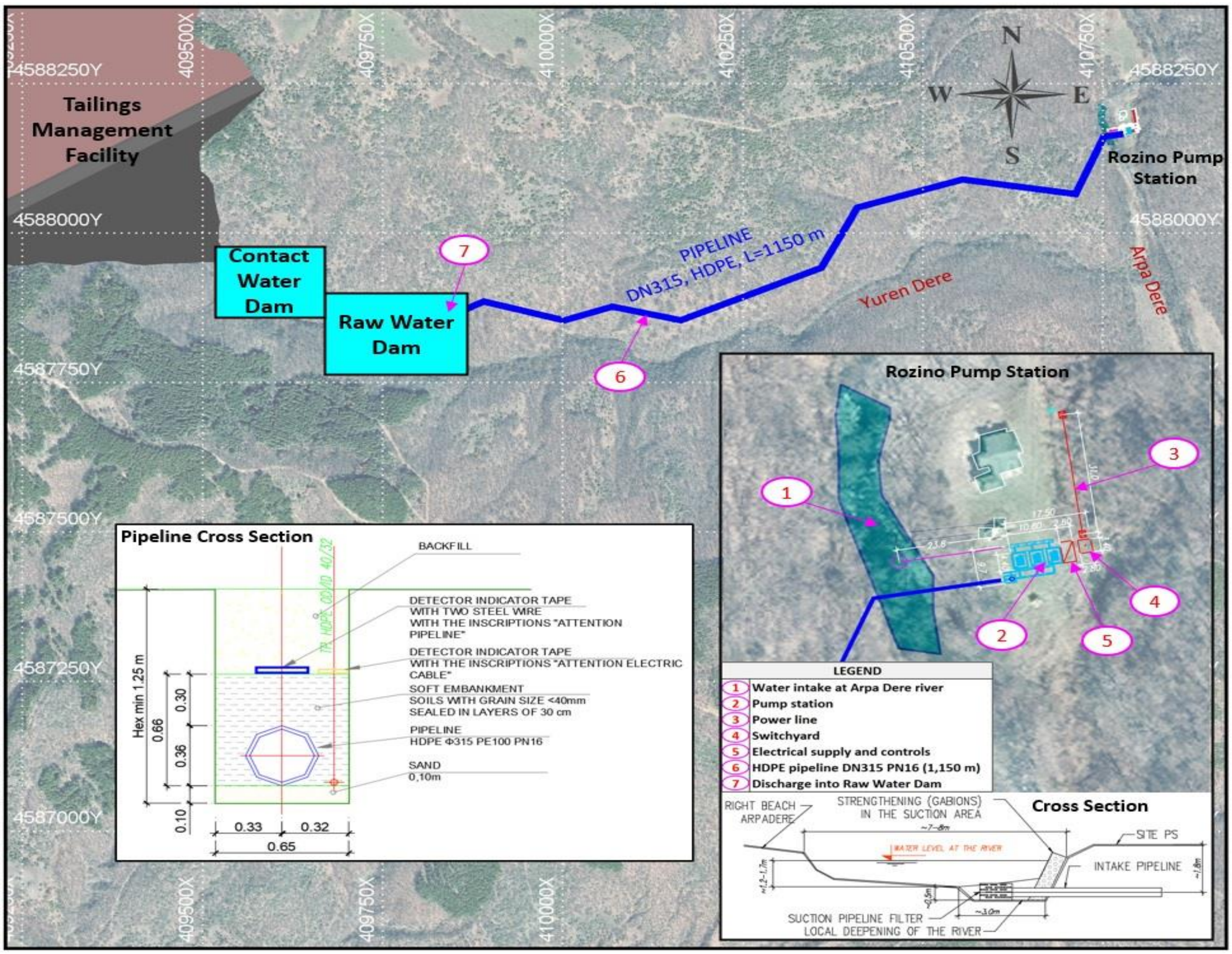


Figure 18-6: Route and infrastructure for the Rozino Project external water supply
Source: Gorubso, 2020

18.5 Waste Rock Dump

A total of 26.5 Mt of waste rock is mined over the life of the mine. Some 3.2 Mt of this waste is utilized as the TMF downstream retention wall. A further 0.4 Mt of the waste mined is utilized in the CWD and RWD walls. The remainder of the waste is stored in the WRD.

18.6 Tailings Disposal

Golder were commissioned by Tintyava Exploration AD (Tintyava) on behalf of to undertake the Pre-feasibility Study design of the Rozino Gold Project TMF. This included providing a design and costing for the facilities required for the disposal of tailings and associated seepage control. Included in the scope was the sizing and design of a CWD directly below the TMF. Downstream from the CWD is the RWD (for the storage and supply of import water (non-contact or fresh) to the Flotation Plant.

Acid-base accounting testwork under the supervision of Mineesia Ltd indicates that the tailings is neither acid generating nor is it anticipated that metal leaching will occur.

The location for the TMF was selected with the objectives of minimising the project footprint and being in close proximity to the Flotation Plant to reduce tailings and return water pumping costs. Thickened tailings disposal was selected as the most cost-effective solution compared with other options (such as paste or dry stack). A downstream raised TMF wall was selected as being the most appropriate for the seismicity of the region. Figure 18-7 shows the TMF site in the Uren dere valley), the contact water dam and the raw water dam in relation to other mining infrastructure and the pit.

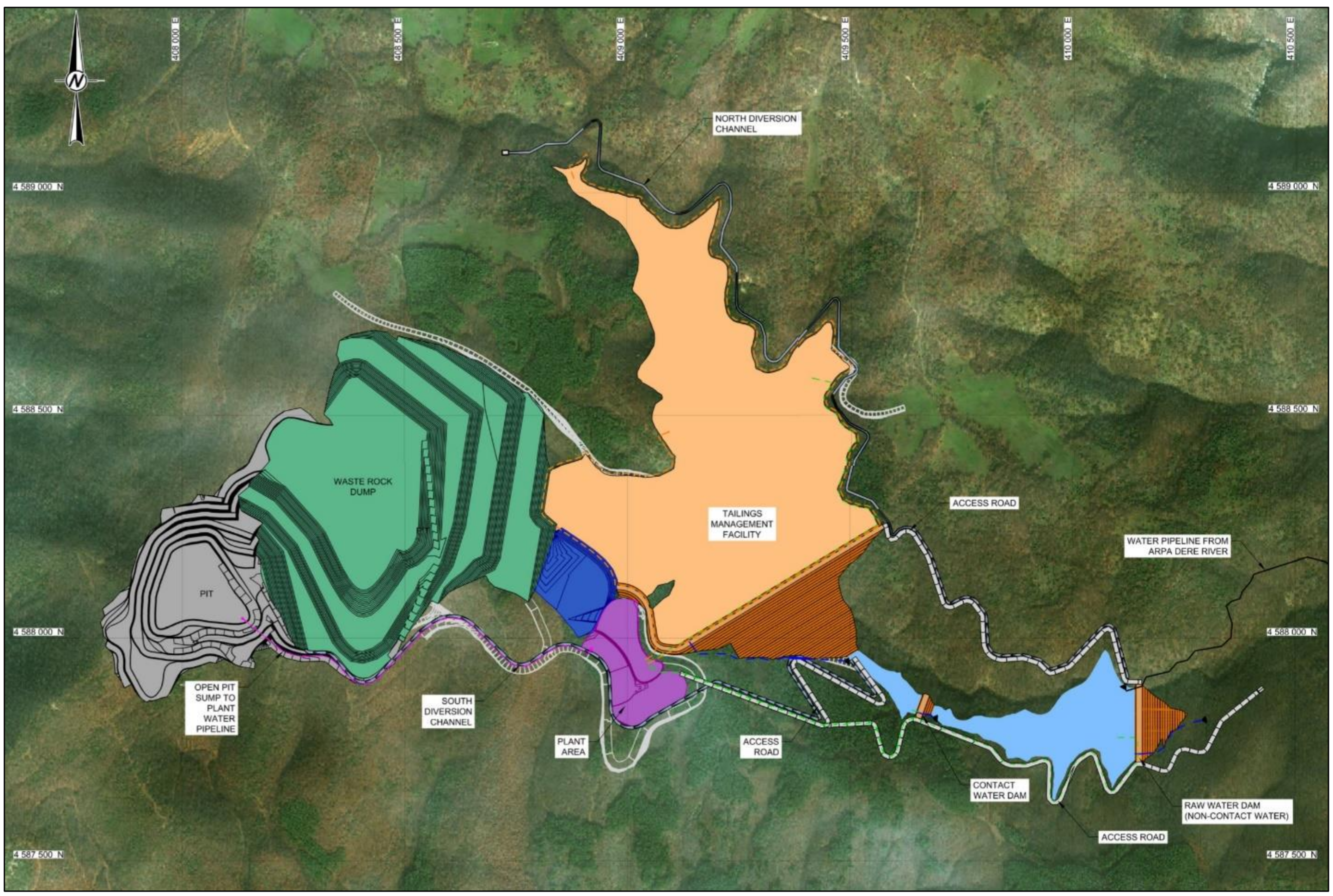


Figure 18-7: Site layout showing relationship of the pit, WRD, TMF, CWD and RWD
Source: Golder, 2020

Site geotechnical studies and laboratory testing were undertaken to characterize the embankment foundations and materials available for construction. Three main geotechnical units were identified: superficial deposits (colluvium/ alluvium), weathered rock (gneiss) and bedrock (gneiss and marble). The superficial deposits are geotechnically unsuitable for embankment foundations and will need to be removed as part of the embankment preparation. The majority of materials required for construction of the embankment will be obtained from locations within the project area, with waste rock from the open pit comprising the majority of material required. The sand for the under-liner zone will come from an external quarry source.

The region is seismically active and classifies as an MSK zone 8.5 to 9.5, with a peak ground acceleration for a return period of 1:10,000 year of 0.3 g selected for design. The seismic case is the dominant mode for stability, and this will be a focus of future work on the embankment design for stability.

Design criteria have been developed, using Canadian Dam Association (CDA) guidelines as a main reference, which Golder consider as the most applicable guidelines for tailings storage design. European regulations on mine waste focus on environmental protection requirements, but do not provide the detailed criteria for design. Other parameters for the design, including throughput and design life, have been provided by Tintyava. Parameters for the water dams, including storage volumes required were developed by Golder, as reported in "Surface Water – Stage 4 – Interpretive Analysis and Design" (Golder 2020).

The TMF has been designed to store 8.575 Mt (6.125 Mm^3) of tailings delivered over a six year period. In the sixth and seventh year, stockpiled low-grade ore will be processed and the tailings will be used to backfill the completed pit (approximately 2.6 Mt). The TMF is constructed with an initial starter wall approximately 37 m high and will be raised on an annual basis to a final height of 67 m.

The TMF cross-section consists of a downstream zone of waste rock with upstream transition zones comprising graded gravels and sands to provide a suitable substrate for an HDPE liner on the upstream face. The transition zones are required to reduce seepage and prevent piping (internal erosion) in the event of damage to the HDPE liner. The TMF is underlain with a drainage system to collect any seepage should this occur.

Tailings will be pumped from the process plant, initially from the embankment but then migrating progressively up the valley. This will be done in order to control the position of the supernatant pond from where a pump barge will return water back to the process plant.

Stability assessments for the embankments were undertaken using Slope/W software, with material parameters developed from laboratory testwork. Required factors of safety for the various stability cases (static, seismic, draw-down) were based on CDA guidelines. Results of the stability assessment indicate that a downstream slope of 1:2.8 (vertical:horizontal) is needed. A steeper slope on the upstream face, whilst acceptable from a stability perspective, increases the difficulty of the transition zone construction and liner installation and is not recommended.

18.7 Site Facilities and Services

As much infrastructure as possible is located close to the Flotation Plant in order to make the operation of the site efficient. The major infrastructure at the Rozino site is:

- ROM stockpile, tip, crusher circuit and conveyor system
- Ball milling circuit
- Flotation- plant and related facilities for dewatering concentrates and tailings materials
- Concentrate weighbridge
- Mine maintenance workshops and warehouses
- Change rooms and administration complex
- Fuel storage and distribution
- Wash bay
- Security building
- Fire protection

- Industrial and effluent waste management
- Site roads
- Communications
- Standby generator.

Figure 18-8: Site infrastructure at the end of the construction phase
Source: Halyard, 2020

18.7.1 General Note on Buildings

Infrastructure buildings are classified as either architectural, control rooms or industrial. Architectural buildings include the administration offices and ablution facilities. Control rooms include the crusher control room and the main process plant control room. Industrial buildings include workshops, stores and buildings that house process equipment.

As far as possible, the proposed buildings will be temporary and semi-mobile in nature taking into consideration the short mine life. They will be easily demobilized and salvageable at closure.

18.7.2 Plant and Site Facilities Pad

The plant site is constructed on a relatively flat ridge line that will be landscaped by excavation and filling (waste rock) to create the platform for plant and site buildings. A total of 45,000 m^3 will be excavated, of which 28,000 m^3 will be required for backfill within the plant site and the remainder will be used in other construction activities where backfill is suitable such as the WRD or the TMF or RWD, depending on material suitability.

18.7.3 ROM Stockpile, Tip, Crusher and Conveyor System

The tip and jaw/cone crusher system feeds a crushed-ore stockpile via a final 120 m long conveyor. Crushed ore is bottom reclaimed from the stockpile and feeds the ball mill located in the flotation building. These facilities are described in more detail in Section 18. Figure 18-9 shows some of the plant area infrastructure to aid in this discussion.

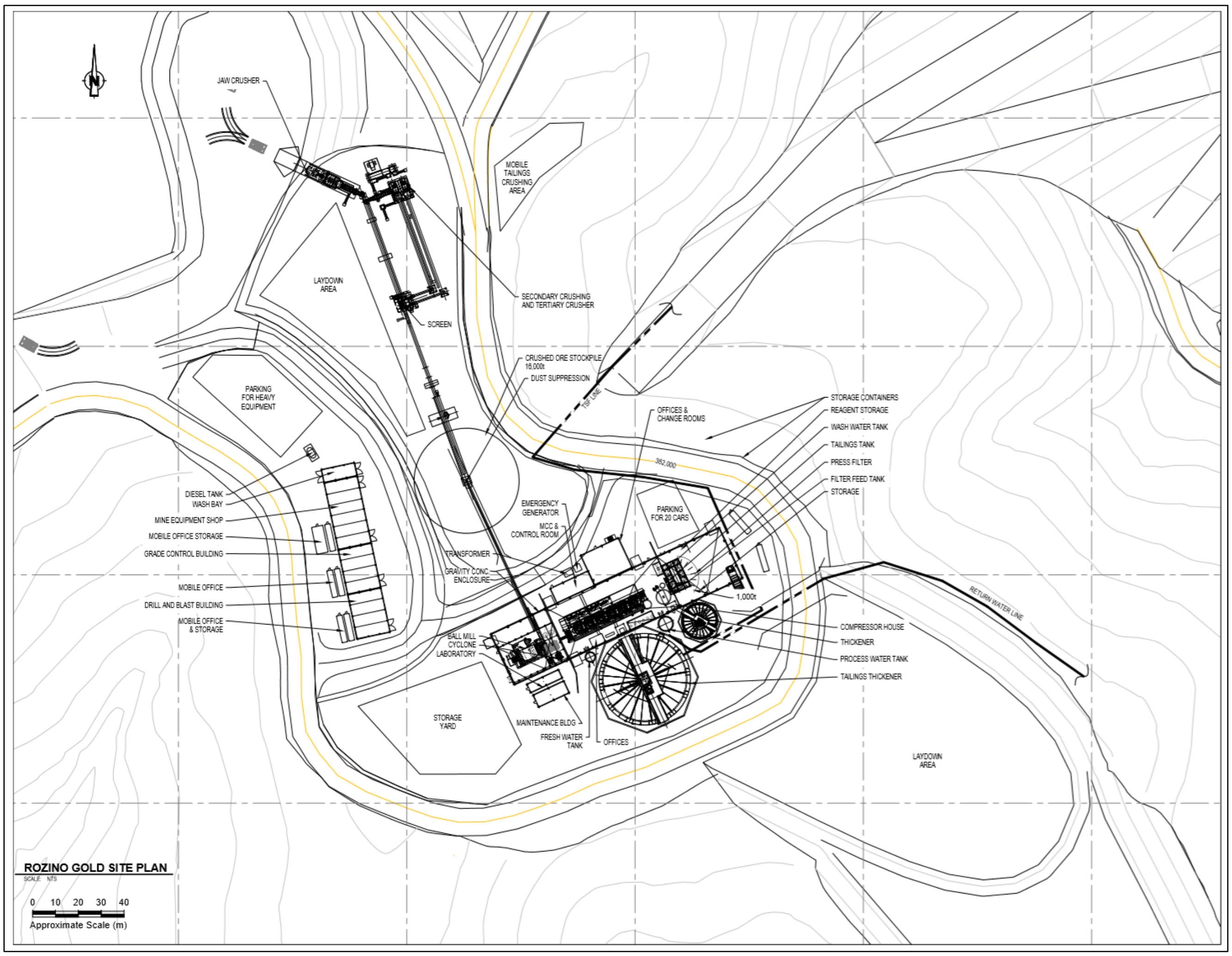


Figure 18-9: Plant area infrastructure at end of construction period
Source: Halyard, 2020

18.7.4 Flotation Plant and Related Facilities

The flotation plant is housed in a building 110 m long, 24 m wide and 32 m high. Adjacent to the flotation building are a concentrate thickener (18 m diameter) and tails thickener (42 m diameter) as well as the plant laboratory and maintenance buildings. These facilities are described in more detail in Section 18.

18.7.5 Concentrate Weighbridge

The 50t capacity concentrate weighbridge 3.1 m x 18 m will be located at the mine entrance and will have automatic data collection. It is to be located together with the security building at the mine entrance.

18.7.6 Mine Workshop and Warehouses

The mine workshop building is a 15 m x 75 m x 12 m high building divided into three work areas. The grade control drilling and drill and blast contractors will have separate work areas to the main mine fleet area. The workshop will provide sufficient area and facilities for the maintenance of the mining and ancillary fleet with 6 bays in the main workshop. The workshop offices will be housed in a portable and modular structure. The warehouse will be located in suitably renovated containers. A heavy equipment parking area is located alongside the workshops.

18.7.7 Change Rooms and Administration Complex

These will be pre-engineered structures:

- Plant offices and change rooms: 14 m x 18 m x 3.5 m
- Administration: 14 m x 18 m x 3.5 m
- Crib and training facility: 15 m x 20 m x 3.5 m

A furniture allocation is included.

18.7.8 Fuel Storage and Distribution

Diesel to supply fuel to light vehicles, the mining fleet and mobile plant and equipment will be stored in a skid mounted diesel tank. The tank will be double walled with an interstitial vacuum monitoring system to confirm containment integrity. The working volume of the tank will be 125,000 l or about two weeks of production. Fuel will be dispensed by means of an electric pump and an automatic shut-off dispensing nozzle. All fuel dispensed will be monitored with a flowmeter showing both a total value and an event value. A safety shutoff valve system will be employed that will shut off the fuel in case of fire. The shutoff valves will be pre-loaded with a tensioned wire system complete with fusible links that will melt under high temperatures to activate the valves.

All fuel required at the mine site will be delivered in tanker trucks by commercial suppliers. The fuel storage area will be bunded to prevent spillage and contamination. Minor quantities of petrol that may be required will be obtained from local fuel distributors. A specific high-volume foam fire extinguisher unit is included in this area.

18.7.9 Wash Bay

A vehicle washdown facility will be located adjacent to the diesel refuelling area. It will comprise a bunded concrete slab sloping to a settling sump. Captured rainfall and diesel spillage from the adjacent diesel refuelling facility will also be directed to this sump. A sump pump will transfer dirty water to an oil/water separator. Contaminated water will be disposed of in accordance with applicable regulations.

18.7.10 Security

All persons entering the mine will be pass through a continuously manned boom gate on the site access road and adjacent to the administration building (4.5 m x 14 m x 3.5 m). Security personnel will monitor and control all vehicles and personnel entering and leaving the site. A site security camera and monitor system are allocated.

A mesh fence will be constructed around the plant and mine service surface facilities. Additional security fencing with lockable access gates will be installed locally around the remote pumping facilities.

Fencing will be installed around the warehouse yard and any other facilities requiring additional security.

18.7.11 Fire Protection

Fire protection in the form of fire hydrants, fire-hose reel cabinets and fire extinguishers will be installed and placed in strategic locations around the facilities in accordance with operational requirements and the relevant regulations. Firefighting water will be supplied from a dedicated water reservoir. Water will be gravity-fed to fire water pumps at the process plant. Jockey, duty and diesel-powered standby pumps will be provided. Various types of fire extinguishers will be provided in areas where water as a means of fire control is unsuitable (electrical facilities such as motor control centres and control rooms). A specific foam fire extinguishing unit is included near to the diesel storage and mine workshop area.

18.7.12 Explosive Storage

Explosives will be delivered on an as-needed basis by the blasting contractor thus eliminating the need for on-site storage.

18.7.13 Industrial and Effluent Waste Management

Sewage from the various plant and mine site buildings will be processed on-site via a waste-water treatment system and treated waters will be cycled to the contact water system. Waste such as hydrocarbons from equipment maintenance and chemical waste from the laboratory will be collected and stored for collection by contractors who will remove this from the site and dispose of it in accordance with the applicable regulations. Office waste and general waste will be collected by a cleaning contractor who will dispose of the waste materials in a solid municipal landfill site.

18.7.14 Site Roads

A 600 m long haul road will connect the exit point of the mine pit to the plant and site facilities area. New unsealed roads and upgraded existing exploration tracks (approximately 2.5 km) will be required to be constructed from the main pit entrance to the TMF wall and water storage facilities wall to allow for their construction.

18.7.15 Communications

The mine site will be linked to the nearest accessible internet using a fibre-optic cable which will support both data and voice communications. A repeater system will provide the infrastructure to enable handheld and mobile radio sets to communicate around the site.

18.7.16 Standby Generator

A 250 kVA, 400 V standby generator is included to ensure emergency power capability in case of loss of line power. Emergency power is provided through a dedicated switchboard for emergency lighting and services only.

19 Market Studies and Contracts

19.1 Market Studies

All doré will be sold through refineries based in Bulgaria or Europe. The relatively small size of sales compared to market demand give no concern to impacts of sales on the metal price. No forward sales are considered. The Qualified Person has reviewed the market studies supplied by Velocity and the results support the assumptions of this Technical Report.

19.1.1 Metal Price

Velocity evaluated market price forecasts for gold using publications and opinion provided by UBS and Haywood Securities Inc.

The metal price selected for the financial analysis was $1,500 per oz. This price is 6.8% above the three-year rolling average of $1,404 per oz (data source: World Gold Council) as of 30 August 2020. The gold spot price on the same day was $1,957 per oz (data source World Gold Council).

19.1.2 Metal Sales Costs

CSA Global completed a market study to determine the terms of sale of doré to smelters in Europe. The quality of the doré was determined through the estimation of the principal metals in the concentrate leachate and predictions of copper cold stripping efficiencies from the PFS metallurgical tests. The possible doré quality was estimated to be 49% gold, 42% silver, 4% copper and 5% other non-payable material (see Section 15.2.18 for further details). However, the terms considered for doré sales terms apply to doré bars with up to 60% gold content. The relative contents of gold, silver, and copper in the doré also depend on the mix of head grades and relative metal recoveries which will vary over the life of the project.

Four smelters were surveyed for metal pay ability, treatment costs, transport, insurance, and refining costs for doré with 50-60% gold content.

A fifth smelter in Bulgaria was considered but negotiations on terms are too early in the process to determine the appropriate terms of doré metal pay ability and sales costs.

Based on the average of the four refineries for doré with 60% gold content, total sales costs were estimated at $8.97 per oz of payable gold. The payable proportion of gold was estimated at 99.8%. No value was given to silver and no penalty was attributed to the content of copper.

No contractual arrangements for shipping or refining exist at this time.

19.2 Contracts Material to the Issuer

Several contracts for construction of the plant, equipment leasing, and infrastructure will be required for the construction and operation of the Rozino Gold Project. The details of the capital and equipment leasing costs are included in Section 21 of this report (Capital and Operating Costs). No contracts or funding arrangements are currently in place or under negotiation.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Introduction

Velocity retained Eco-stim EOOD, a Bulgarian consultancy, to undertake the environmental work necessary to permit the Rozino Project, including an assessment of potential impacts to the environment. This included baseline assessments of surface water, groundwater and the ecology. Mineesia Ltd, a United Kingdom based consultancy, was commissioned to guide the field work and review the findings.

The environmental and social work conducted to support the Project will include:

- Completing an evaluation of potential impacts on the environment resulting from the development of the Project.
- Scheduling the environmental and social permitting requirements.
- Collecting and reviewing available data for environmental studies, assessments or audits, and baseline data gathering.
- Preparation of environmental, social plans, and monitoring programs, including:
  - Impact mitigation plan (also known as the environmental management plan)
  - Sediment and erosion control plan
  - Preliminary closure plan (with the final plan to be developed at least two years prior to end of mine life)
  - Evaluation of the potential for acid rock drainage and metal leaching
  - Spills and emergency response plan
  - Site environmental monitoring plan.
- Comprehensive overview and listing of required permits.

At this stage of the Project mitigation plans are based on standard industry practice and will depend upon the results of impact assessment and site monitoring programs.

20.2 Legal Setting

The necessary permits for the exploration of the Property are obtained and there are no known significant factors or risks that may affect access, title or the right or ability to perform evaluation work on the Property. This includes sampling and testwork required to support the environmental assessment work. The prospecting licence agreement for Rozino was signed with the Minister of Energy and exploration activities are approved by the Ministry of Environment.

Under the Bulgarian Environment Protection Act (EPA) (SG issue 91/25.09.2002 and subsequent amendments) the development of an economically viable Mineral Reserve will require an environmental impact assessment (EIA), in accordance with Directive 2014/52/EU. The Company is also required to prepare a Mine Closure Plan for the Project and submit it to the responsible authorities for approval. The responsible authorities are the Minister of Economy, Energy and Tourism and Minister of Environment and Water. This Closure Plan forms the basis for determining the form and scale of financial guarantees (Article 22g of the Subsurface Resources Act (Promulgated State Gazette No. 23/12.03.1999)) and is still being developed.

The initial opinion of the River Basin Directorate (Ref. PU-02-73, dated 18.07.2019) was that the Project area is located outside of any flood risk areas, so no flood protection is required. It is located within the boundary of a known aquifer (BG3G000PtPg049, which is less than 10% utilized) and not within an area of vulnerable groundwater. Surface water in the area (namely the Byala Reka and its tributaries) are in good environmental condition and there are no known industrial discharges to these rivers. The Water Act protects both surface water and groundwater. Protection measures have been identified by the Basin Directorate to maintain good water quality, including the control of tree-felling and vegetation clearance. The Basin Directorate imposed the following restrictions related to water abstraction:

- No damage to the natural condition of riverbeds, riverbanks, and flood zones.
- No discharge of pollutants to groundwater.

Application of measures in the Water Act to ban/restrict activities that increase the risk of direct or indirect discharge of priority/hazardous substances to groundwater.

According to Ordinance 2/2011 for issuing permits for discharge of wastewater into water bodies Art. 6. (1) states that no permits for new discharges will be issued and no new discharges will be created for waste waters in zones for protection of the waters under art. 119a, items 1, 2, 4 and 5 of the Water Act. The area of Rozino deposit site is determined as such area according to the Water Act.

The key commitment identified by the Basin Directorate is to ensure that there is no discharge of wastewater generated by the staff of the Flotation Plant (i.e. sewerage from ablution facilities) and no waste water discharge points were authorized. A follow up discussion with the Basin Directorate indicated that contaminated contact water is also defined as wastewater and the Project has included a contact water dam to contain and re-use contact water in the Flotation Plant. Water will only be released to the environment if it meets relevant water quality standards.

The Project is located within the Eastern Rhodope Mountains, an area of wide biodiversity. A Compatibility Assessment for the Project is required to comply with the Bulgarian Law on Biodiversity (SG Gazette No. 77/9.08.2002 and subsequent amendments), the European Union Natura 2000 Habitats Directive (EEC Directive 92/43) and Birds Directive 2009/147/EU before the Project could proceed.

The River Basin Directorate determined that there is one surface water body, the Byala Reka, and one groundwater body in the region. There are currently no identified sanitary protection zones surrounding water supply sources that could be affected by the Project. However, there are two Natura 2000 areas which require collection and analysis of water quality information, for at least one year prior to construction. At Rozino baseline data collection commenced at the end of 2019.

An initial Compatibility Assessment was conducted for the approved prospecting licence by Eco-stim in 2017 (Eco-Stim, 2017). Subsequently, a preliminary Compatibility Assessment was completed in 2019 which was used to refine the Project design and minimize potential environmental impacts.

20.3 International Requirements and Guidelines

The Project is classified as a Category A development in accordance with International Finance Corporation (IFC) Guidelines. Mining projects are also classified under Annex I of the EU Directive for Environmental Assessment (2011/92/EU), under Items 4(b) and 19 (should the site surface area exceed 25 ha in extent). As such, the Project is subject to an EIA process, as per Bulgarian legislation.

The primary requirement for any internationally funded project is to comply with local and national regulations of the host country. These regulations are often supplemented by standards and guidelines from international financial institutions, particularly with regard to environmental and social components of the Project. As the national regulations are derived from the EU Directive for Environmental Assessment, no supplementary standards or guidelines are expected to be required for the EIA.

The Project will implement its Environmental Management Programme (EMP) to guide environmental and social management, and stakeholder and community relations. The Project will aim to conform to the environmental and social requirements of the EU Directive for Environmental Assessment, as well as IFC Performance Standards and its associated Environmental Health and Safety guidelines, the International Council for Mining and Metals and Equator Principles where they are relevant to the Project.

20.4 Project Permitting Requirements

In order to be granted a Mining Concession, the following regulatory stages have been completed or are required:

- Prospecting and Exploration Licence Agreement between Tintyava Exploration AD and the Minister of Energy (MoE) (Agreement - 2 May 2017, Permit period extended to 31 July 2022).
- A Commercial Discovery report for exploration including a Bulgarian-compliant mineral resource estimate and a preliminary economic assessment (PEA). These form the basis of submission of an application to the Ministry of Energy (completed, May 2019).
- Defence of the Commercial Discovery for exploration to a Governmental Interdisciplinary Expert Council at the MoE (completed, September 2019).
- Submission of Update of the Commercial Discovery report including the additional drill results used for the PFS Mineral Resources estimates (completed, January 2020).
- Receipt of a Protocol of the Commercial Discovery report Acceptance by the Expert Council at the MoE with recommendation to start the OVOS (EIA) procedure (completed, May 2020).
- An Environmental Impact Assessment procedure that includes:
  - Announcement of an investment intention (Investment Proposal Announcement) that initiates the EIA process
  - EIA scoping phase
  - An environmental impact assessment report
  - Compatibility Assessment report in parallel with the EIA report
  - Public hearing on the EIA documentation
  - Session of the Environmental Expert Council at the Regional Environmental Inspectorate – Haskovo, for examination of the EIA documentation.
- Issuing of the EIA decision by the MoE.
- Declaration of Commercial Discovery and issue of certificate for Commercia Discovery.
- Submission of application for Mining Concession to the Ministry of Energy for mining and processing of mineral reserves.
- Approval of Mining Concession and promulgation of decision in State Gazette.
- Approval of the following to allow the issuing of a Construction Permit by the Chief Municipal Architect (Haskovo):
  - Detailed Site Development Plan (PUP)
  - Change of Land Use
  - Water Management Endorsement
  - Storage of Hazardous Materials
  - Agreement with Service Providers (powerline and power supply).
- Construction activities are monitored by the Directorate of National Construction Control (DNSK and RDNSK) under the Ministry of Regional Development and Public Works.
- Obtaining an Operating Permit from the State Acceptance Commission which will include all regulatory authorities related to the respective construction activities.

20.5 Baseline Environmental Setting

20.5.1 Air Quality

Air quality in the Project area is presumed to be good, given the lack of industry within 10 km of the Project. The nearest cities are Ivaylovgrad (21 km), Krumovgrad (21 km) and Haskovo (59 km), none of which have any heavy industrial activity. The closest government air quality management and assessment station is located at Haskovo (elevation 195 masl), recording particulates, pollen and gases such as nitrogen dioxide, sulphur dioxide and ozone. A station at Krumovgrad collects meteorological data only, and data has been acquired from this station to inform the water assessment.

20.5.2 Soils

The soils found in the area are mostly leached chemozem-smolnitsa soils and cinnamon-coloured forest soils (Andrew, 2008). Agriculture is one of the most prominent sectors in the Ivaylovgrad municipality. However, the Project area is predominantly planted forests. In this area forestry is dominant due to terrain and soil quality. Where crops are grown, these comprise grapes, grain, fodder, tobacco and vegetables.

20.5.3 Noise and Vibration

Given the lack of industry in the area, there are no specific sources of anthropogenic noise and vibration. Intermittent use by road traffic, and agricultural and forestry machinery are the most discernible sources of noise and vibration. Heavy vehicles use the main highway to transport goods, generating the main source of noise and vibration along those routes. Future potential impacts on the roads will be due to increased mine traffic during construction and operation. These will be assessed during the EIA, although potential impacts from the Project itself are likely to be minimal, due to the low population levels in the region. Blasting operations will generate both noise and vibration, but the frequency of blasts will be very low (one per day maximum).

The results of the preliminary Compatibility Assessment (Eco-Stim, 2019) indicated that there may be some disturbance of nesting birds by blasting. To assess the potential effects of blasting, a trial blast was conducted in April 2020. The results are being analysed to provide a preliminary indication of the risks associated with blasting and will be incorporated into the EIA.

20.5.4 Surface Water

The main water resources in the district are the Byala Reka to the south and its tributaries, the Arpa dere and the Uren dere. The East Aegean River Basin Directorate considers these waters to be clean and water quality results from samples taken by Velocity support this.

The Rozino Project is within the catchment of the Byala Reka. There is a seasonal variation in flow in the region with no flow in the tributaries during the dry season (May – October). Monitoring by Velocity has commenced to record the seasonal variation in the Arpa dere and Uren dere, with the installation of v-notch weirs and transducers to capture seasonal high flows (January to March, in the spring thaws). Flow data for Byala Reka are taken using a manual flow meter.

To comply with the opinion of the Basin Directorate, any contact water needs to be collected and reused in the process plant. In contrast, stormwater diversion water can be discharged, provided that it is not contaminated as a result of any Project activity. There are no known restrictions on collecting runoff water from the catchment and using in the process plant. As such, all contact water from the catchment will initially be collected in the contact water dam, until it can be demonstrated that it meets relevant water quality standards. Although it is preferred that some catchment water is returned to water courses downstream of the contact water dam where possible, the intermittent nature of these water courses and the precautionary principle indicates that capturing this water initially is the most suitable approach to water management.

20.5.5 Groundwater

Studies identified groundwater contained in the Paleogene sedimentary rocks and colluvial sediments in river valleys and terraces. Observation wells are installed in and around the Project area to monitor groundwater levels and quality in the Palaeogene rocks, and conduct Packer tests to assess groundwater flows and recharge rates.

Results to date indicate water levels are between 17 m and 40 m below surface. Groundwater quality was tested, with results indicating water quality is relatively good. Paleogene rocks show low permeability characteristics and flow is predominantly through fracture and other discontinuities in the strata. These will not provide appreciable volumes of water and will not be a viable or sustainable source of water for the Project. The alluvial deposits offer some potential for water supply, and further investigation of the alluvial deposits has been conducted for the Byala Reka and Arpa Dere.

20.5.6 Flora and Fauna

Project is located within two Natura 2000 areas. Therefore, data was collected on the biodiversity of the Project area. Studies focussed on bird species with higher conservation status and that could potentially occur in the area, as well as rare mammal species such as the brown bear (*Ursus arctos*), wolf (*Canis lupus*), wild goat (*Rupicapra rupicapra balcanica*), otter (*Lutra lutra*), and marten (*Martes martes*). Within the project footprint 16 habitats (as classified by EUNIS (Davies et al., 2004)) were identified, of which 8 types are potentially directly affected. Of conservation significance are seven types of habitats, which are included in Annex 1 of the Biodiversity Act and therefore subject to conservation in the Rhodopes – Eastern Protected Zone. Of these habitats only three are directly and minimally affected (less than 1% of the total habitat).

The preliminary Compatibility Assessment identified measures to protect bats, reptiles and certain species of birds. These measures included commencing construction outside of bird breeding season and recruiting a qualified ornithologist to survey the Project area prior to construction commencing.

20.5.7 Protected Areas

The Project is located within two Natura 2000 zones (Figure 20-1):

- Protected zone “Rhodopes -– Eastern”, code BG0001032, for the protection of natural habitats and wild fauna and flora
- Protected zone "Byala Reka", code BG0002019, for the protection of wild birds.

As the Project is located within these Natura 2000 areas a Compatibility Assessment is required. The location in or close to Natura 2000 areas does not automatically prevent mining, but there are conditions to be meet. The key objective of the Compatibility Assessment is to ensure the integrity of the Natura 2000 sites. In order to achieve this, the Project commissioned a preliminary Compatibility Assessment (Eco-Stim, 2019) to understand the extent of potential impacts. This assessment identified that the main risk to the Natura 2000 sites was to breeding birds as a result of blasting. No indication that the Project would significantly affect habitat or vegetation communities was provided in this assessment. Nevertheless, the Project design was refined to reduce the impacted areas to less than 1% of the total habitat by type, to comply with the Natura 2000 objectives. A trial blast was commissioned to understand the potential for disturbance of birds; the results will be included in the EIA.

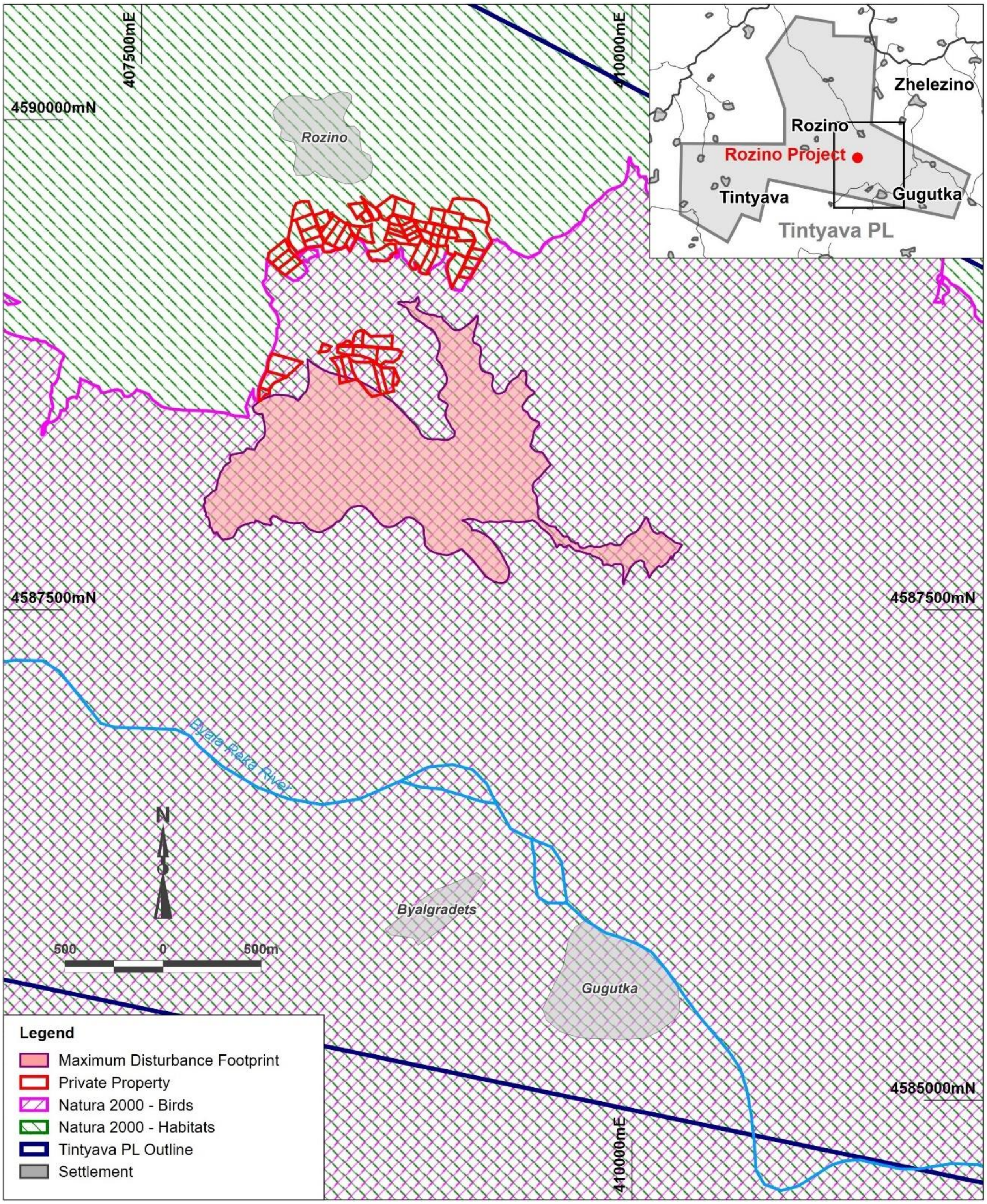


Figure 20-1: Protected areas surrounding Rozino Project
Source: Velocity, 2020

No transboundary impacts are expected because of Project activities.

20.5.8 Baseline Social Setting

Ivaylovgrad and Krumovgrad are the closest municipalities to the Project area. The nearest villages with road links to the Project site are Rozino and Konnitsi (4 km north of Rozino towards Highway 59). Gagutka is located south of the Project and linked by an unpaved track. There are several other villages within a 10 km radius of the proposed development, but these are not likely to be directly impacted by the Project as they are physically well separated by the hilly topography of the region. Inhabitants of the villages in these municipalities are primarily involved in subsistence farming, particularly livestock with tobacco as the cash crop. The other main land use is state controlled forestry. Industry is absent and there is no private-sector employer within a 10 km radius of the Project area.

Based on population records collected by the Ministry of Regional Development and Public Works, local villages have shown decreases in population between 2013 and 2018 (Table 20-1). Rural de-population is anticipated to continue. Rozino village is currently deserted except for some itinerant families occupying several houses.

Table 20-1: Population of villages in the vicinity of the Rozino Gold Project

Village	Population as per permanent and current address* (2018)	Source dated 2013**
Rozino	41	62
Vetrushka	1	No info
Vis	6	4
Byalgradets	42	33
Gugutka	73	82
Lensko	13	18
Pastrook	21	16
Zhelezino	220	228
Konnitsi	79	96
Popsko	23	45
Glumovo	7	11
Belpolitsi	276	300
Pashkul	5	5
Sokolentsi	4	4
Planinets	49	51

**Source: Directorate General "Civil Registration and Administrative Services" to the Ministry of Regional Development and Public Works, dated 15.06.2018.*

***Info source: http://www.guide-bulgaria.com/SC/haskovo/ivaylovgrad/rozino*

20.5.9 Infrastructure

The infrastructure of the area is poorly developed (Caracal, 2008). There is a 20 kV power transmission line 2.5 km from the Project area that supplies electricity to Rozino village. Highway 59 connecting Momchilgrad and Ivaylovgrad (via Krumovgrad) is a dual lane, paved road designed to carry heavy commercial vehicles. There is a single-track tarmac road connecting Highway 59 to Rozino village, and a dirt track connecting the village to the Project site. All villages have access to fresh water, through a network of reservoirs along natural rivers.

The Project will construct a 110 kV powerline from Madzharovo substation, located approximately 20 km north of the Project. Access roads from Highway 59 will require upgrading to support the use of heavy vehicles for construction and operation. Roads will be monitored throughout the Project life to minimize impacts. The EIA will include an assessment of the existing and planned infrastructure and include benefits and impacts of any improvements.

20.5.10 Landscape and Visual

The local terrain is characterized by low mountains with predominantly levelled hills cut by steep valleys and dense vegetation cover in areas. The Project elevation ranges from 70 to 700 masl and averaging about 320 masl. The Project area is bounded to the south by steep cliffs of Tashlaka hill and is segmented by the Byala Reka and its tributaries. As such, development of the Project is not anticipated to detract significantly from the existing landscape. Rehabilitation of the Project site at closure will include profiling and landscaping the WRD and TMF. Revegetation will be with predominantly indigenous species such as oak and black pine, but a mix of other trees and grasses may also be incorporated to restore natural habitats.

20.5.11 Land Acquisition and Resettlement

Land ownership has been assessed in the vicinity of the Project. Most of the land is public land, primarily forestry, although there are some parcels of private land (Figure 20-1). The Project footprint was modified to minimize the potential impact on private land, including a buffer zone to the north. Limited land acquisition is anticipated.

20.5.12 Archaeology and Cultural Heritage

An initial assessment of the archaeology of the area was conducted in August 2016 by examining the Automatic Information System – Archaeological Card of Bulgaria (AIS ACB) and archives. The following sites are adjacent to the site but will not be directly impacted by the Project (Figure 20-2):

- Rozino village:
  - Reference № 1590322 – Early Iron age site, within Golyamata niva locality
  - Reference Site № 1 – Tombstone within Agalat locality.
- Byal Gradets village:
  - Reference № 10003853 – Tombstone
  - Reference № 10003854 – Tombstone.
- Gugutka village:
  - Reference № 1590323 – Prehistorical site and Thracian sanctuary, within Aylyata locality
  - Reference № 1590327 – Thracian sanctuary and chapel
  - Reference № 1590350 – Medieval fortress, within Kaleto locality
  - Reference № 10003846 – Tombstone, within Tumbata locality
  - Reference № 10003847 – Tombstone
  - Reference № 10003863 – Tombstone
  - Reference Site №2 – Ancient and medieval settlement
  - Reference Site №4 – Late ancient and medieval settlement.
- Kazak village:
  - Reference Site №5 – Prehistorical and medieval settlement.
- Pastrook village:
  - Reference № 1590286 – Iron age site, within Arpa bair locality
  - Reference № 1590287 – Tombstone necropolis, within Arpa bair locality.

The Project has developed a “Chance Finds” procedure to manage potential impacts associated with accidental disturbance of archaeology sites.

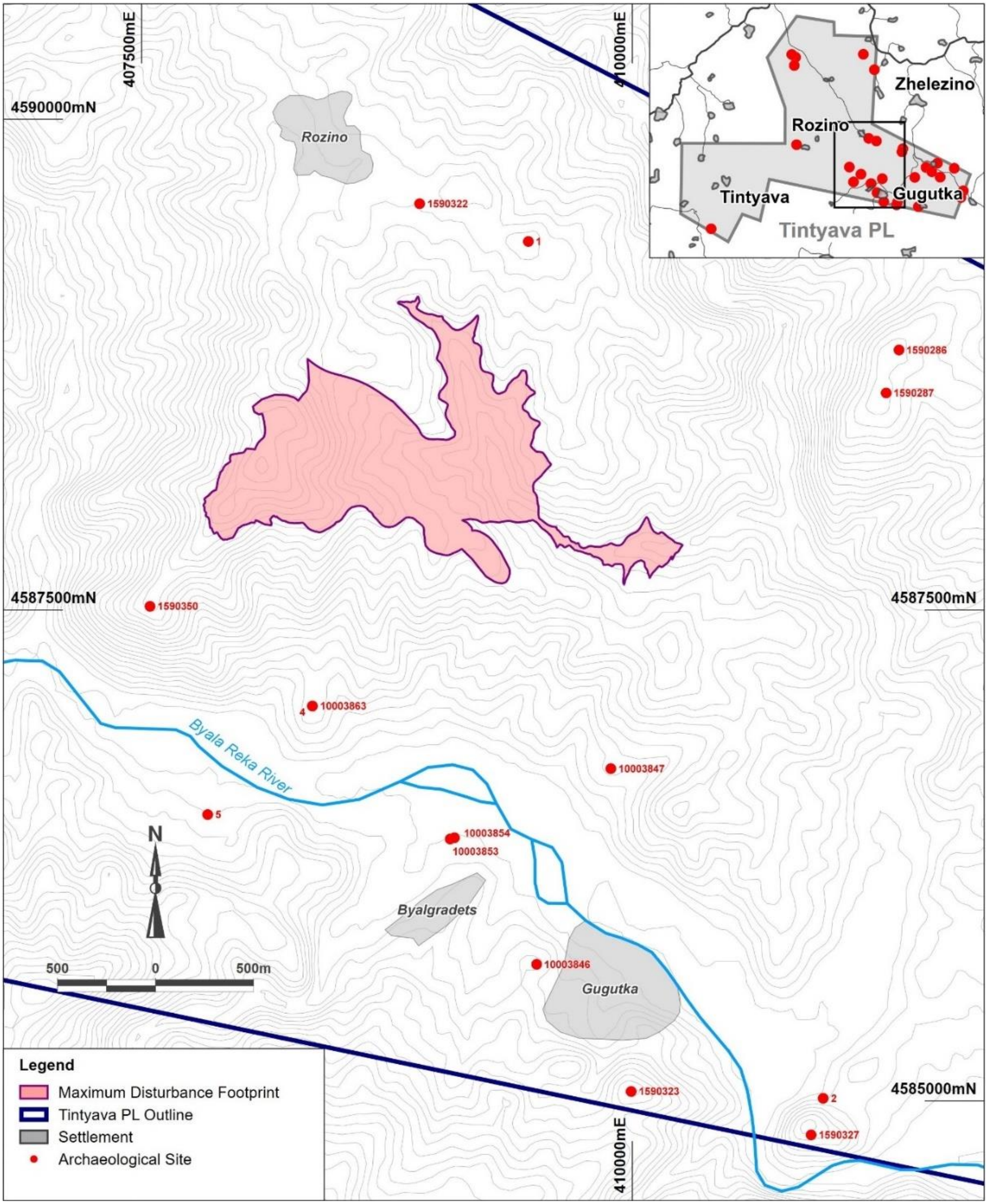


Figure 20-2: Tintyava property archaeological sites and resource extents
Source: Velocity, 2020

20.6 Potential Environmental Impacts

The EIA will include an assessment of the potential environmental and social impacts brought about by the Project. Both national and international environmental consultants have undertaken visits to the Project site and will continue to do so as required. The objective of these studies is to establish baseline ecological, surface water, groundwater, soil, air quality, and noise conditions by:

- Monitoring key receptors, particularly water systems and habitats
- Determination the scope of consultation with local and national stakeholders to identify their specific concerns
- Estimation of the nature and scale of impacts caused by the construction, operation and eventual closure of the Project
- Estimation of mitigation measures to eliminate, avoid, reduce or compensate for adverse environmental impacts
- Evaluation of residual impacts present after mitigation measures have been implemented.

As with all mining projects, this Project is likely to give rise to a range of environmental and social impacts. The assessment is used to understand these potential impacts and mitigate significant issues. An example of this process is the potential for acid rock drainage and metal leaching. This potential has been tested and initial results indicate that this potential is low. Both the ore and the waste rock have low sulphide contents and relatively high neutralizing potential, resulting in net neutralization ratios of > 2 (see MEND Report 1.20.1, December 2009 for more details on net neutralization ratios). Metal leaching testwork results suggest that any leachate from ore and waste rock is not considered harmful to receiving waters and will therefore meet the criteria for disposal as inert waste. It should be noted that this assessment is preliminary; geochemical characteristics of the rock may change in the long term as neutralization compounds and site-specific conditions change.

20.7 Environmental Management Plan

Velocity is committed to managing the impacts of its operations in compliance with international best practice. Assuming the implementation of the proposed mitigation measures, any potential impacts can be managed and controlled through the Environmental Management Plan (EMP). This would enable effective environmental and social development, operation and closure of the Project.

A preliminary EMP was developed for exploration activities and this will be expanded to include the construction and operational phases of the project. A “Chance Finds” procedure and an emergency response plan were developed for the exploration phase. The emergency response plan includes measures to address accidents resulting from work, fires, and power failures or flooding.

20.8 Health and Safety

Construction and operational activities will be conducted in accordance with Bulgarian regulations for labour safety. All personnel will undergo medical screening and site-specific health and safety training prior to commencing work. Only suitably trained personnel will be allowed to operate machinery. Appropriate clothing will be required to be worn, including personal protective equipment. Alcohol consumption will be banned before and during working hours.

20.9 Mine Closure

Closure planning is an integral component of Project planning, particularly for the EIA. It assists in identifying the most appropriate post-mining land uses and closure-related objectives and guides the transition from operations to closure. All closure planning should consider the risks and opportunities presented and develop actions based on sound knowledge. The Project will prepare a Preliminary Mine Closure Plan for approval, in accordance with Bulgarian regulations.

Closure and rehabilitation of the mine site will commence once mining from the Phase 2 pit is complete in Q1/Year 6. A detailed closure plan will be developed and finalized by then to guide these activities. Closure activities will continue during the reclamation and processing of the low-grade ore stockpile. It is anticipated that. Progressive reclamation will be carried out during normal mine operations where circumstances allow, and full mine-site closure will commence during Year 8 and once ore processing is complete

Post-closure management and maintenance objectives will be to ensure that the site achieves a sustainable and maintenance-free status. The proposed overall strategy for the decommissioning and closure of the Project is as follows:

- Decontaminate, dismantle and demolish, as far as practicable, all installations, structures and infrastructure not identified for retention and hand over to another entity
- Safe disposal of all contaminated materials removed during decontamination, dismantling and demolition activities
- Salvage (for sale and/or allocation to other operations) all equipment, and mechanical and electrical plant identified in the asset register as having a residual value or useful life
- Removal from the site as scrap (if economically viable) or dispose as solid waste all equipment, plant and structures not deemed suitable for future refurbishment and/or re-use
- Apply closure design options which are effective, practical and cost effective
- Ensure the site is left in a safe condition
- Where practical, undertake phased closure of the facilities, ensuring retention of facilities required to support the closure process and subsequent post-closure monitoring activities
- Address any potential residual environmental impacts.

It is possible that there may be opportunities after closure to hand over facilities such as the water supply infrastructure to other entities; other opportunities may be considered in the final Closure Plan. Close liaison with local municipalities and local- and regional- government authorities will ensure that site closure conforms to regulatory undertakings and benefits the surrounding communities.

Closure activities commence during the construction period with pre-stripping of topsoil and dumping onto topsoil stockpiles. Revegetation of the area is planned, with the establishment of a nursery and seed harvesting of local species. Purchase of suitable seed stock is possible to supplement seed harvesting, if required. Revegetation will be with predominantly indigenous species such as oak and black pine, but a mix of other trees and grasses may also be incorporated. The waste rock and tailings are considered benign and non-hazardous; no acid rock drainage (ARD) or metal leaching is expected during operations or post-closure. The sulphur content of the tailing's material will be less than 0.1%.

Initial closure activities will focus on the rehabilitation of the waste rock dump (WRD) and tailings management facility (TMF). Benches on the WRD will be cut and filled to produce a landform in keeping with the surrounding landscape. Where possible, the maximum slope angles of the WRD will be approximately 20°. About one Mm^3 of waste rock will be dozed in this process. During the process of dozing, the waste rock will undergo a certain degree of compaction. The surface of the waste rock dump will be ripped to reduce this compaction. Topsoil will be placed over the WRD to a depth of about 0.15 m. The combination of the shallow slope, general compaction of the waste rock and revegetation will minimize the infiltration of precipitation into the WRD, and maximize water runoff and evapotranspiration.

Closure of the TMF and the tailings stored in the Phase 2 pit will commence with the placement of approximately 0.5 m of waste rock over the surface. Topsoil spreading and revegetation will follow the same methodology as the WRD. The placement and levelling of the waste rock will promote water runoff and minimize ponding. This volume of waste rock is not included in the cut-and-fill volume stated for WRD profiling but is expected to be in the order of 200 000 m^3. The TMF wall will not require profiling as it will be built at a slope of 20^o.

The final pit walls not covered by waste rock and tailings will be ripped and dozed where possible to form a smooth profile. It is proposed that a slot be constructed in the southern side of the pit wall in order to make the pit free draining, thereby preventing the formation of surface ponds. This strategy may be re-evaluated closer to closure depending on project dictates and subject to discussions with local communities and authorities. An alternative to this pit closure plan may be to allow the pit to fill with water and form a lake. Any decant from the pit will be handled by appropriate drainage to the contact water dam.

All buildings and structures will be removed including the flotation plant, conveyors, workshops, offices and other ancillary structures. The building structures will be dismantled, and the materials removed from the site for sale, reuse, recycling, or disposal at a registered waste site. All oil, fuels, and processing chemicals will be drained from the equipment and disposed of at a licensed off-site disposal facility. The processing equipment and conveyor structures will be removed from site and sold or recycled. All the disturbed areas will be ripped or ploughed (to increase water infiltration and reduce the potential for surface erosion and instability), levelled, and covered with about 0.15 m of topsoil (except the concrete structures). Revegetation will be as for the WRD. Concrete slabs will remain in situ and covered with about 0.40 m of topsoil, either from stockpiles or imported as necessary. The tailings and water supply pipelines will be removed and disposed of off-site. Any roads that will not be required for post-closure management will be decommissioned.

The water dams below the TMF will be retained post-closure to facilitate water management and, if required, made available for use by the local communities. All water draining from the mine site will be directed to the contact water dam where initial sediment settling will occur. Prior to discharge, the water in the cutoff dam will be tested to ensure that it meets receiving water quality standards in the raw water dam. The RWD will be used as a secondary storage and settling facility for water received from the contact water dam, prior to discharge into the Uren dere water course.

The powerline will be left in situ and handed to the regional power authorities.

Active site management and maintenance is expected to continue for five years after closure. This will entail inspections at appropriate intervals to ensure that any soil erosion is repaired, vegetation density is maintained, the integrity of water control structures is maintained, and the ecology of the area achieves the required status. The period of closure will be undertaken with the assistance of consultants who will advise and report on the status of the rehabilitation. Passive closure is anticipated to continue for a further five years with inspection intervals reduced appropriately. Maintenance will be carried out on an as-required basis. Closure monitoring will be undertaken to document the progression of the mine site from the operational phase to relinquishment.

Closure rehabilitation and post-closure management costs were estimated on the basis of using equipment from the mining operation, contractors experienced in equipment dismantling and salvage, and a range of consultants to advise on the closure methodology and report to the regulatory authorities. A provision of $0.33/processed tonne to cover the cost of closure and post-closure management was allowed for. The closure cost has not been offset by the proceeds from the potential sale of salvaged structures and equipment.

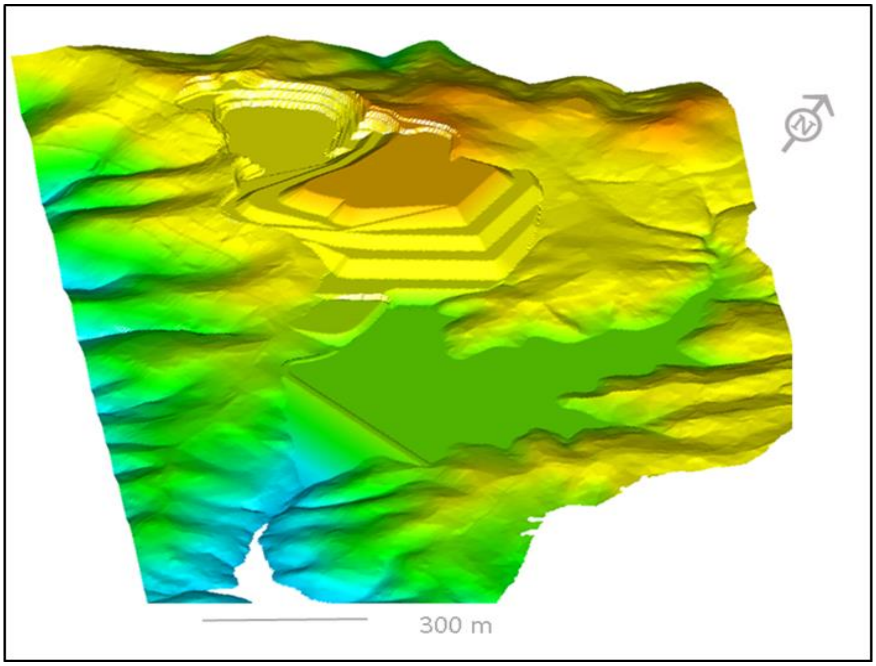


Figure 20-3: Isometric view, Rozino Mine Site at the end of operations (view to northwest)

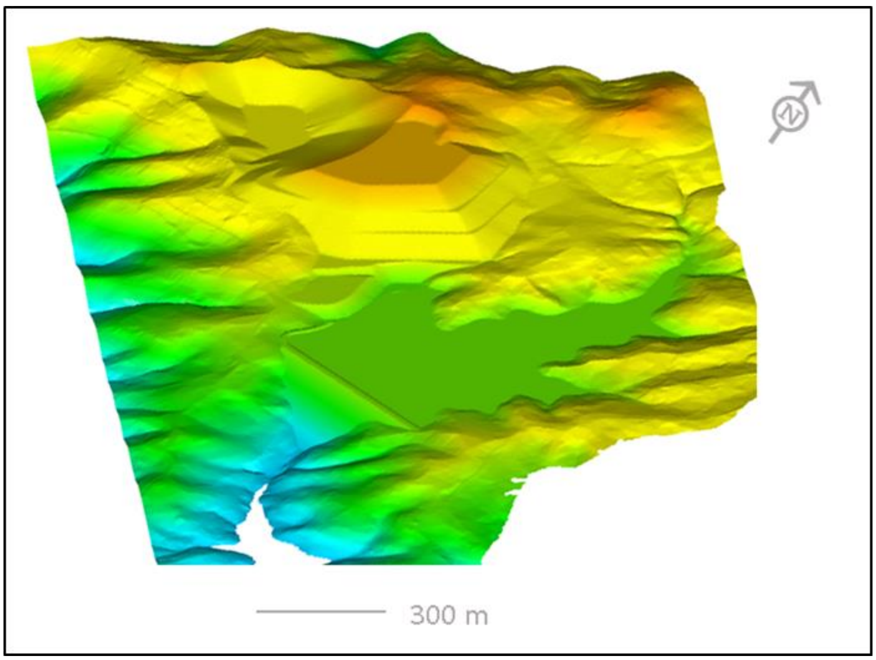


Figure 20-4: Isometric view, Rozino Mine Site after landform smoothing, approximate final rehabilitation

20.10 Environmental Monitoring

Velocity has commenced baseline monitoring for the Project. The objective of baseline monitoring is to characterize environmental conditions, including groundwater, air quality (specifically airborne dust) and ecology, prior to Project construction. Monitoring will continue through the life of the Project to observe any changes that result from the development of the Project. This information will continue to be analysed to inform environmental management of the Project and support the development of mitigation plans for construction, operation and closure phase impacts.

Tintyava has developed and implemented appropriate sampling procedures for water and dust sampling. Water monitoring for surface flow levels, volume and quality has been undertaken since December 2019. Groundwater levels are recorded monthly, and water quality of both surface water and groundwater analysed on a 6-monthly basis.

Interactions with local communities are recorded, along with wildlife observations and any other items of environmental interest. Site environmental activities such as water sampling and social engagement are recorded, and two environmental staff members have been recruited to implement and supervise site monitoring. Camera traps are planned to detect the presence and range of wildlife.

20.11 Public Consultation

As part of the environmental assessment, public consultation and disclosure is required. In order to ensure that the Project is developed and operated in an appropriate manner, Tintyava will incorporate the concept that effective engagement with its stakeholders is an essential component of the assessment process and its on-going 'license to operate'. Velocity is committed to a proactive programme of communications with all relevant stakeholders.

The Project has few stakeholders due to the remoteness of the location. The closest people to the site are itinerant residents at Rozino, with residents and small-scale farmers present at Konnitsi and Gagutka. Public Consultation is required under Bulgarian legislation and formal meetings will be undertaken once the IPA is published and as part of the EIA process.

21 Capital and Operating Costs

21.1 Introduction

This section details a summary of capital and operating cost estimates, with the major components set out in tabular form. Explanations and justifications for the basis for the cost estimates are included.

21.1.1 Base Date and Escalation Rates

The base date for the operating cost estimate is July 2020. All costs are real and, with one exception, escalation factors have not been applied in the cost estimates. The one exception is labour rates which include a labour market escalation factor to reflect Bulgaria joining the EU and the consequent and observed upward pressure on remuneration. The labour market escalation has been formulated from a yearly inflation rate above CPI that commences at 7% in 2020 and decreases to 2% at the end of the project operating life.

21.1.2 Exchange Rate and Base Currency

No allowances were made for fluctuations in exchange rates. The DCF model utilizes US dollars as the base currency as the majority of capital and operating cost estimates are based in US dollars. Where stated (specifically in the output and reporting numbers) a rate of exchange of CAD$0.75 to US$1.00, BGN0.58 to US$1.00, and EUR1.10 to US$1.00 has been used for currency conversions.

21.1.3 Accuracy

The accuracy of the capital and operating cost estimates is considered appropriate to Pre-feasibility Study requirements.

21.1.4 Data Compilation

The capital and operating costs were obtained either from suppliers directly, assembled and verified from other contributors by CSA Global, or from reference databases. Other than are mentioned in this section as having been supplied by CSA Global, the following major contributors of capital and operating cost are:

- Halyard
- Capital cost estimation for all plant and mine building infrastructure
- Flotation plant operating costs
- Unit costs for construction for the TMF; material quantities provided by Golder
- Velocity contracted Dipl. Eng. Nikolay Vassilev Savov to provide powerline capital and line power operating supply costs
- Velocity contracted PiA to provide capital and operating costs for the water supply pipeline
- Velocity contracted MGU to provide capital costs for the upgrading of the access road
- Velocity provided estimates for the cost of upgrading the unsealed portion of the access road
- Velocity provided cost estimates for closure rehabilitation and management.

Some cost areas are dominated by contract rates as supplied by the service providers and these are duly described as such in the text. Applicable verifications to all costs provided by third parties were performed by CSA Global.

21.2 Capital Cost

The Rozino Project total capital expenditure is estimated at $94.8 M. Table 21-1summarizes the main capital items.

Table 21-1: Total capital expenditure

Capital expenditure	$M
Rozino Gold Project Site Preparation	12.6
Mine Infrastructure	10.7
Flotation Plant and Mine Buildings	39.0
TMF incl waste overhaul	9.8
Central Plant Upgrades	1.1
Owner's Administration Costs	2.9
Indirect Costs	2.2
EPCM and Commissioning Costs	7
Contingency	9.6
Total Project CAPEX	94.8

All project costs incurred prior to the declaration of commercial production (but not including sunk costs prior to the construction decision) are considered pre-production capital costs that total $87.1 M (Years -2 and -1 in the DCF). Declaration of commercial production is assumed in the PFS to be at the start of Year 1 when the Flotation Plant is fed with first ore. The remaining $7.8 M of capital expenditure (sustaining capital) will occur over the operating life (Years 1 to 7). Approximately 95% of the sustaining capital is for TMF construction.

Capital expenditure was estimated from quotations and suppliers' costs for equipment or services supplied in Bulgaria.

Mine infrastructure costs include the RWD's, water pipeline, powerline, transformer station and roads.

Table 21-2: Total capital expenditure

Capital expenditure Rozino Gold Project	Unit	LOM total	Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Rozino site preparation											
Equipment Mobilization	$M	0.4	0.2	0.1	0.1	0.0					
Clearing, Grubbing and Topsoil Removal	$M	3.3	1.5	1.5	0.2	0.1					
Mine Pre-strip	$M	8.9	1.4	7.5							
Subtotal	**$M**	**12.6**	**3.1**	**9.1**	**0.3**	**0.1**					
Rozino Mine Infrastructure											
Roads	$M	2.7	2.5	0.2	0.0	0.0					
Water dams RWD and CWD	$M	1.5	1.5								
Water pipeline	$M	0.4	0.4								
RWD overhaul	$M	0.1	0.1								
Powerline 23 km	$M	6.0	2.4	3.6							
Subtotal	**$M**	**10.7**	**6.9**	**3.8**	**0.0**	**0.0**					
Rozino TMF											
Tailings Management Facility (Rev D)	$M	8.9		3.7	2.5		1.5	1.1		0.2	
TMF overhaul capital redirection	$M	0.9		0.2	0.4		0.2	0.1			
Subtotal	**$M**	**9.8**		**3.9**	**2.9**		**1.7**	**1.2**		**0.2**	

Capital expenditure Rozino Gold Project	Unit	LOM total	Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Flotation plant plus mine buildings	$M	39.0	25.3	13.6							
Commissioning Team	$M	0.5		0.2	0.3						
CIP Plant Upgrades	$M	1.1		1.1							
Owners Administration	$M	2.9	1.2	1.7							
Indirects	$M	2.2	1.2	0.8	0.1	0.0	0.0	0.0	0.0	0.0	0.0
EPCM	$M	6.5	3.6	2.5	0.2	0.0	0.1	0.1	0.0	0.0	0.0
Closure (provisioned in OPEX)	$M	0.0									
Subtotal	**$M**	**85.3**	**41.3**	**36.8**	**3.7**	**0.1**	**1.8**	**1.4**	**0.0**	**0.2**	**0.0**
Contingency	$M	9.6	5.3	3.6	0.3	0.0	0.2	0.1	0.0	0.0	0.0
TOTAL Capital Expenditure (incl. sust.)	**$M**	**94.8**	**46.7**	**40.4**	**4.0**	**0.1**	**2.0**	**1.5**	**0.0**	**0.2**	**0.0**
TOTAL Capital Expenditure (excl. sust.)	**$M**	**87.1**	**46.7**	**40.4**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Sustaining Capital Expenditure	$M	7.8			4.0	0.1	2.0	1.5	0.0	0.2	0.0
	$/t	0.67			2.71	0.06	1.12	0.86	0.00	0.11	0.00

21.2.1 Site Preparation

The Rozino Gold Project is a greenfield site. The Project area will require preparation for construction, mining, and processing activities.

21.2.1.1 Equipment Mobilization

This includes only mining equipment that will be utilized by Velocity and the drill and blast contractor in excavation of the pit. Mobilization of other equipment, such as for plant construction and clearing and grubbing is included within those capital estimates.

The mining equipment will require transportation and, in some instances final assembly, by the OEM's to the site. The equipment is considered mobilized from Western Europe over a distance of 2,200 km (as the specific case for BAS, the OEM for the FMX haul trucks). The mobilization cost was estimated from the weight of the equipment and at a standard European road transport cost. This method was applied to 95% of the equipment. In the other 5% of the equipment mobilization cost was estimated as 2.5% of the capital cost. No port or border fees or duties are included as all equipment is provided within the EU.

Table 21-3: Mining equipment mobilization cost ($000s)

Area and equipment	Units (each)	$/unit ($k each)	Total ($k)	Y-2 ($k)	Y-1 ($k)	Y1 ($k)	Y2 ($k)
Pit Load and Haul							
Cat 390 Excavator	3	29.7	88.1	29.7	29.7	29.7	0.0
Volvo FMX 55t 10x4	12	9.9	118.9	29.7	79.2	9.9	0.0
Major Ancillary Fleet							
Cat D8 Dozer	1	16.8	16.8	0.0	0.0	0.0	0.0
35 kl Water Bowser	1	16.8	16.8	0.0	0.0	0.0	0.0
35 kl Fuel Bowser	1	16.8	16.8	0.0	0.0	0.0	0.0
Cat 150 Motor Grader	1	11.6	11.6	0.0	0.0	0.0	0.0
Cat 980 FEL	1	12.2	12.2	0.0	0.0	0.0	0.0
Stockpile Ore to Crusher							
Cat 966 FEL	1	8.3	8.3	0.0	0.0	8.3	0.0

Area and equipment	Units (each)	$/unit ($k each)	Total ($k)	Y-2 ($k)	Y-1 ($k)	Y1 ($k)	Y2 ($k)
Drilling Fleet							
Drill rig Cat 5150	2	21.4	42.8	21.4	21.4	0.0	0.0
TLB (Stemming)	1	2.8	2.8	2.8	0.0	0.0	0.0
Light vehicle	1	0.9	0.9	0.9	0.0	0.0	0.0
Minor Ancillary Fleet							
30t excavator with hammer	1	9.9	9.9	9.9	0.0	0.0	0.0
Transport bus	2	3.9	7.9	0.0	3.9	3.9	0.0
Low loader 80t	1	11.2	11.2	0.0	11.2	0.0	0.0
Lighting plant	4	0.3	1.4	0.7	0.7	0.0	0.0
Water pumps	4	2.8	11.3	0.0	0.0	0.0	11.3
Submersible pumps	2	0.4	0.8	0.8	0.0	0.0	0.0
Compactor	1	2.1	2.1	2.1	0.0	0.0	0.0
Tyre handler	1	12.8	12.8	12.8	0.0	0.0	0.0
Light vehicles	5	0.9	4.5	4.5	0.0	0.0	0.0
TOTAL CAPITAL EXPENDITURE	**32**		**398.4**	**189.3**	**146.1**	**51.8**	**11.3**

21.2.2 Clearing Grubbing and Topsoil Removal

Prior to any excavation the site will require tree felling and removal of all vegetation. This activity will be completed in designated areas ahead of topsoil removal and subsequent construction activities. Estimation of costs for clearing and grubbing depend strongly on the environmental controls imposed, steepness of the topography and amount of vegetation. Clearing and grubbing is a specialized and not frequently applied or available contracting skill. A skilled local provider was not found during the PFS that could provide a complete and confident quote. CSA Global has a database of applicable costs from various projects. A review of the database provided a project with contract terms and costs that was considered appropriately similar and undertaken in a nearby Balkan country in 2019. Site photographs and plans confirmed similar topography and vegetation. An estimated cost for clearing and grubbing, inclusive of equipment mobilization, totals $15,000 per hectare. Clearing and grubbing is considered a specializt task and thus the potential synergies with the Rozino mining team and fleet were not applied.

Table 21-4: Clearing grubbing and topsoil costs

Rozino Clearing Grubbing and Topsoil	Units	Total	Y-2	Y-1	Y1	Y2
Area	ha		57	58	7	3
Costs						
Clear and grub embankment footprints, haul vegetation to designated stockpiles	$M	1.9	0.9	0.9	0.1	0.0
Topsoil Strip, Stack and Haul	$M	1.4	0.6	0.7	0.1	0.0
Subtotal	**$M**	**3.3**	**1.5**	**1.5**	**0.2**	**0.1**

Topsoil stripping, hauling and stacking costs were derived from local quotations and other regional contractors. The cost is inclusive of equipment mobilization. There may be synergies in using the Rozino mining fleet for some activities, but these were not considered for the PFS cost estimation. Topsoil thickness was assessed by Velocity engineers to be approximately 0.15 m thick on average across the Project site. The cost, inclusive of mobilization, is $7.50/m^3.

Two topsoil stockpiles are designated on the Project site, one close to the plant and TMF and the other close to the northern boundary of the mine (see Section 16, Figure 16-8). There were no identified opportunities for topsoil to be redirected to rehabilitation within the life of the project, although this does not preclude the operation seeking these opportunities when further detailed planning is completed. All topsoil in the PFS mine plan will be used in rehabilitation at the end of the mine life.

21.2.2.1 Mine Pre-Strip

Mine pre-stripping during the construction phase is required to provide waste rock for the construction of the TMF, CWD and the RWD. Whilst this phase of mining preferentially targets pit waste rock, some ore will be excavated. Consequently, the ROM ore pad will require construction during this period but will not be constructed to its full design extent. To complete the ROM ore pad, further waste rock will be placed during the operational phase of mining and included in operating costs. The pre-production mine schedule is focussed on producing the least ore possible in order to limit capital costs and reduce the period of stockpiling prior to processing. All mining costs during the capital construction period were estimated using the mine operating costs (see Section 21.4.1. for full details). The pre-stripping cost amounts to $8.9 M.

Most mine operator training occurs in the construction phase. Half of the operators are allocated three months of non-productive training time with two trainers. The remaining half of the operators are considered to arrive fully or near to fully trained. This training cost is included in the mine pre-stripping capital.

A total of 1.8 Mt of waste, 0.1 Mt of low-grade ore and 0.2 Mt of high-grade ore is mined in the pre-stripping period; the strip ratio is around 6.2:1 compared to the LOM stripping ratio of 2.2:1.

21.3 Mine Infrastructure

Mine infrastructure costs include the RWD, CWD, water pumps and pipeline, powerline, transformer station and roads.

21.3.1 RWD's Contact and Non-Contact

The CWD (contact water dam – contact water storage) and RWD (raw water dam – non-contact or freshwater storage) construction costs were estimated at $1.5 M before application of EPCM and contingency. The designs for the dams are discussed in Section 18. Design criteria have been developed, using Canadian Dam Association (CDA) guidelines as a main reference, which Golder consider as the most applicable guidelines. The dams are constructed mostly of compacted (waste) rock fill.

The material quantities and construction specifications were developed by Golder and the unit rates for each cost element were applied based on the same elements estimated for the TMF as undertaken by Halyard. CSA cross referenced all documents to verify the correct application of the cost elements and the total cost.

The rock fill will be delivered from the mine pre-strip development, either directly or via the plant pad construction rock fill, depending on construction timing which will be developed in the FS. Some specialized materials (sand, clay and impermeable wall liners) will be supplied from external quarries and suppliers.

A 10% provision (on total cost) was included for temporary construction roads and removal of unsuitable materials. Overhaul costs for the supply of rockfill was estimated separately. The crushed rock material for the under-liner will be produced by the crushing plant utilized for the TMF.

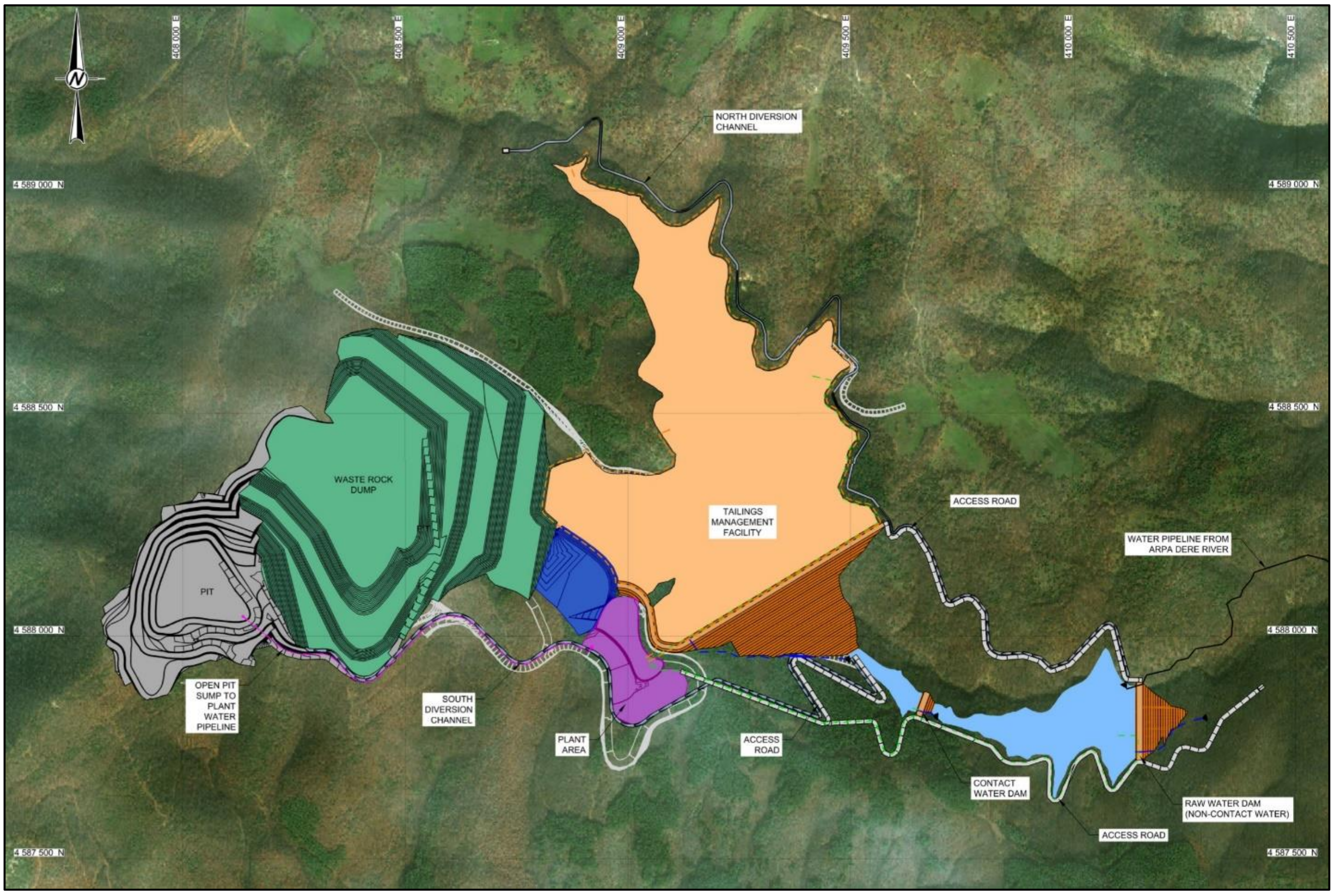


Figure 21-1: General site arrangement
Source: Golder, 2020

21.3.1.1 RWD (and CWD) Overhaul

All mining costs for pit or plant construction are fully estimated for those activities to optimal waste dumping locations. To supply material to the RWD and CWD only the incremental cost of hauling the distance additional to the WRD is considered. The overhaul cost was estimated at $0.4 /t on 0.219 Mt of material for a total of $0.088 M.

21.3.1.2 Water Pipeline

The supply of make-up water from local water sources over a distance of 1.2 km entails the installation of a pump station and a pipeline to deliver water to the raw water dam in Year -1. The cost is shown in Table 21-5which is a summary of the detail as set out in the report provided by PiA, a Bulgarian engineering consultancy.

Table 21-5: CWD construction costs

CWD construction item	$ M
Material supply	$0.237
Construction	$0.305
Liner	$0.092
Total	**$0.634**

Table 21-6: RWD construction costs

RWD construction item	$ M
Material supply	$0.271
Construction	$0.369
Liner	$0.103
Total	**$0.774**

Table 21-7: Make-up water pipeline installation costs

Water pipeline item	$ M
Water intake	$0.006
Pump station pad	$0.013
Pump station enclosure	$0.024
Pump	$0.069
Powerline and control system	$0.119
Pipeline and installation	$0.163
Total	**$0.394**

21.3.1.3 Powerline 23 km

The construction of the 110 kV overhead powerline from the Madzharovo power station to the Flotation Plant site was based on a distance of 23 km. The cost estimate assumed the installation of four galvanized steel pylons per kilometre with applicable cables, insulators, spreaders and lightning protection as set out in the detailed report provided by Dipl. Eng. Nikolay Vassilev Savov. The capital cost also includes land easements, transformers at the Madjarovo substation and the Rozino site, permitting, design, and project coordination. Table 21-8 sets out the summary of the cost estimation.

The power-line construction cost is estimated at $6.0 M.

Table 21-8: Powerline materials and construction cost

Item	Quantity	Unit	Unit cost ($M or $M per km)	$ M
Materials and construction	23 km	$ M per km	0.730	1.681
Connection to Madjarovo substation	1 off	$ M	0.234	0.234
Easement	22 km	$ M per km	0.030	0.669
Rozino site electrical substation (10 MVA)	1 off	$M	3.042	3.042
Transformers 0.4 kV	3 units	$ M per unit	0.026	0.080
Permits, design and coordination	1 off	$ M	0.284	0.284
Total				**5.989**

21.3.1.4 Roads

Four different types of roads are required for the project. All roads are allocated to be constructed by contractors.

Bitumen Access Road

The 12 km bitumen road from Rozino village to the II-59 intersection requires upgrading for heavy vehicles (mostly 30 t capacity concentrate trucks). MGU were employed by Velocity to develop this cost which they provided in a detailed report. The costs were dominated by base layer preparation (45%) and asphalt placement (50%). Culvert and drainage control upgrades were identified along with road signs and markings. The road design includes full upgrade, maintaining the four metre width sealed surface with a one metre shoulder on either side and regulalyr spaced pull-outs. The total cost was estimated at $1.7 M.

Unsealed Access Road

Velocity engineers prepared the cost to upgrade the already existing unsealed track from the end of the sealed road, around the village of Rozino, and to the Project site. The road will remain unsealed but will be upgraded to accept heavy vehicles. This cost was estimated at $0.2 M for the 2 km ($91/m). The road will be single lane with pull-outs.

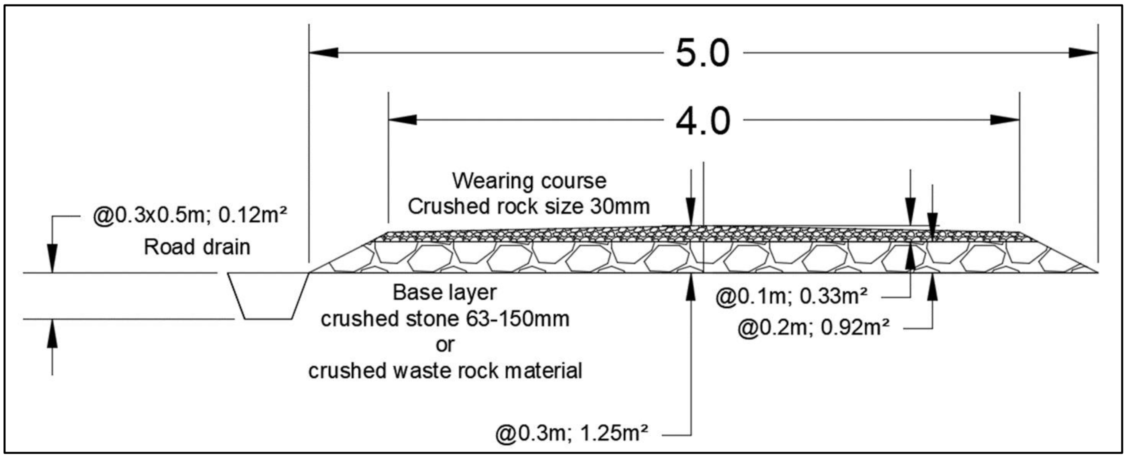


Figure 21-2: Rozino Village by-pass road typical cross section
Source: Velocity, 2020

Haul Roads

Some 4.5 km of haul roads are required within the project area (external to the pit), connecting the pit to the WRD, the TMF and the plant area. Any roads within the pit will be temporary and will be constructed using the mine ancillary equipment. The plant layout has a few short interconnecting service roads that are estimated within the cost of the plant.

All haul roads within the Rozino Project are designed with a 12 m width and dual lanes to accommodate the Volvo FMX 55t mining trucks.

CSA Global estimated the haulage road construction cost, based on a typical profile, to be $182 /m using equipment costs developed for the mine operating costs and productivities based on site observation and typical equipment used for this type of application.

The drainage channel adjacent to the haul road is designed to be constructed from concrete due to the gradient. Costs include signage and safety bund construction. The total cost was estimated at $0.8 M.

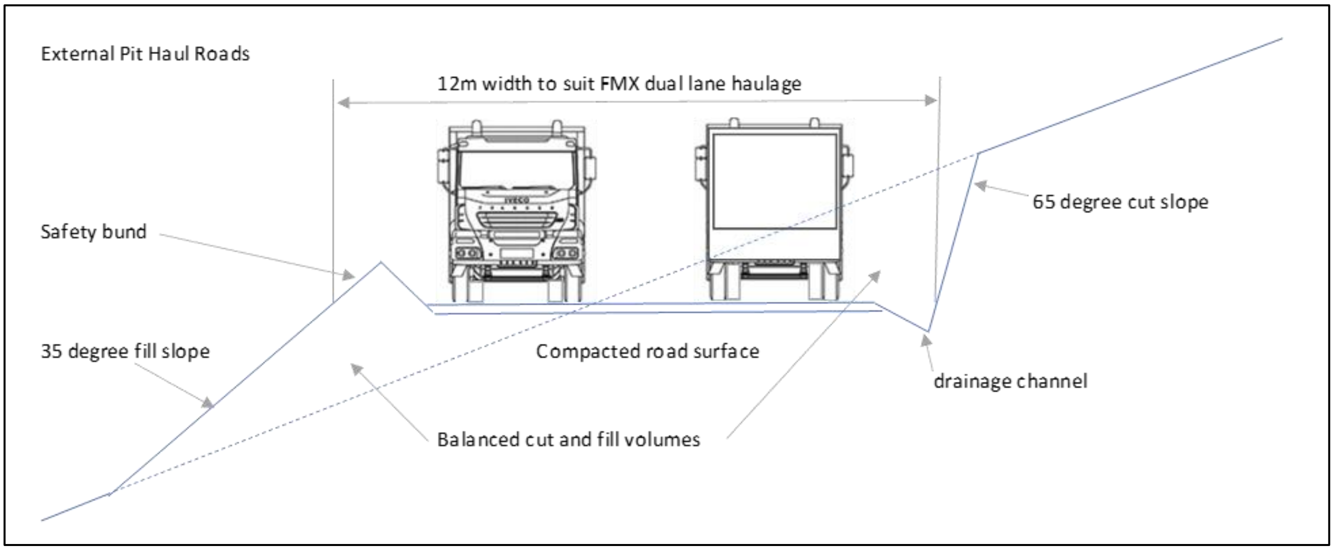


Figure 21-3: Typical cross section of the Project haulage roads

Service Roads

Some service roads will be required to access the water dams. These are estimated to be 350 m long with a single lane for light truck access. The total cost is estimated to be $0.05 M ($120/m).

21.3.1.5 Flotation Plant and Mine Buildings

A short project life was an important design consideration for the construction of the Flotation Plant and ancillary structures. Consequently, the building structures use relatively short-life concepts that not only reduce capital costs but facilitate closure and rehabilitation. The mine facilities include a main mine workshop plus two minor workshops for contractors. Mine warehousing utilizes containers located near to the workshop complex. The workshop complex also includes diesel storage, bunding, a washdown pad and a contaminated oil sump.

The Flotation Plant capital estimate includes site preparation, coarse ore bin, primary jaw crusher, secondary and tertiary cone crusher and screens, crushed ore stockpile, ball mill and cyclone bank, conditioner tank. rougher and single-stage cleaner circuit, tailings thickener, concentrate thickener, concentrate filtration, reagent warehousing, plant housing and other enclosing structures. Ancillary structures include maintenance workshop and containerized parts storage, washdown bay, fuel storage tank and dispensing bowser, security gatehouse, truck weighbridge, sewage treatment, water purification unit, and a general administration building. All concrete foundations, slabs and containments are included.

The total capital estimate before application of indirects, EPCM and contingency is $39.0 M.

Table 21-9 includes the Central Plant upgrades at the Gorubso facility necessary for the ore regrind and gold room. These total $1.1 M (for a grand total of $40.1 M before EPCM, indirects and contingency).

Table 21-9: Capital construction cost for Flotation Plant, mining facilities and Central Plant upgrades

Estimate summary	$ M
Equipment and material supply	
Mechanical equipment	14.7
Structural steel	2.0
Buildings and infrastructure	3.2
Piping and valves	2.3
Electrical and instrumentation	4.3
Freight and transport	2.4
Subtotal – Supply items	**29.0**
Non-supply items	
Civil construction	6.1
Mechanical installation	1.9
Structural steel installation	0.4
Buildings and infrastructure installation	0.7
Electrical installation	1.0
Piping and valves installation	1.1
Subtotal – Non-supply items	**11.1**
NET ESTIMATED PROJECT COST (EXCL INDIRECTS)	**40.1**

Figure 21-4 shows the plant and facilities at the Rozino site as at the end of construction. Further detail is available in Section 17 Recovery Methods.

Figure 21-4: Rozino site as at the end of construction
Source: Halyard, 2020

21.3.2 Tailings Management Facility

The construction cost estimate for the TMF is $8.9 M. This is made up of $8.7 M for the TMF (Table 21-10) and an additional $0.2 M for the relocation of pipes and pumps to allow tailings storage in the pit when mining ceases.

Table 21-10: TMF construction details by year

TMF construction item	Total ($M)	Y-1 ($M)	Y1 ($M)	Y3 ($M)	Y4 ($M)
Preliminary and general	0.350	0.350	-	-	-
Site preparation	1.743	0.786	0.293	0.332	0.332
TMF embankment construction	5.309	1.635	2.043	0.972	0.660
Drainage system	0.029	0.029	-	-	-
Decant system	0.478	0.478	-	-	-
HDPE lining on embankment face	0.424	0.080	0.120	0.112	0.112
Monitoring and instrumentation	0.004	0.004	-	-	-
Pipework and associated items	0.287	0.224	-	0.031	0.031
Miscellaneous	0.074	0.074	-	-	-
Total	**8.700**	**3.659**	**2.456**	**1.448**	**1.136**

The design and estimation of material type and quantities was carried by Golder Associates (UK).

The costs for the construction of the TMF were estimated by Halyard using costs from contractors and suppliers based on Bulgaria. The estimate does not include the incremental cost of waste rock hauled from the pit (the source for all the rock fill), access roads, and clearing, grubbing and topsoil removal. These costs are included in the relevant sections dealing with these aspects of construction. Included in the cost estimate is the supply and installation of a 2 mm HDPE geomembrane under-liner on the TMF upstream wall, and the rental of a crushing and screening plant to produce sized material from waste rock on site. All screened rock for the under-liner for the life of the Project will be produced in Year -1 of the operation.

For the PFS mine plan, the TMF is constructed in three phases. The first phase occurs in Year -1 and Year 1 and raises the TMF to 360 masl. Phase 2 occurs in Year 3 and will raise the TMF to 375 masl. The final phase will raise the TMF to 384 masl in Year 4.

The first phase is designed to minimize pre-production capital whilst also delivering sufficient start-up storage capacity. TMF construction activities will be continuous throughout Year -1 to the end of Phase 1 in Year 1.

There are four major cost elements of the TMF construction:

- The placement and compaction of the rock fill (50%)
- The crushing, screening hauling and placement of the under-liner rock bed (14%)
- The supply and placement of sand for the under-liner bed (10%)
- The supply and placement of the upstream face wall under-liner geomembrane (5%).

A trade-off study between external sourcing of screened material versus hiring a mobile crusher and screen for the under-liner rock was undertaken. The latter option proved to be the most economic option. The crusher and screen will operate in Year -1 and Year 1 only. All sized materials required for Year 3 and Year 4 will be produced during Year 1 and stockpiled until needed.

CSA Global estimated that an additional $0.2 M will be required for pumps and pipelines once the Phase 2 pit (west pit) starts accepting tailings in Year 6. It is assumed the floating decant barge from the TMF will be relocated to the pit storage in Year 6. The pipeline will be specified to handle the pumping of the tailings to the pit at a hydraulic head of approximately 45 m.

21.3.3 Owner's Administration

Owner's administration costs ($2.9 M) were derived from administration costs estimated for the operational phase of the mine. It was assumed that administration costs would increase progressively from about 30% of the full loading at the start of construction and increase to 75% in the final months of construction and plant commissioning. The reader is directed to the operating cost estimate of the administration cost for further details.

21.3.4 TMF Overhaul Capital

Mine waste rock is the most cost-effective material for the bulk fill of the TMF retention wall. The cost difference to haul the waste rock the extra distance between the WRD location and the TMF was estimated in the mine scheduling process. The overhaul cost does not include any fraction of ancillary equipment for road maintenance, mine management or services. Overhaul costs vary from around $0.21 to $0.40 per tonne hauled, depending on the phase of construction, for a total of $0.9 M.

21.3.5 Central Plant Upgrades

A capital estimate for upgrades and additions to the Central Plant was included and reported within Table 21-9 above. The estimate covered the construction of a truck off-load facility, concentrate storage, a re-pulping facility, a stirred mill and cyclone classification, and additions to the gold recovery circuit. The remaining equipment and facilities at the CIP Plant are considered to be of adequate size and condition to accommodate the Rozino concentrate throughput and no further capital expenditure is envisaged. The total capital estimate before application of indirects, EPCM and contingency is $1.1 M.

21.3.6 EPCM, Indirects and Contingency

EPCM costs are estimated for each capital item and aggregate to $6.5 M (9% of the capital construction and equipment cost of $73 M).

Indirect costs are estimated for each capital item and total $2.2 M (3% of the capital construction and equipment cost of $73 M)

Project capital contingency measures 11.3% of total capital (inclusive of EPCM and Indirects). The contingency was evaluated for each cost element and varies from 0% to 15%. In the case of the mine pre-strip, owners administration and overhaul costs, these were produced throughout the operating cost methodology and are considered to include their own internal estimates of required contingency. The operating costs redirected to capital includes some operating contingency and these were therefore not burdened with an additional contingency.

Table 21-11: Contingency, indirect and EPCM percentages and costs

Item	Contingency		Indirect		EPCM	
	%	$ M	%	$ M	%	$ M
Equipment Mobilization	10.0%	0.0	0.0%	0.0	0.0%	0.0
Clearing, Grubbing and Topsoil Removal	10.0%	0.3	0.0%	0.0	0.0%	0.0
Roads	10.0%	0.3	0.0%	0.0	0.0%	0.0
Mine Pre-strip	0.0%	0.0	0.0%	0.0	0.0%	0.0
Owners Administration	0.0%	0.0	0.0%	0.0	0.0%	0.0
TMF overhaul capital redirection	0.0%	0.0	0.0%	0.0	0.0%	0.0
RWD overhaul	0.0%	0.0	0.0%	0.0	0.0%	0.0
Commissioning Team	10.0%	0.1	0.0%	0.0	0.0%	0.0
Powerline 23 km	10.0%	0.7	3.0%	0.2	9.0%	0.5
Flotation Plant plus mine buildings	15.0%	6.8	4.3%	1.7	12.5%	4.9
Tailings Management Facility	10.0%	1.0	3.0%	0.3	9.0%	0.8
Water dams RWD and CWD	10.0%	0.2	3.0%	0.0	9.0%	0.1
Water pipeline	10.0%	0.0	3.0%	0.0	9.0%	0.0
Central Plant Upgrades	15.0%	0.2	4.3%	0.0	12.5%	0.1
Total	**11.3%**	**9.6**	**3.0%**	**2.2**	**9.0%**	**6.5**

21.3.7 Commissioning Team

The start-up of the Flotation Plant is augmented by a small operational commissioning team that will provide operational training and assistance to the site team. The commissioning team are only expected to be on site for the critical first three months. There are additional operator training costs included in the start-up periods for the mine operating cost estimate and all personnel costs include a training component. Commissioning team costs are expected to be $0.5 M.

21.4 Operating Costs

Operating costs were based on the development of equipment productivities, the Rozino local and regional operating environment and contractor quotations or supplier costs for machinery and services in Bulgaria.

Labour costs across all activities were estimated from a detailed labour survey and benchmarking exercise undertaken by a Bulgarian human resources consultant (An-May HR Agency). An adjustment factor to allow for upward pressure in labour rates due to the integration of Bulgaria into the European Union commences at 7% in the first year of construction and reduces to 2% in the last year of production.

The mining operating costs includes the leasing of primary and ancillary mining equipment, drilling and blasting carried out by a contractor, and loading and hauling of ore and waste (including ore rehandling).

Flotation Plant operating costs include all consumable items (balls for the ball mill, reagents, and chemicals) power, external services, and maintenance. A contingency of 7.5% is included.

Concentrate haulage will be provided a by a contractor at a rate of $0.146 per wmt/km. The Central Plant costs include concentrate handling, cyanidation, carbon desorption, electro-winning and refining to produce gold-silver doré.

Mine closure and rehabilitation costs, as well as post-closure management for a period of 10 years were estimated. These costs are reflected as an environmental provision per tonne of ore processed over the operational life of the mine.

Administration costs were developed from first principles and based on Bulgarian labour, service and material input costs.

The average LOM mine operating cost is estimated to be $20.62/t of ore processed.

Table 21-12: LOM operating costs

Operating costs	$/t processed
Mining	8.43
Flotation Plant	7.04
Concentrate Haulage	0.53
Central Plant	2.35
Administration	1.93
Environmental Provision	0.33
All-In OPEX	**20.62**

21.4.1 Mine Operating Cost

The operating cost model is an activity and area-based cost model that has been developed from first principles and estimated on a quarterly basis per tonne mined. The key drivers in the production schedule included material movement by type (waste, low-grade, high-grade), ore type, haul route distances and gradients per period. Additional technical information, including mine and engineering design criteria, was derived from the PFS study.

Mine costs are divided into the following activities:

- Loading
- Hauling
- Drilling
- Blasting
- Drill and Blast support equipment
- Major ancillary fleet
- Minor ancillary fleet
- Pit de-watering

- Stockpile and ROM pad ore rehandle
- Maintenance workshop.

Operational costs are represented by the following elements:

- Capital or lease costs
- Fixed costs
- Energy costs
- Lubricants
- Tyres
- Wear parts
- Explosives and accessories
- Maintenance
- Rebuilds
- Labour
- Insurance.

21.4.1.1 Sources of Information

The operating costs for the study were mostly derived from supplier quotations. Where these were not available, or the provided quotes were found to not meet the request basis, CSA used internal data sources and industry benchmark prices. Grade control drilling costs were calculated from production drilling rates and the assaying costs derived from an industry benchmarked rate.

The operating costs for the primary production fleet (haul trucks, hydraulic excavators, blast-hole drilling rigs and front-end loaders) were obtained from the original equipment manufacturers (OEMs), and developed alongside the equipment performance criteria and expected or predicted fuel, oil and lubricant consumption rates. The major and minor ancillary equipment operating costs were calculated from internal database costs and developed alongside the equipment performance criteria and expected or predicted fuel, oil, and lubricant consumption rates.

Remuneration rates were determined from a review of local labour conducted by a Bulgarian-based human resources practitioner contracted by Velocity Minerals. The in-country labour costs were adjusted to calculate the total cost for each labour category. On-costs added to the base salary rate included:

- Insurance
- Food allowance
- Clothing allowance
- Transportation
- Holiday allowance
- Health insurance
- Annual bonus of 8%
- Food vouchers
- State Social Insurance (SSI).

21.4.1.2 Owner-Operator Principle

The mine plan is premised on an "owner-operated" mining operation. All mining equipment costs include complete coverage of lease-purchase as an operating cost. The lease cost is applied in equal monthly sums for the entire 60-month lease term. No contractor mark-up is included in the "owner-operator" model.

Drill and blast operations were estimated using the concept that a contractor would supply explosives, hole loading and stemming equipment, and management personnel. Costs were estimated for the equipment and supplies as per the owner-operator methods presented in all other sections of this report: a 10% mark-up has been allowed to represent the contractor's governance and profit margin.

Grade control drilling is assumed to be a contracted service and no equipment is purchased (skilled personnel will be supplied by the owner, and assay costs are allowed for separately). Within the capital cost estimate (not included in this section) buildings and container storage are allocated for the drill and blast, and grade control services.

21.4.1.3 Exclusions

The mining operating cost battery limit is at the point that the ore is fed to the crusher.

The estimation does not include:

- Construction of the initial mine access roads (temporary roads within the mining area are covered by ancillary equipment costs)
- Clearing and grubbing
- Topsoil removal.

The costs above are reflected in the appropriate cost categories elsewhere in the PFS report.

21.4.1.4 Equipment Insurance

An allowance of one % per annum on the capital purchase price is estimated for equipment insurance (paid monthly) for the leased primary mining fleet, drilling fleet and minor/major ancillary equipment.

21.4.1.5 Lubricants

Lubricant consumption and costs are estimated from indexed lubricant prices for greases, hydraulic oils, engine and gear oils and coolant. A servicing schedule was used for each of the equipment type's refill capacity to determine lubricant consumption. Where OEM refill frequency was not available, estimated refill intervals were estimated and applied to determine lubricant consumption.

21.4.1.6 Tyres

Tyre replacement cost is estimated from indexed tyre prices per tyre type. Replacement intervals are based on industry benchmarking per equipment type for medium under-foot conditions. Tyre life for the primary haul vehicles is estimated at 1,500 hours which is in-line with road haulage tyres of the same type. The FMX truck with 14 tyres may present a cost saving opportunity during the Feasibility Study once the tyre life under local conditions is understood.

21.4.1.7 Rebuild Costs

Rebuild costs are estimated for each type of equipment and assumes two major overhauls over the mine life. The cost of rebuilds has been assumed to be 60% of the capital purchase price. Rebuild intervals for the primary and secondary equipment are:

- Haul trucks 20,000 hours

- Excavators 20,000 hours
- Drill rigs 15,000 hours
- Major ancillary equipment 20,000 hours

21.4.1.8 Non-Rebuild Maintenance Costs

Maintenance costs comprise costs for routine and scheduled maintenance as recommended by the OEM. The costs comprise items such as filters and spare part replacements. Equipment OEM information that was not available required the use of a single interval applied over the life of the equipment. The scheduled maintenance costs increase as a function of operational hours. The stepped maintenance intervals used in the operational cost model were:

- Haul Trucks 4,000 hours
- Excavators 4,000 hours
- Drill-rigs 5,000 hours
- Major Ancillary Equipment 4,000 hours

In addition to these maintenance costs, a 5% allowance (calculated on the stepped weighted average over life of equipment) for non-planned maintenance, damages and repairs was applied.

21.4.1.9 Fuel Cost and Consumption

The price of diesel fuel used in the operating cost estimate is $1.243/litre based on publicly available indexed diesel prices for Bulgaria. Diesel fuel comprises approximately 15% of the total operating cost. An opportunity to source diesel at more competitive prices has a significant impact on reducing mining operating costs.

Table 21-13 below details the fuel consumption for the various activities and is calculated from cycle time and expected hourly fuel consumption rates for each type of equipment.

Table 21-13: Equipment annual fuel consumption

Fuel burn		Total	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Drill rig	Ml	4.50	0.00	0.16	0.75	0.92	0.90	0.84	0.78	0.15	0.00
D&B support	Ml	0.31	0.00	0.01	0.05	0.06	0.06	0.06	0.05	0.01	0.00
Excavator	Ml	2.44	0.00	0.15	0.47	0.50	0.45	0.41	0.35	0.08	0.04
Haul truck	Ml	3.12	0.00	0.25	0.64	0.56	0.59	0.50	0.42	0.12	0.06
Ore rehandle	Ml	0.33	0.00	0.02	0.06	0.06	0.06	0.06	0.05	0.01	0.00
Major ancillary	Ml	1.31	0.01	0.11	0.23	0.24	0.23	0.23	0.20	0.05	0.01
Minor ancillary	Ml	0.46	0.01	0.05	0.07	0.08	0.08	0.08	0.07	0.02	0.00
Total	**Ml**	**12.46**	**0.02**	**0.73**	**2.26**	**2.42**	**2.37**	**2.18**	**1.92**	**0.45**	**0.10**
Drill rig	l/t	0.11	0.06	0.07	0.10	0.12	0.12	0.12	0.14	0.08	0.00
D&B support	l/t	0.01	0.00	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.00
Excavator	l/t	0.06	0.07	0.07	0.07	0.06	0.06	0.06	0.06	0.04	0.02
Haul truck	l/t	0.08	0.13	0.12	0.09	0.07	0.08	0.07	0.07	0.06	0.03
Ore rehandle	l/t	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.00	0.00
Major ancillary	l/t	0.03	0.57	0.05	0.03	0.03	0.03	0.03	0.04	0.03	0.01
Minor ancillary	l/t	0.01	0.90	0.03	0.01	0.01	0.01	0.01	0.01	0.01	0.00
Total	**l/t**	**0.31**	**1.74**	**0.35**	**0.31**	**0.30**	**0.32**	**0.31**	**0.34**	**0.24**	**0.07**

21.4.2 Activity Descriptions

21.4.2.1 Loading

Loading of ore and waste rock into the 55-tonne capacity haul trucks will be performed using up to three 90 tonne class excavators. The ore and waste, although blasted on 5 m benches, will be extracted on 2.5 m flitches. The CAT 390 hydraulic excavator was selected for this service. The excavator's instantaneous production rate of approximately 672 LCM/hr, bucket capacity of 5.7 LCM and six passes per truck load delivers a productivity of approximately 5,146 hr/yr.

Hourly operating costs are presented below in Table 21-14.

Table 21-14: Excavator hourly operating cost

CAT 390		
Resource area	**Unit**	**Value**
Owning costs	$/hr	36.7
Energy costs	$/hr	69.6
Labour	$/hr	37.6
Lubricants	$/hr	3.4
Tyres	$/hr	0.0
Ground engaging tools	$/hr	18.3
Rebuild costs	$/hr	8.2
Variable maintenance rates	$/hr	18.4
Total hourly cost (excl. Owning)	**$/hr**	**155.5**
Total hourly cost (incl. Owning)	**$/hr**	**192.2**

The mine schedule utilized the hourly operating cost and productivity estimates from the mine schedule to develop the cost per tonne for each material type in each mining period.

21.4.2.2 Hauling

Hauling of ore and waste will be by 55 tonne class modified rock-body Volvo FMX diesel haul trucks. The haul truck productivity is based on "queuing theory" that discounts truck and excavator system productivity for hauling time, loading time, and the number of trucks in the system. The purpose of the "queuing theory" is to determine the lowest unit operating cost for a truck and excavator system but constrained by system productivity and optimal truck numbers. At a bowl capacity of 33 LCM and a 95% fill factor, planned annual productive hours of approximately 5,065 hr/yr are achieved. The total number of trucks utilized varies during the mine schedule but averages 13 over most of the mine life and peaks at 15 in a single quarter. Hourly operating costs are presented below in Table 21-15.

The mine schedule utilizes the hourly operating cost and productivity estimates from the mine schedule to develop the cost per tonne for each material type in each mining period.

Table 21-15: Haul truck hourly operating cost

Volvo FMX 540 10x4 55 t		
Resource area	**Unit**	**Value**
Owning costs	$/hr	16.6
Energy costs	$/hr	21.1
Labour	$/hr	34.9
Lubricants	$/hr	1.0
Tyres	$/hr	11.2
Ground engaging tools	$/hr	5.0
Rebuild costs	$/hr	3.0
Variable maintenance rates	$/hr	16.0
Total hourly cost (excl. Owning)	**$/hr**	**92.2**
Total hourly cost (incl. Owning)	**$/hr**	**108.9**

21.4.2.3 Drilling and Blasting

Drilling and blasting will be completed by a contractor. Costs were developed from first-principle estimates produced by CSA Global, and includes a 10% markup to cover contractor management and profit margin. Explosives will be supplied on a just-in-time basis by the contractor with no on-site storage.

By way of verification, two contract proposals were obtained by Velocity and compared to the first-principles estimate produced by CSA Global (as shown in Table 21-16). CSA Global considers that Contractor 1 will probably not achieve a P_{80} 300 mm for competent ore based on a review of their provided blasting patterns. CSA Global also determined that both third party contractors overestimated the oxide rock blasting requirements. Contractor 2 also over-simplified their cost estimate in that no differentiation was made for ore and waste. For the PFS, the CSA Global estimate was utilized as being most appropriate.

Table 21-16: Drill and blast cost comparison

Mining D&B costs	PFS	Waste $/t				Ore $/t				Total $/t			
	Proportion	Contract1	Contract2	PFS	PEA	Contract1	Contract 2	PFS	PEA	Contract1	Contract2	PFS	PEA
Oxide	0.15	0.67	0.89	0.44	0.47	0.67	0.89	0.43	0.64	0.67	0.89	0.43	0.58
Transitional	0.15	0.67	0.89	0.51	0.87	0.67	0.89	0.73	1.04	0.67	0.89	0.66	0.98
Sulphide	0.70	0.83	1.06	0.85	0.81	0.83	1.06	1.07	0.97	0.83	1.06	0.99	0.91
Total	**1.00**	**0.78**	**1.01**	**0.74**	**0.77**	**0.78**	**1.01**	**0.93**	**0.93**	**0.78**	**1.01**	**0.86**	**0.87**

PFS D&B Costs exclude $0.04/tonne D&B Support Equipment (TLB Stemming).
PFS pre-split costs at $0.21/bcm. PEA Pre-split costs at $0.35/tonne. No pre-split specified in contracts.

A top-hammer blasthole drill rig capable of drilling 76 mm holes was selected. The drill rig will be capable of drilling holes with a diameter 76 mm to 152 mm, at the rate required. To allow for pre-split coverage, the carousel will allow a drill depth of 31 m to be achieved. The planned productivity for the drill rigs was estimated at 6,178 hr/yr. Hourly operating costs are presented in Table 21-17.

The mine schedule utilizes the hourly operating cost and productivity estimates to develop the cost per tonne for each material type in each mining period.

Table 21-17 Drill rig hourly operating cost

MD 5150C		
Resource area	**Unit**	**Value**
Owning Costs	$/hr	30.5
Energy Costs	$/hr	77.1
Labour	$/hr	32.3
Lubricants	$/hr	3.5
Tyres	$/hr	0.0
Ground Engaging Tools	$/hr	100.7
Rebuild Costs	$/hr	10.9
Variable Maintenance Rates	$/hr	42.6
Total Hourly Cost (excl. Owning)	**$/hr**	**267.0**
Total Hourly Cost (incl. Owning)	**$/hr**	**297.5**

To support the drilling and blasting operation a CAT 450F TLB was selected. This was estimated to cost $83 /hr for approximately 350 hr/month. The support equipment hours were estimated at 25% of total drill rig operational time.

21.4.2.4 Major Ancillary Fleet

The major ancillary fleet consists of the following equipment (or equivalent):

- 1 x CAT D8 Dozer (Day works $115 /hr @ 150 hr/month)
- 1 X CAT 980 FEL (Day works $121 /hr @ 150 hr/month)
- 1 X CAT 730 Water bowser (Day works $109 /hr @ 176 hr/month) Note 1
- 1 x CAT 730 Fuel bowser (Day works $109 /hr @ 150 hr/month)
- 1 x CAT 150 Grader (Day works $88 /hr @ 150 hr/month)

Note 1: Water bowser (for dust suppression) hours are based on seasonally adjusted precipitation and evaporation on a quarterly basis (minimum 62 hr/month, maximum 377 hr/month).

Note 2: "Day works" rate is a wet rate inclusive of owning costs. A minimum monthly usage rate of 150 hr/month is assumed for all ancillary equipment.

21.4.2.5 Minor Ancillary Fleet

The minor ancillary fleet consists of the following equipment (or equivalent):

- 1 X CAT 325D rock breaker (Day works $77.5 /hr @ 2,160 hr/yr)
- 2 x 65-seater busses (Day works $57.3 /hr @ 1,500 hr/yr)
- 1 X 80t low-loader with crane (Day works $179 /hr @ 480 hr/yr)
- 4 x Alight MK4 lighting plants (Day works $5.3 /hr @ 3,360 hr/yr)
- 6 x Toyota Hilux light diesel vehicles (LDV) (Day works $14.5 /hr @ 1,440 hr/yr)
- 1 x CAT 980TH tyre handler (Day works $288 /hr @ 480 hr/yr)
- 1 x Dynapac CA602D 19 t compactor (Day works $145 /hr @ 480 hr/yr)

Note: LDV's are for mining maintenance and operations support only.

21.4.2.6 Grade Control

The grade control cost estimate is based on RC drilling on a 5 m x 8 m pattern taking samples at one metre intervals. The 5 m x 8 m pattern is based on benchmark data for similar sized SMU's. Drillholes are planned to be 16 m deep (to give a one metre overlap to the adjacent lower pattern and to avoid taking samples in the broken ore at the bench floor). The PFS assumes that ore and waste are intermingled but that the grade control drilling coverage can be limited to a 1:1 waste : ore ratio.

Contractor costs for grade control drilling were obtained by Velocity at €15 /m. The assay cost was based on other operations in the region for fire assay including supporting costs. Table 21-18details the grade control drilling, sampling and analytical cost estimate.

Table 21-18: Grade control cost

Grade control assumptions	Units	LOM	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Total volume to be grade controlled	bcm	9,256,703	3,308	250,330	1,828,318	2,046,565	1,737,135	1,356,662	1,666,081	368,304	0
Oxide - Waste	bcm	708,042	1,654	92,793	332,133	190,189	74,203	17,071	0	0	0
Trans - Waste	bcm	729,833	0	28,922	217,890	319,300	116,987	46,734	0	0	0
Fresh - Waste	bcm	3,080,860	0	3,450	320,564	495,894	677,378	614,527	784,896	184,152	0
Oxide - Ore	bcm	795,186	1,654	92,793	419,277	190,189	74,203	17,071	0	0	0
Trans - Ore	bcm	765,632	0	28,922	217,890	355,099	116,987	46,734	0	0	0
Fresh - Ore	bcm	3,177,149	0	3,450	320,564	495,894	677,378	614,527	881,185	184,152	0
Ore:Waste volume sample ratio	Ratio	1.0	1.0	1.0	1.1	1.0	1.0	1.0	1.1	1.0	0.0
Grade control drilling											
Drill-hole spacing (m) – Burden	m	5	5	5	5	5	5	5	5	5	0
Drill-hole spacing (m) – Spacing	m	8	8	8	8	8	8	8	8	8	0
Area per drill hole	m^2	40	40	40	40	40	40	40	40	40	0
Number of drill holes	Each/a	15,428	6	417	3,047	3,411	2,895	2,261	2,777	614	0
Sample interval (m)	m	1	1	1	1	1	1	1	1	1	0
Grade Control Bench height	m	15	15	15	15	15	15	15	15	15	0
Dip of holes	°	-90	-90	-90	-90	-90	-90	-90	-90	-90	0
Length of hole per m depth	m	1	1	1	1	1	1	1	1	1	0
Hole overlap per campaign bench	m	1	1	1	1	1	1	1	1	1	0
Metres drilled per period	m	246,845	88	6,675	48,755	54,575	46,324	36,178	44,429	9,821	0
Sample length (each)	m	1	1	1	1	1	1	1	1	1	0
Unsampled collar length	m	1	1	1	1	1	1	1	1	1	0
Hole length	m	16	16	16	16	16	16	16	16	16	0
Number of samples per drill hole within bench	no.	15	15	15	15	15	15	15	15	15	0
Total number of samples per year	no.	231,418	83	6,258	45,708	51,164	43,428	33,917	41,652	9,208	0
RC drilling cost ($318/hr)	$/lin m	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	17.7	0.0
Assay cost	$ ea	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	0.0
Assay and drilling cost	$/lin m	27.7	27.7	27.7	27.7	27.7	27.7	27.7	27.7	27.7	0.0
Grade control cost per year	$	6,675,111	2,385	180,516	1,318,420	1,475,801	1,252,667	978,304	1,201,430	265,588	0
Tonnes of ore per sample	t/sample	49.9	46.4	46.8	50.2	50.2	50.1	50.8	53.8	51.2	0.0
Cost per BCM sampled	**$/m³**	**0.72**	**0.72**	**0.72**	**0.72**	**0.72**	**0.72**	**0.72**	**0.72**	**0.72**	**0.00**
Cost per total BCM	**$/m³**	**0.44**	**0.44**	**0.20**	**0.45**	**0.46**	**0.43**	**0.36**	**0.54**	**0.68**	**0.00**
Cost per mined tonne	**$/t mined**	**0.17**	**0.19**	**0.09**	**0.18**	**0.18**	**0.17**	**0.14**	**0.21**	**0.27**	**0.00**
Cost per ore tonne	**$/tore**	**0.56**	**0.62**	**0.61**	**0.57**	**0.57**	**0.57**	**0.57**	**0.53**	**0.56**	**0.00**

21.4.2.7 ROM Pad High-Grade Ore Rehandle

HG ore will be hauled from the pit to the ROM pad where it is estimated that 75% of the material will be direct tipped into the crusher. Approximately 25% of the HG ore will be sent to the ROM stockpile when ore production is balanced to plant capacity. Above plant capacity, all HG ore is assumed stockpiled (the pad has a ~260 kt stockpile capacity) near the crusher feed bin. Re-handling of the HG ore from these stockpiles will be by a FEL (5.5 m^3 bucket) capable of loading ~500 t/h. A minimum operating rate of 360 hr/month has been allocated for the FEL rehandle in the operational cost model, which generates an HG ore rehandle cost of 0.19 $/t.

The HG ore stockpile is designed for a height of 8 m and a capacity of 260 kt. Cost optimization of the production schedule found that it was necessary to increase the HG ore storage to 500 kt. Due to space constraints some of the HG ore will have to be stockpiled elsewhere which will not permit loader-only re-handling. This re-handling is included in the financial model at 0.58 $/t when the HG material movement exceeds 100 kt per quarter.

The PFS rehandle scheduling assumes that the Oxide, Transitional and Sulphide ores can be blended, which is supported by the metallurgical testwork.

21.4.2.8 Low-Grade Ore Rehandle

When ore mining from the Phase 2 pit is complete, the LG ore (in a stockpile on the WRD) will be loaded into the Volvo FMX 10 x 4 trucks by the large FEL (12.2 m^3 bucket) for delivery (~500 m) to the crusher feed bin. Two trucks will be required to service the FEL and maintain the feed rate required (5,000 tpd). Additional support units required are:

- 1 x D8 dozer operating at 10 hr/month
- 1 x water bowser operating at 10 hr/month
- 1 x grader operating at 10 hr/month
- General maintenance (workshop).

Equipment spares and direct maintenance labour have been included in the equipment hourly operating costs. The cost of loading and hauling the LG ore is calculated at 0.58 $/t.

21.4.2.9 General Workshop

Indirect engineering labour and general workshop costs are included under the engineering costs; personnel requirements were estimated at $19,773 $/month for 5 positions including:

- Workshop Foreman
- Boilermaker
- Auto-electrician
- Boilermakers assistant
- Tyre Fitter.

Other engineering indirect expenses are outlined in Table 21-19below.

Table 21-19: Engineering other indirect costs

Engineering Indirect Expenses	$/pa	$/mth
Tools and Equipment	72,000	6,000
Hydrocarbon Management	254,400	21,200
Tyre Consumables	18,000	1,500
Total	**344,400**	**28,700**

21.4.2.10 Mining Administration Costs

Mine administration costs include items such as mine management personnel, administration, IT costs, consultants, and management light vehicle expenses, and general mine overheads. Table 21-20 and Table 21-21 set out the mine administration costs.

Table 21-20: Mining administration cost

Mine Administration Costs Rozino Project		Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Total
Days per annum		365	365	365	365	365	365	365	365	329	0	
Labour with on-costs	$k	134	238	607	672	672	674	613	134	61	0	3,807
Education/Seminars/Scholarships	$k	1	1	3	3	3	3	3	1	0	0	17
Light Vehicle Lease, Maintenance, Fuel	$k	8	14	37	41	41	41	37	8	4	0	229
Safety supplies	$k	1	2	5	6	6	6	5	1	1	0	34
Supplies for engineering and geology	$k	2	4	11	12	12	12	11	2	1	0	68
Office Supplies	$k	1	2	5	6	6	6	5	1	1	0	34
Mobile Phones and charges	$k	1	2	4	5	5	5	4	1	0	0	27
Consultants and external services	$k	1	2	5	6	6	6	5	1	1	0	34
Miscellaneous pit costs	$k	2	4	11	12	12	12	11	2	1	0	68
Travel Costs	$k	7	13	33	36	36	36	33	7	3	0	204
Software upgrades and support contracts	$k	38	67	172	190	190	191	173	38	17	0	1,076
Computers	$k	1	3	7	7	7	7	7	1	1	0	41
Total	**$k**	**199**	**353**	**899**	**996**	**996**	**999**	**908**	**198**	**90**	**0**	**5,638**
Cost per tonne mined	$/t	16	0.17	0.12	0.12	0.14	0.14	0.16	0.11	0.06	0.00	0.14
Tonnage mined	kt	12	2,116	7,209	7,992	7,334	6,925	5,729	1,876	1,586	0	40,780

Labour positions for mine administration are included in Table 21-21.

Table 21-21: Mining administration labour positions

Position	Count
Mining Manager/Superintendent	1
Mine Clerk	1
Technical services superintendent	1
Senior mining engineer	1
Mine planning/production engineer	1
Chief Geologist	1
Geologist	1
Grade controller	5
Chief Surveyor	1
Mine Surveyor	3

21.4.2.11 Total Mining Operational Labour

Total labour requirements are detailed in Table 21-22. The numbers are inclusive of unavailable labour (personnel on leave) three shifts per day (four crews in total).

Table 21-22: Total Mine Operations Labour Requirements

Labour Complement	Unit	Peak	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Drill Rig Operators	No	12	4	4	12	12	12	12	12	8	0
D&B Support Operators	No	1	1	1	1	1	1	1	1	1	0
Excavator Operators	No	12	8	8	12	12	12	12	12	8	2
Haul Truck Operators	No	60	44	44	52	48	60	48	48	40	8
Major Anc. Equipment Operators	No	7	7	7	7	7	7	7	7	7	5
Minor Anc. Operators	No	16	7	13	16	16	16	16	16	10	0
Indirect Labour	No	0	0	0	0	0	0	0	0	0	0
Engineering Labour	No	53	33	38	49	47	53	47	47	36	4
Total Personnel	**No**	**162**	**104**	**115**	**150**	**144**	**162**	**144**	**144**	**111**	**19**
Personnel in Training (Note 1)	No	66	66	23	0	0	0	0	0	0	0
Operational Personnel	**No**	**162**	**38**	**92**	**150**	**144**	**162**	**144**	**144**	**111**	**19**

Note 1: Some personnel will be recruited ready for work, but it is assumed that half of the personnel will require on-the-job training. Personnel in training are not considered productive for 3 months.

The remuneration scales per position were used to calculate the labour costs. Labour costs were assigned to each position fully loaded.

21.4.2.12 Mine Operating Cost Summary by Resource

The overall operating costs for the mine are summarized by resource in F*igure* 21-5.

Table 21-23 and shown in F*igure* 21-5.

Table 21-23: Operating Cost Summary by Resource (2015-2019)

Total Mining Costs – Resource	Units	LOM	Steady State (Y1 - Y4)
Lease	$M	18.4	11.5
Fixed Costs	$M	11.1	8.2
Power	$M	0.0	0.0
Fuel	$M	15.5	11.5
Lubricants	$M	0.9	0.6
Tyres	$M	2.4	1.8
Wear Parts	$M	9.9	7.5
Explosives	$M	7.7	5.9
Maintenance, Variable	$M	9.3	7.0
Buckets/Bodies/Rims	$M	3.2	2.4
Insurance	$M	0.3	0.2
Operator Labour	$M	17.4	10.7
Maintainer Labour	$M	4.4	2.8
Semi-skilled Labour	$M	3.3	2.1
Skilled Labour	$M	5.6	4.0
Total Mining Operational Cost	**$M**	**109.5**	**72.1**

Unit Mining Costs - Resource	Units	LOM	Steady State (Y1 - Y4)
Lease	$/t	0.45	0.39
Fixed Costs	$/t	0.27	0.28
Power	$/t	0.00	0.00
Fuel	$/t	0.38	0.39
Lubricants	$/t	0.02	0.02
Tyres	$/t	0.06	0.06
Wear Parts	$/t	0.24	0.25
Explosives	$/t	0.19	0.20
Maintenance, Variable	$/t	0.23	0.24
Buckets/Bodies/Rims	$/t	0.08	0.08
Insurance	$/t	0.01	0.01
Operator Labour	$/t	0.43	0.36
Maintainer Labour	$/t	0.11	0.09
Semi-skilled Labour	$/t	0.08	0.07
Skilled Labour	$/t	0.15	0.13
Total Mining Operational Cost	**$/t**	**2.70**	**2.59**

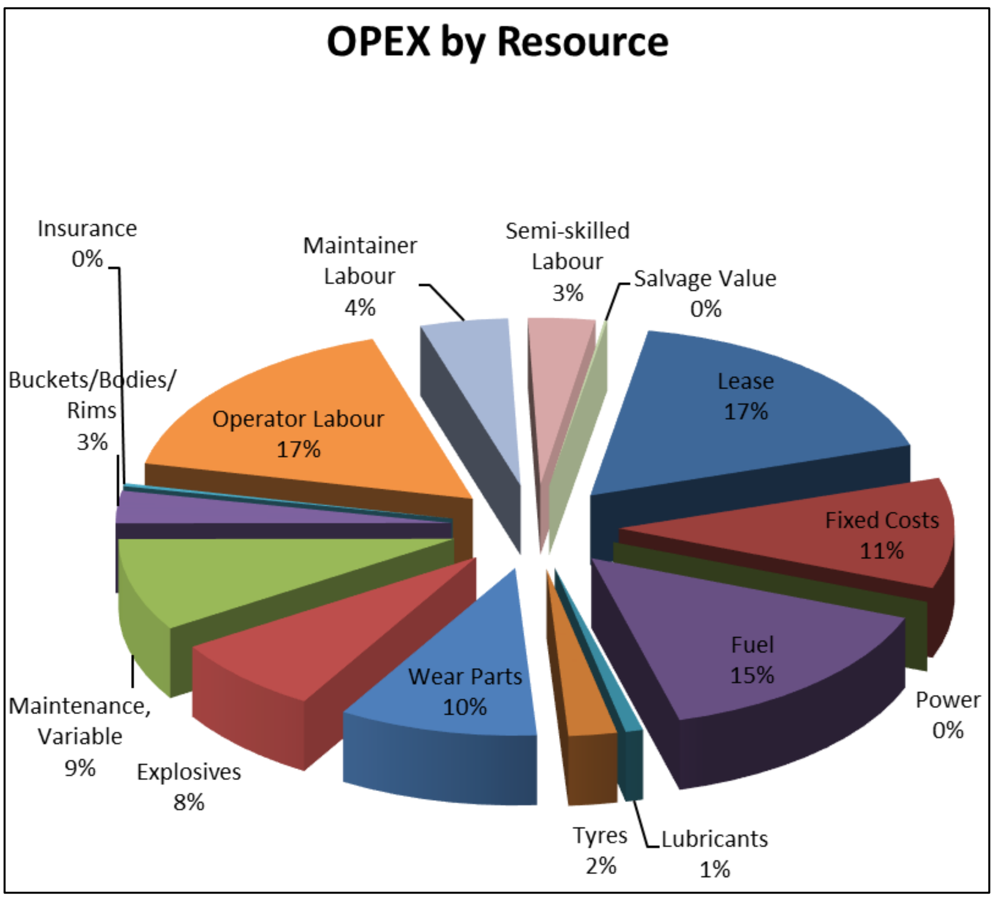


Figure 21-5: Mine operating cost by resource

Note: Costs are shown as $/t material moved, which includes rehandle. Costs are before redirection to capital for Years -2 and -1.

21.4.2.13 Mine Operating Cost Redirection to Capital

Some operating costs are redirected to capital in the financial analysis:

- All mine operating expenses in the construction period (other than overhaul to the TMF). This amounts to $8.9 M ($1.4 M in Year -2 and $7.5 M in Year -1).
- All overhaul of waste to the TMF. This amounts to $0.9 M over the life of the project.

Table 21-24: Summary of Operating Costs with Re-direction to Capital

	Unit	LOM Total	Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Operating Mining Costs	$m	99.7	0.0	0.0	16.8	19.6	20.0	19.0	17.1	6.0	1.1
Drill and Blast	$m	36.1	0.1	1.5	5.5	7.3	7.3	6.4	6.4	1.6	0.0
Load and Haul	$m	44.1	0.8	4.1	7.0	7.2	7.8	7.9	5.8	2.7	0.9
Ancillary, Support Equipment	$m	12.2	0.3	1.4	1.8	1.8	1.9	2.0	1.9	0.8	0.1
Workshop General	$m	2.1	0.0	0.1	0.4	0.4	0.4	0.4	0.3	0.1	0.0
ROM Ore Rehandle Cost plus HG trucked rehandle	**$m**	**2.5**	**0.0**	**0.0**	**0.3**	**0.4**	**0.6**	**0.5**	**0.5**	**0.3**	**0.0**
Dewatering Cost	$m	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0
Mine G&A	$m	5.6	0.2	0.3	0.9	1.0	1.0	1.0	0.9	0.2	0.1
TMF overhaul capital redirection	$m	-0.9	0.0	-0.2	-0.4	0.0	-0.2	-0.1	0.0	0.0	0.0
Mine pre-strip capital redirection	$m	-8.9	-1.4	-7.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0

Costs are shown as $/t total mined or $/t ore mined. The tonnes in this measure do not include rehandle. Including the construction period (Years -1 and -2) the mining costs are $2.70/t total mined. When considering only operating years, the mine operating cost is $2.55/t total mined. The cost per tonne mined including the capital period is shown in Figure 21-6.

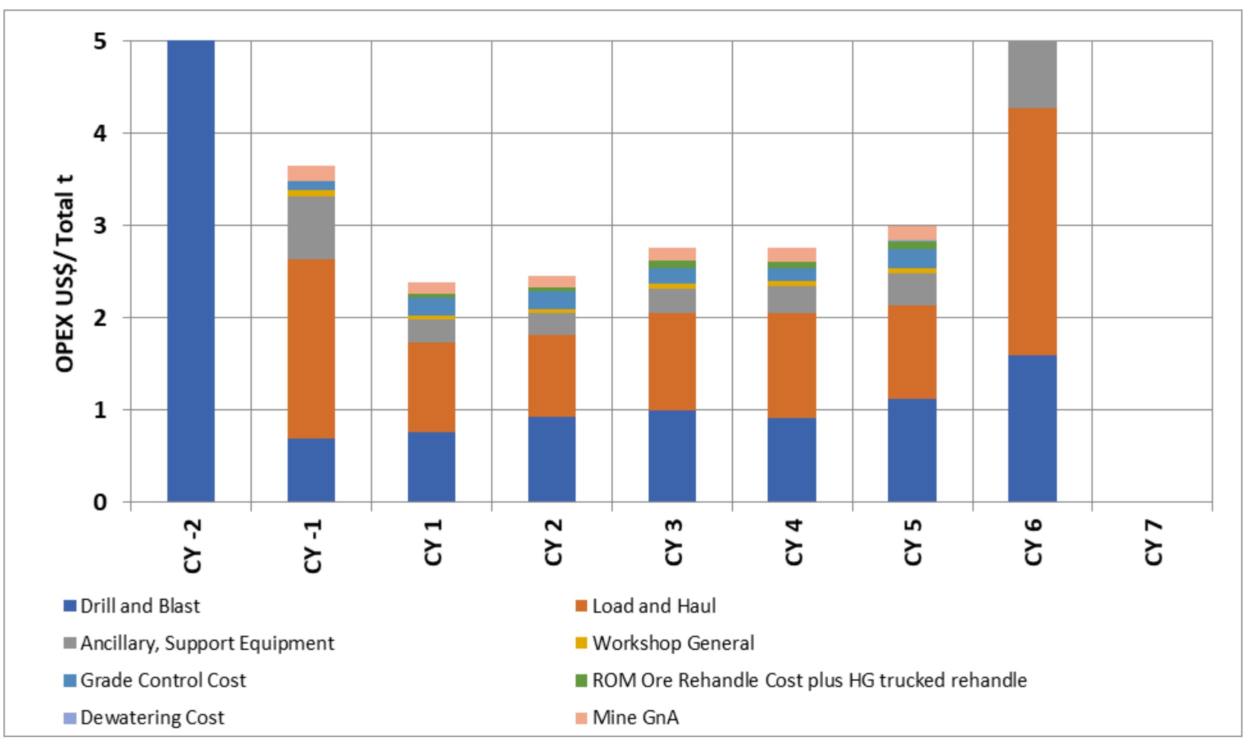


Figure 21-6: Total mine operating cost summary by year

21.4.2.14 Flotation Plant Operating Cost

The Flotation Plant operating costs were estimated from first principles by developing equipment dimensions, capacities, efficiencies and consumable consumption rates. From the testwork data on ore type and reagent consumptions, Oxide and Transitional ore behaved similarly. The Sulphide ore is harder, and more abrasive but was more amenable to flotation than Oxide and Transitional ore and required a separate processing cost estimation.

It was estimated that the ball mill will have a similar throughput rate for Oxide/Transitional ore and Sulphide ore. The ball mill is the key component controlling plant throughput rate. With the same throughput rate for all ore types, the plant labour and external services (essentially consulting services such as specialized maintenance and metallurgical advice) do not alter. The Oxide/Transitional ore has greater reagent consumption than the Sulphide ore as it requires sulphidising chemicals to aid flotation. The Sulphide ore is slightly harder and more abrasive than Oxide/Transitional, so this leads to higher wear rates, hence increased power, consumables (balls and wear plates) and maintenance (equipment replacements parts) costs. It is the different reagent usage that drives most of the difference in the operating costs between the ore types.

Table 21-25: Flotation plant operating costs for Oxide and Transitional ore

Item	Total $/year	$/t feed (ROM)	% of Total Cost
Plant Labour	2,014,260	1.15	15%
Power	2,160,381	1.23	16%
Consumables	2,578,340	1.47	19%
Reagents	2,623,722	1.50	20%
External Services	750,000	0.43	6%
Maintenance	2,250,000	1.29	17%
Contingency (7.5%)	928,253	0.53	7%
Total	**13,304,955**	**7.60**	**100%**

Table 21-26: Flotation plant operating costs for Sulphide ore

Item	Total $/year	$/t feed (ROM)	% of Total Cost
Plant Labour	2,014,260	1.15	17%
Power (Sulphides)	2,331,141	1.33	20%
Consumables	2,830,100	1.62	24%
Reagents	627,693	0.36	5%
External Services	812,500	0.46	7%
Maintenance	2,437,500	1.39	21%
Contingency (7.5%)	828,990	0.47	7%
Total	**11,882,183**	**6.79**	**100%**

An operating contingency of 7.5% was applied.

Flotation Plant battery limits commence at the ROM ore bin hopper, and end with the concentrate being loaded onto trucks for transport to the Central Plant.

21.4.2.15 Start-Up Impacts and Ore Mixes

The Flotation Plant feed schedule is based on a processing nameplate capacity of 1.75 Mtpa with wet commissioning commencing at the start of Year 1 and ramp-up to steady state production 5,000 tpd being achieved within the first year (65% Q1, 85% Q2, 95% Q3, 99% Q4). Figure 21-7 illustrates graphically plant feed tonnages, ore types and the expected gold head grade (DF denotes "Direct feed from pit" and SP denotes "Stockpile subtractions", x-axis is in quarterly periods commencing Period 1 at the start of construction).

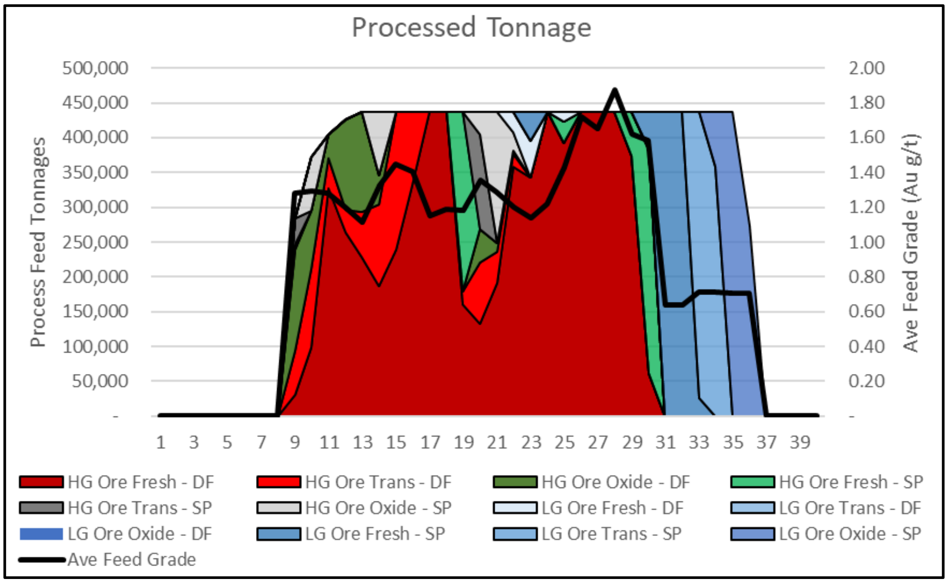


Figure 21-7: Process feed schedule per material type and source

No negative impacts are anticipated from the blending of different ore types in the feed. The plant reagents, consumables, and power are directly variable to throughput rate and ore type blend.

21.4.2.16 Plant labour cost

The plant labour cost estimation is based on a 3-shift/four crew basis. Some positions cover only two shifts (with call-out coverage) and some positions are day shift only. A total of 69 personnel are required to cover plant operations and standard maintenance. Large maintenance tasks will be carried out by specialist contractors included within the external services cost area.

Table 21-27: Flotation Plant Labour Costs

PLANT LABOUR POSITONS					
Position	**Crew**	**Regime**	**Shift**	**Dayshift**	**Total**
Plant Manager / Superintendent	1	Dayshift	0	1	1
Clerks / Secretary	1	Dayshift	0	1	1
Metallurgist	1	Dayshift	0	1	1
Shift boss	1	Shift	4	1	5
Operators	5	Shift	4	0	20
Workers	4	Shift	4	0	16
Light equipment operator / driver	1	Dayshift	0	1	1
Sample Prep	2	Shift	2	0	4
Reagent Mixer	1	Shift	2	0	2
Assay lab: Chief Chemist	1	Dayshift	0	1	1
Assay lab: Chemist	1	Shift	2	0	2
Assay lab: Technician	1	Shift	2	0	2
Maintenance Manager	1	Dayshift	0	1	1
Workshop Clerk / planner	1	Dayshift	0	1	1

PLANT LABOUR POSITONS					
Position	**Crew**	**Regime**	**Shift**	**Dayshift**	**Total**
Workshop foreman	1	Dayshift	0	1	1
Electrical engineer	1	Dayshift	0	1	1
Electrician	1	Shift	2	0	2
Instrumentation and Controls technician	1	Shift	2	0	2
Plumber/Greaser	1	Dayshift	0	1	1
Welder (Boilermaker)	1	Shift	2	0	2
Metalworker	1	Shift	2	0	2
TOTAL	**29**	**-**	**28**	**11**	**69**

21.4.2.17 Power Cost

Power supplied by the 110 kV overhead transmission line will have a cost of $0.059/kWh. The price quoted excludes taxes and is based on the Bulgarian regulated market (existing electricity prices approved by the Decision № Ц-11 of 01.07.2018 of the Commission for Energy and Water Regulation dated 1 July 2018).

The unit rate calculation is based on assumptions as set out in Table 21-28.

Table 21-28: Electricity supply assumptions and costs

Item	Units	Rate
Basic transmission rate	$ / MWh	5.70
Basic network access rate	$ / MWh	0.24
Green energy tax	$ / MWh	11.35
Subtotal	$ / MWh	17.29
Active energy charge for daylight: 16 hours	$ / MWh	50.75
Active energy charge for night: 8 hours	$ / MWh	22.23
Average price before taxes	**$ / MWh**	**58.53**
Average price before taxes	**$ / kWh**	**0.059**

21.4.2.18 Reagent Cost

Reagent costs are higher for Oxide/Transitional than Sulphide ore. It is assumed the reagent costs are not negatively impacted by mixing the ore types; they are shown here separately for clarity. Reagent cost is calculated on the proportion of the different ore types in the feed. All reagent costs are FOB Rozino. Reagent consumption rates were determined by the 2019 and 2020 flotation test work.

21.4.2.19 Consumable Cost

Consumables costs were estimated separately for Oxide/Transitional ore and Sulphide ore on the basis of abrasion rates, and ore hardness and verified against data from similar projects. The costs for each ore type are shown separately for clarity.

Ball consumption was based on the ore characteristics; 0.98 kg/t for Sulphide ore and 0.90 kg/t for Oxide/Transitional ore.

Error! Not a valid link.Error! Not a valid link.

Error! Not a valid link.*Table 21-29: Annual and per Tonne Cost of the Process Plant Consumables for the Sulphide Ore Type*

Area	**Item**	**Total Consumed (steel tpa)**	**Unit Cost ($/t steel)**	**Total Cost ($/year)**	**Total Cost ($/t ore)**
Primary Crushing (Jaw Crusher)	Liners	11.4	3,500	39,900	0.02
Secondary Cone	Liners	12.6	3,500	44,100	0.03
Tertiary Cone	Liners	13.2	3,500	46,200	0.03
Ball Milling	Balls	1,715	1,200	2,058,000	1.18
	Liners	52.5	6,000	315,000	0.18
Chutes	Liners (AR400)	3.6	4,000	14,400	0.01
	Lining (Rubber)			50,000	0.03
Laboratory costs	All in			262,500	0.15
TOTAL		1,808		2,830,100	1.62

21.4.2.20 External Services Cost

External services cover any external contracts required to operate the plant, such as metallurgical testwork, specialized safety services, specialized maintenance labour, technical training and plant management advisory services. The external services cost is estimated at 3% per annum of the equipment capital cost for Oxide/Transitional ore and 3.25% for Sulphide ore. This is based on Halyards operating experience, considering ore type and anticipated general plant duty.

21.4.2.21 Maintenance Cost

Maintenance cost is estimated at 9% of the equipment capital cost on an annual basis for Oxide / Transitional and 9.8% for Sulphide ore. This is based on Halyards operating experience, with consideration of the type of ore and general plant duty expected at the Rozino Gold Project. The maintenance cost covers all routine equipment replacement, parts and any maintenance contractor services that may be utilized. It also covers the mill maintenance (workshop) operating costs.

21.4.3 Concentrate Haulage

Concentrate haulage costs were obtained by a quotation from Gorubso-Kardzhali AD based on 30 tonne capacity trucks hauling the concentrate 85 km to Kardzhali. Approximately twelve truck loads will be required to be delivered daily as the Rozino plant has limited floor storage capacity. The filter system is designed to produce a concentrate at a moisture content of approximately 12% or less before transportation. The concentrate moisture content was set at 12% for costing purposes.

The concentrate haulage cost per tonne was estimated at $13.92/dmt. The haulage rate of $12.43/wmt for the 85 km trip is $0.146/t/km. This is considered a reasonable rate for hauling loose product in Europe.

Concentrate haulage is 2.6% of the total operating cost.

21.4.4 Central Plant

Gorubso-Kardzhali AD provided a quotation to process the concentrate at $62.16/t. The processing cost is all inclusive and delivers doré as the final product.

The concentrate will arrive at the Central Plant, where it will be stockpiled in a fenced and secure shelter. The concentrate will be monitored and tracked through the CIL plant. The concentrate will be loaded into a feeder to a mixing tank by a front-end loader. The concentrate will then pass through a stirred mill and classification system to produce a P_{80} 20 µm product. This product will be thickened before being added to carbon-in-pulp cyanide leach tanks. The carbon will be cold stripped to remove copper before secondary stripping. This will be followed by electro-winning and finally smelting to produce a doré bar. The tailings will be detoxified and discharged to the Gorubso TMF for storage.

The Central Plant cost was also independently estimated by CSA Global to within 5% of the cost provided by Gorubso and thus verified. Although a blended concentrate will be processed, cyanide and detoxification chemical consumption (the two largest costs) was based on sulphide ore concentrate processing.

21.4.5 General Administration

The General Administration cost was developed from direct application of personnel required for the various functions as well as the various overheads related to general mine administration, safety and security, environmental management, and human resources:

- Administration (all fiscal operations, operations manager, external road maintenance, community, and public relations)

- Safety and security
- Environmental and permitting.

21.4.6 Human Resources

Note that direct personnel costs such as meals are not included in administration but are distributed to each person as a labour on-cost.

Labour costs constitute 30% of general administration costs.

A detailed assessment of the administration cost is provided in Section 15.2.14.

The costs were estimated per period and are repeated in Table 21-30.

Administration costs in Year -2 and Year -1 were categorized as capital (owner's administration) and total $2.9 M.

Administration costs average about $3.5 M per annum other than during the construction period. They are reduced when the LG ore is processed in the last two years of the operation.

21.4.6.1 Closure Cost Estimation

Closure and post-closure costs are based on the following assumptions:

- Utilization of mining and ancillary equipment during progressive closure and final closure (Year 8).
- Complete dismantling of surface facilities and the removal of all equipment.
- Concrete slabs remaining in situ and covered by 0.40 m of imported topsoil.
- Placement of topsoil, to a thickness of approximately 0.15 m, on all reclaimed areas from topsoil stockpiles created while clearing, grubbing and pre-stripping at the commencement of mining operations.
- Revegetation of reclaimed areas with 60% indigenous forest and 40 % mixed grassland and other indigenous vegetation.
- Active post-closure management for five years after mining operations terminate, and passive closure for a further period of 5 years.
- Table 21-31 sets out the anticipated costs for closure rehabilitation and post-closure management. Funding for closure will be by means of a provision calculated by total closure cost divided by processed ore tonnes.
- It is recognized that the relatively short life of the operation will mean that many of the structures and pieces of equipment will be able to be salvaged and re-used or sold. No revenue thus generated is considered to defray closure costs.

Table 21-30: Scheduled Administration Costs

ADMINISTRATION COST											
DESCRIPTION	Year	-2	-1	1	2	3	4	5	6	7	Total
Days per annum	days	365	365	365	365	365	366	365	365	329	
TOTAL ADMIN COSTS											
Administration	$k	668	969	1850	2039	2039	2044	1994	1629	1425	13019
Safety and Security	$k	197	286	545	600	600	602	587	480	420	4319
Environmental and Permitting	$k	171	247	471	519	519	521	508	415	363	3734
Human Resources	$k	136	197	376	414	414	415	405	331	290	2978
Total	$k	1172	1699	3243	3572	3572	3582	3494	2856	2497	25688
TONNAGE MILLED											
Total	kt	0	0	1488	1750	1750	1750	1750	1750	1586	11824
PER TONNE MILLED											
Administration	$/unit	0.00	0.00	1.24	1.16	1.16	1.17	1.14	0.93	0.90	1.10
Safety and Security	$/unit	0.00	0.00	0.37	0.34	0.34	0.34	0.34	0.27	0.26	0.37
Environmental and Permitting	$/unit	0.00	0.00	0.32	0.30	0.30	0.30	0.29	0.24	0.23	0.32
Human Resources	$/unit	0.00	0.00	0.25	0.24	0.24	0.24	0.23	0.19	0.18	0.25
Total	$/unit	0.00	0.00	2.18	2.04	2.04	2.05	2.00	1.63	1.57	2.17

Table 21-31: Closure and post-closure costs

Activity	Units	Total $k
Dismantling and disposal of surface structures	$k	422
Ground rehabilitation and revegetation	$k	177
Disposal of hazardous substances and waste	$k	20
Waste rock dump rehabilitation	$k	1,353
Final-pit rehabilitation	$k	255
Tailings management facility rehabilitation	$k	429
Sundry rehabilitation and signage,	$k	24
Closure water monitoring and testing	$k	15
Technical oversight and corporate costs	$k	250
Personnel travelling and accommodation	$k	86
Closure Rehabilitation Total	**$k**	**3,028**
Active post-closure management (Years 8 - 12)	$k	692
Passive post-closure management (Years 13 - 17)	$k	200
Post-closure Total	**$k**	**891**
CLOSURE TOTAL	**$k**	**3,919**
Provision per tonne ore processed	$/t	0.33

22 Economic Analysis

The financial evaluation discussed in this section presents the determination of the key economic performance indicators for the Rozino Gold Project, including the Net Present Value (NPV), the payback period (time in years to redeem the initial capital investment), and the Internal Rate of Return (IRR) for the project. The discounted cash flow (DCF) model is reported at 100% attributable equity. Annual cash flow projections are estimated over the life of the mine based on the estimates of capital expenditures, production costs and sales revenues. The sales revenue is based on the production of a gold doré.

The estimates of initial and sustaining capital expenditures and site production costs were developed specifically for this project and are presented in earlier sections of this report. Total initial and sustaining capital is $87.1 M and $7.8 M respectively (see the report Section 21 - Capital and Operating Costs for details and Table 22-4).

The Rozino Gold Project has cash costs of $699 per oz payable gold. The NPV at assumed long term metal prices using a 5% discount rate is $122.5 M and the internal rate of return is 27% (Table 22-5). Payback of the initial capital occurs 3.0 years after commercial production commences (Table 22-5). The plant recovers a total of 368 koz of gold over a seven year processing life (Table 22-1 and Table 22-4). The seven-year processing life is preceded by a two-year construction period and followed by a period for closure and rehabilitation.

The economic analysis and supporting financial information, including capital and operating costs, were developed in constant dollar terms. The economic analysis uses the Probable Mineral Reserves as described in the Mineral Reserve estimate of this report. Cash flow forecasts on an annual basis using the Mineral Reserves for the base case metal price are included in Table 22-4. Sensitivity analysis charts are presented as Figure 22-1 and Figure 22-2.

22.1 DCF, Exchange Rate, Funding, Corporate Costs and NPV

A standard DCF method of financial valuation is used to value the Rozino Project. The DCF model is reported at 100% attributable equity. Key inputs to the financial valuation such as the ROM production profile, operating costs and capital costs are described in detail in the preceding sections of this report.

The DCF model utilized US$ as the base currency as the majority of capital and operating cost estimates are based in US$.

Cash flows are discounted at 5% to obtain an NPV of the Project.

The financial model does not include any consideration of funding or funding costs. There are no corporate administration overheads applied.

22.2 Working Capital

A working capital fund is applied commencing in the first year of operation to cover 3 months of operating and sustaining capital costs as work in progress. This temporary fund commences at $9.7 M in Year 1 and then decreases each year based on increasing project stability and is fully reclaimed in Year 7. No fees or interest are applied to the working capital fund.

22.3 Taxes and Royalties

Velocity Minerals requested a Bulgarian chartered accountant (Maonl Krivohapkov) to review the tax assumptions in respect of depreciation of exploration and evaluation costs incurred in respect of the Rozino Gold Project. It was determined through that review that Velocity, through Tintyava Exploration AD, have $6.65 M of project losses (exploration and studies) that are deemed applicable to the Rozino Gold Project. These losses are depreciated over the working life of the mine (seven years) and are applied to the DCF. However, the loss due to these costs is excluded.

For the remainder of the capital, a five-year straight-line depreciation method was used to amortize the capital expenditures.

Corporate tax is applied at 10% on positive EBITDA. Total Bulgarian corporate taxes amount to $19.2 M over the life of the project.

A Bulgarian government Concession Royalties is payable and calculated on gross metal sales revenue. The calculation of the Royalty, which is based on project revenues and varies between 0.8 and 4% depending on profitability (see Section 4 for details). A rate of 2%, calculated on the NSR was adopted for the PFS and is considered reasonable.

Any sales taxes expended are considered reclaimable within the year of expenditure.

22.4 Revenue

The DCF schedule assumes a two-week delay in production of concentrate from the Rozino plant at the commencement of processing. It also allows for a two-week build-up of concentrate at the Central Plant prior to processing commencing, and a two-week delay in the of sale of doré. The fixed gold price of $1,500 per oz is applied throughout the project life. Revenues from the sale of Silver are not included.

22.5 Environmental Provision

Reclamation and closure costs were estimated for the Rozino Gold Project and are described in detail in Section 21. The reclamation and closure costs are provided for as an operating expense per tonne of ore processed. The environmental provision is estimated at $3.91 M over the life of the project or $0.33/t of ore processed.

22.6 Key Financial Assumptions and Indicators

Key financial assumptions are presented in Table 22-1to Table 22-3.

Table 22-1: Key project overview and metrics

Project Overview		**Units**	**Value**
Mining	Total ore production	Mt	11.8
	Total waste production	Mt	26.5
	Total mined	Mt	38.3
	Metal mined	Gold koz	465
	Mine life	years	6.9
Processing	Steady state ROM production	Mtpa	1.750
	Years steady state production	years	5.0
	Average production rate	ktpd	4.7
	Average gold head grade	g/t	1.22
	Overall metallurgical recovery	%	79.3
	Payable Au	LOM koz	368
		average kozpa	54

Table 22-2: Summary of LOM Operating Costs (per Tonne of Ore Processed)

Operating costs	Unit	$/tonne
Mining	$/tonne	8.43
Flotation Plant	$/tonne	7.04
Concentrate Haulage	$/tonne	0.53
Concentrate Treatment	$/tonne	2.35
Administration Costs	$/tonne	1.93
Environmental provision	$/tonne	0.33
All-in opex	**$/tonne**	**20.62**

Table 22-3: Summary of total capital costs

Capital Expenditure	$M
Site preparation	13.5
Mine infrastructure	10.6
Flotation Plant and mine buildings	39.0
TMF	8.9
Central Plant upgrades	1.1
Owner's administration costs	2.9
Indirect costs	2.2
EPCM and commissioning costs	7.0
Contingency	9.6
Total Project Capital Expenditure	**94.8**
Initial Capital Expenditure	**87.1**
Sustaining Capital	**7.8**

22.7 Cash Flow Forecasts

The DCF schedule and analysis is presented in Table 22-4.

Table 22-4: Full discounted cash flow analysis

	Unit	LOM Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
MINING PRODUCTION SCHEDULE											
Waste	Mt	26.5		1.8	4.9	5.4	5.1	5.2	3.5	0.5	0.0
Low Grade (LG) ore	Mt	2.6		0.1	0.8	0.8	0.5	0.3	0.2	0.0	0.0
Ag grade	Au g/t	0.68		0.72	0.70	0.68	0.66	0.66	0.64	0.65	0.00
High Grade (HG) ore	Mt	9.2		0.19	1.5	1.7	1.7	1.4	2.1	0.4	0.0
Total mined	Au g/t	1.38		1.35	1.26	1.34	1.21	1.23	1.71	1.52	0.00
Total Ore	Mt	11.8		0.3	2.3	2.6	2.2	1.7	2.3	0.5	0.0
Total	Au g/t	1.22		1.12	1.08	1.12	1.10	1.14	1.63	1.46	0.00
Total Pit Ore and Waste	Mt	38.3		2.1	7.2	8.0	7.3	6.9	5.7	1.0	0.0
High-Grade Ore Rehandle	Mt	0.6		0.0	0.0	0.0	0.3	0.0	0.0	0.3	0.0
Low-Grade Ore Rehandle	Mt	2.5		0.0	0.0	0.0	0.0	0.0	0.0	0.9	1.6
Strip Ratio	w:o	2.2		6.2	2.1	2.1	2.4	3.0	1.5	1.1	0.0
Mining Rate (no rehandle)	ktpd	18.7		5.9	20.1	22.3	20.4	19.3	16.0	2.8	0.0
FLOTATION PLANT PRODUCTION											
Total Ore	Mt	11.8			1.49	1.75	1.75	1.75	1.75	1.75	1.59
	Au g/t	1.22			1.26	1.32	1.22	1.21	1.67	1.12	0.71
	Au koz	465			60	74	69	68	94	63	36
Milling Rate	ktpd	4.7			4.1	4.9	4.9	4.9	4.9	4.9	4.4
Tonnage proportion Oxide	**%**	**16%**			**32%**	**16%**	**5%**	**13%**	**0%**	**0%**	**50%**
Tonnage proportion Transitional	**%**	**16%**			**19%**	**28%**	**14%**	**4%**	**0%**	**0%**	**49%**
Tonnage proportion Sulphide	**%**	**69%**			**49%**	**56%**	**82%**	**83%**	**100%**	**100%**	**2%**
Flotation Mass Pull	%	3.84%			3.51%	3.82%	4.07%	3.90%	4.18%	4.18%	3.10%
Flotation Recovery	% Au	90.4%			84.7%	86.9%	92.3%	92.0%	96.7%	96.4%	75.0%
FLOTATION PLANT TAILINGS											
Tailings to TMF	Mt	8.6			1.4	1.7	1.7	1.7	1.7	0.4	0.0
Tailings to Pit	Mt	2.8			0.0	0.0	0.0	0.0	0.0	1.3	1.5
Total Tailings	Mt	11.4			1.4	1.7	1.7	1.7	1.7	1.7	1.5
Tailings Grade	Au g/t	0.12			0.20	0.18	0.10	0.10	0.06	0.04	0.18
CENTRAL PLANT PRODUCTION											
Concentrate Processed	kt	454.2			47.47	66.31	71.05	68.44	72.00	72.00	56.94

	Unit	LOM Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
	Au g/t	28.8			30.4	30.1	27.8	28.5	37.9	26.9	18.3
CIL Recovery	% Au	87.6%			90.1%	89.3%	87.1%	87.3%	87.9%	85.3%	85.8%
Overall Metallurgical Recovery	%	79.3%			66.7%	76.2%	80.8%	80.2%	81.2%	85.9%	85.4%
Dore Sold	koz	614			64.5	93.5	92.4	91.2	125.7	91.6	54.9
Gold in Dore Sold	Au koz	368			40.2	56.8	55.4	54.7	76.3	54.1	30.8
NET SMELTER REVENUE											
Payable Gold	$M	551.4			60.2	85.0	82.9	81.9	114.2	81.0	46.2
Gold Refining Charge	$M	0.5			0.1	0.1	0.1	0.1	0.1	0.1	0.0
Dore Freight and Insurance	$M	2.7			0.3	0.4	0.4	0.4	0.6	0.4	0.2
NSR Royalty	$M	11.0			1.2	1.7	1.6	1.6	2.3	1.6	0.9
Total NSR	**$M**	**537.1**			**58.7**	**82.8**	**80.8**	**79.8**	**111.2**	**78.9**	**45.0**
Total NSR	**$/t**	**45.43**			**39.46**	**47.30**	**46.17**	**45.60**	**63.56**	**45.07**	**28.35**
OPEX											
Mining	$M	99.7	0.0	0.0	16.8	19.6	20.1	19.0	17.2	6.0	1.1
Flotation Plant	$M	83.3	0.0	0.0	10.7	12.5	12.1	12.1	11.9	11.9	12.0
Concentrate haulage	$M	6.3	0.0	0.0	0.7	0.9	1.0	1.0	1.0	1.0	0.7
Concentrate treatment	$M	27.8	0.0	0.0	3.0	4.1	4.4	4.3	4.5	4.5	3.1
Administration	$M	22.8	0.0	0.0	3.2	3.6	3.6	3.6	3.5	2.9	2.5
Environmental provision	$M	3.9	0.0	0.0	0.5	0.6	0.6	0.6	0.6	0.6	0.5
Total	**$M**	**243.8**	**0.0**	**0.0**	**34.9**	**41.3**	**41.8**	**40.4**	**38.6**	**26.8**	**19.9**
Mining	$/t	8.43	-	-	11.30	11.20	11.46	10.83	9.81	3.42	0.69
Flotation Plant	$/t	7.04	-	-	7.20	7.14	6.94	6.93	6.79	6.79	7.59
Concentrate haulage	$/t	0.53	-	-	0.49	0.53	0.57	0.54	0.58	0.58	0.43
Concentrate treatment	$/t	2.35	-	-	1.98	2.36	2.52	2.43	2.56	2.56	1.94
Administration	$/t	1.93	-	-	2.18	2.04	2.04	2.05	2.00	1.63	1.57
Environmental provision	$/t	0.33	-	-	0.33	0.33	0.33	0.33	0.33	0.33	0.33
Total	**$/t milled**	**20.62**	**0.00**	**0.00**	**23.49**	**23.62**	**23.86**	**23.11**	**22.07**	**15.31**	**12.56**
Capital Expenditure											
Equipment mobilization	$M	0.4	0.2	0.1	0.1	0.0					
Clearing, grubbing and topsoil removal	$M	3.3	1.5	1.5	0.2	0.1					

	Unit	LOM Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Roads	$M	2.7	2.5	0.2	-	-					
Mine pre-strip	$M	8.9	1.4	7.5	-	-	-	-	-	-	-
Owners administration	$M	2.9	1.2	1.7							
TMF overhaul capital redirection	$M	0.9	0.0	0.2	0.4	0.0	0.2	0.1	0.0	0.0	0.0
RWD overhaul	$M	0.1	0.1								
Commissioning team	$M	0.5		0.2	0.4						
Powerline 23 km	$M	6.0	2.4	3.6							
Flotation Plant plus mine buildings on site	$M	39.0	25.3	13.6							
Tailings management facility	$M	8.9		3.7	2.5		1.4	1.1		0.2	
Water dams	$M	1.5	1.5								
Water pipeline	$M	0.4	0.4								
Central Plant upgrades	$M	1.1		1.1							
Indirects	$M	2.2	1.2	0.8	0.1	0.0	0.0	0.0	0.0	0.0	0.0
EPCM	$M	6.5	3.6	2.5	0.2	0.0	0.1	0.1	0.0	0.0	0.0
Subtotal	**$M**	**85.3**	**41.3**	**36.8**	**3.7**	**0.1**	**1.8**	**1.4**	**0.0**	**0.2**	**0.0**
Contingency	$M	9.6	5.3	3.6	0.3	0.0	0.2	0.1	0.0	0.0	0.0
Total	$M	94.8	46.7	40.4	4.0	0.1	2.0	1.5	0.0	0.2	0.0
FINANCIAL SUMMARY											
Depreciation claimed	$M	101.5			19.2	19.2	19.6	19.9	19.9	1.7	2.1
Operating income after depreciation	$M	191.8			4.6	22.3	19.5	19.5	52.7	50.4	23.0
Taxes	$M	19.2			0.5	2.2	1.9	1.9	5.3	5.0	2.3
Net working capital	$M	0.0			9.7	-0.2	-0.4	-1.2	-1.4	-2.5	-3.9
Net pre-tax cashflow	$M	$198.5	-46.7	-40.4	10.0	41.6	37.5	39.1	74.0	54.4	29.0
Cumulative net pre-Tax cash flow	$M		-46.7	-87.1	-77.1	-35.5	2.0	41.1	115.1	169.5	198.5
Net after-tax cash flow	**$M**	**179.3**	**-46.7**	**-40.4**	**9.5**	**39.4**	**35.5**	**37.1**	**68.8**	**49.4**	**26.7**
Cumulative net after-tax cash flow	**$M**		**-46.7**	**-87.1**	**-77.5**	**-38.2**	**-2.6**	**34.5**	**103.3**	**152.6**	**179.3**

22.8 NPV, IRR and Payback

Key financial outcomes are presented in Table 22-5. Earnings before income tax are $293.3 M capital expenditure totals 94.8 M and thus the return on capital employed (ROCE) is 3.1.

Table 22-5: Summary of Economic Results

Analysis Case	Summary of Economic Results	Units	Value
Pre-Tax	NPV @ 0%	$M	198.5
	NPV @ 5%	$M	137.0
	IRR	%	34.7%
	Payback (Project Start)	years	4.9
	Payback (Production Start)	years	2.9
After-Tax	NPV @ 0%	$M	179.3
	NPV @ 5%	$M	122.5
	IRR	%	27.4%
	Payback (Project Start)	years	5.0
	Payback (Production Start)	years	3.0

22.9 Sensitivity Analysis

A number of single parameter sensitivity impacts were estimated. Neither the gold cut-off grade, mine plan nor the processing plan were altered. The Rozino Gold Project's NPV is most sensitive to changes in metal price and factors such as head grade and metallurgical recovery. IRR is most sensitive to changes in capital expenditure. The project is not sensitive to the value of capital contingency applied (within a reasonable range of expectation). A number of standard financial sensitivities are listed in Table 22-6and represented as graphs in Figure 22-1 and Figure 22-2.

Table 22-6: Key Sensitivity Analyses for the Rozino Gold Project

Item	Sensitivities	After-tax IRR (%)	After tax NPV $ M
CAPEX	-25%	46.5	158.0
	Base Case	27.4	122.5
	+25%	15.3	77.0
CAPEX contingency	No contingency applied 0%	28.2	124.7
	Base Case (13%)	27.4	122.5
	Contingency doubled 25%	26.4	119.5
OPEX	-25%	37.9	185.6
	Base Case	27.4	122.5
	+25%	13.8	46.9
Gold price	$1,125	10.2	35.4
	Base Case $1,500	27.4	122.5
	$1,875	41.4	291.1

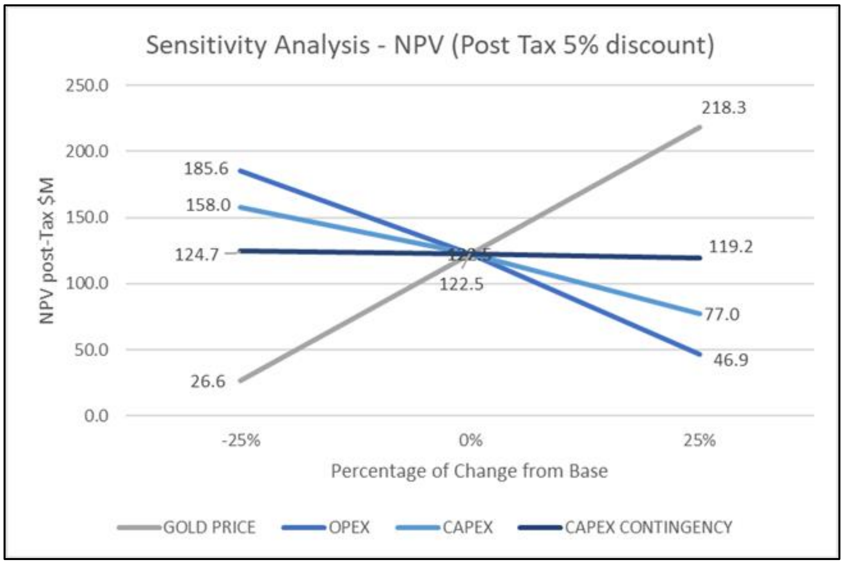


Figure 22-1: *Rozino sensitivity analysis NPV*

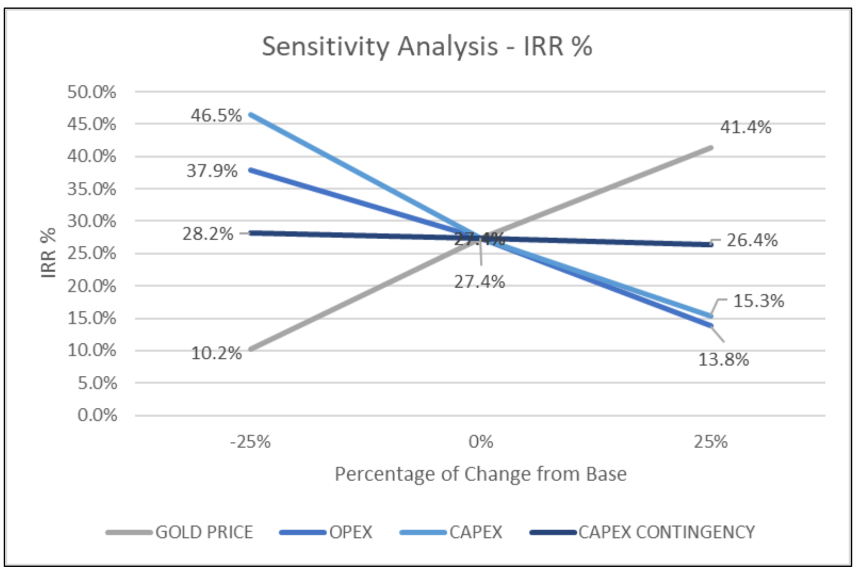
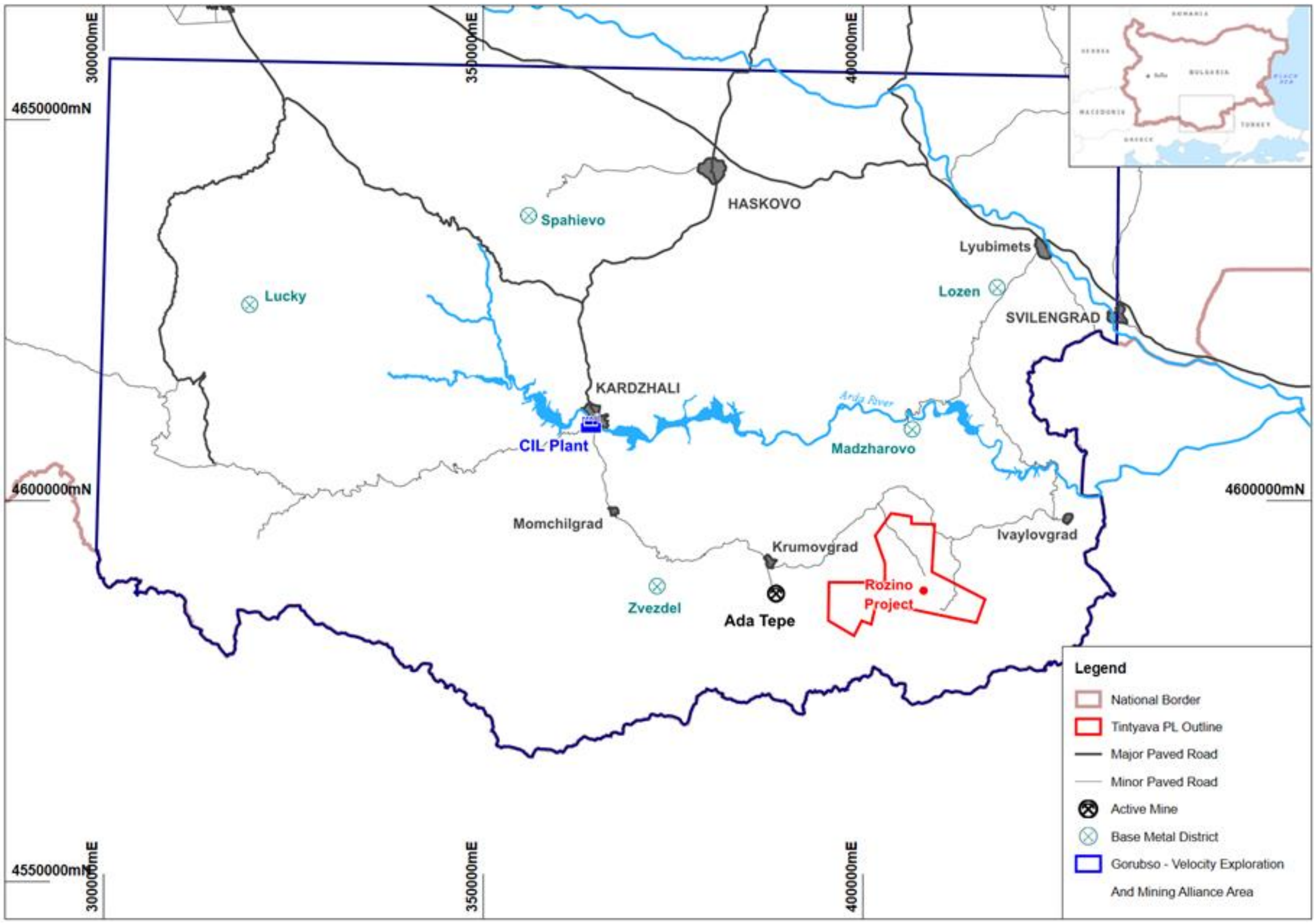


Figure 22-2: *Rozino sensitivity analysis IRR%*

23 Adjacent Properties

The most significant nearby property to Rozino is the Dundee Precious Metals (DPM) Krumovgrad project which is located 20 kilometres to the west



. Mineralization of interest on this property occurs in the Ada Tepe low sulphidation epithermal (LSE) Au-Ag deposit.

At Ada Tepe the highest gold grades occur within veins and breccias that display typical LSE textures and mineral species. Gold occurs as electrum in colloform-banded and lattice-bladed silica-carbonate-adularia veins and hydrothermal breccias. The mineralization is hosted in Paleogene sedimentary rocks just above an unconformable contact with underlying metamorphic basement rocks. The host rocks are variably brecciated and range in grain size from conglomerate to mudstone.

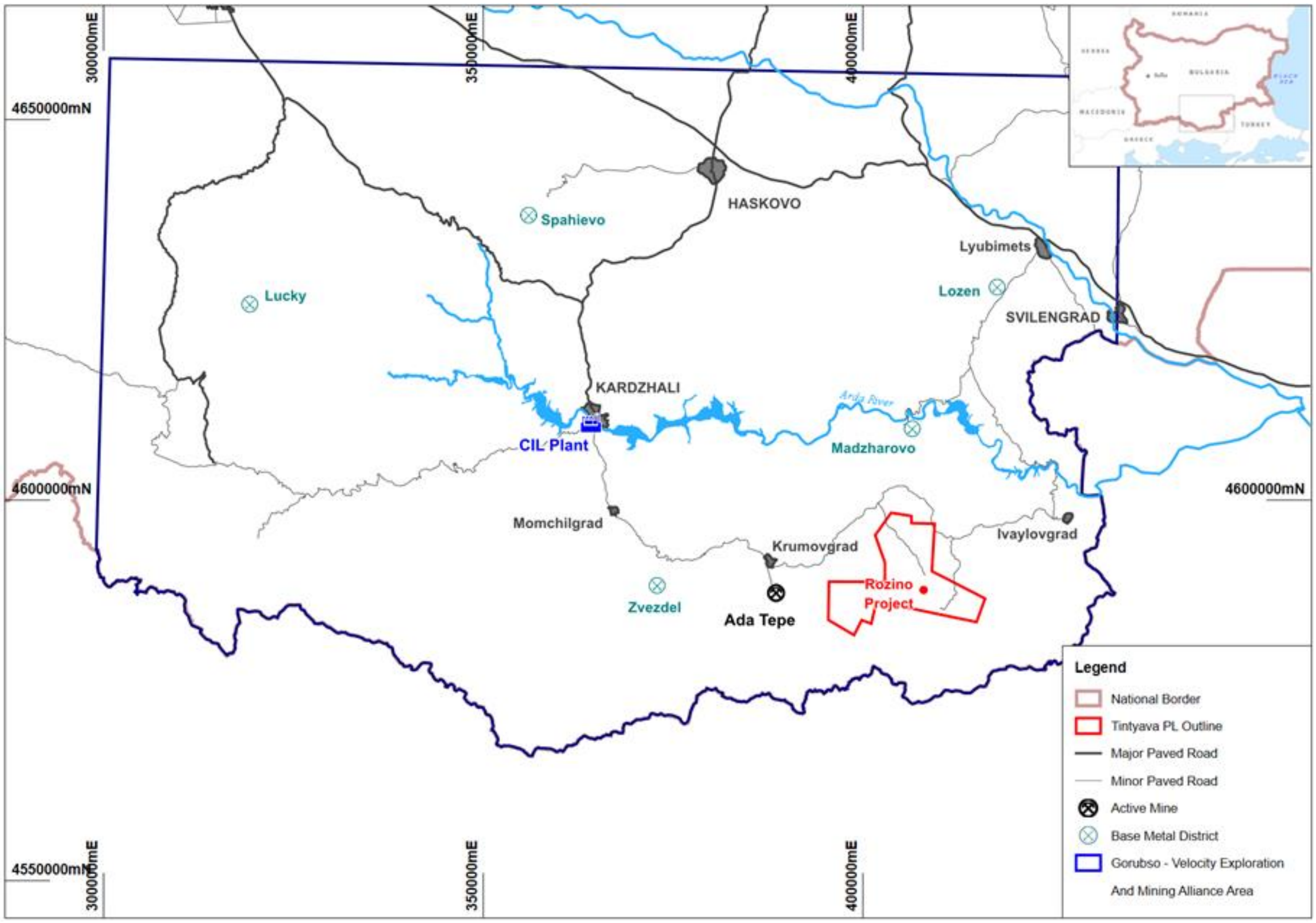


Figure 23-1: Rozino Project and location of DPM's Ada Tepe deposit
Source: Velocity, 2020

DPM holds a 30-year mining concession on the Krumovgrad property. They mine the Ada Tepe deposit using industry standard open cut mining methods and processes ore by conventional crushing, grinding, and flotation. The ore processing plant at Ada Tepe is designed to treat ore at a maximum of 840 ktpa, over an eight-year mine life (Dundee Precious Metals, 2014).

The scientific and technical information disclosed above for the adjacent Ada Tepe deposit is referenced from a NI 43-101 Technical Report on the Ade Tepe Deposit with an effective date of March 21, 2014 and filed on SEDAR under Dundee Precious Metal's profile on November 7, 2017 (Dundee Precious Metals, 2017).

The adjacent property discussed above contains broadly similar geology and mineralization to the Rozino Project. However, the report authors have not been able to independently verify the technical information for this adjacent property, and the information related to the adjacent property is not necessarily indicative of the mineralization on the Rozino Project. The source and date of the technical report related to the adjacent property has been disclosed above and in Section 27.

24 Other Relevant Data and Information

There are no other relevant data or information in this section.

25 Interpretation and Conclusions

25.1 Exploration and Mineral Resources

This report reflects sampling information available for the Property on the 28th of September 2020, including exploration sampling and drilling completed by Hereward and Asia Gold during the mid-2000's and Velocity since 2017. Few details of sampling and assaying are available for drilling by Geoengineering in the 1980's. These holes are not included in datasets used for resource estimation, or exploration and are not detailed in this report.

Exploration sampling of relevance to current exploration comprises soil, stream sediment and rock chip sampling undertaken by Velocity. Results of this sampling support Velocity's interpretation of the Property's geology and are, in the author's opinion, sufficiently suggestive of the potential for deeper mineralization to warrant further investigation including targeted exploration drilling.

The author considers that quality control measures adopted for sampling and assaying of the exploration sampling and drilling of relevance to Mineral Resources and exploration have established that the sampling, and assaying is representative and free of any biases or other factors that may materially impact the reliability of the sampling and analytical results. The author considers that the sample preparation, security and analytical procedures adopted for the Tintyava exploration sampling and drilling provide an adequate basis for the current Mineral Resource estimates and exploration activities.

The author considers that the sample preparation, security and analytical procedures adopted for the Tintyava exploration sampling and drilling provide an adequate basis for the current Mineral Resource estimates and exploration activities.

Geology of the Tintyava area comprises a series of discrete Palaeogene syn-tectonic pull-apart sedimentary basins within metamorphic basement that collectively form the Ivaylovgrad Corridor.

Rozino is a low sulphidation epithermal (LSE) gold deposit hosted dominantly within Palaeogene breccia conglomerate sedimentary rocks as disseminations, replacement and vein mineralization. Alteration is characterized by a quartz, carbonate, chlorite, adularia, pyrite assemblages. The mineralogy consists mainly of pyrite with traces of base metals and rare arsenopyrite. Gold occurs at sulphide mineral boundaries and less commonly as free grains or encapsulated inclusions. Local mineralization development is controlled by the intersection of steep structures sub-parallel to the bounding extensional faults and gently dipping bedding. Drilling has intersected mineralization at Rozino over an area around 800 metres by 1,000 metres to a vertical depth of around 195 metres

Mineral Resource estimates for the Rozino deposit are based on information available for Hereward Asia Gold and Velocity drilling at the Tintyava Property on the 23rd of October 2019. Velocity's diamond drilling provides 82% of the mineralized domain composites informing the estimates with Asia Gold and Hereward drilling contributing 3% and 16% respectively.

Mineral Resources were estimated by Multiple Kriging. Estimated resources include a variance adjustment to give estimates of recoverable resources above gold cut-off grades for selective mining (SMU) dimensions of 4 metres east by 6 metres north by 2.5 metres in elevation. Estimates for mineralization tested by up to approximately 50 metre spaced drilling are classified as Indicated, with estimates for broader and irregularly sampled mineralization assigned to the Inferred category.

The estimates are reported within a 150 metre deep optimal pit shell generated at a gold price of $1,500/oz. The optimization parameters generate a cut-off grade of 0.3 g/t Au, which is selected as the base case for Mineral Resource reporting. Table 14-6 presents Mineral Resources estimated for Rozino at this cut-off grade. The figures in this table are rounded to reflect the precision of the estimates and include rounding errors. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The Indicated Mineral Resources are inclusive of Mineral Reserves.

The Mineral Resource estimates have been classified and reported in accordance with NI 43-101 and the classifications adopted by CIM Council in May 2014 (CIM, 2014).

Velocity consider that untested parts of the Palaeogene basins within the Ivaylovgrad Corridor have potential to host LSE mineralization analogous to that observed at Rozino. First phase surface sampling within the Property has identified several exploration targets. Renaissance drilling has intersected mineralization with at the Rozino South and Kazak exploration areas. Systematic soil surveys identified soil anomalies including two significant gold in soil anomalies at Tumbata which warrant exploration drilling.

Table 25-1: Rozino Indicated and Inferred Mineral Resource Estimates at 0.3 g/t cut off

Effective date of estimates: April 15th 2020			
	Tonnes	Grade	Metal
	(Mt)	(Au g/t)	(Au koz)
Indicated Mineral Resource Estimate	20.5	0.87	573
Inferred Mineral Resource Estimate	0.38	0.8	10

25.2 Development and Mineral Reserve

This Technical Report includes the first Mineral Reserve estimate for the Rozino Gold Project. The Mineral Reserve estimate generated from this new information are presented in Table 15-5. The conversion of Mineral Resources to Mineral Reserves was made using industry recognized methods of determining operational costs, capital costs, mining rate and plant performance. Thus, it is considered to represent actual operational conditions of the proposed mining project. This report has been prepared with the latest information regarding environmental and closure requirements and has set out the type and extent of work required.

It is important to note that permitting for the Rozino Gold Project is not complete. Velocity has initiated the environmental and social impact assessment process. Regulatory permitting procedures to meet Bulgarian regulations and gathering environmental data to improve the design of the Project have also been initiated. Under the Bulgarian Environment Protection Act, the development of an economically viable mining reserve requires an EIA which complies with European Union environmental regulations. The prospecting licence agreement for the Tintyava Property has been signed with the Minister of Energy and exploration activities have been approved by the Ministry of Environment. All necessary permits to conduct the work proposed for the property have been obtained and there are no known significant factors or risks that may affect access, title or the right or ability to perform work on the Property. There are currently no objections to the development of the Project.

The recommended development plan for the Rozino Gold Project includes a low strip ratio (2.2:1) open pit with a simple flotation plant that produces a concentrate that is transported 85 km to the existing Central Plant at Khardjali for the production of gold doré. The mine will operate for seven years, including an initial two years of construction and pre-stripping. The mine will utilize equipment that is commonly available and chosen for its ability to selectively extract ore commensurate with the dimensions of the modelled SMU. The Project development includes training, ore control systems, costs, manning and management that are considered sufficient to provide the necessary control to ore delivery at the specified grade and rate to plant. The mine will be largely owner-operator and use leased equipment. The exception is for drill and blast where a contractor will be responsible for the supply and use of explosives. The mine will have peak ore plus waste mining rate of 22,000 tpd, delivering 5,000 tpd of ore to the Flotation Plant feed for the first five years of life. Low-grade ore will be stockpiled and processed during the subsequent and last two years of the operation after mining is completed. The relatively small pit area, waste rock backfilling requirements and low-grade ore stockpiling within a compact footprint will require diligent planning to ensure that the proposed ore feed rate is achieved. Most mine pre-stripping activities are aimed at the production of waste rock for the water storage dams, the TMF, the ROM stockpile pad, and the surface facilities pad. The waste rock mining plan relies on pit back filling to minimize the environmental impact and reduce operating costs in order to deliver the best economic and environmental outcome.

The Rozino flotation process is considered simple and conventional, with three-stage crushing, ball milling, and concentration by rougher-scavenger-cleaner flotation. The construction of the Flotation Plant and site infrastructure, including access road upgrades and a 23 km powerline will take two years. The Flotation Plant tailings are considered largely inert and are to be stored in a purpose-built TMF that has a downstream lined-wall construction until the pit is complete. Once mining is complete the low-grade ore will be processed and the arising tailings stored in the exhausted pit.

The selected process path is backed by extensive metallurgical testwork that identifies three ore types namely Oxide, Transitional and Sulphide, each with varying performance characteristics. The ore types will not be differentiated and will be processed simultaneously as a blended feed. Predicted gold recovery to doré based on flotation and leach characteristics of the three ore types averages 79.3%. Adequate test-work data is available to provide operating parameters for flowsheet design and major equipment sizing within the contingency allowances normally associated with a PFS.

The Rozino Gold Project has a comprehensive water management plan that integrates precipitation and surface flows, groundwater inflows, and water re-use with operational requirements, but particularly plant requirements. The site water balance indicates that the project has a net water deficit and that water will need to be imported from external sources. The make-up water demand is estimated to range from 125 to 310 km^3 per annum (depending on climatic cycles). The project includes a system of contact and non-contact water separation and management to minimize the environmental impact of downstream receiving water bodies and maximize re-use.

The concentrate produced at the Flotation Plant will be trucked to the Central Plant, about 85 km by road, by approximately 12 trucks per day. The trucking operation is not considered to be a significant risk to public safety and health. The existing Central Plant at Khardjali will require some modifications to enable processing of the concentrate. At the Central Plant the concentrate will be ground to a P_{80} 20µm and then processed through a standard CIL and elution circuit to produce gold doré. The project will produce 368,000 oz of gold in doré over the seven-year mine life, averaging about 55,000 oz per annum.

The initial capital totals $84.8 M over a two-year construction period; sustaining capital is a modest $8.4 M, which is mostly for TMF construction. At a gold price of $1,500 per oz the project generates a cumulative net income after tax of $179 M and an after-tax rate of return of 27%. The project has a three-year payback period after production commences.

The project includes an adequate provision for the rehabilitation and closure of the site in an environmentally sound and sustainable manner when mining operations cease. The project will provide temporary employment for between 300 to 500 construction workers and will generate up to 260 permanent jobs.

25.3 Risks and Opportunities

There are no known likely mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the valuation of the Project. However, the Project must not be considered to be without risk.

Risks to the mine plan not achieving the specified technical and financial parameters set out in the PFS include, but are not limited to:

- Regulatory delays in obtaining the necessary permits.

- Regulatory approval requirements may alter the plan as developed. For instance, by not being able to deposit low-grade ore tailings in the completed pit would alter the plan outcomes.
- The geological model and Mineral Resource estimate were completed using MIK estimation using appropriate methodologies, information, data verification and were applied by world class experts in this field in support of this PFS. Consequently, it is considered that the estimation risk is appropriately minimized. There remains estimation risk associated to the relationship between the size of the SMU and the current data spacing that can be refined no further with the current information. The risk to estimation could possibly be reduced with more intensive data and analysis of that data. Collection of such data is normally appropriate at later stages of study development.
- Inadequate grade control practices and poor mine planning could lead to lower than predicted mining and plant throughput rates, and lower head grades.
- Operating performance of the flotation and CIL processes in actual production may vary to those predicted by metallurgical testwork resulting in lower than predicted gold recovery.
- Prolonged drought conditions leading to a constraint on the sourcing of external water and may lead to a consequential reduction in plant throughput rate.
- Constraints on the road delivery of concentrate to the Central Plant may occur due to road safety incidents, negative community impacts, road conditions or contractor under-performance.
- The cost estimates completed are considered appropriate to support the PFS. However, there are some areas of estimation that whilst generally appropriate, have more risk attached to the level of information support. Two areas that have been noted in the text for further enhancement of estimation accuracy are the drill and blast operation and powerline construction.
- Gold maintaining a price below $1,500/oz for the duration of the project.

Opportunities for improving the performance of the Project include, but are not limited to:

- Integration of water storage into the TMF, thus reducing the number of separate water storage facilities required.
- Gold maintaining a price above $1,500/oz for the duration of the project.
- The project capital cost estimation is based on new equipment. The short project life may allow the procurement of suitably refurbished used equipment, and thus reduce capital costs. A similar strategy may be applied to the mining fleet.
- Silver has not been consistently assayed in the drillholes but is present in the concentrate and leached product. Assaying for silver may enable inclusion in Mineral Resources and Mineral Reserves, and DCF revenue streams.
- Diesel costs were estimated using Bulgarian public rates; it may be possible to negotiate bulk fuel supply pricing and reduce the mine operating cost:
    - Using the owner’s mining equipment for topsoil stacking and hauling may reduce capital cost;
    - Although the drilling completed since October 2019 is not material to the estimate, there are some drill results that may result in an increase in the resource and reserve base, and thus a positive impact to the next stage of project evaluation; and
    - Additional testwork around low-grade mineralization and oxidized ore may allow for improved gold recoveries.

26 Recommendations

26.1 Mineral Resources and Exploration

The author's recommendations for future exploration and resource definition programs are consistent with Velocity's work plan for 2020 and 2021, which targets expansion of the Indicated Mineral Resources at Rozino and definition of additional Inferred Mineral Resources in exploration areas, including mineralization intersected by exploration drilling. The author concurs with the general approach of Velocity's proposed exploration and resource definition programs, and recommends the following work programs with estimated costs summarized in

Table 26-1.

Velocity's work comprises two broad phases:

- Phase One: Exploration drilling in the Rozino area and regional exploration within the sedimentary basins of the Ivaylovgrad Corridor, updating of the Rozino Mineral Resource estimates, and if supported by drilling results estimation of Inferred Mineral resources for the current exploration target areas. The exploration targets are at an early stage of evaluation and it is not certain that the proposed drilling will intersect mineralization, or lead to estimation of additional Inferred Mineral resources.
- Phase Two: Additional drilling of exploration targets and infill drilling aimed at defining Indicated Mineral Resources within the volume of potential Inferred Resources estimated on the basis of Phase One work. The resource infill drilling component of this Phase is contingent on positive Phase One results.

The planned Phase One exploration drilling will prioritize higher grade near surface extensions to existing discoveries at Rozino South and Kazak. Additional drilling is proposed for the area between Kazak and Rozino South where potentially mineralized sedimentary rocks are overlain by younger sedimentary rocks and alluvial sediments.

Velocity consider that the identification of mineralization within the basement at Kazak represents a significant exploration target, and surface exploration including soil sampling and trenching is proposed for this area during Phase One.

Exploration drilling is planned for target areas at Tumbata as identified by soil sampling (Figure 9-4).

The author's specific recommendations relating to Rozino Mineral Resources, the cost of which are included in

Table 26-1, include the following:

- The resource estimates should be updated to reflect drilling and assay information obtained since October 2019
- The reliability of assay results from post October 2019 drilling including inter-laboratory should be further investigated, including inter-laboratory check assaying.

Table 26-1: Proposed Exploration Activities and Budget

Physicals			
Description	**Phase 1**	**Phase 2**	**Total**
Diamond drilling	3,000 m	1,500 m	4,500 m
Soil samples	2,500	500	3,000
Trenching	1,000 m	250 m	1,250 m
Cost ($k)			
Description	**Phase 1**	**Phase 2**	**Total**
Capital	20	20	40
Personnel	250	100	350
Drilling	300	150	450
Geochemistry	120	40	160
Geophysics	20	20	40
Geology	30	30	60
Vehicles/generators	70	50	120
Field and office costs	130	100	230
Contingency	90	60	150
Total	**1,030**	**570**	**1,600**

26.2 Regulatory Processes and Land Acquisition

It is recommended that the EIA process be completed and relevant documentation submitted to the Ministry of Energy for environmental review.

During the EIA process and leading up to the approval of the EIA and the granting of a mining concession it is recommended that private lands be acquired to facilitate Project construction and operation.

26.3 Mining

It is recommended that further mining studies be conducted during the Feasibility Study. In particular:

- Complete an initial and focussed grade control program covering the initial operational period to confirm the SMU gold distribution and estimation characteristics. This will also entail assays, mathematical analysis and modelling.
- Ensure that all drill holes are logged for the ISRM weathering code. This data, combined with metallurgical testing, will allow the development of more accurate and robust geometallurgical recovery models.
- Although the necessity for an automated truck dispatch system is not anticipated and its operational cost is not included in the PFS, evaluation of the potential cost benefits of such a system should be undertaken in the Feasibility Study.
- Further scheduling included in the Feasibility Study to examine the possibility of reducing the variation in truck numbers to +/- one truck per quarter.
- Power supply for the operation is from hydro-generation power plants, which already have reasonable sustainability. Within the context of the global shift away from a carbon-based economy, Velocity should consider the implications of potential legislative changes on energy for material transport and develop positive mitigation strategies. This is recommended to be completed within the Feasibility Study.

26.4 Pit Geomechanical and Surface Geotechnical

Further geotechnical investigative and confirmatory work is recommended to support the Feasibility Study:

- Currently the geomechanical data has been collected from exploration drill holes completed predominately with similar azimuths; this can lead to biases. Additional geomechanical holes should be drilled with an orientation to intersect the likely pit walls and large-scale faults perpendicularly.
- Additional geomechanical holes drilled to target areas of potential wall weakness, particularly in the western and southern areas of the pit. This may improve the understanding of the rock mass characteristics as well as spatial variability across pit domains.
- Additional surface geotechnical drill holes in key TMF, water storage dam and surface facility locations will confirm geotechnical design parameters.
- Structural data analysis showed variation throughout the pit. Although there was no indication of errors in the structural logging, televiewer logging can provide robust and cost effective data along the entire length of boreholes and thus improve geomechanical logging.
- UCS and tensile tests have been conducted for the Pre-feasibility Study. For the Feasibility Study, triaxial and joint shear tests are required to provide a more nuanced understanding of the rock characteristics.
- The Feasibility Study level geotechnical analysis should include kinematic analysis of the large- and small-scale geological structures. In addition to limit equilibrium overall slope analysis, finite element analysis should be undertaken.

26.5 Hydrogeology

The following recommendations are made:

- The hydrogeological properties of the areas in the vicinity of the TMF, CWD and RWD should be characterized in further detail through a targeted site investigation to reduce uncertainty in hydrogeological parameters of the materials underlaying these facilities.
- It is recommended to collect baseline surface water and groundwater quality data in the area of influence of the Project area to assess naturally occurring levels of contaminant concentrations of potential concern (COPC).
- Extended geochemical test work on individual samples is recommended to assess if material with higher acid generation or metal leaching potential is present within the deposit and whether it could impact groundwater or surface water qualities after excavation.
- A numerical hydrogeological model should be developed to support the Feasibility Study. The development of this model should include updated ground characterization information focused on the pit and mine facilities.

26.6 Hydrology

The following recommendations should be considered:

- A more comprehensive design of ancillary water management infrastructure, including flow control structures, energy dissipaters, culverts, riprap lining, etc.
- A more detailed evaluation of closure water management (including water quality considerations).
- Assessment of the effect of snowpack accumulation and snowmelt on water balance results.
- Consideration of alternative or supplementary water supplies (well array, increased abstraction from Arpa dere) on the RWD volume to avoid plant water supply shortfalls.

- Alternative siting of the CWD storage within the plant area given reduced estimates of the CWD storage requirements.
- Review overall water management design to ensure alignment with environmental permitting considerations. In particular, an assessment should be made of any changes to water management that may be necessary to maintain environmental flows downstream of the project in the Uren dere and Arpa dere.

26.7 Metallurgical and Process Optimization

Additional testwork and process characterization is required before the detailed engineering design of the project commences. In particular, the following aspects should be considered:

- Assaying for silver from the existing core reject samples in order for it to be incorporated into the Indicated Resource estimate. This will allow silver revenue to be used for valuation purposes in the Feasibility Study.
- Additional optimization of the reagent suite used for the flotation of Oxide and Transitional ore types and refinement of the reagent regimen for blends of ore types. These tests should use standard composites of the three ore types.
- Future testing on representative samples of the Oxide and Transitional ores to establish whether there is any significant variability. Testing needs to further establish the relationship between Au, sulphide sulphur and oxidation state. Thus, it is recommended to carry out five to ten variability tests for each major oxidation state (ISRM codes 0,1,2,3,4) given that sulphur content is not available for the deposit. The variability tests should cover a range of gold grades as well as ore type.
- Develop a geometallurgical model relating either sulphur or ISRM oxidation state to plant performance within the 3D model on a block-by-block basis. This would increase local estimation accuracy of gold recovery (compared to the current stepped model ore type) and thereby develop more confidence in short-term planning.

26.8 Infrastructure

- Complete a detailed study (adequate to support the Feasibility Study) of the powerline construction including permitting, regulatory requirements and timing, surface rights, cost estimation (capex and electricity supply cost) and general engineering considerations.
- Continue to develop an understanding of external water sourcing options to meet the make-up demand of the Flotation Plant.
- Some areas of the topography in heavily wooded areas have few ground-truth points. It is recommended to undertake ground-based surveys in key areas pit, infrastructure, surface facilities) to ensure that DTM accuracy is within +/- 300mm.
- Whilst the environmental process is on-going, continue to promote in-pit tailings disposal as the lowest environmental impact strategy as well as continuing with a single TMF storage strategy. Define the cost and location of an expanded TMF to include all Mineral Reserves as well as permitting processes for use of the pit as a tailings storage area.

26.9 Environmental

The final Project design is yet to be determined, but by initiating the EIA process early, outcomes of the EIA can be used to improve the Project design and maximize the benefits of the EIA without incurring excessive costs. There are a few potential improvements that the Project should evaluate:

- Improve the water quality monitoring network to fully understand surface water patterns of the area
- Continue to monitor wildlife, particularly Red Dear species, to determine population numbers and the habitat use of potentially impacted and sensitive species

- Schedule construction to commence outside of the normal bird breeding season
- Commission a qualified ornithologist to survey the Project area prior commencement construction
- Maintain an effective public relations policy and programme that is responsive and sensitive to issues and concerns of the surrounding communities.

26.10 Summary

The recommendations set out in the Technical Report cover the period up to the completion of the Feasibility Study and receipt of the EIA certificate. The cost of these studies and programs is estimated to be approximately $3.75 M. The exploration program set out in Section 26.1, at an estimated cost of $1.60 M, is in addition to this. Following EIA certification and a positive construction decision, the project would advance to detailed engineering and full project implementation.

Two phases are recommended, the first is preparatory works and studies to support the Feasibility Study. The second phase includes those activities and studies that can be included within the Feasibility Study.

Table 26-2: Estimated Cost of Studies Leading to the Completion of the Feasibility Study

Recommended Activity / Study	Estimated Cost ($k)
Phase 1 Exploration	
Drilling and geology	1,030
Sub-Total	**1,030**
Phase 1 Feasibility	
Regulatory Processes and Land Acquisition	350
Mining: Verification of Grade Control	150
Mining: Other Studies	75
Pit Geomechanical and Surface Geotechnical	250
Hydrogeology	50
Hydrology and water sourcing	150
Metallurgical and Process Optimization	375
Infrastructure	200
Environmental	75
Sub-Total	**1,675**
Phase 2 Exploration	
Drilling and geology	570
Sub-Total	**570**
Phase 2	
Feasibility Study	2,000
Sub-Total	**2,000**
Grand Total	**5,275**

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Appendix A Abbreviations and Units of Measurement

%	percent
°	degree(s)
°C	degree(s) Celsius
3D	three-dimensional
a	annum
A242	Aerofloat 242
AAS	Atomic Adsorption Spectroscopy (metal analyzer)
AIS-ACB	Automatic Information System – Archaeological Card of Bulgaria
AMSL	above mean sea level
Asia Gold	Asia Gold Corp
Au	gold
bcm	bank cubic meter
BM	Ball Mill
CAD$	Canadian dollars
Caracal	Cambridge Caracal Bulgaria ED
CIC	Carbon in Column
CIL	carbon-in-leach (process)
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre(s)
CN or CNfree	Concentration cyanide free
CNtot	Concentration cyanide free
CNwad	Concentration cyanide weak acid dissociable
Conc or C	Concentrate (stream)
Corg	Concentration Carbon organic
CSA Global	CSA Global Consultants Canada Ltd
Ctot	Concentration Carbon total
$CuSO_4$	Copper sulphate
CWD	contact water dam
DCF	discounted cash flow
DGPS	differential geographic positioning system
DPM	Dundee Precious Metals Inc.
Dmt	dry metric tonne
$/lin m	dollars per linear metre
EBITDA	Earnings before interest taxation depreciation and amortization
Eco-stim	Eco-stim EOOD
EMP	environmental management plan
EPA	(Bulgarian) Environmental Protection Act
EPCM	engineering procurement and construction management
ESIA	Environmental and Social Impact Assessment
ESIA	environmental and social impact assessment

Eurotest	Eurotest Control ED
FA	Fire Assay (gold and silver analysis)
Flot	Flotation (process)
g	gram(s)
g/l	gram(s) per litre
g/t	grams per tonne
Geoengineering	Geoengineering ED
Gorubso	Gorubso-Kardzhali AD
GRG	gravity recoverable gold
H&H	Holland & Holland Consultants
hpa	hours per annum
ha	hectares
Hereward	Hereward Ventures Ltd
HIL	High Intensity Leach (cyanide)
IBECO	incremental break even cutoff grade
IBC	intermediate bulk container
ICP	Inductive Coupled Plasma (multi element analyzer)
IFC	International Finance Corporation
ISA	in-stream analysis
IRR	internal rate of return
JV	joint venture
KBX	Potassium butyl Xanthate
kg	kilogram(s)
km	kilometre(s)
km^2	square kilometre(s)
koz	thousands of ounces
kt	thousands of tonnes
ktpa	thousands of tonnes per year
ktpd	kilotonnes per day
kWh	kilowatt hour
l	litre(s)
LCT	locked cycle test
LG	Lerch-Grossman
LOM	life of mine
LSE	low sulphidation epithermal
$/lin m	dollars per linear metre
m	metre(s)
m^2	square metre(s)
m^3	cubic metre(s)
m^3/a	cubic metre(s)/annum
MCAF	mining cost adjustment factor
MIBC	Methyl Isobutyl Carbinol
MIK	multiple indicator Kriging

Mineesia	Mineesia Ltd
mm	millimetre(s)
Mm^3	million cubic metre(s)
MOE	Ministry of Energy
MOEW	Ministry of Environment and Waters
MPR	MPR Geological Consultants Pty Ltd
Mt	million tonnes
Mtpa	million tonnes per annum
MW	mega watt
MWh	mega watt hour
NAIM-BAS	National Archaeological Institute with Museum at the Bulgarian Academy of Sciences
NI 43-101	National Instrument 43-101
NSR	net smelter return
O_2	Oxygen (gas)
OSA	overall slope angle
oz	troy ounce(s)
OVOS	Bulgarian Environmental Impact Assessment
PAX	Potassium Amyl Xanthate
PEA	preliminary economic assessment
pH	scalar measure of acidity logarithmic
PFS	Pre-feasibility Study
PL	Prospecting Licence
ppm	parts per million
QAQC	quality assurance and quality control (for sampling and assaying)
Qemscan	Quantitative Evaluation of Materials by Scanning Electron Microscopy
RWD	raw water dam
RO	reverse osmosis
ROCE	return on capital employed
ROM	run of mine
SAG	semi-autogenous grinding
SCN or CNS	Concentration thiocyanate (or rhodanine)
SEM	Scan Electron Microscopy
SG	Specific Gravity
SMU	selective mining unit
Ss	Concentration Sulphur as sulphide
Stot or S	Concentration Sulphur total
t	tonne(s) metric
tails or T	Tailings (stream)
tpa	tonnes per annum
tpd	tonnes per day
t/h	tonnes per hour
Tintyava Exploration	Tintyava Exploration ED
Tintyava Property	Tintyava Prospecting Licence Property

TMF	tailings management facility
TMF	tailings management facility
µm	micrometer
US$	US dollars
Velocity	Velocity Minerals Limited
WAI	Wardell Armstrong International Ltd
WGS	World Geodetic System
wmt	wet metric tonne
WRD	waste rock dump
w/v	number of grams of an ingredient in 100 mL of solution
w/w	Weight by weight
XRD	X-ray diffraction
XRF	X-ray fluorescence
Y or CY	Year

Appendix B Certificates of Qualified Persons

Certificate of Qualified Person – Andrew Willis Sharp, PEng (CSA Global)

CSA Global
Mining Industry Consultants

CERTIFICATE OF QUALIFIED PERSON

I, Andrew Willis Sharp, BEng (Mining). PEng, do hereby certify that:

- I am currently employed as Principal Mining Engineer with CSA Global Consultants Canada Ltd. with an office at 1111 W Hastings Street, 15th Floor, Vancouver, B.C., V6E 2J3, CANADA.
- This certificate applies to the Technical Report titled "Revised NI 43-101 Technical Report Pre-Feasibility Study for the Rozino Gold Project, Bulgaria", dated 15 December 2021 with an Effective Date of 28 September 2020, (the "Technical Report") prepared for Velocity Minerals Ltd. ("the Issuer").
- I am registered as a professional engineer in good standing with Engineers and Geoscientists BC. I am a graduate from the University of Curtin, Kalgoorlie (1987). I have been involved or associated with the mining industry since 1987, in Australia, Malaysia, Ghana, Mexico, Papua New Guinea and Canada in production roles for 28 years and 5 years in consulting. In particular to the Rozino Gold Project I have more than 5 years using Indicator Kriging techniques applied to mining and more than 15 years in open cut mining in planning and operational roles. I joined CSA Global in 2019 as Principal Mining Engineer.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I have visited the Rozino Gold Project site and the Central Plant from 30 July to 1 August 2019.
- I am responsible for Section numbers 1.1, 1.2, 1.8, 1.9, 1.11, 1.12, 1.14, 1.15, 1.16, 1.17.2, 2, 3, 4, 5, 15, 16, 18, 19, 21, 22, 23, 24, 25.2, 25.3, 26.2-26.10, and 27 of the Technical Report.
- I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
- I have had no prior involvement with the property that is the subject of the Technical Report.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September 2020

["SIGNED AND SEALED"]
{Andrew Willis Sharp}

____

Andrew Willis Sharp, Peng
Signing Date: 15 December 2021

Certificate of Qualified Person – Gary Patrick, MAusIMM, CP (Met) (Metallurg Pty Ltd)

CERTIFICATE OF QUALIFIED PERSON

I, Gary Patrick, BSc. MAusIMM, CP (Met) do hereby certify that:

- I am currently employed as Principal Consultant with Metallurg Pty Ltd. with an office at Liman Mah, 25 Sokak, Sila Apartman 15-D-10, Konyaalti, Antalya, Turkey, 07070.
- This certificate applies to the Technical Report titled "Revised NI 43-101 Technical Report Pre-Feasibility Study for the Rozino Gold Project, Bulgaria" (reference № R366.2020), dated 15 December 2021 with an Effective Date of 28 September 2020, (the "Technical Report") prepared for Velocity Minerals Ltd. ("the Issuer").
- I hold a BSc. (Chemistry / Extractive Metallurgy) and am a registered Member of the AusIMM (#108090). My experience includes 30 years in operations, metallurgical testwork supervision, flowsheet development, and study work. I have experience in exploration, evaluation and mining of Low Sulphidation Epithermal Deposits.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I visited the project that is the subject of this Technical Report, between 2 October and the Kardzhali treatment facility on the 3 October 2018 for a combined total of 2 days.
- I am responsible for Section numbers 1.6, 1.10, 13 and 17 of the Technical Report.
- I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
- I have had prior involvement with the Rozino project. I co-authored a Technical Report titled "NI 43-101 Technical Report Preliminary Economic Assessment, Rozino Gold Deposit, Bulgaria" with an effective date of the 17 September 2018.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September 2020

["SIGNED"]
{Gary Patrick}

____
Gary Patrick, MAusIMM CP(Met)

Signing Date: 15 December 2021

Certificate of Qualified Person – Carl Steven Nicholas, M.Sc, B.Sc (Hons), DIC, CEnv, MIMMM

CERTIFICATE OF QUALIFIED PERSON

I, Carl Steven Nicholas, M.Sc, B.Sc (Hons), DIC, CEnv, MIMMM, do hereby certify that:

- I am a Chartered Environmental Consultant, with the company of Mineesia Limited, 4 Mace Farm, Cudham, Kent, TN14 7QN, UK.
- This certificate applies to the Technical Report titled "Revised NI 43-101 Technical Report Pre-Feasibility Study for the Rozino Gold Project, Bulgaria" (reference № R366.2020), dated 15 December 2021 with an Effective Date of 28 September 2020, (the "Technical Report") prepared for Velocity Minerals Ltd. ("the Issuer").
- I am a practising Environmental Consultant and registered as a Professional Member of the Institute of Materials, Minerals and Mining (#477471).
- I am a graduate of Imperial College, London, UK with a Masters in Environmental Diagnosis, with a Bachelor of Science (Honours) degree in Biodiversity Conservation and Environmental Management. I have practiced my profession continuously since 2005 and have 15 years practical experience in Environmental Impact Assessments for mining projects.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I visited the Rozino Gold Project between 13 and 17 May 2019. The purpose of the visit was to assess baseline conditions and establish priorities for environmental management and data collection for the project.
- I am responsible for Section numbers 1.13, 4(part), 5(part), 20, 26.9 of the Technical Report.
- I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
- My involvement with the Rozino Gold Project is limited to environmental and social impact assessment work since November 2014 and including the preparation of this technical report.
- As of the effective date of the technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: 28 September 2020

["SIGNED"]
{Carl Steven Nicholas}

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Carl Steven Nicholas, MIMMM

Signing Date: 15 December 2021

Certificate of Qualified Person – Jonathon Abbott, BASc, MAIG (MPR Geological Consultants Pty Ltd)

CERTIFICATE OF QUALIFIED PERSON

As the author of the report titled "Revised NI 43-101 Technical Report Pre-Feasibility Study for the Rozino Gold Project, Bulgaria" prepared for Velocity Minerals Ltd. (the "Issuer") dated 15 December 2021 with an effective date of 28 September 2020 (the "Technical Report"), I, Jonathon Abbott, BASc, MAIG, do hereby certify that:

- I am a Consulting Geologist with MPR Geological Consultants Pty Ltd, 19/123A Colin Street, West Perth, Western Australia, Australia.
- I graduated with a Bachelor of Applied Science in Applied Geology from the University of South Australia in 1990.
- I am a member of the Australian Institute of Geoscientists. I have worked as a geologist for a total of 31 years since my graduation from university. My experience includes mine geology and resource estimation for a range of commodities and mineralization styles. I have been involved in preparation and reporting of resource estimates in accordance with JORC guidelines for 26 years, and National Instrument 43-101 ("NI 43-101") guidelines for approximately 18 years.
- I have read the definition of "qualified person" set out NI 43-101 and certify that by reason of my education, affiliation with a recognized professional association and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
- I have been involved with the Rozino Project since December 2017 and visited the project site on the 25th of February 2018.
- I am responsible for sections 1.3,1.4,1.5,1.7,1.17.1,6,7,8,9,10,11,12,14,25.1,26.1 and 27 (part) of the Technical Report.
- I am independent of the Issuer (within the meaning of Section 1.5 of NI 43-101).
- I have had prior involvement with the Rozino project. Between December 2017 and September 2018, I prepared Mineral Resource estimates for Velocity Minerals Ltd and authored a Technical Report titled "NI 43-101 Technical Report Mineral Resource Estimation for the Rozino Gold Deposit, Republic of Bulgaria" with an effective date of the 21st of March 2018. I was co-author of a Technical Report titled "NI 43-101 Technical Report Preliminary Economic Assessment – Rozino Project Tintyava Property, Bulgaria" with an effective date of the 19th of September 2018.
- I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September 2020

["SIGNED"]
{Jonathon Abbott}

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Jonathon Abbott, MAIG

Signing Date: 15 December 2021

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