NI 43-101 TECHNICAL REPORT
FEASIBILITY STUDY
LAIVA GOLD PROJECT

Raahe, Finland

Prepared For

OTSO GOLD CORP.

Toronto, Ontario, Canada

By

John T. Boyd Company

Mining and Geological Consultants

Denver, Colorado, U.S.A.

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Attention: Mr. Brian Wesson
President & CEO

Subject: NI 43-101 Technical Report
Feasibility Study
Laiva Gold Project
Raahe, Finland

Dear Mr. Wesson:

Please see attached NI 43-101 Technical Report – Feasibility Study for the Laiva Gold Project (Report), located near Raahe, Finland. Otso Gold Corp. (OTSO) has commissioned the John T. Boyd Company (BOYD) to prepare a Feasibility Study based on recent resource development drilling by OTSO and including all previous resource development work and technical studies by OTSO's predecessors and as commissioned by OTSO where such data is duly qualified for inclusion in this Feasibility Study Technical Report.

We are pleased to provide this Technical Report to assist OTSO with its efforts aimed at restarting the Laiva Project for production.

As you are aware, BOYD has worked on this project previously prior to OTSO's involvement, as well as working alongside the OTSO team more recently, resulting in BOYD's deep understanding of both opportunities and challenges associated with all aspects of the Laiva Project.

Completion of this Feasibility Report to rigorous professional standards coupled with our deep understanding of the project, we hope will provide a sound basis for further advancement of the project.

We look forward to engagement by OTSO as required for future advancement of the Laiva Project.

Respectfully submitted,

JOHN T. BOYD COMPANY
By:

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GLOSSARY OF ABBREVIATIONS AND DEFINITIONS

AAS	: Atomic Absorption Spectroscopy.
AG	: Autogenous grinding.
ALS	: ALS Chemex (Sweden).
Assay	: Chemical test performed on a sample of ores or minerals to determine the amount of valuable metals contained.
Au	: Symbol for the element gold.
AusImm	: Australian Institute of Mining and Metallurgy.
AVR	: Acidification, Volatilization Reneutralization.
Bi	: Symbol for the element bismuth.
BOYD	: John T. Boyd Company.
Bulk Mining	: Large scale, mechanized mining without regard to separately mining discrete zones of mineralization.
°C	: Degrees Celsius.
ca	: Circa annum “around”.
CCD	: Counter Current Decantation.
CIL	: The simultaneous dissolution of gold by cyanide solution in agitated tanks generally in the presence of high levels of dissolved oxygen and absorption of gold in solution onto activated carbon.
CIM	: Canadian Institute of Mining, Metallurgy, and Petroleum.
Cm	: Centimeters.
CN	: Cyanide.
Contact	: A geological term used to describe the line or plane along which two different rock formations meet.
CRM	: Certified Reference Material.
Cu	: Symbol for the element copper.
Cut-off Grade	: The lowest grade of mineralized rock that qualifies as ore grade in a given deposit, and is also used as the lowest grade below which the mineralized rock currently cannot be profitably exploited. Cut-off grades vary between deposits depending upon the amenability of ore to gold extraction and upon costs of production.
DCF	: Discounted Cash Flow.
Deposit	: An informal term for an accumulation of mineralization or other valuable earth material of any origin.
Dilution	: Waste rock that is, by necessity, removed together with the ore in the mining process, thereby reducing the mined grade of the ore.
Dip	: The angle at which a vein, structure or rock bed is inclined from the horizontal as measured at right angles to the strike.
$	: US Dollars.
€	: Euro.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

Economic Pit-Shell	: The spatial limits determined by an algorithm to define the optimum minable resources from an unconstrained resource model. The optimum ultimate pit of a mine is defined as the "pit shell", which is the result of extracting the volume of material that provides the total maximum profit while satisfying the operational requirements of safe wall slopes. The ultimate pit limit gives the shape of the mine at the end of its life. Usually this contour is smoothed to produce the final pit outline.
EPA	: Environmental Permit Authority.
Fault	: A break in the Earth's crust caused by tectonic forces which have moved the rock on one side with respect to the other.
Flotation	: A milling process in which valuable mineral particles are induced to become attached to bubbles and float as others sink.
Grade	: Term used to indicate the concentration of an economically desirable mineral or element in its host rock as a function of its relative mass. With gold, this term may be expressed as grams per tonne (g/t) or ounces per tonne (opt).
g/t	: Grams Per Metric Ton.
GPR	: Ground Penetrating Radar.
>	: Greater than.
GTK	: Geologic Survey of Finland.
H	: Hours.
Ha	: Hectares.
Hanging Wall	: The rock on the upper side of a vein or mineral deposit.
HG	: High grade, as in the higher grade gold portion of a process stream
High Grade	: Rich mineralization or ore. As a verb, it refers to selective mining of the best ore in a deposit.
Hydrothermal	: Processes associated with heated or superheated water, especially mineralization or alteration.
ICP	: Inductively coupled plasma analysis, an analytical technique used for elemental determinations.
ID^3	: Inverse distance cubed, a deterministic method for multivariate interpolation with a known scattered set of points, whereby the weighting of the value for a block in a block model is weighted by the cube of the distance to known data points.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

Indicated Mineral Resource	: An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.
Inferred Mineral Resource	: An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
Intrusive	: A body of igneous rock formed by the consolidation of magma intruded into other rock type.
IP	: Induced Polarization Survey.
JORC	: Joint Ore Reserve Committee Australasian Code.
Kg	: Kilograms.
km	: Kilometers.
kV	: Volts x 1000.
kWh/t	: Kilowatt Hours Per Ton.
Leaching	: The separation, selective removal or dissolving-out of soluble constituents from a rock.
<	: Less than.
LG	: Low grade, as in the lower grade gold portion of a process stream.
m	: Meters.
Ma	: Mega-annum (one million years).
m.a.s.l.	: Meters above mean sea level.
Measured Mineral Resource	: A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

Metallurgy	: The science and art of separating metals and metallic minerals from their ores by mechanical and chemical processes.
Metavolcanic	: Being or relating to a type of metamorphic rock originally produced by a volcano, either as lava or tephra, then buried and subjected to high pressures and temperatures, causing it to recrystallize. *Metavolcanic rock is commonly found in greenstone belts.*
mt or t	: Metric tonnes.
Mineral	: A naturally occurring homogeneous substance having definite physical properties and chemical composition and, if formed under favorable conditions, a definite crystal form.
Mineral Resource	: A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals. The term mineral resource used in this report is a Canadian mining term as defined in accordance with NI 43-101 – Standards of Disclosure for Mineral Projects under the guidelines set out in the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM), Standards on Mineral Resource and Mineral Reserves Definitions and guidelines adopted by the CIM Council on December 11, 2005 and recently updated as of May 10, 2014 (the CIM Standards).
Mineral Reserve	: 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 reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study. *Note: no Mineral Reserves are reported in this Technical Study*.
mm	: Millimeters.
NAG	: Non-acid generating waste.
NI 43-101	: National Instrument 43-101 Standards of Disclosure for Mineral Projects.
NORDIC	: Nordic Mines Oy.
nT	: Nanotesla, a unit of magnetic field intensity.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

Open pit or cut	: A form of mining operation designed to extract minerals that lie near the surface. Waste or overburden is first removed, and the mineral is broken and loaded for processing. The mining of metalliferous ores by surface-mining methods is commonly designated as open-pit mining as distinguished from strip mining of coal and the quarrying of other non-metallic materials, such as limestone and building stone.
Outcrop	: An exposure of rock or mineral deposit that can be seen on surface that is not covered by soil or water.
oz or tr oz	: Troy Ounces.
P80	: 80% passing, commonly used to designate particle size 80% of which will pass through a given screen opening
PAG	: Potentially acid generating waste.
Plant	: A building or group of buildings in which a process or function is carried out; at a mine site it will include warehouses, hoisting equipment, compressors, maintenance shops, offices and the mill or concentrator.
ppb	: Parts per Billion.
ppm	: Parts per Million.
QAQC	: Quality Assurance / Quality Control, procedures implemented to assure integrity of results.
Qualified Person (QP)	: Conforms to that definition under NI 43-101 for an individual: (a) to be an engineer or geoscientist with a university degree, or equivalent accreditation, in an area of geoscience, or engineering, related to mineral exploration or mining; (b) has at least five years' experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these, that is relevant to his or her professional degree or area of practice; (c) to have experience relevant to the subject matter of the mineral project and the technical report; (d) is in good standing with a professional association; and (e) in the case of a professional association in a foreign jurisdiction, has a membership designation that (i) requires attainment of a position of responsibility in their profession that requires the exercise of independent judgement; and (ii) requires (A.) a favorable confidential peer evaluation of the individual's character, professional judgement, experience and ethical fitness; or (B.) a recommendation for membership by at least two peers, and demonstrated prominence or expertise in the field of mineral exploration or mining.
RC	: Reverse Circulation Drilling.
RQD	: Rock Quality Designation.
Shoot	: A concentration of mineral values; that part of a vein or zone carrying values of ore grade.
SG	: Specific Gravity.
SMU	: Selective mining unit.
SRK	: SRK Consulting, an independent international consulting group focused on advice and solutions to earth and water resource industries.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

Strike	: The direction, or bearing from true north, of a vein or rock formation measure on a horizontal surface.
Sulphide	: A group of minerals which contains Sulphur and other metallic elements such as copper and zinc. Gold and silver are usually associated with sulphide enrichment in mineral deposits.
Ton or Tonne	: A metric ton of 1,000 kilograms (2,205 pounds).
tpd	: Metric Tonnes Per Day.
Till	: Unsorted material deposited directly by glacial ice and showing no stratification.
µm	: µm (0.001 millimeter).
Vein	: A fissure, fault or crack in a rock filled by minerals that have travelled upwards or laterally from a deep source.
VMS	: Volcanogenic Massive Sulfide.
Vulcan	: Implicit Modeling Software.
VRS GPS	: Virtual Reference Station Global Positioning System.
Waste	: Unmineralized, or rock which is insufficiently mineralized to mine at profit.
XRD	: X-ray diffraction, a rapid analytical technique primarily used for phase identification of a crystalline material.
XRF	: X-ray fluorescence a non-destructive analytical technique used to determine the elemental composition of materials.
Zone	: An area of distinct mineralization.

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1.0 SUMMARY

1.1 Introduction

This Technical Report – Feasibility Study was prepared for Otso Gold Corp (OTSO), a TSX-V listed gold mining company for its existing Laiva Gold Project (LGP) by the John T. Boyd Company (BOYD). It is prepared as an NI 43-101 Technical Report in accordance with Form 43-101F1, and duly signed by its authors as Qualified Persons (QP) as defined under National Instrument 43-101 Part 1, Section 1.1. The purpose of this Technical Report – Feasibility Study is to recommend modifications to past mining practice, and minor adjustments to the process flow sheet in order to provide additional perspective to OTSO's internal determination of value for the benefit of its shareholders and to provide support for further financing of the LGP.

BOYD was commissioned by OTSO to prepare an updated Technical Report – Mineral Resource Update with an Effective Date of 4 October 2021, which preceded the Feasibility Study provided below. The LGP is comprised of existing open pit mines, a nominal 6,000 mtpd processing plant, and all related infrastructure. The LGP has an extensive history of exploration dating back to 1980 when it was first discovered. Since that time, it has gone through a succession of exploration and resource development, culminating in construction of the existing process plant and related facilities and initiation of production mining in January 2012. Since the start of production in 2012, the LGP has operated sporadically by several groups with only marginal success, largely due to poor mining practices resulting in massive mine dilution and undercapitalization. OTSO has taken over management of the LGP and have engaged appropriately skilled mining and process professionals to manage the operation.

This Feasibility Study includes revisions to mining practices and minor modifications to the existing process plant that result in the production and economic projections contained herein.

1.2 Property Description and Location

Location

The Laiva Gold Project is located in Northern Ostrobothnia, western Finland, centered on 2528500 mE, 7161000 mN (Finnish Coordinate System, Zone 2), and is situated 20 km southeast of the nearest town, Raahe, and 70 km directly south of Oulu which is the closest regional center. See Figure 1.1, following this page.

Figure 1.1 – Location of Laiva Gold Project

Mineral and Surface Tenure

The LGP is comprised of two license areas: Laiva and Oltava. The Laiva area comprises the advanced stage Laiva Gold Mine and satellite exploration projects at Mussuneva and Kaukainen, see Figure 1.2, following this page. Oltava is located 12 km south of Laiva and is an early-stage exploration property (Figure 1.2).

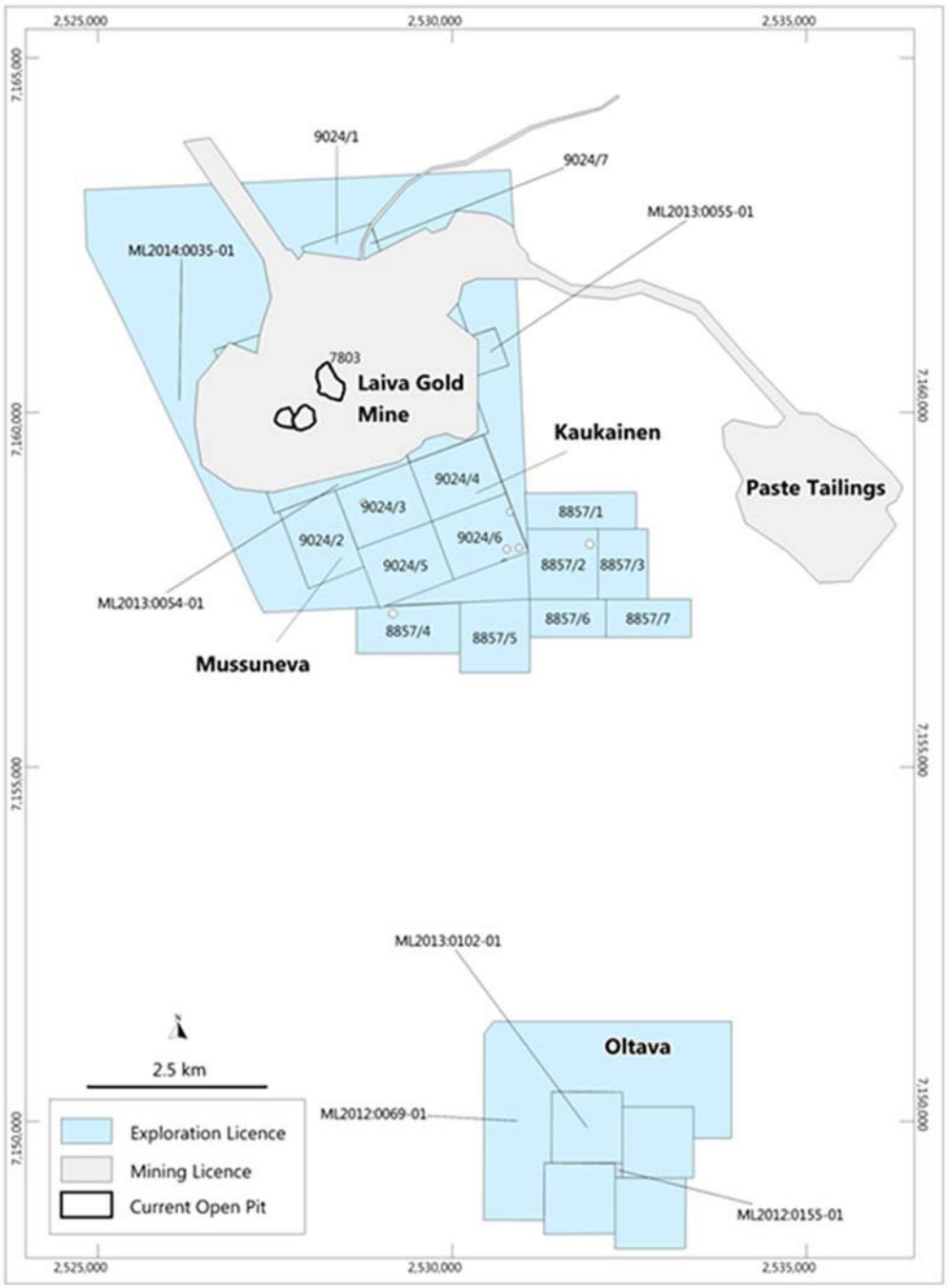

Figure 1.2 – Map Showing Otso Gold Corp Mineral Rights and Registration Numbers

The LGP is comprised of exploration licenses totaling 33.6 km^2 and a single 16.9 km^2 mining license, for a total landholding of 50.5 km^2. The location and registration numbers of the licenses which form the LGP are shown in Figure 1.2 above.

OTSO holds 900 ha of the surface area within the mining license area. The remainder of the surface rights at the LGP within the mining and exploration licenses are held by private landowners and forestry companies. The mining license permits OTSO to conduct mining related activities and the surface rights are considered sufficient for mining operations. The area of the mining license is uninhabited with limited commercial logging in the vicinity of the project area.

In addition to the annual exploration fees, compensation is payable to landowners within the exploration license in the event that exploration activity such as drilling causes damage (for example, removal of trees). The compensation must be approved by the mining authority prior to commencement of the activities.

At Oltava, surface rights are held by a joint venture windfarm project between Tornator Oy and Taaleritehdas Oy. This joint venture was granted a permit for construction of a windfarm over the same area as the exploration licenses subsequent to the granting of exploration licenses to OTSO.

Royalties

A 0.15% royalty is payable to landowners within the mining permit and is paid pro rata depending on the area of land they own within the mining permit. The royalty is based on the average annual value of metal produced in the subject period.

A 2.5% NSR royalty is payable to PFL RAAHE HOLDINGS LLP, in connection with settlement of previous project debt.

1.3 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

Accessibility

The LGP is readily accessed from the town of Raahe, North Ostrobothnia, located 20 km northwest of the project with a population of approximately 20,000. Raahe is located 600 km north of Helsinki and 80 km southwest of Oulu and is located on a major coastal highway (E8). A port is located at Raahe, as well as a steel plant. The closest airport is located at Oulu which has direct flights to Helsinki. Alternatively, Raahe is easily accessed by rail or air from Helsinki to Kokkola and then by vehicle from Kokkola to the mine site.

From Raahe, the project is reached via public paved roads (Road 88) to the village of Kopsa, see Figure 1.3 below. A paved road from Kopsa leads to a turn onto a 4 km long gravel access road which leads directly to the Laiva Mine site and offices. Access is possible year-round. A separate exploration office and core storage facility is located 6 km east of the mine site and can be accessed via gravel roads from the mine, or paved road from Raahe. The Oltava project is accessed from Raahe via Kalajoki and Polusperä.

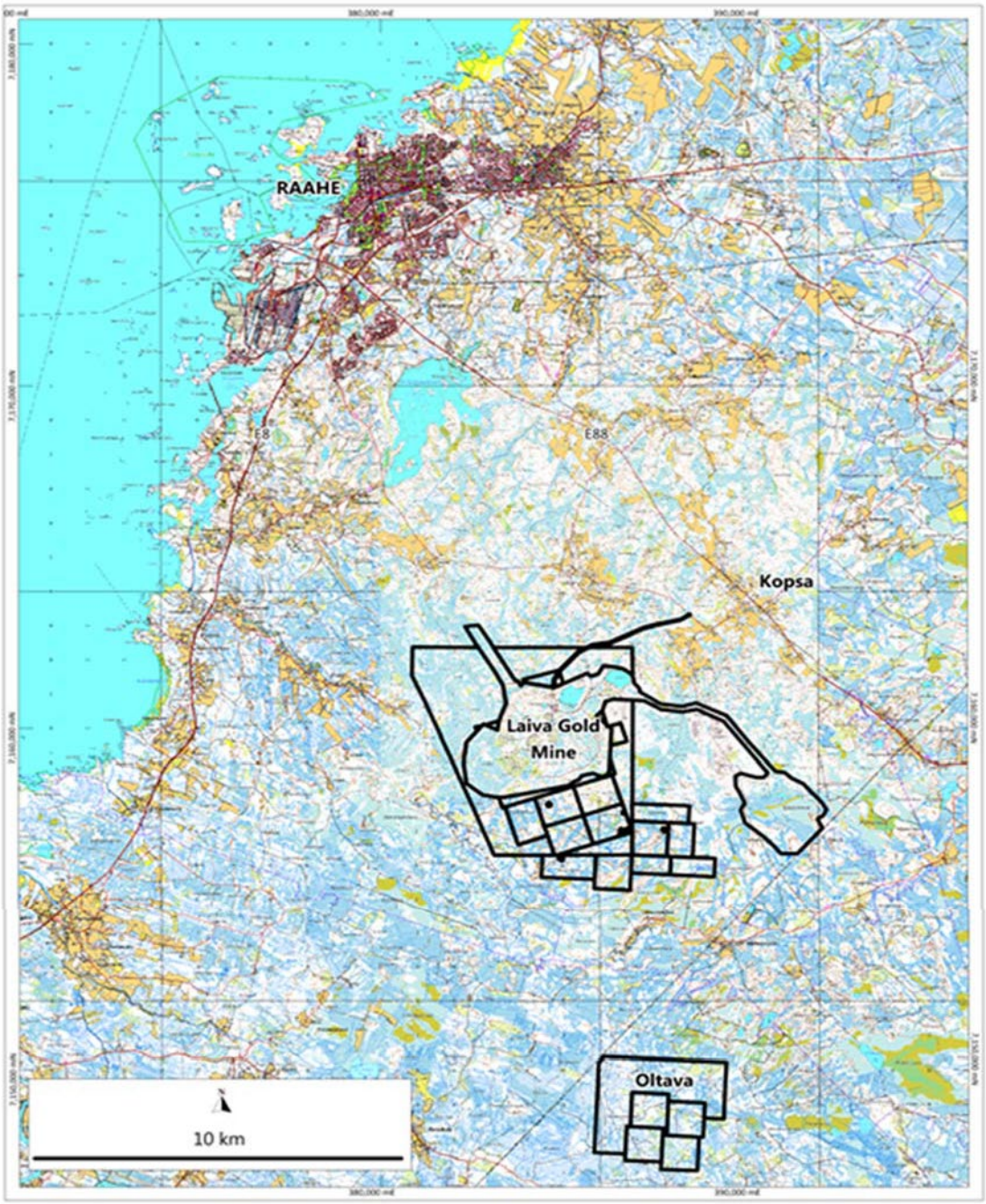

Figure 1.3 – Laiva Gold Project License Areas Access

Access within the Laiva property is via all-weather gravel roads constructed by the previous operator as well as existing forestry and public gravel roads. Access is generally good across both the Laiva and Oltava projects, with temporary access routes required for exploration drill sites located within forested areas. The primary restriction to access is in areas of peat bogs, where some ground preparation is required to provide vehicular access and stable drill platforms.

Physiography

The project area is generally flat lying with gentle topography within an area of Boreal Forest. Broad, shallow depressions contain peat bogs, and the area is incised by shallow streams. Elevation varies between 30 m.a.s.l in the northwest and 100 m.a.s.l inthe east. The mine site elevation varies between 50 and 60 m.a.s.l.

Vegetation is comprised of pine, spruce, and birch trees with occasional grassed cleanings and boggy areas. The following aerial photograph (supplied by OTSO) of the Laiva GoldMine shows physiography and site infrastructure in Figure 1.4 below:

Figure 1.4 – Laiva Gold Mine Showing Physiography and Site Infrastructure

Climate

North Ostrobothnia is subject to a sub-arctic climate, typified by mild summers and cold winters, with average temperatures of -9°C in winter and 15°C in summer. Extreme temperatures of -35°C in winter and 30°C in summer are rarely observed. Annual precipitation is relatively low, averaging between 300 and 600 mm with most precipitation occurring in the summer months. The ground is snow covered for much of

winter, although the snow rarely exceeds 1 m deep and has not historically affected the operating season for exploration and mining activities.

Mine, Plant, and Infrastructure

The LGP consists of two open pit mines currently being restarted and a 6,000-tonne-per-day mill and processing plant, including crushers, autogenous and pebble grinding mills, flash flotation concentrator, gravity circuit with centrifugal and spiral concentrators, and high and low grade CIL leach circuits.

Tailings facilities include a lined high-grade tailings pond adjacent to the CIL plant, and a low-grade tailings paste storage repository fed by a slurry pipeline located 7 km east of the process plant. Separate waste dumps for inert and sulphide-rich material are used and located proximal to the pits.

1.4 History

The LGP was discovered through boulder sampling by an amateur prospector in 1980. The area was subsequently acquired by Outokumpu Oy (Outokumpu) in 1981 who explored the area until 1986 when it relinquished the licenses. The area was later acquired by Endomines Oy (Endomines in 1999 who explored the project until it was later acquired. The LGP was acquired from Endomines by Nordic Mines AB, a Swedish public company through its wholly owned subsidiary company Nordic Mines Marknad AB, also a Swedish company. Nordic Mines Marknad AB in turn was the previous 100% owner of Nordic Mines Oy, (NORDIC a Finnish company. Subsequently, NORDIC was acquired by Fire Steel Resources Inc. (FTR). Finally, after abandonment to its lender, OTSO took over NORDIC and the LGP.

The Oltava Project was first recognized in the 1950s through regional exploration and drilling conducted by the Geological Survey of Finland (GTK. The project was briefly explored by Rautaruukki Oy in 1984 and was owned by Outokumpu between 1985 and 1986. GTK conducted a second phase of exploration at Oltava between 1994 and 2002, prior to the acquisition of Oltava by NORDIC in 2005, followed by the sale to FTR and finally, the transfer to OTSO.

1.5 Geologic Setting and Mineralization

Finland is part of the Archean-Palaeoproterozoic Fennoscandian Shield, which comprises the Archean Karelian and Kola cratons flanked by the Paleoproterozoic Caledonide and Svecofennian orogenic belts to the west and southwest. Phanerozoic sedimentary rocks are juxtaposed against the shield to the east.

The Laiva Gold Deposit is located in the Svecofennian orogenic belt, which formed during northeast vergent collision between multiple calc-alkaline volcanic arc complexes and microcontinents with the foreland Karelian Craton approximately 1,900 Ma. The contact between the Karelian Craton and the Svecofennian orogenic belt is marked by the deep, crustal scale, northwest trending Raahe-Ladoga suture, which is located approximately 50 km north of the Laiva Gold Deposit and can be traced through Finland into Sweden. See Section 7.0 herein, Figure 7.1.

The accretionary history of the Svecofennian orogenic belt is complex and comprises microcontinent accretion, continental extension, continent-continent collision, orogenic collapse and then stabilization at ca. 1,770 Ma (Nironen, 2017). Several stages of Svecofennian granitic magmatism are recognized, comprising syn-orogenic (1,930 to 1,850 Ma) and late-orogenic (1,850 to 1,770 Ma) stages (Hanski, 2016).

Between 1,860-1,890 Ma an orogen-parallel shear regime was initiated resulting in bimodal volcanism and intrusion of associated plutons. This stage of the Svecofennian orogeny is recognized as the most important in terms of regional orogenic gold mineralization (Eilu et al., 2003).

The region experienced relative tectonic quiescence until rifting associated with formation of the Iapetus Ocean between 560 and 400 Ma, followed by exhumation, weathering and erosion of the Fennoscandian Shield between 300 and 60 Ma. Pleistocene glacial till covers much of the low-lying areas of the shield (Eilu et al., 2003).

1.6 Deposit Type

Laiva is classed as an orogenic gold deposit. Orogenic gold deposits form during subduction-related accretion or collision and are spatially related to second and third order faults or shear zones proximal to crustal-scale zones of deformation. In addition, orogenic gold deposits are commonly discovered proximal to granitoid intrusions, although the nature of the relationship between gold mineralization and granitoid magmatism is unclear.

Orogenic gold deposits are characterized by formation in the late stages of orogenesis in compressional or transpressional regimes. Gold is typically deposited synchronous with ductile shearing and/or folding, and occurs in shear zones, fold hinges and lithological contacts. Mineralization forms as discrete veins, sheeted vein arrays or disseminated bodies with variable grade continuity, controlled principally by the local structural geology and chemistry of the host rock (Groves et al., 2003).

Orogenic gold deposits display a metal association of gold-silver, arsenic, boron, bismuth, antimony, tellurium, and tungsten. Vertical and lateral metal zonation is usually subtle, with gold mineralization often recognized over significant vertical intervals of >1 km.

The above characteristics are largely evident at Laiva, where gold mineralization is hosted by an array of sheeted quartz-sulphide veins and mineralized shear zones, cross cutting greenschist facies metavolcanic rocks and quartz diorite intrusions. Mineralization at Laiva is often associated with anomalous arsenic, and more rarely with copper, molybdenum, bismuth, and telluride. The presence of molybdenum and copper in association with gold is somewhat atypical of orogenic gold deposits and may be influenced by the host rock geochemistry, may represent an ore forming fluid of magmatic origin, or represent a separate stage of mineralization related to higher temperature hydrothermal fluids. This is reflected in bedrock geochemistry where molybdenum appears to occur peripheral to the gold mineralization at Laiva and Oltava, and where copper is not always associated with gold mineralization.

Mapping of gold and trace elements including arsenic, molybdenum, and copper has been proven to be an effective exploration tool at Laiva and should be continued, in conjunction with further regional interpretation of structural controls on mineralization at Laiva. An increased understanding of the structural controls to mineralization is likely to lead to better evaluation of exploration potential and will also assist with the potential definition of high-grade shoots at Laiva.

1.7 Exploration

Under previous management, NORDIC commenced a phased regional geochemical sampling program in 2005, including an initial assessment of historic regional geochemical and geophysical data obtained from previous owners, followed up by surficial till sampling, base of till and bedrock sampling of anomalous zones identified from surficial till sampling, and finally channel sampling at Mussuneva of anomalous zones identified from bedrock sampling and drilling of the Laiva and Oltava prospects.

In 2010, NORDIC, under previous ownership commissioned a detailed mapping and structural geology interpretation, conducted by Specialized Geological Mapping Ltd (Pratt, 2010). In 2016, NORDIC conducted a trial Ground Penetrating Radar (GPR) survey at the Laiva Gold Deposit. The methodology and results of the exploration conducted by NORDIC and the previous operators are summarized below.

Following acquisition by FTR, NORDIC did not conduct additional significant exploration work. Since transfer of the LGP from FTR, OTSO has conducted significant additional drilling, which is also summarized below.

The exploration work programs conducted to date by previous operators demonstrate that top of bedrock sampling is an effective exploration tool for the mineralization styles observed in the Laiva area. Several exploration targets exist proximal to the current mineral resource and operational area, as well as satellite targets at Mussuneva Kaukainen and Oltava:

- Near-mine exploration potential exists to the north of the North Pit, associated with a northeast trending structure defined by a gold-arsenic anomaly in bedrock sampling. The New Hope area immediately west of the North Pit is small but has demonstrated strong gold-arsenic in top of bedrock sampling. Infill top of bedrock sampling and a detailed ground magnetic survey is recommended for both areas.

- Additional top of bedrock sampling between the Mussuneva and Kaukainen prospects is warranted to infill a gap in sampling between the two prospects and potentially link the two bedrock$_{80}$ gold-arsenic-copper anomalies. Detailed ground magnetic surveys are also recommended. This will assist in exploration drill program planning in this area.

- Widely spaced, top of bedrock sampling is warranted at Oltava, where two gold anomalies remain open along strike and a large proportion of the license holding remains untested.

Further studies into the gold grain size distribution, metal associations, structural geology, and timing of granite magmatism relative to mineralization are recommended in order to improve gold recovery, refine the geological model, and potentially develop additional areas of exploration potential.

Results from the recent drilling campaign conducted by OTSO in 2021, along with previous drilling form the basis for the recent BOYD Resource update with an effective date of 4 October 2021. Drilling data to date used in this resource estimate update is presented in Section 10.0 herein.

1.8 Drilling

Several phases of drilling have been conducted at Laiva by Outokumpu, Endomines, and NORDIC, under various operators between 1981 and 2014. The majority of drilling has been conducted in the mine area, with a limited number of historic exploratory holes drilled at Mussuneva, Kaukainen, and Oltava.

For the purposes of mineral resource estimation, only the holes drilled by NORDIC under various operators have been considered by BOYD. The drilling conducted by Outokumpu and Endomines is considered to be historic (pre-2000) and the data cannot be qualified under NI 43-101 requirements. The data from these historic programs is however, a useful tool in exploration planning.

No drilling was being performed at the time of BOYD's site visits; therefore, the operating procedures have not been observed first-hand by the authors. The drilling database contains exploration, geotechnical, grade control and blasthole data. The exploration and grade control data were used for mineral resource estimation. Only the drilling conducted by NORDIC under various operators which pertains to the mineral resource estimate at Laiva is discussed in the following sections. See Figure 1.5 below for drill hole collar locations and traces.

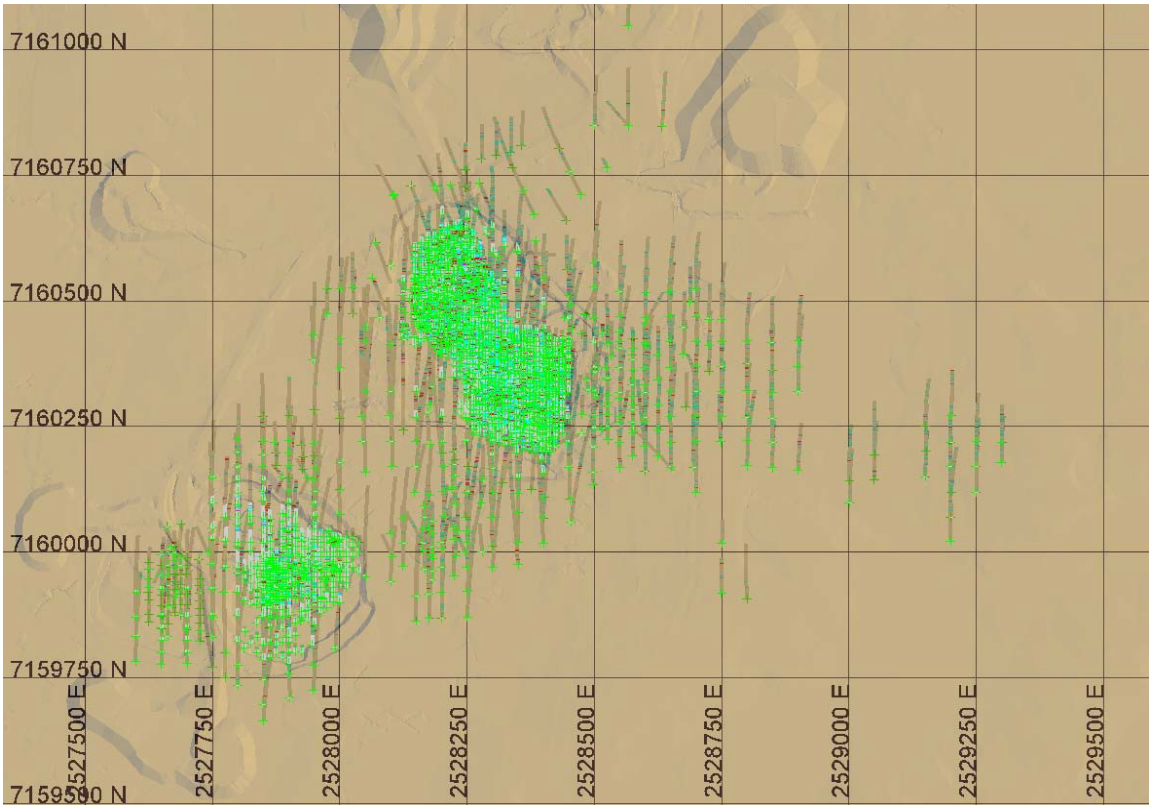

Figure 1.5 – Drill Hole Locations and Traces

Diamond drill core recovery was routinely measured during core logging and recorded in the detailed descriptions of the lithology database. Core recoveries are typically greater than 95% and often 100%, with no bias observed between core recovery and gold grade. A visual inspection of core and core photos by BOYD confirmed the excellent diamond drill core recovery.

Drilling by NORDIC has defined gold mineralization at the Laiva Gold Mine over an area of 1,300 m by 600 m and to a maximum depth of 250 m below surface. The drilling and sampling procedures have been conducted to industry standards and the results of the drilling program are considered adequate for mineral resource estimation purposes.

Overall the drilling, logging, and sampling procedures as well as the database provided are considered reliable and adequate for mineral resource estimation purposes.

BOYD strongly recommended at the time that future sample intervals are determined by lithology and presence of mineralization. While a nominal sample length of 1 m is generally acceptable, the sample interval should be further determined by the geologist, where a sample is terminated at a lithological contact or change in mineralization or alteration. This would allow better granularity in assessment of assay results and provide for a more accurate estimate of gold grade and mineralized interval width in proximity to post mineral intrusive units.

Oriented core should be obtained as regularly as possible for infill drill holes in order to better define the structural controls on mineralization and provide for more accurate geological modelling of the mineralized zones, dikes, and faults.

1.9 Sample Preparation, Analysis, and Security

Under the first operator, NORDIC employed the Labtium Oy, Finland (formally GTK Geolaboratory) and ALS Chemex (Sweden) (ALS) laboratories for sample preparation and analysis for all exploration samples, including channel samples, diamond drill core and RC chips. Both laboratories were independent of NORDIC and are certified under the following schemes:

- GTK Geolaboratory/Labtium Oy: T025 (EN ISO/IEC 17025), Finnish Accreditation Service (FINAS).
- ALS Chemex (Sweden): ISO/IEC 17025, SWEDAC Ackreditering.

Grade control samples were prepared and analyzed at NORDIC's on-site laboratory, which was managed by independent contractors (CRS). This facility is not certified.

Under FTR and OTSO operation of NORDIC, both used CRS Laboratories located in Kempele, Finland, for sample preparation and CN soluble and ICP analysis, with on shipment to MSALABS Ltd. (ISO 9001) in Langley, BC, Canada, for 30 g fire assay with AA finish.

Under previous ownership, exploration samples were reportedly held in custody of the site geologist until return to the exploration office, where samples were stored in locked sheds until dispatch to the preparation laboratory by courier. Upon delivery to the preparation laboratory, samples were receipted, and a report was sent to the company by electronic mail confirming the sample numbers and safe receipt of samples. Under FTR and OTSO management, chain of custody was similar, but samples were dispatched in sealed containers to the nearby CRS lab by company truck. A dispatch sheet was submitted to CRS and the sample numbers were checked by CRS upon sample receipt.

In BOYD's opinion, the above chain of custody arrangements maintained the validity and integrity of samples prior to analysis. Chain of custody under both FTR and OTSO was handled similarly to previous practices. BOYD has reviewed practices by both FTR and OTSO and can confirm appropriate chain of custody under both entities.

1.10 Data Verification

Sample preparation, security and analyses have in general been reported to be conducted to industry standards and the data therefrom are adequate for use in mineral resource estimation.

However, the QAQC program implemented by previous operators of NORDIC has some deficiencies in that blind, external blanks and CRM have not been added to the sample stream, and sample batches of variable sizes were dispatched. Further, the selection, frequency, and location within a batch of samples for laboratory blanks, CRM and pulp duplicate samples were determined by the laboratory and not NORDIC. The result is that duplicate samples were not routinely taken of mineralized samples, and blank samples were not inserted following suspected elevated gold mineralized samples.

In addition, the field duplicate samples selected from half cut drill core were of a very low grade (average 0.19 g/t gold from 52 samples) and therefore the results are of limited use in characterizing the mineralization.

The QAQC program instituted by OTSO was a significant improvement to past practices in that previous BOYD recommendations were implemented to:

- Dispatch samples in batches containing 20 samples, comprised of 16 field (core or RC chip) samples, one CRM, one blank, one field duplicate, and a pulp (laboratory) duplicate.
    - The CRM and blank material should be selected by the operator and inserted blind to the laboratory.
    - Where possible, blank samples should be inserted immediately following a suspected elevated gold mineralized sample to assess the potential for contamination during sample preparation.
    - Similarly, field and pulp duplicate samples should be selected from mineralized samples to better assess repeatability.
- Use of external CRM material inserted blind to the laboratory will give greater control over the analysis of accuracy and precision at the selected laboratory, and therefore greater confidence in the results used for mineral resource estimation purposes.
- Control limits for blanks and CRM should be set in order to pass or fail a batch of samples. This will allow continuous monitoring of each batch of samples and assay results. Batches that fail should be immediately re-assayed, and when all QAQC samples in a batch have passed, the results should be signed-off by the Competent Person as final and acceptable for public disclosure and use in mineral resource estimation.

1.11 Mineral Processing and Metallurgical Testing

Process design criteria for the restart of the LGP treatment plant were developed by completing the following key tasks:

- Conduct detailed evaluation of all operating information from previous operating periods with an emphasis on a 14-month period of continuous operation by previous operators.
- Evaluate comminution test work to determine maximum and optimum production rates for the existing comminution circuits and to identify modifications to improve the circuits. Comminution test work recently completed in 2021 included:
    - Rod mill work index
    - Ball mill work index
    - JK drop test

- Evaluate results from the metallurgical test program completed in 2021 at Resource Development Inc. (RDi) located in Wheat Ridge, Colorado. The RDi test program utilized samples from each of the primary rock types from the LGP composited from individual drill hole sections. Treatment parameters evaluated included:
    - Flash flotation
    - Gravity concentration
    - Cyanide leaching of concentrate tailings
- Evaluate results from all previous metallurgical test work completed prior to the 2021 RDi test program.

The most critical plant operating parameter was determined to be the proportion of gold in the plant feed reporting to the high grade (HG CIL circuit. During previous periods of operation, the HG CIL circuit consistently achieved greater than 97% gold extraction. However, the proportion of contained gold in the feed was generally around 40% reporting to the HG CIL circuit.

BOYD determined that inadequate capacity in the concentrate regrind circuit to grind to suitable particle size (P_{80} 45 µm) for feed to the HG CIL circuit was the limiting factor affecting full utilization of the HG CIL circuit capacity. Of only slightly lesser importance was failure to maximize pre-leach gold concentration in the flash flotation and gravity concentration circuits. Accordingly, the primary focus in developing the updated process design criteria was to maximize utilization of the HG circuit by increasing regrind capacity and optimization of pre-concentration circuits ahead of CIL.

In order to achieve the process plant “name plate” capacity of nominal 250 mtph or 2 million mtpy at required grind for best overall cost-effective recovery, revisions to the feed to the primary AG mill need to be implemented. Past operations only rarely recorded actual throughput at these rates, and when actually achieved generally resulted in reduced plant recovery. Comminution test work conducted by past operators showed a wide range of grinding work indices, which resulted in considerable variability in comminution performance during operation.

In order to better understand the variability among ore types, BOYD reviewed ore reserves by rock type and recommended further comminution testing to accurately determine comminution performance by rock type. In addition, analysis by BOYD’s comminution expert determined that feeding the primary autogenous grinding mill (AG with crushed product at nominal 200 mm particles (20% of feed to serve as grinding media and feeding secondary crushed product at nominal 25 mm particles (80% of feed would result in 1 reduced energy cost, 2 increased through put to name plate capacity,

and 3) result in ground product (through AG and pebble mills in closed circuit) of P_{80} 100 mm product, as required for optimum process recovery.

Large bulk samples were collected by OTSO project geologists for new comminution test work. Samples were collected for the two primary rock types, quartz diorite and mafic volcanics. Together, these two rock types make up approximately 90% of the current resource. The new test work included:

- Rod mill bond work index.
- Ball mill work index.
- JK drop test.

Based on the results of an evaluation of the 2021 comminution test work and all available historical operating information, the calculated mill circuit maximum production rate is 2.0 M mtpy with a target mill final product particle size of P_{80} 100 µm.

BOYD recommends restarting the mill at an initial feed rate of 1.5 M mtpy, gradually increasing to 2.0 M mtpy provided recoveries are maintained. Overall plant operating parameters should be optimized at each production rate before increasing to the next increment.

A laboratory scale metallurgical test program was completed in 2021 to evaluate the LGP treatment flow sheet utilized in the existing treatment plant. The test program was conducted by Resource Development Inc. (RDi) located in Wheat Ridge, Colorado.

The RDi test program included evaluation of the plant unit operations utilizing a QD composite sample with a gold grade similar to the estimated gold grade for the plant feed. The test program included the testing of flash flotation at P_{80} 1.5 mm and gravity concentration followed by tailings cyanidation at P_{80} 75 µm and P_{80} 105 µm.

The primary purpose of the test program was to develop process design criteria for minor, but important revisions to the existing treatment plant.

Primary conclusions from these results include:

- The 100 µm and 75 µm particle size samples both responded well to flotation concentration, gravity concentration and cyanidation of the concentrate tailings producing tailings grades that are essentially the same for both particle sizes.

- The 75 µm provided a superior overall recovery along with having a 10% calculated head grade. This result indicates that following restart of the plant, it wouldbe warranted to evaluate mill particle sizes finer than the initial restart target of P80 100 µm.
- Approximately 60% of the contained gold was recovered into a float/gravity concentrate representing approximately 6% of the weight of the feed for both samples.
- The flotation concentrate contained a higher percentage of the contained gold than the gravity concentrate for both samples.
- Both samples were floated at the same particle size of P80 = 1.5 mm; therefore, the higher gold flotation recovery in the 75 µm sample is likely due to differences in the laboratory flotation procedures.

1.12 Mineral Resource Estimates

BOYD developed three-dimensional (3D) geological models for the deposit using Maptek Pty. Ltd.'s Vulcan software. The procedures used to model and estimate the mineral resources consist of the following steps:

1. Assemble and validate the exploration (drillhole) database.
2. Load the exploration database into Vulcan and validate the results.
3. Develop 3D wireframe models of the mineralized domains and surrounding rock masses for each deposit using previously developed cross-sectional interpretation and advanced implicit modelling techniques.
4. Examine the various sampling lengths and establish a composite length for assay composites.
5. Create a block model based on the resource area geology and mineralized domains developed in Step 3 above.
6. Determine, based on lognormal probability charts of the assay data, the threshold gold grade to limit the area of influence of high-grade gold assays.
7. Flag the sample composites by their intersection with the various mineralized domains as developed in Step 3 above.
8. Using the composites from Step 7 above, develop variograms for gold grades in each potentially mineralized domain.
9. Develop grade estimation parameters and interpolate block grades.
10. Flag the blocks located above or below topography.

11. Run the post-interpolation script that determines mineral resource classification, block density, and rock codes for use in pit optimization.

12. Validate the block grade estimates using QQ plots and visual inspection against the underlying drillhole samples.
13. Export the block model into a format suitable for loading into Geovia's Whittle pit optimizer.

14. Import the block model into the Whittle pit optimizer.

15. Determine economic pit limits to constrain the open pit mineral resource estimates using Whittle's pit optimization tools.

16. Import the pit optimization results into Vulcan.

17. Determine a grade shell of the Whittle pit results and flag the model for material within the Whittle pit limits and material outside of the Whittle pit limits.

18. Determine mineral resources inside the Whittle pit shell and underground mineral resources outside of the Whittle pit shell.

All of the available exploration, grade control, channel samples, and blast holes were reviewed as part of the current mineral resource estimate. The total of these drill holes is 18,835 amounting to 260,657.9 meters in length. Drilling occurred in 1982 through 1986; 1999; 2005 through 2014; 2018 through 2019; and 2021. Drilling consisted of 536 diamond core exploration holes totaling 78,841.1 meters; 139 reverse circulation exploration holes totaling 16,363.7 meters; 78 channel samples totaling 361.9 meters; 14,918 blastholes totaling 92,137.0 meters; 201 grade control diamond core holes totaling 6,836.0 meters; and 2,963 grade control reverse circulation drill holes totaling 66,118.3 meters.

There are indications that additional drilling was completed in the past, but no records could be located to confirm this.

Much of the data is not considered NI 43-101 compliant and was removed from the final database used for the mineral resource estimate. This included any drillholes drilled prior to 2001 as well as all of the blasthole data. The authors consider that the blasthole data did not have the same level of QA/QC as the other exploration data and contained significant downhole gold contamination that tended to smear gold grades into adjoining waste zones. The result of this is a significantly reduced drill hole database that is NI 43-101 compliant.

Data from geologic field mapping as well as geologic and assay data from the drill hole database were used to determine four distinct structural domains to develop structural orientations for variogram development as well as the limiting mineralized grade shells. The process involved examining the gold bearing quartz veins and structures and determining an average azimuth and dip for each domain.

A significant crossing structure cuts across the center of the deposit striking to the northeast. This structure, called the Red Shear zone, breaks the deposit into two major structural domains (east and west) and controls the apparent strikes on both sides of the deposit. Generally, to the west of the Red Shear zone mineralized structures strike to the northeast and dip to the southeast. On the east side of the Red Shear zone, mineralized structures strike due east to slightly southeast and dip to the south and southwest.

Domains were identified by common strikes as well as proximity to the Red Shear zone. Using this analysis, four distinct structural domains were identified:

- Domain 1 – Mineralization within this domain averaged an azimuth of 76 degrees with a -80-degree dip to the southeast. This domain is mainly in the South Pit area west of the Red Shear zone.
- Domain 2 – Mineralization within this domain averaged an azimuth of 72 degrees with a -80-degree dip to the southeast. This domain adjoins the other three domains (1, 3, and 4) and is located west of the Red Shear zone as well as domains 1 and 2. It occurs in the North Pit areas.
- Domain 3 – Mineralization within this domain averaged an azimuth of 115 degrees with a -80-degree dip to the south. This domain adjoins domains 2 and 4 and is located west of the Red Shear zone. This domain is in the North Pit area only.
- Domain 4 - Mineralization within this domain averaged an azimuth of 95 degrees with a -80-degree dip to the southwest. It occurs east of the Red Shear zone and adjoins the other domains (1, 2, and 3). It is in both the North and South pits.

The resource classification used for the Laiva deposit is based on which pass generated a block grade estimate as well as the distance to the nearest neighbor (measured and indicated only). The resource classification used was:

- Measured – Blocks estimated in Pass 1 (minimum of four composites) with a maximum nearest neighbor distance as listed in Tables 14.11 through 14.14 for each domain are classified as measured.
- Indicated – Blocks estimated in Pass 2 (minimum of three composites) with a maximum nearest neighbor distance as listed in Tables 14.11 through 14.14 for each domain as indicated.
- Inferred – Blocks estimated in Pass 3 (minimum of two composites) are classified as inferred.

The gold grade populated block model was reviewed to ensure reasonableness. These checks included:

- An overall review of the estimated metal values.
- The impact of gold grade capping on the mineral resource.
- QQ plots of the block model versus the composites.
- A section-by-section comparison between the ID3 metal values and the underlying drillholes.
- A statistical comparison of the raw assay values versus the composite values versus the block values.

The overall block metal grades were visually examined to confirm that all the estimation parameters were honored and kept within the individual mineralized domains. Each of the cross sections was reviewed and the underlying drillholes were checked to determine that the original metal grade closely matched the estimated block metal grade without exceeding it. Cross-sections were examined, and assay intervals agreed with the overlying estimated block model metal grades.

Table 1.1 – Laiva Mineral Resource Estimate

Mining Method	Resource Classification	Gold Cutoff Grade (g/t)	Tonnes	Au g/t	Au Troy Ozs
Open Pit - High Grade	Measured	0.700	654,000	1.930	40,600
Open Pit - High Grade	Indicated	0.700	6,573,000	1.655	349,700
Open Pit - High Grade	***Measured + Indicated***	***0.700***	***7,227,000***	***1.680***	***390,300***
Open Pit - High Grade	Inferred	0.700	4,670,000	1.746	262,200
Open Pit - Low Grade	Measured	0.323	531,000	0.487	8,300
Open Pit - Low Grade	Indicated	0.323	6,247,000	0.488	98,000
Open Pit - Low Grade	***Measured + Indicated***	***0.323***	***6,778,000***	***0.488***	***106,300***
Open Pit - Low Grade	Inferred	0.323	4,012,000	0.488	62,900
Total Open Pit	Measured	0.323	1,185,000	1.283	48,900
Total Open Pit	Indicated	0.323	12,820,000	1.086	447,700
Total Open Pit	***Measured + Indicated***	***0.323***	***14,005,000***	***1.103***	***496,600***
Total Open Pit	Inferred	0.323	8,682,000	1.165	325,100
Underground	Measured	1.886	0	0.000	0
Underground	Indicated	1.886	26,000	3.072	2,600

(1) The effective date for this mineral resource estimate is 4 October 2021 and is reported on a 100% ownership basis. This estimate is an update using additional assays and exploration drilling as well as updated economics.

(2) Mineral Resources are calculated at a gold price of US$1,600 per troy ounce.

(3) The mineral resources presented above are global and do not include a detailed pit or underground design, only an economic pit-shell was used to determine the in-pit mineral resources. The underground mineral resources are that material outside of the in-pit mineral resources above the stated underground cutoff grade.

(4) Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.

(5) The mineral resources presented here were estimated using a block model with a block size of 5 m by 5 m by 5 m sub-blocked to a minimum block size of 2.5 m by 2.5 m by 2.5 m using ID^3 methods for grade estimation. All mineral resources are reported using an open pit gold cut-off of 0.323 g/t Au and an underground gold cut-off of 1.886 g/t Au. Open pit mineral resources above 0.700 g/t are considered high grade material while material between 0.323 g/t and 0.700 g/t are considered low grade.

(6) The mineral resources presented here were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10, 2014.

(7) Figures are rounded, and totals may not add correctly.

1.13 Mineral Reserve Estimates

Mineral reserves for the Laiva Gold Deposit have been estimated by BOYD as summarized in Table 1.2, following this page.

Mineral reserves have been estimated for the North and South pits from the undiluted block model and then diluted as described in Section 16.0 herein. Mineral reserves are the result of work described in Section 16.0.

Table 1.2 – Laiva Diluted Mineral Reserves

Mining Method	Reserve Classification	Cutoff	Tonnes	Au g/t	Au Troy Ozs
Open Pit - High Grade	Proven	0.700	678,000	1.853	40,400
Open Pit - High Grade	Probable	0.700	4,780,000	1.622	249,300
Open Pit - High Grade	**Proven + Probable**	**0.700**	**5,458,000**	**1.651**	**289,700**
Open Pit - Low Grade	Proven	0.323	547,000	0.466	8,200
Open Pit - Low Grade	Probable	0.323	4,488,000	0.468	67,500
Open Pit - Low Grade	**Proven + Probable**	**0.323**	**5,035,000**	**0.468**	**75,700**
Total Open Pit	Proven	0.323	1,225,000	1.234	48,600
Total Open Pit	Probable	0.323	9,268,000	1.063	316,800
Total Open Pit	**Proven + Probable**	**0.323**	**10,493,000**	**1.083**	**365,400**

(1) The mineral reserve estimates were prepared by Sam Shoemaker, Reg'd Member SME (who is also an independent Qualified Person), reported using the 2014 CIM Definition Standards, and have an effective date of September 29, 2021.

(2) Mineral Reserves are mined tonnes and grade delivered to a stockpile in front of the primary crusher.

(3) Mineral reserves are reported at a cut-off grade of 0.323 g/t Au.

(4) Cut-off grade assumes US$1,600/oz Au; US$2.55/oz selling costs; and uses a 82% metallurgical recovery for high grade material (0.700 g/t Au and higher) and a 80% metallurgical recovery for low grade material (0.23 to 0.700 g/t Au). The cut off-grade covers processing costs of US$11.78/t for high grade and US10.22/t for low grade, administrative (G&A) costs of $2.20/t. Ore material has been diluted by 5% with a 10% of the undiluted gold grade.

(5) Mined tonnes and grade are based on an SMU of 5.0 m x 5.0 m x 5.0 m, including additional mining losses estimated for the removal of isolated blocks (surrounded by waste).

CIM Standards on Mineral Resources and Reserves Definitions and Guidelines (1) define 'Probable Mineral Reserve' as "the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified."

The effective date of the mineral reserve estimate is September 29, 2021.

The ultimate pit limit is based on the economic Lerchs-Grossmann algorithm design on all sides, with no external factors constraining the pit. The final pit plans are presented in Section 16.0.

The mine plan developed in this report is based on Measured and Indicated resources only. There is an opportunity to upgrade some minor amounts of the inferred resource mineralization to ore classification with additional infill drilling.

1.14 Mining Methods

The Laiva Gold Deposit is composed of a single project with two mining areas: the previously developed South Pit area in the southwestern portion of the project; and the previously developed North Pit area in the northern portion of the project. These two mining areas develop the same mineral resource and are shown in Figure 1.6 below.

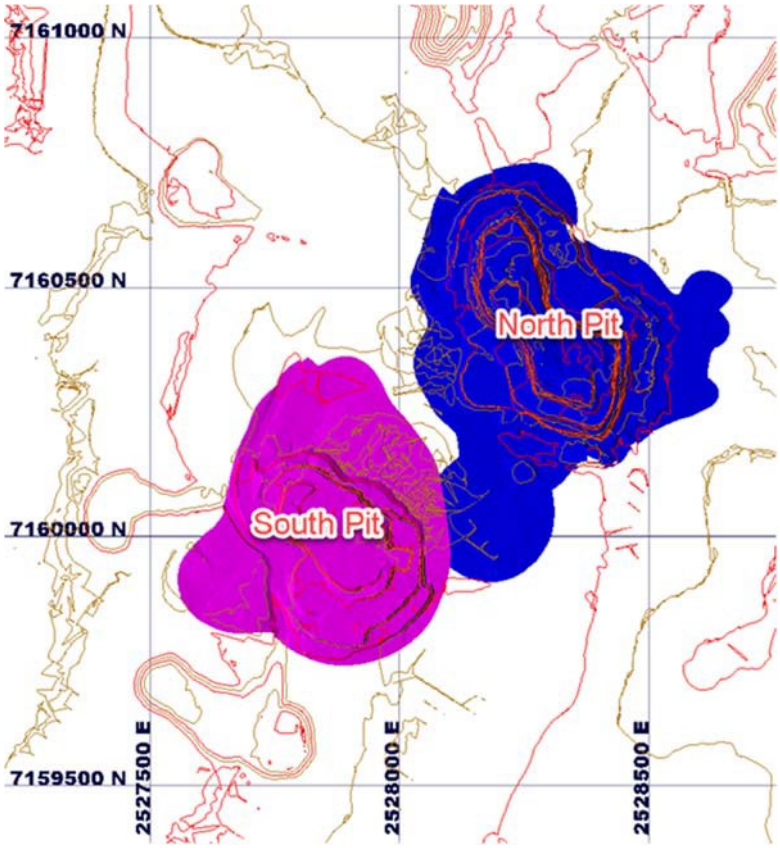

Figure 1.6 – Laiva Project Mining Areas

Pit shells were determined for the Laiva deposit from a Whittle economic pit optimization described below. Based on these initial pit optimization results, a pit shell was selected to design the phased pits that make up the two mining areas for the feasibility study.

Standard mining technology has been utilized to create an open pit at each of the two mining areas. The South Pit has approximate dimensions of 620 m south to north by 540 m east to west and a maximum depth of 200 m below current ground level. The South is composed of three mining phases. The North Pit has approximate dimensions of 840 m south to north by 600 m east to west and a maximum depth of 170 m below current ground level. The North Pit is composed of four mining phases. All of the pit shells were

selected from the Whittle open-pit optimization discussed below and were used to guide the design of two mining areas.

A mine contractor has been engaged by OTSO who will be used for most mining activities including site preparation, haul road construction and maintenance, mineralized material and bulk waste drilling and blasting, excavation and haulage of mineralized material and waste, management of waste dumps, oversize breakage, and pit dewatering. The mine contractor will provide all mining equipment. The contractor will provide operator training, supervision, mine consumables, and maintenance facilities for contractor's operations. Mine maintenance personnel will be supplied by the contractor for the contractor's fleet. OTSO will provide mine technical services including blast design, blasthole layout, ore grade control, mine planning, and blasthole sampling. Orica will provide explosives storage on-site. Explosives, blasting agents, fuel and other consumables will be provided by established suppliers.

The Laiva resource model uses a fixed 5 m X 5 m X 5 m parent block size (sub-blocked to 2.5 m X 2.5 m X 2.5 m) which is considered the smallest selective mining unit (SMU) for the deposit and is suitable for the planned mining equipment fleet. Based on this selected SMU size, no additional waste dilution other than internal included waste was deemed appropriate for the Resource update, as ore zones are gradational defined by cut-off grade boundaries.

For the mineral reserve reporting, a fixed tonnage dilution of 5 percent was selected as the tonnage dilution factor. The grade of the diluting tonnes is 10 percent of the original block grade. Diluting material is drawn from nearby waste material which decreases in grade with increasing distance from the mineralized zone.

In order to design the various required phased designs for the Laiva Gold Deposit, BOYD completed new pit optimizations to determine the optimal economic pit configuration for the overall project. To accomplish this task, BOYD used the Whittle open-pit optimization software. This software uses the industry standard Lerchs-Grossmann algorithm to determine an optimal pit shape using various economic, geotechnical, and metallurgical parameters. A Whittle optimization was completed on the resource model using only measured and indicated mineral resources (inferred material is considered waste) and a final conceptual pit shell was determined. This pit shell was then used for feasibility study level detailed phased pit designs and production scheduling.

Mine planning geotechnical parameters were determined by BOYD in conjunction with historical operating practice and are shown in Table 1.3 below.

Table 1.3 – Laiva Geotechnical Parameters

Item	Value
Inter-ramp Slope (degrees)	53.0
Face Angle (degrees)	80.0
Safety Bench (m)	5.78
Benching	Double
Bench Height (m)	5.0

Initial feasibility study level operating costs were calculated and used to provide the economic basis for the various Whittle pit optimization runs that were completed. All runs only included measured and indicated mineral resource material. Inferred mineral resource material is considered to be waste and was treated as such in all aspects of the feasibility study.

Different mining phases were designed in accordance with the recommended bench configurations provided as tabulated above in Table 1.3. Double benching of 5 m production benches was determined to be feasible in all geologic units. Catch or safety benches with a width of 5.78 m are used in all designed phases. These safety benches are applied on every other bench (10 m vertically). Ore mining is planned on 5 m benches while waste mining is planned on 10 m double height benches.

Two-way haul roads, 26 m wide at a 10% grade, were designed in most cases where higher traffic may require extra width for safe and efficient passing of trucks. To maximize ore recovery at depth, the final benches of each pit floor were designed with single-lane access (13 m width with a 12% grade). Safety berms were designed in accordance with the recommendations provided by BOYD.

The Laiva South Pit consists of three phases. Phase 1 is the starter pit while phases 2, and 3 expand the starter pit. Table 1.4, following this page, describes the in-pit diluted mineral reserves for South Pit while Figure 1.7, following this page, shows the individual phase layouts.

Table 1.4 – Laiva South Pit Diluted Mineral Resources

Class	Material	Cutoff	Au (g/t)	Tonnes	Au (t.ozs)
Measured	High-Grade	0.700	2.085	371,000	24,900
Indicated	High-Grade	0.700	1.754	2,172,000	122,500
Total M+I	***High-Grade***	***0.700***	***1.802***	***2,543,000***	***147,400***
Measured	Low-Grade	0.323	0.474	229,000	3,500
Indicated	Low-Grade	0.323	0.474	1,644,000	25,100
Total M+I	***Low-Grade***	***0.323***	***0.474***	***1,873,000***	***28,600***
Measured	All	0.323	1.470	600,000	28,400
Indicated	All	0.323	1.203	3,816,000	147,600
Total M+I	***All***	***0.323***	***1.239***	***4,416,000***	***176,000***
Waste	All	---	---	40,944,000	---
Total Material Tonnes		---	---	45,360,000	---
Stripping Ratio		---	---	9.27	---

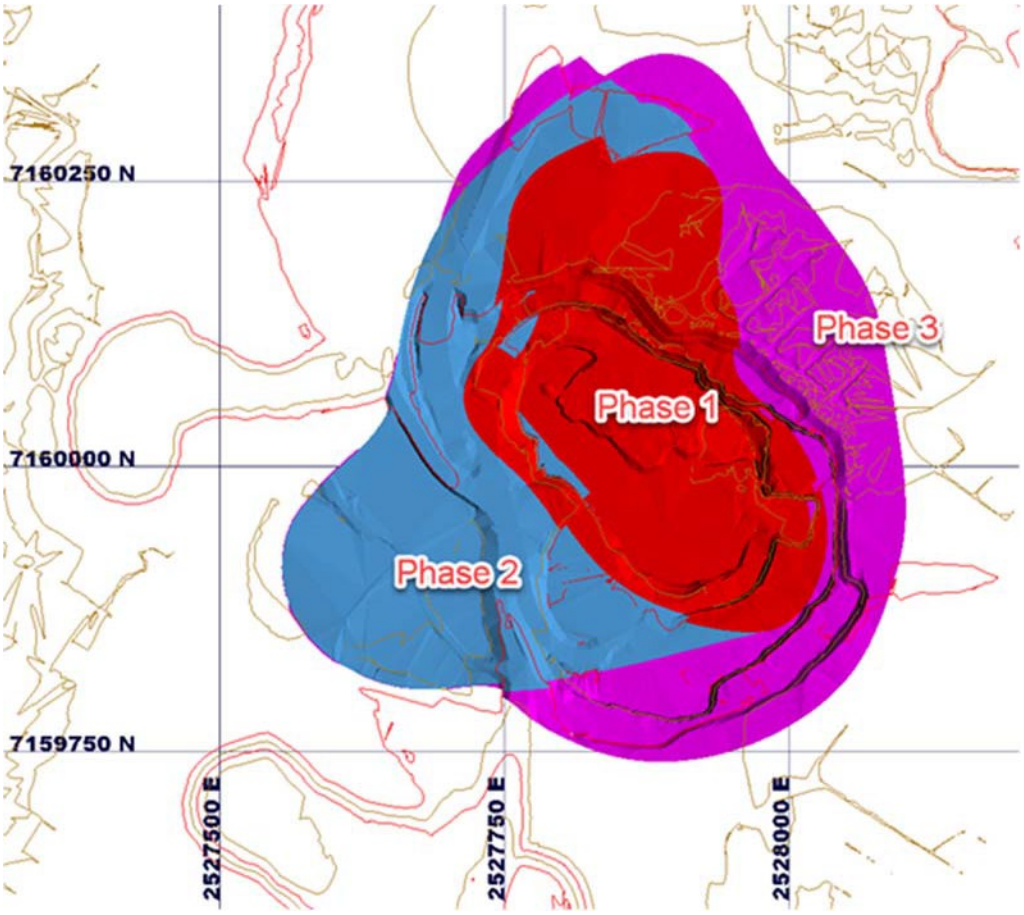

Figure 1.7 – Laiva South Pit Phased Design

The North pit consists of four phases. Phase 1 is the starter pit with Phases 2 and 3 being expansions of that phase. Phase 4 is developed below Phase 3 at the end of the mine life. Table 1.5 below describes the in-pit mineral resources for the North Pit while Figure 1.8 below, shows the individual phase layouts.

Table 1.5 – Laiva North Pit Diluted Mineral Resources

Class	Material	Cutoff	Au (g/t)	Tonnes	Au (t.ozs)
Measured	High-Grade	0.700	1.570	308,000	15,500
Indicated	High-Grade	0.700	1.513	2,607,000	126,800
Total M+I	***High-Grade***	***0.700***	***1.519***	***2,915,000***	***142,300***
Measured	Low-Grade	0.323	0.459	318,000	4,700
Indicated	Low-Grade	0.323	0.465	2,843,000	42,500
Total M+I	***Low-Grade***	***0.323***	***0.464***	***3,161,000***	***47,200***
Measured	All	0.323	1.006	626,000	20,200
Indicated	All	0.323	0.966	5,450,000	169,300
Total M+I	***All***	***0.323***	***0.970***	***6,076,000***	***189,500***
Waste	All	---	---	41,960,000	---
Total Material Tonnes		---	---	48,036,000	---
Stripping Ratio		---	---	6.91	---

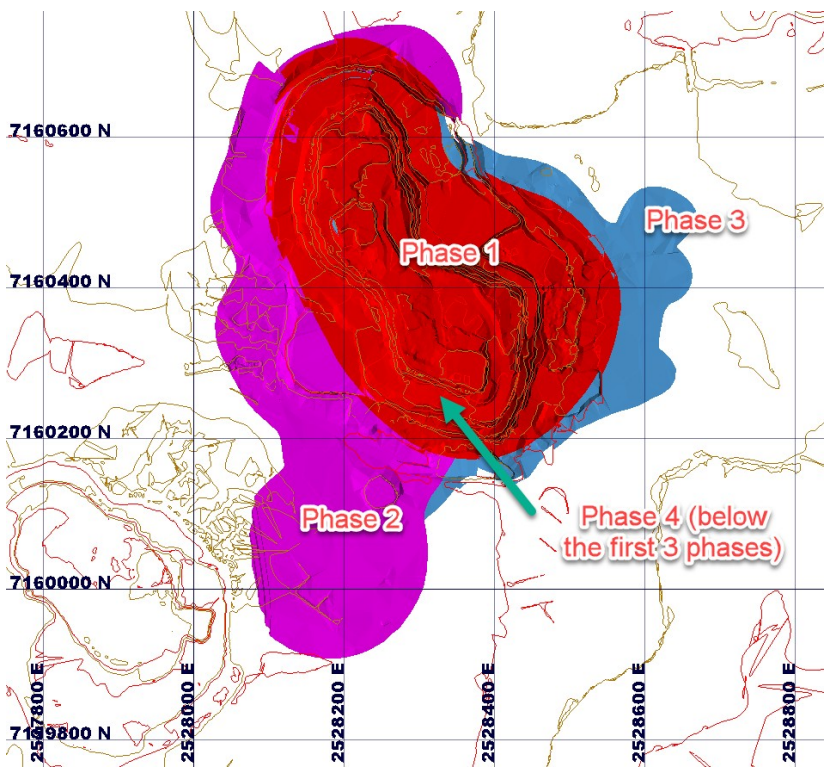

Figure 1.8 – Laiva North Pit Phased Design

Table 1.6 – Laiva LOM Annual Mine Production Schedule from Proven and Probable Reserves

Year	2021	2022	2023	2024	2025	2026	Totals
Diluted High-Grade Ore Production							
Tonnes	404,000	1,860,000	953,000	287,000	988,000	967,000	**5,459,000**
Gold Grade (g/t)	1.641	1.539	1.535	1.732	1.763	1.847	**1.651**
Contained Gold (troy ounces)	21,300	92,100	47,000	16,000	56,000	57,400	**289,800**
Diluted Low-Grade Ore Production							
Tonnes	377,000	1,906,000	1,012,000	186,000	734,000	819,000	**5,034,000**
Gold Grade (g/t)	0.462	0.465	0.465	0.477	0.475	0.473	**0.468**
Contained Gold (troy ounces)	5,600	28,500	15,100	2,900	11,200	12,400	**75,700**
Diluted Total Ore Production Tonnes							
Tonnes	781,000	3,766,000	1,965,000	473,000	1,722,000	1,786,000	**10,493,000**
Gold Grade (g/t)	1.072	0.996	0.984	1.239	1.214	1.217	**1.083**
Contained Gold (troy ounces)	26,900	120,600	62,100	18,900	67,200	69,800	**365,500**
Waste Tonnes	7,876,000	22,604,000	19,682,000	12,004,000	12,435,000	8,260,000	**82,861,000**
Total Tonnes	8,657,000	26,370,000	21,647,000	12,477,000	14,157,000	10,046,000	**93,354,000**
Stripping Ratio	10.08	6.00	10.02	25.38	7.22	4.62	**7.90**

As previously noted, all reasonable measures to minimize mining dilution are critical to the success of the LGP. Resource development drilling as indicated in previous sections is adequate in BOYD's opinion for production of the resource estimate set forth in Section 14.0. The steeply dipping sheeted vein swarm style of mineralization in mineralized zones however, requires higher definition for effective mining with minimal mining dilution.

It should be clearly understood that OTSO is responsible for directing mining activities, including:

- Mine Design
- Production Sequencing
- Definition/Grade Control Drilling
- Detailed Mapping
- Ore And Waste Blasting Design
- Grade Control at The Face
- Truck Dispatch When Hauling Ore
- Regular Survey of Excavated Volumes
- Reconciliation of Resource Estimates to Actual Material Mined

It should be clear that the mining contractor is engaged to move bcm/tonnes and is not responsible for mine management. Previous operators were unsuccessful in large measure due to too much reliance on the mining contractor for the tasks enumerated above.

1.15 Recovery Methods

The Laiva Gold Mine Process facilities were constructed in 2011 for a reported 233 million Euro constructed cost. It operated for a total of seventeen months; with few problems associated with equipment operation or flow sheet design. Essentially, the same process facilities and flow sheet will be utilized for future operations with only minor modifications to enhance productivity and improve key process parameter measurement and recording systems.

The mineral processing plant is located approximately 1 km northeast from the South and North open pits on the LGP site. Processing consists of three primary circuit components, including: crushing and grinding, pre-concentration by flash flotation and gravity concentration, followed by High and Low grade CIL leach circuits. Design throughput rate is 250 tonnes per hour (6,000 mtpd/2 M mtpy).

Ore from active pits will be transported to a ROM stockpile area to be stacked in accordance with estimated grade and rock type to accommodate optimized blending of plant feed. Stockpiled ore in appropriate proportionate blend will be delivered to the primary feed hopper after segregation of oversize. BOYD recommends a Caterpillar 966K or equal to segregate oversize, perform blending, and supply the primary feed hopper.

The primary feed hopper with a capacity of approximately 80 cubic meters, having grizzly bars spaced at nominally 1 m X 1 m feeds directly onto a Metso VF661-2V vibrating feeder. This unit feeds directly into a Nordberg/Metso C160 jaw crusher.

The jaw crusher typically has a closed side setting of 200 mm. This unit has the capacity to deliver approximately 800 mtph at a closed side setting of 200 mm, which is more than ample for feed to the process plant.

Crusher discharge is via a chute to a Metso variable speed VMOU 25/10-5060/B vibrating feeder for belt loading.

Primary crushed ore is delivered by a 1,200 mm wide inclined conveyor to a single point discharge under a covered semi-live stockpile with an estimated live capacity of 7,500 to 10,000 metric tonnes.

Beneath the stockpile are four discreet feed ports spaced at approximately 15 meters apart. Each feed port is equipped with a Metso VMOU 25/10-5060/B vibrating feeder independently controlled, which feeds onto a single belt in the tunnel beneath the stockpile for delivery to the primary autogenous grinding mill.

A mechanical scale on the mill feed conveyor controls the feed rate to the Outotec Autogenous Mill (AG). The installed power is 3,000 Kw. The inside diameter of the AG-Mill is 6.5 m and the inside length is 8.5 m.

Water is added to the AG-Mill in order to reach 65 weight-% solids in the finest (-8 mm) discharge stream.

A double drum trommel screen after the AG -Mill separates the discharge into three sized products:

1. Fines -8 mm
2. Chips 8 - 50 mm
3. Pebbles 50 - 150 mm

The fines material, together with the discharge from the Pebble Mill, report to the classification sieve or can be diverted directly to the Pebble Mill feed if required.

The two largest product sizes of the AG-Mill can circulate back to the AG-Mill for regrind if needed through the Pebble Crusher System. Usually the coarsest material (pebbles) from the AG are used as grinding media in the Pebble Mill when pebbles are needed, and the power intake of the Pebble Mill enables it. The excess pebbles and chips (8-150 mm) are transported to the pebble crush system as required by conveyor belt.

To increase the Mill throughput, two cone crushers in series re-crush the excess critical size (8-150 mm) from the AG-Mill discharge material. The feed to the cone crushers is designed to be 87 tph (35% of AG-Mill feed). The critical size is re-crushed down to -20 mm before returned back to primary milling.

The conveyor belt under the primary crusher transports the crushed product to a double deck screen (40 and 9 mm), on top of the secondary cone crusher. The screen separates three products:

1. Fines –9 mm
2. Mid-size 9 - 40 mm
3. Oversize +40 mm

Mid-size product is crushed in the secondary cone crusher, Fines (-9 mm and oversize (+40 mm are bypassed.

If the primary cone crusher cannot be used, the secondary cone crusher still can operate if the oversized material, which is too large for the secondary crusher to handle, is screened off.

Underflows from the primary cyclone cluster are ground in closed circuit in the Pebble Mill. The grinding media are 50-150 mm sizes pebbles from the AG-Mill. The power draw from the Pebble Mill is controlled by the pebble material flow from the AG-Mill. The installed power is 3,000 kW. The inside diameter of the Pebble-Mill is 6.5 m and the inside length is 8.5 m.

Fines discharged from the AG-Mill and Pebble mill are pumped to a two-deck classifying sieve with 3 mm opening on the upper deck and no screen included on the bottom deck. Material +3 mm reports back to the AG-Mill and -3 mm material is pumped to the primary classifying cyclone and on to Flash Flotation and gravity separator circuits.

A cyclone battery, including a total of 12 units, is used to classify particles ready for low grade leaching. The cyclone underflow with the coarse particles reports back to the Pebble Mill for regrind. The cyclone overflow is the feed to the low grade leaching circuit. The targeted particle size of cyclone overflow currently is P_{80} ~100 µm.

From the sieve, the undersized flows by gravity to a sump and is pumped to the flash flotation circuit. The Flash Flotation unit (Metso Unit Cell RCS 30 produces high grade concentrate of gold-bearing sulphide minerals including arsenopyrite and pyrite. Copper sulphate (CuSO4 is added to activate arsenopyrite and PAX (Potassium Amyl Xanthate and Aerophine 3418A are used as collectors. The produced concentrate is directed to the concentrate regrind circuit for onward direction to the high grade CIL circuit.

The tailings from flash flotation are circulated back to the primary cyclone cluster for re-classification. Underflow from the primary cyclone cluster is returned to the Pebble Mill for further grinding.

Overflow from the primary cyclone bank is forwarded to gravity concentration. The gravity concentration circuit feed is split by a cyclone circuit with clone underflow fed to spirals (6 ea Multotec triple starts circuit and the cyclone overflow fed to the Knelson concentrator (Knelson CVD64-D40.

The target split particle size for the gravity concentration circuit is 105 µm. The particle size fed to the spiral circuit is approximately P_{80} = 68 µm. The particle sizefed to the Knelson concentrator is approximately P_{80} = 150 µm.

The two gravity concentrates are directed to the concentrate regrind circuit for additional grinding before subsequent high grade CIL.

The target particle size for the high grade concentrate regrind circuit is P_{80} = 45 µm.

The cyanide leach facility incorporates standard Carbon in Leach methodology. The average nominal annual mill throughput of 197 mtph was utilized for mass balance purposes. It has been assumed that the plant will operate 8,000 hours per year (assuming 90% overall availability). The feed to the cyanidation plant is comprised of two streams, including concentrate from the gravity circuits and flash flotation to high grade CIL and the tailings from each to low grade CIL.

Design of the low grade leach circuit, carbon adsorption, elution, and gold recovery circuit, with respect to retention time and gold production, is based on a nominal throughput of 187 mtph containing 0.434 g /t Au, 0.004 % As, and 0.022% S.

Design of the high grade leach circuit capacity, carbon adsorption, elution, and gold recovery circuit, as well as retention time, gold production, etc., is based on a nominal throughput of 10 tonnes per hour containing 51.4 g/t Au, 22.7 g/t Ag, 1.2 % As, and 5.9% S.

The upgrade concentrate regrind circuit will have the capacity to produce up to 13 mtph of P_{80} = 45 µm product. The increased high grade grinding capacity will ensure that all gravity and flotation concentrates will be reground and report to the high grade leach circuit. During previous operations, the undersized regrind mill resulted in a portion of the

concentrates reporting to the low grade leaching circuit, thereby substantially lowering overall gold recovery.

Proposed Modifications to Existing Process Plant

OTSO has rented primary and secondary diesel-powered crushing units to provide finer feed (nominal 25 mm) to the AG-Mill, blended with 200 mm feed produced by the existing C160 jaw crusher. Though the ratio may vary, roughly 20% of the feed will be 200 mm particles and 80% will be 25 mm particles. The coarse fraction is intended to serve as grinding media in the AG-Mill.

While the rented crusher units are included in this Feasibility analysis, BOYD strongly recommends replacing these units with a permanent electrically powered secondary cone crusher coupled with appropriate split of feed diversion from the existing primary crusher (Norberg/Metso C160), which has more than ample capacity at any closed side setting to supply total feed to the process plant.

That said, the rented diesel powered units engaged by OTSO include a track-mounted jaw crusher (Metso LOKOTRAK LT 106) and a track mounted cone crusher (METSO LOKOTRACK LT 2OOHP) to bridge the essential requirement for blended particle feed size to the AG-Mill, which is critical to achieve maximum plant throughput.

BOYD recommends several minor modifications to the flash flotation and gravity concentration circuit, which include only minor piping changes and installation of one cyclone cluster for particle size separation to narrow the range of particle size processed by the Knelson centrifugal concentrator and the spirals. Also, OTSO is completing installation of a previously existing, but uninstalled Koppers KW-200 tower mill for regrind of high grade flash float and gravity concentrates prior to high grade leaching.

The objective of these minor modifications is to:

1. Expose all mill feed to flash flotation and to gravity concentration.
2. Divide flows to the Knelson concentrator and spirals to narrow the particle size seen by each and to apply the Knelson centrifugal concentrator to the finer fraction and spirals to the coarser fraction.
3. Maximize concentrate feed to the high grade leach circuit by adjusting the mass pull from flash flotation and the gravity concentrators for regrind in the tower mill.

In BOYD's opinion, the foregoing minor modifications should result in maximization of overall recovery and can easily be adjusted to hone in on optimum results and changing feed conditions.

1.16 Project Infrastructure

Project Infrastructure in this instance can generally be construed to include site access, power supply, water supply, waste dumps, TSF facilities, and site drainage. As an existing project including mine, mill, and infrastructure which was in production as recently as early 2018, all infrastructure is in place and remains in a near-new state.

Ample grid power is available to the site. An overhead line supplies 3-phase power at 120 kv ,50 hz to a 20,000 kva substation located on site. Grid service is generally highly reliable notwithstanding moderately harsh winter weather in the north of Finland. Power is moderately expensive at approximately US $0.10 per kwh inclusive of demand and all other charges currently. No significant on-site standby generation capacity is present at Laiva.

Water is plentiful in northern Finland. Process makeup water not otherwise available from the freshwater collection and treatment system from tailings water recycle, pit pumping, and site run-off is expected to be sufficient to provide process water. Typical past consumption has been about 35,000 cu meters (9.25 million us gallons per month. Under current operating conditions, purchased makeup water is not expected to be required except in periods of exceptional drought (rare in this region).

Treated potable water is supplied by the local municipality via pipeline to the Laiva site. Cost for treated potable municipal water is relatively high at €1.02 per cubic meter delivered or US$0.005 per US gallon. Only site domestic water consumption at nominally 1,500 cu m/year is considered as a component of infrastructure cost in this Technical Report.

Mining waste is divided into potentially acid generating waste (PAG) and benign, non-acid generating waste (NAG). The waste rock is differentiated based on sulfide content, with waste containing higher levels of sulfide material reporting to the PAG dump area and waste with lower content reporting to the NAG waste storage area.

The PAG waste must be deposited on an impermeable underliner of HDPE or locally available processed peat. The underliner is placed on a prepared surface of crushed gravel topped with sand and protected from above by a layer of sand overlain by crushed gravel. Drainage water from the PAG waste is collected and pumped to a collection pond where it is mixed with other site drainage before being pumped to the sea.

Similar to the mine waste dumps, the tailings storage facilities (TSF) are divided between high sulfide and low sulfide TSFs. High sulfide tailings from the High Grade leach circuit are delivered to HDPE lined cells. Tailings from the High Grade CIL circuit also contain elevated levels of arsenic resulting from the concentrated presence of arsenopyrite.

Following exit from the High Grade CIL circuit, the tailings pass through the Inco air/SO2 for cyanide destruction, before being pumped to the isolation cell. Each cell has a nominal capacity of 117,000 dry metric tons of High Grade tailings. A second cell was completed shortly before shutdown of operations by the previous operator. The remaining life of HG tailings cell #2 (already constructed) is estimated at approximately 30 months at planned operating levels.

The project had previously permitted a total of five HG tailings cells. Accordingly, a third HG tailings cell will need to be constructed for use beginning mid-year 2023, followed by a fourth cell estimated for use beginning early in 2024, and finally a fifth cell for use beginning mid-year 2025 based on the current minable reserves. Cost for this work is capitalized as Sustaining Capital. Production beyond that will require amendment of the permit for HG tailings deposition.

The relatively benign Low Grade tailings which account for the bulk of the tailings disposal (240 mtph LG vs 10 mtph HG) are pumped via a 7.39 km pipeline to a dewatering plant before being deposited as paste at approximately 55% solids. The elevation gain from the mill to the paste plant is approximately 60 meters. Low Grade tailings are pumped at approximately 35% solids through a 300 mm id steel pipeline. Pumping over the entire distance is accomplished by three Metso VASA HD507-150 pumps arranged in series for a total input of 396 kW. The pumping system has 100% on-line spare redundancy, which can be instantly activated with automatic valves via a “Y” arrangement, see Figure 1.9, following this page.

Figure 1.9 – Paste Tailings System

All site drainage, including site run-off, run-off from PAG waste, mine dewatering, paste thickener overflow, and drainage from deposited paste is collected and pumped to a temporary storage basin with a maximum capacity of approximately 700,000 cubic meters. Excess combined clear water effluent from the storage basin not recycled as process water is pumped through a 17.42 km pipeline to an offshore submerged discharge point. Maximum discharge rate with existing pumps and pipeline is approximately 240 cubic meters per hour (1,056 usgpm).

1.17 Market Studies and Contracts

Gold and minor silver in the form of doré bullion is expected to be delivered for sale to a recognized refinery for further processing with the resulting gold and silver being credited to OTSO's metal account for on-sale. As BOYD is not aware of any hedging agreements pertinent to OTSO, it is assumed that sales will be made at then current spot prices for gold and silver sold through the refiner.

Key provisions of the refining contract still under negotiation and as advised by OTSO, which are contained in Section 22.0, Economic Analysis, in this Technical Report, are as follows:

Gold Dore Bullion

Quantity: Min. 30 kg per shipment

Quality: Min. 70% Au, balance non-deleterious

Rates: In USD per Net Doré ounce received

Gold Contents	Min. 70% Au	**Min. 80% Au**	Min. 90% Au
Treatment Charges (T/C)*	$0.34	**$0.32**	$0.28
Returnable Gold	99.90%	**99.92%**	99.95%
Returnable Silver**	99.00%		
OUTTURNS Gold/Silver***	Five business days after delivery at PAMP		Four business days after delivery at PAMP.

*Minimum T/C USD 450.00 per lot.
**Minimum 1% Ag to apply for recovery.
***or upon processing start with Customer's appointed representative.

Please note such outturn timing will only apply under normal processing capacity circumstances.

1.18 Environmental Studies, Permitting and Social or Community Impact

The Laiva Mine and process plant were fully permitted for operation and were fully operated for the period February 2012 through December 2013 and the process plant only continued to operate, processing low-grade stockpile material until March 2014. Since then, the operation was on active care and maintenance until restarted by FTR in August 2018 (processing November 2018). The site was placed back to care and maintenance in April 2019. All environmental permits are active and have legal validity. No permit requirements existing at this stage.

In 2018, several studies on the LG TSF area structures were completed. These studies included stability assessment for the existing walls, tailings quality assessment, and danger risk assessment. Based on the information received from these studies, the LG TSF area structures have been re-assessed by Geobotnia and Knight Piesold in 2021. The latest studies have included new design for perimeter dams, tailings deposition model and detailed discharge plan for the first 12 months of production. All works

recommended in these studies are currently being completed. Designs have been conditionally approved by applicable regulatory authorities for continued operation.

All permits are valid, and no permit related issues are existing. Permit compliance has been evaluated annually by related authorities, with the most recent evaluation in October 2021.

Environmental monitoring programs have been updated for the 2021 restart program. Final approval of the programs is expected momentarily and is expected to be pro forma, and as such are incorporated in this Technical Report. Environmental monitoring is currently being performed in accordance with the requirements during mine production. These monitoring programs were approved in 2018 with no change. Monitoring includes internal water quality monitoring, monitoring of discharge water quality, monitoring of surrounding creeks and rivers, groundwater monitoring, dust and air quality monitoring, monitoring for noise and vibration, monitoring of mine waste (waste rock and tailings), biological monitoring including sediment load, and monitoring of discharge to the sea including fishery impacts.

During both operational and care and maintenance periods, site environmental monitoring was completed based on separate monitoring programs for each requirement. Monitoring data is annually reported to regulatory authorities by an external consultant.

Monitoring requirements include monitoring of environment related structures (e.g., pipelines and pump stations) and dam safety monitoring. Monitoring performance is annually reported to relevant regulatory authorities by the then current operator.

All mining activities are located within the mine lease area. The area has existing and a valid “town plan” in place approvals. The closest communities are located over 5 km away from the mine site. Due to current major forestry activities in the surrounding area, dust is limited to mine site areas, thereby having no impact on forestry designated zones or local communities (verified by continuous monitoring under previous and current operator).

Excess site water, net of process recycle is discharged via pipeline to the Bothnia Sea, as described in Section 18.0. The water quality for necessary discharge to the sea is also monitored monthly. Given the quality of the water discharged to the sea from operations to date, no issues from future operations can be reasonably predicted. The relatively minor adjustments to the process circuit discussed previously may actually reduce the volume of discharge and attendant water quality.

1.19 Capital and Operating Costs

As previously described, the LGP is fully constructed with an operational mine, process plant, and all necessary infrastructure. The project commenced operation in 2012 and operated sporadically through early 2019 under two previous operators who were unsuccessful largely due to ineffective mining practices that lead to excessive mine dilution coupled with undercapitalization in both instances.

Through a business arrangement with FTR's (the immediate predecessor in interest) lender, OTSO has taken over the LGP effective in December 2020. Since that time, OTSO has worked to perform geophysical surveys, additional resource development drilling, engage a mining contractor, clean up and repair mill components and infrastructure in preparation of a re-start of operations ramping up from September 2021. The collective cost, including all of the foregoing plus negotiating agreements with lenders and trade creditors left by FTR, totals US $8.812 million all-in preproduction capital, which OTSO has elected to treat as sunk cost for purposes of the forward-looking economic analysis of the LGP.

The basis for the capital and operating costs summarized in Table 1.7 below, and Table 1.8, following this page, considers mining and processing all current Proven and Probable reserves as determined herein, over a project life (LOM) of 5.3 years based on annual process capacity of the existing plant. As the deposit remains open in all directions and resource development drilling is planned to continue, it is expected that project life will be increased by some amount which is not currently determinate. The cost of the additional resource development will be considered deferred capital to be amortized over the additional gold ounces discovered. Accordingly, this potential additional resource development is not included in the Technical Report.

Table 1.7 – Summary of Estimated Capital Costs (US$)

Item	Pre -Production	Sustaining	Total
Mine		400,000	400,000
Processing		8,085,800	8,085,800
Infra. & Indir.		7,680,000	7,680,000
Contingency		1,131,600	1,131,600
Total	**8,812,000***	**17,297,400**	**26,109,400**

* Not considered in economic analysis.

Table 1.8 – Summary of Estimated Operating Costs (US$)

Section	Cost (US$) ($/t Ore)
Mining	18.39
Processing	11.02
Infrastructure	0.49
Indirects	0.20
Tech Services	0.47
Safety	0.19
Environmental	0.56
G&A	1.73
Total Operating Cost	33.05

All-in Sustaining Costs (AISC)

A summary of estimated all-in operating costs is presented below in Table 1.9 below.

Table 1.9 – Estimated All-in Operating Cost Summary (US$)

	2021	2022	2023	2024	2025	2026	Totals
Mining Cost Total	13,440,388	42,944,466	33,575,983	18,010,231	22,078,643	16,401,672	**146,451,384**
Processing Cost Total	3,362,018	16,852,434	16,822,849	16,905,970	16,905,970	16,905,970	**87,755,212**
Infrastructure and Indirect Total	342,210	1,026,630	1,026,630	1,026,630	1,026,630	1,026,630	**5,475,360**
Site Services Total	298,548	1,886,216	1,886,216	1,886,216	1,886,216	1,886,216	**9,729,633**
G&A Cost Total	656,676	2,626,704	2,626,704	2,626,704	2,626,704	2,626,704	**13,790,196**
Other Cost Total	35,965	3,115,909	3,170,985	1,529,353	2,390,055	3,390,936	**13,633,203**
Total All-in Operating Cost	**18,135,806**	**68,452,361**	**59,109,368**	**41,985,106**	**46,914,219**	**42,238,129**	**276,834,988**
All-in Operating Cost $/mt Ore	23.22	18.18	30.08	88.87	27.25	23.65	**34.76**
All-in Operating Cost $/Troy Oz Au Equ prod.	1,210.24	957.10	935.54	1,316.51	793.49	695.44	**918.38**

Table 1.10 – Estimated Capex and All-in Sustaining Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
Capex Total	2,653,600	9,531,458	256,800	4,341,958	256,800	256,800	**17,297,416**
Opex Total	18,135,806	68,452,361	59,109,368	41,985,106	46,914,219	42,238,129	**276,834,988**
Total AISC	**21,338,489**	**77,983,819**	**59,366,168**	**46,327,064**	**47,171,019**	**42,494,929**	**294,681,487**
AISC Cost $/mt Ore	69.94	50.99	38.89	30.20	30.75	27.70	**37.00**
AISC Cost $/Troy Oz Au Equ produced	1,422.57	1,089.27	938.69	1,451.24	797.06	698.99	**976.63**

1.20 Economic Analysis

As previously noted, the LGP is a fully developed with existing pits, process plant, and all infrastructure. Accordingly, this Feasibility Study Technical Report does not contain preliminary facilities design work as is more typical. Additionally, the LGP has reached steady state operation under previous ownership/management, from which key operating parameters and costs can be drawn.

Updated mine design based on additional resource drilling and estimation as well as revised detailed mining plans have been developed. A mining contractor to perform all mining functions on a unit price basis has been engaged by OTSO. Based on the foregoing, BOYD has estimated from first principles mine capital and operating costs based on the contractor unit price schedule.

Process equipment remains generally unchanged with the addition of a parallel rental crusher line to produce finer feed to the primary milling circuit plus other minor flow sheet adjustments for optimized performance. The rented crusher units are under fixed monthly contract rates, which have been incorporated into the plant operating costs. Based on results of recent metallurgical testing, as discussed in Section 13.0 herein, BOYD has estimated from first principles plant capital and operating costs.

Infrastructure facilities are unchanged from historic. BOYD has estimated from first principles project infrastructure and indirect capital and operating costs based on historic figures updated to current pricing.

G&A and other costs have been provided to BOYD by OTSO. BOYD has reviewed these costs in detail and determined that they are reasonable for use in this Technical Report.

Pre-production capital for each of the following tasks and amounts have been provided by OTSO, as booked. BOYD has reviewed these costs and has no reason to doubt the accuracy thereof.

Economic Assumptions

Gold Price

Gold price was taken at a fixed rate of $1,600/tr oz, conservatively supported by the 200-day trailing average London PM Fix quoted by Kitco at US $1,750 per tr oz Au, as well as the three-year volume weighted trailing average gold price also at approximately US $1,750 as quoted by Kitco based on London PM Fix.

Gold price was held constant at $1,600 per tr oz Au for the current LOM of 5.3 years.

Exchange Rate and Inflation

The USD to Euro exchange in September 2021 was approximately US $1.18 to 1.00 € and has been trending lower since that time. That said, most forecasts expect the ratio to continue in a relatively narrow trading range at least through 2023. BOYD examined the difference in the pre-tax NPV at 5% discount rate over the range of 1.15:1 and 1.21:1 exchange rate and determined that a variance of approximately 2% to the pre-tax NPV

at 5% occurs. Given the relative insensitivity to exchange rate within forecast parameters, BOYD has elected to fix the exchange rate at US $1.18 to 1.00 € for economic analysis in this Technical Report.

Finnish inflation is expected to be relatively modest at “around 1.2% in 2022 and 1.5% in 2023”, according to trading economics.com. However, even modest inflation generally manifests as higher gold prices at fixed currency exchange rates, so BOYD has included no inflation given its decision to maintain a fixed gold price for the duration of the current LOM.

Corporate Taxation

Finnish corporate tax is currently 20% of adjusted income. Permitted adjustments to income include standard depreciation, tax loss carried forward (subject to certain expiry dates), and amortization of capitalized advance expenses. There is currently no standard depletion allowance in Finland.

BOYD has included both pre and post-tax calculations of key economic parameters in its analysis.

Royalties

OTSO is required to pay 0.15% of gold sales revenue to the holders of surface rights for occupation and use of surface not directly held by OTSO. In addition, OTSO is required to pay a 2.5% NSR to PFL Raahe Holdings LP as part of a partial settlement of a previous FTR debt obligation.

Mined ore will be deposited on the ROM ore pad and stockpiled in accordance with grade and ore type for plant feed blending. Processing rate and schedule are independent of the mining schedule. The process production schedule is shown in Table 1.11, following this page.

Process Production Schedule

The existing LGP process plant is limited to approximately 1.5 - 2.0 million metric tonnes per year. It’s based on primary grinding capacity as supported by test work and historical throughput that achieved output particle size suitable for reasonable recovery for down stream process steps in aggregate. Adoption of additional crushing equipment (rented and included in Opex) to reduce the 80% of the particles size to P_{80} 25 mm for feed to the existing primary grinding circuit (3,000 Kw AG-Mill) was deemed essential by BOYD to achieve this annual production target.

It may be possible to feed up to 2 million metric tonnes per year without compromising overall gold recovery based on recent test work. However, BOYD recommends the production rates shown in Table 1.11 below, based on the optimized mine plan and the need for further plant optimization during ramp up. Actual operating experience and availability of increased minable high grade as resource development continues could suggest increasing mill throughput closer to 2 million metric tonnes per year.

Table 1.11 – Mill Production Schedule

Plant Production	2021	2022	2023	2024	2025	2026	Total
Plant Feed							
High Grade							
Tonnes (mt)	305,085	1,529,269	1,382,039	286,650	988,050	966,700	**5,457,793**
Grade (g/mt)	1.659	1.579	1.551	1.732	1.763	1.847	**1.651**
Low Grade							
Tonnes (mt)	-	-	144,546	1,247,477	546,077	567,427	**2,505,527**
Grade (g/mt)	-	-	0.465	0.465	0.467	0.468	**0.466**
Total							
Tonnes (mt)	305,085	1,529,269	1,526,584	1,534,127	1,534,127	1,534,127	**7,963,320**
Grade (g/mt)	1.659	1.579	1.398	0.702	1.301	1.337	**1.278**

Table 1.12 below, summarizes the LOM cash flows for the project, while Table 1.13, following this page, presents the annual production and cash flow schedule.

Table 1.12 – LOM Cash Flow Summary (US$)

Net Operating Revenue	**472,300,097**
Less All-in Operating Costs	(275,717,047)
Operating Margin (EBITDA)	196,583,050
Less Sustaining Capital	(17,297,400)
Undisc Pre-Tax CF ex sunk	**178,736,551**
Post-Tax	
EBITDA	196,583,050
Depreciation Adjustment	(40,739,818)
Loss Carried Forward Adjustment*	(111,355,978)
Net Taxable Income	44,601,470
Tax Payable	(8,920,294)
After Tax Income (ex-sunk)	187,662,756
Undisc After Tax CF ex sunk**	**169,816,257**

* Note, this includes existing NOL expirations
** Note, sunk capital is included in NOL carried forward

Table 1.13 – LOM Production and Cash Flow Summary

Parameter	2021	2022	2023	2024	2025	2026	Total
Mining							
Ore Mined (mt)	781,042	3,765,669	1,965,280	472,415	1,721,928	1,785,762	**10,492,096**
Waste Mined (mt)	7,875,958	22,603,664	19,682,053	12,003,918	12,435,004	8,260,102	**82,860,700**
Total Mined (mt)	8,657,000	26,369,333	21,647,333	12,476,333	14,156,932	10,045,864	**93,352,796**
Strip Ratio (W:O)	10.1	6.0	10.0	25.4	7.2	4.6	**7.9**
Processing							
Ore Processed (mt)	305,085	1,529,269	1,526,584	1,534,127	1,534,127	1,534,127	**7,963,320**
Grade Processed (gr/mt)	1.659	1.579	1.398	0.702	1.301	1.337	**1.278**
Gold Recovery (%)	90.2%	90.2%	90.2%	90.2%	90.2%	90.2%	**90.2%**
Net Payable Gold (tr oz)	14,675	70,038	61,872	31,230	57,898	59,476	**295,188**
Financial							
Net Revenue (USD)	23,479,310	112,060,403	98,994,429	49,967,959	92,636,228	95,161,769	**472,300,097**
Operating Cost (USD)	18,135,025	68,384,771	59,040,583	41,951,931	46,862,374	42,164,573	**276,539,257**
Sustaining Capital (USD)	2,653,600	9,531,458	256,800	4,341,958	256,800	256,800	**17,297,416**
Undiscounted Pre-tax CF	2,141,602	34,300,755	39,898,719	3,771,108	45,667,216	52,957,152	**178,736,551**
Net Tax	-	-	-	-	-	8,920,294	**8,920,294**
Undiscounted Post-tax CF	2,141,602	34,300,755	39,898,719	3,771,108	45,667,216	44,036,858	**169,816,257**

Discount Rate and Resulting NPV Analysis

BOYD determined discounted cash flow results on a pre and post-tax basis over a range of discount rates including 5.0%, 6.2%, 8.0%, and 15.0%.

As a discount rate of 5.0% appears to be the most common rate used in NI 43-101 Technical Reports filed to date in 2021, this rate was chosen as the base case to provide a direct comparison with other recently filed Technical Reports. While most used a 5% discount rate, a limited few used an 8% discount rate so BOYD included analysis at this discount rate as well.

The calculated discount rate by the Capital Asset Pricing formula using the three month t-bill rate a/o 28 September 2021 quoted at 1.50% for the risk free rate, a market beta of 0.90 based on a basket of 86 gold equities for 2021 to date, and a risk premium of 5.30%, again based on a basket of Canadian gold equities with international operations, yielded an indicated discount rate of 6.27% based on the CAPM formula, a standard metric.

This compares very favorably with the reported weighted average cost of capital (WACC) for a basket of ten Canadian gold producers at 6.17%. Accordingly, BOYD also analyzed the NPV at a discount rate of 6.20% for a comparison combining the two.

Finally, BOYD also analyzed the NPV using a discount rate of 15.0% at the request of a large shareholder of OTSO.

The resulting pre and post-tax NPV at the discount rates noted above is shown below in Table 1.14.

Table 1.14 – Pre and Post Tax NPV

Discount Rate (%)	Pre-Tax NPV (US$)	After-Tax NPV (US$)
5.0	152,932,950	146,030,015
6.2	147,642,666	141,139,869
8.0	140,260,307	134,307,193
15.0	116,747,228	$112,467,418

Note: Both pre and post-tax values do not consider sunk cost, except in so far as sunk costs serve as an adjustment to taxable income in the post-tax case.

Sensitivity Analysis

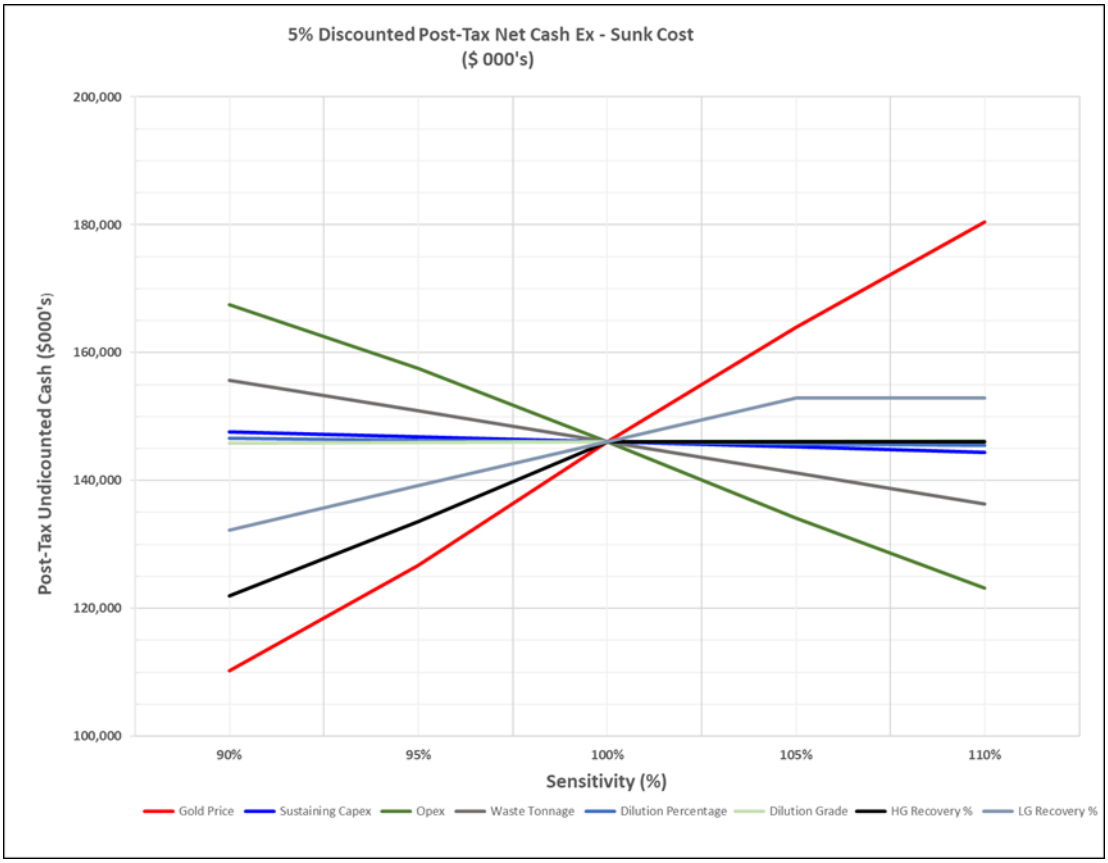

Figure 1.10 – After-Tax Sensitivity at 5% Discount Rate

5% Discounted Post -Tax Net Cash Ex-Sunk Costs ($ 000's)

Sensitivity Parameter	90%	95%	100%	105%	110%
Gold Price	110,195	126,758	146,030	163,979	180,399
Sustaining Capex	147,643	146,837	146,030	145,223	144,417
Opex	167,529	157,544	146,030	134,145	123,169
Waste Tonnage	155,658	150,887	146,030	141,173	136,317
Dilution Percentage	146,586	146,307	146,030	145,754	145,480
Dilution Grade	145,837	145,933	146,030	146,127	146,223
HG Recovery %	121,947	133,507	146,030	146,030	146,030
LG Recovery %	132,259	139,145	146,030	152,915	152,915

A graphical representation of post-tax NPV is shown for the range of discount rates examined as Figure 1.11, following this page.

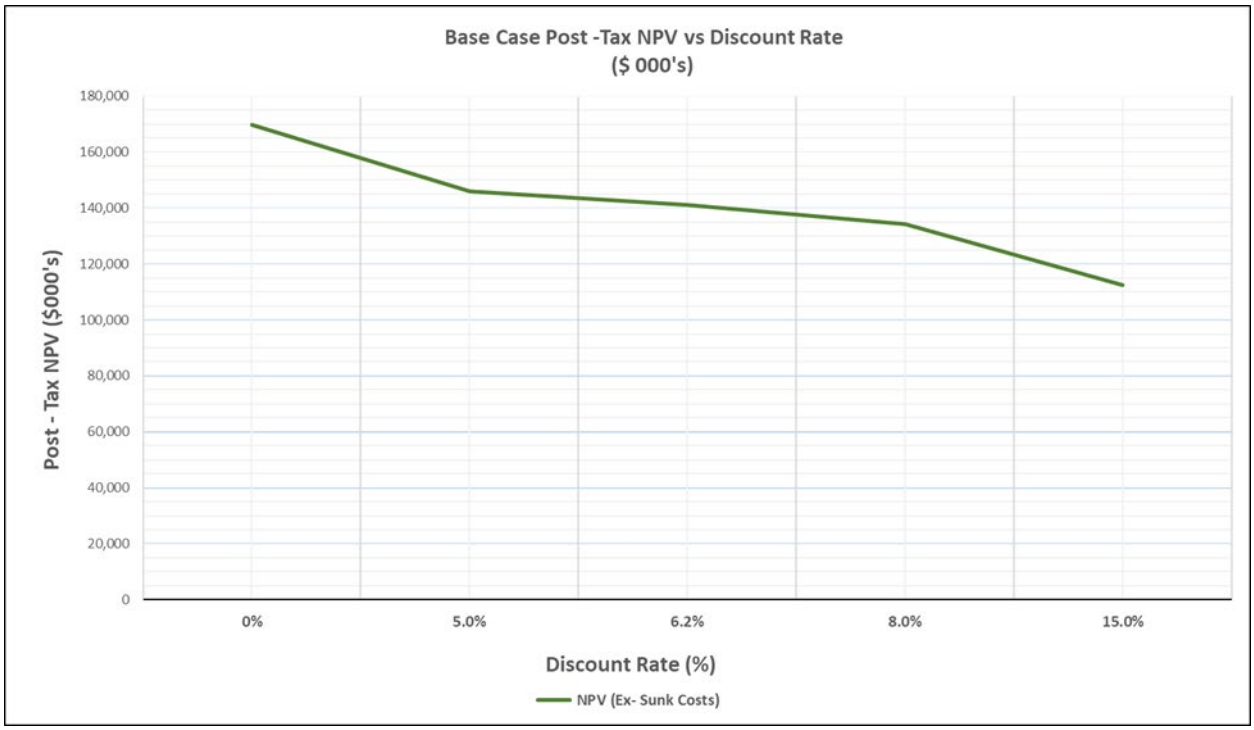

Figure 1.11 – Post-Tax NPV at Multiple Discount Rates Analyzed

Base Case Post-Tax NPV ($ 000's)					
Discount Rate (%)	**0%**	**5.0%**	**6.2%**	**8.0%**	**15.0%**
NPV (Ex- Sunk Costs)	169,816	146,030	141,140	134,307	112,467

Economic Analysis Conclusions

From this forward-looking economic analysis, which is based on the LOM provided by the current Reserves determined herein, BOYD concludes that the project economics are generally favorable if sound mining and processing practices as recommended in this Technical Report are followed. The foregoing coupled with potentially significant resource development potential on untested portions of the LGP are encouraging for further project prosecution.

1.21 Adjacent Properties

There are no adjacent properties proximal to the LGP not currently held by the Issuer.

1.22 Other Relevant Data and Information

All relevant data and information regarding the LGP is included in other sections of this

Technical Report.

BOYD is not aware of any other data that would make a material difference to the quality of this Technical Report or make it more understandable, or without which the report would be incomplete or misleading.

1.23 Interpretations and Conclusions

Drilling, Sampling, and QAQC Practices

Drilling practices have generally been to an acceptable standard in previous programs. However, drill core sampling methodology under previous programs required revision to terminate samples at lithological contacts, changes in alteration, and separate sampling of mineralized intervals for a minimum sample width of 20 cm and maximum width of 1 m. In addition, it was recommended that oriented drill core should be collected using a Reflex or similar tool which should become standard operating procedure. All of the foregoing has been implemented under OTSO's recent drilling program.

Owing to the complexity of the mineralized body at Laiva, BOYD recommended closely spaced short-hole RC drilling ahead of actual mining to define the mineralized zones more precisely to further minimize mining dilution. This too has been implemented by OTSO.

The QAQC program has previously relied upon the internal procedures employed by the independent laboratories used for sample preparation and analysis, with little evidence of continual assessment of control limits per sample batch or remedial action. While the QAQC results demonstrate good performance overall, implementation of a more rigorous external QAQC program was recommended in future programs to better assess the potential for contamination during sample preparation as well as laboratory accuracy and precision. Selection of duplicate samples to better represent the mineralization at Laiva was also recommended. This too has been resolved under OTSO's recent and on-going program.

Metallurgical Testing

Previous metallurgical test work conducted on the Laiva deposit indicates that gold is typically fine-grained, free milling, and amenable to cyanide leach processing. Further test work was commissioned by OTSO to better characterize the gold grain size distribution, petrology, mineralogy, and optimum grind size. This work, though not yet fully complete has permitted a more detailed review of potential improvements to the existing mineral processing flow sheet as specifically recommended herein in Section 17.0.

Existing Process Plant as Amended

During the relatively stable 14 month period of previous operations, the Laiva process plant operated generally as expected given the characteristics and grade of feed supplied to the plant. However, plant performance was significantly impaired by mining practices which lead to excessive mine dilution issues that were further exacerbated by reliance on the previous resource model, which lacked necessary definition of mineralized zones.

BOYD completed an evaluation of historical operational information, results from the metallurgical test program completed in 2021 and results from the several phases of previous metallurgical test work to provide the basis for the estimated point forward gold recovery of 90% at a grinding circuit product size of P_{80} 100 µm.

The existing Laiva process facilities with the planned minor modifications should be able to achieve production rates for future operations which start at 1.5 M tonnes per year for year 1 ramping up to 2.0 M tonnes per year by year 3.

Potential to increase the plant production rate to over 2.0 M tons per year is likely dependent on the maximum feed rate that can be fed to the HG leaching while achieving plus 95% gold recovery. Therefore, to increase overall production rates above 2.0 M tonnes, if so desired, additional HG CIL tanks will likely be required.

The most promising option to substantially increase overall gold production from the processing would be the addition of a vat cyanide leach circuit for the treatment of the lower grade portions of the ore body. This addition would allow concurrent treatment of HG and LG ores.

Procedures previously utilized to monitor cyanide levels in the tailings discharge system included, in BOYD's opinion, excessive sampling frequency with samples collected at the point of discharge from the process plant. Based on a review of the discharge permit, the sample point should actually be at the point of slurry discharge into the tailings facility and sample frequency should be significantly reduced.

In BOYD's overall opinion, the addition of the additional primary and secondary crusher units will improve primary grinding throughput, while maintaining suitable product particle size so to maintain downline gold recovery.

Recommended minor piping modifications should maximize recovery from the gravity circuit, whereby the combined gravity circuit will see all tailings from the flash flotation unit, as compared to only part of the flow previously. Also, the particle size range will be narrowed for both the Knelson concentrator, focused on the finer fraction, and the spirals focused on the coarser fraction for better recovery from each.

Finally, installation of the regrind tower mill will permit maximum feed to the high grade leach circuit, which has demonstrated high recovery of gold at greater than 95% Au recovery.

Exploration Potential

In BOYD's opinion, the LGP likely holds favorable exploration potential. Near-mine exploration potential exists to the north of the North Pit, associated with a northeast trending structure defined by a gold-arsenic anomaly in bedrock sampling. Further stripping of ±3 m deep surficial till to allow channel sampling and mapping will be required to advance this area. In addition, the mineralization at the Laiva Gold Mine remains open at depth. The controls on high grade mineralization require better understanding in order to assess the down dip potential.

1.24 Recommendations

Resources/Reserves and Mining

Continue With Program of Closely Spaced RC Definition Drilling

Given the complexity of the Laiva mineralized body, in BOYD's opinion, it is not cost effective to drill resource development core holes on a tighter spacing than is required by definition of measured and indicated resources. Very closely spaced short RC definition drilling will be more cost-effective in developing higher precision for the mineralized zone necessary to minimize mining dilution for attendant higher grade mill feed.

Implement Detailed Bench Mapping to Advance Structural Understanding of the Deposit

A better understanding of the structures, including sequence of events and implications for mineralization will improve overall understanding of the deposit both for continued resource development and effective mining strategies. Also, detailed mapping of intrusive bodies pre and post-mineralization, as well as intrusive impacts on primary mineralization should lead to a better understanding of potential zones of secondary enrichment through re-mobilization of gold. Mapping should be laid out on 1 m X 1 m grid and surveyed for accurate geo-reference.

Continued Infill and Step Out Core or RC Drilling for Additional Resource Development for Priority Targets as Above

Based on good comparative results from recent diamond core drilling to RC drilling to the East of the existing North Pit, BOYD recommends instituting an RC drilling program for in-fill and step out resource development to be followed up with limited diamond core drilling to confirm accuracy of RC results, perhaps to include one twinned core hole per 10 RC holes.

It is essential that the RC drilling rig(s) be equipped with high pressure, high volume compressors to assure clearing of the holes with no lag in cutting returns given the potential for groundwater presence in the holes. The RC rigs should be equipped with auto samplers receiving the underflow from the velocity sink cyclone to ensure consistent sampling.

Processing

- Evaluate alternative crushing circuit options to eliminate rented crushers to optimize feed to the grinding circuit.
- Utilize results from the completed metallurgical test program to develop detailed optimization plans for all process unit operations.
- Develop metallurgical balance with adequate detail for all process related planning and reporting requirements following plant restart.
- Plan detailed sampling and analyses program for initial operating period to provide the results required to optimize the split between low-grade and high-grade leach circuits. The program should include:
    - Sampling procedures that ensure the collection of representative samples for the feed and discharge of each of the concentrate recovery systems.
    - Accurate determination of particle size distribution of all samples.
    - Quick turn-around analyses of gold and arsenic concentrations in solution and solids samples.
    - Quick turn-around determination of mineralogical composition of solids samples.
    - Bottle roll cyanide leach testing of samples, primarily for HG and LG CIL circuit samples.
- Maximize feed rate to HG circuit while achieving plus 95% gold recovery. Feed rate to the HG leach circuit is likely the limiting parameter to increase the overall plant feed rate.
- Evaluate modifications to the LG CIL circuit to reduce processing costs. LG CIL circuit potential modifications should include:
    - Reduction in leach solution cyanide concentration.
    - Reduction in air addition rate.
    - Reduction in number of CIL tanks being utilized.
    - Conversion to CIP.

- Implement a detailed carbon activity test program.
- Complete a detailed trade-off study comparing the economics of utilizing a new vat cyanidation circuit verses the existing CIL circuit for treatment of the LG zones of the ore body.
- Complete trade-off study of potential benefits for installation of a cyanide recovery circuit. A cyanide recovery circuit could potentially provide significant cost savings and environmental advantages.
- Complete leach solution cyanide analysis of samples from all locations related to the tailings discharge facility and prepare report to confirm the ability to monitor tailings discharge cyanide concentrations at the point of slurry discharge into the tailings storage facility.

Cost Estimate for Recommended Programs

Resources/Reserves and Mining

BOYD recommends the foregoing program be conducted for one year to be followed by reassessment of efficacy and continuing opportunities thereafter. The total estimated cost of the recommended programs for a one-year period is shown in Table 1.15 below.

Table 1.15 – Total Estimated Cost of the Recommended Resource/Reserves Programs

Item	Total Program Cost (US $/yr)	Comment
Definition Drilling	525,000	RC Drilling plus PAL Assay
Detailed Bench Mapping	25,200	Mapping and Surveying
Geochemical Surveys	100,000	Sample Collection and Analysis
Geophysical Surveys	180,000	Mag and ADR
RC Drilling	360,000	6,000m @ US $40/m drill + US $20/m assay
Diamond Core Drilling Verification Check	155,000	1,000m @ US $135/m drill + US $20/m assay
Total	**1,345,200**	Reassess after one year

Estimated Cost for Process Related Programs

Mush of the estimated process recommendations should be routinely included in operating costs, which are already accounted for in this Technical Report. The trade-off studies referenced above may require assistance from external consultants, but are not likely to exceed US $100,000.In the alternative, the trade-off studies may be undertaken by OTSO staff if time and expertise is available.

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2.0 INTRODUCTION

2.1 Client Name and Purpose

This Technical Report – Feasibility Study was prepared for Otso Gold Corp (OTSO), a TSX-V listed gold mining company for its existing Laiva Gold Project (LGP) by the John T. Boyd Company (BOYD). It is prepared as an NI 43-101 Technical Report in accordance with Form 43-101F1, and duly signed by its authors as Qualified Persons (QP) as defined under National Instrument 43-101 Part 1, Section 1.1. The purpose of this Technical Report – Feasibility Study is to recommend modifications to past mining practice, and minor adjustments to the process flow sheet in order to provide additional perspective to OTSO's internal determination of value for the benefit of its shareholders and to provide support for further financing of the LGP.

2.2 Terms of Reference

BOYD was commissioned by OTSO to prepare an updated Technical Report – Mineral Resource Update with an Effective Date of 4 October 2021, which preceded the Feasibility Study provided below. The LGP is comprised of existing open pit mines, a nominal 6,000 mtpd processing plant, and all related infrastructure. The LGP has an extensive history of exploration dating back to 1980 when it was first discovered. Since that time, it has gone through a succession of exploration and resource development, culminating in construction of the existing process plant and related facilities and initiation of production mining in January 2012.

Since start of production in 2012, the LGP has operated sporadically by several groups with only marginal success, largely due to poor mining practices, resulting in massive mine dilution and undercapitalization. The LGP management has now been taken over by OTSO who have engaged appropriately skilled mining and process professionals to manage the operation.

This Feasibility Study includes revisions to mining practices and minor modifications to the existing process plant that result in the production and economic projections contained herein.

The scope of work commissioned to BOYD by OTSO generally included:

- Assist OTSO technical staff with design of further resource development drilling program.
- Produce an NI 43-101 Technical Report – Resource Update based on further drilling.
- Design revisions to mining program to result in significant reduction to mining dilution from previous mining efforts.
- Oversee additional metallurgical test work based on representative samples of mineralized material.
- Recommend minor process modifications to improve process recovery:
  - Crushing and grinding circuits
  - Flash flotation and gravity circuits
  - Improve distribution to high grade CIL circuit
  - Optimize low grade CIL circuit
- Complete an NI 43-101 Technical Report – Feasibility Study embracing all of the foregoing.

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3.0 RELIANCE ON OTHER EXPERTS

This Technical Report incorporates and relies on details of the surface and subsurface mineral ownership from other experts including OTSO staff. OTSO has assisted with and has provided much of the background information for Section 20.0 “Environmental Studies, Permitting and Social or Community Impact”. This information has been reviewed and confirmed to the extent possible absent a detailed legal review. BOYD has reviewed this information in detail for its acceptance for use in this Technical Report.

BOYD was previously engaged by a former operator, Fire Steel Resources Inc, (FTR) to provide an NI 43-101 compliant PEA on the LGP with an Effective Date of 6 April 2018 and had previously duly qualified historic data for use in this Technical Report where applicable, including:

- Arthur, Dr John, “Nordic Mines, Laiva Project Mineral Resource Update”, May 2016.
- KJ Mining Consulting, “Financial Evaluation of Laiva Gold Mine – Finland”, May 2016.
- SRK Consulting (UK) Limited, “The Laiva Gold Mine Updated Business Plan 2014 – Technical Report, March 2015.
- SRK Consulting (UK) Limited, “Mineral Resource Update for the Laiva Gold Mine, Finland, December 2013.
- CSA Global Pty Ltd, “Summary Report – Mineral Resource and Ore Reserve Update”, 7 December 2012.
- Outotec, “Laiva Gold Plant Feasibility Study Volume 1 Summary”, May 2009, Revised January 2010.
- Outotec, “Laiva Gold Plant Feasibility Study Volume 2 Ore Reserve and Mine”, May 2009, Revised January 2010.

All other data input used in this Technical Report was directly overseen or produced from first principles by BOYD.

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4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Description and Ownership

The Laiva Gold Project is located in Northern Ostrobothnia, western Finland, centered on 2528500 mE, 7161000 mN (Finnish Coordinate System, Zone 2), and is situated 20 km southeast of the nearest town, Raahe, and 70 km directly south of Oulu which is the closest regional center, see Figure 4.1 below.

Figure 4.1 – Location of Laiva Gold Project

The LGP is comprised of two license areas: Laiva and Oltava. The Laiva area comprises the advanced stage Laiva Gold Mine and satellite exploration projects at Mussuneva and Kaukainen, see Figure 4.2 below. Oltava is located 12 km south of Laiva and is an early-stage exploration property (Figure 4.2).

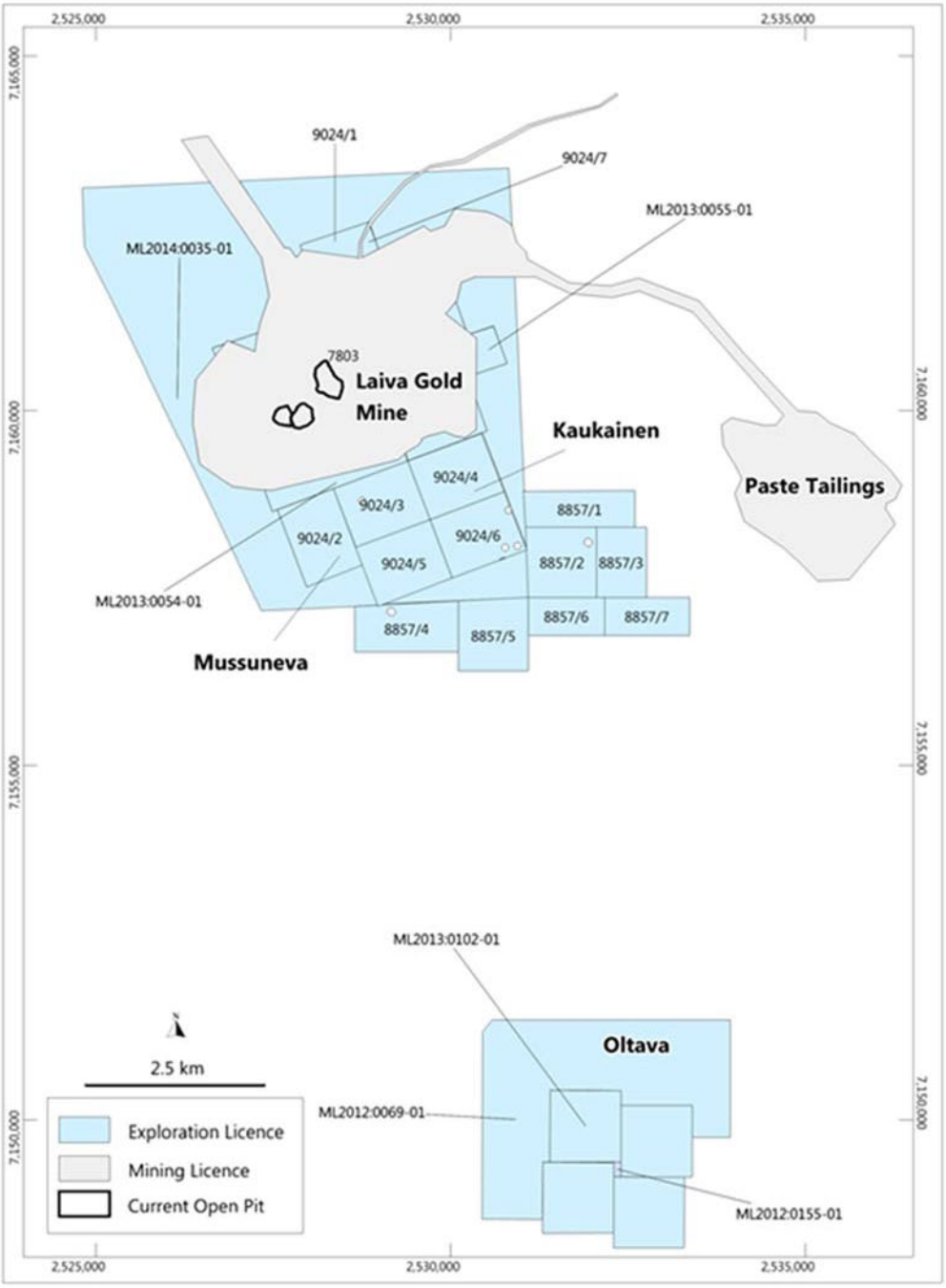

Figure 4.2 – Map Showing Otso Gold Corp Mineral Rights and Registration Numbers

It should be noted that exploration licenses are currently being renewed, which is considered to be a pro forma matter. Currently a part of the Oltava area designated as ML2013:0102-01 is on hold, to be determined.

4.2 Mineral Tenure

Exploration and mining licenses are governed by the Mining Act that was issued by the Ministry of Employment and Economy and is effective from 1st July 2011. The Finnish Safety and Chemicals Agency is the mining authority responsible for administration of permitting.

The LGP is comprised of exploration licenses totaling 33.6 km^2 and a single 16.9 km^2 mining license, for a total landholding of 50.5 km^2. The location and registration numbers of the licenses which form the LGP are shown in Figure 4.2, Page 4-2, and the current status of licenses is shown in Table 4.1, following this page.

Exploration licenses are valid for an initial period of four years and can be extended for three-year periods up to a maximum of 15 years. An annual report must be submitted to the mining authority detailing the exploration activity and results at each exploration property, as well as payment of an annual exploration fee to each landowner. The exploration fee is calculated as a rate per hectare, which varies depending on the age of the exploration license, as follows:

- Years 1 – 4: €20/ha
- Years 5 – 7: €30/ha
- Years 8 – 10: €40/ha
- Years 11 – 15: €50/ha

The mining license was granted on 31st October 2008 and is valid until 31st October 2023. The term of the mining license can be extended either for an additional fixed term of ten years, or until further notice in order to exploit the deposit.

Table 4.1 – Exploration License Status

Project	Registration Number	Status	Granted
Laiva	9024/1	under renewal	28.4.2014
	9024/2	under renewal	
	9024/3	under renewal	
	9024/4	under renewal	
	9024/5	under renewal	
	9024/6	under renewal	
	8857/1	under renewal	23.11.2012
	8857/2	under renewal	
	8857/3	under renewal	
	8857/4	under renewal	
	8857/5	under renewal	
	8857/6	under renewal	
	8857/7	under renewal	
	ML2014:0035-01	under renewal	31.5.2006
	ML2013:0054	under renewal	27.9.2005
	ML2013:0055	under renewal	27.10.2005
Oltava	ML2012:0155	under renewal	19.8.2014
	ML2013:0102	under renewal	17.11.2008
	ML2012:0069	under renewal	28.7.2016

4.3 Surface Rights

OTSO holds 900 ha of the surface area within the mining license area. The remainder of the surface rights at the LGP within the mining and exploration licenses are held by private landowners and forestry companies. The mining license permits OTSO to conduct mining related activities and the surface rights are considered sufficient for mining operations. The area of the mining license is uninhabited with limited commercial logging in the vicinity of the project area.

In addition to the annual exploration fees, compensation is payable to landowners within the exploration license in the event that exploration activity such as drilling causes damage (for example, removal of trees). The compensation must be approved by the mining authority prior to commencement of the activities.

At Oltava, surface rights are held by a joint venture windfarm project between Tornator Oy and Taaleritehdas Oy. This joint venture was granted a permit for construction of a windfarm over the same area as the exploration licenses subsequent to the granting of exploration licenses to OTSO.

4.4 Royalties

A 0.55% royalty is payable to landowners within the mining permit and is paid pro rata depending on the area of land they own within the mining permit. The royalty is based on the average annual value of metal produced in the subject period.

A 2.5% NSR royalty is payable to PFL RAAHE HOLDINGS LLP, in connection with settlement of previous project debt.

4.5 Discussion

OTSO holds sufficient rights and title in order to conduct exploration (subject to license renewal in progress) at the LGP and mining at the Laiva Gold Mine. Periodic license renewal approvals are required from the mining authority in order to maintain these licenses in good standing. The renewals are routine ministerial matters and pose no issue, so long as operations remain in compliance with regulatory requirements. No other significant factors or risks that may affect title or the right or ability to perform work on the property have been identified by BOYD.

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5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Laiva Gold Project is readily accessed from the town of Raahe, North Ostrobothnia, located 20 km northwest of the project with a population of approximately 20,000. Raahe is located 600 km north of Helsinki and 80 km southwest of Oulu and is located on a major coastal highway (E8). A port is located at Raahe, as well as a steel plant. The closest airport is located at Oulu which has direct flights to Helsinki, see Figure 5.1 below. Alternatively, Raahe is easily accessed by rail or air from Helsinki to Kokkola and then by vehicle from Kokkola to the mine site.

Figure 5.1 – Location of Laiva Gold Project

From Raahe, the project is reached via public paved roads (Road 88) to the village of Kopsa, see Figure 5.2 below. A paved road from Kopsa leads to a turning onto a 4 km long gravel access road which leads directly to the Laiva Mine site and offices. Access is possible year-round. A separate exploration office and core storage facility is located 6 km east of the mine site and can be accessed via gravel roads from the mine, or paved road from Raahe. The Oltava project is accessed from Raahe via Kalajoki and Polusperä.

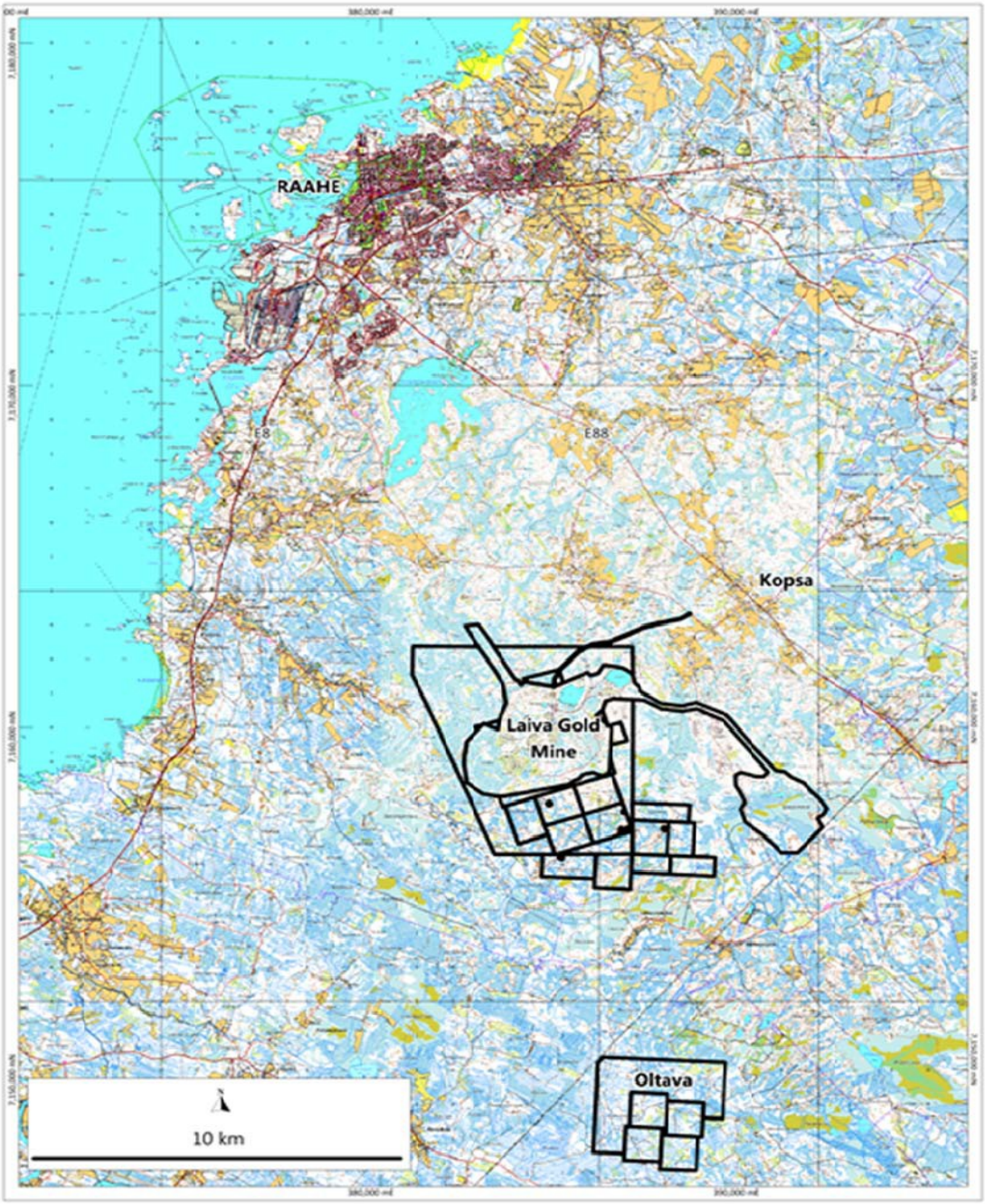

Figure 5.2 – Laiva Gold Project License Areas Access

Access across the Laiva property is via all-weather gravel roads constructed by the previous operator as well as existing forestry and public gravel roads. Access is generally good across both the Laiva and Oltava projects, with temporary access routes required for exploration drill sites located within forested areas. The primary restriction to access is in areas of peat bogs, where some ground preparation is required to provide vehicular access and stable drill platforms.

5.2 Physiography

The project area is generally flat lying with gentle topography within an area of Boreal Forest. Broad, shallow depressions contain peat bogs, and the area is incised by shallowstreams. Elevation varies between 30 m.a.s.l in the northwest and 100 m.a.s.l in the east. The mine site elevation varies between 50 and 60 m.a.s.l.

Vegetation comprises pine, spruce, and birch trees with occasional grassed cleanings and boggy areas. The following aerial photograph (supplied by OTSO) of the Laiva Gold Mine shows physiography and site infrastructure in Figure 5.3 below:

Figure 5.3 – Laiva Gold Mine Showing Physiography and Site Infrastructure

5.3 Climate

North Ostrobothnia is subject to a sub-arctic climate, typified by mild summers and cold winters, with average temperatures of -9°C in winter and 15°C in summer. Extreme temperatures of -35°C in winter and 30°C in summer are rarely observed. Annual precipitation is relatively low, averaging between 300 and 600 mm with most precipitation occurring in the summer months. The ground is snow covered for much of winter, although the snow rarely exceeds 1 m deep and has not historically affected the operating season for exploration and mining activities.

5.4 Mine Plant and Infrastructure

The Laiva Gold Project consists of two open pit mines currently being restarted and a 6,000-tonne-per-day mill and processing plant, including crushers, autogenous and pebble grinding mills, flash flotation concentrator, gravity circuit with centrifugal and spiral concentrators, and high and low grade CIL leach circuits.

Tailings facilities include a lined high-grade tailings pond adjacent to the CIL plant, and a low-grade tailings paste storage repository fed by a slurry pipeline located 7 km east of the process plant. Separate waste dumps for inert and sulphide-rich material are used and located proximal to the pits, see Figure 5.4, following this page.

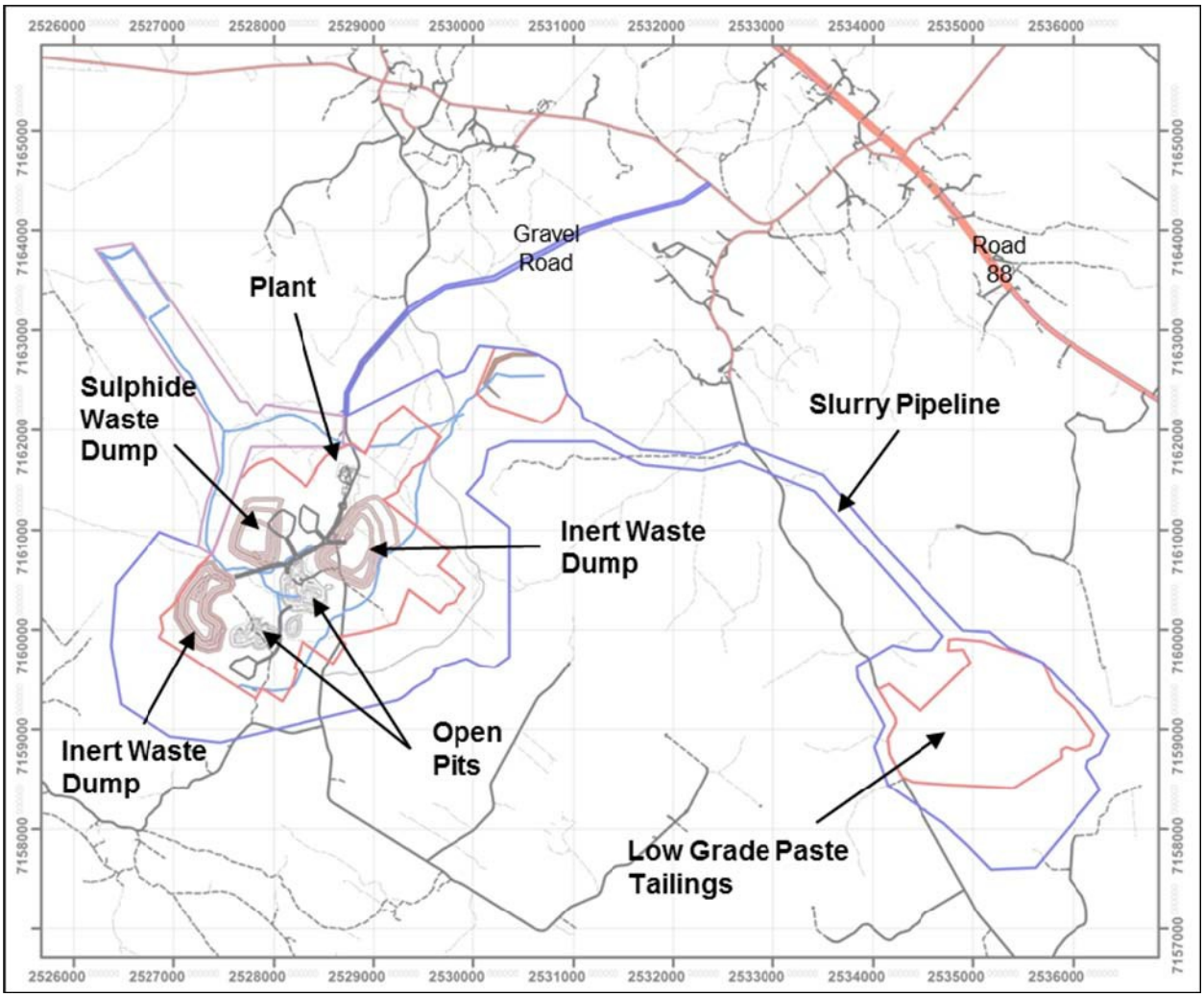

Figure 5.4 – Laiva Site Plan

A laboratory located at the plant site is currently staffed by a contractor (CRS) and includes a PAL1000 cyanide leach machine and AAS spectrometer, with a capacity of 8,000 samples per week. This laboratory will be used for grade control samples from the mine, as well as process control.

The LGP has sufficient water resources for mining and processing operations. Groundwater pumped from the pits and waste-water returned from the paste tailings is pumped and stored in a purpose-built reservoir, which provides process water for the plant. Excess water in the reservoir is discharged through an 18 km long pipe into the Gulf of Bothnia.

The mine site has access to two sources of grid power – 110 kV power is supplied from the main grid to the east and 20 kV is supplied from the village of Mattilanpera to the north.

A diesel storage area to supply the mine fleet is located at the mine site.

The project at full labor compliment is expected to employ 132 staff including both expatriates and Finnish Nationals.

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6.0 HISTORY

6.1 Discovery and Prior Ownership

The Laiva Gold Project was discovered through boulder sampling by an amateur prospector in 1980. The area was subsequently acquired by Outokumpu Oy (Outokumpu) in 1981 who explored the area until 1986 when they relinquished the licenses. The area was later acquired by Endomines Oy (Endomines) in 1999 who explored the project until it was later acquired. The Laiva Gold Project was acquired from Endomines by Nordic Mines AB, a Swedish public company through its wholly owned subsidiary company Nordic Mines Marknad AB, also a Swedish company. Nordic Mines Marknad AB in turn was the previous 100% owner of Nordic Mines Oy, (NORDIC) a Finnish company. Subsequently, NORDIC was acquired by Fire Steel Resources Inc. (FTR). Finally, after abandonment to its lender, OTSO took over NORDIC and the LGP.

The Oltava Project was first recognized in the 1950s through regional exploration and drilling conducted by the Geological Survey of Finland (GTK). The project was briefly explored by Rautaruukki Oy in 1984 and was owned by Outokumpu between 1985 and 1986. GTK conducted a second phase of exploration at Oltava between 1994 and 2002, prior to the acquisition of Oltava by NORDIC in 2005, followed by the sale to FTR and finally, the transfer to OTSO.

6.2 Historic Exploration

6.2.1 GTK

6.2.1.1 Laiva

Regional airborne magnetic, radiometric, and electromagnetic data were collected by GTK between 1982 and 1983 as part of the National Aerogeophysical Mapping Program. The data are available for the entire Laiva Project area thus providing a useful tool for regional mapping and structural interpretation.

6.2.1.2 Oltava

GTK conducted regional soil sampling, ground geophysical surveys and diamond drilling at the Oltava Project in the 1950's targeting massive sulphide base metal mineralization. Drilling intercepted narrow, high grade gold mineralization (including 1 m at 10.9 g/t gold) associated with quartz-arsenopyrite veins.

GTK recommenced exploration at Oltava between 1994 and 2002, when the area was revisited to assess the gold potential and later (2000 to 2002) for palladium potential. The exploration program comprised boulder sampling, trenching, shallow percussion drilling and inclined diamond drilling, totaling 3,484.8 m. Multiple auriferous quartz-arsenopyrite veins were intercepted, as well as a small lens of gold-molybdenum bearing quartz with mineralized intervals up to 1 m at 72.6 g/t gold. However, GTK decided to relinquish the project due to the perceived limited size potential.

6.2.2 Outokumpu Oy

6.2.2.1 Laiva

Outokumpu conducted a series of boulder and till sampling and top of bedrock percussion drilling on a 100 m by 50 m grid at Laiva. Gold anomalies were followed up with a diamond core drilling campaign totaling 6,926 m which defined mineralization at Laiva North Pit over an area of approximately 400 m by 400 m and Laiva South Pit over an area of 250 m by 200 m. Their exploration work resulted in several historic mineral resource estimates, see Section 6.3, following this page.

Outokumpu also conducted geophysical surveys including ground magnetic, Bouger gravity, and induced polarization over a maximum area of 3 km by 2 km. The results demonstrated that metavolcanic rocks gave a more intense magnetic response relative to the quartz diorite and granitoids. Results from IP (chargeability and resistivity) were inconclusive and were not conducted over the main body of mineralization. See Section 8.0.

Two shallow pits were excavated in what is now recognized as the North Pit to expose the bedrock and collect bulk samples for metallurgical test work. In total, Outokumpu extracted 5,000 t of material with a grade of 2.6 g/t gold. Details regarding the test work and results are not known.

Outokumpu's exploration work also resulted in the definition of gold prospects at Mussuneva and Kaukainen, which were both followed up with a total of six shallow (<120 m) diamond drill holes. The drilling at Mussuneva and Kaukainen intercepted narrow zones of sheeted quartz-sulphide veins with a similar character and grade to that observed at Laiva.

6.2.2.2 Oltava

Outokumpu drilled three holes totaling 13.5 m at Oltava but did not conduct any other exploration work.

6.2.3 Endomines Oy

Endomines drilled 10 diamond holes totaling 772 m at Laiva in 1999. The holes were largely drilled as infill to the Outokumpu holes. Results of the drilling were used for historical mineral resource estimation, presented in Section 6.3.

6.3 Historical Resource and Reserve Estimates

Historical mineral resource and reserve estimates have been conducted by the previous operators at the Laiva Gold Mine. There are no historical resource estimates for the Mussuneva, Kaukainen, or Oltava projects.

The results of each historical resource estimate for the Laiva Gold Mine are summarized below.

6.3.1 Outokumpu Historical Mineral Resource Estimates

Outokumpu produced two mineral resource estimates in 1984 and 1985, which were reported to the mining authority. The key assumptions, parameters, methods, and results of the estimates are presented in Table 6.1 below:

Table 6.1 – Outokumpu Historic Internal Mineral Resource Estimates

Date	Company	Deposit	Category	Tonnes	Gold (g/t)	Contained Gold (oz)	Methodology
1984 Code:	Outokumpo Internal	Laiva North	Measured	-	-	-	In-house polygonal using a 2 g/t gold cut-off and minimum mineralized interval of 2 m.
			Indicated	413,000	4.30	57,103	
			Inferred	137,000	3.10	13,656	
		Laiva South	Measured	-	-	-	
			Indicated	190,000	5.60	34,212	
			Inferred	-	-	-	
1985 Code:	Outokumpo Internal	Laiva South	Measured	-	-	-	In-house polygonal using a 3 g/t gold cut-off and minimum mineralized interval of 1 m.
			Indicated	52,100	9.20	15,412	
			Inferred	-	-	-	

The Outokumpu mineral resource estimates used categories defined as Measured, Indicated, and Inferred; however, the reporting code is not known, and the estimation methodology is not considered by BOYD to be appropriate. The historical estimates produced by Outokumpu are thus considered unreliable. The area in question has since been extracted and the historical mineral resource estimate is no longer relevant. A qualified person has not done sufficient work to classify the Outokumpu estimate as a mineral resource.

6.3.2 Endomines Historical Mineral Resource Estimates

Endomines produced a mineral resource estimate in 2005, which was reported to the mining authority. The key assumptions, parameters, methods, and results of the estimates are presented in Table 6.2 below:

Table 6.2 – Endomines Mineral Resource Estimates

Date	Company	Deposit	Category	Tonnes	Gold (g/t)	Contained Gold (oz)	Methodology
2005 Code:	Endomines Internal	Laiva North	Measured	-	-	-	Block model with a 10 x 3 x 10 m block size using a 2 g/t gold cut-off.
			Indicated	460,000	3.20	47,331	
			Inferred	250,000	3.00	24,116	
		Laiva South	Measured	-	-	-	Block model with a 5 x 3 x 5 m block size using a 2 g/t gold cut-off.
			Indicated	290,000	4.10	38,232	
			Inferred	150,000	3.80	18,328	

The Endomines mineral resource estimate used categories defined as Measured, Indicated, and Inferred; however, the reporting code is not known, and the estimation methodology is considered inappropriate by BOYD. Therefore, the estimate produced by Endomines is considered by BOYD to be unreliable. The mineralization included in the historical estimate has since been partially extracted and the historical mineral resource estimate is no longer relevant. A qualified person has not done sufficient work to classify the Endomines estimate as a mineral resource.

6.3.3 NORDIC Mines Previous Mineral Resource and Reserve Estimates

NORDIC Mines commissioned several mineral resource and reserve estimates between 2008 and 2015 using independent consultants. The results are summarized below.

6.3.3.1 NORDIC Historic Mineral Resource and Reserve Estimates

The mineral resource estimates prepared by NORDIC between 2008 and 2012 were reported in accordance with JORC (2008) and CIM (2005). These mineral resource and reserve estimates are therefore considered historical in nature and unreliable.

The historical mineral resource and reserve estimates reported by NORDIC between 2008 and 2012 used Measured, Indicated, and Inferred resource categories and Proven and Probable reserve categories as defined by the JORC 2008, which are equivalent to the categories used in reporting of mineral resources under NI 43-101. The key assumptions, parameters, methods, and results of the estimates are presented in Table 6.3, following this page. The historical mineral resource estimates are superseded by the mineral resource estimate presented in Section 14.0, and do not include material removed by mining. Therefore, the historical mineral resource and reserve estimates

reported by NORDIC are not considered relevant. A qualified person has not done sufficient work to classify the NORDIC estimates as mineral resources.

Table 6.3 – Historic Resource Estimate Commissioned by NORDIC Under Previous Operator

Date	Company/ Consultant	Deposit	Category	Tonnes (Mt)	Gold (g/t)	Contained Gold (koz)	Methodology
2008	Nordic	Primary Resource	Measured	2.94	2.12	200	Ordinary Kriging into 25 m x 10 m x 20 m blocks using a 0.8 g/t gold cut-off, constrained to a broad wireframe and exclusive of granite waste rock. Divided into two domains, North and South.
	P.A Dowd		Indicated	7.53	2.36	571	
Code:	JORC (2008)		Inferred	4.10	2.40	316	
		Additional Resource	Measured	-	-	-	Ordinary Kriging into 25 x 10 x 20 m blocks, to only include mineralization between 0,5 and 0,8 g/t gold, constrained to a broad wireframe and exclusive of granite waste rock. Divided into two domains, North and South.
			Indicated	5.27	0.63	107	
			Inferred	-	-	-	
2008	Nordic	Primary Resource	Measured	3.58	2.36	270	The Mineral Resource was estimated within constraining block volumes based upon lower cut-off's of 0.5, 0.8 and 1g/t Gold, which have been combined into a composite model. Gold grades were estimated using Ordinary Kriging. A top cut of 30g/t Gold has been applied to the sample data. The resource is reported for block grades exceeding a 0.5g/t Gold grade.
5th October (UK)	CSA Global		Indicated	8.50	2.13	580	
Code:	CIM (2005)		Inferred	3.59	1.94	220	
	NI 43-101	Additional Resource	Measured	0,94	0.63	20	
			Indicated	3.57	0.62	71	
			Inferred	-	-	-	

Date	Company/ Consultant	Deposit	Category	Tonnes (Mt)	Gold (g/t)	Contained Gold (koz)	Methodology
2009	Nordic	Primary Resource	Measured	4.89	2.33	366	The Mineral Resource was estimated within constraining block volumes based upon lower cut-off's of 0.5, 0.8 and 1g/t Gold, which have been combined into a composite model. Gold grades were estimated using Ordinary Kriging. A top cut of 30g/t Gold has been applied to the sample data. The resource is reported for block grades exceeding a 0.5g/t Gold grade.
30th March	CSA Global (UK)		Indicated	9.50	2.15	657	
Code:	CIM (2005)		Inferred	4.17	2.00	268	
	NI 43-101	Additional Resource	Measured	1.18	0.63	24	
			Indicated	3.99	0.63	81	
			Inferred	-	-	-	

Table 6.3 – Continued

Date	Company/ Consultant	Deposit	Category	Tonnes (Mt)	Gold (g/t)	Contained Gold (koz)	Methodology
2011 11th July	Nordic CSA Global (UK)	Primary Resource	Measured	4.88	2.32	364	The Mineral Resource was estimated within constraining block volumes based upon gold lower cut-off's of 0.5, 0.8 and 1g/t Gold, which have been combined into a composite model. Gold grades were estimated using Ordinary Kriging. A top cut of 30g/t Gold has been applied to the sample data. The resource is reported for block grades exceeding a 0.5g/t Gold grade.
			Indicated	9.88	2.14	680	
Code:	JORC (2008)		Inferred	3.62	1.81	211	
		Additional Resource	Measured	1.18	0.62	24	
			Indicated	4.14	0.62	83	
			Inferred	-	-	-	
		Global Estimate	Proved Reserve	5.76	1.83	339	Whittle Pit Optimization using a gold price of $535/oz, mining recovery of 90% and mining dilution of 10%.
			Probable Reserve	7.15	1.86	427	
2012 8th May	Nordic CSA Global (UK)	Primary Resource	Measured	5.85	2.06	388	The Mineral Resource was estimated within constraining block volumes based upon a lower cut-off of 0.8 g/t Gold, which have been combined into a composite model. Gold grades were estimated using Ordinary Kriging and a recoverable resource was estimated using Uniform Conditioning and Local Uniform Conditioning was applied to produce SMU blocks. A top cut of 30g/t Gold has been applied to the sample data. The resource is reported for block grades exceeding a 0.5g/t Gold grade.
			Indicated	14.17	2.03	925	
Code:	JORC (2008)		Inferred	2.14	2.07	142	
		Additional Resource	Measured	2.04	0.64	42	
			Indicated	5.35	0.64	110	
			Inferred	-	-	-	
		Global Estimate	Proved Reserve	7.30	1.56	366	Whittle Pit Optimization using a gold price of $750/oz, mining recovery of 90% and mining dilution of 10%.
			Probable Reserve	9.58	1.63	502	
2012 8th May	Nordic CSA Global (UK)	Primary Resource	Measured	5.21	2.08	349	The Mineral Resource was estimated within constraining block volumes based upon a lower cut-off of 0.8 g/t Gold, which have been combined into a composite model. Gold grades were estimated using Ordinary Kriging and a recoverable resource was estimated using Uniform Conditioning and Local Uniform Conditioning was applied to produce SMU blocks. A top cut of 30g/t Gold has been applied to the sample data. The resource is reported for block grades exceeding a 0.5g/t Gold grade and was updated using as-mined topography.
			Indicated	14.02	2.03	915	
Code:	JORC (2008)		Inferred	2.14	2.07	142	
		Additional Resource	Measured	1.83	0.64	38	
			Indicated	5.30	0.64	109	
			Inferred	-	-	-	
		Global Estimate	Proved Reserve	6.41	1.57	324	Whittle Pit Optimization using a gold price of $750/oz, mining recovery of 90% and mining dilution of 10%.
			Probable Reserve	8.78	1.60	452	

Note: Numbers May Differ Between Reports Due to Rounding.

6.3.3.2 NORDIC Later Mineral Resource and Reserve Estimates

NORDIC released two mineral resource estimates and a mineral reserve estimate prepared in accordance with JORC (2012) by SRK in 2013 and 2015 and Dr. John Arthur/SRK in 2016. These estimates used Measured, Indicated, and Inferred category resources and Proven and Probable Reserves which are equivalent to the categories defined under NI 43-101. The key assumptions, parameters, methods, and results of the estimates are presented below in Table 6.4.

Table 6.4 – Reserve Estimate Commissioned by NORDIC Under Previous Operator

Date	Company/ Consultant	Deposit	Category	Tonnes (Mt)	Gold (g/t)	Contained Gold (koz)	Methodology
2013 1st November Code:	Nordic SRK Consulting (UK) JORC (2012)	Global Estimate	Measured	-	-	-	Constrained by a 0.3 g/t gold leapfrog shell. Mineral resource estimate used a 0.6 g/t gold cut-off and Ordinary Kriging into blocks 20 m x 20 m x10 m, sub-divided using Localized Uniform Conditioning into block measuring 10 m x m 5 m x 10 m. Composites >30 g/t gold were restricted to a 10m sphere of influence. Mineral resources were reported within a Whittle Pit Shell using a gold price of $1,750/oz.
			Indicated	15.42	1.49	739	
			Inferred	3.15	2.28	231	
2015 1st January Code:	Nordic SRK Consulting (UK) JORC (2012)	Global Estimate	Measured	-	-	-	Constrained by a 0.3 g/t gold leapfrog shell. Mineral resource estimate used a 0.6 g/t gold cut-off and Ordinary Kriging into blocks, sub-divided using Localized Uniform Conditioning into 4 m x 4 m x 5 m. Composites >30 g/t gold were restricted to a 10 m sphere of influence. Mineral resources were reported within a Whittle Pit Shell using a gold price of $1,510/oz.
			Indicated	15.97	1.52	780	
			Inferred	3.22	2.08	215	
		Global Estimate	Proven Reserve	-	-	-	Reported at a cut-off grade of 0.6 g/t gold using an assumed gold price of $1,184/oz.
			Probable Reserve	9.37	1.19	360	
2016 May Code:	Nordic SRK Consulting (UK) / Dr John Arthur JORC (2012)	Global Estimate	Measured	-	-	-	Constrained by a 0.3 g/t gold leapfrog shell. Mineral resource estimate used a 0.3 g/t gold cut-off and Ordinary Kriging into blocks, sub-divided using Localized Uniform Conditioning into 4 m x 4 m x 5 m. Composites >30 g/t gold were restricted to a 10m sphere of influence. Mineral resources were reported within a Whittle Pit Shell using a gold price of $1,400/oz.
			Indicated	24.32	1.13	885	
			Inferred	4.37	1.64	231	

Note: Numbers May Differ Between Reports Due to Rounding.

These mineral resource estimates commissioned by NORDIC were considered not relevant by BOYD as the parameters used were not considered appropriate.

Subsequently, FTR commissioned BOYD to produce a revised pit constrained mineral resource estimate presented in Table 6.5 below.

Table 6.5 – Laiva Open-Pit Constrained Mineral Resource Estimate as of 9 August 2017

Classification	Au g/t	Tonnes	Contained Au (troy ozs)
Measured	1.132	355,000	13,000
Indicated	1.248	3,442,000	138,000
Measured & Indicated	1.237	3,797,000	151,000
Inferred	1.531	9,030,000	445,000

In the same Technical Report, BOYD also recommended to FTR that further work including infill drilling, reassessment of previous mining practices, revision of detailed mine plans, metallurgical test work and optimization of mineral processing be conducted in support of a mineral reserve estimate in accordance with NI 43-101.

6.4 Production History

Under NORDIC the Laiva Mine entered production in January 2012 and mining ceased in December 2013. The mill continued to operate, processing low-grade stockpile material until March 2014. Plant availability averaged approximately 80% throughout the operating period.

Performance is summarized in the following graph in Figure 6.1 below.

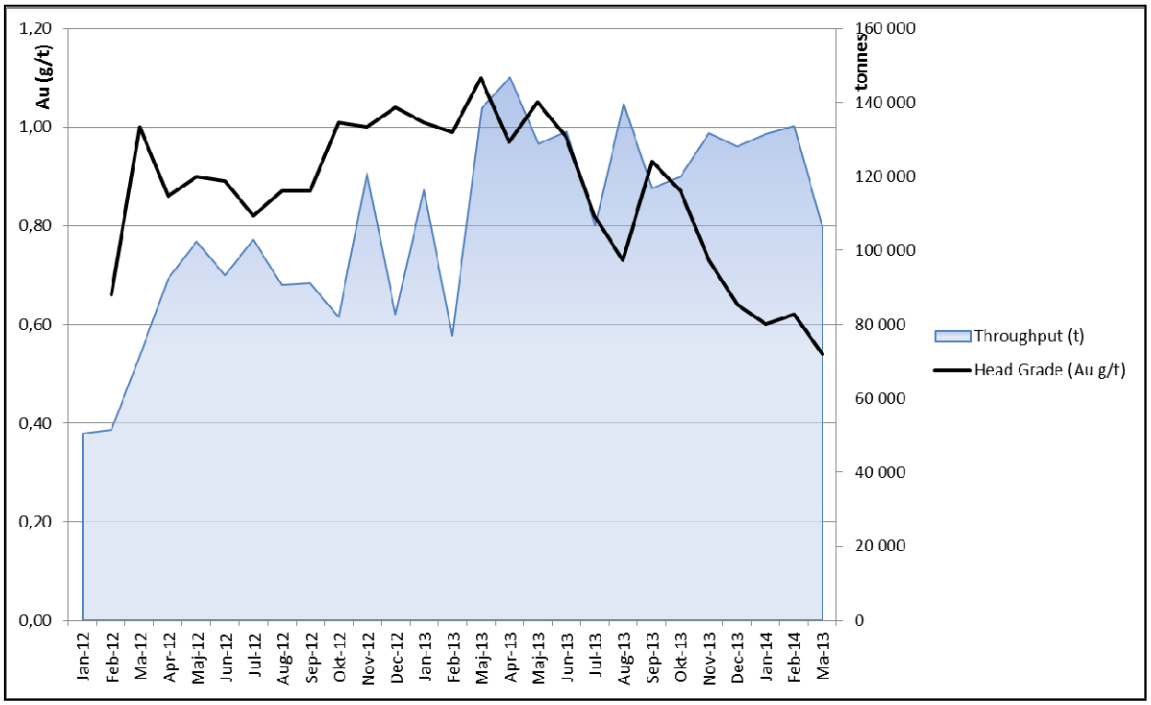

Figure 6.1 – Previous Production by NORDIC from the Laiva Gold Mine

Mining was conducted from two open pits, North Pit and South Pit. The North Pit currently measures approximately 500 m by 280 m and is 60 m deep. The South Pit measures approximately 320 m by 240 m and is up to 10 m deep. Both pits were then flooded. An area measuring 260 m by 240 m located immediately west of the South Pit has been stripped to approximately 5 m deep but there has been no production from this area to date.

Total gold production under NORDIC management was 2,241 kg from 2.8 million tonnes of ore processed, for an average LOM head grade of 0.9 g/t gold and a gold recovery of between approximately 79% and 85%. In total, 7.7 million tonnes of waste material, including 2.3 million tonnes of sulphide-rich waste, was produced with a strip ratio of 2.8:1, waste tonnes to ore tonnes. The mine operated at an average all-in sustaining cost of approximately US$1,760 per ounce gold.

Subsequently FTR acquired the LGP from NORDIC. Production history under FTR's tenure is questionable, but as reported for the production period from August 2018 through March 2019 this included:

- 534,000 mt "ore" grading 0.761 g/mt Au
- 3,381,000 mt waste
- Strip Ratio 6.33:1 waste:"ore"

It should be noted that during FTR's operation, against BOYD's strong recommendations contained in the PEA, all mining was apparently conducted utilizing 9-meter bench height, and "ore" and waste were blasted together resulting in extreme dilution of the mineralized material being fed to the mill.

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7.0 GEOLOGIC SETTING AND MINERALIZATION

7.1 Geotectonic Setting

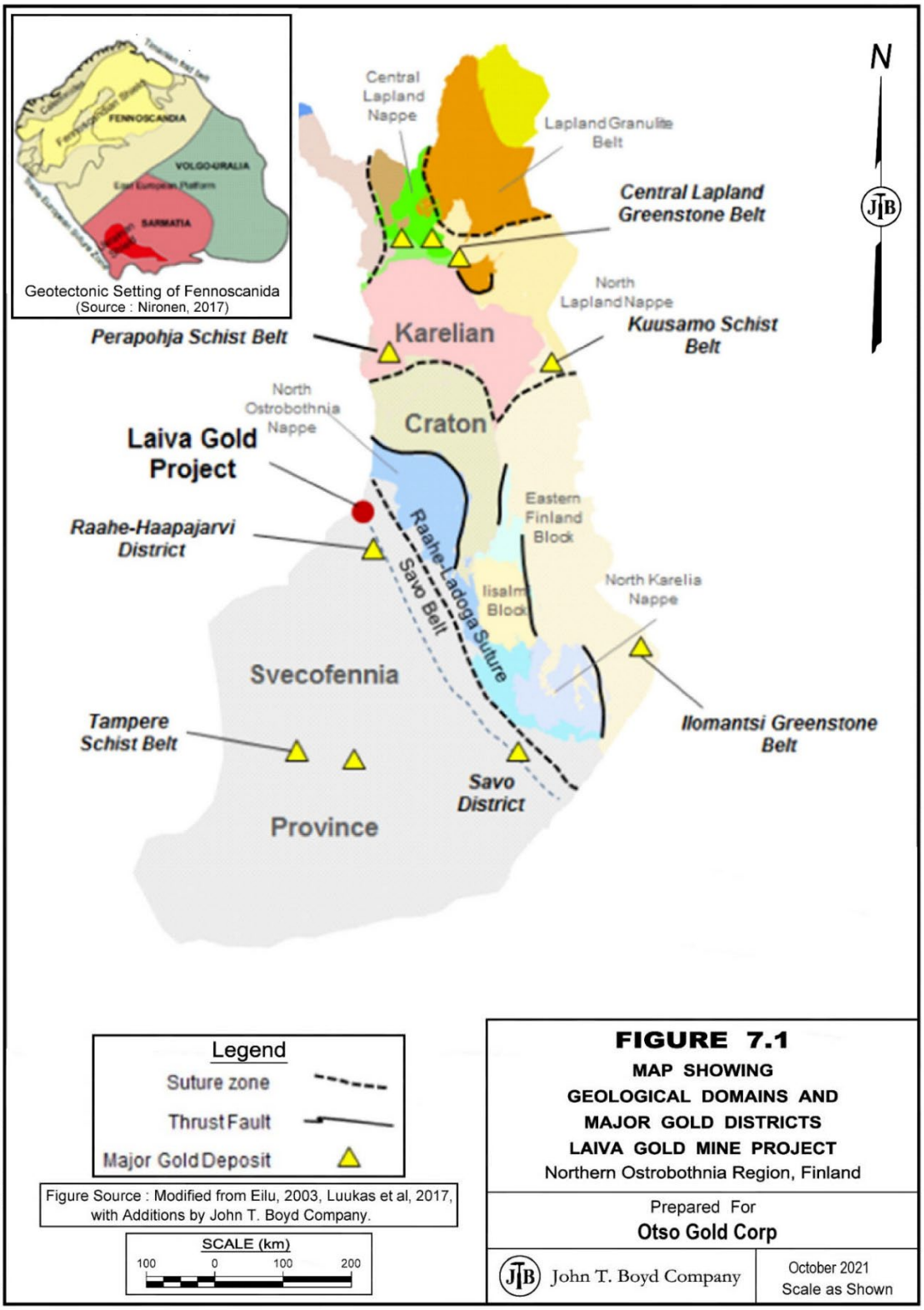
Finland is part of the Archean-Palaeoproterozoic Fennoscandian Shield, which comprises the Archean Karelian and Kola cratons flanked by the Paleoproterozoic Caledonide and Svecofennian orogenic belts to the west and southwest, see Figure 7.1, following this page. Phanerozoic sedimentary rocks are juxtaposed against the shield to the east.

The Laiva gold deposit is located in the Svecofennian orogenic belt, which formed during northeast vergent collision between multiple calc-alkaline volcanic arc complexes and microcontinents with the foreland Karelian Craton approximately 1,900 Ma. The contact between the Karelian Craton and the Svecofennian orogenic belt is marked by the deep, crustal scale, northwest trending Raahe-Ladoga suture (Figure 7.1), which is located approximately 50 km north of the Laiva Gold Deposit and can be traced through Finland into Sweden.

The accretionary history of the Svecofennian orogenic belt is complex and comprises microcontinent accretion, continental extension, continent-continent collision, orogenic collapse and then stabilization at ca. 1,770 Ma (Nironen, 2017). Several stages of Svecofennian granitic magmatism are recognized, comprising syn-orogenic (1,930 to 1,850 Ma) and late-orogenic (1,850 to 1,770 Ma) stages (Hanski, 2016).

Between 1,860-1,890 Ma an orogen-parallel shear regime was initiated resulting in bimodal volcanism and intrusion of associated plutons. This stage of the Svecofennian orogeny is recognized as the most important in terms of regional orogenic gold mineralization (Eilu et al., 2003).

The region experienced relative tectonic quiescence until rifting associated with formation of the Iapetus Ocean between 560 and 400 Ma, followed by exhumation, weathering and erosion of the Fennoscandian Shield between 300 and 60 Ma. Pleistocene glacial till covers much of the low-lying areas of the shield (Eilu et al., 2003).


Geotectonic Setting of Fennoscanida
(Source : Nironen, 2017)
FENNOSCANDIA
VOLGO-URALIA
SARMATIA
N
Central Lapland Nappe
Lapland Granulite Belt
Central Lapland Greenstone Belt
North Lapland Nappe
Kuusamo Schist Belt
Perapohja Schist Belt
Karelian
Craton
North Ostrobothnia Nappe
Laiva Gold Project
Raahe-Haapajarvi District
Raahe-Ladoga Suture
Savo Belt
Eastern Finland Block
Iisalmi Block
North Karelia Nappe
Svecofennia
Ilomantsi Greenstone Belt
Tampere Schist Belt
Savo District
Province
Legend
Suture zone
Thrust Fault
Major Gold Deposit
Figure Source : Modified from Eilu, 2003, Luukas et al, 2017, with Additions by John T. Boyd Company.
SCALE (km)
100 0 100 200
P:\CAD_GROUP-jfg\3832.001\FIGURE 7-1.DWG
FIGURE 7.1
MAP SHOWING
GEOLOGICAL DOMAINS AND
MAJOR GOLD DISTRICTS
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown

7.2 Regional Geology

The Svecofennian orogenic belt is divided into three arc complexes: the Savo Belt, central Svecofennia, and southern Svecofennia. The Laiva gold deposit is located in the 450 km long Savo Belt, immediately adjacent to the Raahe-Ladoga suture. The Savo Belt comprises supracrustal mafic-ultramafic volcanic and volcanosedimentary rocks as well as accretionary prism nappe complexes (Luukas et al., 2017), which have been intruded by mafic plutons and several generations of granitoid stocks and dikes. These Svecofennian units are variably folded and faulted and display lower greenschist through to amphibolite facies metamorphism. Younger mafic dikes and sills formed during a later period of extension and cross cut the Svecofennian stratigraphy.

Gold mineralization in the Savo Belt occurred post-peak metamorphism and resulted in the formation of two main gold districts; the Raahe-Haapajärvi district in the northwest in which the Laiva gold deposit is located, and the Savo District in the southeast. The Savo Belt also hosts major VMS (copper, nickel, lead and zinc) deposits and occurrences hosted in the supracrustal sequences.

The Raahe-Haapajärvi gold district is a northwest-southeast oriented belt comprising meta-volcanic and meta-sedimentary units, intruded by intermediate plutons and late-orogenic granitoids. The district is situated on a north-south trending fault located between two major, sub-parallel, northeast trending shear zones, and has been subject to regional shearing and ductile deformation.

7.3 Local Geology

The Laiva-Oltava Project area is comprised of lozenges of mafic metavolcanic rocks with intercalated meta-volcanosedimentary horizons, located primarily in the northwestern parts of the project area. Metagreywackes occur as boudins and slivers to the east and west of the metavolcanic units. These supracrustal units typically strike north-south, sub-parallel to the regional structural fabric, see Figure 7.2, following this page. Quartz diorite to granodiorite plutons intrude the supracrustal strata and occupy the central and eastern part of the project area. All units are intruded by granitic plutons and dikes and later mafic (dolerite) dikes (see Figure 7.3 on page 7-5).

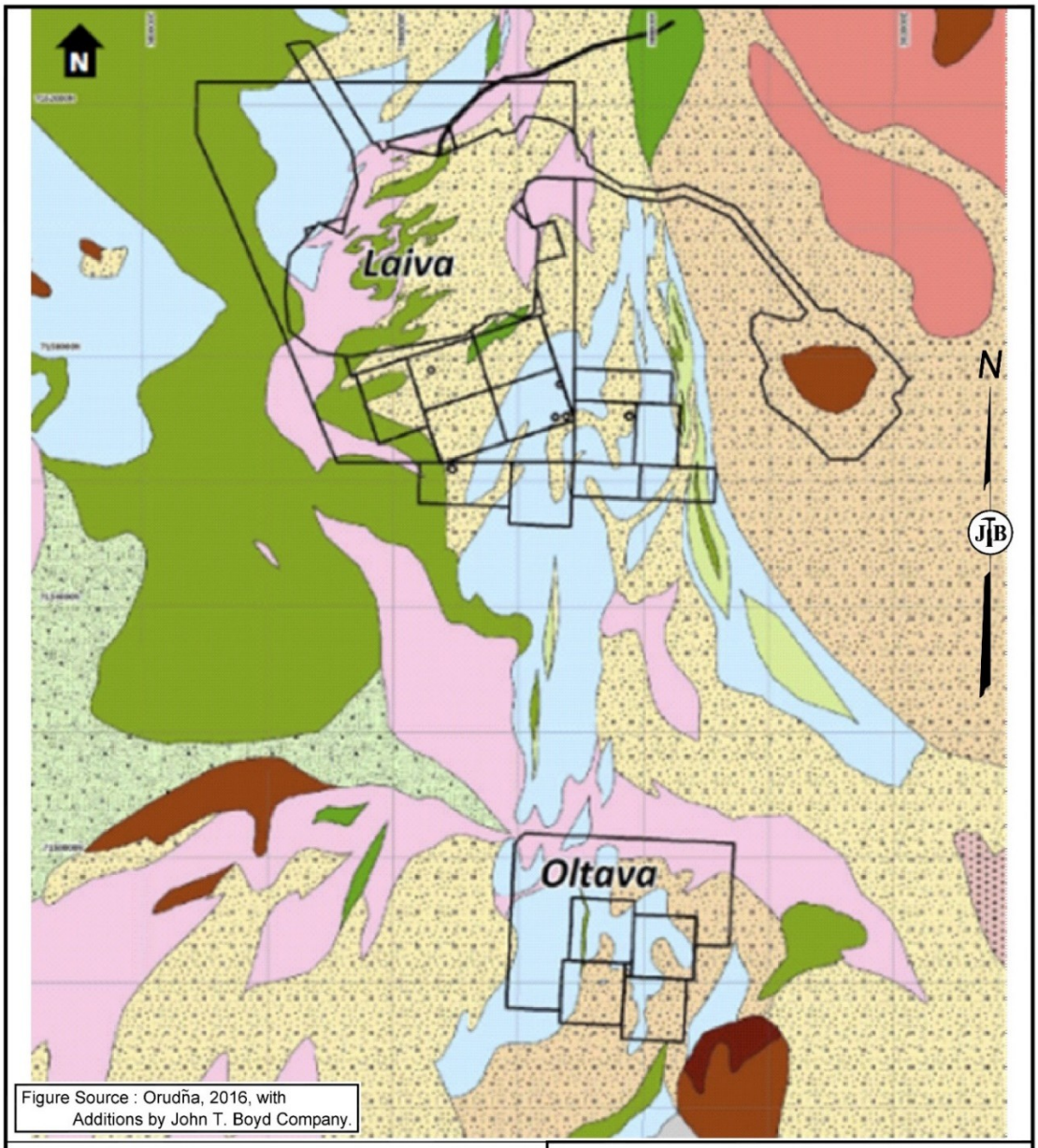


Figure Source : Orudña, 2016, with Additions by John T. Boyd Company.

Legend

- Mafic Volcanic Rocks
- Intermediate Volcanic Rocks
- Intermediate Tuffaceous Sandstones
- Felsic Volcanic Rocks
- Siliclastic Sediments (Greywackes)
- Granite
- Porphyritic Granite
- Pyroxene Granite
- Grandodiorite
- Quartz Diorite
- Peridotite
- Gabbro
- Amphibolite

SCALE (km)
1 0 1 2

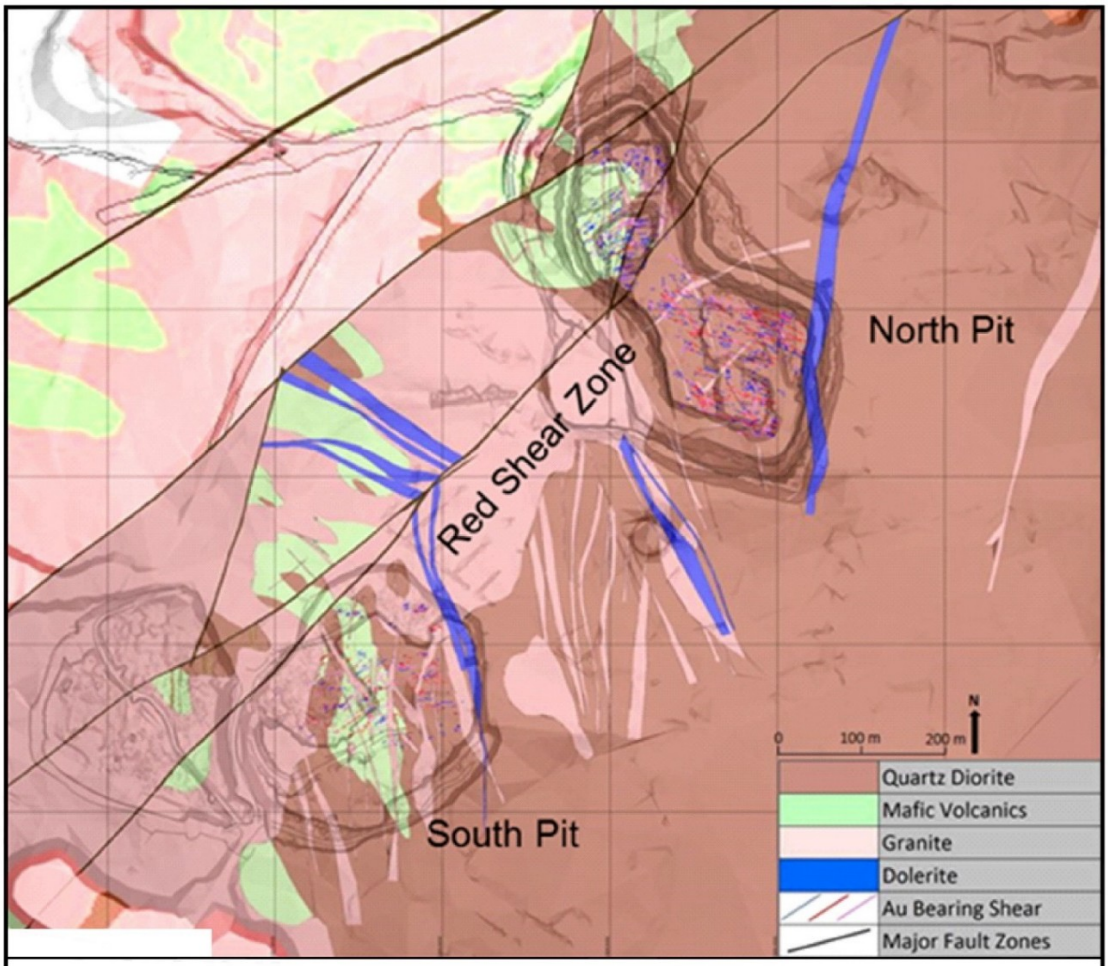
FIGURE 7.2
MAP SHOWING GEOLOGY OF THE LAIVA - OLTAVA AREAS LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
Otso Gold Corp

John T. Boyd Company

October 2021
Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 7-2.DWG



N

Figure Source : Nordic Mines Oy, with Additions by John T. Boyd Company.

SCALE (m)
100 0 100 200

FIGURE 7.3
MAP SHOWING
GEOLOGY OF THE
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
Otso Gold Corp

John T. Boyd Company

October 2021
Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 7-3.DWG

The entire project area is overlain by glacial till and sand deposits between 3 m and 15 m thick. The glacial flow direction is inferred as approximately southeast and is an important factor to consider when assessing regional geochemical data.

7.4 Mine Area

7.4.1 Lithostratigraphy

The mine area is comprised of mafic meta-volcanic (termed Uralite Porphyrite by previous operators) intercalated with rare, metamorphosed volcano sedimentary horizons, intruded by quartz diorite plutons and dikes. Together, the metavolcanic and quartz diorite are the main host to mineralization, see Figure 7.3, Page 7-5. Mafic metavolcanics occur as pendants and xenoliths in the central and western parts of the project area, in contact with a zoned quartz diorite pluton to the east and a granite pluton to the west (Figure 7.3). Mafic metavolcanic rocks are typically massive, fine-grained, equigranular with weak localized foliation. Quartz diorite is typically massive, medium-grained, and equigranular, with transitional contacts into medium-grained, porphyritic quartz diorite and medium-grained, equigranular granodiorite. The quartz diorite and metavolcanic rocks are weakly magnetic and display a weak localized schistose fabric associated with regional scale deformation.

Several stages of granitoid magmatism are observed intruding the mafic metavolcanic and quartz diorite rocks. The majority of granite is post-mineral and occurs as granite dikes, sills, and plutons; however, relatively minor narrow (<2 m thick) pre-mineral pegmatite dikes are also observed. A north-south trending granite pluton dominates the western part of the project area. In the western part of the North Pit, the granite pluton dips shallowly northeast and truncates mineralization at depth.

Granite dikes and sills typically occur proximal to and sub-parallel or perpendicular to the contact between the granite pluton and metavolcanic rocks. The granitic dikes and sills vary in thickness between <1 m to >4 m and form a conjugate pattern. In the South Pit area, the granite dikes form a dense network, encapsulating xenoliths of mineralized metavolcanic rocks, see Figure 7.4, following this page.

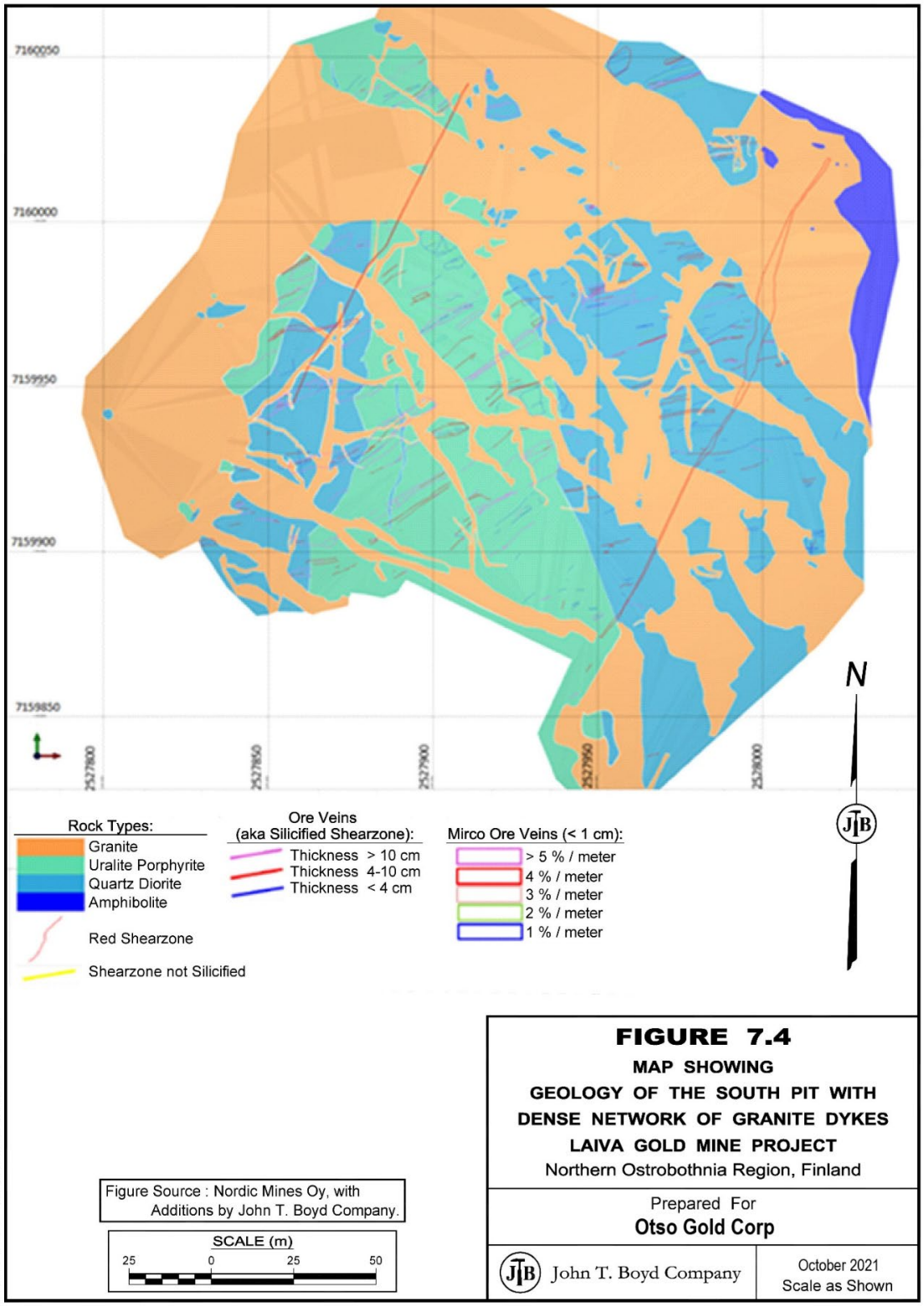


FIGURE 7.4

MAP SHOWING
GEOLOGY OF THE SOUTH PIT WITH
DENSE NETWORK OF GRANITE DYKES
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
Otso Gold Corp

John T. Boyd Company

October 2021
Scale as Shown

All units are intruded by dolerite sills and dikes up to 10 m wide. The entire project area is overlain by Quaternary glacial till and localized deposits of sand which are between 3 m and 15 m thick.

7.4.2 Structure

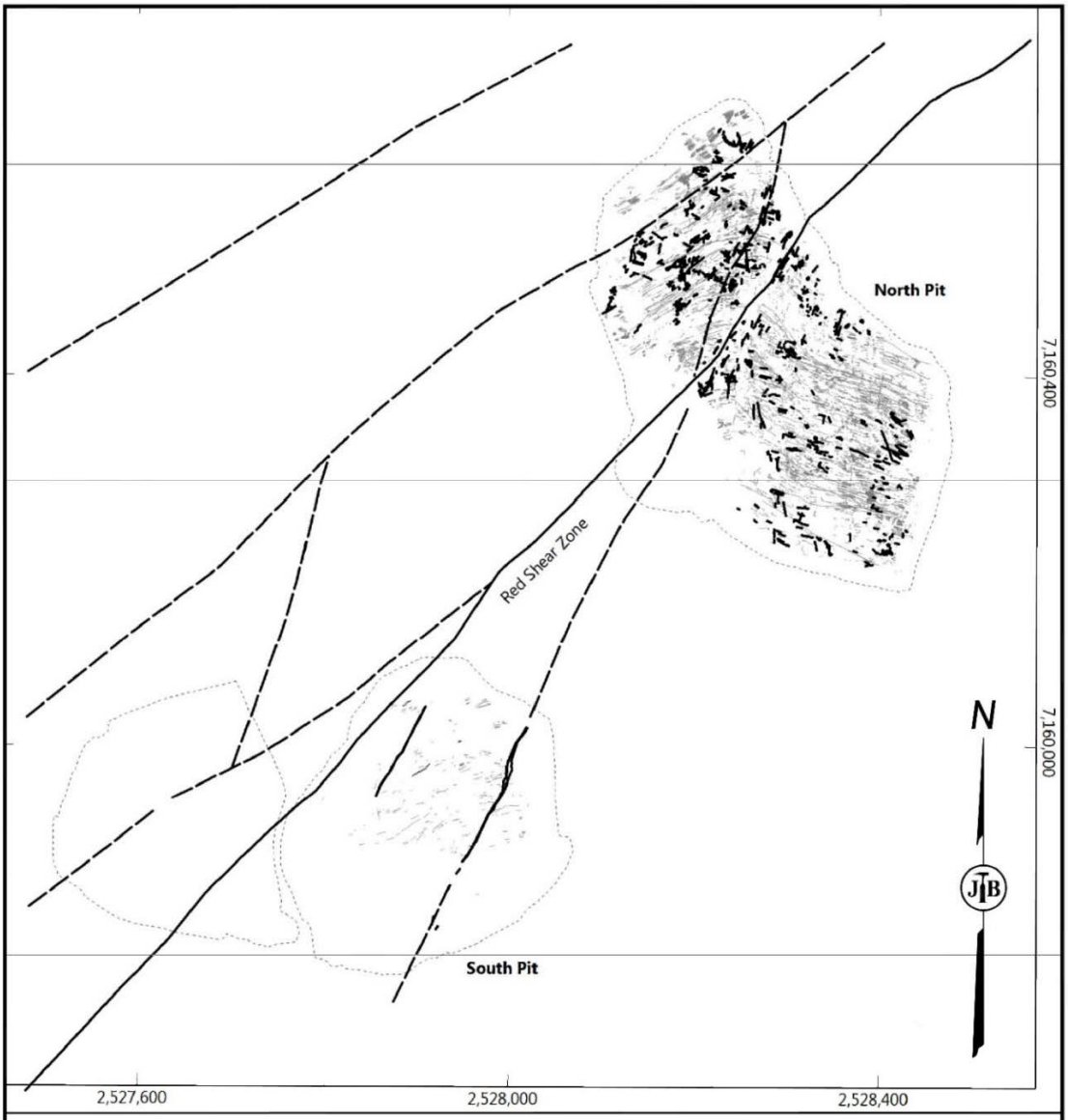
Structure has been mapped during grade control work and drill core logging, as well as inferred from ground magnetic data. Much of the structural understanding of the Laiva Gold Mine has been developed recently by Pratt (2010), SRK (2014), and BOYD (2017).

The structural geology of Laiva is complex, manifesting as poorly-defined large-scale folding of schists, weak localized schistosity in mafic metavolcanic and quartz diorite units, and localized development of foliation in some granite dikes. Parasitic folding of granite dikes, especially within meta-volcanic rocks is commonly observed. Several generations of faults are recognized, comprising mylonitic shear zones which are inferred to be synchronous with mineralization and later strike-slip and reverse faults which offset mineralization. Veins associated with most stages of deformation are observed at Laiva; however, vein paragenesis requires further investigation. The main structural elements observed at the Laiva Gold Mine are summarized below.

7.4.2.1 Faults and Shear Zones

Mylonitic shear zones are the main host to mineralization. The mylonitic shear zones display variable strike due to late ductile deformation associated with a northeast trending strike-slip fault (the Red Shear Zone, Figure 7.3, Page 7-5), but in general strike approximately east-west. The main orientations of mylonitic shear zones are shown in Figure 7.5, following this page. The mylonitic shear zones are hosted in quartz diorite and mafic volcanic rocks, post-date pegmatite dikes and are cross-cut by granite and diorite dikes. They are typically 1 to 20 cm wide, steeply south dipping (75 degrees to 85 degrees) and planar and occur as sub-parallel lineaments with weak foliation recognized between closely spaced shear zones.

Several post-mineral, northeast trending brittle-ductile strike-slip faults are observed or inferred cross cutting the mylonitc shear zones. The faults post-date granitic intrusions but their paragenetic relationship with dolerite dikes has not been observed. The faults strike northeast and dip steeply southeast. The Red Shear Zone is the best defined of these faults (Figure 7.3, Page 7-5, and Figure 7.5, following this page), which displays a dextral shear sense, strikes northeast and dips steeply southeast. Lateral displacement in the order of 10 m is indicated by inference.



Legend
- Current Open Pit
- Fault (Inferred)
- Fault (Observed)
- Mylonitic Shear

Figure Source : Nordic Mines Oy, with Additions by John T. Boyd Company.

SCALE (m)
100 0 100 200

FIGURE 7.5
MAP SHOWING
STRUCTURAL GEOLOGY OF THE
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
Otso Gold Corp

John T. Boyd Company

October 2021
Scale as Shown

P:\CAD_GROUP-jtg\3832.001\FIGURE 7-5.DWG

Mylonitic shear zones and granite dikes in the hanging wall and footwall of the Red Shear Zone in the North Pit display drag folding over tens of meters as shown in Figure 7.6 below:

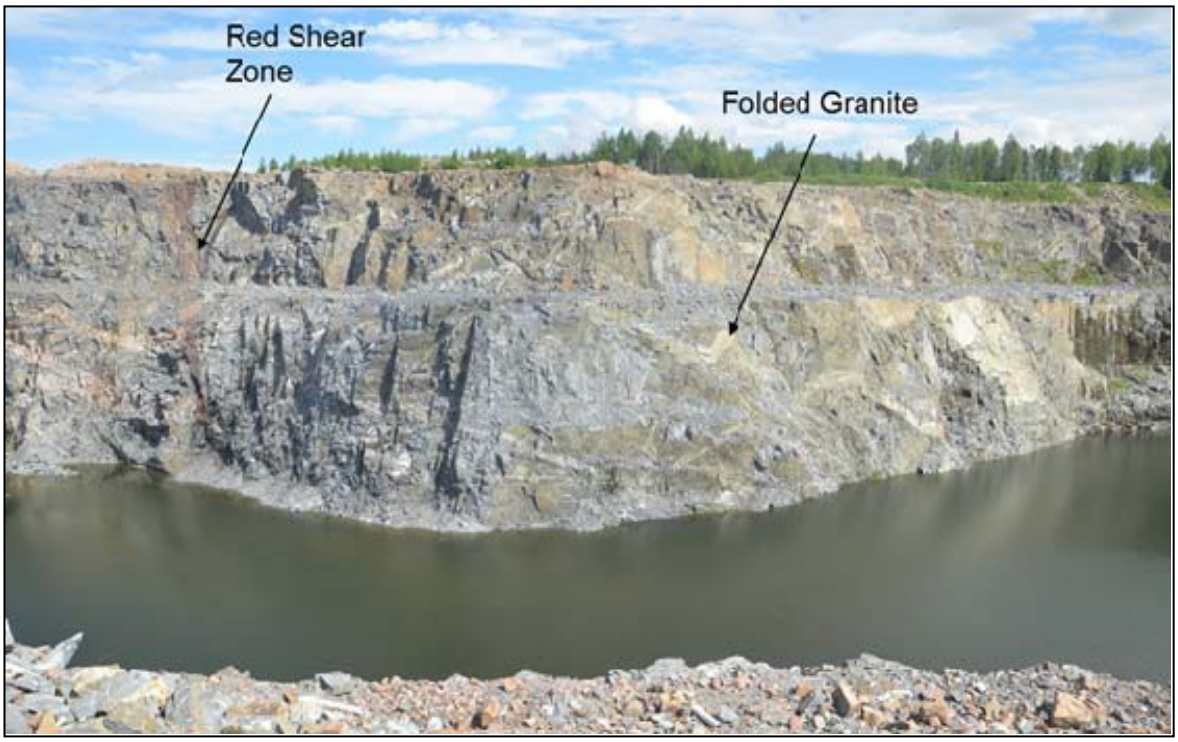


Figure 7.6 – View Northeast Over the North Pit, Showing the Red Shear Zone and Late Granite Intrusions

The Red Shear Zone displays late brittle deformation in the form of cataclastic monomict breccia with a carbonate matrix and strong hematite staining. The Red Shear Zone is typically barren of gold mineralization, with occasional low grades reported from drilling intercepts. It is postulated that these isolated gold values are associated with clasts of mineralized host rock within the fault, rather than indicating that the Red Shear Zone itself is mineralized.

The distinctive hematite staining associated with the Red Shear Zone is also observed on faults in the South Pit. The exact relationship between these faults in the North and South Pits requires further investigation, as the Red Shear Zone is known to locally modify the orientation of mineralization in the North Pit, therefore is likely to have a similar influence on the orientation of mineralization in the South Pit (see Figure 7.5, Page 7-9).

North-northeast striking, sub-vertical structures are observed in the eastern part of the North Pit. These faults display a reverse sense of movement, with a displacement of 1 m to 2 m. These faults have not been mapped in detail as they were only observed during mapping of the pits during grade control and were not recognized in drill core. Their influence on mineralization in terms of offset is minimal and their exclusion as a plane in the geological modelling should not have a material impact on mineral resource estimation. However, continued mapping of these structures will be required during grade control work.

Shallow thrust faults with cm-scale displacement are observed in the eastern part of the North Pit. Their paragenetic relationship to other faults has not been determined. Pratt (2010) postulated that the presence of shallowly dipping S1 schistosity may represent the presence of regional scale, relatively flat lying thrusts or shear zones adjacent to or underlying the Laiva project area.

7.4.2.2 Veins

Veins occur as pre-mineral bucky quartz, syn-mineral blue-grey quartz-sulphide veins hosted within mylonitic shear zones, and post-mineral white-grey quartz-carbonate veins.

Pre-mineral bucky quartz veins display folding and offset by faulting, and are observed cross cutting metavolcanic, quartz diorite, and pegmatite. These veins typically strike sub-parallel to mineralized shears but also occur in a variety of other orientations and have been folded and boudinaged (see below). These veins typically dip 70 degrees to 80 degrees to the south.

Figure 7.7 – Mylonitic Shear with Quartz-Sulphide Vein (Center) and Deformed Pre-mineral Bucky Quartz Veins Displaying Fault Drag. South Pit

Syn-mineral quartz-sulphide veins occupy mylonitic shear zones with a steep dip (75 degrees to 85 degrees to the south) and general east-northeast to east-southeast trend either side of the Red Shear Zone. En-echelon vein sets occur oblique to mylonitic shear

zones, as well as tension gashes which occur at high angles to the mylonitic shear zones, shown in Figure 7.8 below.

Figure 7.8 – Planar, Sheeted Quartz-Sulphide Veins and High Angle, Short Scale Tension Gashes in South Pit. Hammer Head for Scale

Post mineral medium- to coarse-grained, quartz-carbonate veins cross cut most units and occur at high angles to the mineralized shear zones.

7.4.2.3 Granite and Dolerite Dikes

Granite dikes and sheets, or sills, display highly variable dip and strike, but typically strike north-south and dip east to east-northeast with variable steepness. Dikes with roughly east-west strike and north dip, again with variable steepness, are less commonly observed in the North and South Pits.

Granite dikes are inferred to locally offset mineralization by up to 25 m. It is not clear from drill core observations whether the offset is a faulted displacement, or a fault drag causing parasitic folding of the mineralized zones proximal to dikes. Small scale fault drag of mineralized zones at the contacts with granite dikes is observed in the South Pit and provides some evidence that the mineralized zones are locally folded proximal to granite contacts, see Figure 7.9 below.

Figure 7.9 – Sheared Contact Between Metavolcanic Hosted Mineralized Shear Zone and Granite Dyke

7.4.3 Alteration

Alteration at Laiva is weak and comprises a rare, narrow (mm-scale) sericite-pyrite selvedge to mineralized shear zones, see Figure 7.10 below. Rare moderate albite-epidote-pyrite alteration is observed as a selvedge to mineralized shear zones in some metavolcanic units, possibly associated with volcano-sedimentary protoliths. Mafic phenocrysts display a localized magnetite destructive chlorite alteration proximal to mineralized zones.

Figure 7.10 – Sericite Selvedge to mm Scale Quartz-Pyrite-Arsenopyrite Veins Hosted in Sheared Mafic Metavolcanic

A weak, pervasive secondary biotite and trace pyrite-chlorite alteration is observed throughout the metavolcanic rocks and is possibly a metamorphic assemblage.

7.4.4 Mineralization

Mineralization is comprised of sheeted quartz-sulphide vein arrays within multiple, sub-parallel, mylonitic shear zones, hosted in metavolcanic and quartz diorite rocks. Mineralization occurs in two main bodies, the North Pit with Eastern Extension in the north, and the South Pit in the south, see Figure 7.11, following this page.

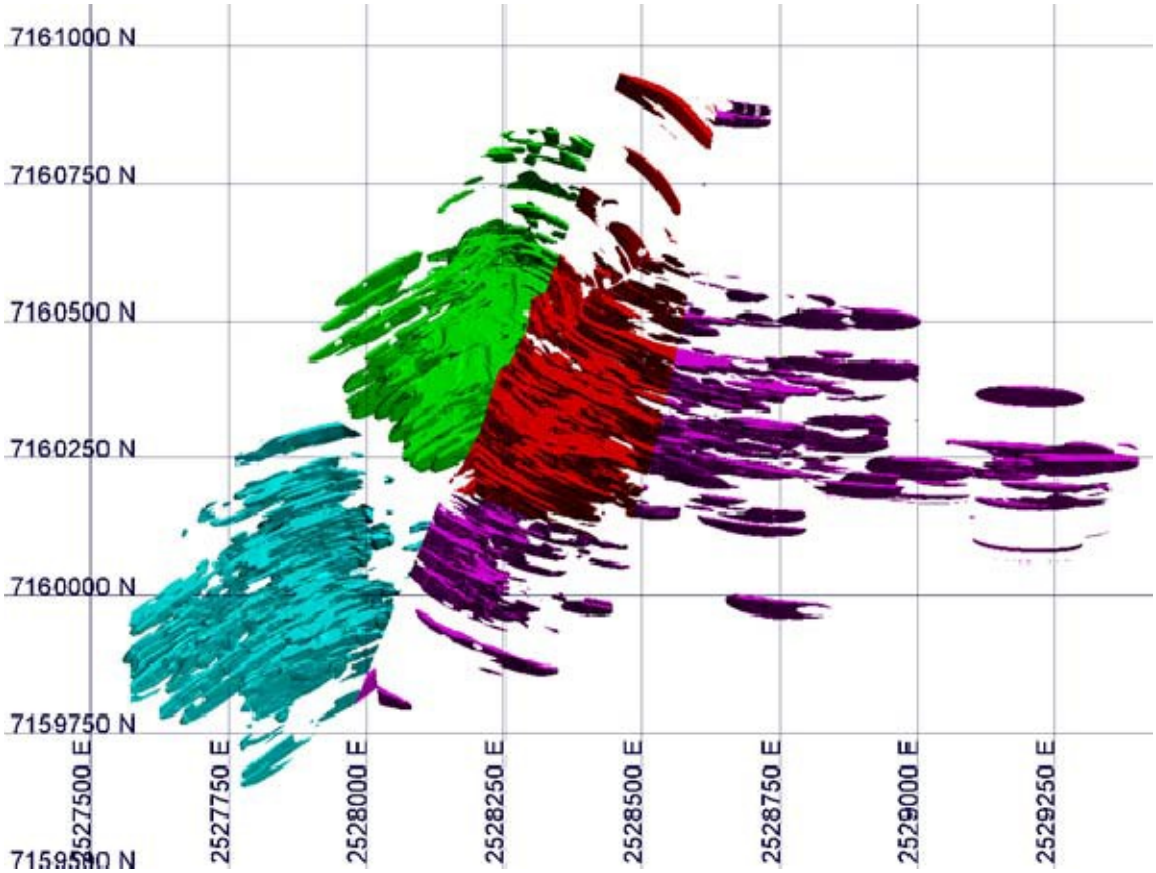


Figure 7.11 – Plan View of Mineralized Shear Zones Forming the Laiva Gold Deposit
RSZ – Red Shear Zone

At the North Pit, mineralized shear zones can be traced over a strike length of up to 600 m, with the mineralized zone (including the Eastern Extension) occurring over an area of 1,300 m by 450 m. In the South Pit area, mineralization occurs over an area of 500 m by 400 m. Mineralization has been intercepted at depths of up to 250 m below surface and is open down dip, apart from the western part of the North Pit where mineralization is truncated by the granite pluton. In both the North and South Pits, the mineralized shear zones vary in width between <1 m to >10 m.

The mineralized shear zones display distinct structural trends in different parts of the deposit, with the orientation being dependent on proximity to the Red Shear Zone. West of the Red Shear Zone, mineralized shear zones strike east-northeast and dip steeply (78 degrees to 85 degrees) south-southeast. Immediately east of the Red Shear Zone, mineralized shear zones strike east-southeast and dip steeply west-southwest, transitioning to an almost easterly strike with southerly, steep dip progressively to the east.

Within the mineralized shear zones, vein density varies between 1 and 29 veins per meter. Mineralized shear zones display sharp contacts with the host rocks and there is no gold grade associated with the narrow selvedges. In general, veins are more closely spaced in the metavolcanic rocks compared to the veins hosted in quartz diorite, and there is a positive correlation between vein density and gold grade. This increased vein density and gold grade in metavolcanic rocks compared to quartz diorite may be attributed to contrasting rheology and/or chemical composition of the host rocks.

Individual veins are typically planar, narrow (2 to 5 mm, rarely 2-5 cm) and display localized pinching and swelling, although the mineralized structure may be continuous over tens of meters, shown in Figure 7.12 below.

Figure 7.12 – Exposed Mineralized Shear Zone in the South Pit

Quartz veins are composed of very fine-grained, massive, blue-grey quartz with variable, disseminated and blebby, fine- to medium-grained pyrite, arsenopyrite, pyrrhotite, and chalcopyrite (shown below). Molybdenite is rarely observed, mainly in the Eastern Extension mineralization. Other gangue minerals including lollingite, epidote, tremolite, and scheelite have been identified from petrology studies. The following photograph, Figure 7.13, following this page, shows quartz-arsenopyrite mineralized shear hosted in metavolcanic.

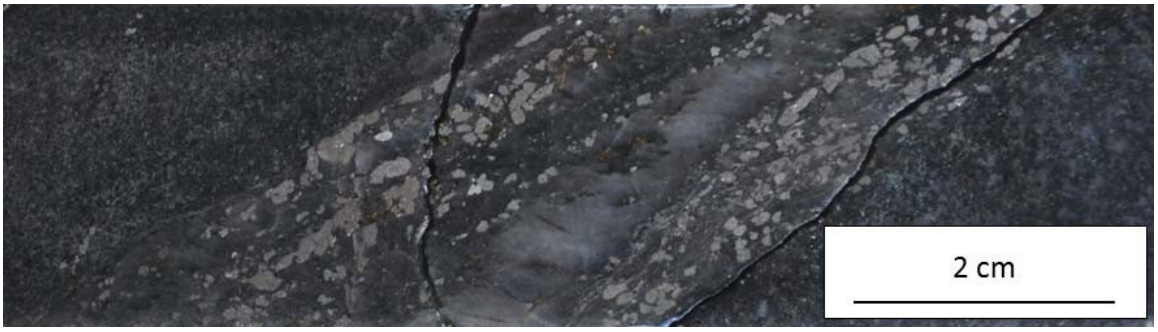


Figure 7.13 – Quartz-Arsenopyrite-Pyrrhotite Mineralized Shear Hosted in Metavolcanic

Gold mineralization principally occurs as fine-grained (sub 10 µm, Titley and Fourie, 2009), free grains and displays a positive correlation with arsenic, bismuth, and telluride. Gold rarely occurs as Maldonite (Au2Bi) and as inclusions within arsenopyrite.

Gold grade continuity is variable along strike and down dip. While structural continuity and mineralization is continuous along strike and down dip, elevated grades of >1 g/t gold display a localized character. Higher grades of >5 g/t gold are relatively common but generally dispersed within a mineralized structure. Elevated gold grades are also observed at the contacts with granite dykes, potentially indicative of gold remobilization during granitoid emplacement. Further research into these localized controls on higher gold grades is required to identify the potential for high grade shoots, which may lead to better definition and constraint of elevated grade for exploration and resource estimation purposes.

Mineralization is cross-cut by granite and dolerite intrusives as well as locally offset by faults. Intrusive units typically stope out mineralization rather than offset it, although some small-scale shearing and possible parasitic folding of mineralization at granite contacts is observed. The Red Shear Zone is inferred to have caused fault drag folding and dextral offset of mineralization. Lesser, meter-scale offset by northeast trending link structures in the North Pit is less well defined from current mapping; however, the small scale of offset is not of material concern for resource estimation purposes.

7.5 Mussuneva-Kaukainen

The prospects at Mussuneva and Kaukainen are hosted in the same metavolcanic and quartz diorite rocks as Laiva. Structural geology of the Mussuneva-Kaukainen prospect is poorly understood due to the extensive glacial till cover and limited exploration work.

Mineralization is hosted in east-northeast trending, sub-vertical, sheeted, sub-parallel quartz-sulphide (pyrite-arsenopyrite-pyrrhotite) veins with a similar character to that observed at Laiva. Mineralization is inferred from bedrock sampling to occur over a strike length of >1 km. An historic diamond drill hole drilled by Outokumpu in 1986 intercepted 1.95 m at 1.20 g/t gold and 1 m at 1.68 g/t gold. Further exploration work is required to fully determine the extent and character of mineralization at Mussuneva-Kaukainen.

7.6 Oltava

Oltava is located in the same Svecofennian suite as Laiva. Mineralization is hosted in metavolcanic, mica gneiss, and quartz diorite which is intruded by pre-mineral pegmatite and post-mineral granitoid dykes. All units are intruded by dolerite dykes.

In the south of the Oltava license area, mineralization is hosted in east-west striking, sub-vertical, sheeted veins composed of quartz-arsenopyrite with a similar character to mineralization at Laiva. The strike extent and depth of the veins is not well understood, however is currently inferred to be in the order of tens of meters. Individual veins are typically less than 5 cm wide and lenticular.

In the central part of the Oltava license area, quartz-arsenopyrite-molybdenite ± tourmaline veins and breccias hosted in quartz diorite are reported from historic work conducted by GTK (see Sections 6.0 and 10.0). A 5 m long, 2 m wide, and 30 m deep lens of quartz-molybdenite intercepted by GTK returned significant high grade gold results as follows in Table 7.1.

Table 7.1 – Historic Drilling at Oltava
Significant Results from Historic Drilling by GTK at Oltava

Hole ID	From (m)	To (m)	Interval (m)	Estimated True Width (m)	Au (g/t)
OLT324	17.0	18.7	1.7	1.2	20.2
OLT325	26.5	28.7	2.2	1.5	18.0
OLT326	15.2	17.5	2.3	1.6	33.7
OLT328	31.3	32.7	1.4	1.0	13.0

These results are historic in nature, have not been verified by a qualified person, and therefore cannot be relied upon. The results from historic drilling warrant follow up exploration, however.

8.0 DEPOSIT TYPE

Laiva is classed as an orogenic gold deposit. Orogenic gold deposits form during subduction-related accretion or collision and are spatially related to second and third order faults or shear zones proximal to crustal-scale zones of deformation. In addition, orogenic gold deposits are commonly discovered proximal to granitoid intrusions, although the nature of the relationship between gold mineralization and granitoid magmatism is unclear.

Orogenic gold deposits are characterized by formation in the late stages of orogenesis in compressional or transpressional regimes. Gold is typically deposited synchronous with ductile shearing and/or folding, and occurs in shear zones, fold hinges and lithological contacts. Mineralization forms as discrete veins, sheeted vein arrays or disseminated bodies with variable grade continuity, controlled principally by the local structural geology and chemistry of the host rock (Groves et al., 2003).

Orogenic gold deposits display a metal association of gold-silver, arsenic, boron, bismuth, antimony, tellurium, and tungsten. Vertical and lateral metal zonation is usually subtle, with gold mineralization often recognized over significant vertical intervals of >1 km.

The above characteristics are largely evident at Laiva, where gold mineralization is hosted by an array of sheeted quartz-sulphide veins and mineralized shear zones, cross cutting greenschist facies metavolcanic rocks and quartz diorite intrusions. Mineralization at Laiva is often associated with anomalous arsenic, and more rarely with copper, molybdenum, bismuth, and telluride. The presence of molybdenum and copper in association with gold is somewhat atypical of orogenic gold deposits and may be influenced by the host rock geochemistry, may represent an ore forming fluid of magmatic origin, or represent a separate stage of mineralization related to higher temperature hydrothermal fluids. This is reflected in bedrock geochemistry where molybdenum appears to occur peripheral to the gold mineralization at Laiva and Oltava, and where copper is not always associated with gold mineralization.

Mapping of gold and trace elements including arsenic, molybdenum, and copper has been proven to be an effective exploration tool at Laiva and should be continued, in conjunction with further regional interpretation of structural controls on mineralization at Laiva. An increased understanding of the structural controls to mineralization is likely to lead to better evaluation of exploration potential and will also assist with the potential definition of high-grade shoots at Laiva.

9.0 EXPLORATION

9.1 Introduction

Under previous management, NORDIC commenced a phased regional geochemical sampling program in 2005, including an initial assessment of historic regional geochemical and geophysical data obtained from previous owners, followed up by surficial till sampling, base of till and bedrock sampling of anomalous zones identified from surficial till sampling, and finally channel sampling at Mussuneva of anomalous zones identified from bedrock sampling and drilling of the Laiva and Oltava prospects.

In 2010, NORDIC, under previous ownership commissioned a detailed mapping and structural geology interpretation, conducted by Specialized Geological Mapping Ltd (Pratt, 2010). In 2016, NORDIC conducted a trial Ground Penetrating Radar (GPR) survey at the Laiva Gold Deposit. The methodology and results of the exploration conducted by NORDIC and the previous operators are summarized below.

Following acquisition by FTR, NORDIC did not conduct additional significant exploration work. Since transfer of the LGP from FTR, OTSO has conducted significant additional drilling, which is also summarized below.

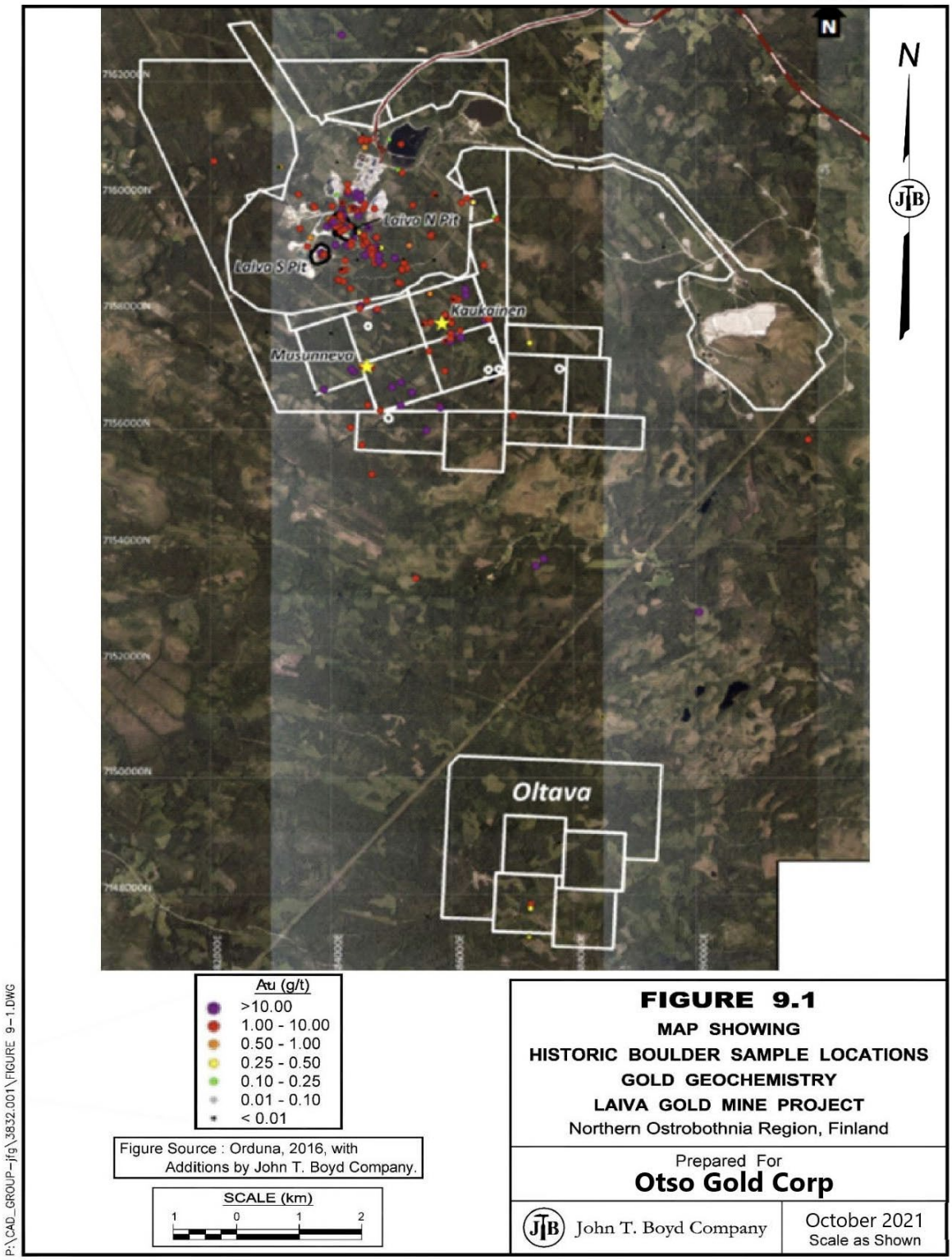
9.2 Boulder Sampling

A database sourced from GTK containing 475 historic surface boulder samples collected by Outokumpu and GTK was used by NORDIC as a regional vector to areas warranting surficial till sampling.

The historic boulder samples were collected as rock chips from mineralized boulders observed at surface and were therefore highly selective. However, this regional prospecting method led to the discovery of the Laiva Gold Deposit (by an amateur prospector) and generation of exploration targets at Mussuneva and Kaukainen. The survey method of sample location for boulder samples is unknown but was likely by compass triangulation.

Boulder samples were analyzed for gold, arsenic, copper, and zinc. The gold geochemistry of boulder samples is shown in Figure 9.1, following this page. The laboratory, sample preparation and analysis methodologies are not known and due to the historic nature of the samples the results cannot be relied upon. However, the results

do provide a useful guide to exploration and have been used by NORDIC to assist with planning of exploration programs.



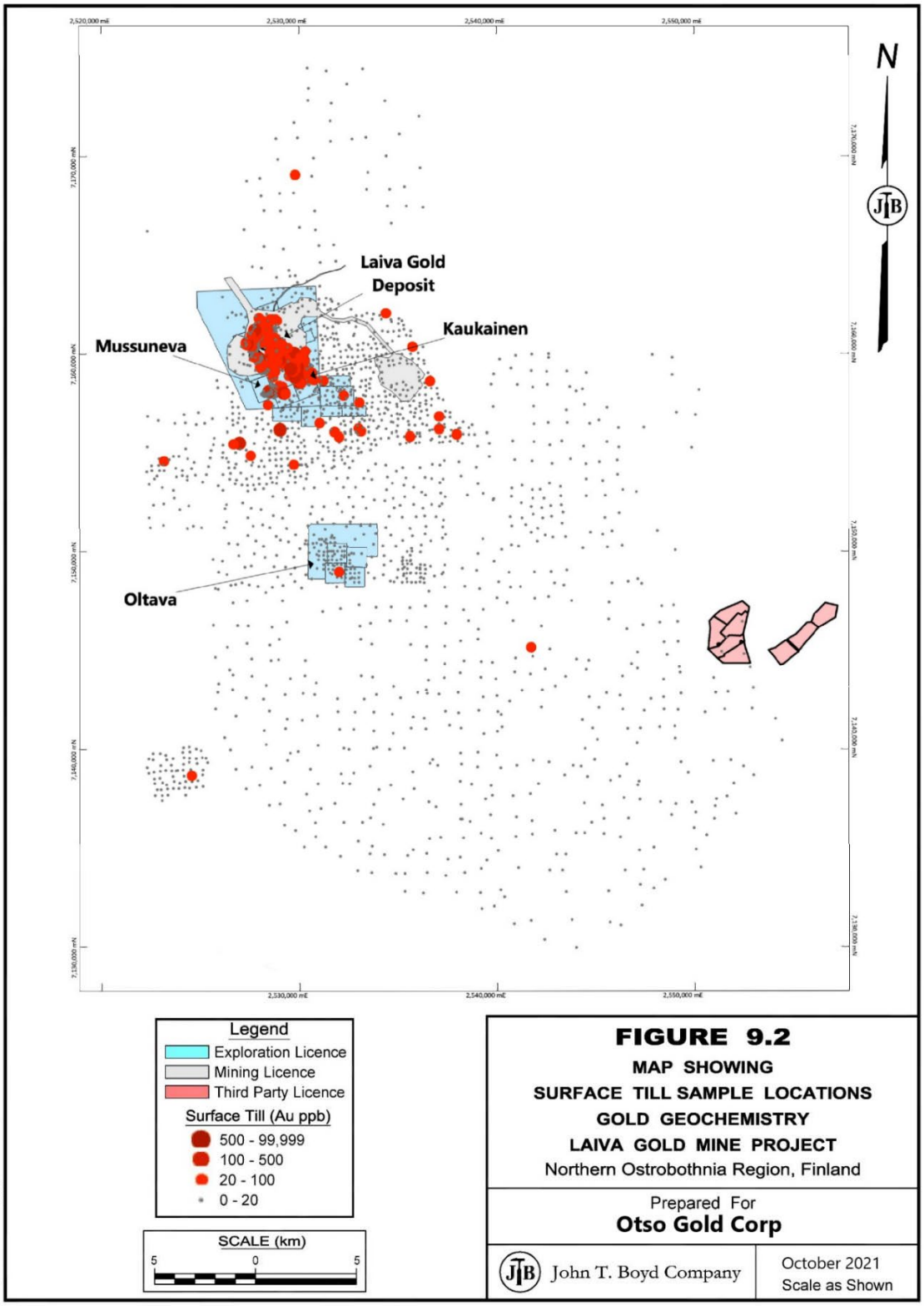
9.3 Surficial Till Sampling

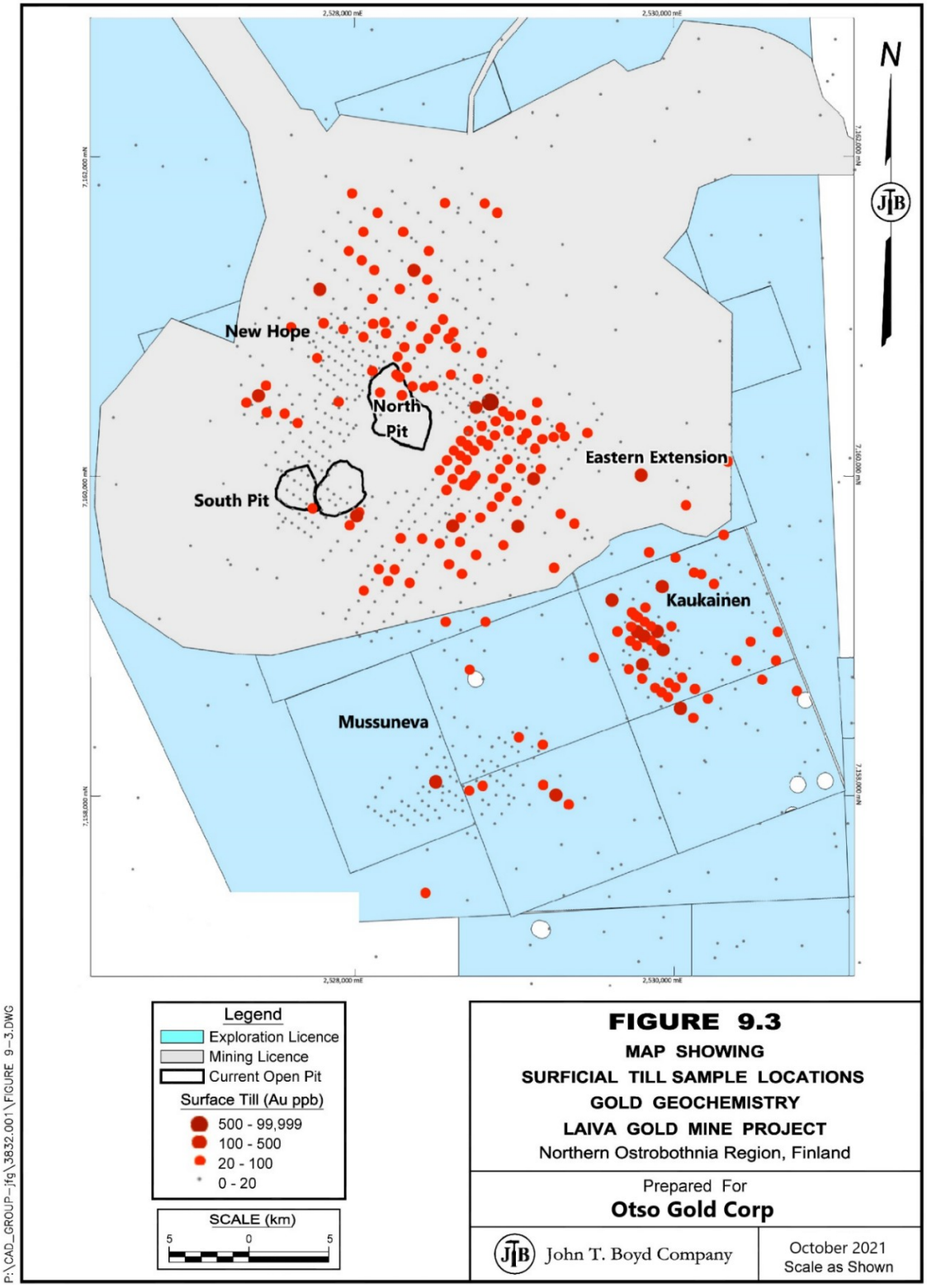
Under previous management, NORDIC collected 2,841 surface till samples over an area of approximately 46 km by 34 km, see Figure 9.2, following this page. Samples were collected on an initial approximate 1 km grid pattern, with sample spacing varying depending on access and ground conditions. Follow up surface till sampling was conducted on a grid spacing of approximately 80 m to 50 m on priority areas at Laiva, Mussuneva, and Kaukainen, see Figure 9.3, Page 9-5. Follow up grids of approximately 400 m to 200 m grid spacing were conducted at Oltava and other unnamed prospects (Figure 9.2).

Samples were collected from hand dug pits to approximately 30 cm deep with each sample weighing approximately 1 kg. Sample locations were surveyed using hand held GPS.

Results demonstrate low-grade gold and arsenic anomalism overlying the Laiva, Mussuneva and Kaukainen prospects, and clearly define distinct zones of gold anomalism at each prospect (Figure 9.3). The southeasterly glacial flow direction is represented by sporadic gold in soil anomalism southeast of the South Pit and Eastern Extension areas at Laiva, and up to 3.5 km southeast of the Mussuneva and Kaukainen prospects (Figure 9.2). A dispersed gold in soil anomaly immediately north of Laiva is also observed in the geochemical data. The cause of this gold anomaly warrants further exploration.

Oltava is less well defined from the surficial till sampling, with only one sample assaying greater than 20 ppb gold (Figure 9.2).

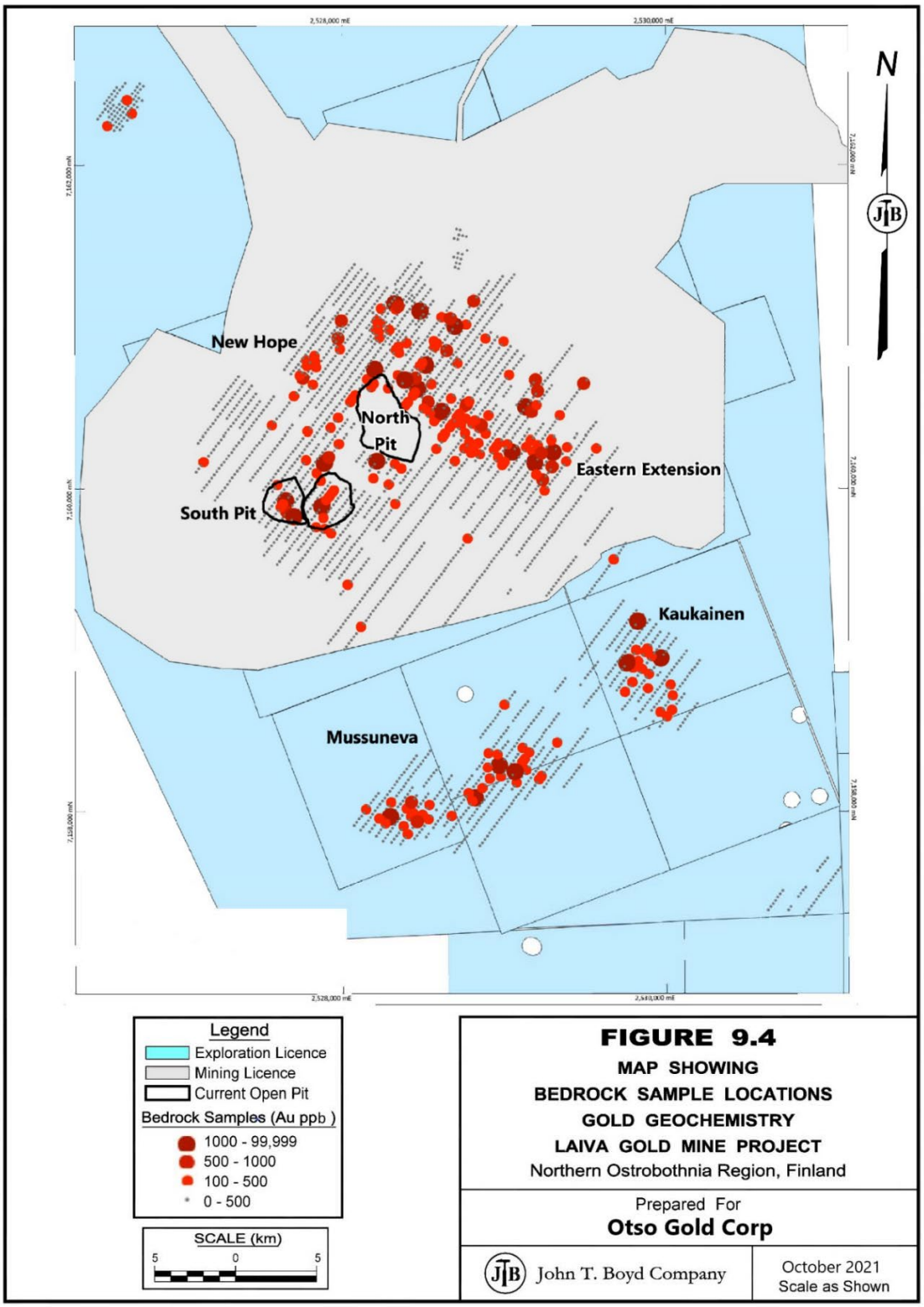

N
Laiva Gold Deposit
Kaukainen
Mussuneva
Oltava
Legend
Exploration Licence
Mining Licence
Third Party Licence
Surface Till (Au ppb)
500 - 99,999
100 - 500
20 - 100
0 - 20
SCALE (km)
5
0
5
FIGURE 9.2
MAP SHOWING
SURFACE TILL SAMPLE LOCATIONS
GOLD GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-2.DWG

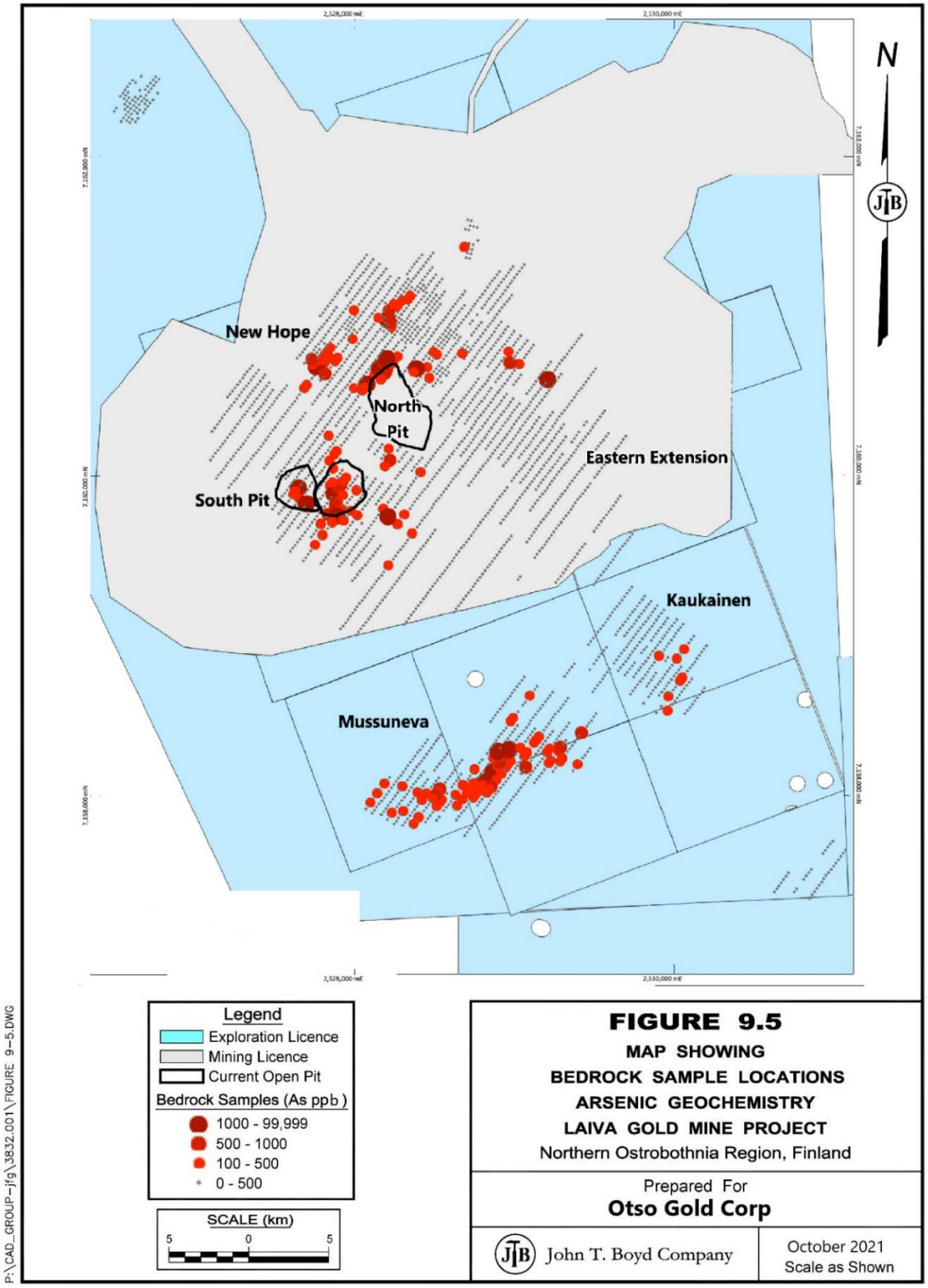

N
New Hope
North Pit
South Pit
Eastern Extension
Kaukainen
Mussuneva
Legend
Exploration Licence
Mining Licence
Current Open Pit
Surface Till (Au ppb)
500 - 99,999
100 - 500
20 - 100
0 - 20
SCALE (km)
5
0
5
FIGURE 9.3
MAP SHOWING
SURFICIAL TILL SAMPLE LOCATIONS
GOLD GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-3.DWG

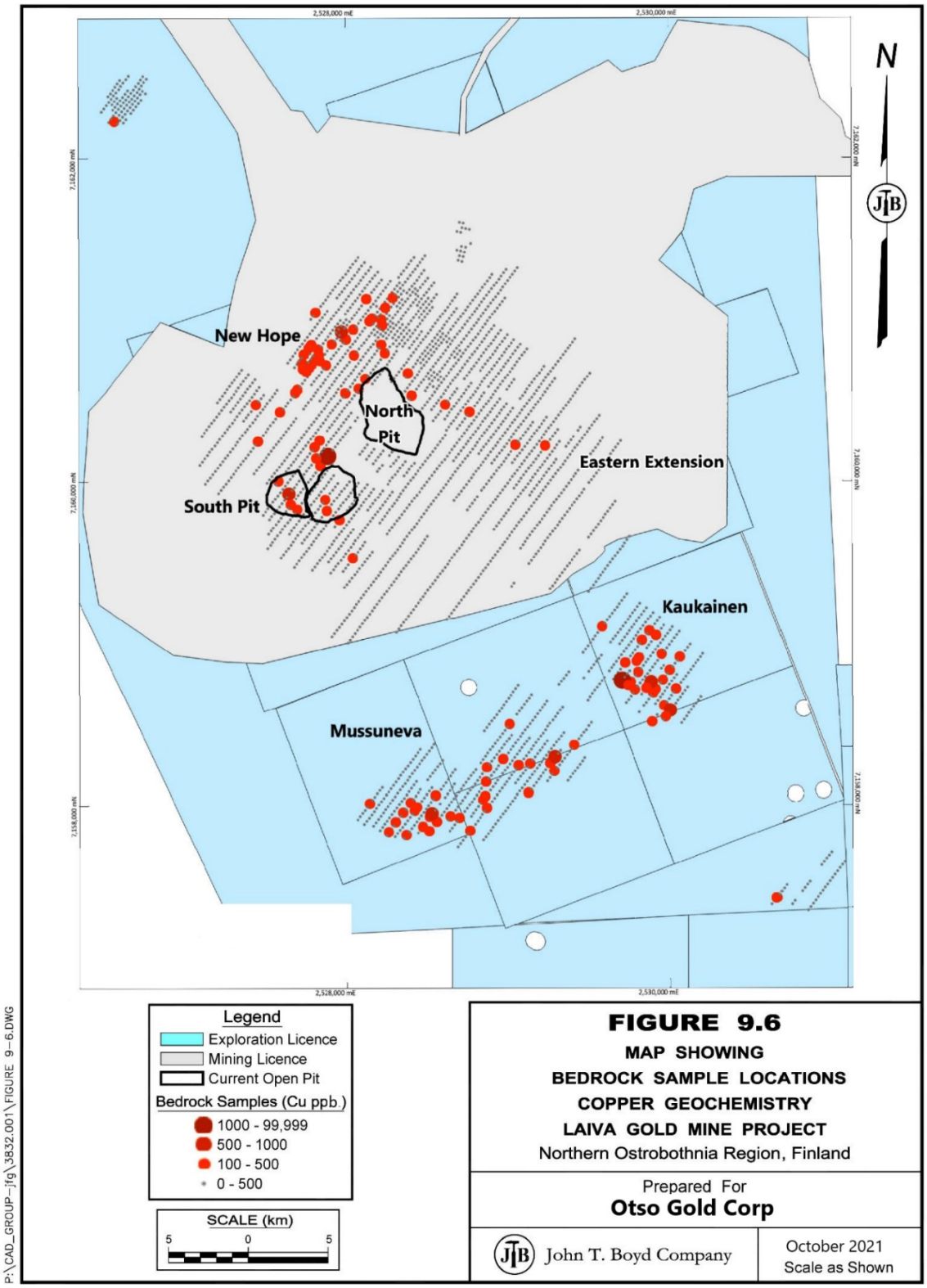
9.4 Base of Till and Top of Bedrock Sampling

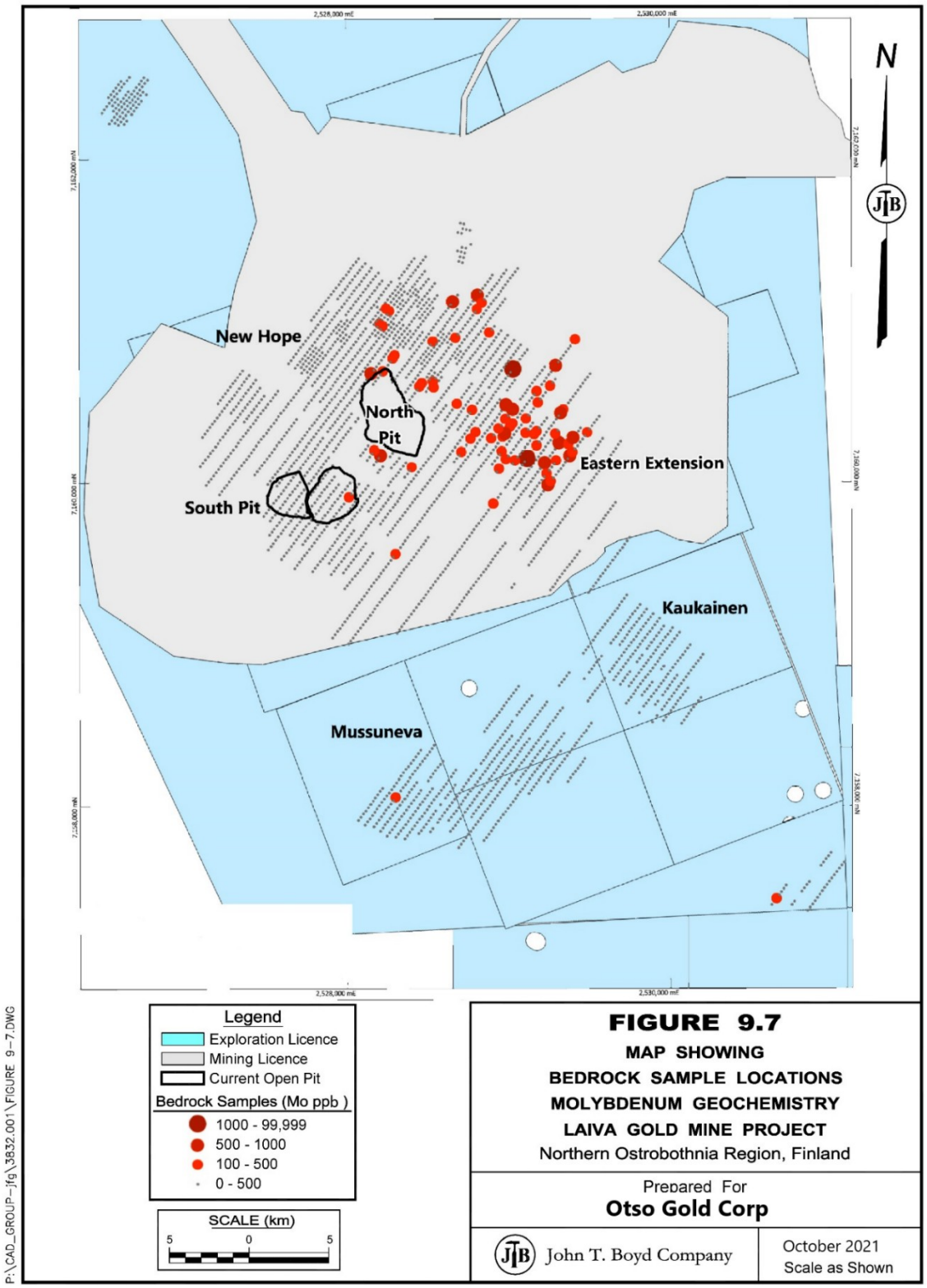
Base of till and bedrock sampling work was conducted by NORDIC purely for regional geochemical exploration purposes and was not used in mineral resource estimation work. Therefore, the methodology and results are discussed in this section.

Base of till and top of bedrock sampling was conducted simultaneously using continuous percussion drilling as follow up on anomalous areas identified from the surficial till geochemistry at Laiva, Mussuneva, Kaukainen and Oltava. The sampling was performed on various grids of 200 m by 25 m, 100 m by 50 m, 50 m by 50 m and occasionally 25 m by 25 m spacing (see Figures 9.4 through 9.7, on the following pages), with samples collected from the bottom meter of till and the first 2 m of bedrock, at which point the hole was abandoned. Holes were drilled to 2 m into bedrock to ensure the hole was not inadvertently terminated in a boulder within the glacial till. Holes were vertical. The lithology, depth, thickness and sample number were recorded, and the collar coordinate was surveyed using hand-held GPS.


N
New Hope
North Pit
South Pit
Eastern Extension
Kaukainen
Mussuneva
Legend
Exploration Licence
Mining Licence
Current Open Pit
Bedrock Samples (Au ppb)
1000 - 99,999
500 - 1000
100 - 500
0 - 500
SCALE (km)
5
0
5
FIGURE 9.4
MAP SHOWING
BEDROCK SAMPLE LOCATIONS
GOLD GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-4.DWG


N
New Hope
North Pit
South Pit
Eastern Extension
Kaukainen
Mussuneva
Legend
Exploration Licence
Mining Licence
Current Open Pit
Bedrock Samples (As ppb)
1000 - 99,999
500 - 1000
100 - 500
0 - 500
SCALE (km)
5
0
5
FIGURE 9.5
MAP SHOWING
BEDROCK SAMPLE LOCATIONS
ARSENIC GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-5.DWG


N
New Hope
North Pit
South Pit
Eastern Extension
Kaukainen
Mussuneva
Legend
Exploration Licence
Mining Licence
Current Open Pit
Bedrock Samples (Cu ppb.)
1000 - 99,999
500 - 1000
100 - 500
0 - 500
SCALE (km)
5
0
5
FIGURE 9.6
MAP SHOWING
BEDROCK SAMPLE LOCATIONS
COPPER GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-jtg\3832.001\FIGURE 9-6.DWG


N
New Hope
North Pit
South Pit
Eastern Extension
Kaukainen
Mussuneva
Legend
Exploration Licence
Mining Licence
Current Open Pit
Bedrock Samples (Mo ppb)
1000 - 99,999
500 - 1000
100 - 500
0 - 500
SCALE (km)
5
0
5
FIGURE 9.7
MAP SHOWING
BEDROCK SAMPLE LOCATIONS
MOLYBDENUM GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-Jfg\3832.001\FIGURE 9-7.DWG

9.4.1 Laiva Area

The bedrock sampling program identified gold, arsenic, and molybdenum anomalies to the east, north, and west of the Laiva North Pit. Between 200 m and 400 m north of the North Pit, two sub-parallel, 500 m long, east-southeast trending gold anomalies 150 m apart are observed (see Figure 9.4, Page 9-7). The western part of this zone displays elevated concentrations of arsenic and lesser copper (Figure 9.5, Page 9-8 and Figure 9.6, Page 9-9). The anomaly was partially tested with six diamond drill holes; however, the area requires further exploration.

The Eastern Extension prospect comprises a 750 m by 180 m, east-southeast trending gold anomaly with coincident but low-grade molybdenum (Figure 9.7, Page 9-10). Arsenic and copper are not associated with the mineralization. This area was followed up with extensive diamond drilling by NORDIC (see Section 10.0).

The New Hope prospect is located 300 m west of the current North Pit and comprises a 150 m by 50 m gold-arsenic-copper anomaly. The prospect was followed up with two channel samples and three exploration drill holes (see Section 10.0). Results returned a maximum assay of 1 m at 4 g/t gold.

The Mussuneva prospect was discovered as a result of the bedrock sampling and is located 1.7 km south of the Laiva South Pit. Mussuneva comprises a significant 1.3 km by 0.3 km, east-northeast trending gold-arsenic-copper anomaly (Figures 9.4 to 9.7, above) hosted in metavolcanic rocks.

The Kaukainen prospect was also discovered as a result of bedrock sampling, located 500 m east-northeast of the Mussuneva prospect and 1.2 km south of the Eastern Extension prospect (Figure 9.4, Page 9-7). The Kaukainen gold-copper in bedrock anomaly is approximately 300 m by 200 m with sporadic, low concentrations of arsenic. A gap in sampling between the Mussuneva and Kaukainen prospects indicates that the prospects are potentially related to the same mineralized structure. The Mussuneva and Kaukainen prospects were drilled by Outokumpu in the 1980s; however, no recent drilling has been conducted.

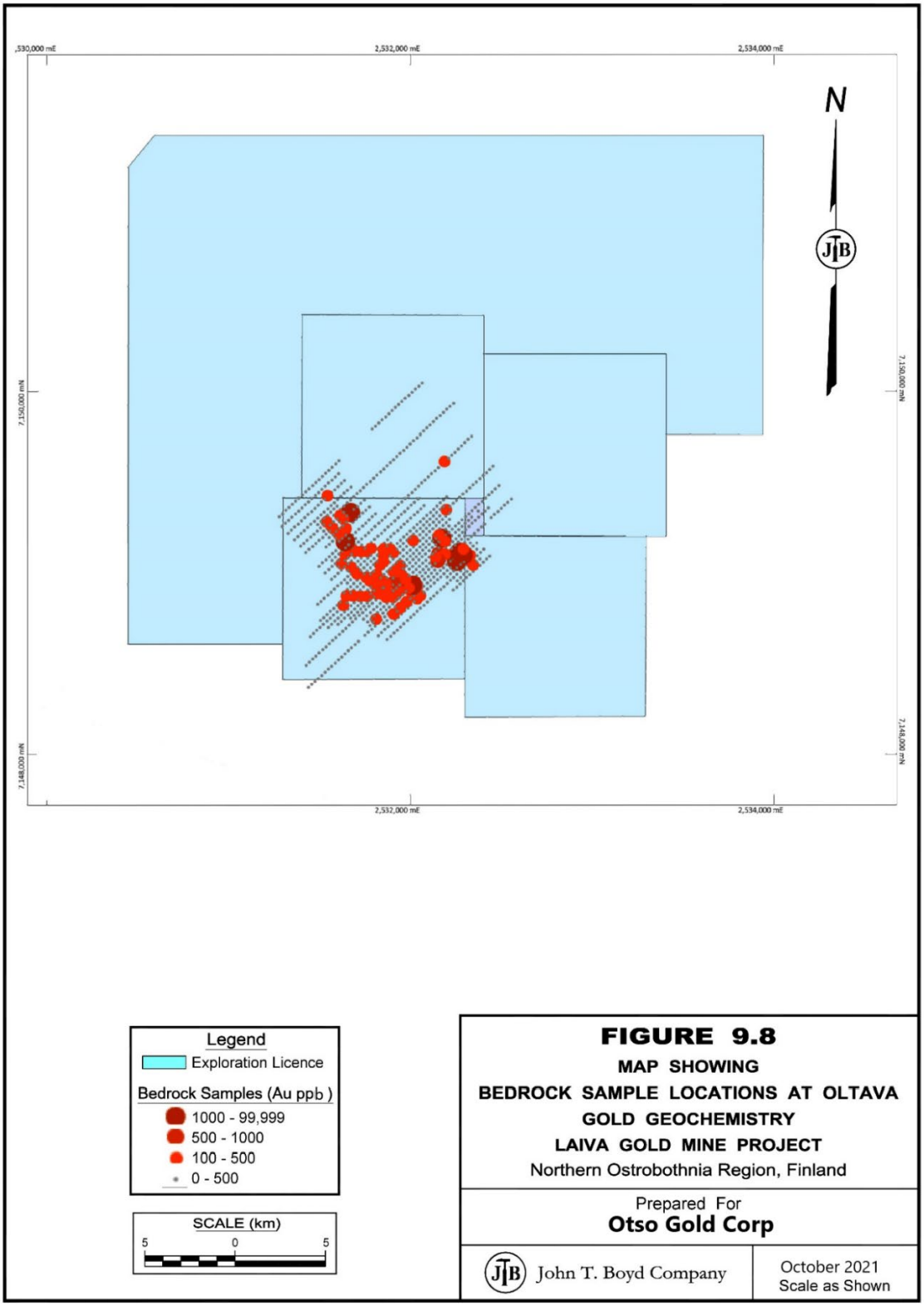
The results of base of till and bedrock sampling at Laiva display a strong correlation between gold, arsenic, and copper anomalism at the Laiva North and South Pit, New Hope and Mussuneva prospects. Strong gold anomalism is observed at the Eastern Extension and Kaukainen prospects; however, arsenic is not anomalous at the Eastern Extension prospect and is weak at the Kaukainen prospect. It is inferred that elevated arsenic and copper is more strongly associated with gold mineralization hosted in

metavolcanic rocks, and that gold mineralization hosted in quartz diorite generally contains lower concentrations of arsenic and copper.

9.4.2 Oltava

Bedrock sampling at Oltava defined two distinct, sub-parallel southeast trending gold anomalies, measuring 650 m by 300 m and 250 m by 150 m (see Figures 9.8 to 9.11, on the following pages). Both zones are located within a wider zone of anomalous arsenic-copper geochemistry, measuring approximately 1.0 km by 0.8 km, and occurring as an envelope to the gold anomalies. A distal molybdenum geochemical signature is observed to the west and northwest of the gold anomalies and is open to the west.

A relatively small (4.6 km2) area of the Oltava Project (total 21.5 km2) has been covered by bedrock sampling. The surficial till sampling did not demonstrate significant gold in soil anomalism despite the significant anomalous results from bedrock sampling. Therefore, it is recommended that the rest of the Oltava license area is sampled at an initial 400 m line spacing with 50 m between samples to explore the potential for mineralization elsewhere at the Oltava Project.


2,530,000 mE
2,532,000 mE
2,534,000 mE
7,150,000 mN
7,148,000 mN
N
JTB
Legend
Exploration Licence
Bedrock Samples (Au ppb)
1000 - 99,999
500 - 1000
100 - 500
0 - 500
SCALE (km)
5
0
5
FIGURE 9.8
MAP SHOWING
BEDROCK SAMPLE LOCATIONS AT OLTAVA
GOLD GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-8.DWG

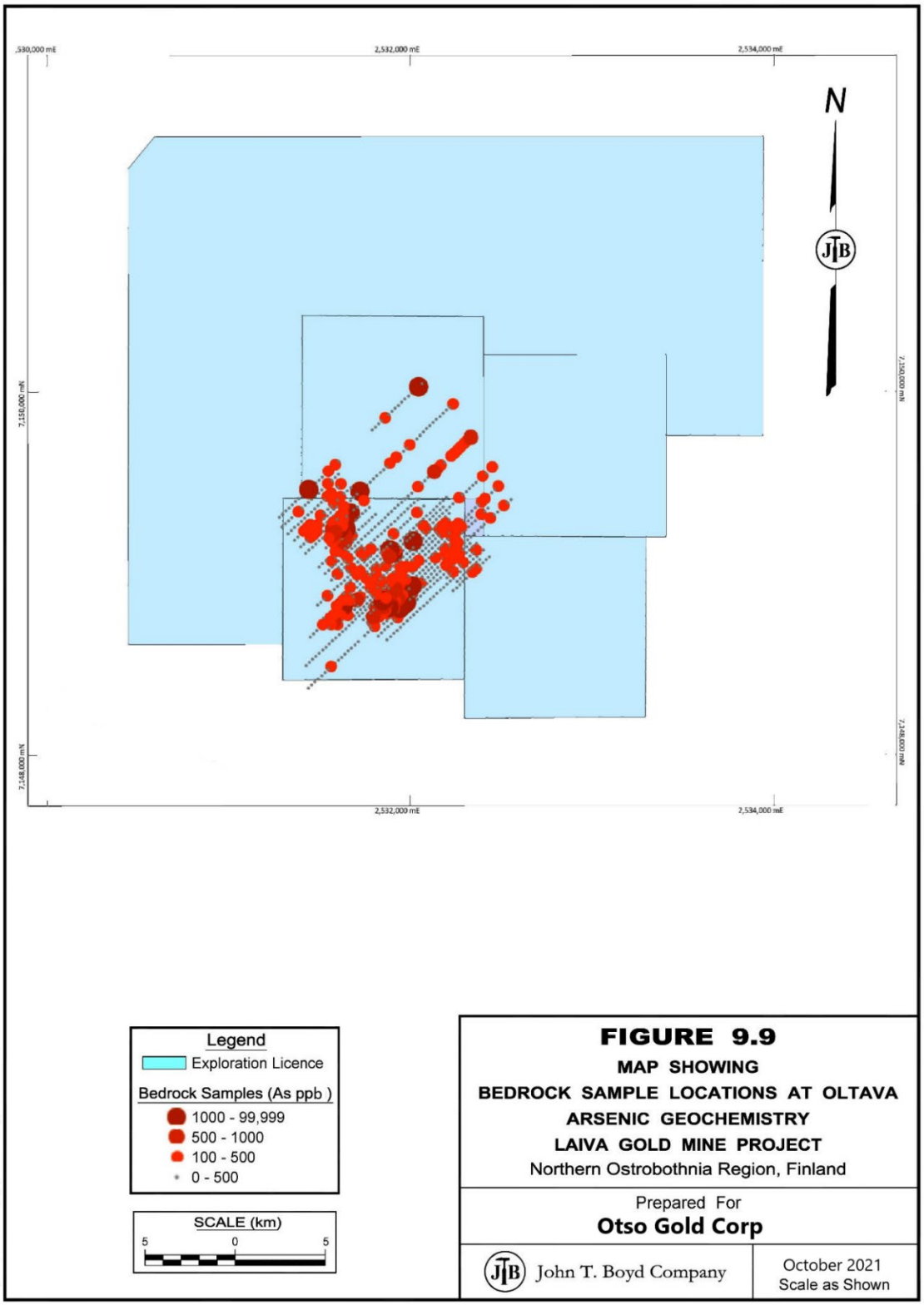


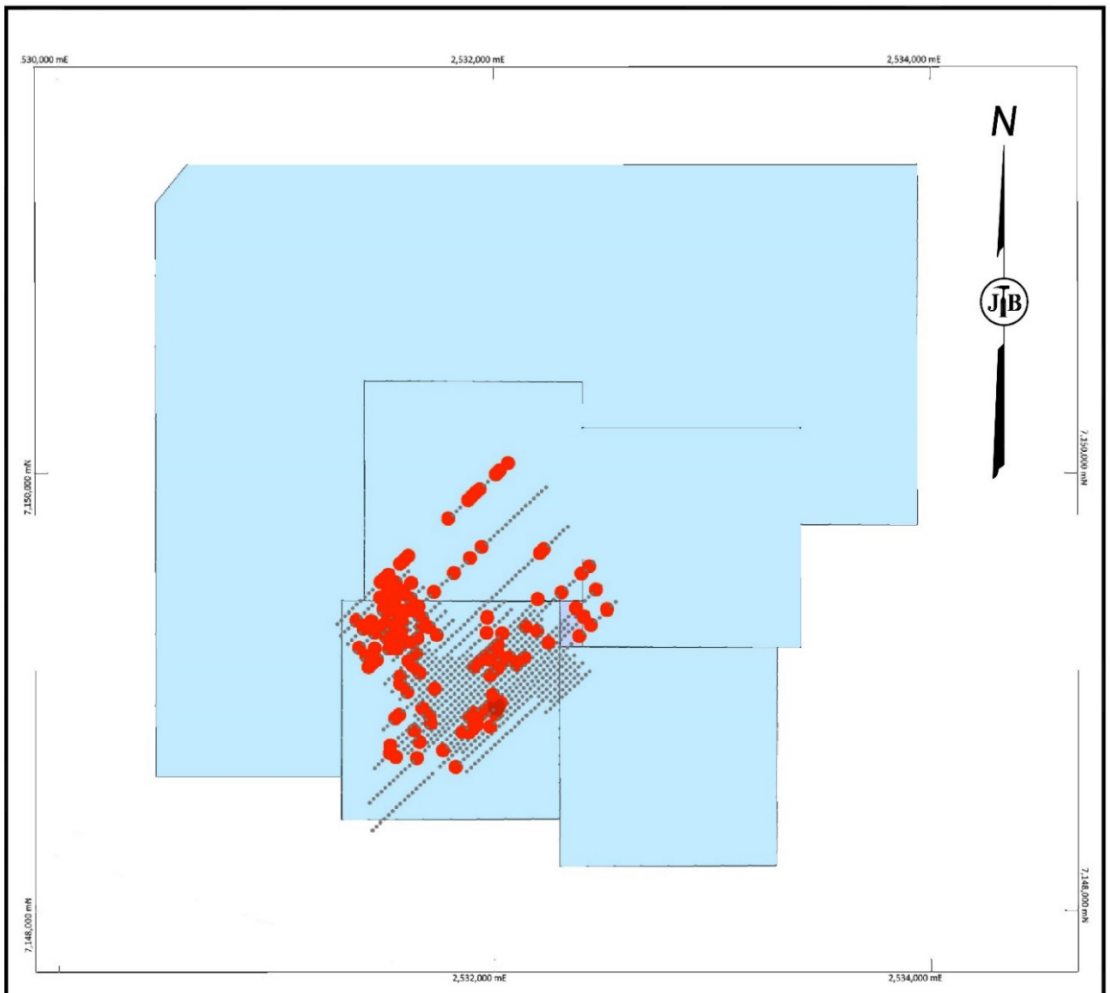
FIGURE 9.9

MAP SHOWING
BEDROCK SAMPLE LOCATIONS AT OLTAVA
ARSENIC GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
Otso Gold Corp

John T. Boyd Company

October 2021
Scale as Shown



Legend
Exploration Licence
Bedrock Samples (Cu ppb)
1000 - 99,999
500 - 1000
100 - 500
0 - 500

SCALE (km)
5 0 5

FIGURE 9.10
MAP SHOWING
BEDROCK SAMPLE LOCATIONS AT OLTAVA
COPPER GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
Otso Gold Corp

John T. Boyd Company

October 2021
Scale as Shown

P:\CAD_GROUP-_jfg\3832.001\FIGURE 9-10.DWG

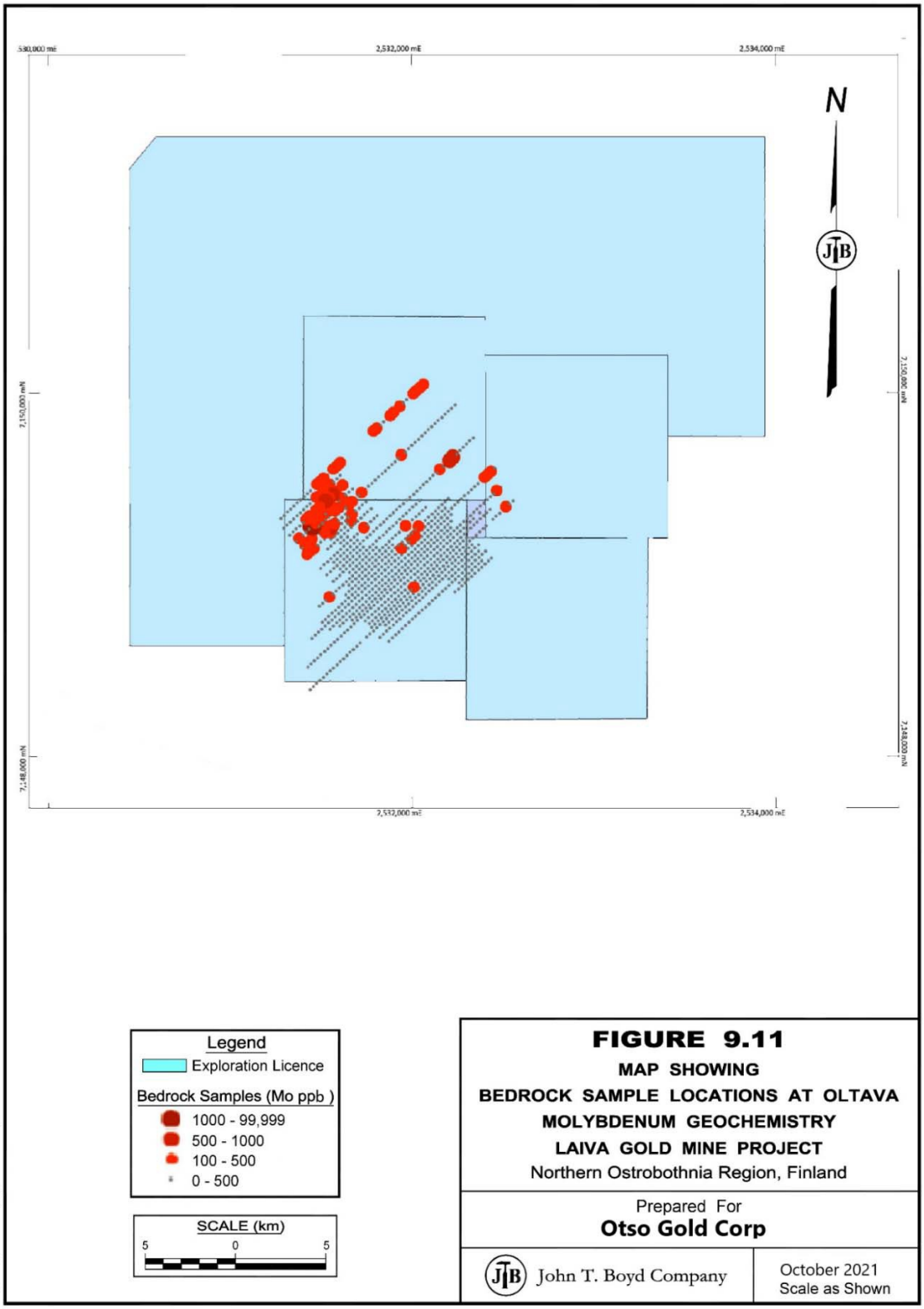


FIGURE 9.11

MAP SHOWING

BEDROCK SAMPLE LOCATIONS AT OLTAVA

MOLYBDENUM GEOCHEMISTRY

LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

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Otso Gold Corp

John T. Boyd Company

October 2021
Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 9-11.DWG

9.5 Channel Sampling

Under previous management, NORDIC conducted stripping of surficial glacial till deposits at the Mussuneva prospect over a small, approximately 50 m by 50 m area to expose the bedrock. Sheeted quartz veins were exposed, and channel sampled using a hand-held rock saw. The channels were cut approximately 5 cm wide and 5 cm deep and were sampled at 1 m intervals.

NORDIC also conducted channel sampling of exposures in the open pits during grade control, using the same methodology described above. The aim of this work was to assist with mapping and determination of gold bearing structures. The channel sampling was not used for resource estimation purposes or grade control purposes, as the channel samples were targeted to specific structures and were not placed at regular spacing or orientations and did not always test the entire width of mineralized structures. The results from these channel samples are therefore considered useful for mapping but are not sufficiently reliable or adequate for mineral resource estimation or grade control purposes due to their highly selective nature.

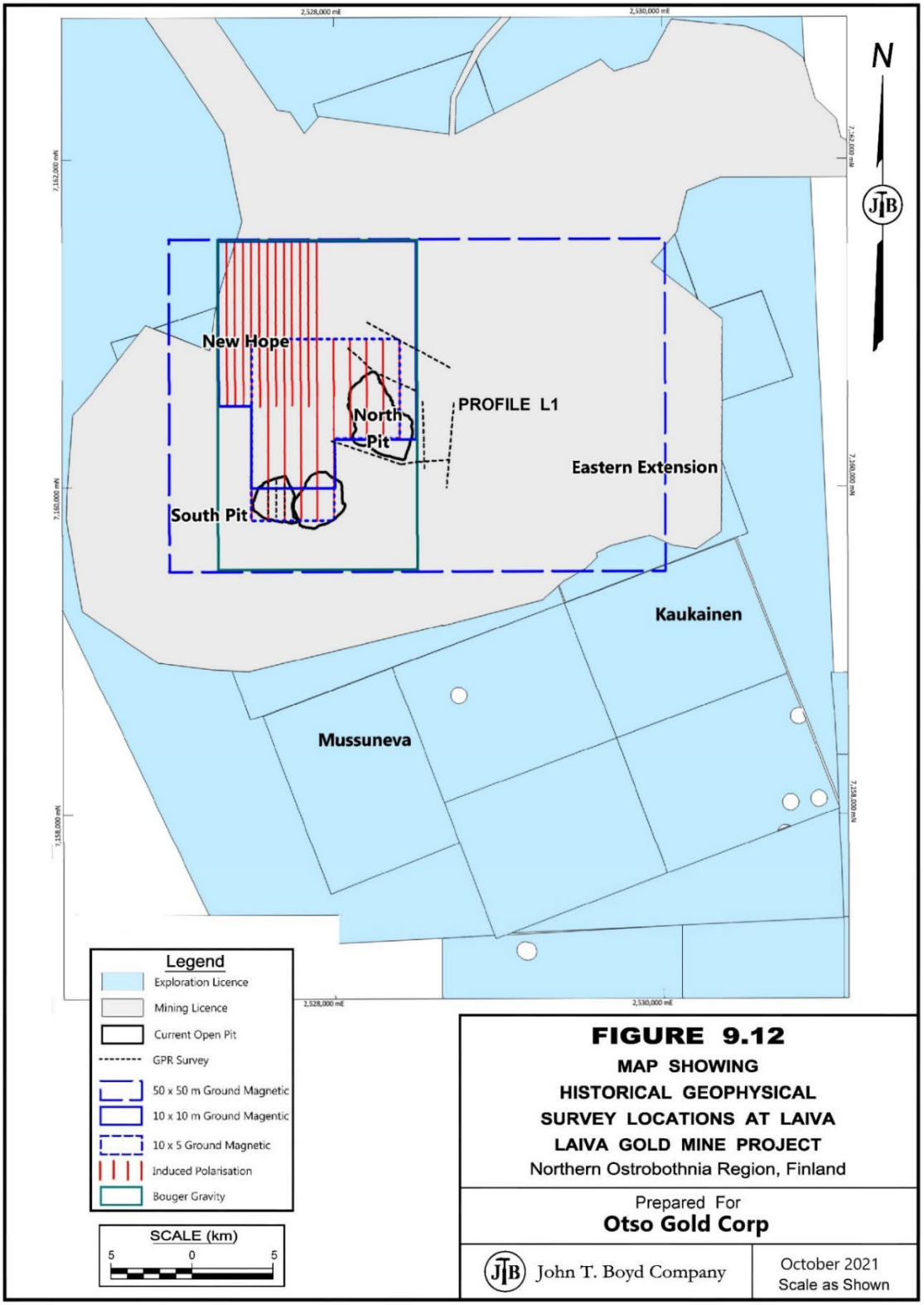
9.6 Geophysical Surveys

Airborne geophysical surveys were flown over the Laiva area by GTK in the 1980s, and Outokumpu commissioned Bouger gravity, magnetic and induced polarization (IP) surveys between 1981 and 1985.

In 2006, NORDIC, under previous ownership, commissioned Astrock Geophysics to compile, process, and interpret geophysical data available from the GTK and Outokumpu datasets. In 2010, NORDIC commissioned a trial GPR survey at the Laiva gold deposit.

The locations of the geophysical surveys conducted at Laiva are shown on Figure 9.12, following this page. The results of the Astrock (2006) interpretation, 2010 GPR trial and BOYDs own observations are summarized below.

BOYD understands that recently OTSO commissioned Adrok PGM Ltd to conduct field testing of its ADR geophysical survey methodology to determine efficacy for future application to enhance drill targeting. The final report dated 4 August 2021 shows promising results with high probability drill targets to be tested.


2,528,000 mE
2,530,000 mE
7,162,000 mN
7,160,000 mN
7,158,000 mN
7,162,000 mN
7,160,000 mN
7,158,000 mN
N
New Hope
North Pit
South Pit
PROFILE L1
Eastern Extension
Kaukainen
Mussuneva
Legend
Exploration Licence
Mining Licence
Current Open Pit
GPR Survey
50 x 50 m Ground Magnetic
10 x 10 m Ground Magentic
10 x 5 Ground Magnetic
Induced Polarisation
Bouger Gravity
SCALE (km)
5
0
5
FIGURE 9.12
MAP SHOWING
HISTORICAL GEOPHYSICAL
SURVEY LOCATIONS AT LAIVA
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland
Prepared For
Otso Gold Corp
John T. Boyd Company
October 2021
Scale as Shown
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-12.DWG

9.6.1 GTK Airborne Surveys

Regional magnetic, radiometric, and electromagnetic surveys were flown across the entire Laiva and Oltava Project areas between 1982 and 1983. The surveys were flown on east-west oriented lines at a spacing of 200 m.

The LGP is located in an area of weak, flat magnetic intensity bound to the east and west by north-south striking magnetic highs. These are inferred to represent metavolcanic rocks to the west and metasediments to the east. Weakly elevated apparent resistivity is coincident with metavolcanic rocks, and strongly elevated apparent resistivity is associated with the metasedimentary rocks to the east of the Laiva Project.

The GTK airborne data provide a useful source of information for regional interpretation of bedrock lithology and structure but are too low in resolution for defining geology and mineralization at the prospect scale. NORDIC, under previous management has used these data to some extent in their exploration planning, and it is recommended that further regional interpretation of the bedrock lithology and structure using the airborne data is conducted to better understand the controls on mineralization and geological setting at Laiva, Mussuneva, Kaukainen, and Oltava. This in turn will lead to a greater understanding of exploration potential in the region in BOYD's opinion.

9.6.2 Bouger Gravity Data

Outokumpu commissioned a 2.4 km2 ground gravity survey located coincident with the North Pit and South Pit at Laiva. The data display a weak correlation between positive residual gravity anomalies and magnetic highs. A north trending positive residual gravity anomaly to the north of the North Pit potentially represents a dyke or other structure but is not associated with mineralization.

9.6.3 Ground Magnetic Data

Three ground magnetic surveys were commissioned by Outokumpu, comprising:

- 3 km by 2 km survey area, comprised of a 50 m line spacing and 10 m station spacing.
- 1.5 km by 1.2 km survey area, comprised of a 10 m spaced line and station survey.
- 1.0 km by 0.9 km survey area, comprised of a 10 m line spacing and 5 m station spacing.

The magnetic data displays positive magnetic anomalies associated with metavolcanic rocks; however, the highest peak values are around 750 nT above the base level and the magnetic anomalies are therefore relatively weak (Lehtonen, 2006). SRK (2014) conducted an initial structural interpretation using the magnetic data, and further detailed interpretation of the magnetic data is recommended by BOYD.

9.6.4 Induced Polarization Data

Dipole-dipole array IP surveys were conducted on north-south oriented lines with a line separation of 100 m, 50 m, or 25 m, with a constant station interval of 20 m. The areas covered by the IP surveys were not ideally placed and only a wide (100 m) spacing was used over the North and South pits. Accordingly, results are inconclusive.

9.6.5 Ground Penetrating Radar Data

Under previous management, NORDIC commissioned a trial GPR survey in 2016, conducted by Terravision Radar Ltd. The survey was conducted over nine profiles for a total line length of 4 km and comprised the use of two antenna spacing of 10 m and 6 m to yield maximum depths of penetration of 200 m and 100 m, respectively. The aim was to test the viability of GPR for mapping structures (faults), differentiating between mineralized and non-mineralized rock, and differentiating between granite and quartz diorite (Howarth, 2016). Terravision was provided with drill hole data and geological maps and models to assist with their interpretation.

Results demonstrate peaks in deeper sections of the profiles which are inferred by Terravision as shadows underlying conductive mineralized bodies. Xenoliths of metavolcanic rock within granite are represented by featureless, low amplitude signals and sub-horizontal reflectors are observed in the presence of shallow dipping granite dykes. Late faults are also inferred from the offset of the GPR signal in profile.

9.6.6 2018 Ground Magnetic Survey

FTR conducted a ground magnetic survey at Laiva using a GSM 19 base station set to 15 second acquire and a GSM19W mobile unit with built-in DGPS set at a 0.2 second acquire. A total of 500 line kilometers were completed over the mine and areas immediately surrounding.

A nominal first pass line spacing of 50 m was used. The program was designed to assist with structural interpretation and distinction of mafic-rich lithologies which typically host the highest-grade mineralization. Unfortunately, results of this survey have apparently been lost during the transition from FTR to OTSO management. If this data can be found

and verified, in BOYD's opinion, this may be helpful in development of priority drill targets.

9.7 Structural Geology Interpretation

In 2010, under a previous operator, NORDIC commissioned Specialized Geological Mapping Ltd to conduct detailed geological mapping of exposures in the North Pit, determine the controls on mineralization, and make recommendations for near-mine and regional exploration (Pratt, 2010). Mapping was conducted by hand on 1:50 and 1:100 scale base maps and digitized by NORDIC staff, using surveyed reference points. Twenty channel samples were also collected and submitted for gold analysis by fire assay and multielement ICP-AES analysis.

Pratt (2010) made several important observations relating to controls on gold mineralization at Laiva, summarized below and incorporated into Section 7.0 (Geology and Mineralization):

- Some of the multiple phases of granitic magmatism were syn-tectonic and possibly a source of metal, with a gold, bismuth, tungsten, tellurium, and molybdenum geochemical signature recognized at Laiva.
- Drag folding deformed the mineralized zones and is principally associated with a dextral shear sense along the Red Shear Zone.
- Mineralization is hosted within mylonitic shear zones, and while the principle orientation of mineralized zones varies between east-southeast and east-northeast, some small scale north-northeast trending quartz veins (the Blue Shear Zone) returned assays of up to 10 g/t gold.

9.8 Results of Exploration

The exploration work programs conducted to date by previous operators demonstrate that top of bedrock sampling is an effective exploration tool for the mineralization styles observed in the Laiva area. Several exploration targets exist proximal to the current mineral resource and operational area, as well as satellite targets at Mussuneva-Kaukainen and Oltava:

- Near-mine exploration potential exists to the north of the North Pit, associated with a northeast trending structure defined by a gold-arsenic anomaly in bedrock sampling. The New Hope area immediately west of the North Pit is small but has demonstrated strong gold-arsenic in top of bedrock sampling. Infill top of bedrock sampling and a detailed ground magnetic survey is recommended for both areas.

- Additional top of bedrock sampling between the Mussuneva and Kaukainen prospects is warranted to infill a gap in sampling between the two prospects and potentially link the two bedrock gold-arsenic-copper anomalies. Detailed ground magnetic surveys are also recommended. This will assist in exploration drill program planning in this area.

- Widely spaced, top of bedrock sampling is warranted at Oltava, where two gold anomalies remain open along strike and a large proportion of the license holding remains untested.

Further studies into the gold grain size distribution, metal associations, structural geology, and timing of granite magmatism relative to mineralization are recommended in order to improve gold recovery, refine the geological model, and potentially develop additional areas of exploration potential.

Results from the recent drilling campaign conducted by OTSO in 2021, along with previous drilling form the basis for the recent BOYD Resource update with an effective date of 4 October 2021. Drilling data to date used in this resource estimate update is presented in Section 10.0 herein.

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10.0 DRILLING

10.1 Introduction

Several phases of drilling have been conducted at Laiva by Outokumpu, Endomines, and NORDIC, under various operators between 1981 and 2014. The majority of drilling has been conducted in the mine area, with a limited number of historic exploratory holes drilled at Mussuneva, Kaukainen, and Oltava (see Table 10.1 below and Tables 10.2, 10.3, and 10.4, following this page).

Table 10.1 – Drilling Campaigns Conducted at Laiva Pre FTR and OTSO

Operator	Period	Series	Type	No. of Holes	Meterage (m)
Outokumpu	1981-1987	LAI01-LAI95	Core	95	6,926
Endomines	1991-2005	LAD1-LAD12	Core	10	772
		Subtotal - Historic Drilling			7,698
Nordic Exploration					
	2005-2011	LAN501-LAN906	Core	209	42,670
	2007-2011	LAN703-LAN953	RC	92	13,350
	2010	1100001-1100040	Core	35	5,075
	2011	LAN11001-LAN11044	Core	30	8,801
	2012	LAN12011-LAN12053	Core	28	3,532
		Subtotal - Nordic Exploration			73,428
Nordic Geotechnical					
	2008	LAG1-LAG4	Core	4	607
	2010	AGMILL1-AGMILL11	Core	11	181
	2010	K200-K206	Core	6	93
	2010	PMILL22-PMILL24	Core	3	49
		Subtotal - Nordic Geotechnical			930
Nordic Grade Control					
	2011	511001-5110572	RC	412	8,143
	2011-2012	51188001-51299434	RC	1419	31,459
	2012	512000241-512010831	RC	47	1,008
	2012	12S001-13S021	RC	51	1,194
	2012	5110088B-512SA893E	RC	17	363
	2013	13S022-13S364	RC	117	2,838
	2013	5130007-5131058	RC	711	16,616
	2013	5130774-5131055	Core	92	3,010
	2013	LS1-LS119	Core	100	3,965
		Subtotal - Nordic Grade Control			68,596
Nordic Blasthole					
	2011-2012	712001-7100237	Blasthole	243	1,733
Total Nordic and others					144,687
Total pre FTR and OTSO Database					152,385

Table 10.2 – Drilling Conducted at Oltava Pre FTR and OTSO

Operator	Period	Series	Type	No. of Holes	Meterage (m)
GTK	1950	OLT001-OLT002	Core	2	204.4
	1951-1955	LUK046-LUK052	Core	7	944.5
		OLT003-OLT030	Core	28	2,756.3
	1971	POL001-POL003	Core	3	322.0
Outokumpu	1981	P7, P10, P20	Core	3	13.5
GTK	1995	LUK310-LUK312	Core	3	93.9
	1998	OLT318-OLT347	Core	36	2,530.6
	2002	OLT348-OLT352	Core	5	860.3
Nordic	2011	OLA001-OLA012	Core	12	1,839.6
Total Pre FTR and OTSO Historic					7,725.4

Table 10.3 – Drilling Conducted at Laiva by FTR

Operator	Period	Series	Type	No. of Holes	Meterage (m)
FTR	2018		RC	47	3,044
Total FTR				47	3,044

Table 10.4 – Drilling Conducted at Laiva by OTSO

Operator	Period	Series	Type	No. of Holes	Meterage (m)
OTSO	2019		Core	22	2,125.0
OTSO	2021		Core	78	7,458.8
Total OTSO					9,583.8
Grand Total Drilling ex blastholes for Laiva				**3,636**	**163,280**

For the purposes of mineral resource estimation, only the holes drilled by NORDIC under various operators have been considered by BOYD. The drilling conducted by Outokumpu and Endomines is considered to be historic (pre-2000) and the data cannot be qualified under NI 43-101 requirements. The data from these historic programs is however, a useful tool in exploration planning.

No drilling was being performed at the time of BOYD's site visits; therefore, the operating procedures have not been observed first-hand by the authors. The drilling database contains exploration, geotechnical, grade control and blasthole data. The exploration and grade control data were used for mineral resource estimation. Only the drilling conducted by NORDIC under various operators which pertains to the mineral resource estimate at Laiva is discussed in the following sections.

10.2 Exploration and Grade Control Drilling

The drilling conducted by NORDIC under all management groups at Laiva was largely inclined at -45 degrees mostly with a northerly azimuth (000 degrees). Some holes were oriented northwest for exploration north of the north pit, and four geotechnical holes were oriented at variable azimuths for design purposes.

Exploration drilling was conducted by NORDIC, under various management groups, on a nominal 50 m by 50 m grid, closing to 25 m by 25 m in some areas of the North and South Pits. Exploration and resource development drilling was conducted using both diamond core holes (85,265 m) and RC holes (78,015 m) for a total of 163,280 m.

Grade control drilling in rare instances included in the above totals was previously conducted on a 12.5 m by 6.5 m grid pattern, inclined at 45 degrees and oriented normal to strike of the mineralized zone. Subsequently, under OTSO, short hole grade control drilling/definition drilling, <u>not included in the Mineral Resource estimate</u>, has been conducted on a 6 m across strike and 8 m along strike of the mineralized zone, as indicated by resource drilling, as recommended by BOYD, and which in BOYD's opinion is adequate for defining mineralized zones relative to bulk waste to minimize mining dilution.

Drilling statistics are shown in Table 10.1, Page 10-1, and Tables 10.2 through 10.4, Page 10-2 (Source: NORDIC).

Exploration and grade control hole collars and drill traces used in the BOYD mineral resource estimate are shown in Figure 10.1, following this page.

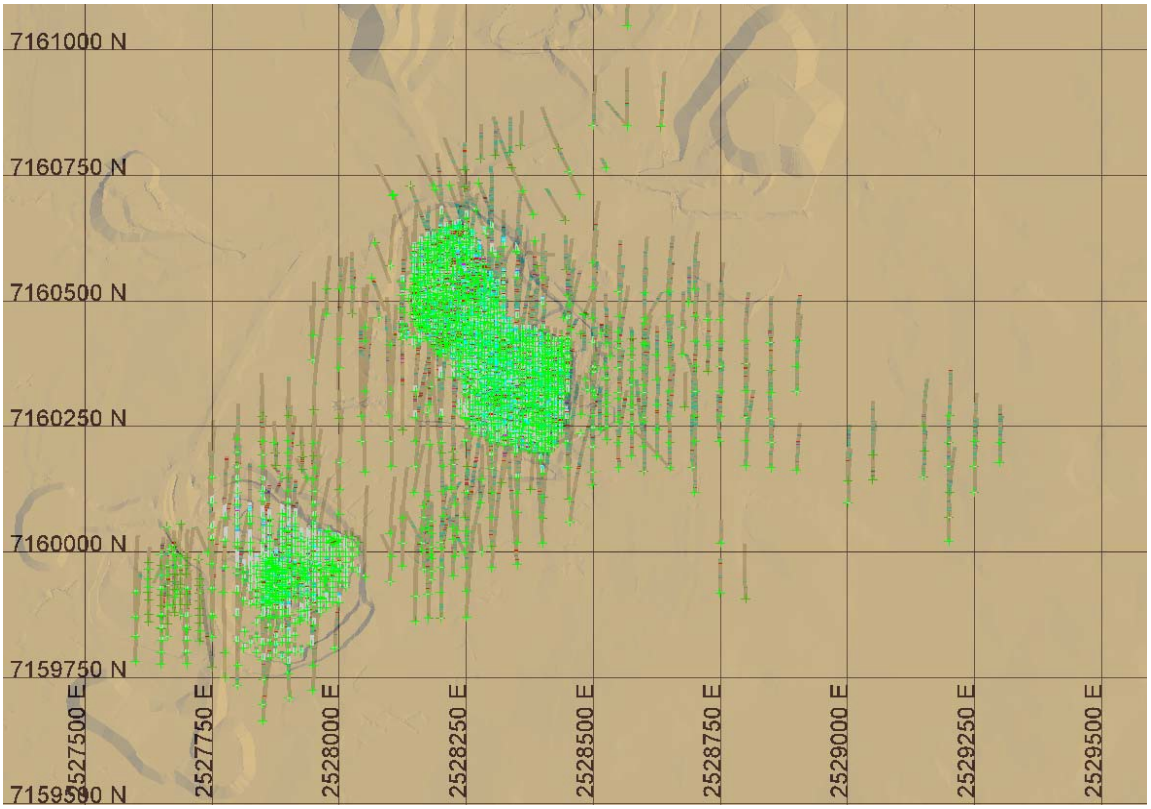

Figure 10.1 – Laiva Drill Hole Collars and Traces

10.3 Diamond Drilling Procedures

Diamond drilling was conducted by contractors using wireline double tube barrels and produced two core sizes over the course of the NORDIC drilling program. Holes LAN501-510 produced a core diameter of 42 mm with the remainder of diamond core drilled by NORDIC using a diameter of 50 mm (NQ2). Run lengths were typically 3 m.

At the end of each run, drill core was placed by the driller into core boxes which were marked with a box number. The driller inserted a block marked with the run depth at the end of each run. The geologist would transport the drill core to the company core logging facility, where each run was fit on a core rack, marked with an orientation line and measured for RQD and recovery calculations. The core was placed back into the core box, photographed, geologically logged and marked up for sampling prior to cutting in half with a core saw. After sampling was complete, the core boxes containing half core were stacked in a covered and locked storage facility owned by NORDIC.

Specific gravity measurements were taken from 1 m sections of whole drill core using the Archimedes water displacement method, whereby the whole core sample was weighed in air and then weighed in water in order to calculate the mass. This is considered appropriate for the style of mineralization, alteration, and host rock at Laiva as the rocks are competent and non-porous. Samples were selected to represent the rock type and mineralization per drill hole.

Samples of half core were collected over 1 m intervals downhole, regardless of lithological contacts or changes in alteration or mineralization. Samples were typically only collected from visually mineralized intervals as well as several meters into the wall rock either side of the mineralized interval. Samples were individually numbered, and a sample number tag was inserted into the core box. The sample was then placed into a numbered plastic bag prior to submission to the laboratory and stored securely at NORDIC's logging facility prior to dispatch.

At the end of hole, collars were cemented with a plug and a PVC pipe, which was labelled with the hole number.

10.4 Reverse Circulation Drilling Procedures

Exploration RC drilling was conducted by a contractor (Järvsö Borr) with a hole diameter of 112 mm. Grade control RC drilling was conducted by several contractors using a hole diameter of 127 mm. Chips were split using a rig mounted riffle splitter with a ratio of 70:30 and samples were collected every 1 m downhole. Rock chips were sieved per sample and retained in a chip tray for logging. Samples were collected in cloth bags and labelled with a sample number by the geologist and stored on a pallet at the rig before being transported to the company's secure logging facility prior to dispatch to the laboratory.

10.5 Sample Recovery

Diamond drill core recovery was routinely measured during core logging and recorded in the detailed descriptions of the lithology database. Core recoveries are typically greater than 95% and often 100%, with no bias observed between core recovery and gold grade. A visual inspection of core and core photos by BOYD confirmed the excellent diamond drill core recovery.

Recovery during both exploration and grade control RC drilling was not routinely measured or recorded; however, the geologist was reported to be on site during drilling and would note where loss was suspected. No significant zones of core loss are reported within mineralized intervals and there is no bias between RC chip recovery and gold grade.

10.6 Collar Surveys

For exploration drill holes, planned hole coordinates were provided to the surveyor and the collar was located by GPS. A backsight and foresight was also marked by the surveyor. Following completion of each exploration hole, the collar was typically surveyed by VRS GPS to a high degree (mm scale) of accuracy and occasionally by total station survey.

Planned grade control collars were located in the pit by surveyors using differential GPS and checked following completion of the hole. The accuracy of drill hole collar surveys is considered sufficient for mineral resource estimation purposes.

10.7 Downhole Deviation Surveys

Downhole deviation surveys were conducted on exploration drill holes using Reflex equipment and were typically performed following completion of the hole with measurements taken at 3 m to 5 m intervals as well as at the end of the hole.

Downhole deviation surveys were only rarely conducted on grade control drill holes; however, the results demonstrate that the deviation is minor for grade control holes due to the relatively short (<30 m) downhole depth. Downhole deviation surveys were conducted using Reflex equipment with measurements taken at 3-m intervals as well as at the end of the hole.

10.8 Core Orientation Surveys

Core orientation surveys were conducted sporadically throughout the exploration diamond drill program using the spear method. The core is largely unoriented. Orientation measurements recorded the downhole depth, alpha and beta angle, and description of each feature. The quality of orientation measurements is reported by NORDIC and SRK (2014) to be low due to the use of the spear method. However, holes drilled under OTSO management were all oriented using highly accurate Reflex ACT equipment.

Four geotechnical holes (LAG1 to LAG4) were drilled for pit design and were oriented.

10.9 Logging

The diamond drill core was geologically logged to record lithology, color, grain size, texture, alteration and mineralization, structure, and quartz vein density. Intervals greater than 20 cm were generally recorded as a separate unit. RC chips were logged to record lithology, color, grain size, texture, alteration, and mineralization.

Exploration drill core was also geotechnically logged to measure the RQD and estimate the Rock Tunneling Quality Index (Q-value).

Logging was conducted directly into a digital database and there are no hard copies of logging sheets.

10.10 Database

Under all operators, NORDIC's drill data is maintained in a Microsoft Access database, which is administered by the Exploration Manager. Geologists enter the collar, survey, sample location, geotechnical and geological logging data into a copy of the database, which is then appended to the master database by the Exploration Manager. Assay results returned from the laboratory were imported to the Access database by automatically matching the sample numbers.

10.11 Twin Holes

No twin holes were specifically conducted by NORDIC to compare results between diamond core and RC drilling. In order to assess the reliability of RC grade control data, BOYD selected RC grade control holes that had been drilled within 5 m of a diamond exploration hole and compared the width and gold grade of mineralized intervals for each pair. Eight pairs were selected and demonstrate good correlation, with no evidence of bias in interval width or gold grade between the two drilling methods. The results do demonstrate local variability in gold grade, which is to be expected with the style of mineralization observed at Laiva and has been considered in the mineral resource estimation methodology and mineral resource classification (see Section 14.0).
In 2009, CSA Global conducted a statistical analysis of 1,102 exploration RC results versus 1,085 exploration diamond hole results and concluded that the results were comparable and that results from both drilling techniques were equally suitable for mineral resource estimation purposes. Following a review of the data, BOYD concurs with this assessment.

10.12 Trial Grade Control Program 2014

In 2014, under a previous operator, NORDIC undertook a trial grade control program under the supervision of SRK (UK). The aim was to evaluate the optimum sample spacing for grade control purposes and to compare the results between RC chip sampling and blasthole sampling.

Grade control drilling between 2012 and 2013 was conducted at a line spacing of 12.5 m with 10 m between holes on each section. The 2014 trial program infilled the historic grade control drilling to a spacing of 6.25 m by 5 m in two areas, one in the North Pit and one in the South Pit. A total of 85 holes for 2,121 m were drilled in the North Pit trial area, and 89 holes for 1,941 m were drilled in the South Pit trial area. Blastholes were also drilled in the same trial locations at a nominal spacing of approximately 3 m by 3 m, totaling 264 holes for 2,640 m in the North Pit and 364 holes for 3,640 m in the South Pit.

The results from the above RC and blasthole infill drilling were compared with results from the earlier wider-spaced grade control drilling by modelling the results and interpolating grade using several methods:

- Modeling of veins to create wireframes using Leapfrog software.
- Modeling of grade shells using Leapfrog software.
- Interpolation of grade into (1) a single domain and (2) separate grade domains of <0.6 g/t and >0.6 g/t gold, combined to give a percent weighted block gold grade.

The above methods were applied to results obtained from RC and blasthole sample data for comparison of results between RC and blasthole data as well as against the mineral resource estimate. Results demonstrated that blasthole data allowed for more accurate estimations of grade and tonnage compared to the wider spaced RC drilling, and that the previous grade control spacing of 12.5 m by 10 m was inappropriate. In BOYDs opinion, this is indicative of local gold grade variability and reinforces the interpretation by BOYD that mineralization occurs in discrete, relatively narrow mineralized shear zones. This interpretation therefore necessitates a more tightly constrained geological model than previously employed.

10.13 Results of Drilling Programs

Drilling by NORDIC has defined gold mineralization at the Laiva Gold Mine over an area of 1,300 m by 600 m and to a maximum depth of 250 m below surface. The drilling and sampling procedures have been conducted to industry standards and the results of the drilling program are considered adequate for mineral resource estimation purposes.

10.14 Comments on Section 10

Overall the drilling, logging, and sampling procedures as well as the database provided are considered reliable and adequate for mineral resource estimation purposes.

BOYD strongly recommended at the time that future sample intervals are determined by lithology and presence of mineralization. While a nominal sample length of 1 m is generally acceptable, the sample interval should be further determined by the geologist, where a sample is terminated at a lithological contact or change in mineralization or alteration. This would allow better granularity in assessment of assay results and provide for a more accurate estimate of gold grade and mineralized interval width in proximity to post mineral intrusive units.

Oriented core should be obtained as regularly as possible for infill drill holes in order to better define the structural controls on mineralization and provide for more accurate geological modelling of the mineralized zones, dikes, and faults.

All of the foregoing recommendations have been adopted for drilling conducted by OTSO.

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11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

11.1 Introduction

Under the first operator, NORDIC employed the Labtium Oy, Finland (formally GTK Geolaboratory) and ALS Chemex (Sweden) (ALS) laboratories for sample preparation and analysis for all exploration samples, including channel samples, diamond drill core and RC chips. Both laboratories are independent of NORDIC and are certified under the following schemes:

- GTK Geolaboratory/Labtium Oy: T025 (EN ISO/IEC 17025), Finnish Accreditation Service (FINAS).
- ALS Chemex (Sweden): ISO/IEC 17025, SWEDAC Ackreditering.

Grade control samples were prepared and analyzed at NORDIC's on-site laboratory, which was managed by independent contractors (CRS). This facility is not certified.

Under FTR and OTSO operation of NORDIC, both used CRS Laboratories located in Kempele, Finland, for sample preparation and CN soluble and ICP analysis, with on shipment to MSALABS Ltd. (ISO 9001) in Langley, BC, Canada, for 30 g fire assay with AA finish.

11.2 Chain of Custody

Under previous ownership, exploration samples were reportedly held in custody of the site geologist until return to the exploration office, where samples were stored in locked sheds until dispatch to the preparation laboratory by courier. Upon delivery to the preparation laboratory, samples were receipted, and a report was sent to the company by electronic mail confirming the sample numbers and safe receipt of samples. Under FTR and OTSO management, chain of custody was similar, but samples were dispatched in sealed containers to the nearby CRS lab by company truck. A dispatch sheet was submitted to CRS and the sample numbers were checked by CRS upon sample receipt.

Grade control samples were retained in custody of the grade control geologist on-site and samples were transported by company vehicle to CRS for sample preparation and analysis. A dispatch sheet was submitted to CRS and the sample numbers were checked by CRS upon sample receipt.

Samples were submitted to the laboratories in batches of variable quantities, rather than in batches of a standard number of samples.

In BOYD's opinion, the above chain of custody arrangements maintained the validity and integrity of samples prior to analysis.

Chain of custody under both FTR and OTSO was handled similarly to previous practices. BOYD has reviewed practices by both FTR and OTSO and can confirm appropriate chain of custody under both entities.

11.3 Sample Preparation

Labtium and ALS prepare samples using similar, industry standard methods, comprising drying, jaw crushing, sub-sampling using a riffle splitter, pulverizing the split sample and retention of sub-sample(s) for analysis(es). Grade control samples were dried at 105°C, with a 500 g sub-sample collected by mat rolling and channel cutting of the chip sample. Grade control samples were not crushed or pulverized prior to analysis. The specific details of sample preparation at Labtium and CRS/MSA are shown in Figure 11.1, following this page, and Figure 11.2, Page 11-4, respectively.

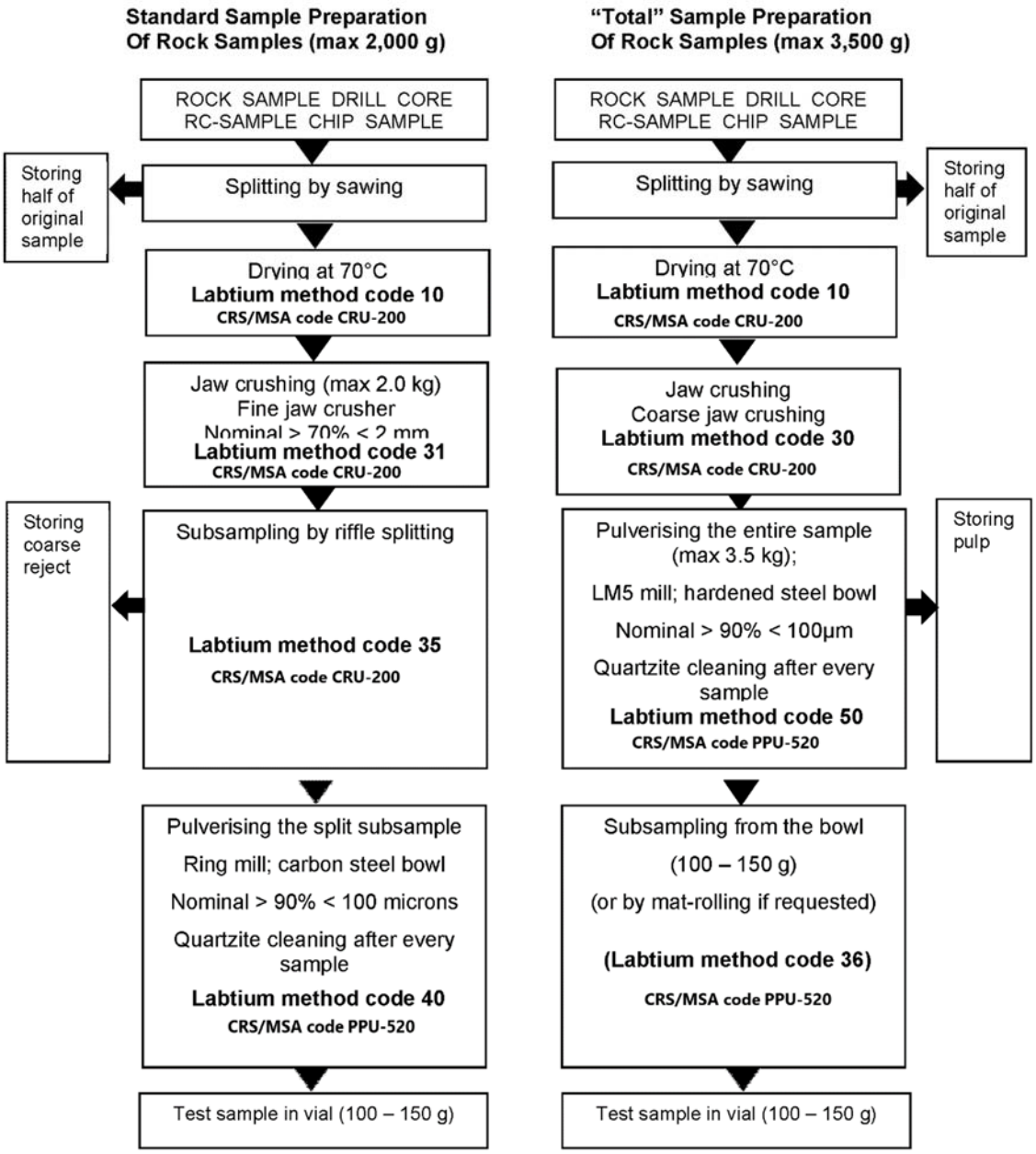


Figure 11.1 – Sample Preparation Flow Sheet

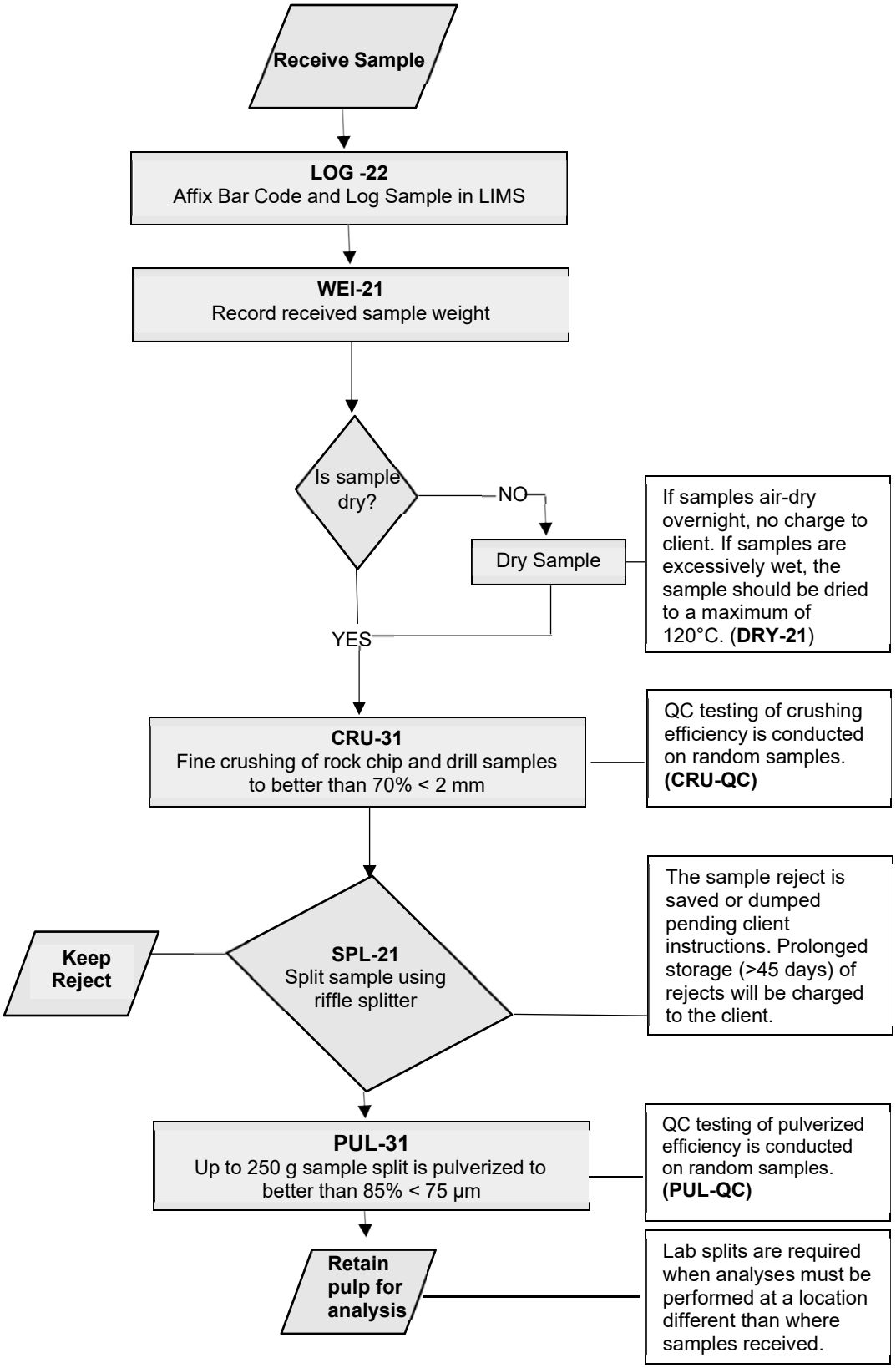


Figure 11.2 – Sample Preparation Flow Sheet (ALS Chemex)

11.4 Analyses

Exploration samples were routinely analyzed for gold by 50 g fire assay and multi-element ICP/AES. The codes, methods and elements analyzed at Labtium and ALS Chemex are summarized in Table 11.1 below.

Table 11.1 – Analytical Methods

Analytical Methods and Codes for Exploration Samples

Laboratory	Analytical Code	Analytical Method	Elements
Labtium Oy/GTK	510P	Four-acid digest ICP-AES	Ag, As, Cd, Co, Cr, Cu, Fe, Mn, Mo, Ni, Pb, S, Sb, Zn
	705P	50 g Fire Assay with ICP-AES finish	Au
CRS/MSA	ICP-130	Four-acid digest ICP-AES	35 element ICP
	FAS-211	30 g Fire Assay with AA finish	
	PALO-AAS	CN leach w AA finish	
ALS Chemex	ME-ICP61	Four-acid digest ICP-AES	Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn
	Au-ICP22	50 g Fire Assay with ICP-AES finish	Au

Grade control samples were assayed for gold only, using a pulverize and leach machine (PAL1000). The PAL1000 contains 52 steel pots. Each pot was filled with a 500 g sample split, 2.6 kg of grinding media and 500 ml of water with 7.5 g sodium cyanide, LeachWell and sodium hydroxide. The samples were simultaneously leached and pulverized for 2.5 hours at 60 rpm. The solution was then analyzed for gold by AAS. The steel pots were washed, and the grinding media replaced for the next sample.

11.5 Quality Control and Quality Assurance

Quality Control and Quality Assurance (QAQC) of exploration samples was undertaken through the insertion of blanks and CRM into the sample stream at random intervals. The blank and CRM material was selected by the laboratory, as was the frequency of QAQC sample insertion.

NORDIC did not insert independent, blind blank or CRM into their exploration sample submissions and have relied on the QAQC procedures conducted by the analytical laboratories. NORDIC conducted field duplicate analysis on select half drill core samples

as well as an umpire test between the Labtium and ALS laboratories on mineralized half core samples.

QAQC of grade control samples was undertaken by CRS and included insertion of two pulp duplicate samples and two CRM samples in every run of 52 samples. NORDIC did not insert any blind blank or CRM samples, nor did NORDIC conduct any duplicate analysis between 2011 and 2013. During a trial RC grade control program in 2014, NORDIC inserted two duplicate samples, two CRM samples and a blank sample in every run of 52 samples.

The results of the QAQC programs are summarized below.

11.5.1 Blanks

11.5.1.1 Labtium and ALS Blanks

In total, 1,708 blank samples were analyzed for gold by Labtium, shown in Figure 11.3 below, and 381 samples were analyzed for gold by ALS. The maximum blank assay in Labtium data is 0.3 g/t gold and the maximum blank assay in ALS data is 0.003 g/t gold. In total, 97% and 98% of blank samples assayed at Labtium and ALS, respectively, assayed either below detection limit or within two times the detection limit.

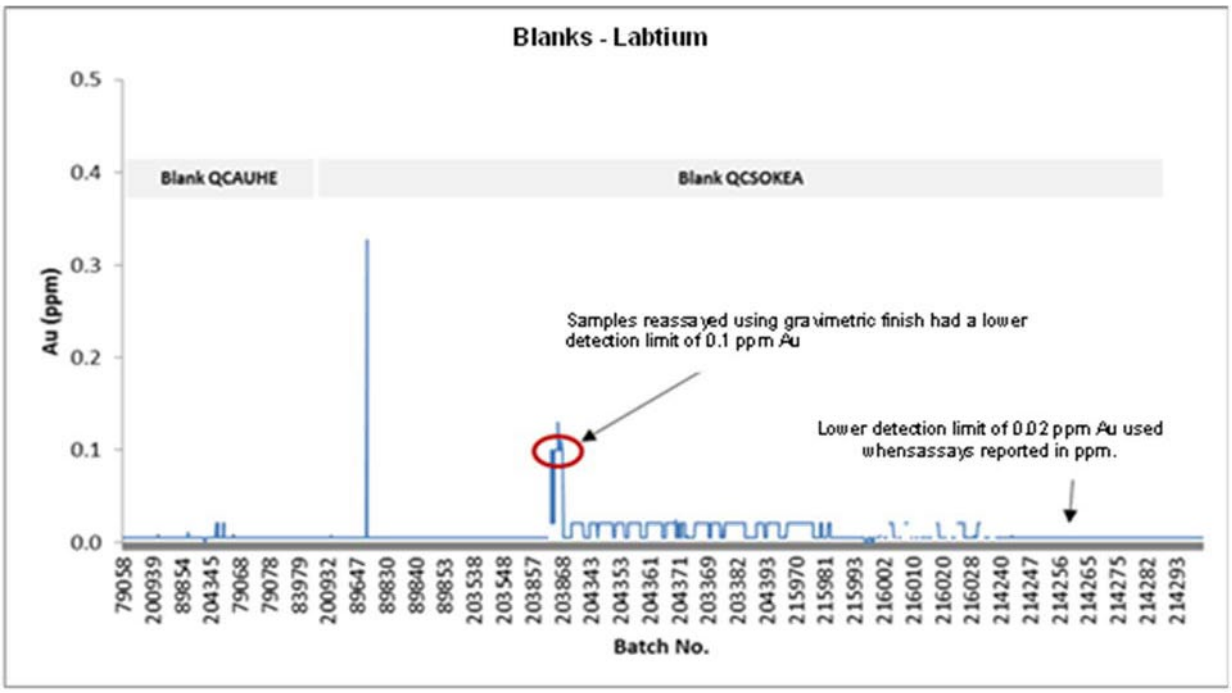


Figure 11.3 – Labtium Blank Results

The results demonstrate that no significant contamination occurred during sample preparation. However, the batch containing a blank sample with 0.3 g/t gold should have been re-assayed in order to determine the potential for contamination of other samples in the batch.

Further, as the blank material and location within a batch was selected by the laboratory, it is difficult to fully assess the potential for contamination during preparation of high-grade samples. In future, it is recommended that the operator inserts blind blank samples after suspected high grade gold samples, in order to assess the potential for contamination during sample preparation with greater vigor.

11.5.1.2 CRS Blanks

Between 2011 and 2013, no blanks were analyzed by CRS during the grade control sampling. In 2014, 205 blank samples were analyzed by CRS and all results were below detection limit (0.1 ppm gold).

11.5.2 Certified Reference Material

Labtium, ALS, and CRS utilized a mix of internal gold CRM and external, commercially available gold and base metal CRM. Only the gold CRM results have been assessed by NORDIC historically and in this study.

In order to assess the CRM results, BOYD collated the gold assay results for each CRM and plotted the results against the certified mean and ±2 and ±3 standard deviations from the mean, where this data has been provided. In instances where a certified mean and standard deviation value was not available, BOYD calculated the laboratory mean and standard deviation to derive the ±2 and 3 standard deviations from the mean for each CRM.

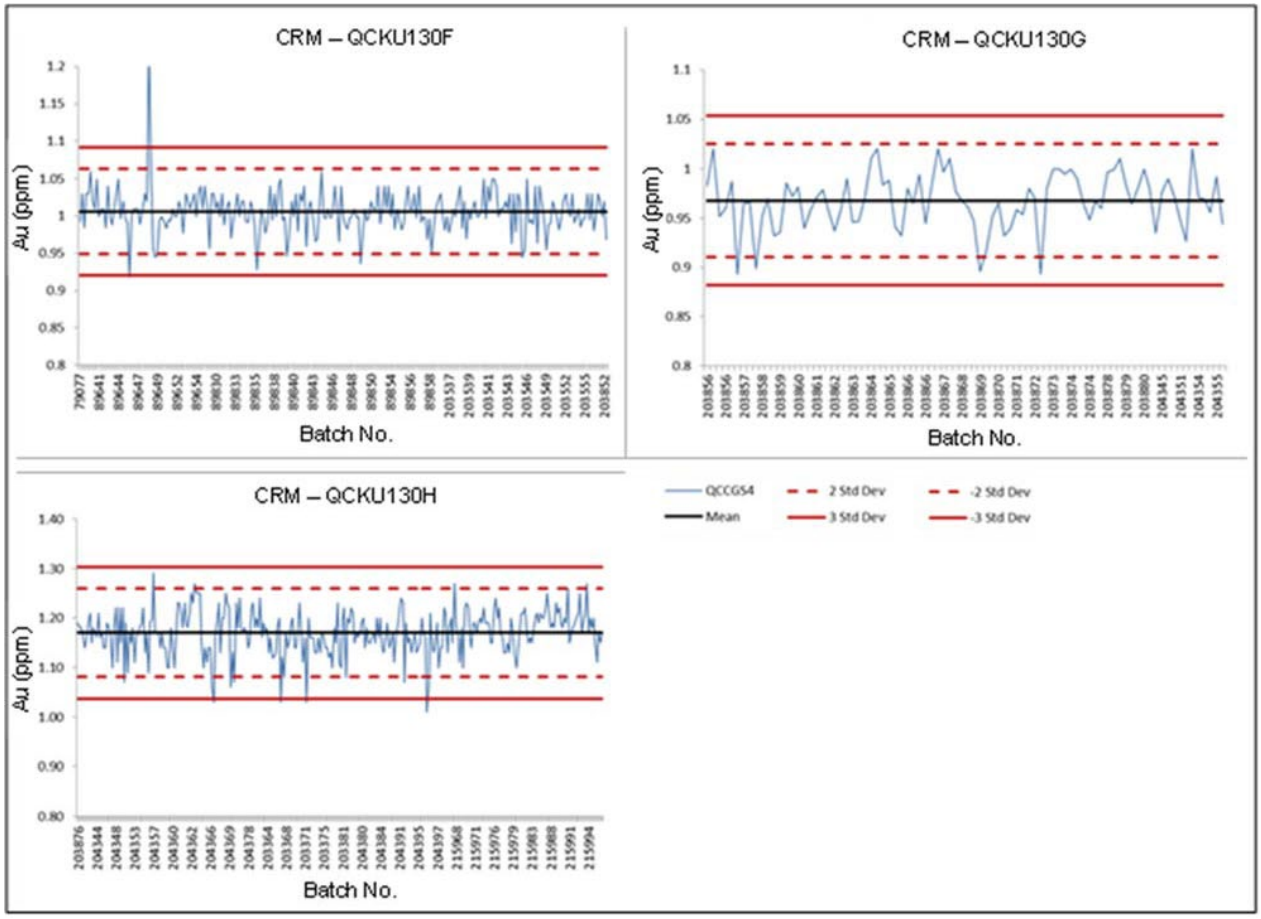
11.5.2.1 Labtium CRM

Labtium utilized nine different CRM during the course of NORDIC's exploration program. The internal gold CRM assayed by Labtium are reported by Labtium together with the sample assay results. Only the data mean, and data standard deviations are reported for the CRM, and the certificate. BOYD assumed that any CRM assayed for gold is a gold CRM, although this is not explicitly stated in the data provided.

Results of the Labtium CRM assays indicate that there is no significant bias and that assay results are generally to a high level of precision, with the majority of results falling within ±2 standard deviations from the mean.

However, out of a total 1,713 CRM assays, 11 (0.6%) assayed outside of ±3 standard deviations from the mean, indicative of either sample mis-numbering, contamination or analytical error. Where a CRM falls outside of ±3 standard deviations from the mean, it is recommended that the entire batch is re-assayed and the CRM value reassessed, to ensure that the assays for the batch are not over- or under-stated. The limited number of CRM failures in the Labtium dataset is not of material concern to the mineral resource estimate.

Overall, the range of mean gold grade in the CRM used by Labtium is quite small, ranging from mean values of between 0.98 and 2.12 g/t gold. Whilst this approximately reflects the average grades observed at Laiva, it would be beneficial in the future to analyze some CRM at a higher grade of between 5 to 10 g/t gold to assess the laboratory performance on samples that closely match the higher-grade mineralized intervals at Laiva.



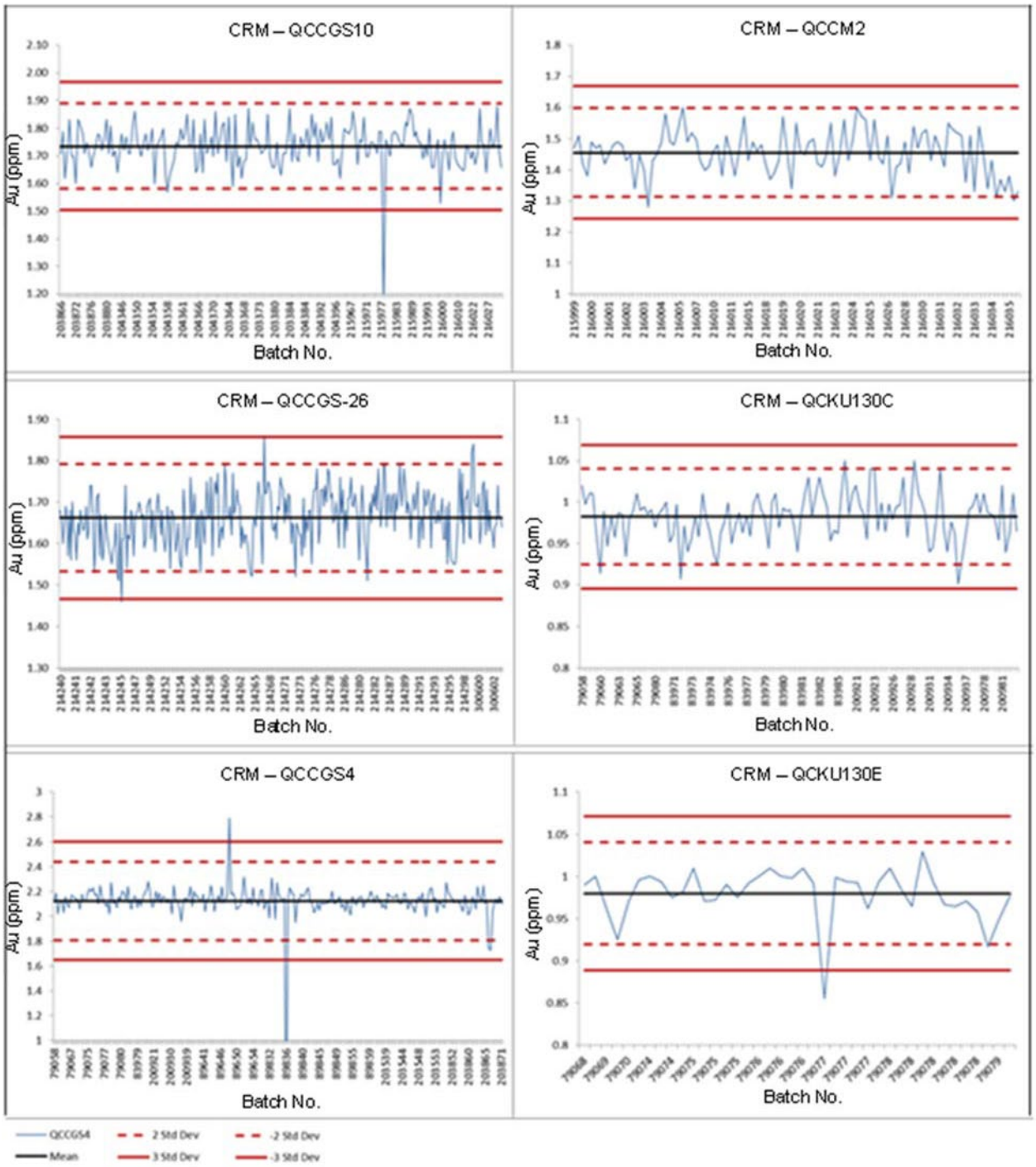


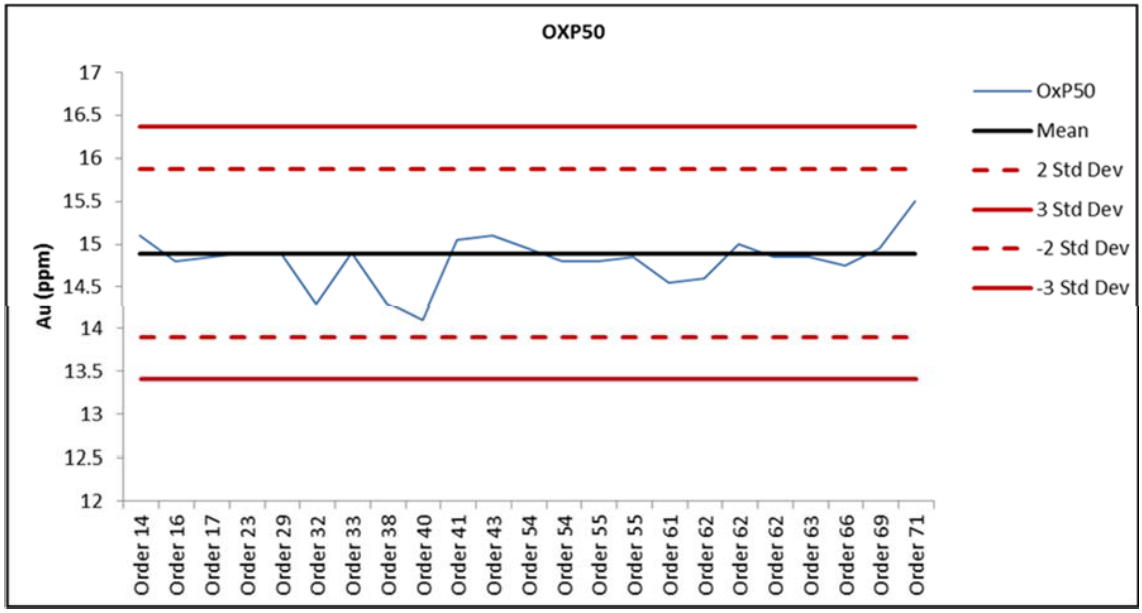
Figure 11.4 – CRM Performance (Labtium)

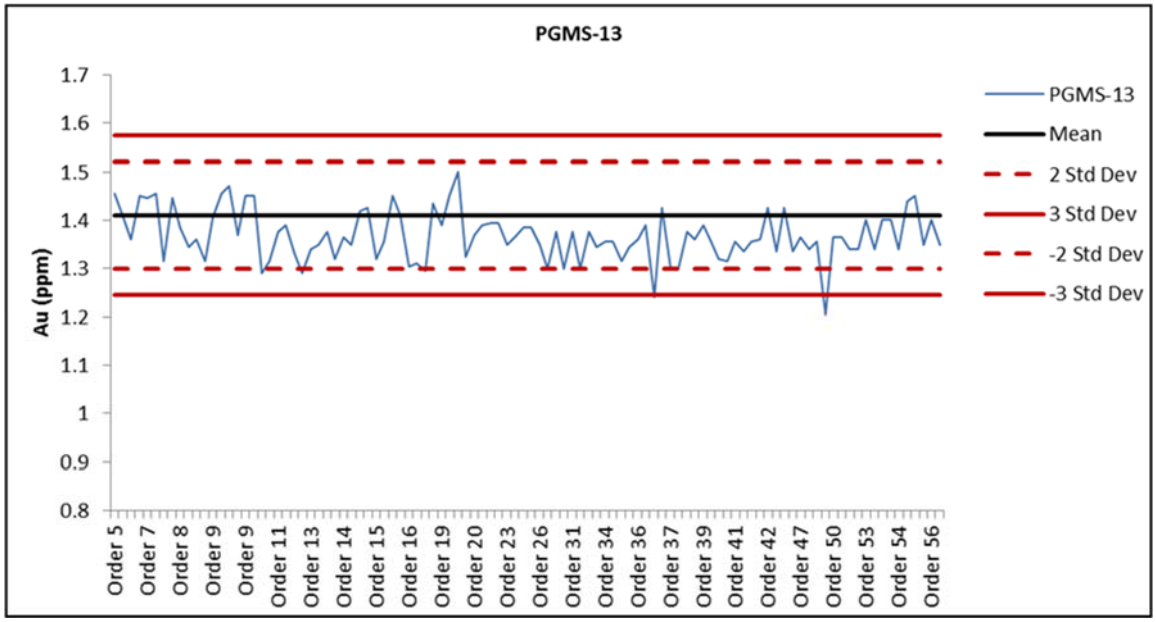
11.5.2.2 ALS CRM

ALS utilized 27 different gold CRM during the course of NORDIC's exploration program and analyzed a total of 1,128 gold CRM samples. The CRMs used had a large range of mean gold grade, between 0.04 and 14.9 g/t gold. The CRM gold grades selected are considered by BOYD to be representative of the gold grades observed at Laiva.

The certified mean and standard deviation were only available to BOYD for 11 of the 27 CRM analyzed by ALS, equating to 655 CRM samples. Of these, 23 of the CRM samples (3.5%) fell outside of ±2 standard deviations from the mean and seven (1%) fell outside of ±3 standard deviations from the mean. This is indicative of a high degree of accuracy and precision in the results obtained from ALS. Assay results for CRM OXP50, PGMS13 and ST252 display a weak negative bias to results when compared with the certified mean assay value, shown in the following graphs in Figure 11.5 below and following this page. However, the results are typically well within ±2 standard deviations from the mean and there is no material concern.

Of the 473 CRM samples without a certified mean and standard deviation, five samples (1.1%) fell outside of ±3 standard deviations from the mean and seven (1.4%) fell outside of ±2 standard deviations from the mean. This is indicative of a high degree of precision in the results obtained from ALS.





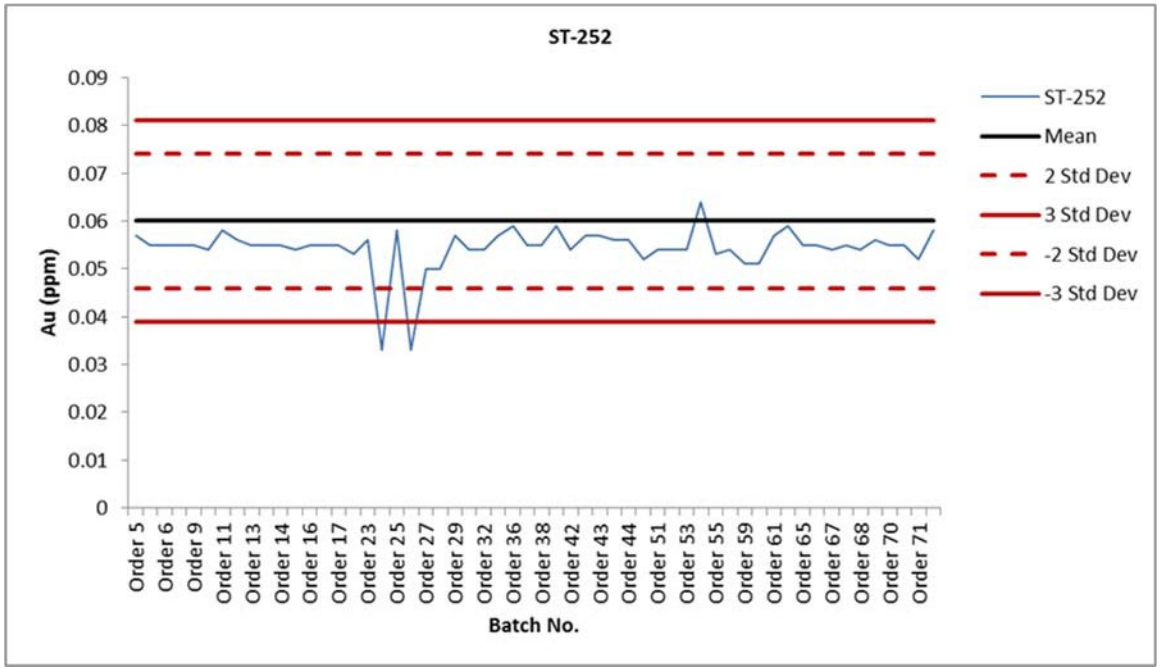


Figure 11.5 – ALS CRS Results

11.5.2.3 CRS CRM

Between 2011 and 2013, CRS created their own internal, hybrid CRM which was prepared by mixing barren sand with CRM sourced from Geostats International Pty and Rocklabs. The CRM from Rocklabs and Geostats had certified mean values of 24.88 ppm gold and 48.53 ppm gold, respectively. CRS created their hybrid CRMs by adding 10 g of CRM to 490 g of barren sand to create material with approximate grades of 0.50 and 0.97 ppm gold.

The size fraction(s) of the barren sand and the source of the barren sand is not known to BOYD. BOYD considers the practice of producing an internal standard from diluting a commercially available CRM to be improper, as the method of homogenization and size fraction(s) of the barren sand is not known. Further, a standard deviation for the hybrid CRM is not given and the estimated mean value was derived from a basic formula rather than analysis of the material. In addition, the original CRM selected were only certified for analysis by 50 g fire assay or aqua regia digest, not cyanide leach tests.

Results from the analyses of both hybrid CRM show inconsistent results, which is most likely due to inadequate homogenization of the CRM sample by CRS and intended analysis by fire assay or aqua regia.

For the hybrid CRM derived from the Rocklabs material and using the standard deviation of the CRS analytical results, 24% of the CRM samples were within ±1 standard deviation, 93% were within ±2 standard deviation and 98% were within ±3 standard deviation, as shown in Figure 11.6 below.

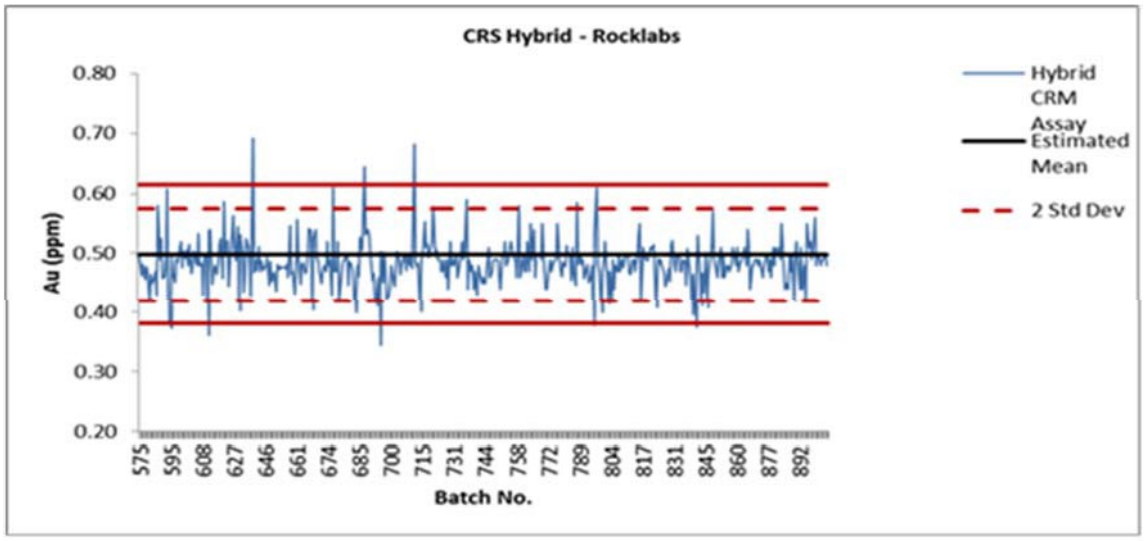


Figure 11.6 – CRS Hybrid CRM Results, 0.50 ppm Gold Showing Data Standard (Deviation)

For the hybrid CRM derived from the Geostats material and using the standard deviation of the CRS analytical results, 61% of the CRM samples were within ±1 standard deviation, 93% were within ±2 standard deviation and 99% were within ±3 standard deviation, as shown in Figure 11.7 below.

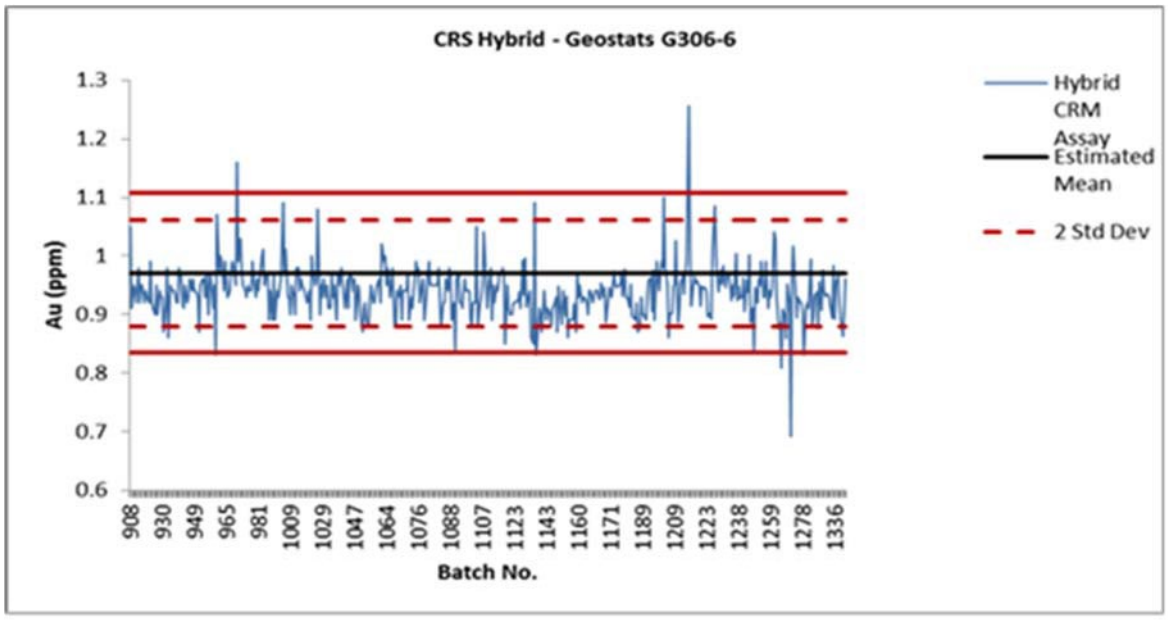


Figure 11.7 – CRS Hybrid CRM Results, 0.97 ppm Gold
(Showing Standard Deviation Derived from the CRS Assay Results)

11.5.3 Duplicate Samples

11.5.3.1 Laboratory Duplicate Samples

Laboratory duplicate samples assayed for gold show a strong correlation in all of the Labtium, ALS, and CRS datasets, with R2 values of 0.9978, 0.9969, and 0.9945, respectively, as shown in Figures 11.8 and 11.9, following this page. This demonstrates good repeatability, as expected in a homogenized sample.

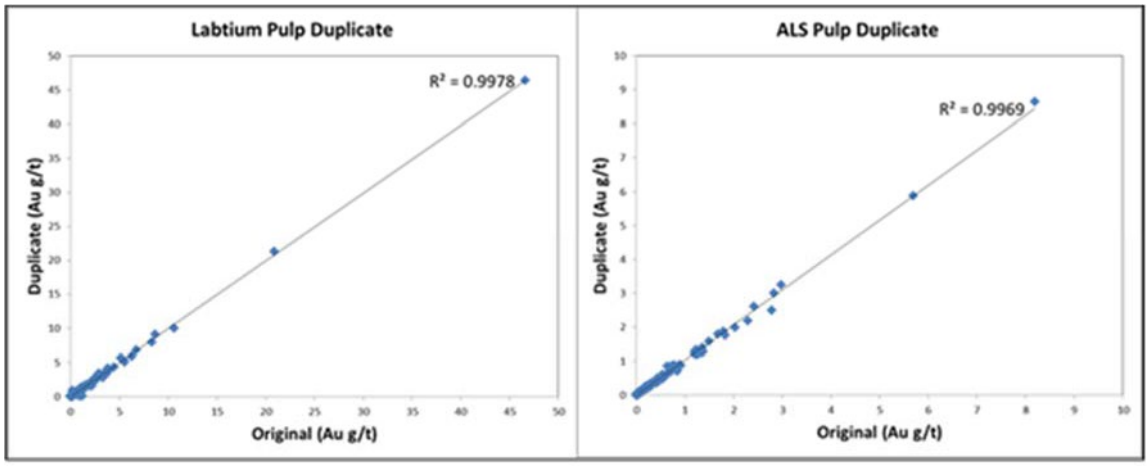


Figure 11.8 – Laboratory Duplicate Results from Labtium and ALS

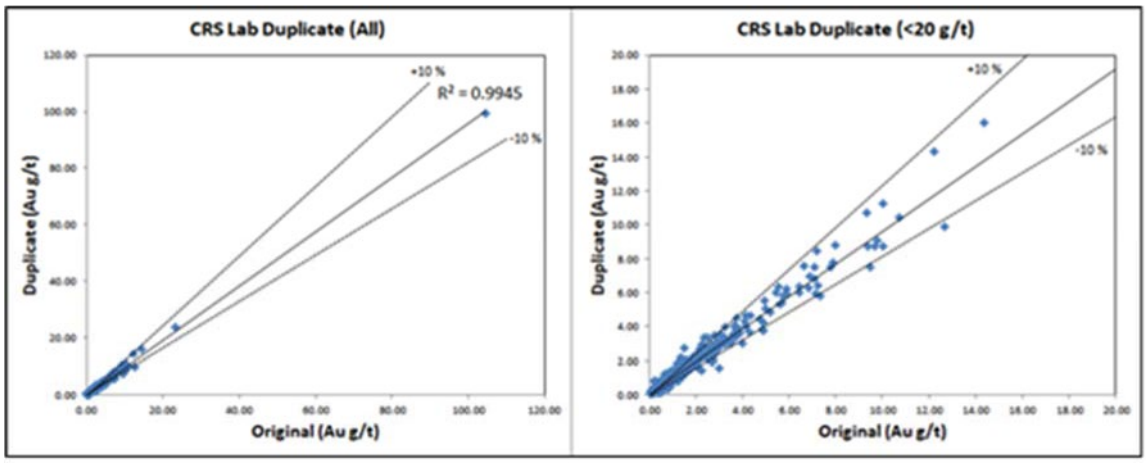


Figure 11.9 – Laboratory Duplicate Results from CRS

11.5.3.2 Crush Reject Duplicate Samples

Under all operators, NORDIC conducted a re-assay program of 523 crush reject samples that had been stored by NORDIC at the exploration office. Samples were re-assayed by both fire assay and bottle roll cyanide leach at Labtium. Results from the fire assay and bottle roll duplicate samples demonstrate excellent repeatability with R2 values of 0.9749 and 0.9654, respectively, as shown in Figure 11.10, following this page.

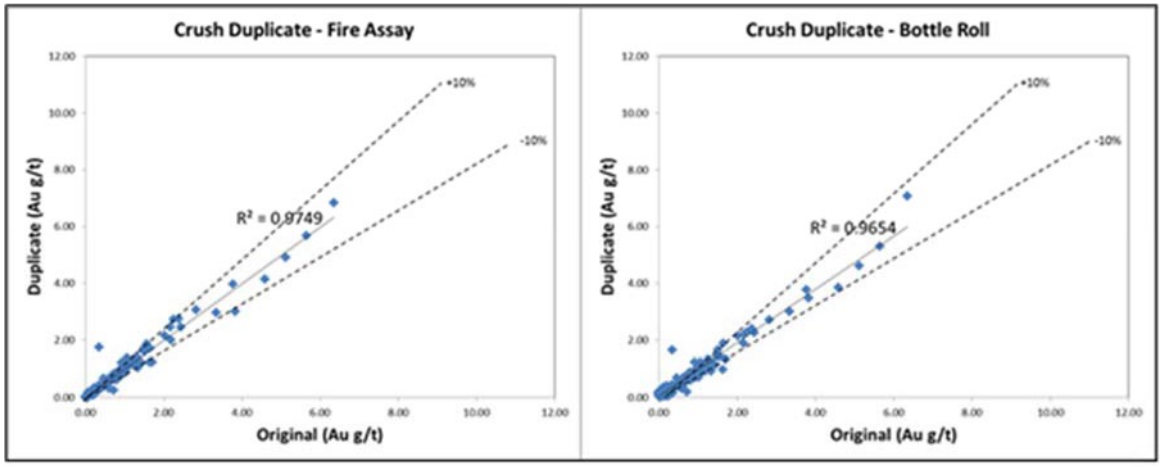


Figure 11.10 – Crush Duplicate Results

11.5.3.3 Field Duplicate Samples

Under all operators, NORDIC conducted field duplicate analysis of 56 half cut drill core samples. Results show a moderate correlation with a R2 value of 0.7916, shown in Figure 11.11 below. However, the average grade of the field duplicate samples is 0.19 g/t gold and only two samples assayed greater than 1 g/t gold. The limited data and very low, sub-economic grade of field duplicate samples are not considered meaningful for determination of gold grade repeatability, nor are the data representative of mineralization at Laiva. Further analysis of gold grade variability and grain size distribution is warranted.

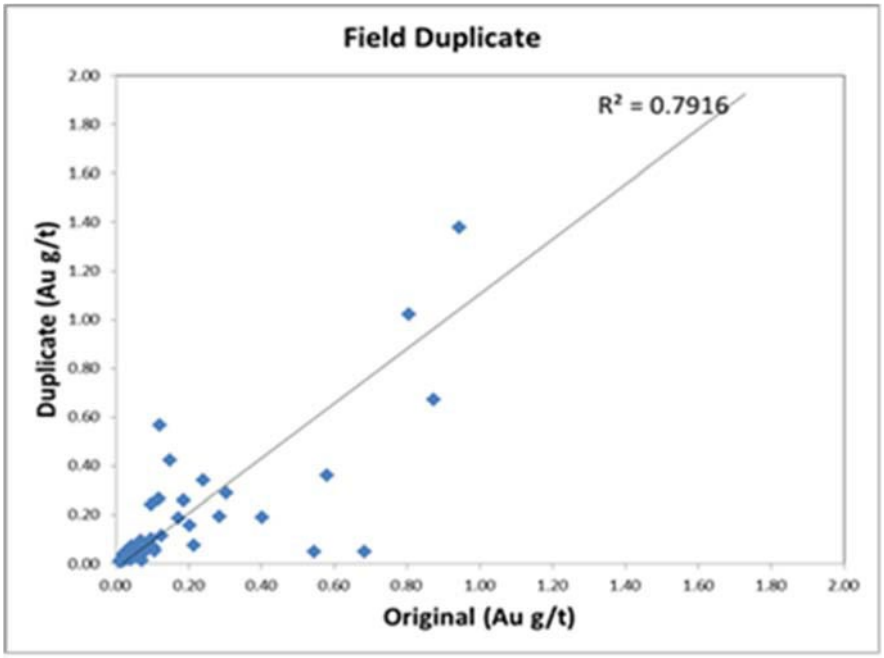


Figure 11.11 – Field Duplicate Results from Half Core Samples

During the 2014 RC grade control trial program, 213 field duplicate samples were analyzed by CRS. Results demonstrate excellent repeatability, as shown in Figure 11.12 below.

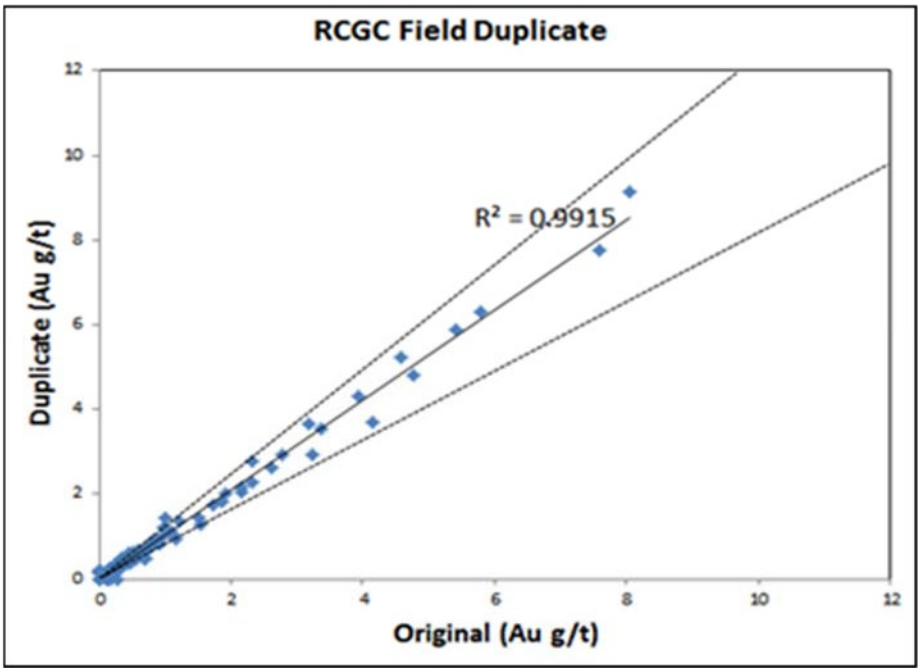


Figure 11.12 – Field Duplicate Results from RC Grade Control Program

11.5.3.4 Check Assays

Under previous operators, NORDIC implemented a check assay program whereby 103 pulp rejects originally assayed at ALS were re-assayed at Labtium, and 110 samples originally assayed at Labtium were re-assayed by ALS. In both instances samples were analyzed by 50 g fire assay.

The check assays conducted by Labtium on ALS original assays shows a strong correlation and all samples assayed within 10% of the original gold grade reported by ALS, shown on the graph in Figure 11.13, following this page.

The check assays conducted by ALS on Labtium original assays also shows a strong correlation in the lower range of gold grades (up to 2 g/t gold). There is poorer correlation for samples greater than 2 g/t gold, although there is only a small sample population in this range, shown in Figure 11.13.

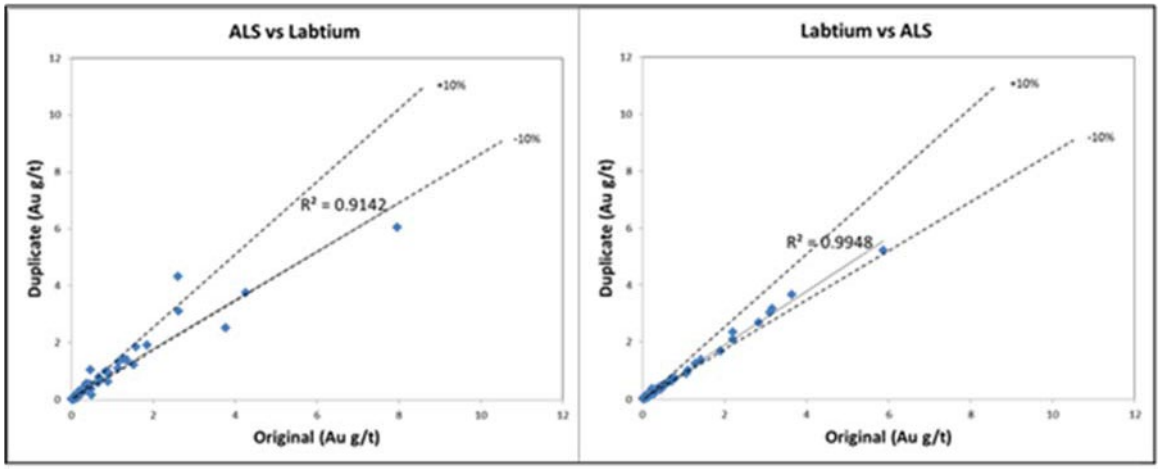


Figure 11.13 – Check Assay Results

11.6 Conclusions

Sample preparation, security and analyses have in general been reported to be conducted to industry standards and the data therefrom are adequate for use in mineral resource estimation.

However, the QAQC program implemented by previous operators of NORDIC has some deficiencies in that blind, external blanks and CRM have not been added to the sample stream, and sample batches of variable sizes were dispatched. Further, the selection, frequency, and location within a batch of samples for, laboratory blanks, CRM and pulp duplicate samples were determined by the laboratory and not NORDIC. The result is that duplicate samples were not routinely taken of mineralized samples and blank samples were not inserted following suspected elevated gold mineralized samples.

In addition, the field duplicate samples selected from half cut drill core were of a very low grade (average 0.19 g/t gold from 52 samples) and therefore the results are of limited use in characterizing the mineralization.

The QAQC program instituted by OTSO was a significant improvement to past practices in that previous BOYD recommendations were implemented to:

- Dispatch samples in batches containing 20 samples, comprised of 16 field (core or RC chip) samples, one CRM, one blank, one field duplicate, and a pulp (laboratory) duplicate.
    - The CRM and blank material should be selected by the operator and inserted blind to the laboratory.

  - Where possible, blank samples should be inserted immediately following a suspected elevated gold mineralized sample to assess the potential for contamination during sample preparation.
  - Similarly, field and pulp duplicate samples should be selected from mineralized samples to better assess repeatability.
- Use of external CRM material inserted blind to the laboratory will give greater control over the analysis of accuracy and precision at the selected laboratory, and therefore greater confidence in the results used for mineral resource estimation purposes.
- Control limits for blanks and CRM should be set in order to pass or fail a batch of samples. This will allow continuous monitoring of each batch of samples and assay results. Batches that fail should be immediately re-assayed, and when all QAQC samples in a batch have passed, the results should be signed-off by the Competent Person as final and acceptable for public disclosure and use in mineral resource estimation.

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12.0 DATA VERIFICATION

12.1 Resource Estimate

The following measures were taken by BOYD to verify the data presented by NORDIC, FTR, and OTSO for mineral resource estimation purposes:

- A visual check of assay intervals, sample numbers, downhole depth, and geological logs was conducted through inspection of randomly selected drill core during BOYD's site visit. Driller's blocks at the end of each run were checked against the drill core length and depth intervals labelled on the core boxes. No errors were observed.

- BOYD checked field locations of some drill collars against the collar file by comparing the surveyed collar location against coordinates measured from a hand-held GPS device. No significant errors were observed.

- The drill hole database was verified by checking for overlapping sample intervals, duplicated data, variations in drill hole orientation, sample intervals deeper than the end of hole, and missing assay, survey or lithological data. BOYD also conducted a check of random sample intervals, sample numbers and assay results against the original laboratory certificates. Minor errors were corrected with input from NORDIC staff, and holes that could not be verified were removed from the database. Only one hole was removed from the database and there were no other major errors in the drill hole database.

- BOYD conducted a visual check of mineralized intercept width and gold grade between closely-spaced RC grade control and exploration diamond drill holes.

- BOYD reviewed the block models provided by NORDIC which had been used for previous mineral resource estimates at Laiva. BOYD elected to reinterpret the mineralized zones based on BOYD's observations from drill core, exposures in the open pits, and results from detailed bench mapping conducted by NORDIC during mining operations. The aim of BOYD's reinterpretation was to represent the narrow, steeply dipping nature of mineralization and constrain the gold grade within the mineralized shear zones more accurately. The result was a revised geological interpretation and wireframes of mineralized zones, which were used to visually verify the position and gold grade of mineralized blocks in the revised block model.

There were no observed limitations or failure to conduct any of the data verification. Accordingly, in BOYD's opinion, the data set is adequate for use in Mineral Resource estimation upon which this Technical Report is based.

12.2 Property Rights and Mineral Tenure

Given the LGP operating history since before 2013, it is clear that operations have been authorized since then by regulatory authorities and landowners. The mining concession is up for renewal in 2023. Given the importance to the local, economy and the current existing surface disturbance, under Finish law, in BOYD's opinion, the renewal process is likely to be a proforma matter. For similar reasons, BOYD expects that exploration concessions under renewal, which are critical to ongoing mine development will also be renewed on a proforma basis.

Information relating to property rights and mineral tenure has been supplied by OTSO, and legal due diligence has not been independently performed. Nonetheless, BOYD considers the foregoing evidence to be verified and suitable for use in this Technical Report.

12.3 Environmental Permits and Social Considerations

Similar to the discussion in Section 12.2, above, OTSO has provided information regarding permitting and social matters to BOYD, including evidence thereof. While BOYD has not completed a legal investigation of the foregoing, ongoing successful operation coupled with BOYD's understanding of permitting requirements required under Finnish Law and regulations thereunder lead us to believe the information is adequately verified for inclusion in this Technical Report.

12.4 Existing Facilities

12.4.1 Introduction

The LGP is a fully developed facility consisting of two adjacent open pit gold mines, a fully operational nominal 2 million metric tonne per year process facility, and all related infrastructure including high and low grade tailings storage facilities (TSF), reagent storage facilities, shops, warehouses, and administrative offices.

BOYD personnel have visited the LGP on several occasions from March 2017 through September 2021 on behalf of the prior operator, FTR and the current operator, OTSO.

12.4.2 Mine

BOYD personnel have personally inspected the North and South pits, mine haul roads, waste storage facilities, ROM ore pad, and pit dewatering systems. BOYD has in its files design data for these facilities and has field verified the accuracy of this data and therefore considers this data to be suitable for use in this Technical Report.

12.4.3 Process Plant

BOYD personnel have personally inspected the entire process plant in detail. BOYD has in its files metallurgical testing data, design criteria, P&ID's, engineering drawings, and operating records and has field verified the accuracy of this data where applicable and therefore considers this data to be suitable for use in this Technical Report.

12.4.4 Infrastructure

BOYD personnel have personally inspected all components of infrastructure, including without limitation TSF's, shops, warehouse facilities, reagent storage facilities, site water collection reservoir, and pumping system for discharge of excess site water to the sea. BOYD has in its files, design criteria, engineering drawings, and operating data where applicable and therefore considers this data to be suitable for use in this Technical Report.

12.5 Contracts

BOYD has been provided pertinent contract information by FTR and OTSO which are contained in its files. BOYD has reviewed this contract information and determined the terms to be consistent with industry standards for the intended purpose, and concluded that terms provided in these contracts, where applicable, are suitable for use in this Technical Report.

12.6 All Other

All other data used in preparation of this Technical Report was developed by BOYD from first principals.

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13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Process design criteria for the restart of the LGP treatment plant were developed by completing the following key tasks:

- Conduct detailed evaluation of all operating information from previous operating periods with an emphasis on a 14-month period of continuous operation by previous operators during the period December 2013 through January 2014.
- Evaluate comminution test work to determine maximum and optimum production rates for the existing comminution circuits and to identify modifications to improve the circuits. Comminution test work recently completed in 2021 included:
  - Rod mill work index
  - Ball mill work index
  - JK drop test
- Evaluate results from the metallurgical test program completed in 2021 at Resource Development Inc. (RDi) located in Wheat Ridge, Colorado. The RDi test program utilized samples from each of the primary rock types from the LGP composited from individual drill hole sections. Treatment parameters evaluated included:
  - Flash flotation
  - Gravity concentration
  - Cyanide leaching of concentrate tailings
- Evaluate results from all previous metallurgical test work completed prior to the 2021 RDi test program.

The most critical plant operating parameter was determined to be the proportion of gold in the plant feed reporting to the high grade (HG) CIL circuit. During previous periods of operation, the HG CIL circuit consistently achieved greater than 97% gold extraction. However, the proportion of contained gold in the feed was generally around 40% reporting to the HG CIL circuit.

BOYD determined that inadequate capacity in the concentrate regrind circuit to grind to suitable particle size (P_{80} 45 µm) for feed to the HG CIL circuit was the limiting factor affecting full utilization of the HG CIL circuit capacity. Of only slightly lesser importance was failure to maximize pre-leach gold concentration in the flash flotation and gravity concentration circuits. Accordingly, the primary focus in developing the updated process design criteria was to maximize utilization of the HG circuit by increasing regrind capacity and optimization of pre-concentration circuits ahead of CIL.

In order to achieve the process plant “name plate” capacity of nominal 250 mtph or 2 million mtpy at required grind for best overall cost-effective recovery revisions to the feed to the primary AG mill need to be implemented. Past operations only rarely recorded actual throughput at these rates, and when actually achieved generally resulted in reduced plant recovery. Comminution test work conducted by past operators showed a wide range of grinding work indices, which resulted in considerable variability in comminution performance during operation.

In order to better understand the variability among ore types, BOYD reviewed ore reserves by rock type and recommended further comminution testing to accurately determine comminution performance by rock type. In addition, analysis by BOYD’s comminution expert determined that feeding the primary autogenous grinding mill (AG) with crushed product at nominal 200 mm particles (20% of feed) to serve as grinding media and feeding secondary crushed product at nominal 25 mm particles (80% of feed) would result in 1) reduced energy cost, 2) increased through put to name plate capacity, and 3) result in ground product (through AG and pebble mills in closed circuit) of P_{80} 100 mm product, as required for optimum process recovery.

13.2 Operating History

The Laiva Mill was operated continuously for approximately two years during the period 2012 to 2014. The mill reports from the most stable operating periods were evaluated in detail to determine the primary problematic areas and to develop a priority list of plant modifications.

The mill production rate during this period averaged approximately 210 mtph verses the design rate of 250 mtph. The ground ore particle size ranged from P_{80} 110 µm to 130 µm verses a design particle size of P_{80} 75 µm.

Identified operational issues include:

- Overall plant availabilities typically less than 85%, well below the expected range of plus 90%.
- Regular interruptions in the comminution circuit resulting in inconsistent production rates and variable particle size distributions in the feed to the recovery plant.
- Low-grade circuit consistently achieved gold extractions less than 80%.
- Operating records did not include industry standard metallurgical balance for process facilities.

- Operating records did not include a description of a well-planned preventative maintenance program.

Identified operational areas with positive performance include:

- High-grade circuit consistently achieved plus 97% gold extraction.
- Overall cyanide consumption was reasonable at an estimated average consumption of 0.47 kg of cyanide per tonne of ore.
- All components of the carbon circuit met or exceeded the design criteria.
- The electrowinning and smelter facilities operated according to specifications.

13.3 Comminution Test Work

Large bulk samples were collected by OTSO project geologists for new comminution test work. Samples were collected for the two primary rock types, quartz diorite and mafic volcanics. Together, these two rock types make up approximately 90% of the current resource. The new test work included:

- Rod mill bond work index.
- Ball mill work index.
- JK drop test.

Based on the results of an evaluation of the 2021 comminution test work and all available historical operating information, the calculated mill circuit maximum production rate is 2.0 mtpy with a target mill final product particle size of P_{80} 100 µm.

BOYD recommends restarting the mill at an initial feed rate of 1.5 M mtpy, gradually increasing to 2.0 M mtpy provided recoveries are maintained. Overall plant operating parameters should be optimized at each production rate before increasing to the next increment.

The calculated power draws for the two primary ore types at key production rate increments are:

1.5 MTPY Throughput

- P_{80}=100 µm for 8,000 hours at 187.5 mtph.
- Power draw metavolcanic rock type (MV) per JK drop test 12.3 kW/mt or 2,306 kW required in AG Mill.

- Power draw for quartz diorite rock type (QD) per JK drop test 11.4 kW/mt or 2,137 kW required in AG Mill.

1.75 MTPY Throughput

- P_{80}=100 µm for 8,000 hours at 218.75 mtph.
- Power for MV per JK drop test 12.3 kW/mt or 2,690 Kw required in AG Mill.
- Power draw for QD per JK drop test 11.4 kW/mt or 2,493 Kw required in AG Mill.

2.0 MTPY Throughput

- P_{80}=100 µm for 8,000 hours at 250.0 mtph.
- Power for MV per JK drop test 12.3 kW/mt or 3,074 Kw required in AG Mill.
- Power draw for QD per JK drop test 11.4 kW/mt or 2,849 Kw required in AG Mill.

The actual past production rates for the approximate 14-month operating period were:

- Primary grind particle size of P_{80} 110 - 130 µm at 210 tonnes per hour.
- The calculated Bond Rod Mill Work Index was 20.5 kWh/tonne.
- The mill was typically operated at 210-240 mtph with an AG mill power draw 2,650-2,988 kW and with a Pebble Mill power of 2,826-2,975 kW.
- Total available average power 5,476 kw to 5,963 kw between the two mills.

The results of the 2021 work index test work are as follows:

- Sample MV Bond Rod Mill Work Index: 13.41 kWh/tonne.
- Sample MV Bond Ball Mill Work Index: 14.92 kWh/tonne.
- Sample QD Bond Rod Mill Work Index: 14.88 kWh/tonne.
- Sample QD Bond Ball Mill Work Index: 17.23 kWh/tonne.

Previous power draw calculations were based upon back calculations to an operating WI of 18.1kWh/tonne

Integrated JKW and SMC tests were completed on bulk samples of the QD and MV rock types. The data was analyzed to determine the JKSimMet and SMC Test comminution parameters. These test procedures are the industry standard method to estimate autogenous and pebble mill operating parameters.

Results of the JKDW and SMC tests were evaluated by BOYD's comminution expert, which were used to confirm the comminution design criteria utilized to develop process capital and operating costs.

13.4 Metallurgical Test Work 2021

A laboratory scale metallurgical test program was completed to evaluate the LGP treatment flow sheet utilized in the existing treatment plant. The test program was completed by Resource Development Inc. (RDi) located in Wheat Ridge, Colorado.

The RDi test program included evaluation of the plant unit operations utilizing a QD composite sample with a gold grade similar to the estimated gold grade for the plant feed. The test program included the testing of flash flotation at P_{80} 1.5 mm and gravity concentration followed by tailings cyanidation at P_{80} 75 µm and P_{80} 105 µm.

The primary purpose of the test program was to develop process design criteria for minor, but important revisions to the existing treatment plant.

13.4.1 Sample Description

The primary ore zone rock type is QD, which makes up over 70% of the current reserve. It was also determined that the gold bearing mineral components of the second largest rock type, MV, are similar to those found in QD. Accordingly, BOYD selected a QD composite sample for the metallurgical test work designed to evaluate the plant flow sheet.

Composites were selected to ensure that the samples would be representative of the QD zones of the ore zones as they are now known. Individual drill hole sample splits from completed resource development programs were blended to produce the composite samples. The individual drill hole samples that had been crushed to a particle size of approximately P_{80} 1.5 mm were thoroughly blended from reject splits remaining from samples submitted for assay. The entire remaining quantity of the P_{80} 1.5 mm reject from each selected drill hole sample were shipped to RDi for compositing.

13.4.2 Head Analyses

Duplicate splits of the QD were analyzed for determination of sample variability. Analysis of the two splits included:

- Fire assay for gold and silver.
- Multi-element ICP.
- Sulfur speciation.

A single split of the QD composite was then submitted for mineralogical analyses that included:

- XRF and XRD.
- Optical minerology.

Results from the mineralogical analyses for the QD composites are presented later in this report in Section 13.4.3.3.

13.4.2.1 Chemical Analysis

A summary of the results from the chemical analytical procedures for the two splits are presented in Table 13.1, following this page.

Table 13.1 – Head Analyses of QD Average Grade Composite Sample

Element	QD Average Grade Split A	QD Average Grade Split B
Au, g/mt	1.678 Orig / 1.687 Dup	1.515 Orig / 1.517 Dup
Ag, g/mt	0.4 Orig / 0.2 Dup	0.5Orig / 0.7 Dup
Sulfide S %	0.03	0.03
Sulfate S %	0.02	0.03
Total S %	0.05	0.06
Al %	6.78	6.64
Ca %	3.62	3.64
Fe %	4.95	5.11
K %	1.95	1.97
Mg %	2.53	2.57
Na %	2.11	2.11
P %	0.25	0.25
Ti %	0.29	0.30
As ppm	573	640
Ba ppm	375	378
Be ppm	<2	<2
Bi ppm	<2	<2
Cd ppm	6	6
Co ppm	22	23
Cr ppm	204	193
Cu ppm	60	62
Hg ppm	0.77	0.69
La ppm	11	12
Li ppm	56	58
Mn ppm	629	639
Mo ppm	2	<1
Ni ppm	18	18
Pb ppm	13	<10
Sb ppm	<2	<2
Sc ppm	19	20
Se ppm	<5	<5
Sn ppm	<10	<10
Sr ppm	308	311
Ta ppm	<10	<10
Te ppm	10	10
Tl ppm	<10	<10
U ppm	<10	<10
V ppm	143	145
W ppm	86	112
Zn ppm	67	55

The primary conclusions from these results are:

- Both samples had gold head grades similar to the average grade of the HG portion of the reserve.
- The gold grades of the two samples varied by about 10% which confirms the previously reported variability of gold grade within a given rock type.
- The silver grades of the two samples varied considerably but are considered to be insignificant since silver grades are so low.
- The sulfide sulfur contents of the two samples were low indicating that the majority of minerals that make up the flotation and gravity concentrates are not sulfides and may not be susceptible to concentration by flotation but will likely be concentrated by gravity means if particle size ranges are relatively narrow.

13.4.3 Design Criteria Test Results

Initial testing of the process flow sheet was completed generally on the average grade QD composite sample. Two separate runs were completed with seven kg charges. The test procedure consisted of completing flash flotation at P_{80} 1.5 mm to collect approximately 2% of the starting mass into the flotation concentrate for both tests. This was followed by grinding flotation tails to two target particle sizes of P_{80} 100 µm and P_{80} 75 µm.

The ground flotation tails were then subjected to gravity concentration to recover approximately 4% of the starting mass into the gravity concentrate. Duplicate splits from the gravity concentration tailings were then cyanide leached to determine gold extraction. The flotation and gravity concentrates were submitted for assay analysis, including XRF/XRD.

A diagram of the flow sheet utilized is shown in Figure 13.1, following this page.

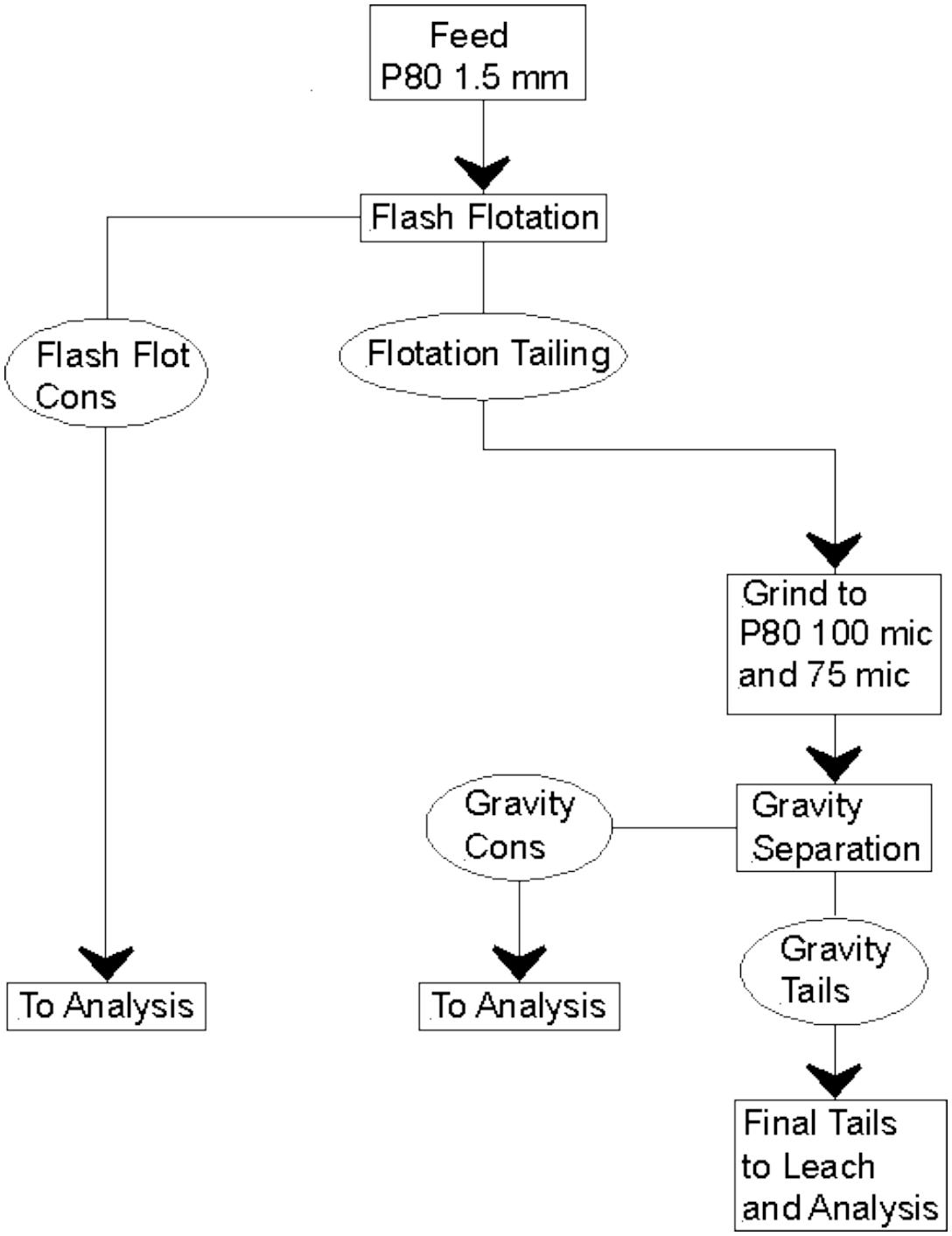

Figure 13.1 – Plant Simulation Test Program Flow Sheet

A summary of the overall results for the float/gravity/cyanidation test program are presented in Table 13.2 below.

Table 13.2 – Summary of Process Recovery Results for QD Average Grade Composite

Sample	Overall Au Recovery %	Calc. Head Grade (Au g/mt)	Flotation Concentrate			Gravity Concentrate			Tail Leach	
			Recovery %		Grade (Au g/mt)	Recovery %		Grade (Au g/mt)	Average Au Extraction %	Average Residue Grade (Au g/mt)
			Wt.	Au		Wt.	Au			
P_{80} 100 µm	91.5	1.82	2.3	33.6	25.99	4.2	25.1	10.85	79.4	0.17
P_{80} 75 µm	92.7	2.01	2.0	41.7	42.59	3.7	21.1	11.32	80.2	0.16

It should be noted that the overall Au recoveries in this table are presented only for generation of concentrates and do not include leaching of the concentrates. Therefore, the estimated plant recoveries would include a 0.97 recovery factor for the leaching of the concentrates, which results in estimated overall recovery for the 100 µm sample of 89.7% and 90.8% for the 75 µm samples respectively.

The primary conclusions from these results are:

- The 100 µm and 75 µm particle size samples both responded well to flotation concentration, gravity concentration and cyanidation of the concentrate tailings producing tailings grades that are essentially the same for both particle sizes.
- The 75 µm provided a superior overall recovery along with having a 10% calculated head grade. This result indicates that following restart of the plant, it wouldbe warranted to evaluate mill particle sizes finer than the initial restart target of P_{80} 100 µm.
- Approximately 60% of the contained gold was recovered into a float/gravity concentrate representing approximately 6% of the weight of the feed for both samples.
- The flotation concentrate contained a higher percentage of the contained gold than the gravity concentrate for both samples.
- Both samples were floated at the same particle size of P_{80} = 1.5 mm; therefore, the higher gold flotation recovery in the 75 µm sample is likely due to differences in the laboratory flotation procedures. Exactly duplicating laboratory flash flotation testsis difficult, primarily because of the short flotation time period and the many flotation variables.

Table 13.3 – Summary of XRD Results for Flotation and Gravity Concentrates

Mineral	Weight %			
	Float Con Test 1	Float Con Test 2	Gravity Con Test 1	Gravity Con Test 2
Plagioclase feldspar	31	27	17	18
Clinoamphibole	35	42	64	66
Mica/illite	12	17	10	7
Quartz	9	5	<5	<5
K-feldspar	8	<5	-----	-----
Chlorite	<3	5	-----	-----
Arsenopyrite	<3	<2	<1	-----
Ilmenite	-----	-----	<2	<2
Apatite	-----	-----	<2	-----
Unidentified	<5	<5	<5	<5

13.4.3.1 Flash Flotation and Gravity Test Results

Two flotation/gravity tests were completed to compare gravity concentration and subsequent concentrate tailings, followed by cyanidation at two particle sizes of P_{80} 75 and 100 µm. Both samples were floated at a particle size of approximately P_{80} 1.5 mm.

The target concentrate quantities were 2% of the feed for flotation and 4% of the feed for gravity concentration for a total concentrate quantity equal to approximately 6% of the feed. These concentration quantities were targeted to match the proposed plant restart concentrate production quantities for flash flotation and gravity concentration. A concentrate production rate equal to 6% of the total plant feed at 158 mtph results in concentrate production of approximately 9.5 mtph. A 158 mtph feed rate in the plant relates to an annual production rate of 1.5 M tons per year.

Test One 1.5 mm/100 µm

A 1.0 kg split of the 1.5 mm QD composite sample was floated in a laboratory scale Denver 3.2 liter float cell for two minutes with air addition.

Flotation reagent additions were:

- PAX at 14 g/mt of ore.
- 3418A at 14 g/mt of ore.
- AF65 at 6 g/mt of ore.

The flotation tailings were ground in a standard laboratory scale rod mill for 32 minutes. The ground flotation tailings were then fed to a Diester laboratory scale shaking table to recover the gravity concentrate.

A summary of the flotation and gravity concentration test results for the 100 µm sample are presented in Table 13.4 below.

Test Two 1.5 mm/75 µm

A 1.0 kg split of the 1.5 mm quartz QD composite sample was floated in a laboratory scale Denver 3.2 liter float cell for two minutes with air addition.

Flotation reagent additions were:

- PAX at 13 g/mt of ore.
- 3418A at 13 g/mt of ore.
- AF65 at 5 g/mt of ore.

The flotation tailings were ground in a standard laboratory scale rod mill for 44 minutes. The ground flotation tailings were then fed to a laboratory scale Diester shaking table to recover the gravity concentrate.

A summary of the flotation and gravity concentration test results for the 75 µm sample are presented in Table 13.5 below.

Table 13.4 – Float-1.5mm/Gravity-100 µm

Product	Weight %	Au g/t	Ag g/t	Sulfide %	As %	Au % Dist.
Feed calc	100.0	1.8	0.2			100.0
Float con	2.3	26.0	5.9	1.2	1.27	33.6
Gravity con	4.2	10.9	1.8	0.2	0.45	25.1
Tails	93.4	0.8				41.3
Combined Con calc	6.5	16.2	3.3	0.6	0.70	58.7

Table 13.5 – Float-1.5mm/Gravity-75 µm

Product	Weight %	Au g/t	Ag g/t	Sulfide %	As %	Au % Dist.
Feed calc	100.0	2.0	0.3			100.0
Float con	2.0	42.6	12.9	1.7	1.64	41.7
Gravity con	3.7	11.3	1.9	0.2	0.39	21.1
Tails	94.3	0.8				37.3
Combined Con calc	5.7	22.3	5.8	0.7	0.82	62.8

The primary conclusions from these test results are:

- Approximately 60% of the contained gold was recovered into a float/gravity concentrate that represented approximately 6% of the weight of the feed. These values will be utilized for the process design criteria.
- The difference in the flotation recoveries for the two splits of the sample confirms the variability of the mineralogy of the ore body for a given rock type.

13.4.3.2 Flotation/Gravity Tailings Cyanidation Results

Duplicate splits of the 100 µm and the 75 µm flotation/gravity concentrate tailings samples were cyanide leached in bottle rolls for 96 hrs.

The cyanide leach conditions were:

- 40% solids.
- Cyanide solution concentration of 1.0 g/l.
- pH target of 11.0 by lime addition.

100 µm Concentrate Tailings Sample

A summary of the results of the cyanide bottle duplicate tests of the 100 µmconcentrate tailings sample are presented below.

Duplicate Sample 1

Au Extraction **= 77.5 %**
Calculated Head Au **= 0.82 g/t**
Final Tail Au Assay **= 0.19 g/t**
Cyanide Consumption **= 0.122 kg/t,** as NaCN
Lime Added **= 2.104 kg/t,** as $Ca(OH)_2$

Duplicate Sample 2

Au Extraction **= 81.2 %**
Calculated Head Au **= 0.78 g/t**
Final Tail Au Assay **= 0.15 g/t**
Cyanide Consumption **= 0.182 kg/t,** as NacCN
Lime Added **= 2.234 kg/t,** as $Ca(OH)_2$

75 µm Concentrate Tailings Sample

A summary of the results of the cyanide bottle duplicate tests of the 75 µmconcentrate tailings sample are presented below.

Duplicate Sample 1

Au Extraction **= 80.9 %**

Calculated Head Au **= 0.79 g/t**

Final Tail Au Assay **= 0.15 g/t**

Cyanide Consumption **= 0.182 kg/t,** as NacCN

Lime Added **= 2.234 kg/t,** as $Ca(OH)_2$

Duplicate Sample 2

Au Extraction **= 79.5 %**

Calculated Head Au **= 0.80 g/t**

Final Tail Au Assay **= 0.16 g/t**

Cyanide Consumption **= 0.122 kg/t,** as NacCN

Lime Added **= 2.264 kg/t,** as $Ca(OH)_2$

The primary conclusions from these results are:

- The final gold extractions for all four samples were essentially the same at approximately 80%. This is the value that will be utilized for the process design criteria.
- Low tailings grades were achieved for all four samples at approximately 0.15 g/t Au.
- The cyanide consumptions for all four samples were in the low range and match closely to the estimated consumption utilized in the design criteria, which was based on previous plant operating history.

The cyanide leach kinetics for the four final tailings bottle roll tests are present in Table 13.6 below.

Table 13.6 – Cyanide Leach Kinetics for the Four Final Tailings Bottle Roll Tests

Au Extraction %	100 µm Duplicate 1	100 µm Duplicate 2	75 µm Duplicate 1	75 µm Duplicate 2
6 hrs	67.2	72.3	73.8	72.8
24 hrs	75.6	79.2	80.7	77.6
48 hrs	76.8	78.6	80.1	78.7
72 hrs	78.0	81.7	81.4	78.2
96 hrs	77.5	81.2	80.9	79.5

The primary conclusions from these results are:

- All four samples exhibited fast leach kinetics with approximately 90% of the leachable gold extracted in the first six hours of bottle roll leaching.
- Evaluation of modifications to the LG CIL circuit to reduce processing costs is warranted.
- LG CIL circuit modifications could include:
  - Reduction in leach solution cyanide concentration
  - Reduction in air addition rate
  - Reduction in number of CIL tanks being utilized.
  - Conversion to CIP

13.4.3.3 Mineralogical Analyses Results

Following are the mineralogical analyses results for the QD metallurgical composite.

The mineralogical analysis was completed by DCM Science Laboratory located in Wheat Ridge, Colorado. Analysis consisted of microscopic evaluation of a prepared thin section of a split of the QD composite sample.

The results of the mineralogical analysis are presented below.

Microscopic Description

Major Mineralogy : Amphibole 25% *Plagioclase 25%* *Quartz 20%*
Microcline 15%

Biotite 10% *Muscovite/Sericite 4%* *Sphene 1%*

Trace Mineralogy: Apatite, Chlorite, Zircon, Epidote, Pyrite, Marcasite, Chalcopyrite, Calcite,Chalcopyrite, Pyrrhotite, Au

In thin section this material represents granitic rock fragments primarily composed of hard silicates. Dark green to light brown hornblende is the primary mineral and occurs as subhedral toeuhedral prisms up to 500 µm in size.

The amphibole is commonly associated with euhedral books of biotite/muscovite, anhedral grains of sphene and small grains of zircon. Some of the amphibole shows light chloritization along cleavage planes and grain boundaries.

Plagioclase with a bulk composition of albite/oligoclase is the dominant feldspar and occurs as subhedral to euhedral grains with a size variation of 50 µm to approximately 450 µm. Many grains show light to moderate sericitization. The plagioclase is often associated with anomalous blue colored grains of epidote and occasional inclusions of

secondary calcite. Late-stage grid twinned microcline is present in significant amounts and occurs as interstitial anhedral grains up to 200 µm in size. The potassium feldspar shows no weathering and is water clear.

Quartz is well represented and occurs as mosaic aggregates and large liberated fragments that appear to have originated from vein quartz. Quartz grain size varies greatly from 2 µm to over 1 mm. Larger grains commonly carry small inclusions of sulfides.

Collectively sulfide mineralogy accounts for approximately 1% of the sample's mass and is represented by several types. Arsenopyrite is the dominant type and occurs as anhedral to euhedral rhomb shaped grains that vary in size from 1 µm to 200 µm. Pyrite is present in slightly lesser amounts and generally occurs as anhedral grains up to 200 µm in size. Larger grains commonly carry small inclusions/attachments of marcasite, chalcopyrite and rare pyrrhotite. Chalcopyrite and pyrrhotite also occur as inclusions in some silicates with a grain size up to 75 µm. Although a small population of pyrite shows minor oxidation, the sulfides are generally alteration free. An extensive search under high magnification identified one grain of Au. The Au occurs as a minute, rounded grain in coarse quartz with a grain size measurement of approximately 4 µm.

Examples of the most pertinent mineralogical components of the QD sample are shown in the following micrographs in Figure 13.2 through 13.7, on the following pages.

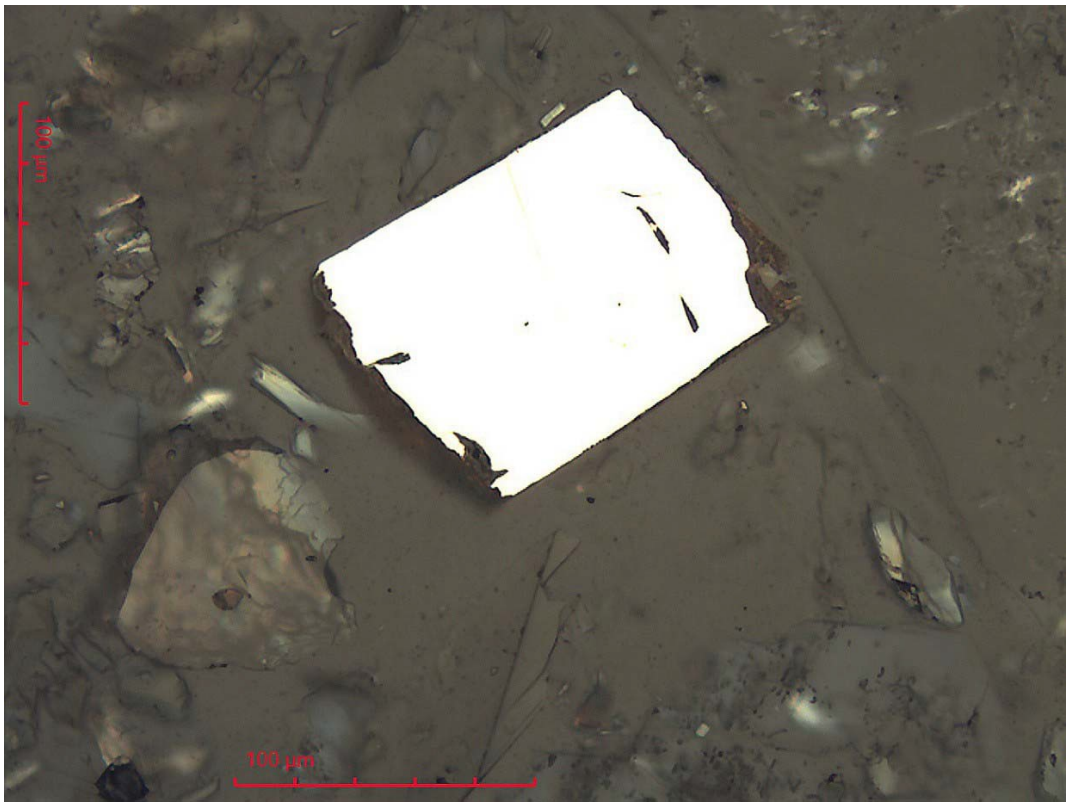

Figure 13.2 – Client Sample No. Laiva QD Avg
Cubic Grain of Liberated Pyrite – 200X RL

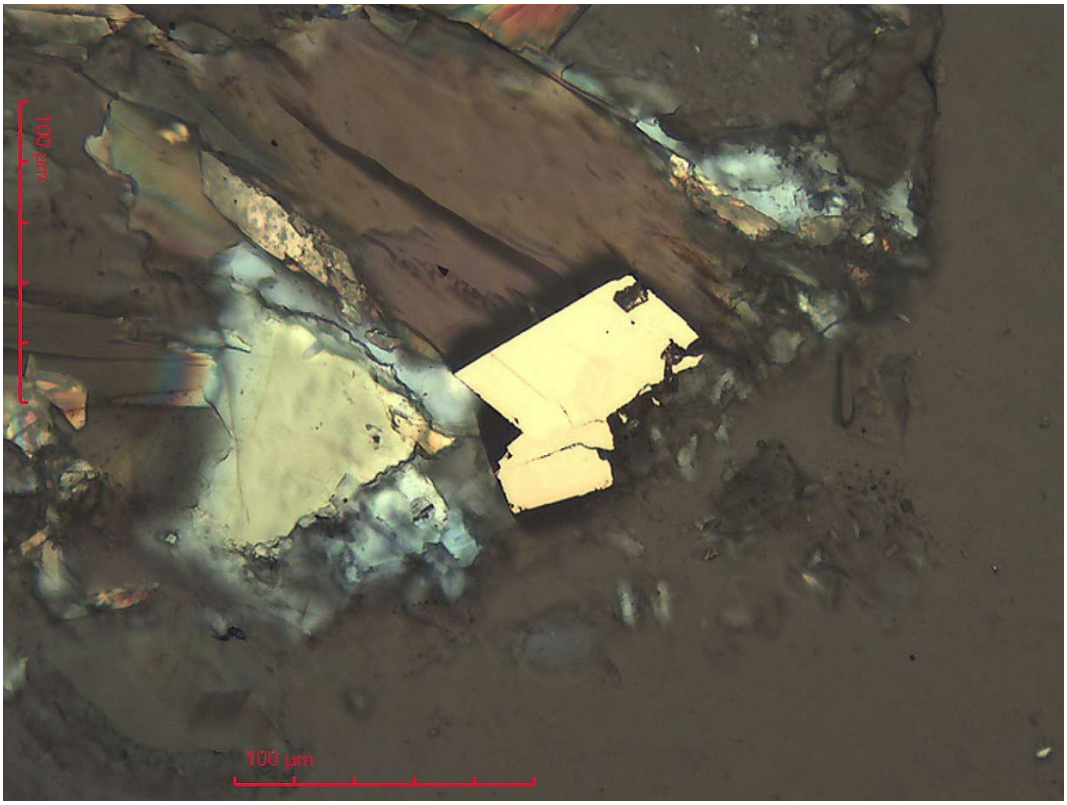

Figure 13.3 – Client Sample No. Laiva QD Avg
Rhomb Shaped Arsenopyrite with Amphibole and Quartz – 200X RL

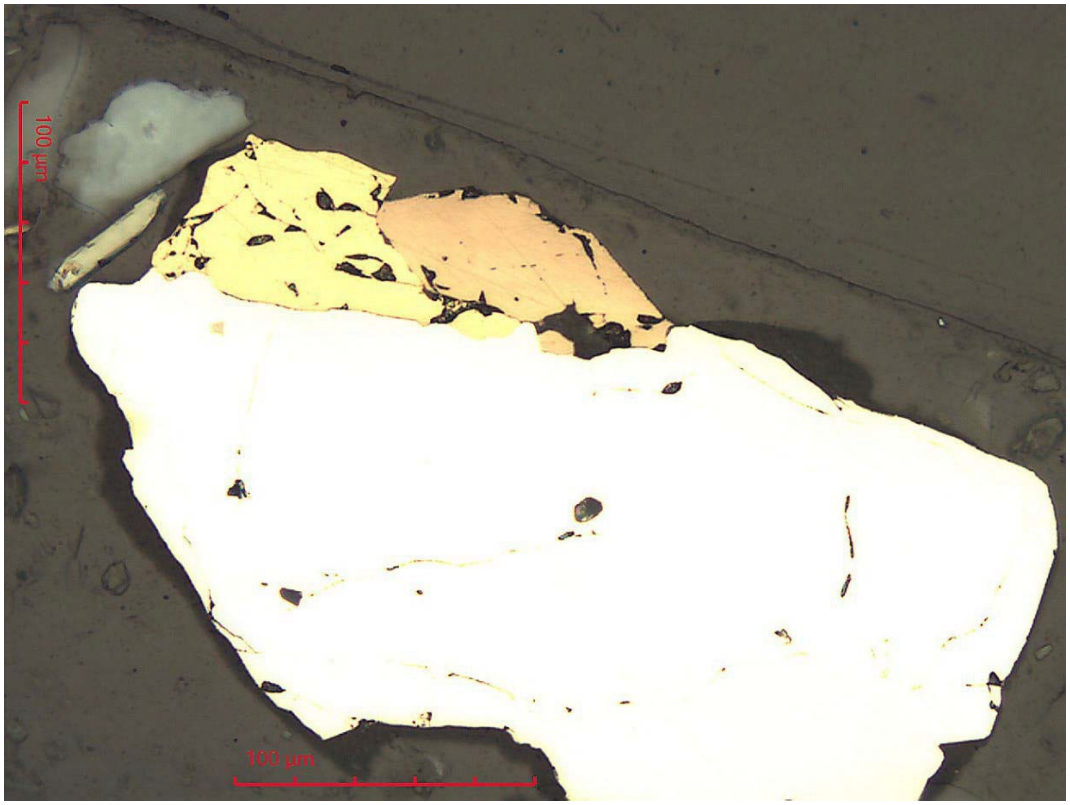

Figure 13.4 – Client Sample No. Laiva QD Avg
Large, Liberated Grain of Pyrite with Attachments
of Yellow Chalcopyrite and Brown Pyrrhotite –200X RL

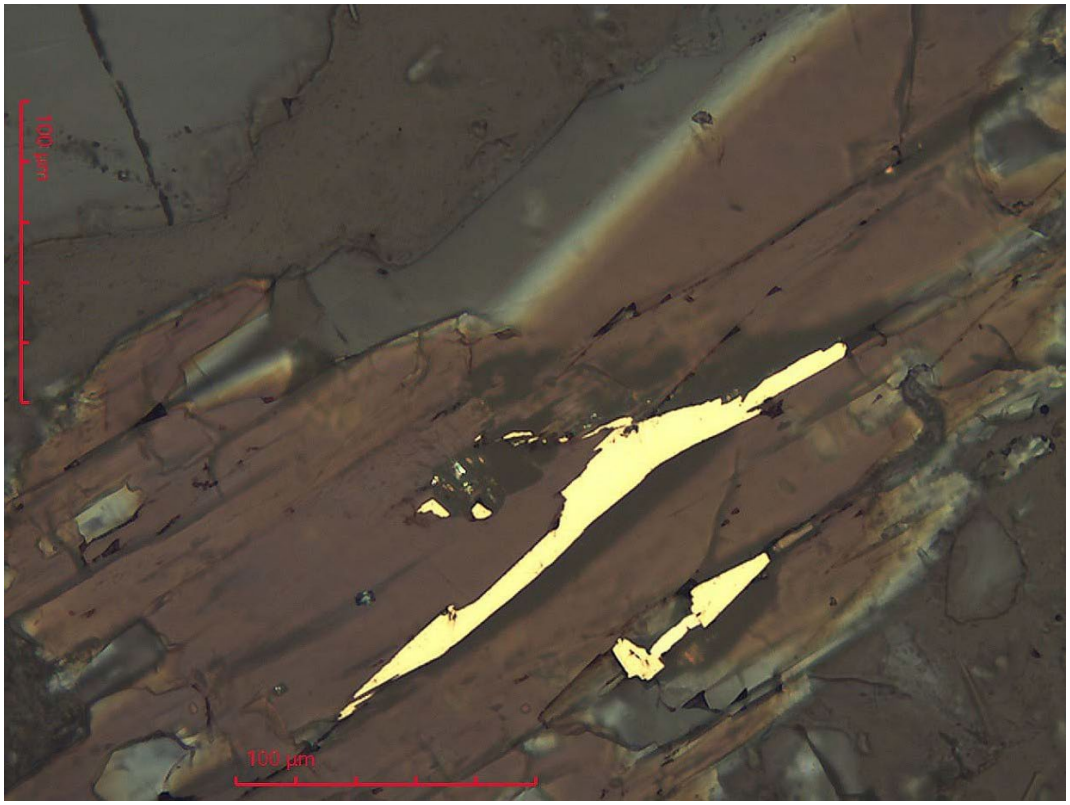

Figure 13.5 – Client Sample No. Laiva QD Avg
Thin Veins of Yellow Chalcopyrite between Amphibole Cleavage – 200X RL

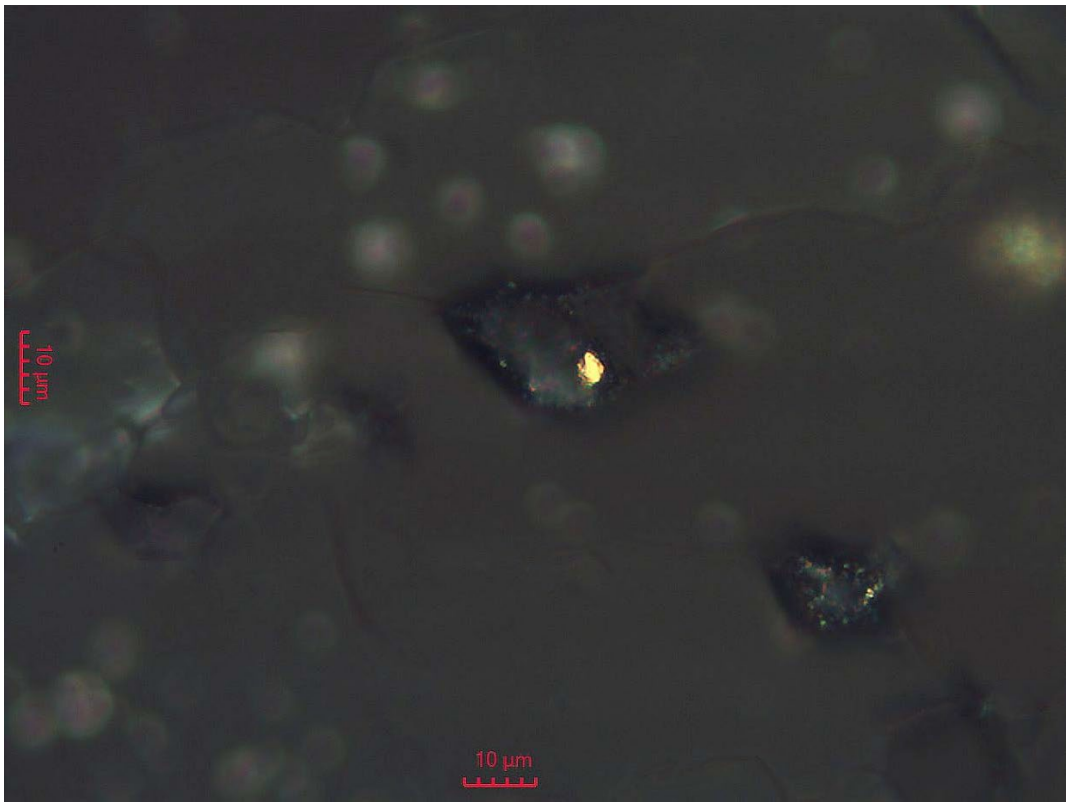

Figure 13.6 – Client Sample No. Laiva QD Avg Minute 4 µm Grain of Au in Quartz – 500X RL

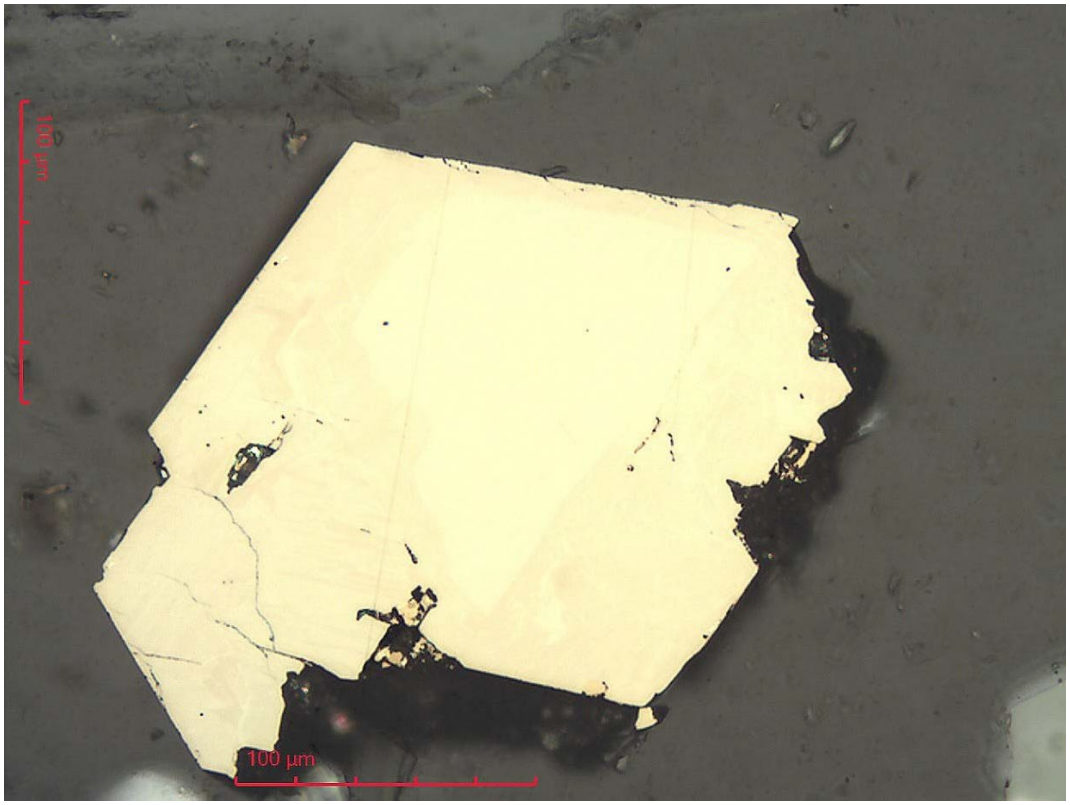

Figure 13.7 – Client Sample No. Laiva QD Avg
Large, Liberated Grain of Arsenopyrite – 200X RL

A summary of the XRD and XRF analyses of the single split is presented in Table 13.7 and 13.8, following this page.

Table 13.7 – Summary of XRD Results for QD Average Grade Composite

Mineral	Formula	Weight %
Plagioclase feldspar	$(Na,Ca)Al(Si,Al)_3O_8$	37
Clinoamphibole	$(Na,K)(Ca,Na)_2)(Mg,Fe,Al,)_5$ $(Si,Al)_8O_{22}(OH,F)_2$	30
Quartz	SiO_2	12
K-feldspar	$KAlSi_2O_8$	10
Mica/illite	$(K,Na,Ca)(Al,Mg,Fe)_2(Si,Al)_4O_{10}(OH,F)_2$	8
Unidentified	-------	<5

Table 13.8 – XRF Results for QD Average Grade Composite

	Shown in Wt %
Na_2O	3.12
MgO	4.29
Al_2O_3	13.1
SiO_2	59.5
P_2O_5	0.18
S	0.06
Cl	<0.02
K_2O	2.49
CaO	6.09
TiO_2	0.48
MnO	0.10
Fe_2O_3	6.80
BaO	0.04
	Shown in ppm
As	423
U	<10
Th	<10
Nb	<10
Co	<10
Cr	204
Cu	56
Sn	<20
Mo	<10
Ni	13
Pb	<10
Sr	334
V	146
W	67
Zn	47
Zr	86
Rb	77
Y	18

The XRF and XRD analyses of the QD sample confirm the mineralogical composition as compared to mineralogical speciation completed during field sample logging of individual drill samples and the optical mineralogical analysis of the metallurgical composite QD sample utilized for the RDi metallurgical test program.

13.5 Historical Metallurgical Test Work

All historical metallurgical test work results were evaluated in detail by BOYD to aid in the planning of the 2021 metallurgical test program.

Previously completed metallurgical test work utilized samples of considerably higher gold and sulfide grades than those values for the current resource. Therefore, the results from the previous test work represent primarily the highest-grade zones to be mined after restart. The results from the previous metallurgical test work completed on high-grade samples were primarily utilized to develop restart process plant operating parameters that ensure optimum gold recovery from the high-grade zones in the ore body.

A detailed description of the historical metallurgical test work is presented in the following sections.

13.5.1 Introduction

A preliminary metallurgical test program was completed utilizing a Laivakangas gold ore sample from Endomines Oy. The preliminary test program was completed at the Technical Research Centre of Finland Mineral Processing facilities in October and November 1997.

Gold grade in the sample tested was 7.7 g/t. Arsenic and copper content were 0.26 wt% and 0.02 wt% respectively. The main sulfides contained in the samples were: arsenopyrite, pyrrhotite, chalcopyrite, and loellingite.

Gold was found to occur as inclusions within arsenopyrite and loellingite. The largest gold grains were ranged to 30 – 400 µm but the majority were less 10 µm. Minor gold occurred in silicates.

The ore was found to be amenable to direct cyanidation with a gold extraction of 85% at a P80 = 75 µm after 24 hours of cyanide leaching in a stirred reactor. The reported cyanide and lime consumptions were both in the low range with a cyanide consumption of 0.5 kg/t and lime consumption of 1.3 kg/t.

Work Index determined by Mergan ball mill grinding was 15.3 kWh/h at a grind size of P_{80} = 60 µm.

Standard bulk flotation tests were completed on the ore sample. The sample was found to be amenable to flotation with a gold recovery of 85.4% into a concentrate that represented 3.2% of the mass of the feed sample.

Pilot scale test work on bulk samples collected by NORDIC was conducted in October and November 2007 at GTK/Mineral Processing, Outokumpu. The main objective of the test work was to define design criteria for AG milling and methods for processing mineralization by gravity concentration followed by cyanide leaching, and possibly flotation. The objective was to find the most economic process and the lowest costs demonstrating reasonable gold recovery.

Pilot plant test work was performed at GTK's pilot plant in Outokumpu between 12 October and 16 November 2007. The pilot plant operation consisted of the following process stages:

1. Crushing and homogenization of the samples.
2. Two-stage AG milling.
3. Gravity separation.
4. Fine flotation.
5. Thickening the produced process products for further leaching tests.
6. Bench-scale cyanide leach tests on selected products of the pilot plant.
7. Gravity separation.
8. Fine flotation.
9. Thickening the produced process products for further leaching tests.
10. Bench-scale cyanide leaching tests on selected products of the pilot plant.

13.5.2 Bulk Samples

Three bulk samples were collected by NORDIC in 2007 for the pilot scale test work. The samples were all located in Domain 2 in the North Pit and were collected from blasted rock in the base of the pit. Sample descriptions are given in Table 13.9 below.

Table 13.9 – Sample Descriptions

			Head Grade			
Sample ID	**Host Rock**	**Weight (tonnes)**	**Au (g/t)**	**As (%)**	**Cu (%)**	**S (%)**
Type A	Metavolcanic	110	1.37	0.046	0.016	0.215
Type B	Quartz Diorite - Sulphide Poor	300	2.69	0.022	0.018	0.113
Type D	Quartz Diorite - Sulphide Rich	340	1.17	0.054	0.009	0.183

Types B and D were described as sulphide poor and sulphide rich, respectively; however, the analytical results do not indicate a significant variance in Sulphur content between the two samples. Therefore, BOYD considers the Types B and D samples to represent the QD hosted mineralization as a whole. There are no detailed descriptions of the samples; therefore, it is not possible to comment on how the distinction was made between sulphide rich and sulphide poor ore types.

BOYD considers the samples collected in 2007 to be representative of the mineralization types and gold grades at Laiva. It would be beneficial in the future to submit additional test sample from the South Pit (Domain 1) and east of the Red Shear Zone (Domains 3 and 4).

13.5.3 Pilot Scale Test Work Results

Results from the pilot scale tests showed that the mineralization could be ground autogenously to the desired fineness P_{80} 75 to 85 µm with an energy consumption of 17 kWh/t to 20 kWh/t. In the tests with Type A mineralization, energy consumption to grind to P_{80} 88 µm was 16.9 kWh/t. In the tests with Type B mineralization, the respective figures were P_{80} 76 µm at 19.7 kWh/t. The majority of the grinding tests were run on Type D mineralization, which averaged P_{80} 87 µm with energy consumption of 19.0 kWh/t.

The principal flow sheet developed for gravity concentration included a Falcon C400 centrifugal separator rougher followed by an LG7 spiral cleaner. The spiral concentrate was a final product and taken out of the circuit, and Falcon tails and spiral tails were combined and further reported to a 150-liter flash flotation cell. From flash flotation, the concentrate was taken out as a final product, and the flash flotation tails reported to the primary classifying cyclone. The cyclone underflow reported into a pebble mill and the overflow reported to fine flotation as gravity tails. Thus, the gravity circuit concentrate, or “high-grade concentrate”, consisted of spiral concentrate and flash flotation concentrate.

The final step in the metallurgical test program included evaluation of the cyanide leachability of the high grade and low-grade samples produced during flotation and gravity concentration testing.

As the high-grade concentrate contained the majority of the recoverable gold, it was vital to achieve maximum gold recovery from it. Two process options were investigated.

- Intensive cyanidation followed by solid liquid separation and CCD wash, with direct electrowinning of the gold from the filtrate.
- Intensive CIL cyanidation followed by elution of the loaded carbon and electrowinning.

After a series of pilot bench scale electrowinning tests, it was determined that the very high copper concentrations in the pregnant liquor affected cell performance and the first option was rejected as impractical. The studies were therefore concentrated on the second option.

The final test work focused on re-grinding particle size, oxygen uptake rate, cyanide concentration, mixing and impeller selection, preg-robbing considerations, and pulp density, which were carried out on the high-grade concentrate.

The final results from high grade leach tests are presented in Table 13.10 below.

Table 13.10 – Extraction Data Concentrate Leach 25% and 40% Solids Kinetics

Test ID	Assay Head Au g/t	Calc. Head Au g/t	Residue Au g/t	Extraction Au %	NaCN kg/t	Lime kg/t	O_2 g/m^3	pH	Pulp Density % w/w	Pre Condition h	Leach Time h
YR 39	122	112	4.90	95.6	5.2	1.1	21	11.0	25	4	12
YR 40	122	101	5.66	94.4	5.2	1.0	22	11.0	40	4	12
YR 41	122	116	4.37	96.2	7.1	1.2	20	11.1	25	4	24
YR 42	122	104	4.25	95.9	5.9	1.0	22	11.0	40	4	24
YR 43	122	112	3.54	96.8	9.2	1.1	22	11.1	25	4	36
YR 44	122	102	3.63	96.5	8.2	1.1	21	11.2	40	4	36
YR 45	122	116	3.42	97.0	9.6	1.3	23	11.2	25	4	48
YR 46	122	108	3.42	96.8	8.7	1.0	22	11.0	40	4	48

Depending on the leach time, gold recovery from 95% to 97% was achieved from the high-grade concentrate. Low grade leach on gravity and flash float tails, confirming test work in December 2008, indicted that a 12 h reaction time was required and that gold extraction would be in the range 0.5 to 0.6 g/t. Total cyanide consumption was again 0.4 kg/t and lime consumption 0.5 kg/t.

Table 13.11, following this page, presents the results of the final low grade leach tests; after 16-24 h of leaching the residue assay showed a gold grade of 0.15 g/t as it in the beginning of the leach had been 0.73 g/t.

Table 13.11 – Results of the Final Low Grade Leach Tests

Test No.		37	38	39	40	41	42	43	44	Average
Leach Time	h	16	16	16	16	24	24	24	24	
Liquid Phase	ml g	814.0	815.0	807.0	809.0	813.0	815.0	806.0	807.0	810.8
Volume	g/t	542.8	543.6	537.7	539.2	542.1	542.9	537.1	538.0	540.4
Solids Mass		0.66	0.66	0.72	0.72	0.66	0.66	0.72	0.72	0.69
Head Assay	g/t	0.17	0.17	0.14	0.14	0.14	0.14	0.16	0.16	0.15
Residue Assay	µg/ml	0.38	0.43	0.40	0.39	0.36	0.37	0.38	0.37	0.39
Solution Assay	µg	309.3	350.5	322.8	315.5	292.7	301.6	306.3	298.6	312.1
Gold in Soln	µg	89.6	89.7	75.3	75.5	73.2	73.3	85.9	86.1	81.1
Gold in Residue										
Total Gold	µg	398.9	440.1	398.1	391.0	365.9	374.8	392.2	384.7	393.2
Extracted Gold	g/t	0.57	0.64	0.60	0.59	0.54	0.56	0.57	0.56	0.58
Calc Head	g/t	0.73	0.81	0.74	0.73	0.67	0.69	0.73	0.72	0.73
Calc Head/Assay Head		1.11	1.12	1.03	1.01	1.02	1.05	1.01	0.99	0.73

In order to complete historical resource evaluation and historical mine planning and scheduling, it was necessary to show the effect of head grade variation on both flotation and float tails leach recovery. Specially selected samples were taken covering a target head grade range of 0.5 g/t to 3.5 g/t by approximately 0.5 g/t intervals. Flotation after ball mill grinding and leaching of the flotation tails was carried out. Figure 13.8 below shows the leach tails versus flotation feed grade dependency.

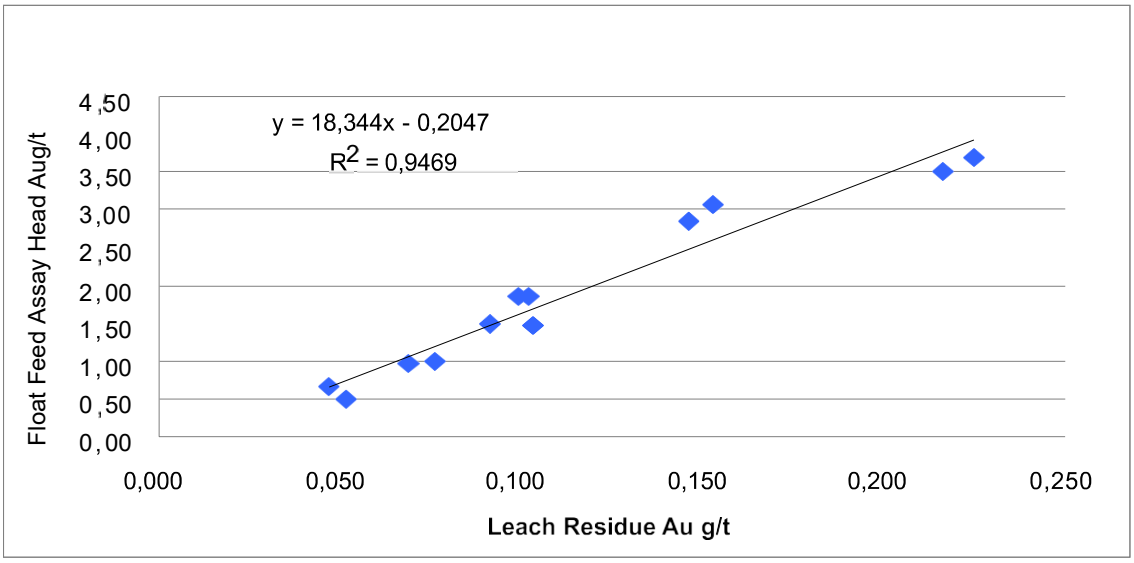

Figure 13.8 – Leach Residue vs Float Feed Assay Head

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14.0 MINERAL RESOURCE ESTIMATE

14.1 Overview

BOYD was requested by OTSO to provide an updated mineral resource estimate for its Laiva Gold Deposit located near the town of Raahe, in the Northern Ostrobothnia Region of Western Finland. BOYD had previously developed an initial NI 43-101 compliant mineral resource estimate in August 2017. An update to that estimate was completed by Coffey Geotechnics Ltd., a Tetra Tech Company (COFFEY) in 2019. The mineral resource estimate presented in this chapter supersedes all previous mineral resource estimates and was completed using new and updated exploration drilling. This mineral resource estimate was undertaken by Sam J. Shoemaker, Jr. of BOYD, QP. The effective date of this mineral resource estimate is 17 September 2021.

14.1.1 Classification

All the mineral resources as reported within this document have been classified in accordance with the Definition Standards of the Canadian Institute of Mining and Metallurgy:

A *measured mineral resource* is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling, and testing, and is sufficient to confirm geological and grade or quality continuity between points of observation.

A measured mineral resource has a higher level of confidence than that applying to either an indicated mineral resource or an inferred mineral resource. It may be converted to a proven mineral reserve or to a probable mineral reserve.

Mineralization or other natural material of economic interest may be classified as a measured mineral resource by the Qualified Person (QP) when the nature, quality, quantity, and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

An *indicated mineral resource* is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling, and testing, and is sufficient to assume geological and grade or quality continuity between points of observation.

An indicated mineral resource has a lower level of confidence than that applied to a measured mineral resource and may only be converted to a probable mineral reserve.

Mineralization may be classified as an indicated mineral resource by the QP when the nature, quality, quantity, and distribution of data allow a confident interpretation of the geological framework and a reasonable assumption of the continuity of mineralization. The QP must recognize the importance of the indicated mineral resource category to the advancement of the feasibility of the project. An indicated mineral resource estimate is of sufficient quality to support a pre-feasibility study, which can serve as the basis for major development decisions.

An *inferred mineral resource* is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

An inferred mineral resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings, and drillholes. Inferred mineral resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed pre-feasibility or feasibility studies, or in the life of mine plans and cash flow models of developed mines. Inferred mineral resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, and geological and grade/quality continuity of a measured or indicated mineral resource; however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an indicated or measured mineral resource. Under these circumstances, it may be reasonable for the QP to report an inferred mineral resource if the QP has taken steps to verify the information meets the requirements of an inferred mineral resource.

Measured and indicated mineral resources, when combined, are also referred to as M+I in tables in this report.

14.2 Laiva Gold Deposit Mineral Resource Estimate

The Laiva Gold Deposit was actively mined from 2011 through 2014 and was developed in two operating open pits, the North Pit, and the South Pit. Due to declining gold prices as well as lower than expected gold grades, the operation was closed until the project was acquired by FTR in July 2017. The mine was actively mined by FTR in the last half of 2018 through the first quarter of 2019 at which time the mine closed again because of excessive dilution created during the mining process. The deposit has had extensive exploration drilling before, during, and after its operating period and much of that data had not been considered in previous mineral resource estimates. The mineral resource estimate presented within this document adds that additional data to the previously used exploration data to produce a new mineral resource estimate in accordance with NI 43-101 standards.

14.2.1 Previous Mineral Resource Estimates

The most recent mineral resource estimate on the Laiva Gold Deposit was completed by COFFEY with an effective date of 18 September 2019. This estimate was reported to be NI 43-101 compliant. However, the mineral resources reported were global and not constrained by an economically optimized open pit. The results of this mineral resource estimate are shown in Table 14.1 below.

Table 14.1 – Previous Mineral Resource Estimate, Coffey Geotechnics 2019

Mining Method	Resource Classification	Tonnes	Au g/t	Au Troy Ozs
Open Pit	Measured	1,780,000	1.54	88,100
Open Pit	Indicated	5,811,000	1.56	291,500
Open Pit	***Measured + Indicated***	***7,591,000***	***1.56***	***379,600***
Open Pit	Inferred	24,677,000	1.53	1,214,700

The estimate in Table 14.1, Page 14-3, is superseded by the mineral resource estimates presented in this report.

14.2.2 Mineral Resource Estimation Procedure

BOYD developed three-dimensional (3D) geological models for the deposit using Maptek Pty. Ltd.'s Vulcan software. The procedures used to model and estimate the mineral resources consist of the following steps:

1. Assemble and validate the exploration (drillhole) database.
2. Load the exploration database into Vulcan and validate the results.
3. Develop 3D wireframe models of the mineralized domains and surrounding rock masses for each deposit using previously developed cross-sectional interpretation and advanced implicit modelling techniques.
4. Examine the various sampling lengths and establish a composite length for assay composites.
5. Create a block model based on the resource area geology and mineralized domains developed in Step 3 above.
6. Determine, based on lognormal probability charts of the assay data, the threshold gold grade to limit the area of influence of high-grade gold assays.
7. Flag the sample composites by their intersection with the various mineralized domains as developed in Step 3 above.
8. Using the composites from Step 7 above, develop variograms for gold grades in each potentially mineralized domain.
9. Develop grade estimation parameters and interpolate block grades.
10. Flag the blocks located above or below topography.
11. Run the post-interpolation script that determines mineral resource classification, block density, and rock codes for use in pit optimization.
12. Validate the block grade estimates using QQ plots and visual inspection against the underlying drillhole samples.
13. Export the block model into a format suitable for loading into the Geovia's Whittle pit optimizer.
14. Import the block model into the Whittle pit optimizer.
15. Determine economic pit limits to constrain the open pit mineral resource estimates using Whittle's pit optimization tools.
16. Import the pit optimization results into Vulcan.
17. Determine a grade shell of the Whittle pit results and flag the model for material within the Whittle pit limits and material outside of the Whittle pit limits.
18. Determine mineral resources inside the Whittle pit shell and underground mineral resources outside of the Whittle pit shell.

14.2.3 Units

The Laiva block model values are expressed in metric units. Metric units are used for coordinates and for the block sizes (the block size is fixed at 5 m by 5 m by 5 m sub-blocked to 2.5 m by 2.5 m by 2.5 m) and are also used for tonnage factors (metric tons per cubic meter). Grades are interpolated in metric units (grams per metric ton) since these are the units used by the lab to report the gold grades. The mineral resource estimate also reports gold ounces as troy ounces.

14.2.4 Laiva Gold Deposit Data

14.2.4.1 Drill Holes

All of the available exploration, grade control, channel samples, and blast holes were reviewed as part of the current mineral resource estimate. The total of these drill holes is 18,835 amounting to 260,657.9 meters in length. Drilling occurred in 1982 through 1986; 1999; 2005 through 2014; 2018 through 2019; and 2021. Drilling consisted of 536 diamond core exploration holes totaling 78,841.1 meters; 139 reverse circulation exploration holes totaling 16,363.7 meters; 78 channel samples totaling 361.9 meters; 14,918 blastholes totaling 92,137.0 meters; 201 grade control diamond core holes totaling 6,836.0 meters; and 2,963 grade control reverse circulation drill holes totaling 66,118.3 meters.

There are indications that additional drilling was completed in the past, but no records could be located to confirm this.

Much of the data is not considered NI 43-101 compliant and was removed from the final database used for the mineral resource estimate. This included any drillholes drilled prior to 2001 as well as all of the blasthole data. The authors consider that the blasthole data did not have the same level of QA/QC as the other exploration data and contained significant downhole gold contamination that tended to smear gold grades into adjoining waste zones. The result of this is a significantly reduced drill hole database that is NI 43-101 compliant. Table 14.2, following this page, shows the final NI 43-101 compliant drill hole database. Figure 14.1, Page 14-7, shows the collar locations for the exploration drilling database.

Table 14.2 – Summary of Exploration Drilling used in the Laiva Mineral Resource Estimate

Year	Diamond		Reverse Circulation		Channel		Grade Control-DH		Grade Control-RC		Totals All Types	
	Number	Meters	Number	Meters	Number	Meters	Number	Meters	Number	Meters	Number	Meters
2005	20	2,323.5	0	0.0	3	50.2	0	0.0	0	0.0	23	2,373.7
2006	85	16,626.3	0	0.0	0	0.0	0	0.0	0	0.0	85	16,626.3
2007	27	6,310.0	29	3,930.0	0	0.0	0	0.0	0	0.0	56	10,240.0
2008	66	14,026.2	63	9,389.7	0	0.0	0	0.0	0	0.0	129	23,415.9
2009	7	1,318.3	0	0.0	0	0.0	0	0.0	0	0.0	7	1,318.3
2010	54	5,398.1	0	0.0	12	103.2	0	0.0	0	0.0	66	5,501.3
2011	38	11,505.1	0	0.0	0	0.0	0	0.0	631	12,942.0	669	24,447.1
2012	28	3,532.5	0	0.0	46	112.1	0	0.0	1312	29,271.3	1,386	32,915.9
2013	0	0.0	0	0.0	0	0.0	201	6,836.0	845	19,837.0	1,046	26,673.0
2014	0	0.0	0	0.0	0	0.0	0	0.0	175	4,068.0	175	4,068.0
2015	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0
2016	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0
2017	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0
2018	0	0.0	47	3,044.0	0	0.0	0	0.0	0	0.0	47	3,044.0
2019	22	2,125.0	0	0.0	0	0.0	0	0.0	0	0.0	22	2,125.0
2020	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0	0	0.0
2021	78	7,458.8	0	0.0	0	0.0	0	0.0	0	0.0	78	7,458.8
Totals	**425**	**70,623.5**	**139**	**16,363.7**	**61**	**265.5**	**201**	**6,836.0**	**2,963**	**66,118.3**	**3,789**	**160,206.9**

14.2.4.2 Assays

Of the 131,290 gold assays available (in the NI 43-101 compliant database) on 12 September 2021, all were used for the mineral resource estimation. For unsampled intervals, gold values were set to 0.000 g/t. Assay values used included both fire and PAL assays. Fire assays were always used except where only PAL assays were available. Total assayed sample length is 131,793 m.

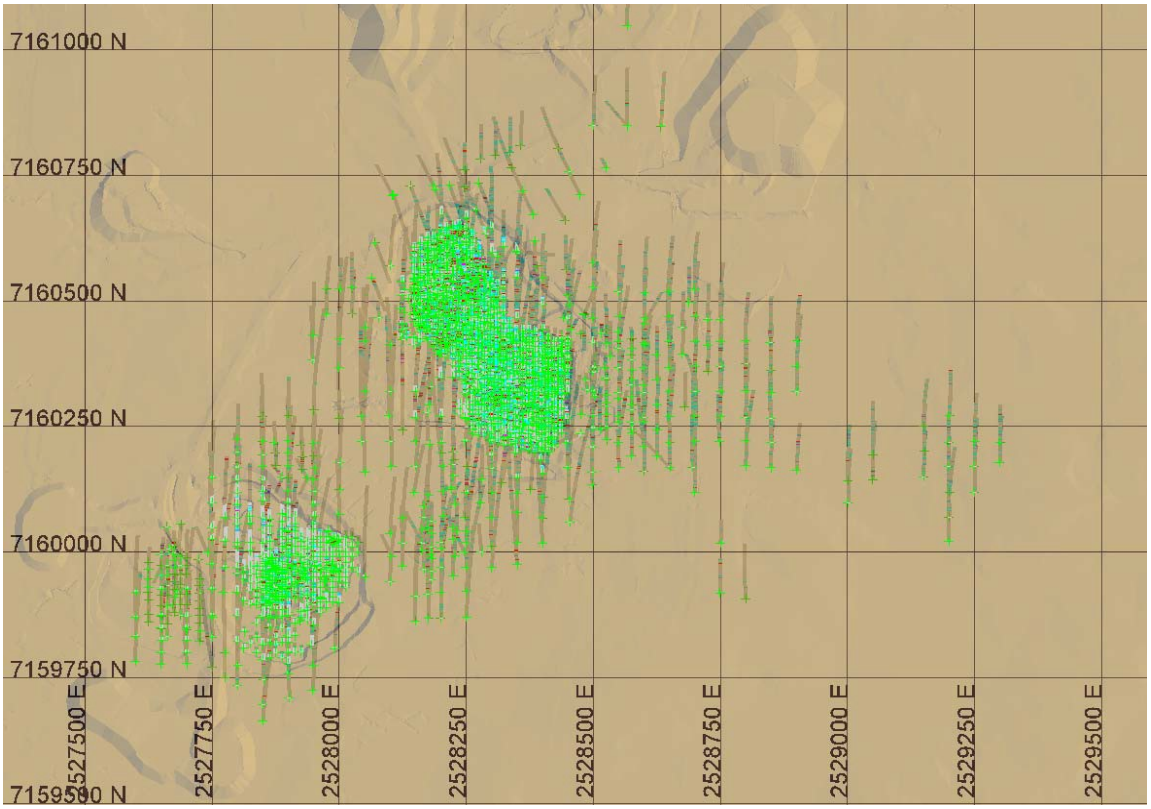


Figure 14.1 – Laiva Drill Hole Locations

14.2.4.3 Density

There have been 5,876 density measurements completed on the Laiva Gold Deposit to date. Densities used in the block model are shown in Table 14.3 below.

Table 14.3 – Densities Used in the Block Model

Rock Type	Rock Description	Average Density (t/m^3)
GR	Granite	2.616
MIR	Dolerite	2.817
MV	Mafic Volcanics	2.785
QD	Quartz Diorite	2.719
UNKN	Unknown	2.718

In areas not covered by the geologic model (described below) a density of 2.718 tonnes/m^3 was used. The impact of gold grade versus density was also examined. This examination showed that there was little to no correlation between gold grade and density. Figure 14.2, following this page, shows the relationship between the gold value and density.

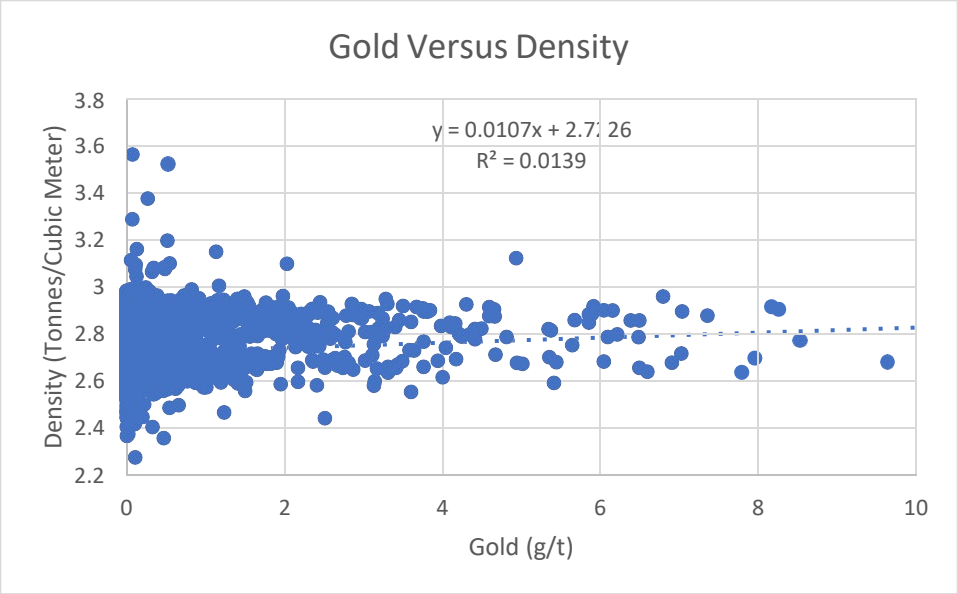


Figure 14.2 – Gold Values Versus Density

14.2.4.4 Topography

The topography of the area around the Laiva Gold Deposit is shown in Figure 14.3, following this page. All contours are expressed in meters above sea level. Minor contours are every 5 m while major contours are every 10 m. The topographic is based on the post mining 2014 topography as well as the progress maps for the 2018 and 2019 mining operations. The updated topography is the result of deducting the 2018 and 2019 mining from the post mining 2014 topography.

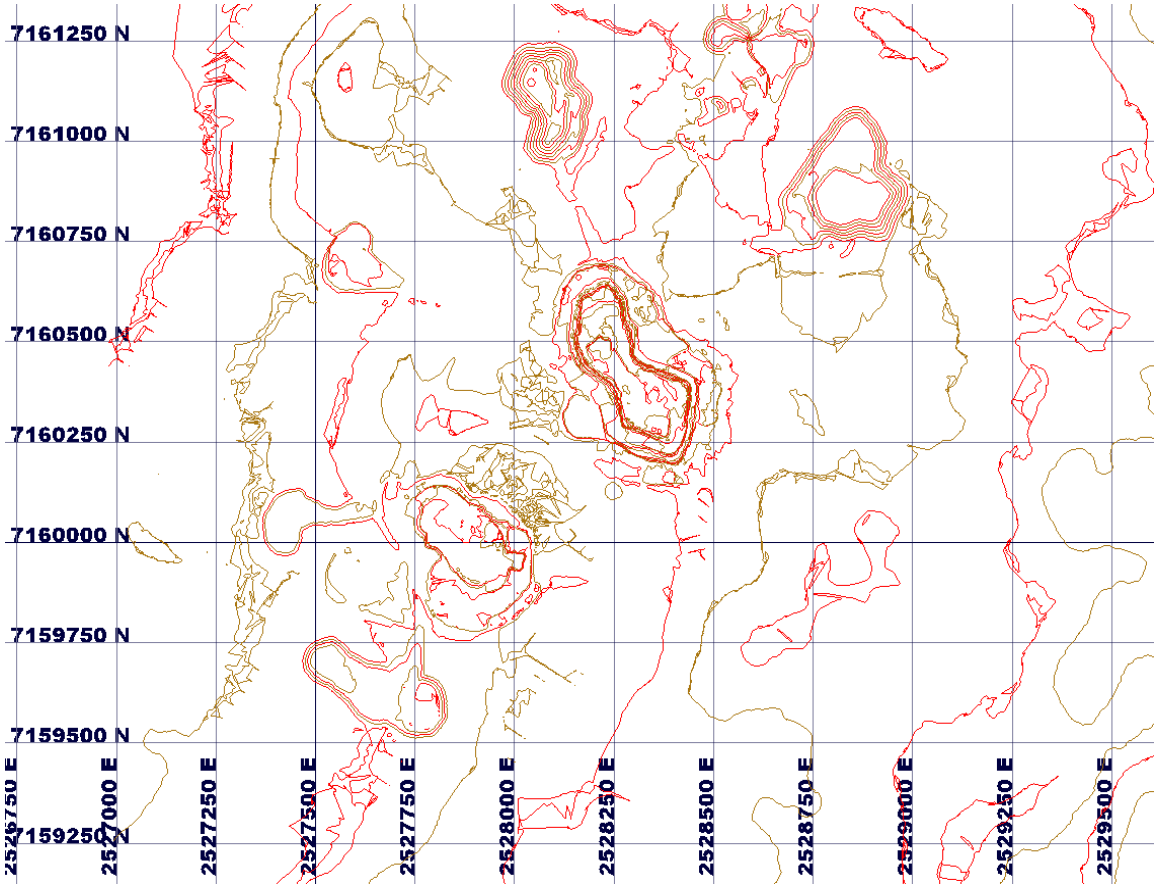


Figure 14.3 – Laiva Topography

14.2.5 Laiva Gold Deposit Data Analysis

14.2.5.1 Geologic Model

The geology of the Laiva Gold Deposit is made up of various intrusive units. All of the units can have gold mineralization although granite plutons, granite dikes and sills, and dolerite dikes were excluded from any grade estimate. A model developed in 2015 by SRK continues to be the best interpretation of the geology at the property. This model was used to define the general geology within the deposit as well as excluding granites and dolerites from grade estimation. In areas not covered by the geologic model (due to lack of information) the geology was flagged as “unknown”. The majority of mineralization occurs within shear zones in the quartz diorite unit.

Figure 14.4, following this page, shows a plan view of the geologic model.

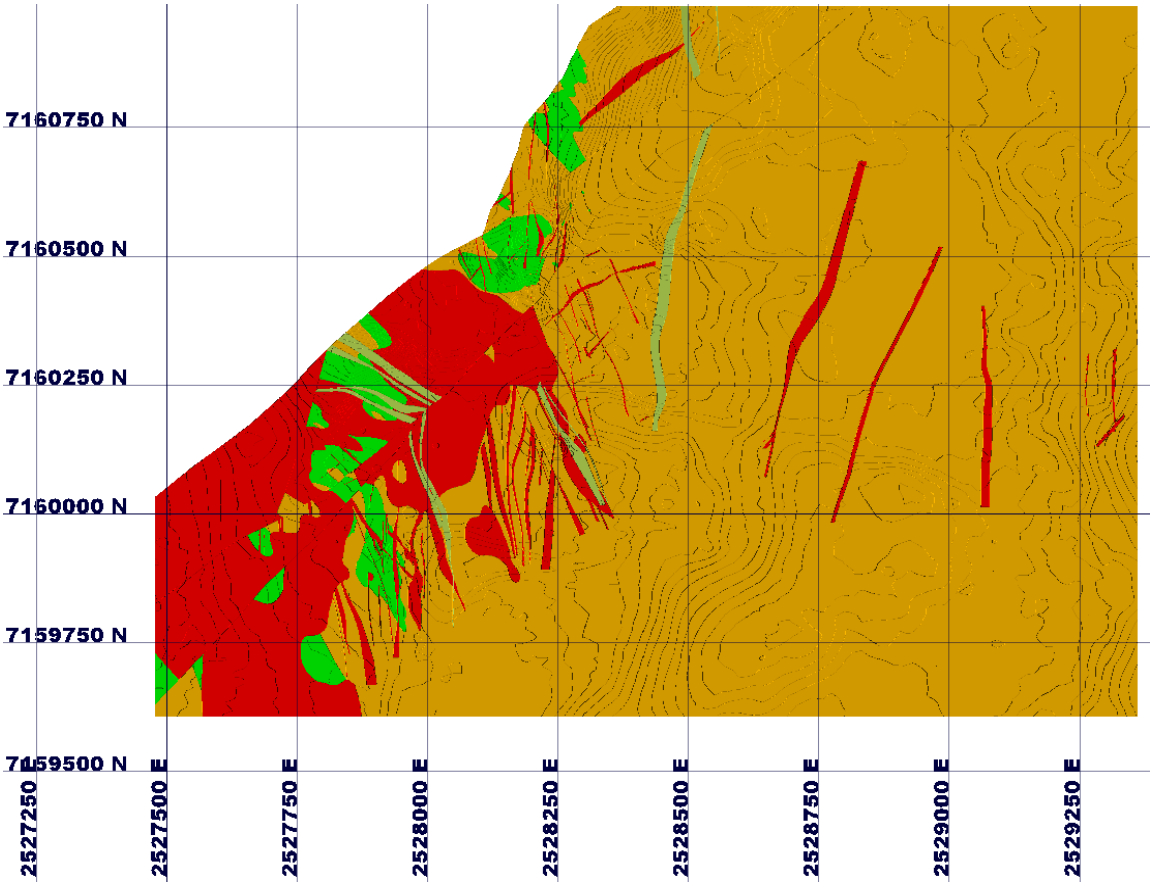


Figure 14.4 – Laiva Geologic Model (red = granites, green = uralite porphyry, light green = dolerites, tan = quartz diorite)

14.2.5.2 Structural Domains and Mineralized Solids

Data from geologic field mapping as well as geologic and assay data from the drill hole database were used to determine four distinct structural domains to develop structural orientations for variogram development as well as the limiting mineralized grade shells. The process involved examining the gold bearing quartz veins and structures and determining an average azimuth and dip for each domain.

A significant crossing structure cuts across the center of the deposit striking to the northeast. This structure, called the Red Shear zone, breaks the deposit into two major structural domains (east and west) and controls the apparent strikes on both sides of the deposit. Generally, to the west of the Red Shear zone mineralized structures strike to the northeast and dip to the southeast. On the east side of the Red Shear zone, mineralized structures strike due east to slightly southeast and dip to the south and southwest.

Domains were identified by common strikes as well as proximity to the Red Shear zone. Using this analysis, four distinct structural domains were identified:

- Domain 1 – Mineralization within this domain averaged an azimuth of 76 degrees with a -80-degree dip to the southeast. This domain is mainly in the South Pit area west of the Red Shear zone.

- Domain 2 – Mineralization within this domain averaged an azimuth of 72 degrees with a -80-degree dip to the southeast. This domain adjoins the other three domains (1, 3, and 4) and is located west of the Red Shear zone as well as domains 1 and 2. It occurs in the North Pit areas.

- Domain 3 – Mineralization within this domain averaged an azimuth of 115 degrees with a -80-degree dip to the south. This domain adjoins domains 2 and 4 and is located west of the Red Shear zone. This domain is in the North Pit area only.

- Domain 4 - Mineralization within this domain averaged an azimuth of 95 degrees with a -80-degree dip to the southwest. It occurs east of the Red Shear zone and adjoins the other domains (1, 2, and 3). It is in both the North and South pits.

Overall structural domain areas are shown in Figure 14.5 below.

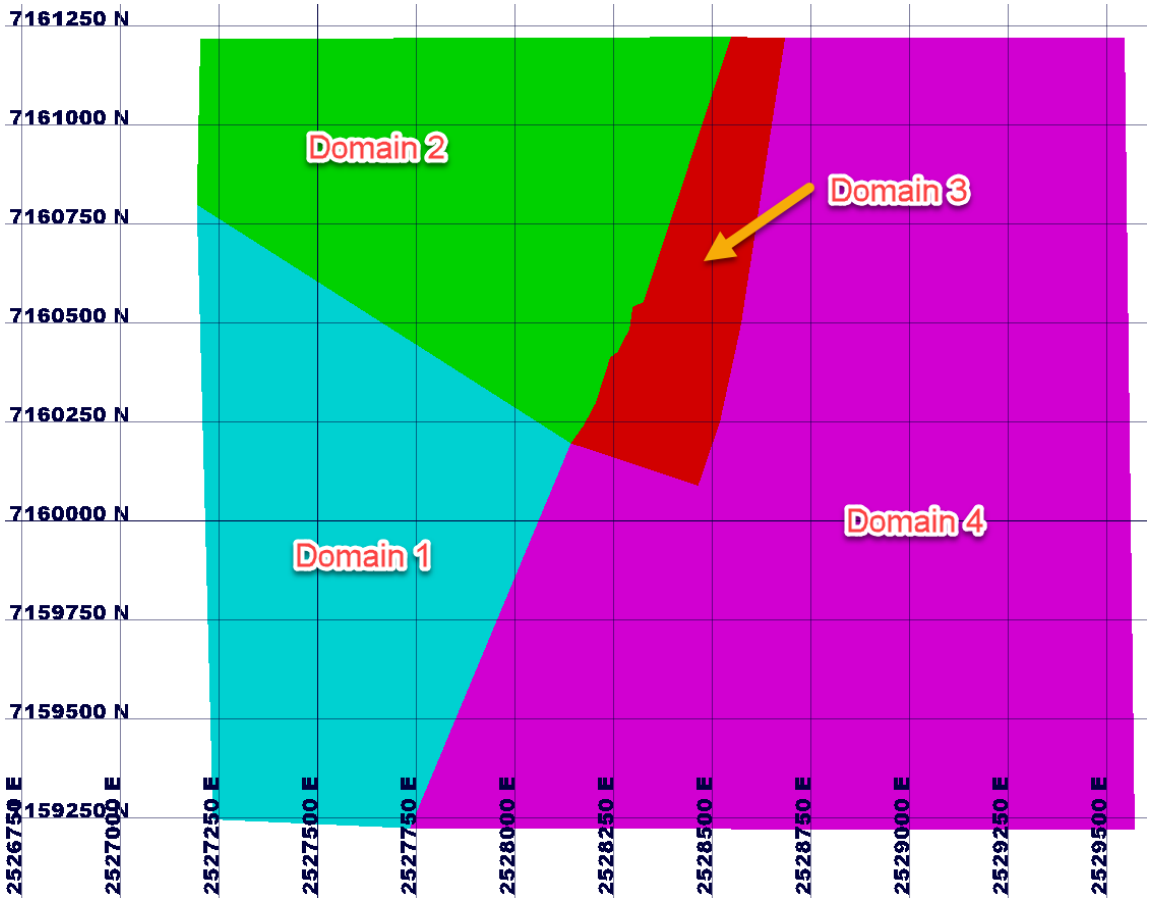


Figure 14.5 – Laiva Structural Domains

To produce the individual mineralized solids, a grade shell was generated using the observed strikes and dips as well as a trend surface for each domain at a gold cutoff grade of 0.3 g/t. These shells were then checked against the underlying drillhole assays to ensure that the shell represented the gold cutoff selected. Figure 14.6 below, shows the interpreted structural domains as well as the mineralized solids.

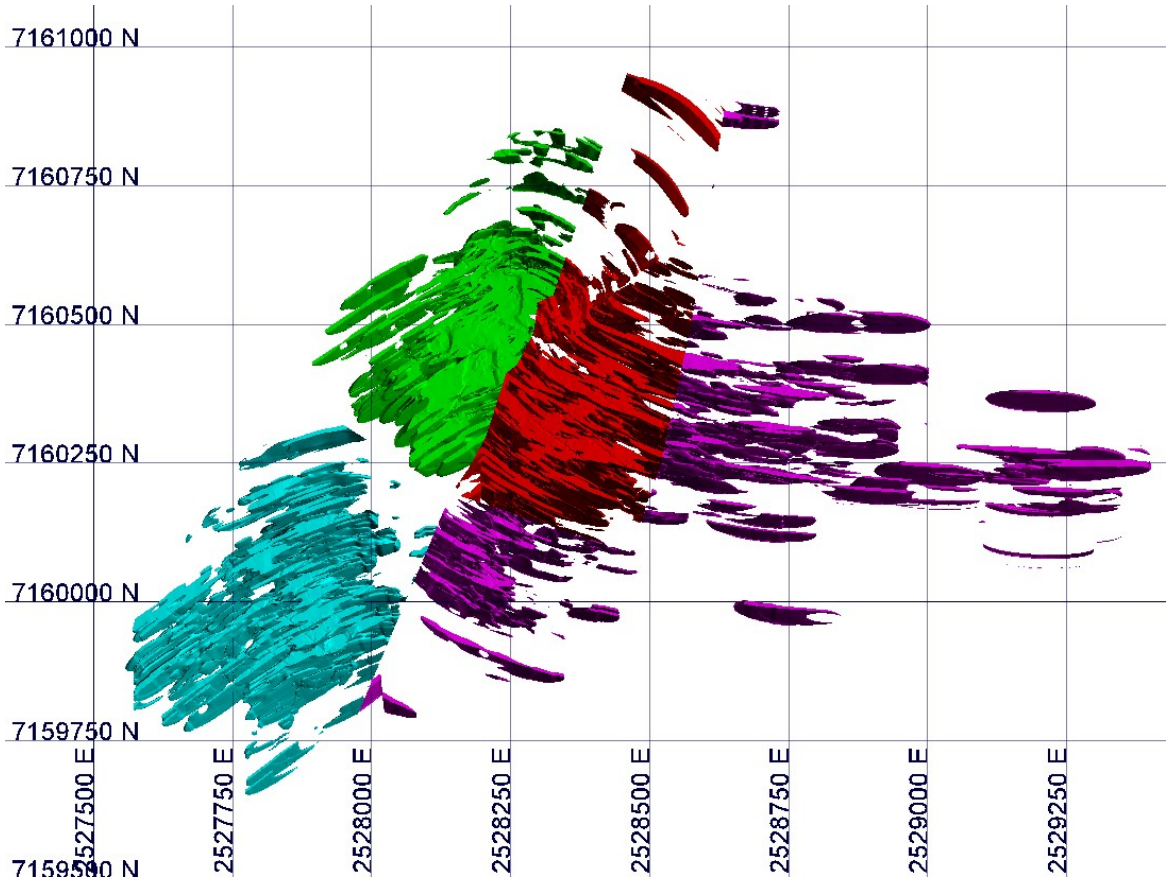


Figure 14.6 – Laiva Structural Domain Orientation and Mineralized Solids

These four domains were used to flag the composites for later variography as well as statistics.

14.2.5.3 Drillhole Descriptive Statistics

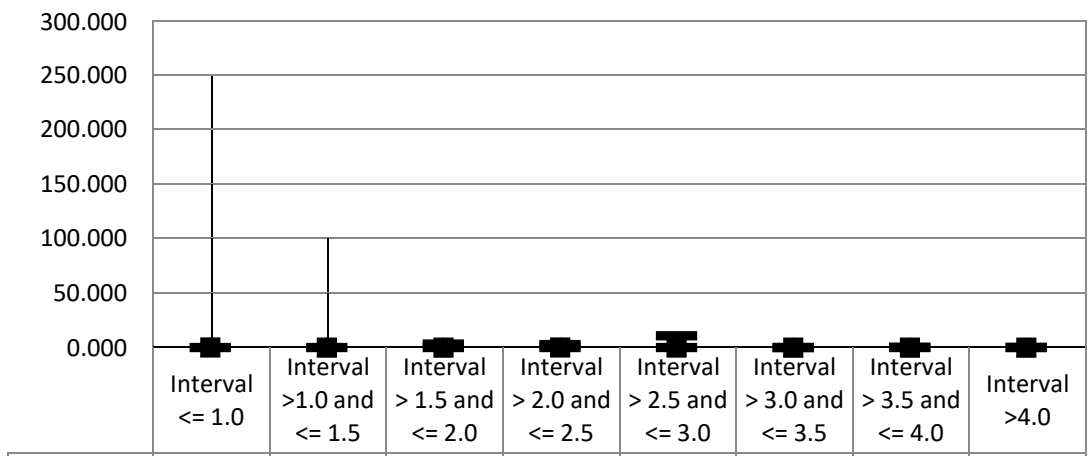
Descriptive statistics were generated for each individual domain, as well as the overall exploration database for gold. The results of this analysis are shown in Table 14.4, following this page.

Table 14.4 – Drillhole Descriptive Statistics

Item	Domains All	1	2	3	4
Number of samples:	44,679	5,150	17,021	19,978	2,530
Minimum:	0.010	0.010	0.010	0.010	0.010
Maximum:	240.000	65.100	46.100	93.300	240.000
Range:	239.990	65.090	46.090	93.290	239.990
Average:	1.011	1.135	1.057	0.970	0.781
Standard deviation:	2.458	2.781	1.949	2.260	5.001
Variance:	6.042	7.734	3.799	5.108	25.010
Coef. of variance:	2.431	2.450	1.844	2.330	6.403

14.2.5.4 Compositing

Sample length statistics were run on the assay database examining the number of samples for sample lengths in 0.5 m increments from 1.0 m through a total length of 4.0 m. The purpose of this analysis is to determine what sample length was associated with the total number of samples. The boxplot in Figure 14.7 below, shows the results of this analysis.



	Interval <= 1.0	Interval >1.0 and <= 1.5	Interval > 1.5 and <= 2.0	Interval > 2.0 and <= 2.5	Interval > 2.5 and <= 3.0	Interval > 3.0 and <= 3.5	Interval > 3.5 and <= 4.0	Interval >4.0
Q1	0.001	0.001	0.001	0.023	0.005	0.030	0.000	0.010
▬ Min	0.000	0.000	0.000	0.001	0.000	0.030	0.000	0.000
■ Median	0.060	0.055	0.029	0.070	0.044	0.030	0.001	0.010
▲ Mean	0.422	0.353	0.185	0.288	1.041	0.030	0.085	0.013
▬ Max	240.000	91.005	3.180	2.620	10.700	0.030	0.254	0.100
Q3	0.313	0.215	0.147	0.286	0.081	0.030	0.191	0.010
NSamples	128776	2305	147	19	11	1	3	28

Figure 14.7 – Laiva Composite Lengths

In examining the results of this analysis, most samples with potentially economic gold mineralization were taken at a length of 1.0 m or less. A total of 98.1% of all assays were taken at 1 meter or less containing 98.4% of the total contained metal. Based on this, a composite length of 1.0 m was selected and applied within the confines of the mineralized domains. Composites less than 1.0 m were divided by the run length (1.0 m). This composite length was selected to better reflect the actual breakdown of the mineralization in the individual drillholes within each mineralized zone.

14.2.5.5 High Value Gold Grade Limits

High outlier metal values can skew the resulting grade estimate if they are not accounted for with some sort of limitation or grade capping value applied to the underlying assay database. To determine this, a lognormal probability plot was generated for gold in each mineralized domain.

To determine high-value gold grade outliers, several methods were considered. These included a 1 troy ounce gold grade cap, the mean plus the standard deviation, four times the mean, five times the mean, lognormal, and decile analysis. All of these methods were reviewed, and the resulting potential grade caps/threshold were determined. For the Laiva Gold Deposit, the lognormal graph was considered the best method to establish a capping/threshold value. This is due to the very smooth lognormal results in all estimation domains.

Threshold metal grades were selected from the lognormal plot at the point where the data starts to break up or where there is a significant slope change in the plot. The lognormal probability plots for gold found in each mineralized domain are shown in Figures 14.8 through 14.11, following this page.

The lognormal probability graphs were used to determine a gold threshold grade to limit the area of influence of gold grades higher than the threshold. The area of influence was developed using indicator variograms to determine the size and extents of above threshold gold-bearing areas by producing a high gold grade search ellipsoid. This search ellipsoid was used to determine the area of influence of above threshold gold grades. This process was completed on all the potentially mineralized domains and the selected metal threshold grades are shown in Table 14.5, Page 14-17.

Threshold gold grades were applied during the grade estimation runs to limit the influence of the higher-grade outliers in the composites. The extreme outliers were used to hard cap gold grades at gold values that exceeded this number. This cap was determined using a lognormal graph and selecting a value where the extreme outliers appeared to lose lognormal continuity.

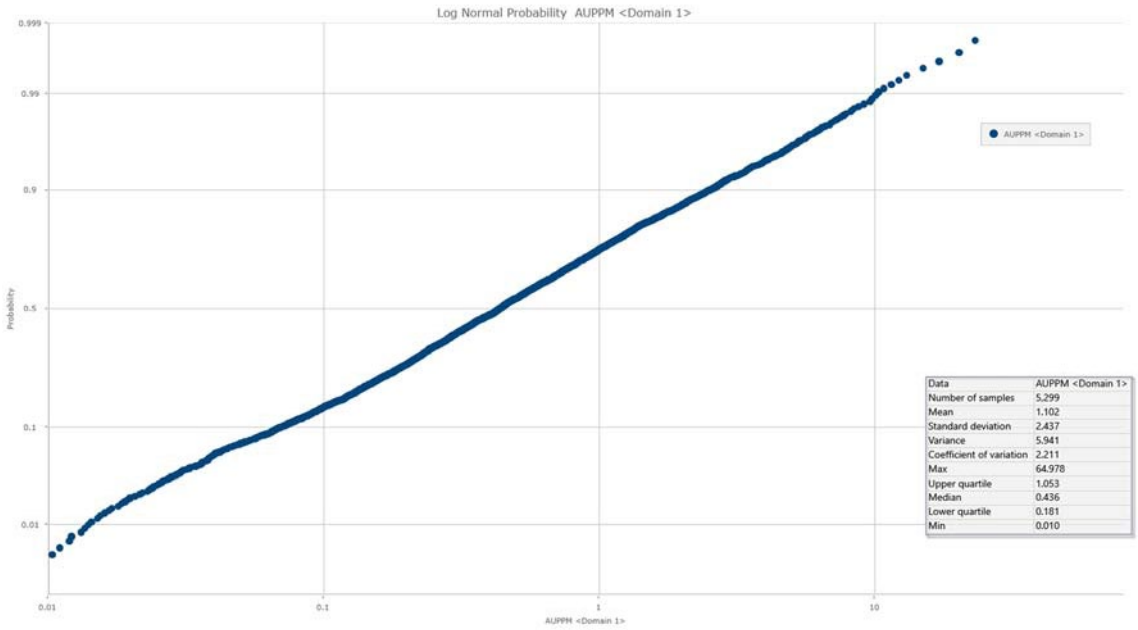


Figure 14.8 – Log-Normal Probability Plot Domain 1

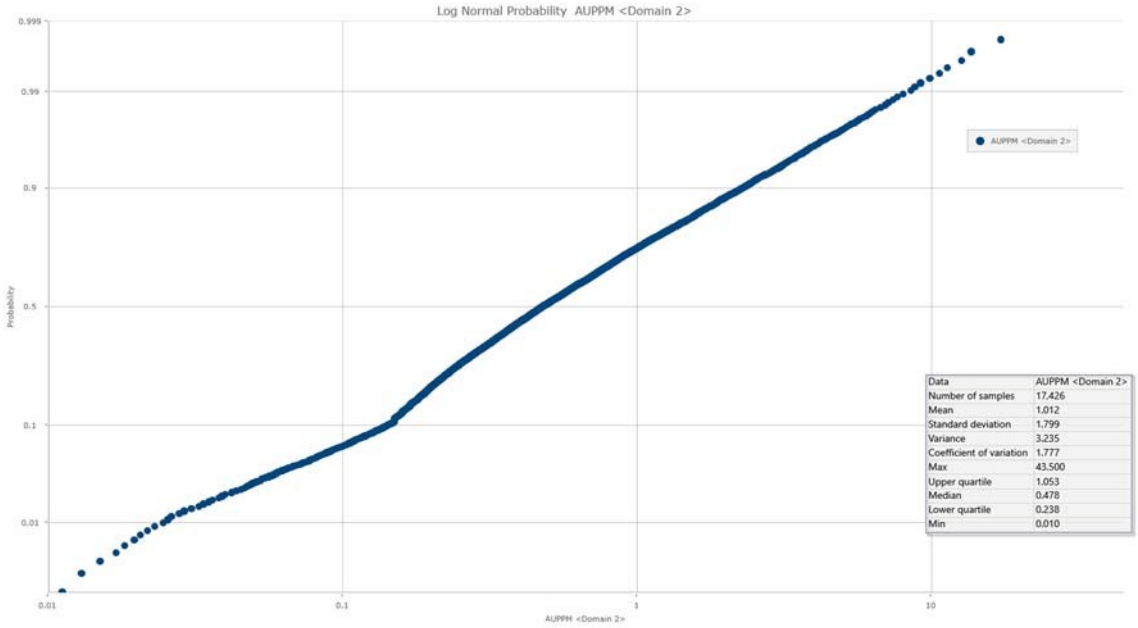


Figure 14.9 – Log-Normal Probability Plot Domain 2

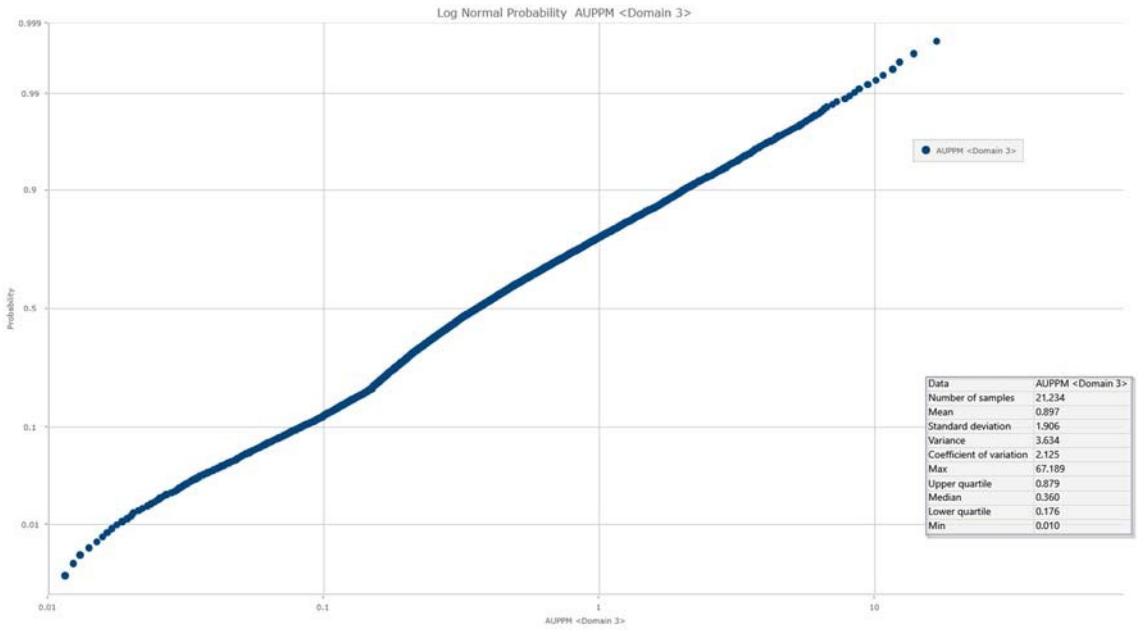


Figure 14.10 – Log-Normal Probability Plot Domain 3

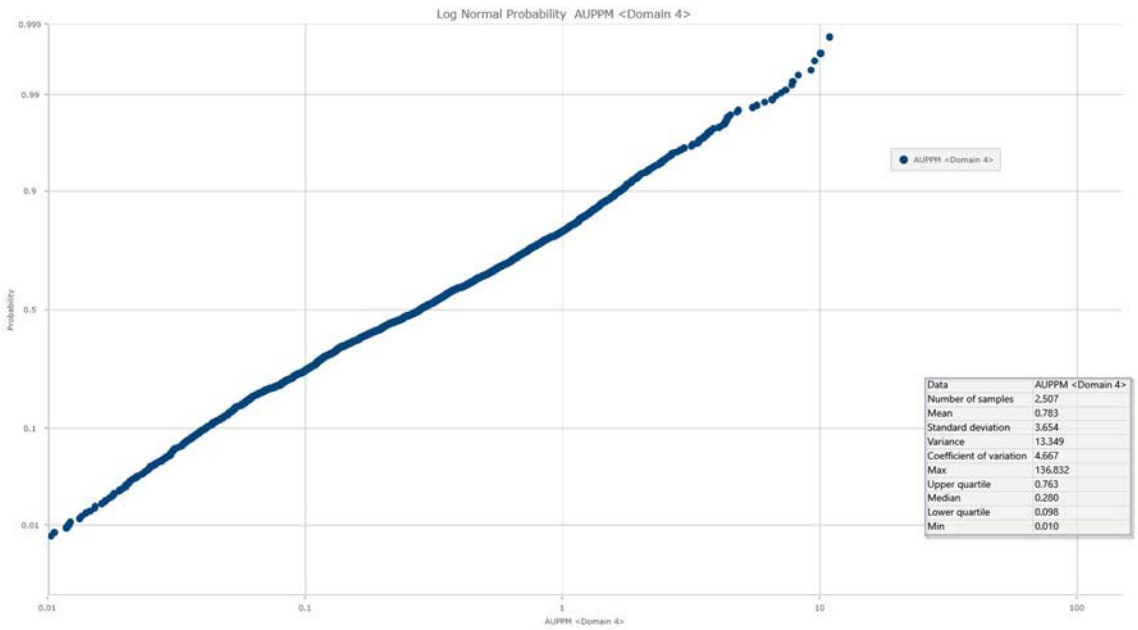


Figure 14.11 – Log-Normal Probability Plot Domain 4

Table 14.5 – Laiva Outliers by Domain

Item	Domain 1	Domain 2	Domain 3	Domain 4
Extreme Outliers (g/t)	32.0	27.0	28.0	12.0
Threshold Cap (g/t)	25.0	25.0	25.0	10.0
Azimuth (degrees)	76.0	72.0	115.0	95.0
Plunge (degrees)	0.0	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0	-80.0
Range - Major (m)	16.0	13.9	13.7	25.4
Range - Semi-Major (m)	7.7	4.2	9.7	25.1
Range - Minor (m)	2.5	1.1	2.2	3.1

14.2.6 Search Ellipsoid

The search ellipsoids for the grade estimation were developed using variograms for each domain. Variograms were run in each domain for gold using the same structural orientations used to develop the mineralized solids. These variograms are shown in Figures 14.12 below, and Figures 14.13 through 14.15, following this page.

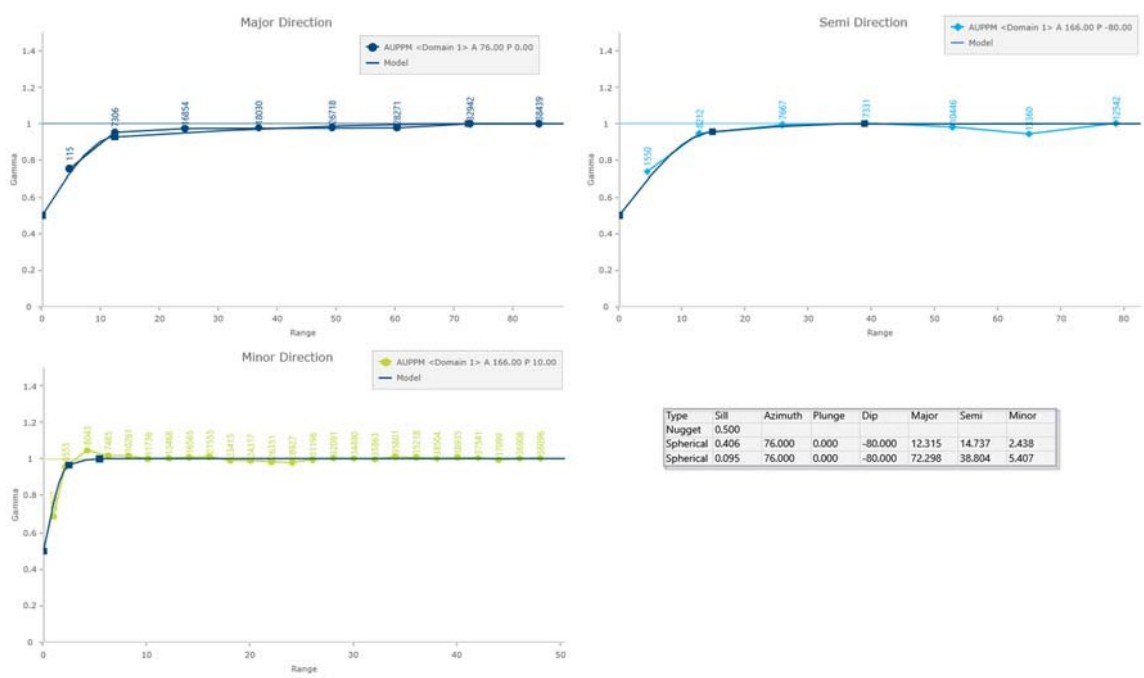
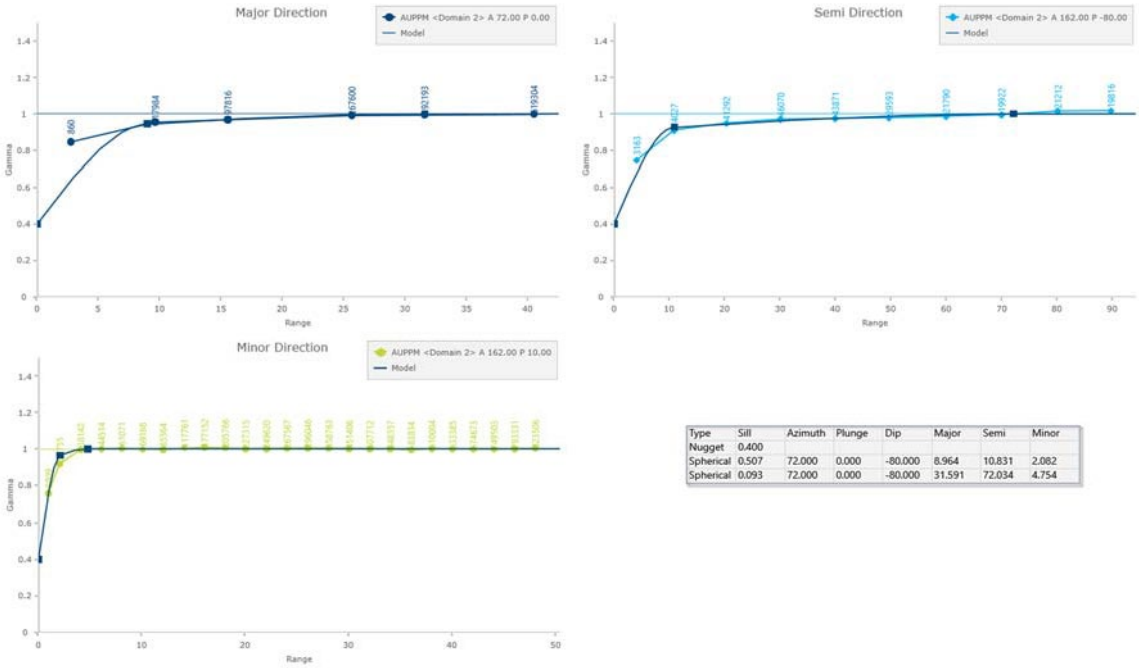
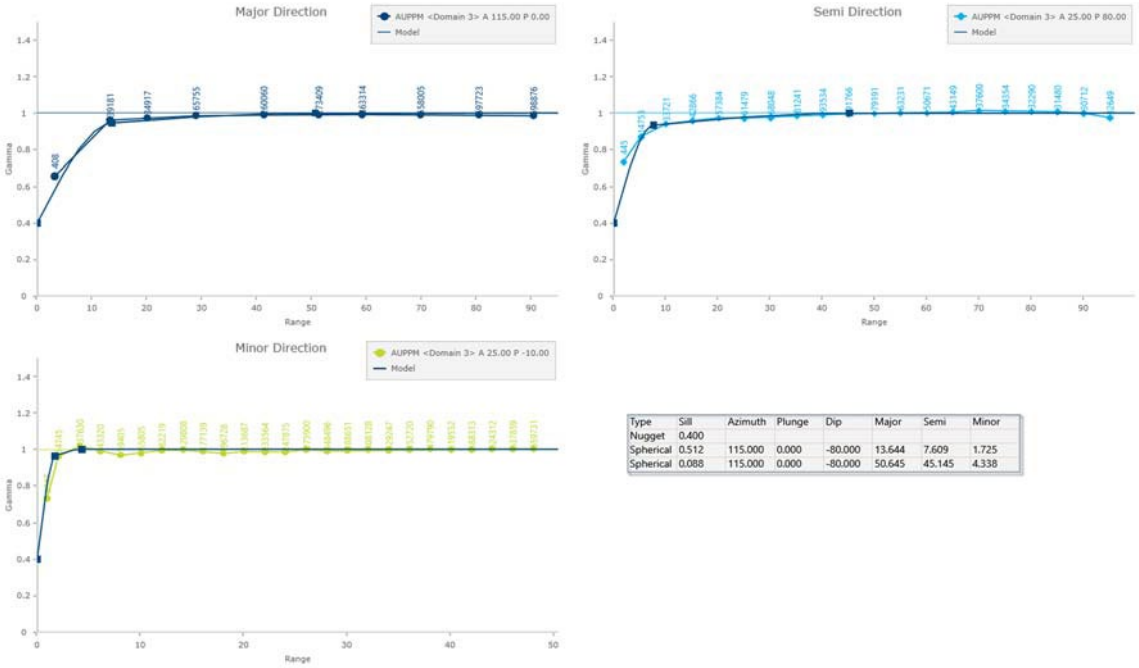


Figure 14.12 – Domain 1 Variograms



Type	Sill	Azimuth	Plunge	Dip	Major	Semi	Minor
Nugget	0.400						
Spherical	0.507	72.000	0.000	-80.000	8.964	10.831	2.082
Spherical	0.093	72.000	0.000	-80.000	31.591	72.034	4.754

Figure 14.13 – Domain 2 Variograms



Type	Sill	Azimuth	Plunge	Dip	Major	Semi	Minor
Nugget	0.400						
Spherical	0.512	115.000	0.000	-80.000	13.644	7.609	1.725
Spherical	0.088	115.000	0.000	-80.000	50.645	45.145	4.338

Figure 14.14 – Domain 3 Variograms

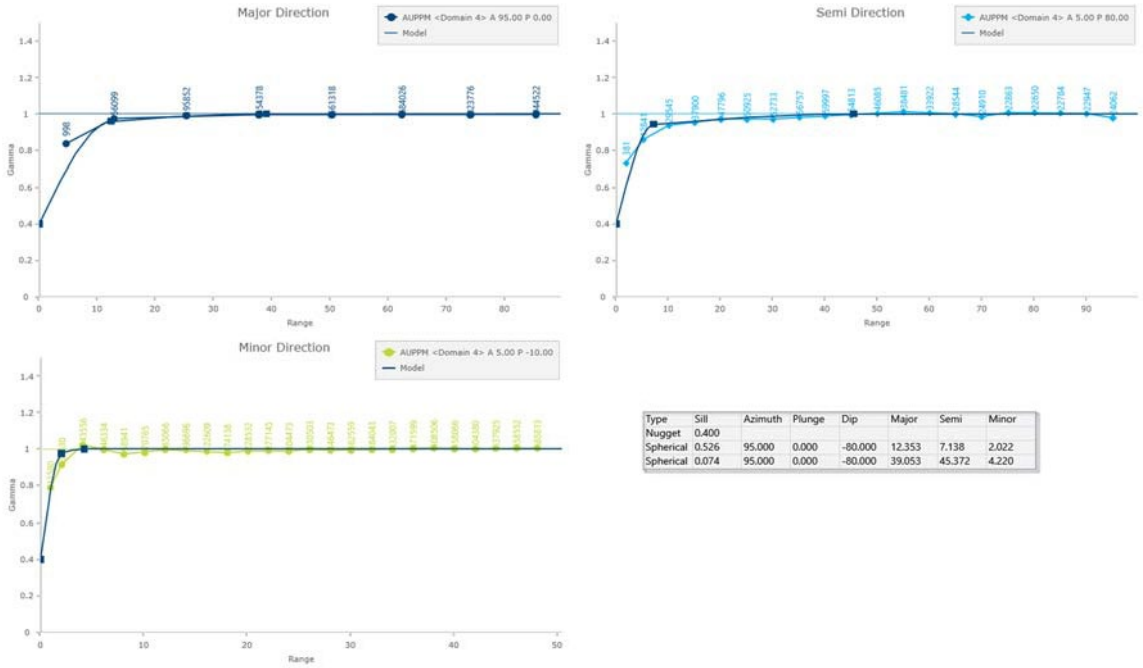


Figure 14.15 – Domain 4 Variograms

Based on these analyses, the search ellipsoid for each mineralized domain was established as shown in Table 14.6 below, and Tables 14.7 through 14.9, following this page.

Table 14.6 – Domain 1 Search Parameters

Item	Pass 1	Pass 2	Pass 3
Search Parameters			
Major Range (m)	72.3	72.3	72.3
Semi-Major Range (m)	38.8	38.8	38.8
Minor Range (m)	5.4	5.4	5.4
Azimuth (degrees)	76.0	76.0	76.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0

Table 14.7 – Domain 2 Search Parameters

Item	Pass 1	Pass 2	Pass 3
Search Parameters			
Major Range (m)	31.6	31.6	31.6
Semi-Major Range (m)	72.0	72.0	72.0
Minor Range (m)	4.8	4.8	4.8
Azimuth (degrees)	72.0	72.0	72.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0

Table 14.8 – Domain 3 Search Parameters

Item	Pass 1	Pass 2	Pass 3
Search Parameters			
Major Range (m)	50.6	50.6	50.6
Semi-Major Range (m)	45.1	45.1	45.1
Minor Range (m)	4.3	4.3	4.3
Azimuth (degrees)	115.0	115.0	115.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0

Table 14.9 – Domain 4 Search Parameters

Item	Pass 1	Pass 2	Pass 3
Search Parameters			
Major Range (m)	39.1	39.1	39.1
Semi-Major Range (m)	45.4	45.4	45.4
Minor Range (m)	4.2	4.2	4.2
Azimuth (degrees)	95.0	95.0	95.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0

14.2.7 Block Model

Table 14.10 below, shows the Laiva block model extents. Figure 14.16, following this page, shows a typical block model section of the mineralized domain.

Table 14.10 – Block Model

Item	X	Y	Z
Origin	2,527,300	7,159,350	-300
Offset Minimum	-	-	-
Offset Maximum	2,205	1,800	500
Parent Block Size (m)	5.00	5.00	5.00
Child Block Size (m)	2.50	2.50	2.50
Bearing Dip/Plunge	90.00	-	-

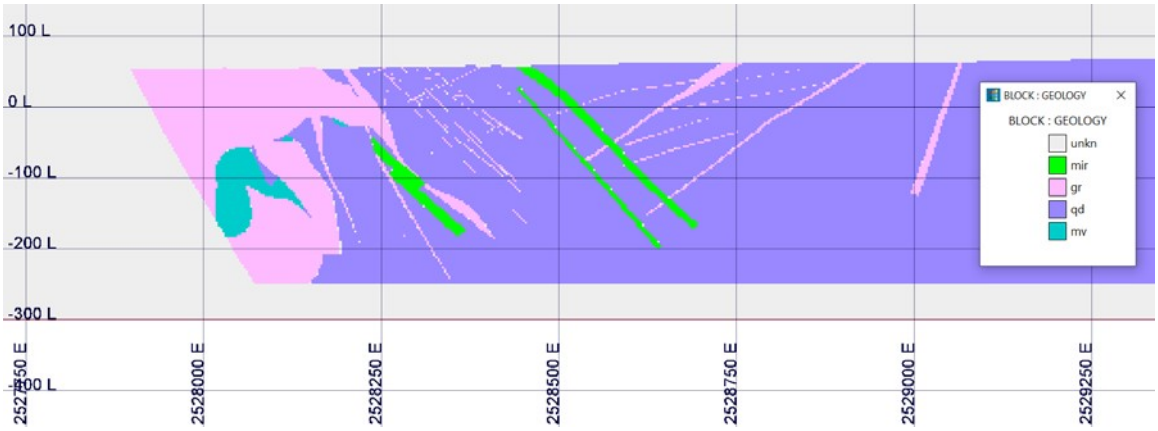


Figure 14.16 – Typical Cross-Section (7,160,400N) through the Laiva Block Model

Four different block models were created for the mineral resource estimate. The purpose of these different block models was to consider the impact of gold grade capping on the total contained metal content in the block models. The four block models included:

- No Cap Model – This block model assumed that no gold grade capping was applied.
- Hard Cap Model – This block model used a fixed hard cap to minimize the impact of high-grade outliers. This model was used as the basis for the mineral resources reported for the Laiva Gold Deposit.
- Threshold Cap Model – This block model used a gold grade cap in each domain above which a limited area of influence was applied.
- Hybrid Cap Model – This block model used both a threshold gold cap and an extreme outlier hard gold cap to limit the impact of higher gold grades.

These four block models were used to examine the impact of gold grade capping on the final mineral resource estimate.

14.2.8 Laiva Mineral Resource Estimation

A 3D block model was constructed in Vulcan that was constrained by the mineralized domains described above. The current topographic surface was used to flag the topographic variable (vtopo). This variable is set to 100 for a block 100% below the surface and to 0% for a block 100% above the surface. A topo-adjusted density (rdensity) was assigned using the following formula:

$$rdensity = density * (vtopo/100)$$

This procedure ensures that blocks along the topographic surface have the correct density applied during pit optimization functions.

No attempt was made to apply a block percentage (percent of the block that is mineralized material and waste). Blocks are in or out of the mineralized domain. Grade interpolation runs were set up for only that material within the mineralized domain for gold. Only domains 1 through 4 were estimated.

Using the composited assays described above, block grade interpolations were run in each mineralized domain for gold. Runs were completed using inverse distance (ID), inverse distance squared (ID^2), inverse distance cubed (ID^3), inverse distance to the fifth (ID^5), ordinary kriging (OK), and nearest neighbor (NN). Three passes were run to allow for use in resource classification. Only composites and blocks flagged as within the same mineralized domain were considered in the grade estimation. Grade estimation parameters are shown in Tables 14.11 below and Tables 14.12 through 14.14, on the following pages.

Table 14.11 – Domain 1 Grade Estimation Parameters

	Pass		
Item	**1**	**2**	**3**
Search Parameters			
Major Range (m)	72.3	72.3	72.3
Semi-Major Range (m)	38.8	38.8	38.8
Minor Range (m)	5.4	5.4	5.4
Azimuth (degrees)	76.0	76.0	76.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0
Search Ellipsoid			
Azimuth (degrees)	76.00	76.00	76.00
Plunge (plunge of the azimuth in degrees)	0.00	0.00	0.00
Dip (degrees)	-80.00	-80.00	-80.00
Major (m)	72.3	72.3	72.3
Semi-Major (m)	38.8	38.8	38.8
Minor (m)	5.4	5.4	5.4
Estimation Parameters			
Minimum Number of Composites	8	4	2
Maximum Number of Composites	10	10	10
Maximum Composites Per Drill Hole	2	2	2
Maximum Distance to Nearest Neighbor (m)	14.5	43.4	---
Resource Classification	Measured	Indicated	Inferred

Table 14.12 – Domain 2 Grade Estimation Parameters

Item	Pass 1	Pass 2	Pass 3
Search Parameters			
Major Range (m)	31.6	31.6	31.6
Semi-Major Range (m)	72.0	72.0	72.0
Minor Range (m)	4.8	4.8	4.8
Azimuth (degrees)	72.0	72.0	72.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0
Search Ellipsoid			
Azimuth (degrees)	72.00	72.00	72.00
Plunge (plunge of the azimuth in degrees)	0.00	0.00	0.00
Dip (degrees)	-80.00	-80.00	-80.00
Major (m)	31.6	31.6	31.6
Semi-Major (m)	72.0	72.0	72.0
Minor (m)	4.8	4.8	4.8
Estimation Parameters			
Minimum Number of Composites	8	4	2
Maximum Number of Composites	10	10	10
Maximum Composites Per Drill Hole	2	2	2
Maximum Distance to Nearest Neighbor (m)	6.3	19.0	---
Resource Classification	Measured	Indicated	Inferred

Table 14.13 – Domain 3 Grade Estimation Parameters

Item	Pass 1	Pass 2	Pass 3
Search Parameters			
Major Range (m)	50.6	50.6	50.6
Semi-Major Range (m)	45.1	45.1	45.1
Minor Range (m)	4.3	4.3	4.3
Azimuth (degrees)	115.0	115.0	115.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0
Search Ellipsoid			
Azimuth (degrees)	115.00	115.00	115.00
Plunge (plunge of the azimuth in degrees)	0.00	0.00	0.00
Dip (degrees)	-80.00	-80.00	-80.00
Major (m)	50.6	50.6	50.6
Semi-Major (m)	45.1	45.1	45.1
Minor (m)	4.3	4.3	4.3
Estimation Parameters			
Minimum Number of Composites	8	4	2
Maximum Number of Composites	10	10	10
Maximum Composites Per Drill Hole	2	2	2
Maximum Distance to Nearest Neighbor (m)	10.1	30.4	---
Resource Classification	Measured	Indicated	Inferred

Table 14.14 – Domain 4 Grade Estimation Parameters

Item	Pass 1	Pass 2	Pass 3
Search Parameters			
Major Range (m)	39.1	39.1	39.1
Semi-Major Range (m)	45.4	45.4	45.4
Minor Range (m)	4.2	4.2	4.2
Azimuth (degrees)	95.0	95.0	95.0
Plunge (plunge of the azimuth in degrees)	0.0	0.0	0.0
Dip (degrees)	-80.0	-80.0	-80.0
Search Ellipsoid			
Azimuth (degrees)	95.00	95.00	95.00
Plunge (plunge of the azimuth in degrees)	0.00	0.00	0.00
Dip (degrees)	-80.00	-80.00	-80.00
Major (m)	39.1	39.1	39.1
Semi-Major (m)	45.4	45.4	45.4
Minor (m)	4.2	4.2	4.2
Estimation Parameters			
Minimum Number of Composites	8	4	2
Maximum Number of Composites	10	10	10
Maximum Composites Per Drill Hole	2	2	2
Maximum Distance to Nearest Neighbor (m)	7.8	23.5	---
Resource Classification	Measured	Indicated	Inferred

14.2.9 Laiva Resource Classification

The resource classification used for the Laiva deposit is based on which pass generated a block grade estimate as well as the distance to the nearest neighbor (measured and indicated only). The resource classification used was:

- Measured – Blocks estimated in Pass 1 (minimum of four composites) with a maximum nearest neighbor distance as listed in Tables 14.11 through 14.14 for each domain are classified as measured.
- Indicated – Blocks estimated in Pass 2 (minimum of three composites) with a maximum nearest neighbor distance as listed in Tables 14.11 through 14.14 for each domain as indicated.
- Inferred – Blocks estimated in Pass 3 (minimum of two composites) are classified as inferred.

14.2.10 Laiva Mineral Resource Estimate Validation

The gold grade populated block model was reviewed to ensure reasonableness. These checks included:

- An overall review of the estimated metal values.
- The impact of gold grade capping on the mineral resource.
- QQ plots of the block model versus the composites.

- A section-by-section comparison between the ID^3 metal values and the underlying drillholes.
- A statistical comparison of the raw assay values versus the composite values versus the block values.

The overall block metal grades were visually examined to confirm that all the estimation parameters were honored and kept within the individual mineralized domains. Each of the cross-sections was reviewed and the underlying drillholes were checked to determine that the original metal grade closely matched the estimated block metal grade without exceeding it. Cross-sections were examined, and assay intervals agreed with the overlying estimated block model metal grades. An example cross-section is shown in Figure 14.17 below. An example plan view section is shown in Figure 14.18, following this page.

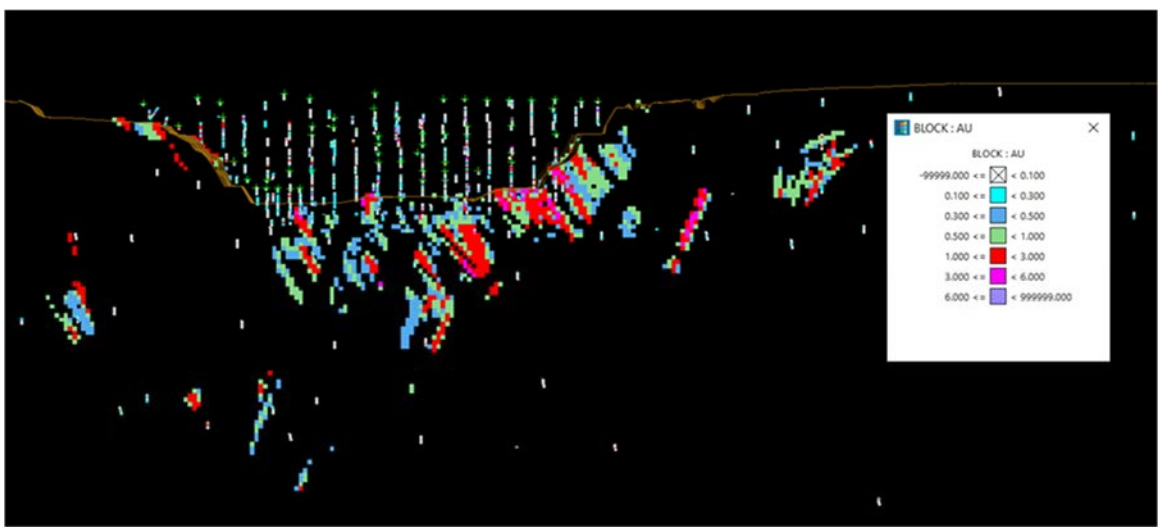


Figure 14.17 – Typical Cross-Section (7,160,400N) for Visual Validation

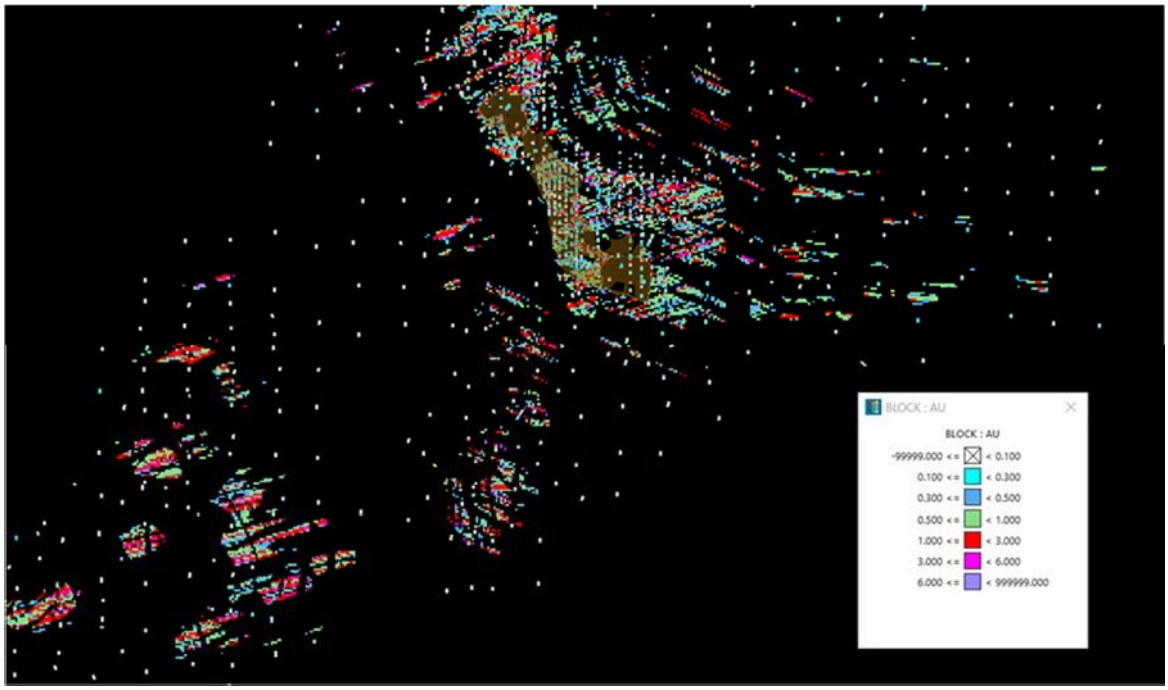


Figure 14.18 – Typical Plan View Section (elevation of -5 below sea level) for Visual Validation

A statistical comparison of the raw assay values versus the composite values versus the estimated block values was run and is shown in Table 14.15 below.

Table 14.15 – Laiva Mineral Resource Statistics (all domains)

	Domains				
Item	**All**	**1**	**2**	**3**	**4**
1-Meter Composite Data					
Number of samples:	46,729	5,293	17,684	21,309	2,443
Minimum:	0.010	0.010	0.010	0.010	0.010
Maximum:	142.155	60.197	43.500	67.054	142.155
Range:	142.145	60.187	43.490	67.044	142.145
Average:	0.957	1.102	1.011	0.896	0.788
Standard deviation:	2.063	2.401	1.796	1.906	3.706
Variance:	4.256	5.765	3.226	3.633	13.734
Coef. of variance:	2.156	2.179	1.776	2.127	4.703
Hard Cap Block Model Results					
Number of blocks:	1,049,525	254,670	227,628	391,104	176,123
Minimum:	0.010	0.010	0.010	0.010	0.010
Maximum:	24.607	24.338	23.925	24.607	9.921
Range:	24.597	24.328	23.915	24.597	9.911
Average:	0.734	0.904	0.729	0.644	0.694
Standard deviation:	1.078	1.393	1.054	0.939	0.814
Variance:	1.162	1.940	1.111	0.882	0.663
Coef. of variance:	1.469	1.541	1.446	1.458	1.173

The various mineralized domain QQ plots of the block model estimated ID^3 gold grades versus the composites are shown in Figures 14.19 and 14.20 below, and Figures 14.21 through 14.22, following this page.

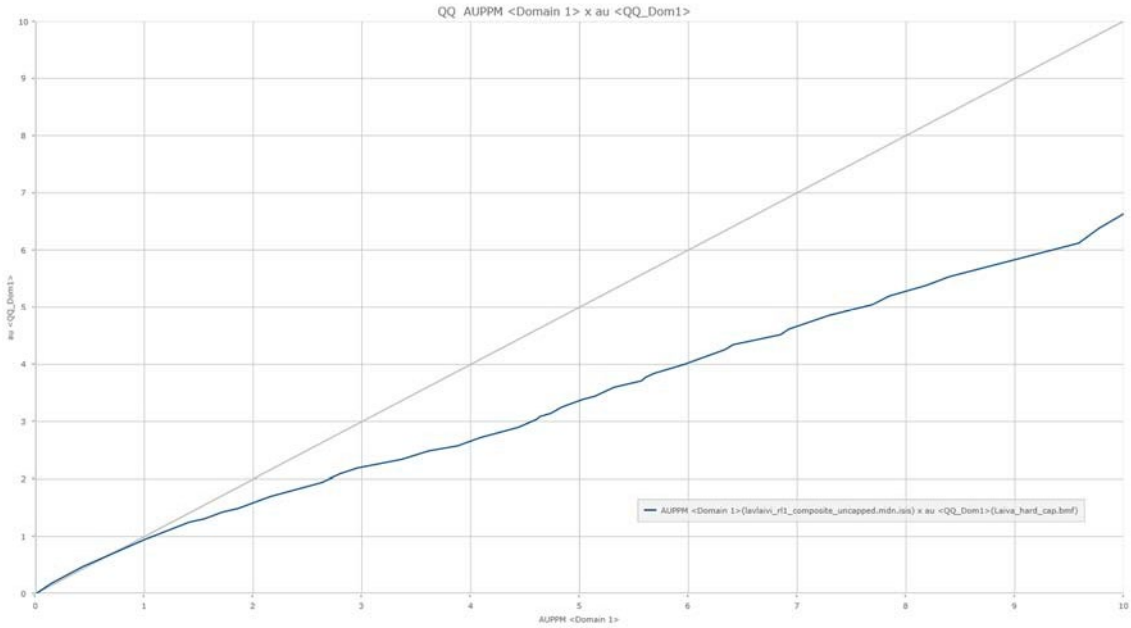


Figure 14.19 – Laiva QQ Plot Domain 1

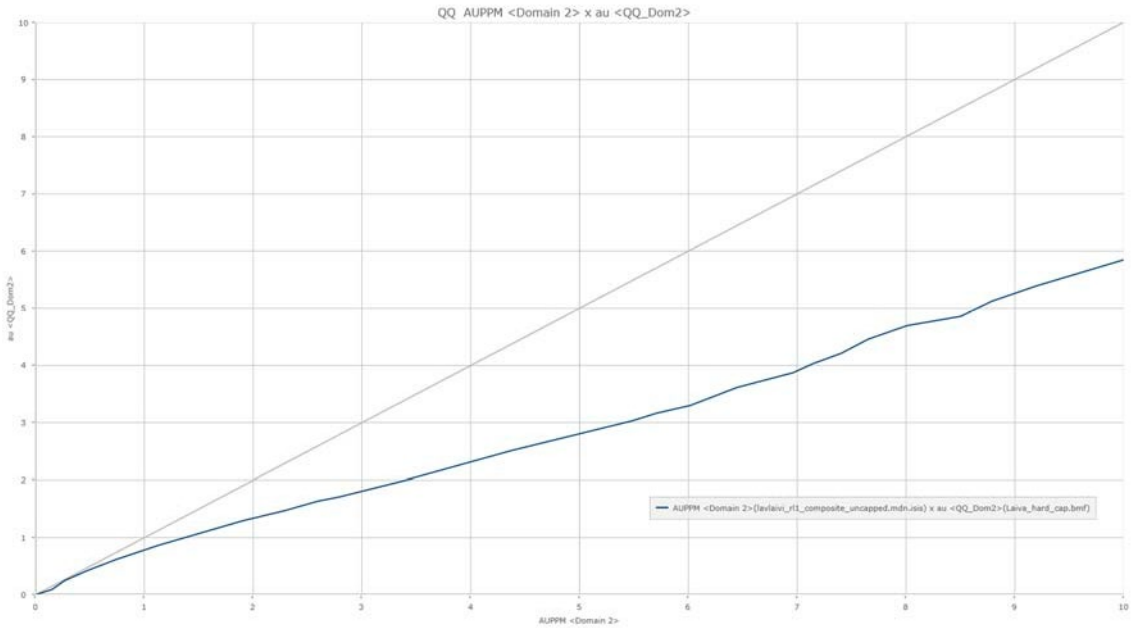


Figure 14.20 – Laiva QQ Plot Domain 2

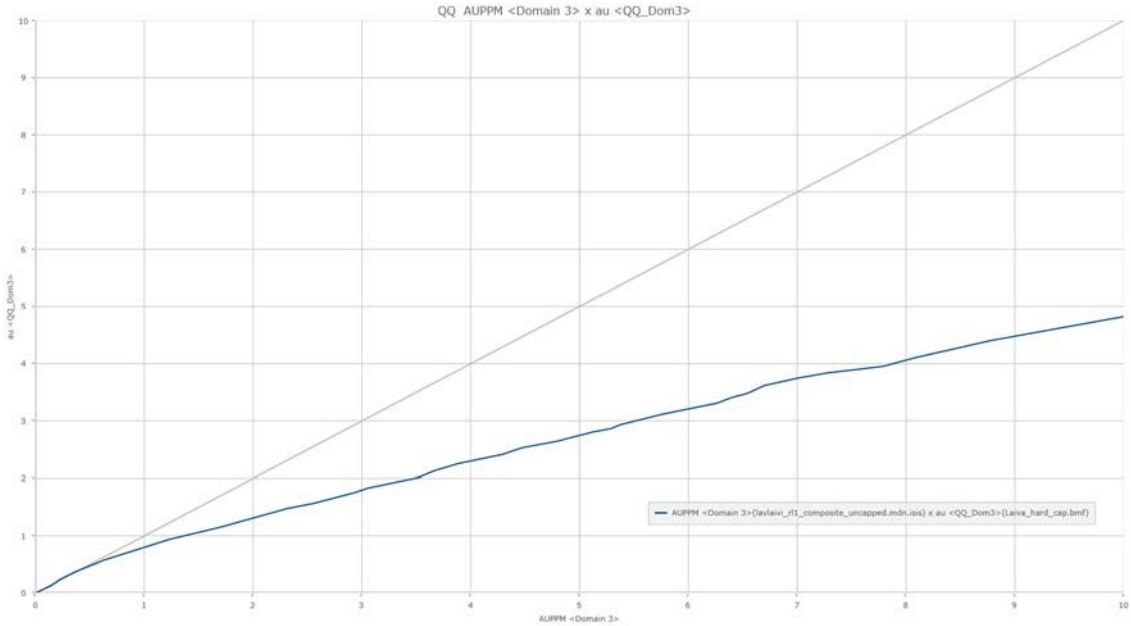


Figure 14.21 – Laiva QQ Plot Domain 3

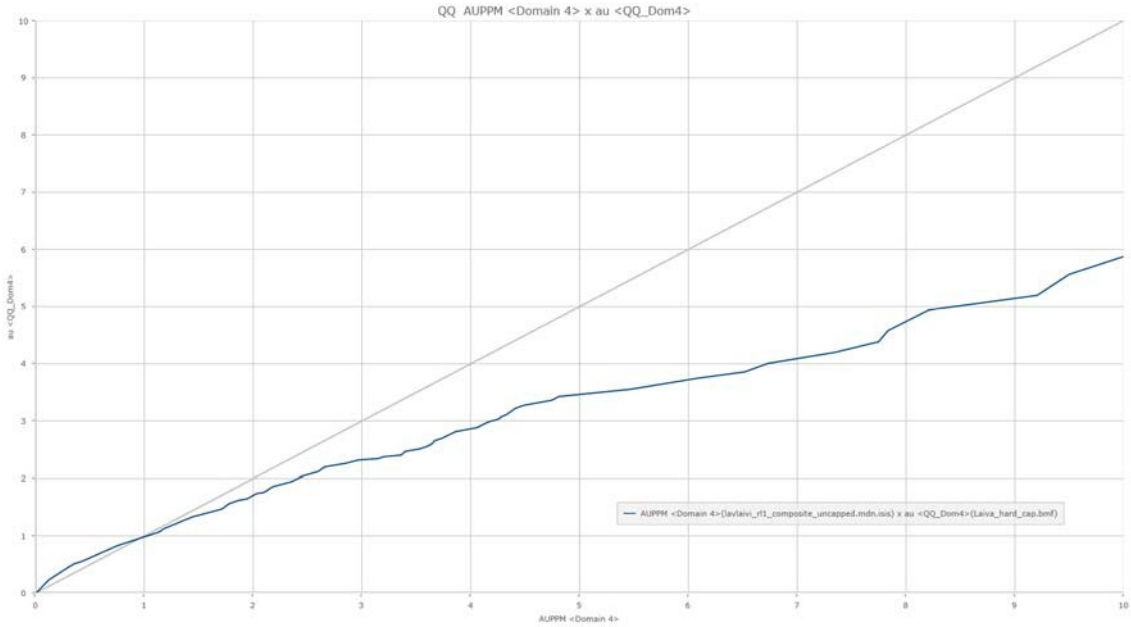


Figure 14.22 – Laiva QQ Plot Domain 4

The impact on total metal content of gold grade capping is shown in Figure 14.23, following this page. The impact of gold grade capping at Laiva showed that the hard-capped block model (used for mineral resources) contained 97.4% of the no capping

block model contained gold ounces. The threshold capped block model contained 97.7% of the no capping block model contained ounces. The hybrid capped model contained 96.6% of the no capping block model contained ounces. It is the opinion of BOYD that the hard capped model represents the best estimate of the in-situ mineral resource at Laiva and was selected for mineral resource reporting.

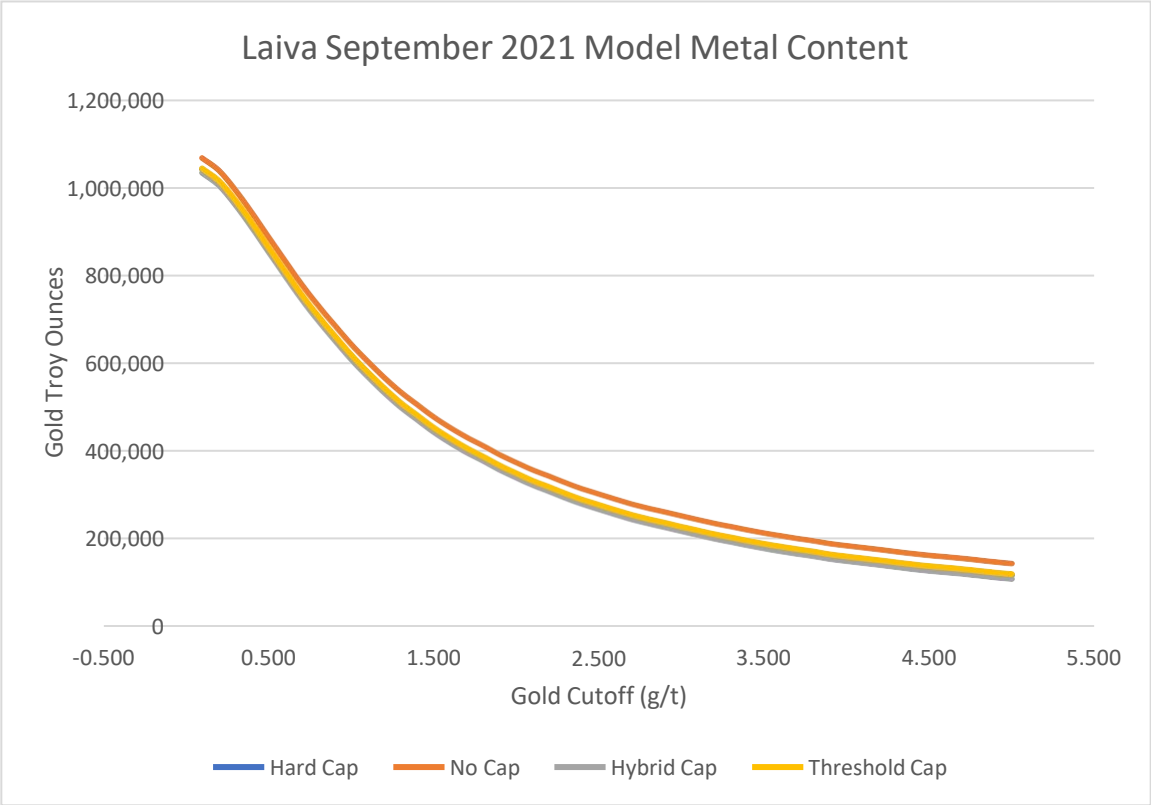


Figure 14.23 – Laiva Impact of Gold Grade Capping

14.3 Laiva Mineral Resource Estimate

The Laiva mineral resources may be amenable to a combination of open pit and underground mining methods. BOYD developed a conceptual pit shell (the economic open pit shell) using the Lerchs-Grossman method as provided by the Whittle software within which the portions of the block model that show “reasonable prospects for economic extraction” by open pit mining. From this shell, a conceptual open pit mine was designed and used to constrain the mineral resources. Portions of the block model that are external to the conceptual pit shell but satisfy cut-off grade criteria for an appropriate underground extraction method, are considered to show “reasonable prospects for economic extraction” by underground mining methods.

14.3.1 Economic Assumptions

The operating assumptions (economic and gold recovery) used for the Whittle economic open pit optimization as well as underground cut-off grade are shown in Table 14.16, following this page. These assumptions are based on the inputs for the feasibility study metallurgical and economic parameters currently being developed.

For mineral resource estimation, a cut-off grade of 0.323 g/t gold was used for open pit, and a cut-off grade of 1.886 g/t gold was used for underground. The assumed overall pit slope in Whittle was assumed to be 53.0° as per historical operating practice. None of the pit slopes includes an allowance for ramps.

Using these assumptions, a Whittle economic pit optimization was completed, and an economic open pit shell generated. This pit shell was used to design a conceptual open pit, which is shown in Figure 14.24 below.

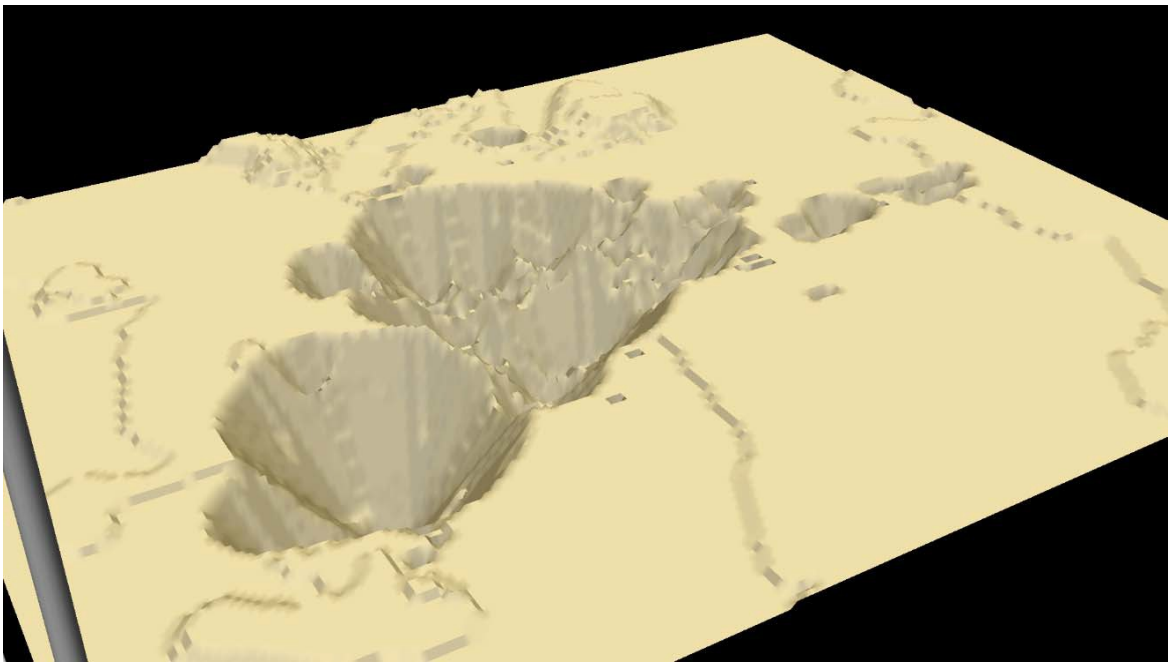


Figure 14.24 – Laiva Whittle Mineral Resource Open Pit Shell

The operating assumptions (economic and gold recovery) used for the Whittle economic pit optimization are shown in Table 14.16 below.

Table 14.16 – Laiva Economic Assumptions

Item	Units	Laiva
Open Pit Waste Mining Cost	US$/Waste Tonne	$1.90
Open Pit Ore Mining Cost	US$/Ore Tonne	$2.73
Underground Mining Cost	US$/Ore Tonne	$75.00
Mill - High Grade Processing Cost	US$/Mill Ore Tonne	$11.78
Mill - Low Grade Processing Cost	US$/Mill Ore Tonne	$10.22
G&A Cost	US$/Feed Tonne	$2.20
Mill Gold Recovery (at cutoff)	%	92.0%
Mill - Low Grade Gold Recovery (at cutoff)	%	80.0%
Mill Annual Capacity (Both High & Low Grade)	Tonnes	1,500,000
Gold Price	US$/Troy Ounce	$1,600.00
Selling Cost	US$/Troy Ounce	$2.55
Land Tenure Royalty	%	0.55%
Open Pit Calculated Mill - High Grade Cutoff	g/t	0.354
Open Pit Incremental Mill - High Grade Cutoff	g/t	0.314
Open Pit Calculated Mill - Low Grade Cutoff	g/t	0.369
Open Pit Incremental Mill - Low Grade Cutoff	g/t	0.323
Underground Calculated Cutoff	g/t	1.886

14.3.2 Pit Constrained Mineral Resource Estimate

BOYD's mineral resource estimate for the Laiva deposit is provided in Table 14.17, following this page.

Table 14.17 – Laiva Mineral Resource Estimate

Mining Method	Resource Classification	Gold Cutoff Grade (g/t)	Tonnes	Au g/t	Au Troy Ozs
Open Pit - High Grade	Measured	0.700	654,000	1.930	40,600
Open Pit - High Grade	Indicated	0.700	6,573,000	1.655	349,700
Open Pit - High Grade	***Measured + Indicated***	***0.700***	***7,227,000***	***1.680***	***390,300***
Open Pit - High Grade	Inferred	0.700	4,670,000	1.746	262,200
Open Pit - Low Grade	Measured	0.323	531,000	0.487	8,300
Open Pit - Low Grade	Indicated	0.323	6,247,000	0.488	98,000
Open Pit - Low Grade	***Measured + Indicated***	***0.323***	***6,778,000***	***0.488***	***106,300***
Open Pit - Low Grade	Inferred	0.323	4,012,000	0.488	62,900
Total Open Pit	Measured	0.323	1,185,000	1.283	48,900
Total Open Pit	Indicated	0.323	12,820,000	1.086	447,700
Total Open Pit	***Measured + Indicated***	***0.323***	***14,005,000***	***1.103***	***496,600***
Total Open Pit	Inferred	0.323	8,682,000	1.165	325,100
Underground	Measured	1.886	0	0.000	0
Underground	Indicated	1.886	26,000	3.072	2,600
Underground	***Measured + Indicated***	***1.886***	***26,000***	***3.072***	***2,600***
Underground	Inferred	1.886	303,000	3.067	29,900
Open Pit + Underground	Measured	0.323	1,185,000	1.283	48,900
Open Pit + Underground	Indicated	0.323	12,846,000	1.090	450,300
Open Pit + Underground	***Measured + Indicated***	***0.323***	***14,031,000***	***1.107***	***499,200***
Open Pit + Underground	Inferred	0.323	8,985,000	1.229	355,000

(1) The effective date for this mineral resource estimate is 4 October 2021 and is reported on a 100% ownership basis. This estimate is an update using additional assays and exploration drilling as well as updated economics.

(2) Mineral Resources are calculated at a gold price of US$1,600 per troy ounce.

(3) The mineral resources presented above are global and do not include a detailed pit or underground design, only an economic pit-shell was used to determine the in-pit mineral resources. The underground mineral resources are that material outside of the in-pit mineral resources above the stated underground cutoff grade.

(4) Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.

(5) The mineral resources presented here were estimated using a block model with a block size of 5 m by 5 m by 5 m sub-blocked to a minimum block size of 2.5 m by 2.5 m by 2.5 m using ID^3 methods for grade estimation. All mineral resources are reported using an open pit gold cut-off of 0.323 g/t Au and an underground gold cut-off of 1.886 g/t Au. Open pit mineral resources above 0.700 g/t are considered high grade material while material between 0.323 g/t and 0.700 g/t are considered low grade.

(6) The mineral resources presented here were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10, 2014.

(7) Figures are rounded, and totals may not add correctly.

14.3.3 Changes from the Previous Mineral Resource Estimate

At Laiva, changes from recent previous mineral resource estimate include additional drilling in 2019 and 2021 as well as the addition of drillholes that were previously not included in the mineral resource estimate. This has significantly changed the mineral resource estimate from the previous 2019 estimate. These changes are shown in Figures 14-25 and 14-26 below.

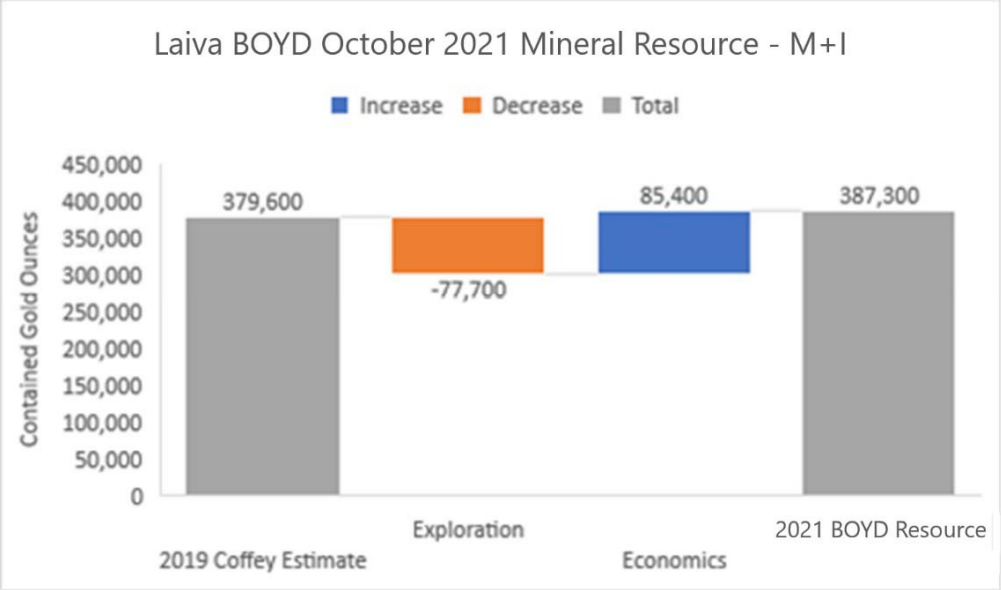


Figure 14.25 – Changes to Measured and Indicated since the 2019 Mineral Resource Estimate

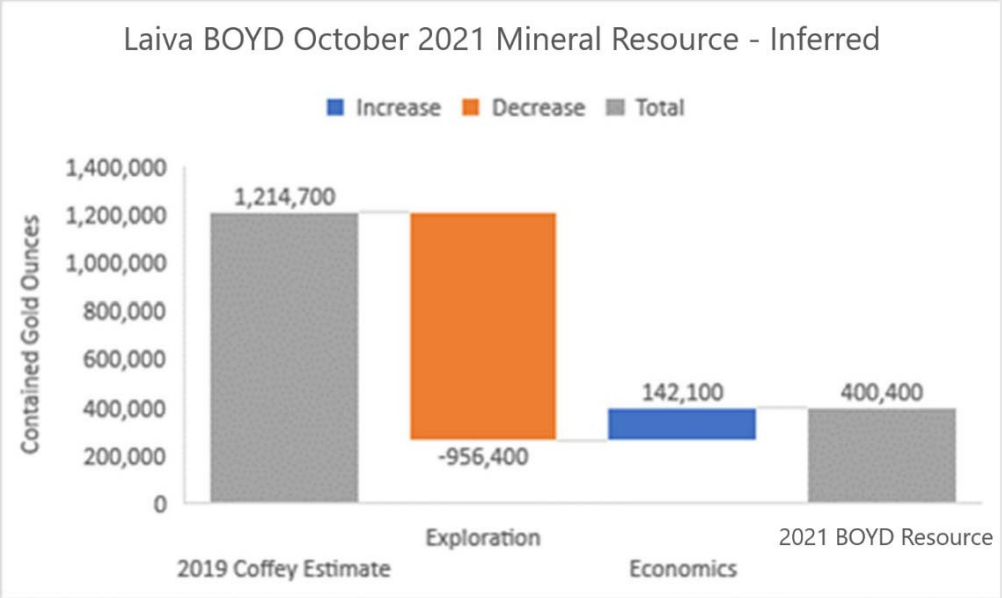


Figure 14.26 – Changes to Inferred since the 2019 Mineral Resource Estimate

15.0 MINERAL RESERVE ESTIMATES

15.1 Mineral Reserve Estimate

Mineral reserves for the Laiva gold deposit have been estimated by BOYD as summarized in Table 15.1 below.

Mineral reserves have been estimated for the North and South pits from the undiluted block model and then diluted as described in Section 16.0 of this report. Mineral reserves are the result of work described in Section 16.0 of this report.

Table 15.1 – Laiva Diluted Mineral Reserves

Mining Method	Reserve Classification	Cutoff	Tonnes	Au g/t	Au Troy Ozs
Open Pit - High Grade	Proven	0.700	678,000	1.853	40,400
Open Pit - High Grade	Probable	0.700	4,780,000	1.622	249,300
Open Pit - High Grade	**Proven + Probable**	**0.700**	**5,458,000**	**1.651**	**289,700**
Open Pit - Low Grade	Proven	0.323	547,000	0.466	8,200
Open Pit - Low Grade	Probable	0.323	4,488,000	0.468	67,500
Open Pit - Low Grade	**Proven + Probable**	**0.323**	**5,035,000**	**0.468**	**75,700**
Total Open Pit	Proven	0.323	1,225,000	1.234	48,600
Total Open Pit	Probable	0.323	9,268,000	1.063	316,800
Total Open Pit	**Proven + Probable**	**0.323**	**10,493,000**	**1.083**	**365,400**

(1) The mineral reserve estimates were prepared by Sam Shoemaker, Reg'd Member SME (who is also an independent Qualified Person), reported using the 2014 CIM Definition Standards, and have an effective date of September 29, 2021.

(2) Mineral Reserves are mined tonnes and grade delivered to a stockpile in front of the primary crusher.

(3) Mineral reserves are reported at a cut-off grade of 0.323 g/t Au.

(4) Cut-off grade assumes US$1,600/oz Au; US$2.55/oz selling costs; and uses a 82% metallurgical recovery for high grade material (0.700 g/t Au and higher) and a 80% metallurgical recovery for low grade material (0.23 to 0.700 g/t Au). The cut off-grade covers processing costs of US$11.78/t for high grade and US10.22/t for low grade, administrative (G&A) costs of $2.20/t. Ore material has been diluted by 5% with a 10% of the undiluted gold grade.

(5) Mined tonnes and grade are based on an SMU of 5.0 m x 5.0 m x 5.0 m, including additional mining losses estimated for the removal of isolated blocks (surrounded by waste).

(6) Figures are rounded, and totals may not add correctly.

15.2 CIM Standards Definitions Applied

CIM Standards on Mineral Resources and Reserves Definitions and Guidelines (1) defines A ‘Proven Mineral Reserve as “the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.”

CIM Standards on Mineral Resources and Reserves Definitions and Guidelines (1) define ‘Probable Mineral Reserve’ as “the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.”

The effective date of the mineral reserve estimate is September 29, 2021.

The ultimate pit limit is based on the economic Lerchs-Grossmann algorithm design on all sides, with no external factors constraining the pit. The final pit plans are presented in Section 16.0.

The mine plan developed in this report is based on Measured and Indicated resources only. There is an opportunity to upgrade some minor amounts of the inferred resource mineralization to ore classification with additional infill drilling.

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16.0 MINING METHODS

16.1 Introduction

The Laiva Gold Deposit is composed of a single project with two mining areas: the previously developed South Pit area in the southwestern portion of the project; and the previously developed North Pit area in the northern portion of the project. These two mining areas develop the same mineral resource and are shown in Figure 16.1 below.

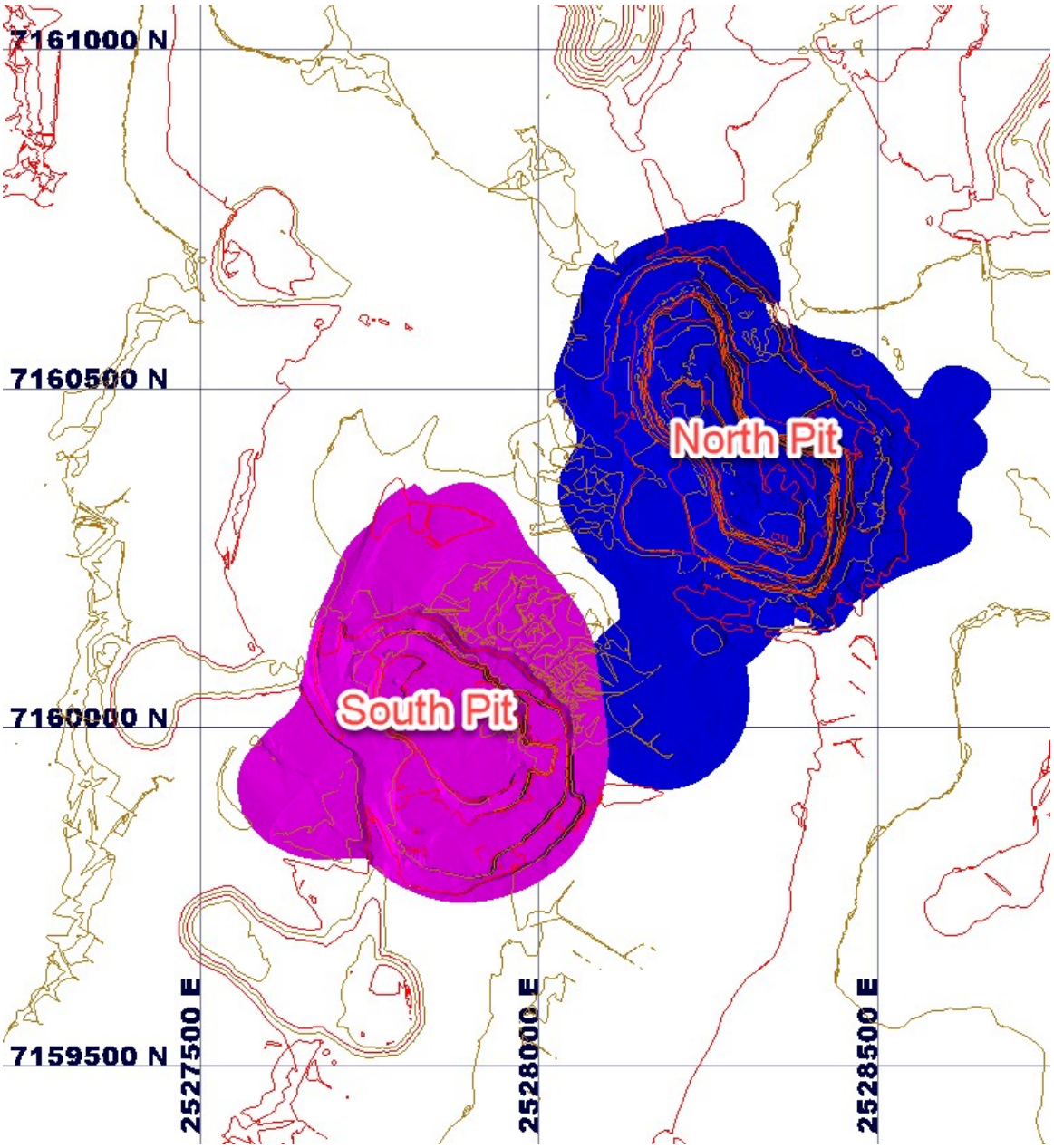


Figure 16.1 – Laiva Project Mining Areas

Pit shells were determined for the Laiva deposit from a Whittle economic pit optimization described below. Based on these initial pit optimization results, a pit shell was selected to design the phased pits that make up the two mining areas for the feasibility study.

Standard mining technology has been utilized to create an open pit at each of the two mining areas. The South Pit has approximate dimensions of 620 m south to north by 540 m east to west and a maximum depth of 200 m below current ground level. The South is composed of three mining phases. The North Pit has approximate dimensions of 840 m south to north by 600 m east to west and a maximum depth of 170 m below current ground level. The North Pit is composed of four mining phases. All of the pit shells were selected from the Whittle open-pit optimization discussed below and were used to guide the design of two mining areas.

A mine contractor has been engaged by OTSO who will be used for most mining activities including site preparation, haul road construction and maintenance, mineralized material and bulk waste drilling and blasting, excavation and haulage of mineralized material and waste, management of waste dumps, oversize breakage, and pit dewatering. The mine contractor will provide all mining equipment. The contractor will provide operator training, supervision, mine consumables, and maintenance facilities for contractor's operations. Mine maintenance personnel will be supplied by the contractor for the contractor's fleet. OTSO will provide mine technical services including blast design, blasthole layout, ore grade control, mine planning, and blasthole sampling. Orica will provide explosives storage on-site. Explosives, blasting agents, fuel and other consumables will be provided by established suppliers.

16.2 Mine Design

The ultimate pit limits selected for the two mining areas for the Laiva Gold Deposit were selected based on Whittle open-pit economic optimizations. Together the mining areas will be developed using seven distinct phases designed to approximate an optimal extraction sequence. The phased pit designs are based on slope design parameters and benching configurations provided by BOYD as well as historic operating practice. A mine production schedule was prepared by BOYD using Maptek's Chronos scheduling software.

16.2.1 Mining Dilution

The Laiva resource model uses a fixed 5 m X 5 m X 5 m parent block size (sub-blocked to 2.5 m X 2.5 m X 2.5 m) which is considered the smallest selective mining unit (SMU) for the deposit and is suitable for mining equipment fleet planned. Based on this selected

SMU size, no additional waste dilution other than internal included waste was deemed appropriate for the Resource update, as ore zones are gradational defined by cut-off grade boundaries.

For the mineral reserve reporting, a fixed tonnage dilution of 5 percent was selected as the tonnage dilution factor. The grade of the diluting tonnes is 10 percent of the original block grade. Diluting material is drawn from nearby waste material which decreases in grade with increasing distance from the mineralized zone.

16.2.2 Whittle Pit Optimization

In order to design the various required phased designs for the Laiva Gold Deposit, BOYD completed new pit optimizations to determine the optimal economic pit configuration for the overall project. To accomplish this task, BOYD used the Whittle open-pit optimization software. This software uses the industry standard Lerchs-Grossmann algorithm to determine an optimal pit shape using various economic, geotechnical, and metallurgical parameters. A Whittle optimization was completed on the resource model using only measured and indicated mineral resources (inferred material is considered waste) and a final conceptual pit shell was determined. This pit shell was then used for feasibility study level detailed phased pit designs and production scheduling.

Mine planning geotechnical parameters were determined by BOYD in conjunction with historical operating practice and are shown in Table 16.1 below.

Table 16.1 – Laiva Geotechnical Parameters

Item	Value
Inter-ramp Slope (degrees)	53.0
Face Angle (degrees)	80.0
Safety Bench (m)	5.78
Benching	Double
Bench Height (m)	5.0

Initial feasibility study level operating costs were calculated and used to provide the economic basis for the various Whittle pit optimization runs that were completed. All runs only included measured and indicated mineral resource material. Inferred mineral resource material is considered to be waste and was treated as such in all aspects of the feasibility study. The feasibility study economics used for the Whittle optimization are shown in Table 16.2, following this page.

Table 16.2 – Laiva Feasibility Study Whittle Economics

Item	Units	Laiva
Open Pit Waste Mining Cost	US$/Waste Tonne	$1.90
Open Pit Ore Mining Cost	US$/Ore Tonne	$2.73
Underground Mining Cost	US$/Ore Tonne	$75.00
Mill - High Grade Processing Cost	US$/Mill Ore Tonne	$11.78
Mill - Low Grade Processing Cost	US$/Mill Ore Tonne	$10.22
G&A Cost	US$/Feed Tonne	$2.20
Mill Gold Recovery (at cutoff)	%	92.0%
Mill - Low Grade Gold Recovery (at cutoff)	%	80.0%
Mill Annual Capacity (Both High & Low Grade)	Tonnes	1,500,000
Gold Price	US$/Troy Ounce	$1,600.00
Selling Cost	US$/Troy Ounce	$2.55
Royalty	%	0.55%
Open Pit Calculated Mill - High Grade Cutoff	g/t	0.354
Open Pit Incremental Mill - High Grade Cutoff	g/t	0.314
Open Pit Calculated Mill - Low Grade Cutoff	g/t	0.369
Open Pit Incremental Mill - Low Grade Cutoff	g/t	0.323

Whittle optimizations were completed on the Laiva mineral resource model using the economic parameters in Table 16.2 above, as well as the geotechnical parameters presented in Table 16.1, Page 16-3. The initial Whittle runs used only the overall inter-ramp slope. Results from these runs were examined and the overall slope was flattened to reflect the inclusion of ramps to the design. The addition of ramps to the Whittle results decreased the overall pit slope from 53 degrees to 44 degrees.

The Whittle runs using the ramp reduced slopes as well as the economics presented in Table 16.2 above were used to generate the final pit shells that were then used for the phased and ultimate pit designs on each deposit. Figure 16.2, following this page, shows the resulting Whittle pit shells used for design.

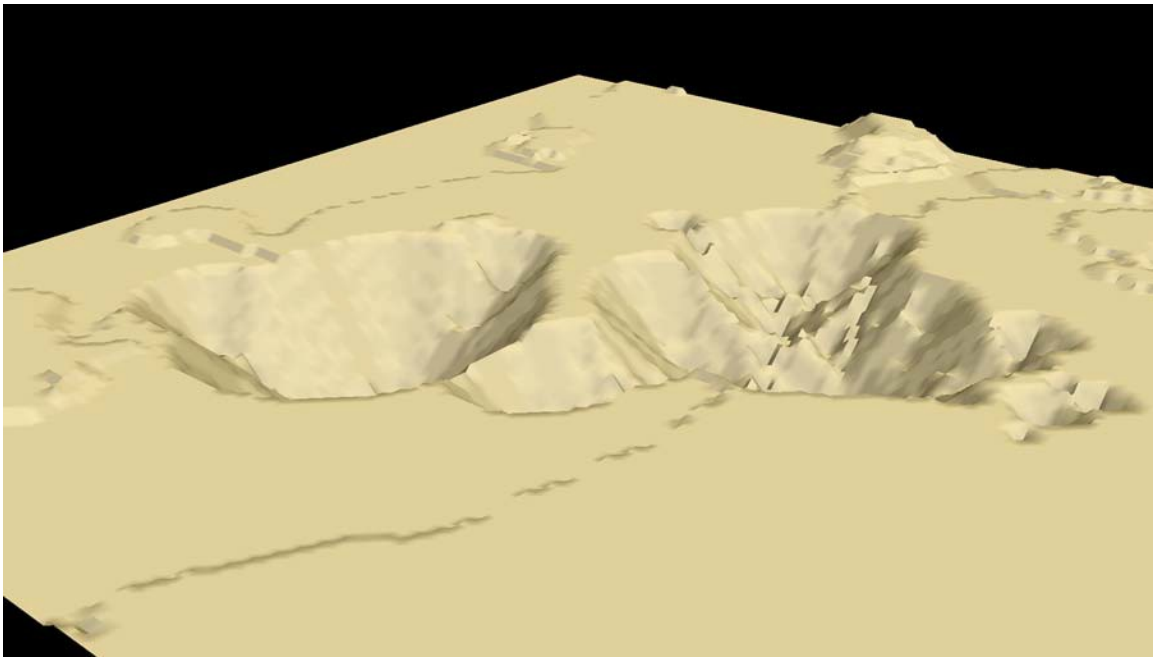


Figure 16.2 – Laiva PEA Whittle Pit Shell

A total of 71 pit shells were examined from the Laiva Whittle results. These pit shells ranged from a 0.3 revenue factor (gold price of US$480/troy ounce) up to a revenue factor of 1.5 (gold price of US$2,400/troy ounce). The revenue factor 1 pit shell (gold price of US$1,600/troy ounce) was selected for the detailed feasibility study phased designs. The Whittle pit by pit graph of the results is shown below in Figure 16.3, following this page.

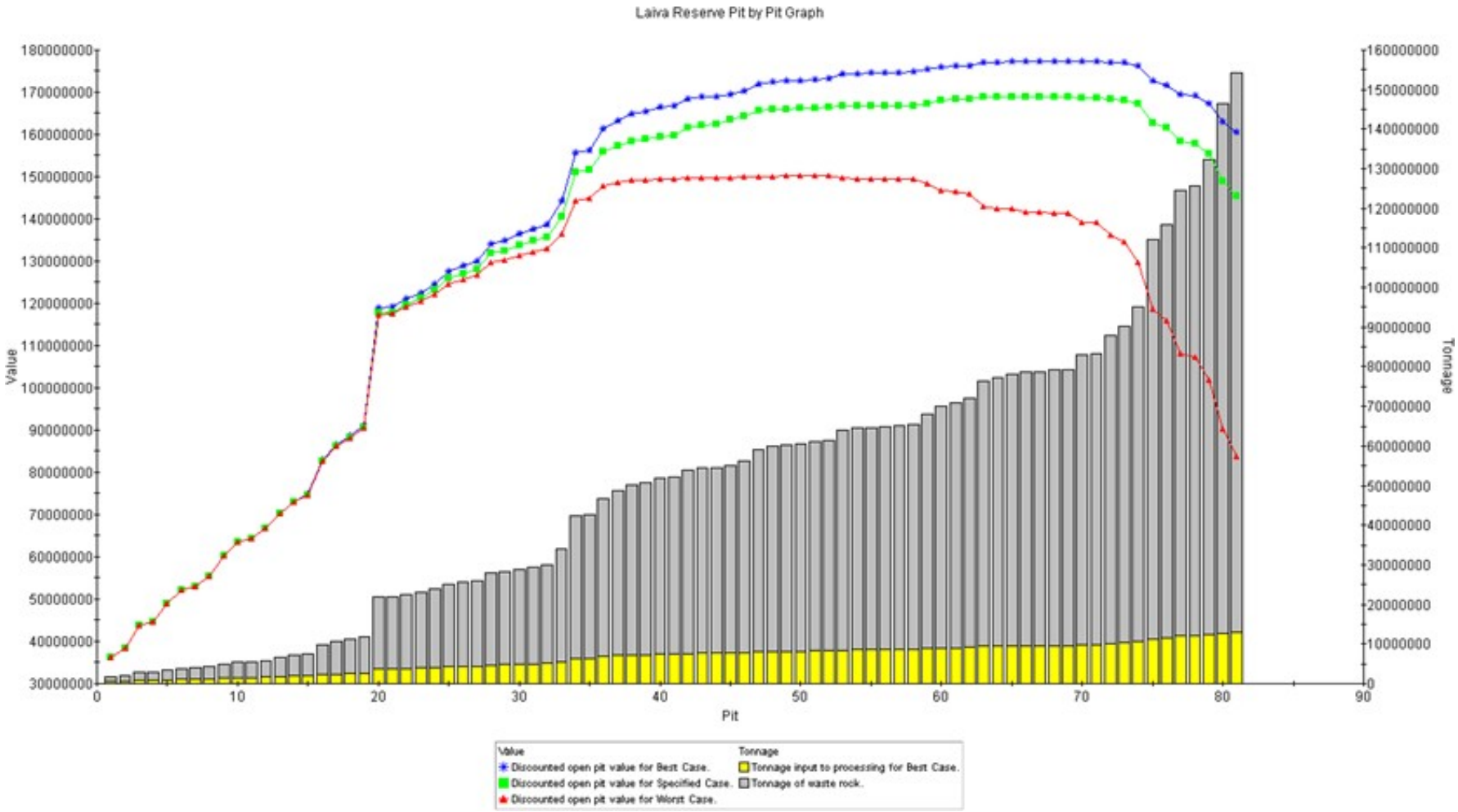


Figure 16.3 – Laiva Whittle Pit by Pit Results

16.2.3 Design Criteria

Different mining phases were designed in accordance with the recommended bench configurations provided as tabulated above in Table 16.1, Page 16-3. Double benching of 5 m production benches was determined to be feasible in all geologic units. Catch or safety benches with a width of 5.78 m are used in all designed phases. These safety benches are applied on every other bench (10 m vertically). Ore mining is planned on 5 m benches while waste mining is planned on 10 m double height benches.

Two-way haul roads, 26 m wide at a 10% grade, were designed in most cases where higher traffic may require extra width for safe and efficient passing of trucks. To maximize ore recovery at depth, the final benches of each pit floor were designed with single-lane access (13 m width with a 12% grade). Safety berms were designed in accordance with the recommendations provided by BOYD.

16.2.4 Phased Designs

16.2.4.1 South Pit Phased Designs

The Laiva South Pit consists of three phases. Phase 1 is the starter pit while phases 2, and 3 expand the starter pit. Table 16.3 below describes the in-pit diluted mineral reserves for South Pit while Figures 16.4 through 16.7, following this page, show the individual phase layouts.

Table 16.3 – Laiva South Pit Diluted Mineral Resources

Class	Material	Cutoff	Au (g/t)	Tonnes	Au (t.ozs)
Measured	High-Grade	0.700	2.085	371,000	24,900
Indicated	High-Grade	0.700	1.754	2,172,000	122,500
Total M+I	***High-Grade***	***0.700***	***1.802***	***2,543,000***	***147,400***
Measured	Low-Grade	0.323	0.474	229,000	3,500
Indicated	Low-Grade	0.323	0.474	1,644,000	25,100
Total M+I	***Low-Grade***	***0.323***	***0.474***	***1,873,000***	***28,600***
Measured	All	0.323	1.470	600,000	28,400
Indicated	All	0.323	1.203	3,816,000	147,600
Total M+I	***All***	***0.323***	***1.239***	***4,416,000***	***176,000***
Waste	All	---	---	40,944,000	---
Total Material Tonnes		---	---	45,360,000	---
Stripping Ratio		---	---	9.27	---

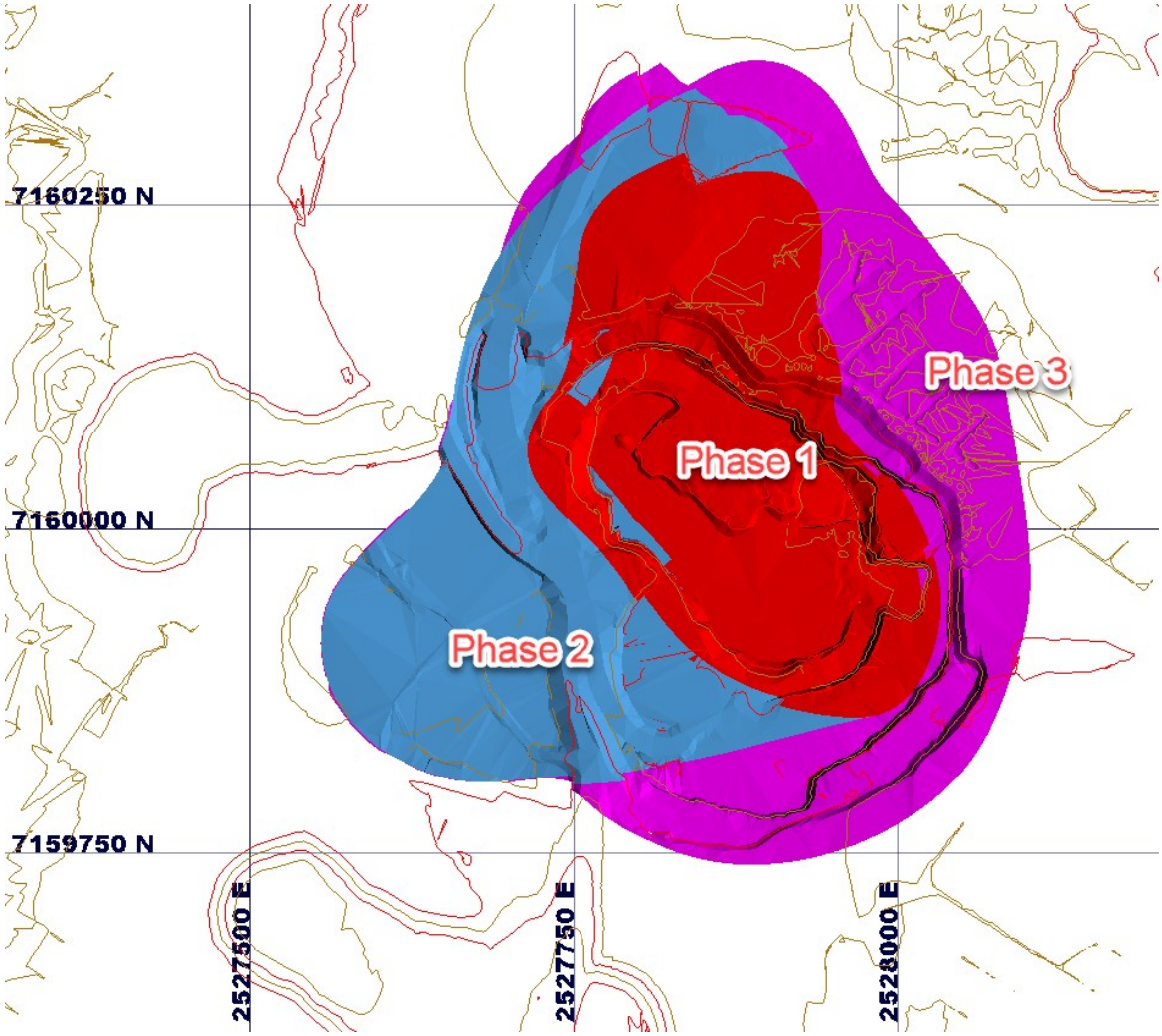


Figure 16.4 – Laiva South Pit Phased Design

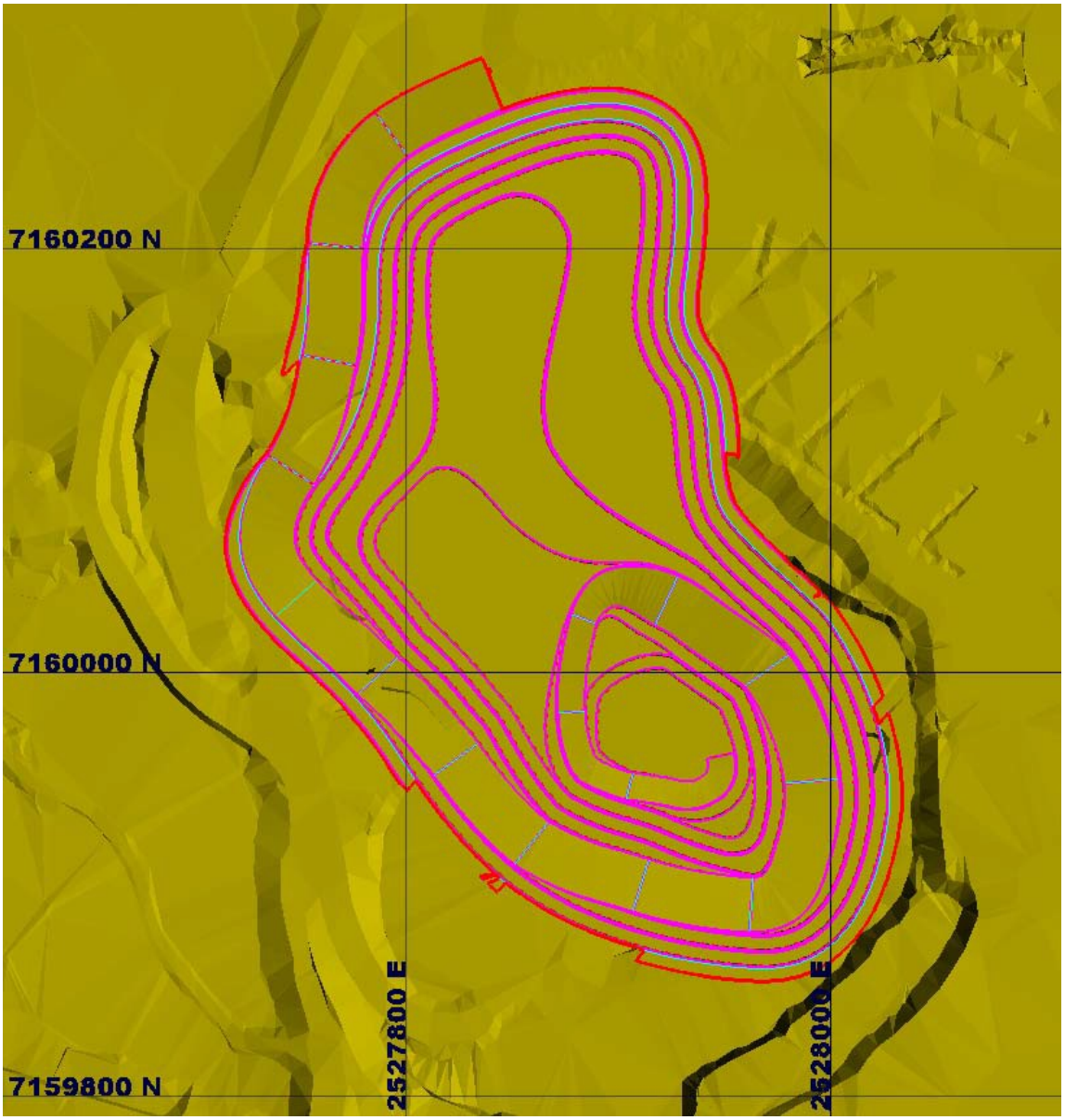


Figure 16.5 – Laiva South Pit Phase 1 Design

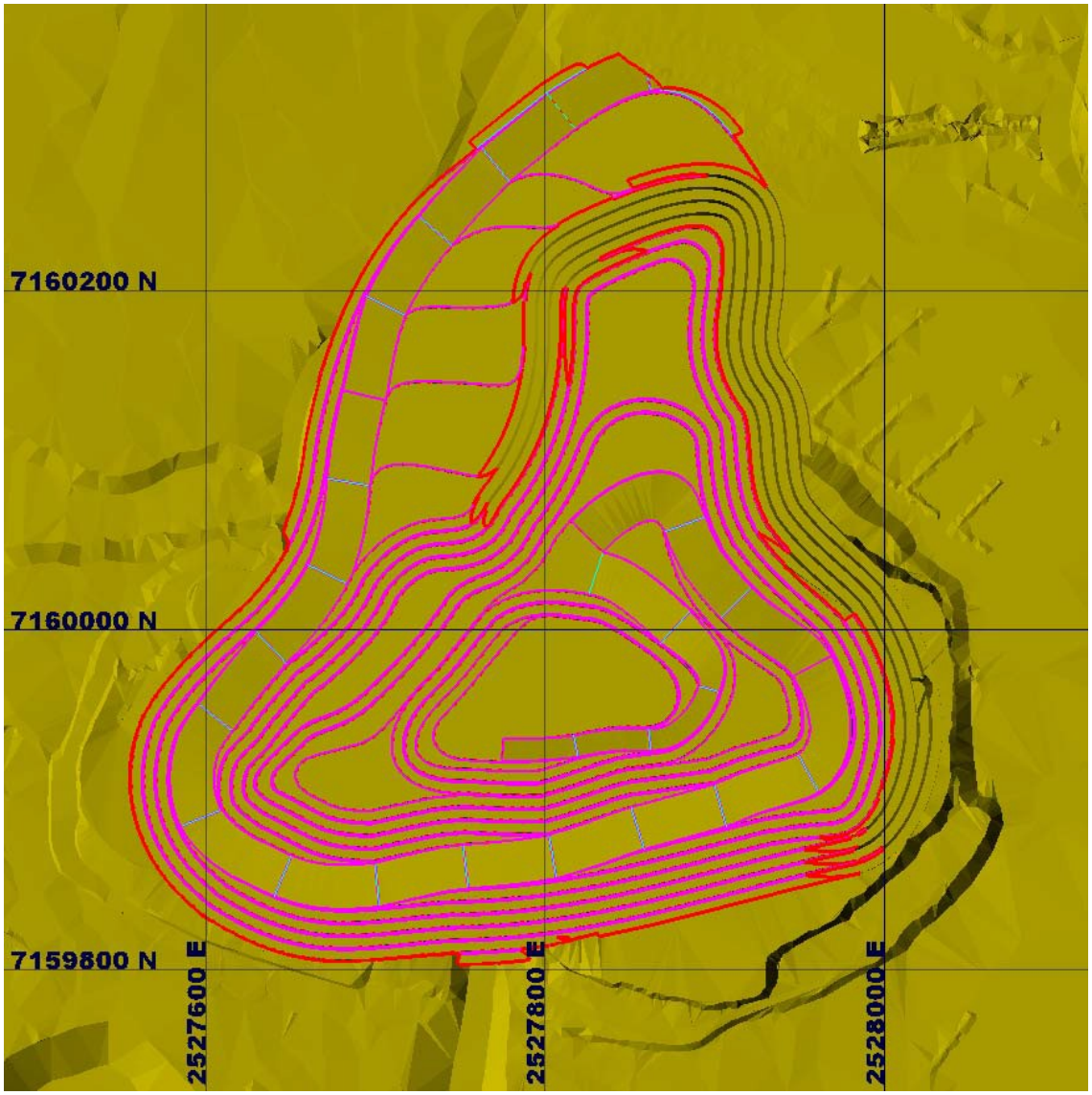


Figure 16.6 – Laiva South Pit Phase 2 Design

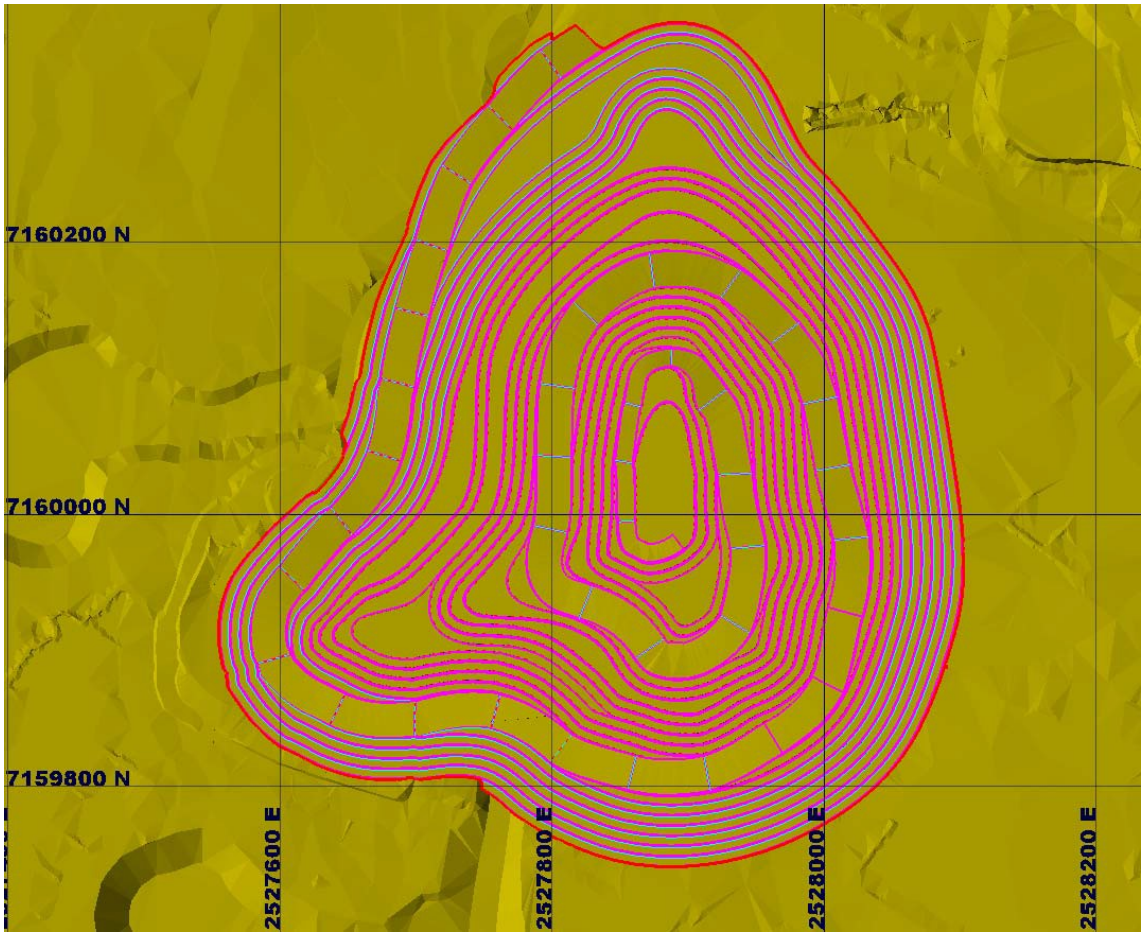


Figure 16.7 – Laiva South Pit Phase 3 Design

16.2.4.2 North Pit Phased Designs

The North pit consists of four phases. Phase 1 is the starter pit with Phases 2 and 3 being expansions of that phase. Phase 4 is developed below Phase 3 at the end of the mine life. Table 16.4 below describes the in-pit mineral resources for the North Pit while Figures 16.8 through 16.12, following this page, show the individual phase layouts.

Table 16.4 – Laiva North Pit Diluted Mineral Resources

Class	Material	Cutoff	Au (g/t)	Tonnes	Au (t.ozs)
Measured	High-Grade	0.700	1.570	308,000	15,500
Indicated	High-Grade	0.700	1.513	2,607,000	126,800
Total M+I	***High-Grade***	***0.700***	***1.519***	***2,915,000***	***142,300***
Measured	Low-Grade	0.323	0.459	318,000	4,700
Indicated	Low-Grade	0.323	0.465	2,843,000	42,500
Total M+I	***Low-Grade***	***0.323***	***0.464***	***3,161,000***	***47,200***
Measured	All	0.323	1.006	626,000	20,200
Indicated	All	0.323	0.966	5,450,000	169,300
Total M+I	***All***	***0.323***	***0.970***	***6,076,000***	***189,500***
Waste	All	---	---	41,960,000	---
Total Material Tonnes		---	---	48,036,000	---
Stripping Ratio		---	---	6.91	---

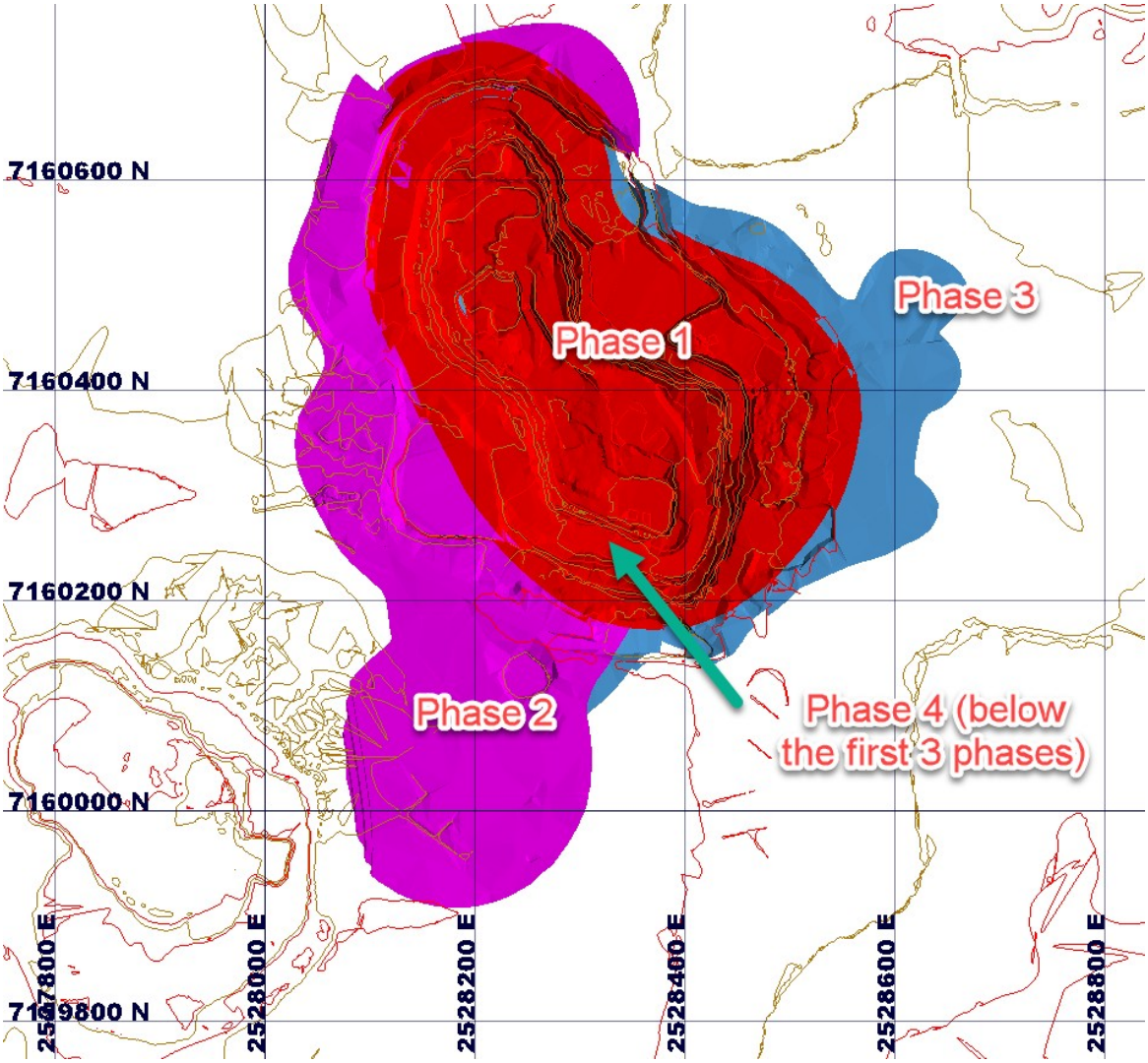


Figure 16.8 – Laiva North Pit Phased Design

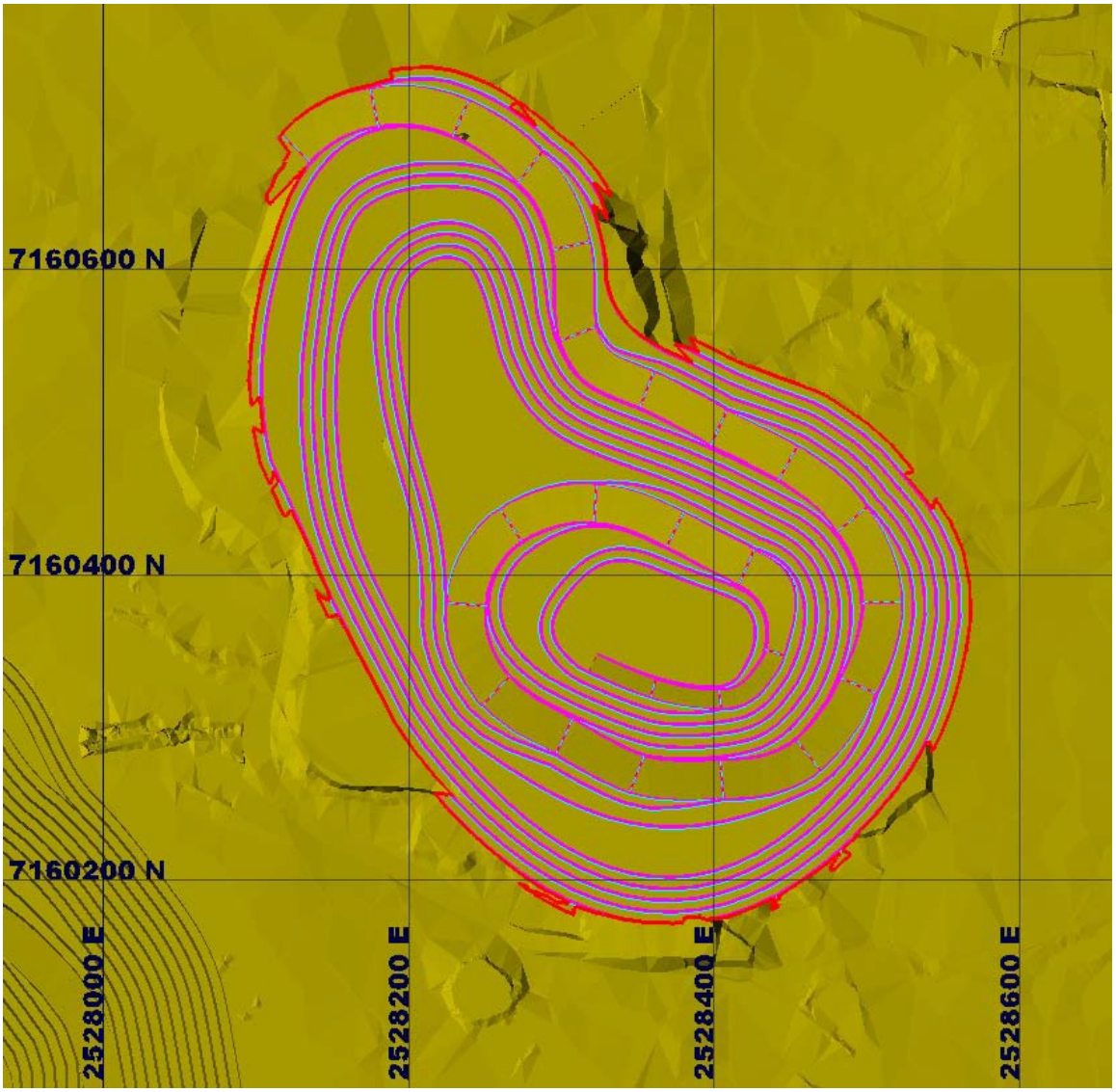


Figure 16.9 – Laiva North Pit Phase 1 Design

Figure 16.10 – Laiva North Pit Phase 2 Design

Figure 16.11 – Laiva North Pit Phase 3 Design

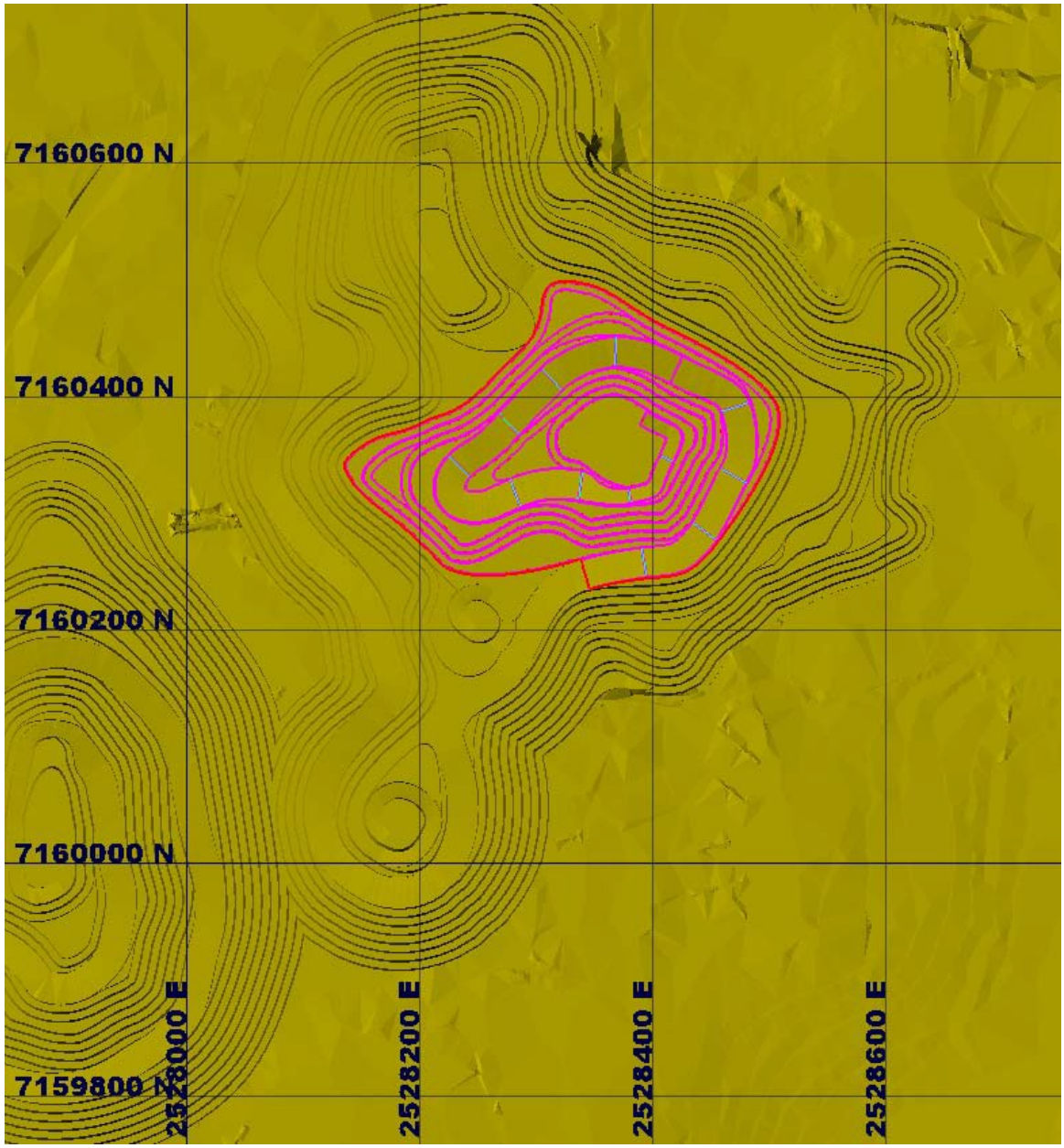


Figure 16.12 – Laiva North Pit Phase 4 Design

16.2.5 Waste Storage Areas

Waste disposal areas for the Laiva Project will continue to be the existing waste disposal areas developed during previous mining operations and are shown in Figure 16.13, following this page.

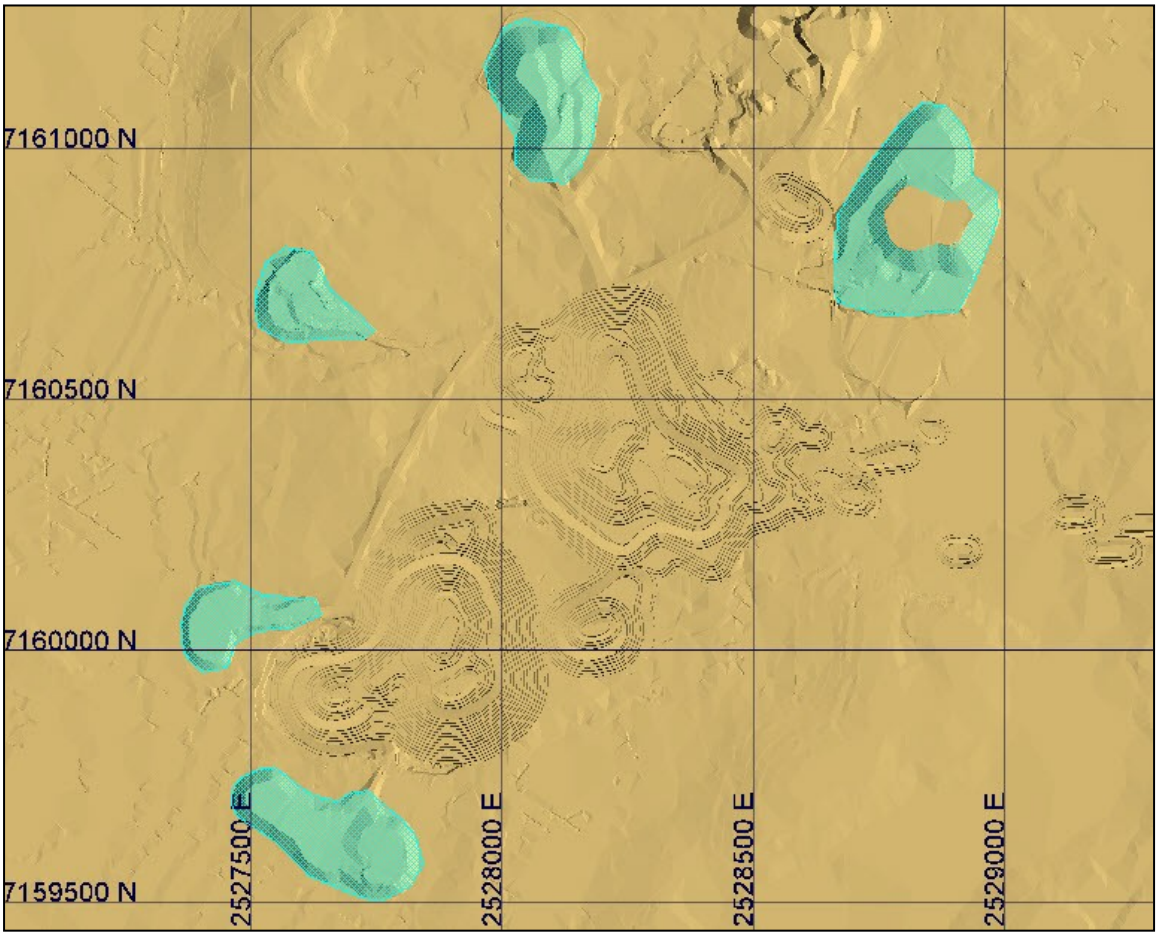


Figure 16.13 – Laiva Existing Waste Disposal Areas (shown in green)

16.3 Mine Production Schedule

A mine production schedule was prepared using Maptek's Chronos scheduling software. Ore selection was based on the economic cutoff described in Table 16.2, Page 16-4 (0.323 g/t). The production schedule has no pre-production period and assumes full plant capacity (1,500,000 tonnes per year) as soon as ore is available to feed it. As much as possible, low-grade material (0.323 g/t to 0.700 g/t gold) is stockpiled for later processing. This is to ensure a maximum gold grade during the earlier years of the project. Table 16.5 and Figure 16.14, following this page, shows the LOM production schedule on an annual basis. Figures 16.15 through 16.20, on the following pages, show the annual pit progress.

Table 16.5 – Laiva LOM Annual Mine Production Schedule from Proven and Probable Reserves

Year	2021	2022	2023	2024	2025	2026	Totals
Diluted High-Grade Ore Production							
Tonnes	404,000	1,860,000	953,000	287,000	988,000	967,000	**5,459,000**
Gold Grade (g/t)	1.641	1.539	1.535	1.732	1.763	1.847	**1.651**
Contained Gold (troy ounces)	21,300	92,100	47,000	16,000	56,000	57,400	**289,800**
Diluted Low-Grade Ore Production							
Tonnes	377,000	1,906,000	1,012,000	186,000	734,000	819,000	**5,034,000**
Gold Grade (g/t)	0.462	0.465	0.465	0.477	0.475	0.473	**0.468**
Contained Gold (troy ounces)	5,600	28,500	15,100	2,900	11,200	12,400	**75,700**
Diluted Total Ore Production Tonnes							
Tonnes	781,000	3,766,000	1,965,000	473,000	1,722,000	1,786,000	**10,493,000**
Gold Grade (g/t)	1.072	0.996	0.984	1.239	1.214	1.217	**1.083**
Contained Gold (troy ounces)	26,900	120,600	62,100	18,900	67,200	69,800	**365,500**
Waste Tonnes	7,876,000	22,604,000	19,682,000	12,004,000	12,435,000	8,260,000	**82,861,000**
Total Tonnes	8,657,000	26,370,000	21,647,000	12,477,000	14,157,000	10,046,000	**93,354,000**
Stripping Ratio	10.08	6.00	10.02	25.38	7.22	4.62	**7.90**

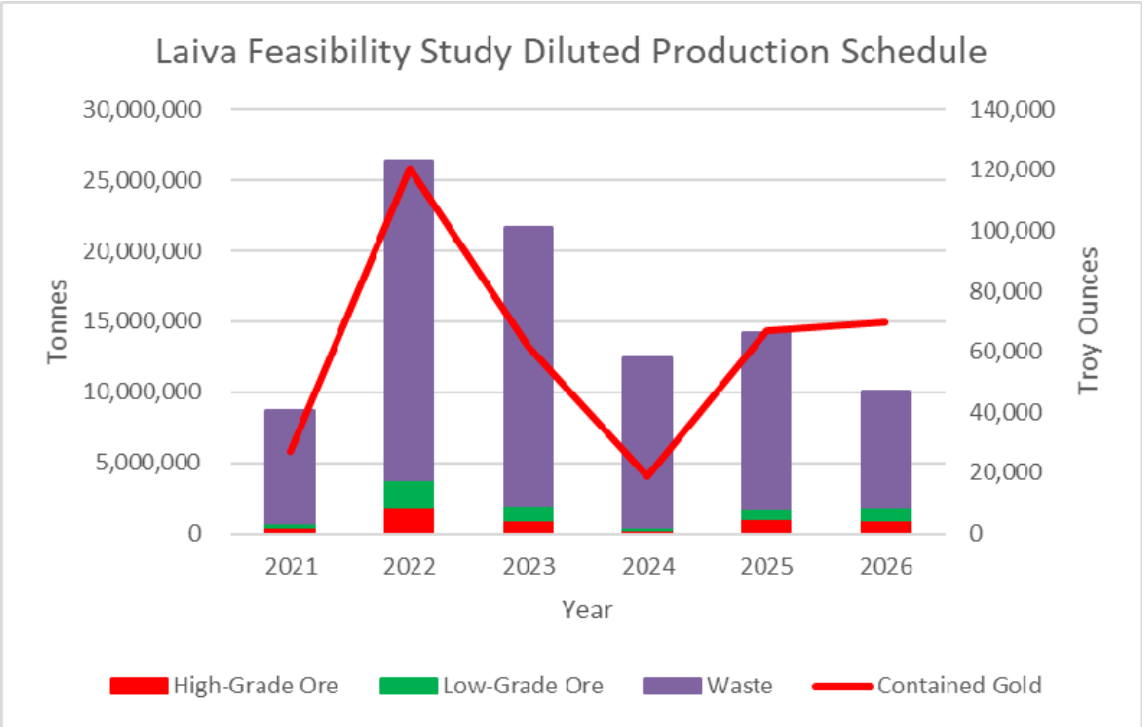


Figure 16.14 – Laiva Diluted Production Schedule

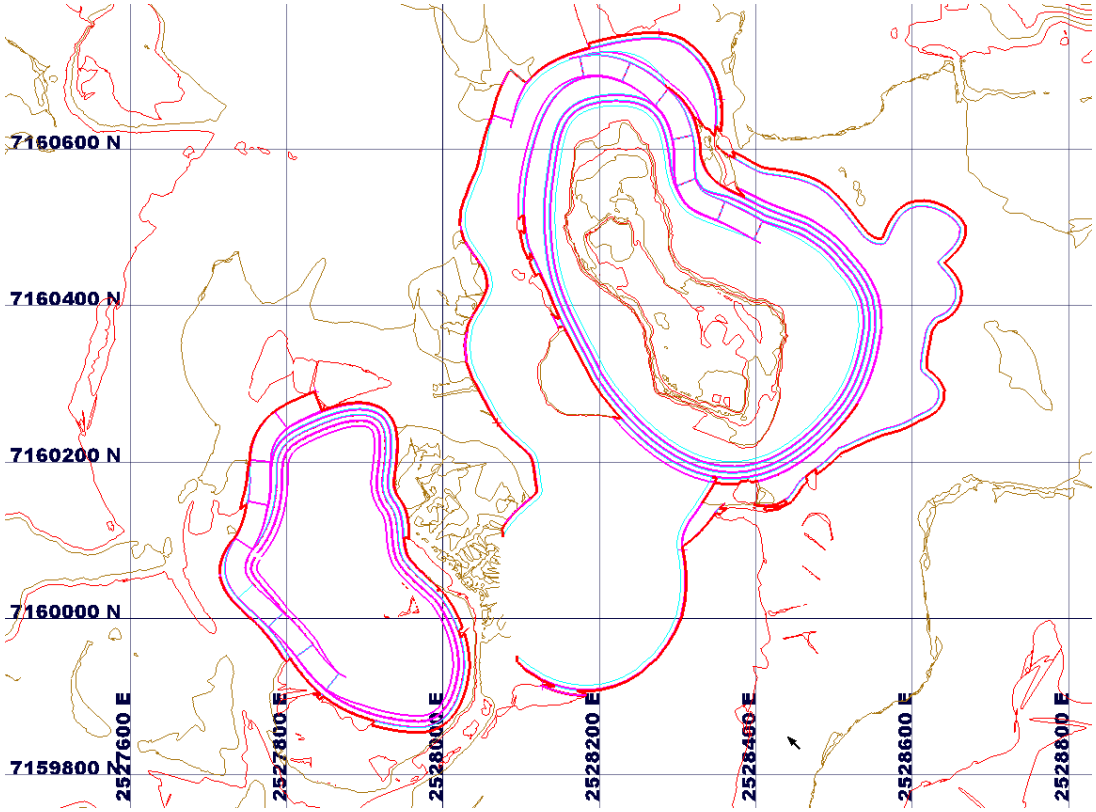


Figure 16.15 – Laiva Pit Progress to the End of 2021

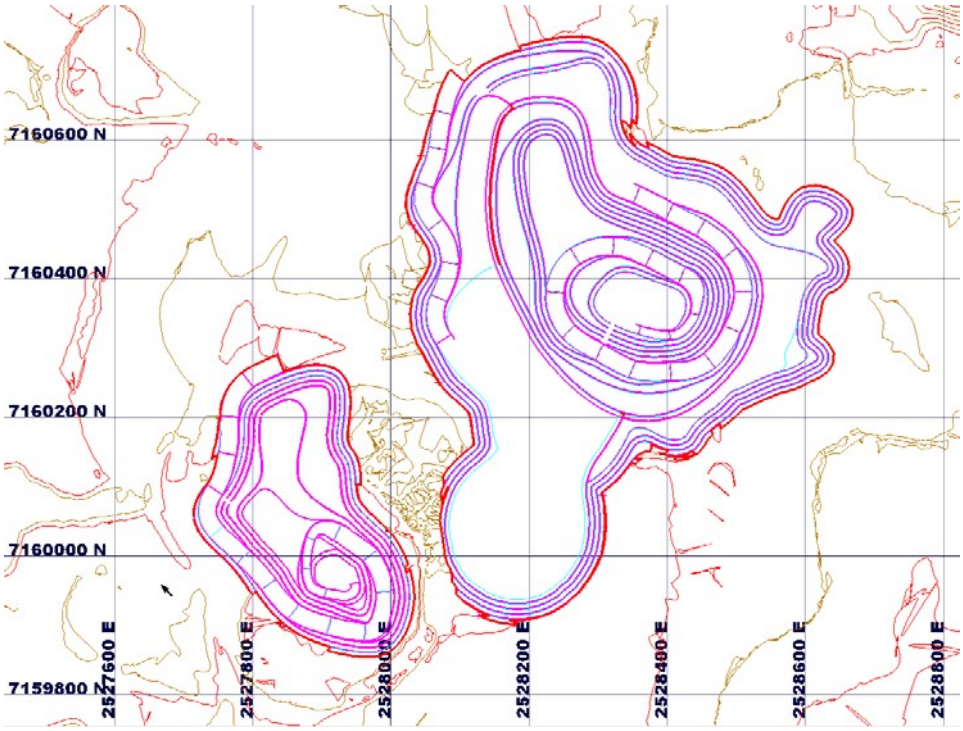


Figure 16.16 – Laiva Pit Progress to the End of 2022

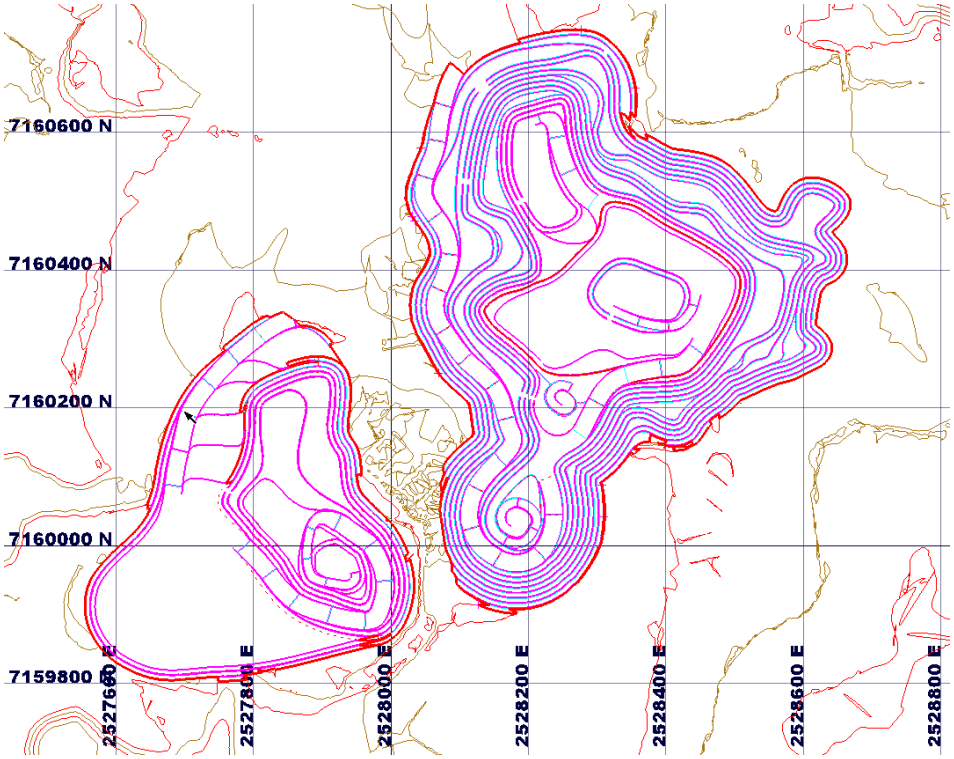


Figure 16.17 – Laiva Pit Progress to the End of 2023

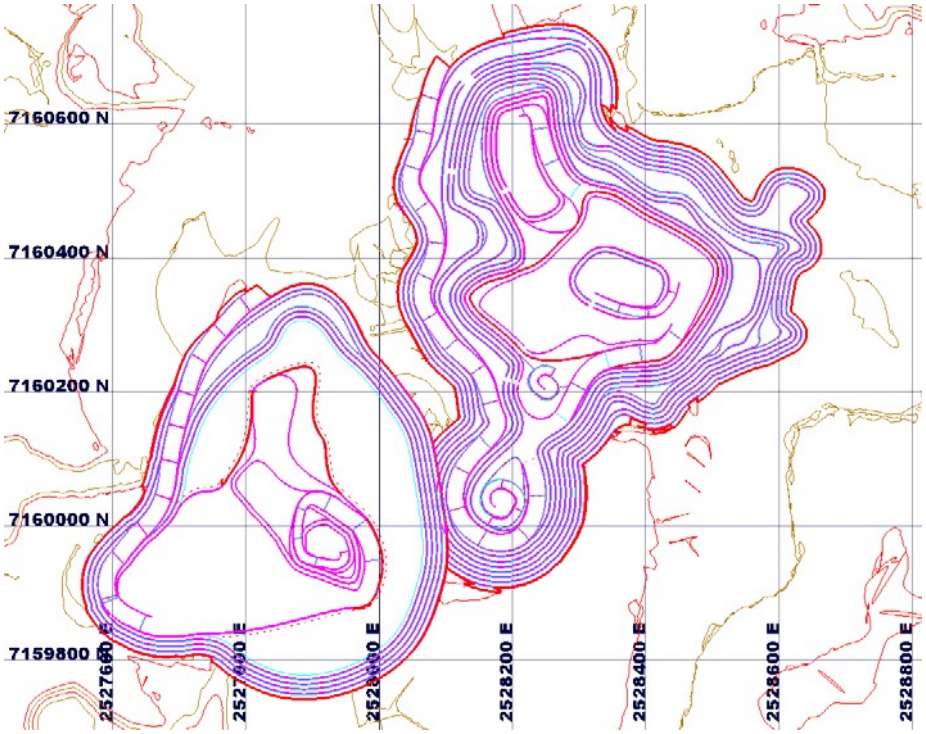


Figure 16.18 – Laiva Pit Progress to the End of 2024

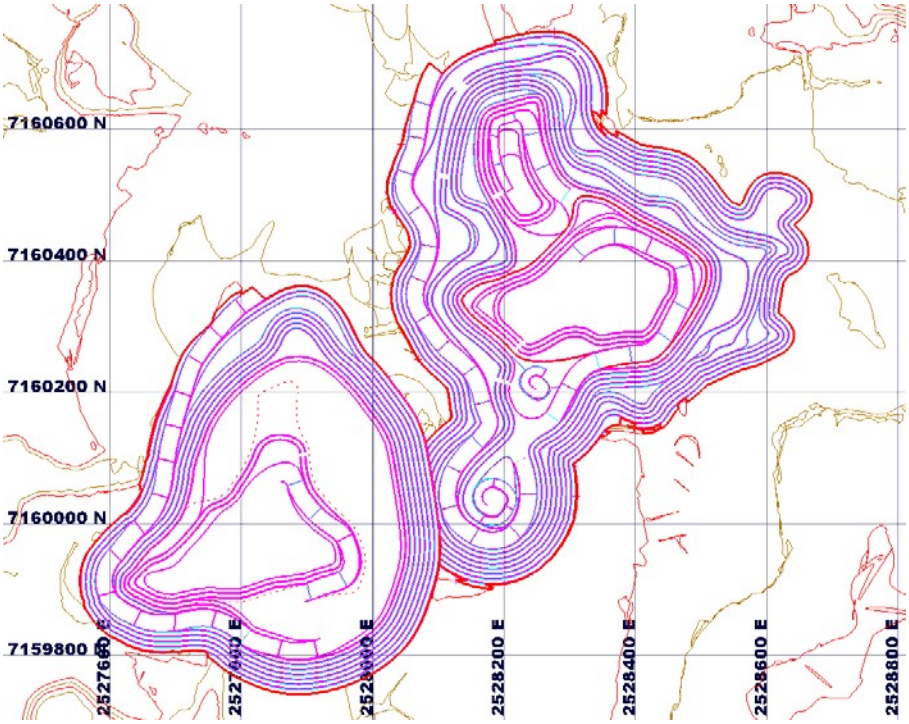


Figure 16.19 – Laiva Pit Progress to the End of 2025

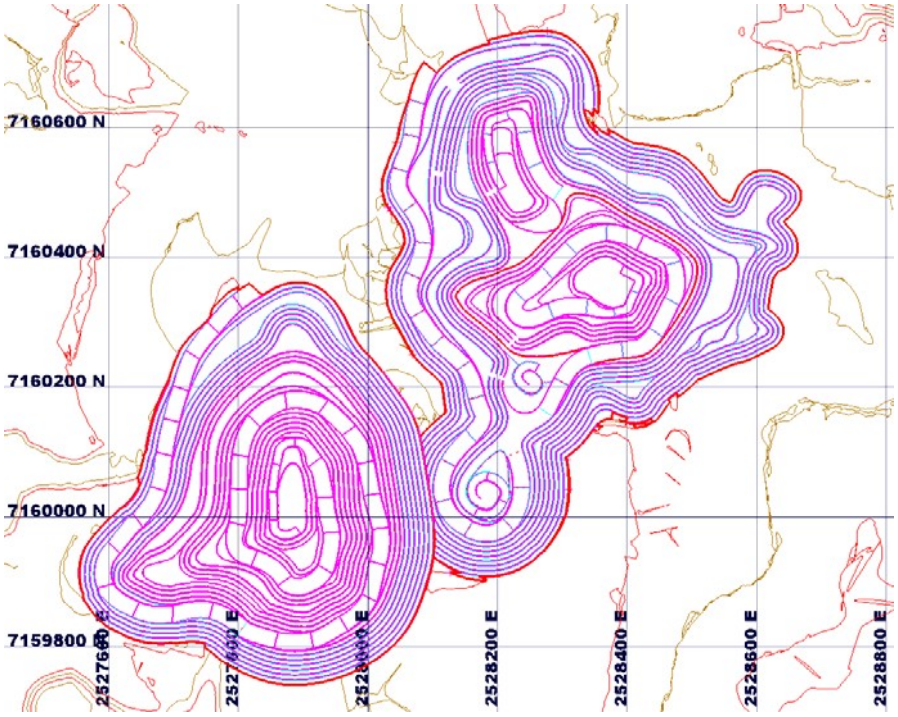


Figure 16.20 – Laiva Pit Progress to the End of Current Mine Life (2026)

16.4 Mine Operations

16.4.1 Mining Activities by Contractor

Since most mining activities will be completed by a mine contractor, no attempt was made to determine contractor mining fleet requirements as well as unit mining costs since they are a function of the mining contract. Unit costs for each component of work contracted are listed in Table 16.6 below. The unit costs were calculated from the contract entered into by OTSO with E. Hartikainen OY. Table 16.7 below summarizes average parameters resulting from production modeling.

Table 16.6 – Mine Contracting Costs

Mine Contracting Costs	Unit Cost (US $/mt)	Ave Annual Cost at Full Production ($M)
Bulk Waste Removal ave Total	1.33	20.952
Mineralized Material Mining Total	1.94	3.886
Mine Support Services ($/total ton mined)	0.10	1.778
Mine Contracting Total		**26.617**

Table 16.7 – Average Mine Production Statistics P&P Reserves Only

Parameter	Average Result
Ore Tons to Mill Including Mining Dilution (mt/yr)	1,998,500
Ore Grade to Mill Including Mining Dilution (g/t)	1.083
Total Waste (mt/yr)	15,783,000
Total Mining (mt/yr)	17,781,000
Strip Ratio (waste: ore)	7.9
Nominal Mining Days per Year	357
Ore Tons per Day (mt/d)	5,600
Waste Tons per Day (mt/d)	44,200
Total Tons per Day (mt/d)	49,800

16.4.2 Production Drilling and Blasting

Given the sheeted vein style of the ore body within the mineralized zone, strict attention to details for drilling and blasting procedures are critical to minimization of mining dilution. Mineralized material and nearby waste will be mined on 5 m sub-benches to minimize inherent dilution at the hanging and foot wall contacts of a dipping mineralized zone (± 80 degrees). Mineralized zones will be defined by closely spaced definition drilling discussed below. The resource block model will be updated based on definition drilling to tightly define suitable zones for extraction of mineralized material. Three dimensional coordinates for mineralized zone outlines will be derived from the model and will be surveyed to define the projected mineralized zones on each bench for drilling by contractor crews. A buffer of near-mineral waste amounting to approximately 15% will be included within the area defined for “mineralized zone” drilling to provide for imprecision in mineralized zone definition resulting from the foregoing procedure.

Bulk hanging wall waste will first be blasted and removed. The mineralized zone will then be drilled and blasted with the free face oriented parallel to the strike of the mineralized zone.

Hanging wall waste will be blasted and mucked prior to blasting the mineralized zone. Mineralized zone blast patterns and delay periods will be laid out by OTSO engineers as necessary to minimize ore and waste mixing to minimize heave, while at the same time achieving good fragmentation.

BOYD strongly recommends blasthole drilling be drilled in the plane of the dip of the mineralized zone, determined on average to be -80 degrees from the horizontal in order to minimize blasting of triangles of waste on the hanging wall and footwall of the mineralized zone.

While blast round design is somewhat variable depending on multiple factors, in keeping with long-standing and well-established relationships, BOYD recommends initial parameters (which have been applied in this Technical Report) shown in table 16.8, following this page, and as illustrated in Figure 16.21, Page 16-26. These parameters will likely need to be adjusted based on actual operating experience to achieve optimum results.

Table 16.8 – Ore and Waste Blasting Design Parameters

Key Assumptions	
Sub-bench Height for Ore and Near Ore (m)	5 m
Full Bench Height for Bulk Waste Only (m)	10 m
Bench Face Batter Angle (deg from horizontal)	-80 deg
Ore and Near Ore BH Diameter (nominal mm)	89 mm
Bulk Waste BH Diameter (nominal mm)	127 mm
Bottom Prime Only	Y
Initial Parameter Calculations	
Ore and Near Ore	
Bench Height (m)	5.00 m
Indicated Burden (m)	3.00 m
Indicated Spacing (m)	4.00 m
Indicated Subdrilling (m)	1.00 m
Indicated Stemming (m)	2.40 m
Hole Length (m)	6.10 m
Charge Length (m)	3.70 m
Loading density (kg/m)	6.84 kg/m
Powder Factor (kg/cu m)	0.41 kg/cu m
Bulk Waste	
Bench Height (m)	10.00 m
Indicated Burden (m)	4.70 m
Indicated Spacing (m)	5.90 m
Indicated Subdrilling (m)	1.50 m
Indicated Stemming (m)	3.80 m
Hole Length (m)	11.60 m
Charge Length (m)	7.80 m
Loading density (kg/m)	13.93 kg/m
Powder Factor (kg/cu m)	0.39 kg/cu m

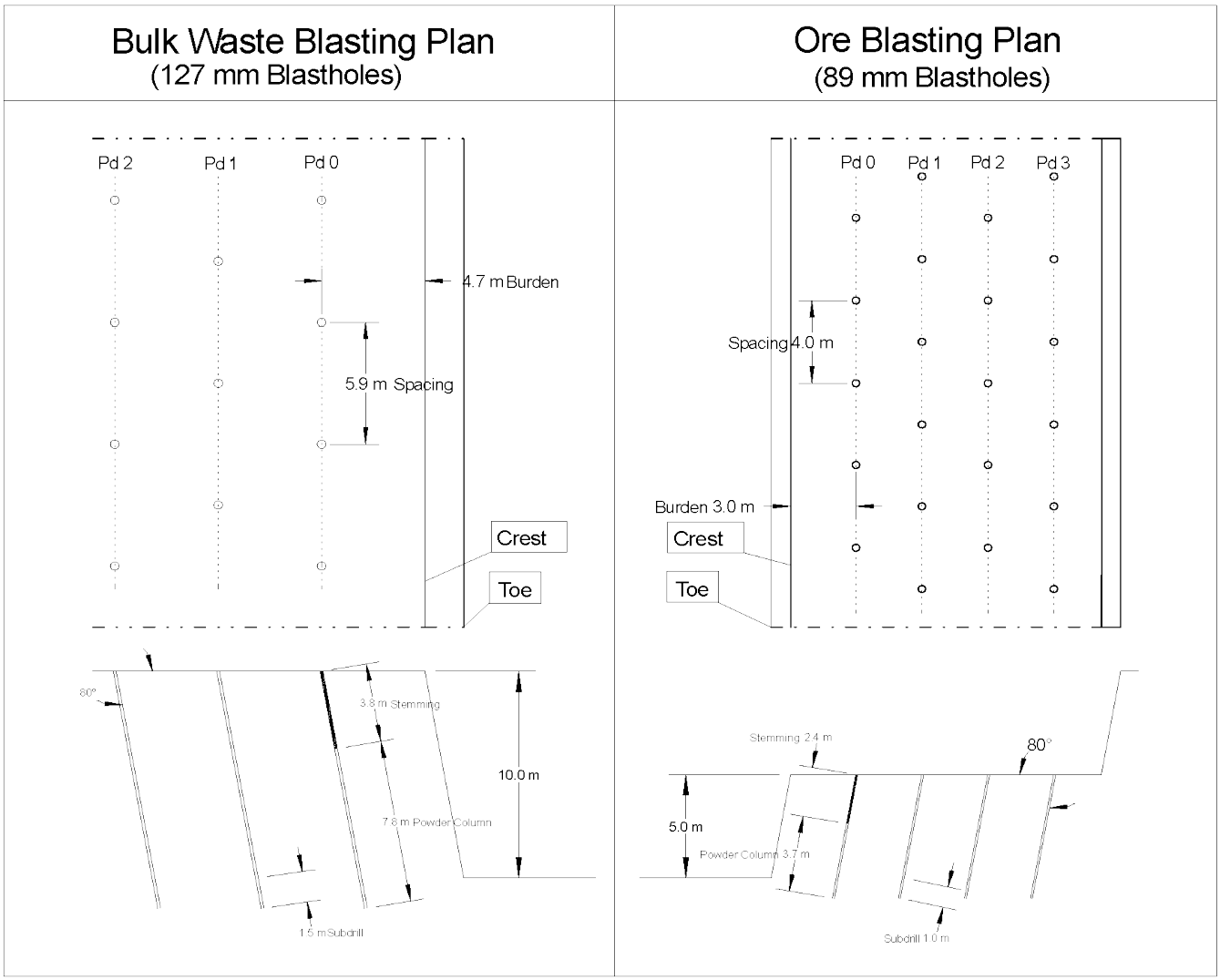


Figure 16.21 – Waste and Ore Blasting Schematic

16.4.3 Grade Control

As previously noted, all reasonable measures to minimize mining dilution are critical to the success of the LGP. Resource development drilling as indicated in previous sections is adequate in BOYD's opinion for production of the resource estimate set forth in Section 14.0. The steeply dipping sheeted vein swarm style of mineralization in mineralized zones however, requires higher definition for effective mining with minimal mining dilution.

14.4.3.1 Definition/Grade Control Drilling

Accordingly, BOYD has recommended and OTSO has adopted a definition/grade control drilling program to achieve the appropriate spatial resolution of mineralized zones as set forth below. In areas of indicated mineralized zones based on the geologic model, RC holes are to be drilled from the hanging wall side of the mineralized zone with an azimuth normal to strike of the mineralized zone at a dip of nominal 45 degrees. Fences of holes are to be spaced 8 meters apart along strike, with spacing of 6 meters across strike. Hole depth is intended to cover two sub-benches in depth.

Holes are to be sampled from collar to TD on one-meter intervals using a rig mounted rotary or riffle splitter. Samples are to be assayed using the on-site PAL 1000 CN soluble leach analysis. Results from the assays will then be modelled on site to identify a minable zone for accurate blasting layout to separately drill, blast, and muck identified ore and waste zones, as shown in Figure 16.22, following this page.

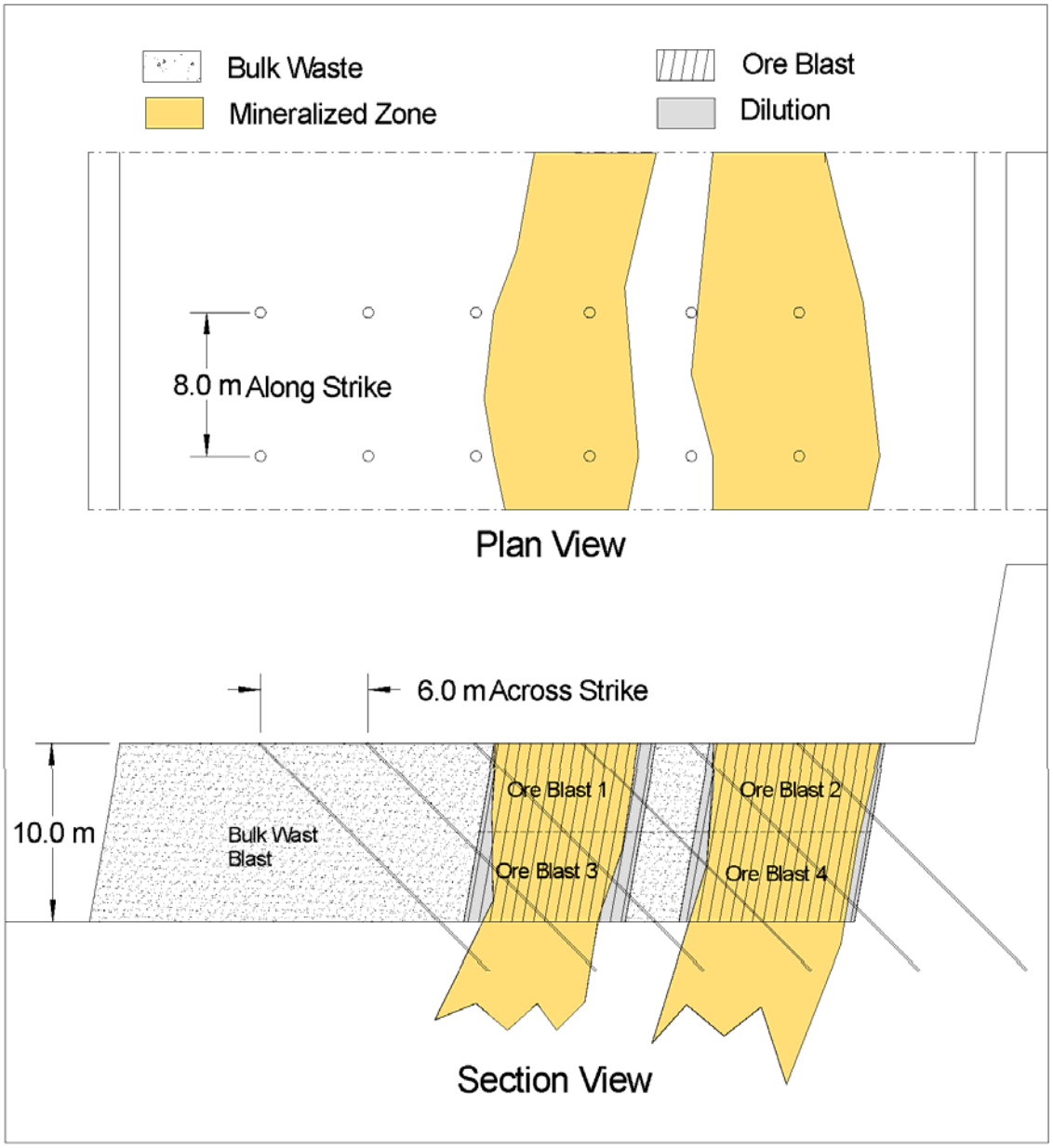


Figure 16.22 – Definition Drilling Schematic

16.4.3.2 Grade Control Direction at the Face

In addition to definition drilling and detailed mapping of mineralized zone/waste interface, BOYD also recommends monitoring ore loading activities at the face by appropriately trained grade control geologists or technicians. Ore loading should be restricted to day shift only.

Following each ore blast, the resulting muck pile should be visually inspected and marked out with marking paint to differentiate probable high grade, low grade and waste by rock characteristics. Duties should include regular visual inspection of the muck pile to ensure the character of material loaded is consistent with the mineralized zone. The grade control geologist/technician should also direct haul truck dispatch to the appropriate stockpile location on the ROM pad by communicating with each departing truck via sign board or radio contact with the driver. Stockpiles on the ROM pad should be clearly marked with signage to ensure dispatched loads are dumped in the appropriate location.

16.5 General Comments

It should be clearly understood that OTSO is responsible for directing mining activities, including:

- Mine Design
- Production Sequencing
- Definition/Grade Control Drilling
- Detailed Mapping
- Ore And Waste Blasting Design
- Grade Control at The Face
- Truck Dispatch When Hauling Ore
- Regular Survey of Excavated Volumes
- Reconciliation of Resource Estimates to Actual Material Mined

It should be clear that the mining contractor is engaged to move bcm/tonnes and is not responsible for mine management. Previous operators were unsuccessful in large measure due to too much reliance on the mining contractor for the tasks enumerated above.

17.0 RECOVERY METHODS

17.1 Introduction

The Laiva Gold Mine Process facilities were constructed in 2011 for a reported 233 million Euro constructed cost. It operated for a total of seventeen months; with few problems associated with equipment operation or flow sheet design. Essentially, the same process facilities and flow sheet will be utilized for future operations with only minor modifications to enhance productivity and improve key process parameter measurement and recording systems.

17.2 Existing Processing Plant and Facilities

17.2.1 Existing Flow Sheet

The mineral processing plant is located approximately 1 km northeast from the South and North open pits on the LGP site. Processing consists of three primary circuit components, including: crushing and grinding, pre-concentration by flash flotation and gravity concentration, followed by High and Low grade CIL leach circuits. Design throughput rate is 250 tons per hours (6,000 mtpd / 2 M mtpy). Process flow sheets for the as-built process facilities are shown as preliminary, but are in fact final in Figures 17.1 through 17.4, following this page.

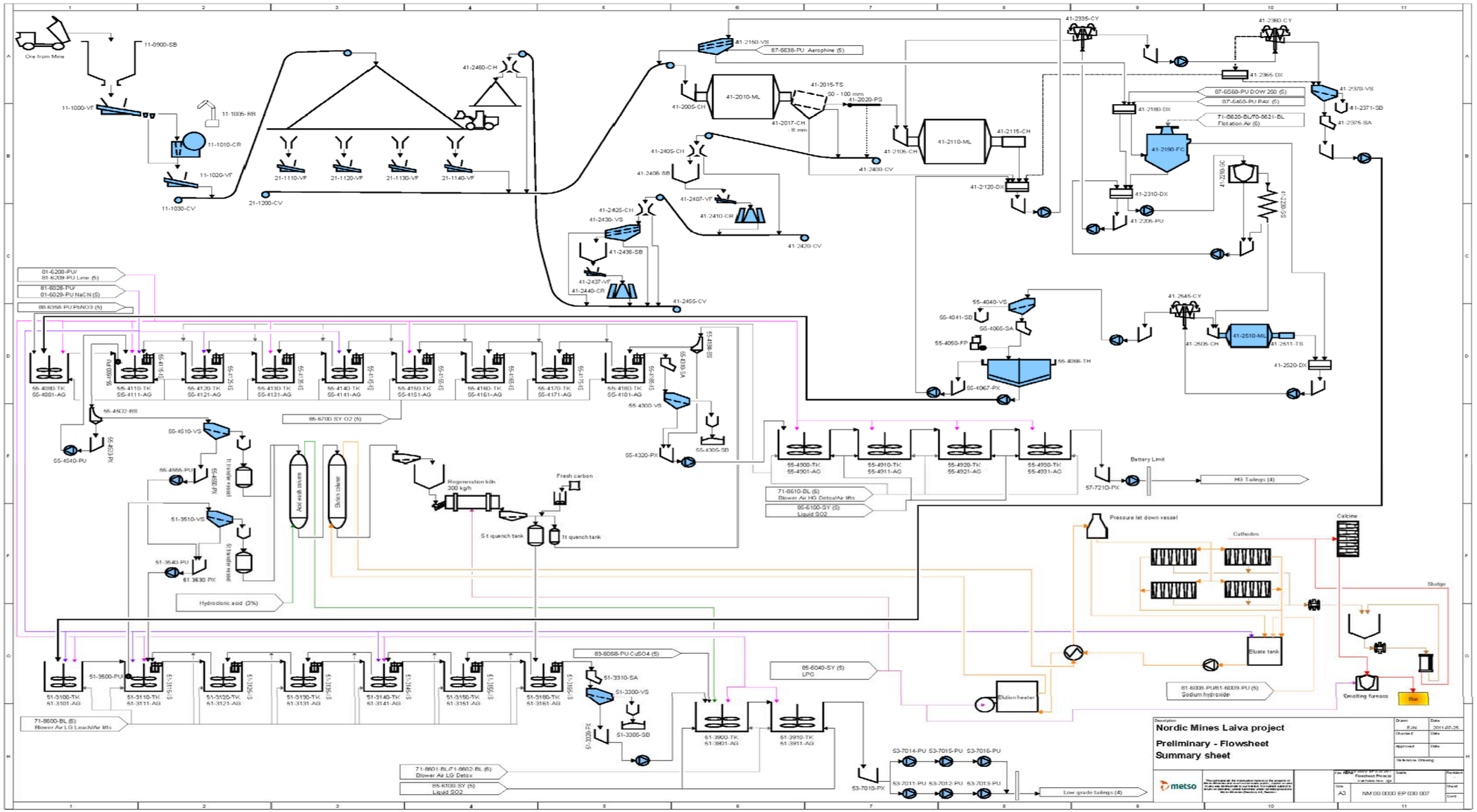
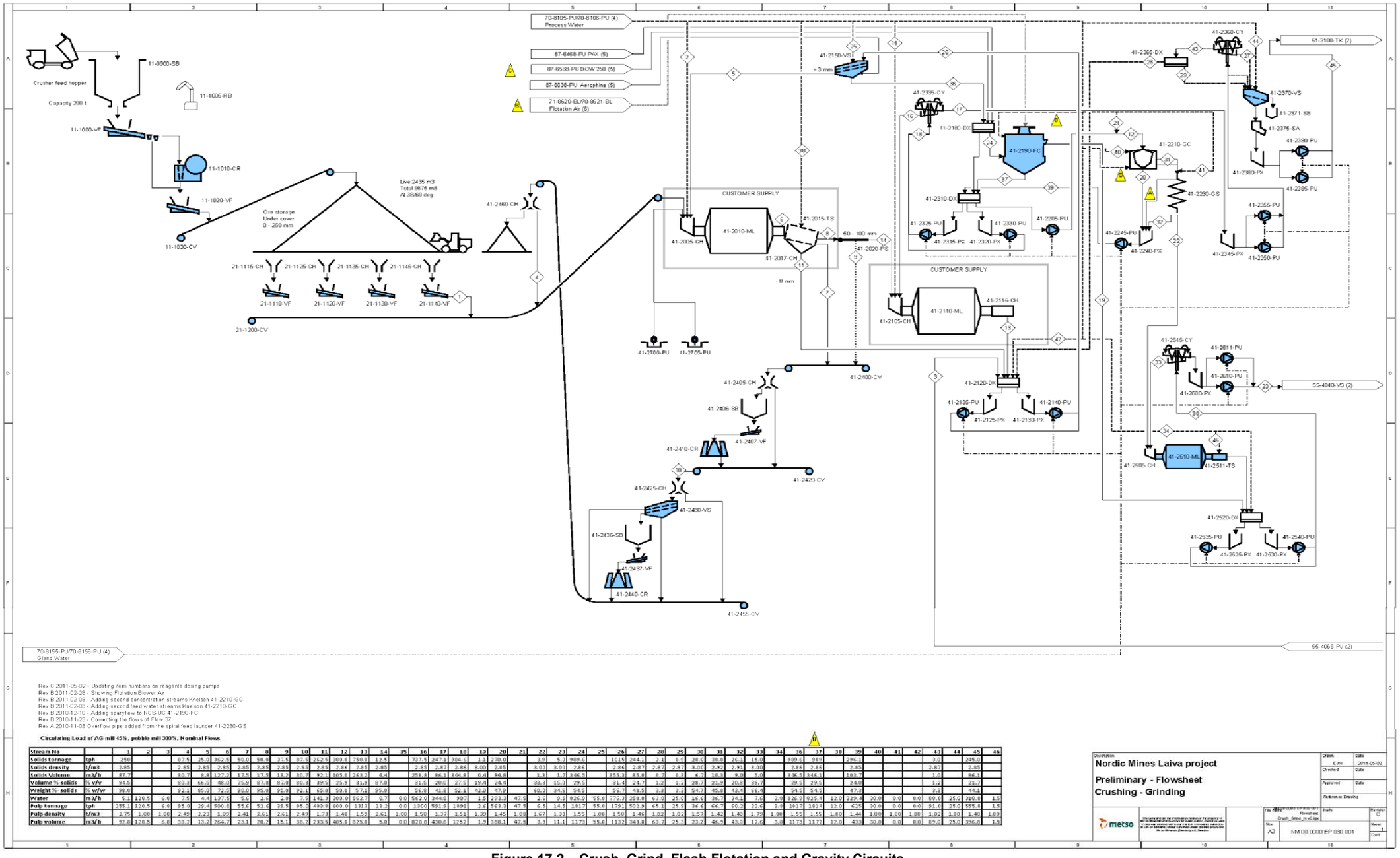


Figure 17.1 – Summary Flow Sheet



Stream No		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	36	37	38	39	40	41	42	43	44	45	46
Solids tonnage	tph	250			87.5	25.0	362.5	50.0	50.0	37.5	87.5	262.5	300.0	750.0	12.5		737.5	247.3	984.0	1.3	270.0		3.9	5.0	989.0		1015	244.1	2.1	0.9	20.0	30.0	26.1	15.0		989.0	989		296.1				3.0		245.0	
Solids density	t/m3	2.85			2.85	2.85	2.85	2.85	2.85	2.85	2.85	2.85	2.86	2.85	2.85		2.85	2.87	2.86	8.00	2.85		8.00	8.00	2.86		2.86	2.87	2.87	2.87	8.00	2.92	2.91	8.00		2.86	2.86		2.85				2.87		2.85	
Solids Volume	m3/h	87.7			30.7	8.8	127.2	17.5	17.5	13.2	30.7	92.1	105.8	263.2	4.4		258.8	86.1	344.8	0.4	94.8		1.3	1.7	346.5		355.3	85.8	0.7	0.3	6.7	10.3	9.0	5.0		346.5	346.5		103.7				1.0		86.1	
Volume %-solids	% v/v	94.5			80.3	66.5	68.0	75.9	87.0	87.0	80.8	89.5	25.9	31.9	87.0		31.5	20.0	27.5	19.0	24.0		86.8	15.0	29.5		31.4	24.7	1.2	1.2	28.7	21.9	20.8	89.7		29.5	29.5		24.0				1.2		23.2	
Weight %-solids	% w/w	90.0			92.1	85.0	72.5	90.0	95.0	95.0	92.1	65.8	59.8	57.1	95.0		56.8	41.8	52.1	42.0	47.9		60.0	34.6	54.5		56.7	48.5	3.3	3.3	54.7	45.0	43.4	66.4		54.5	54.5		47.3				3.3		44.1	
Water	m3/h	5.1	120.5	6.8	7.5	4.4	127.5	5.6	2.6	2.0	7.5	141.3	200.0	562.7	0.7	0.0	562.0	344.8	907	1.5	203.3	47.5	2.6	9.5	826.9	55.8	776.3	259.8	63.0	25.0	16.6	26.7	24.1	7.5	3.8	826.9	825.4	12.0	329.4	30.0	0.0	0.0	60.0	25.0	310.0	1.5
Pulp tonnage	tph	255.1	120.5	6.8	95.0	29.4	500.0	55.6	52.6	39.5	95.0	403.8	600.0	1313	13.2	0.0	1300	591.8	1891	2.6	563.3	47.5	6.5	14.5	1817	55.8	1791	502.9	65.1	25.9	36.6	66.7	60.2	22.5	3.8	1817	1814	12.0	625	30.0	0.0	0.0	91.0	25.0	555.8	1.5
Pulp density	t/m3	2.75	1.00	1.00	2.40	2.23	1.99	2.41	2.61	2.61	2.49	1.73	1.48	1.59	2.61	1.00	1.50	1.37	1.53	1.39	1.45	1.00	1.67	1.39	1.55	1.00	1.50	1.46	1.02	1.02	1.57	1.42	1.40	1.79	1.00	1.55	1.55	1.00	1.44	1.00	1.00	1.00	1.02	1.00	1.40	1.00
Pulp volume	m3/h	92.8	120.5	6.8	38.3	13.2	264.7	23.1	20.2	15.1	38.2	233.5	405.8	825.8	5.0	0.0	870.8	430.8	1232	1.9	388.1	47.5	3.9	11.1	1173	55.8	1132	343.8	63.7	25.3	23.2	46.9	43.0	12.6	3.8	1173	1172	12.0	433	30.0	0.0	0.0	89.0	25.0	396.8	1.5

Figure 17.2 – Crush, Grind, Flash Flotation and Gravity Circuits

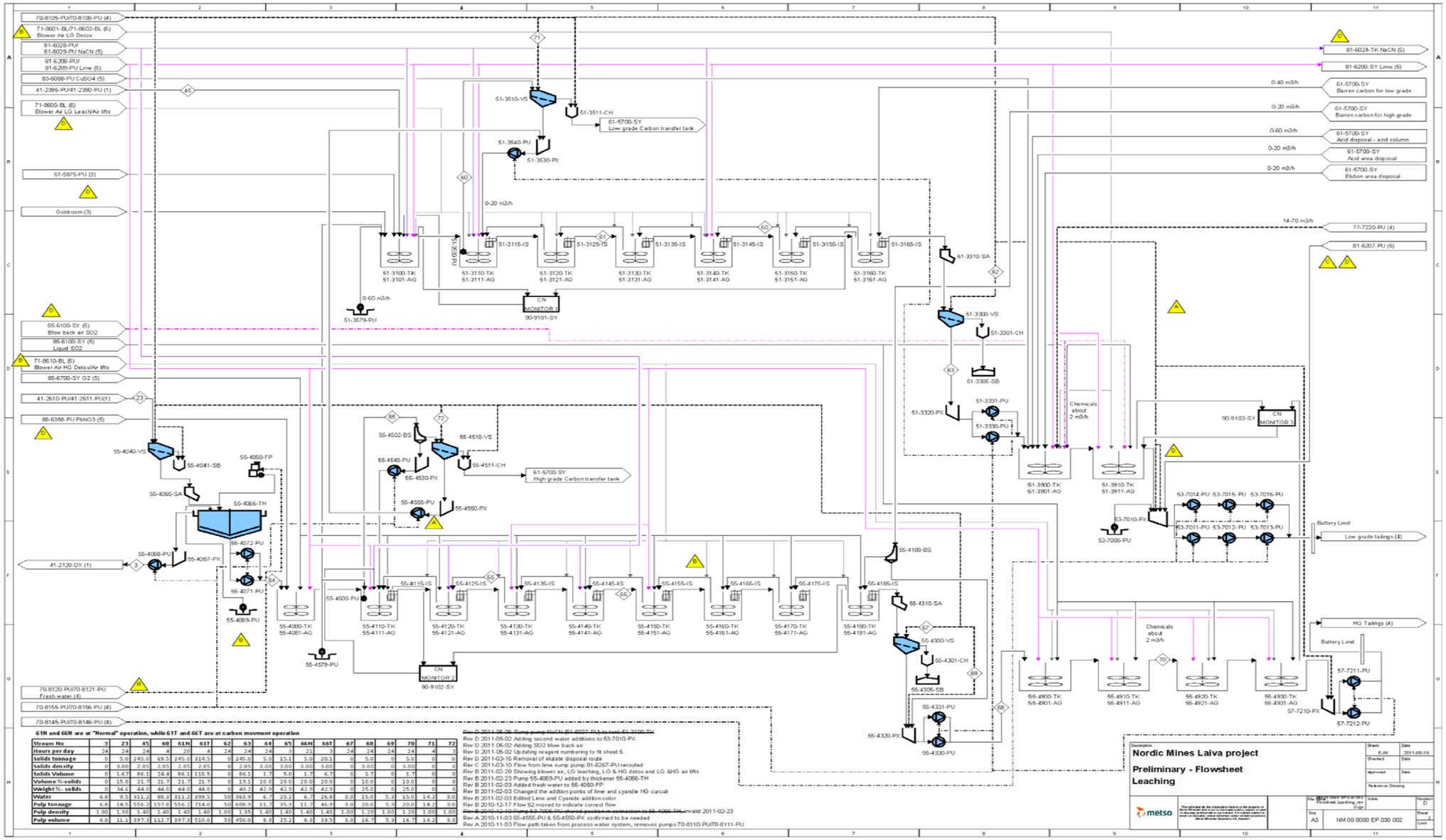


Figure 17.3 – High and Low Grade CIL Circuits

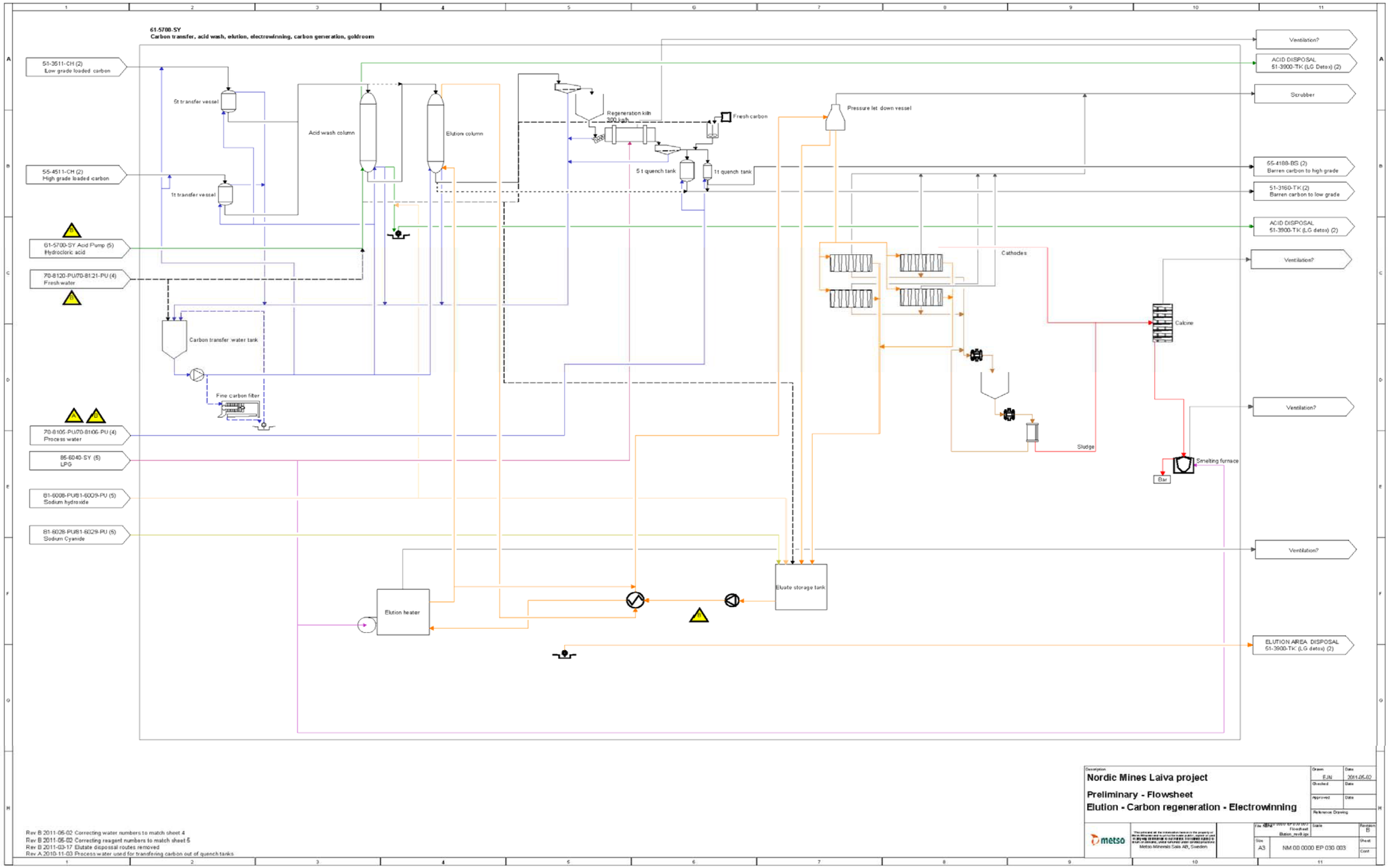


Figure 17.4 – Elution, Carbon Regen, Electrowinning

17.2.2 Existing Plant Layout and Major Equipment

The following Figures 17.5 through 17.13, following this page, include general arrangement drawings of the existing LGP process plant.

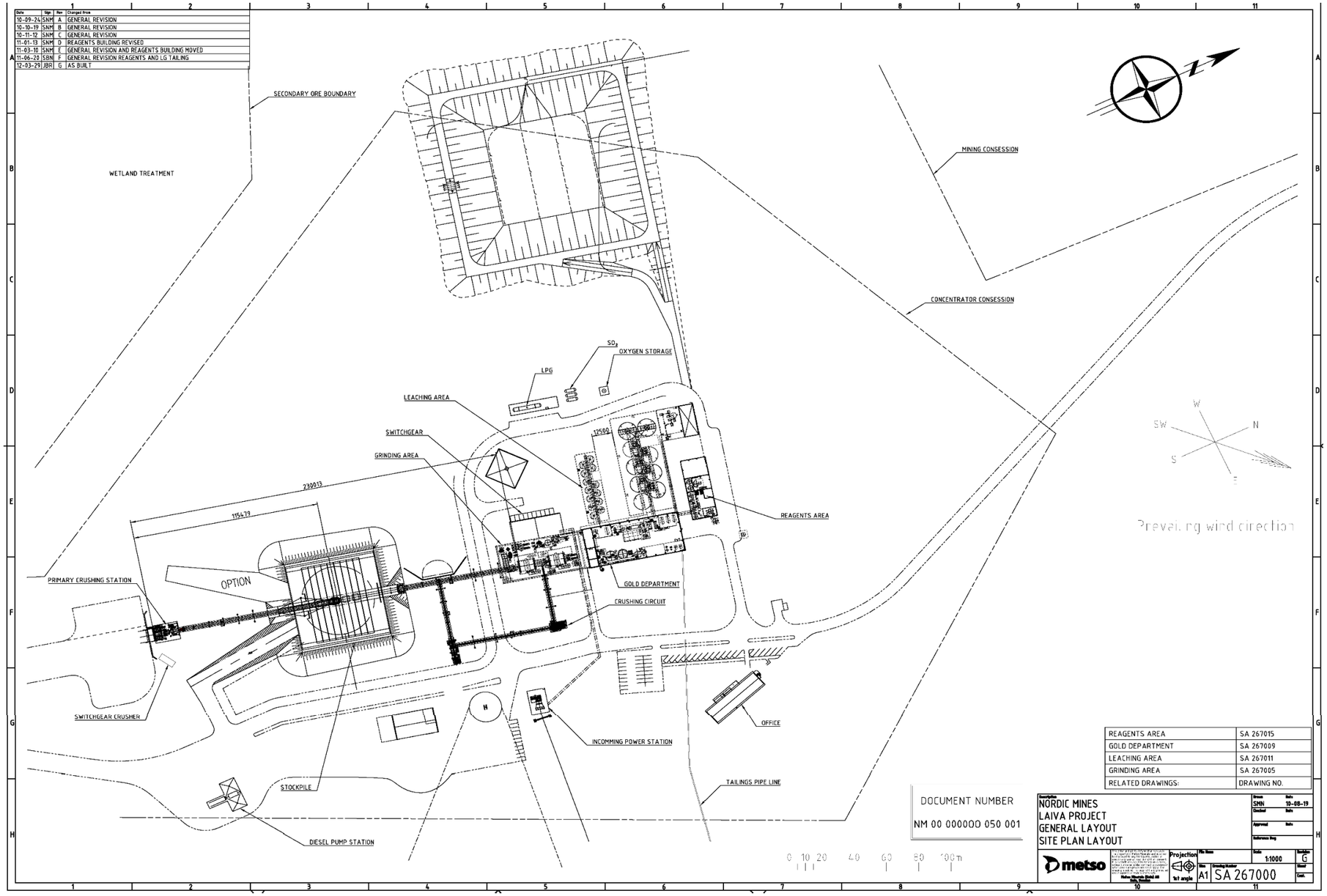


Figure 17.5 – Site Plan Layout

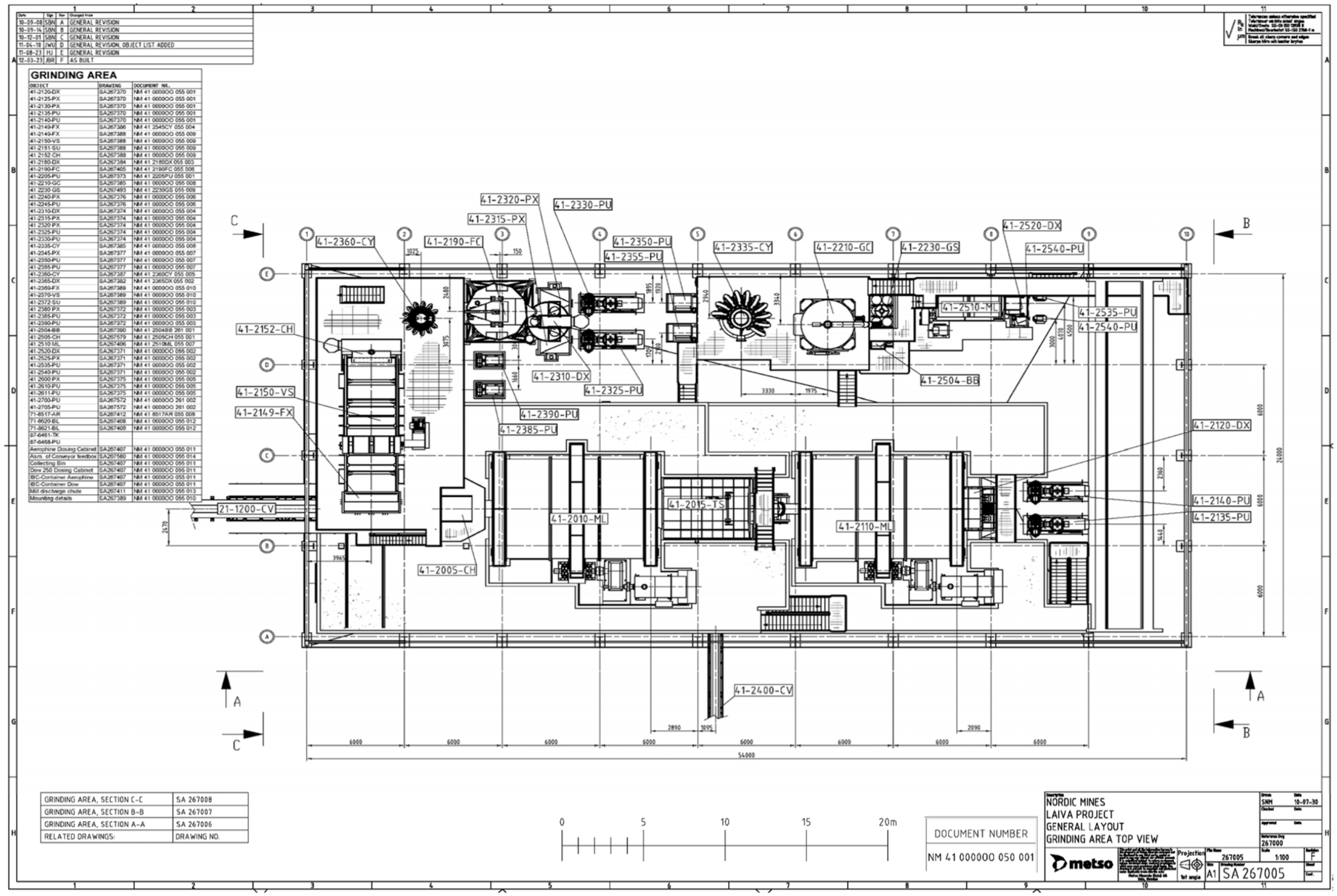
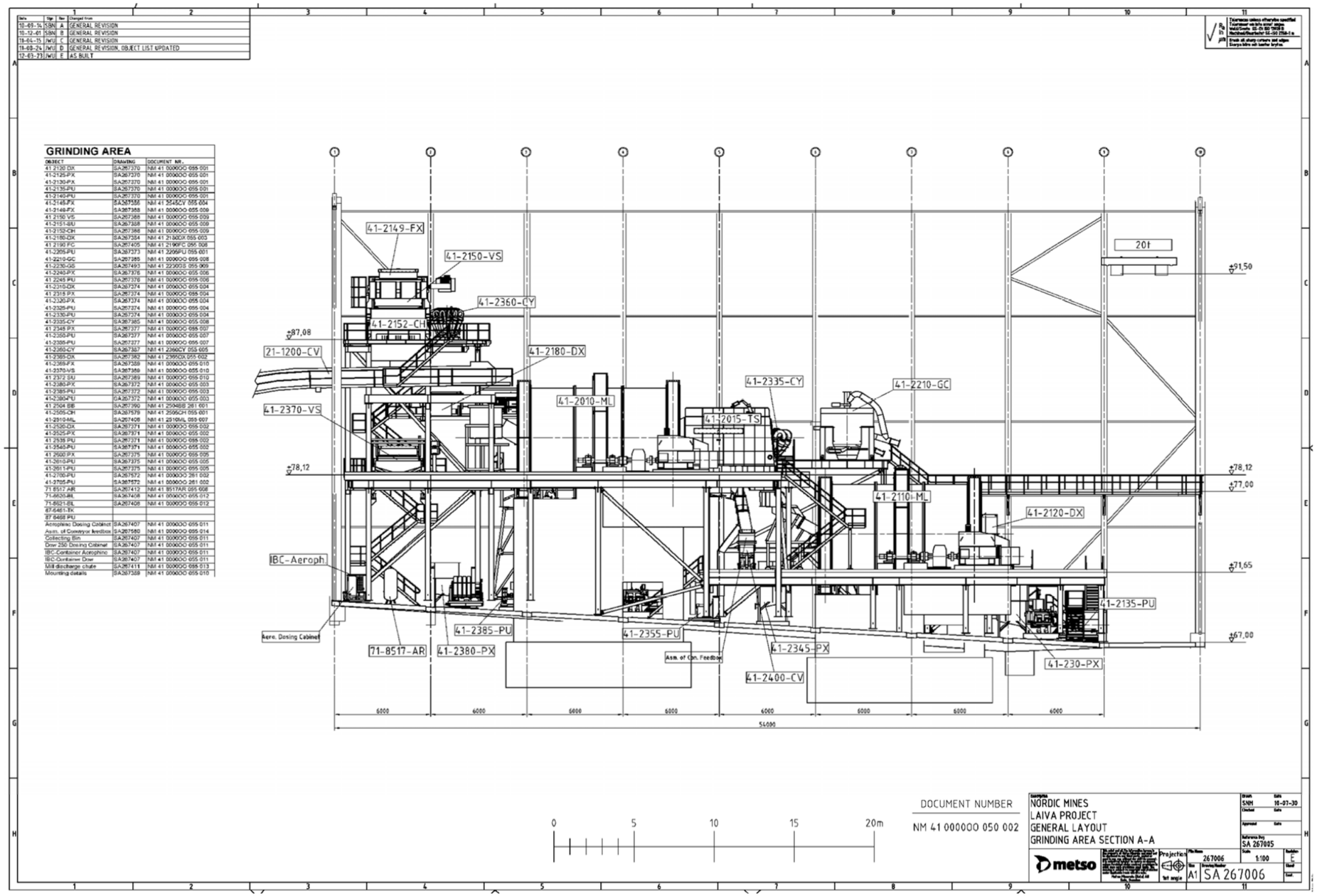


Figure 17.6 – GA Grinding Area Plan View



GRINDING AREA

OBJECT	DRAWING	DOCUMENT NR.
41-2120-DX	SA267370	NM 41 000000-055-001
41-2125-PX	SA267370	NM 41 000000-055-001
41-2130-PX	SA267370	NM 41 000000-055-001
41-2135-PU	SA267370	NM 41 000000-055-001
41-2140-PU	SA267370	NM 41 000000-055-001
41-2145-PX	SA267386	NM 41 2545CV-055-004
41-2149-FX	SA267388	NM 41 000000-055-009
41-2150-VS	SA267388	NM 41 000000-055-009
41-2151-BU	SA267388	NM 41 000000-055-009
41-2152-CH	SA267388	NM 41 000000-055-009
41-2180-DX	SA267384	NM 41 2180DX-055-003
41-2190-FC	SA267405	NM 41 2190FC-055-006
41-2205-PU	SA267373	NM 41 2206PU-055-001
41-2210-GC	SA267385	NM 41 000000-055-008
41-2230-GS	SA267493	NM 41 2230GS-055-009
41-2245-PX	SA267376	NM 41 000000-055-006
41-2245-PU	SA267376	NM 41 000000-055-006
41-2315-DX	SA267374	NM 41 000000-055-004
41-2315-PX	SA267374	NM 41 000000-055-004
41-2325-PX	SA267374	NM 41 000000-055-004
41-2325-PU	SA267374	NM 41 000000-055-004
41-2330-PU	SA267374	NM 41 000000-055-004
41-2335-CY	SA267385	NM 41 000000-055-008
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87-6468-PU		
Aerophinc Dosing Cabinet	SA267407	NM 41 000000-055-011
Asm. of Conveyor feedbox	SA267580	NM 41 000000-055-014
Collecting Bin	SA267407	NM 41 000000-055-011
Dow 250 Dosing Cabinet	SA267407	NM 41 000000-055-011
IBC-Container Aerophinc	SA267407	NM 41 000000-055-011
IBC-Container Dow	SA267407	NM 41 000000-055-011
Mill discharge chute	SA267411	NM 41 000000-055-013
Mounting details	SA267389	NM 41 000000-055-010

Figure 17.7 – Grinding Area Section A - A View

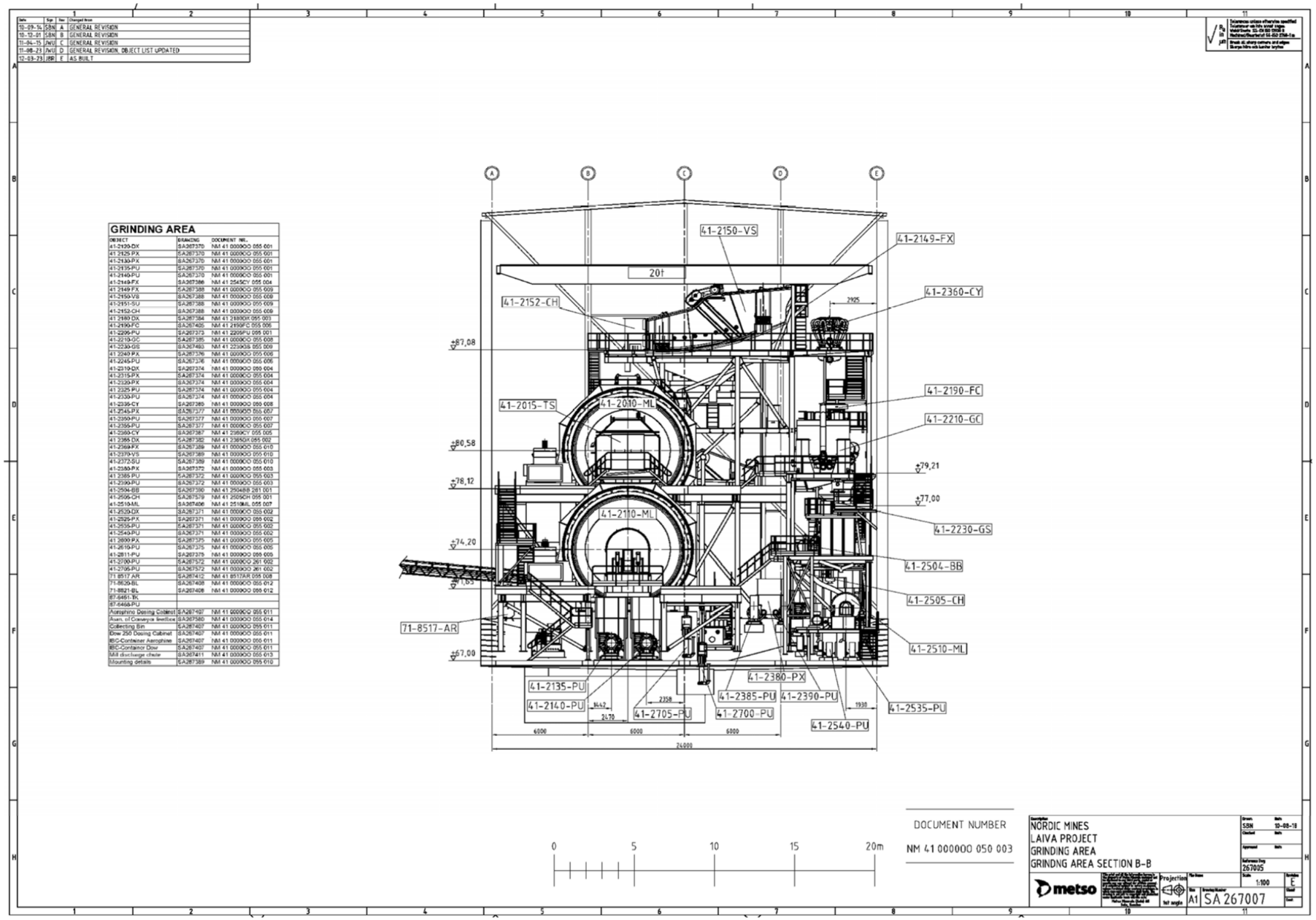
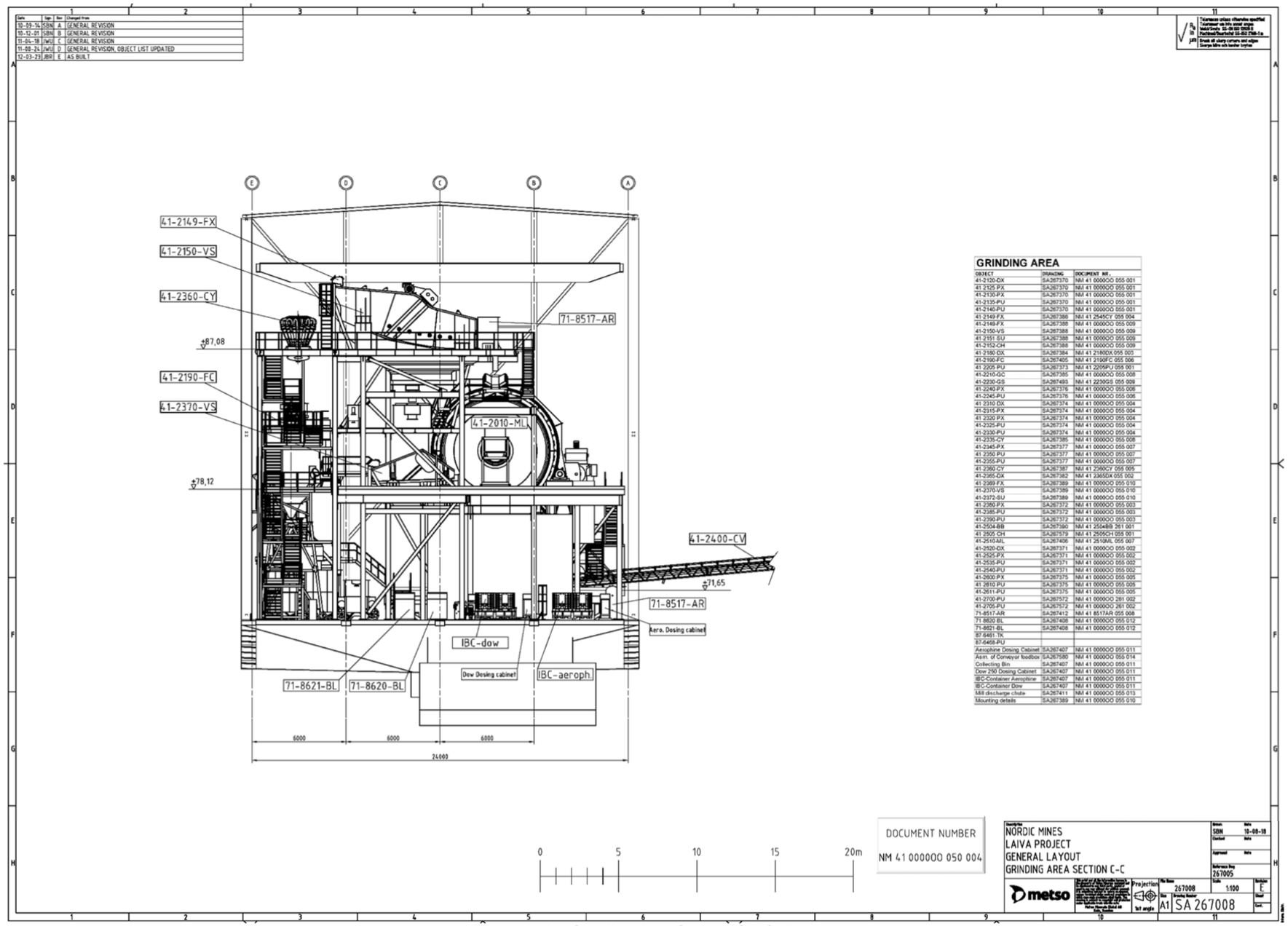


Figure 17.8 – Grinding Area Section B - B View



Date	Sign	Rev	Changed from
10-09-14	SBN	A	GENERAL REVISION
10-12-01	SBN	B	GENERAL REVISION
11-04-18	JWU	C	GENERAL REVISION
11-08-24	JWU	D	GENERAL REVISION, OBJECT LIST UPDATED
12-03-23	JBR	E	AS BUILT

GRINDING AREA

OBJECT	DRAWING	DOCUMENT NR.
41-2120-DX	SA267370	NM 41 0000OO 055 001
41-2125-PX	SA267370	NM 41 0000OO 055 001
41-2130-PX	SA267370	NM 41 0000OO 055 001
41-2135-PU	SA267370	NM 41 0000OO 055 001
41-2140-PU	SA267370	NM 41 0000OO 055 001
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41-2152-CH	SA267388	NM 41 0000OO 055 009
41-2180-DX	SA267384	NM 41 2180DX 055 003
41-2190-FC	SA267405	NM 41 2190FC 055 006
41-2205-PU	SA267373	NM 41 2205PU 055 001
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41-2355-PU	SA267377	NM 41 0000OO 055 007
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71-8621-BL	SA267408	NM 41 0000OO 055 012
87-6461-TK		
87-6468-PU		
Aerophine Dosing Cabinet	SA267407	NM 41 0000OO 055 011
Asm. of Conveyor foodbox	SA267580	NM 41 0000OO 055 014
Collecting Bin	SA267407	NM 41 0000OO 055 011
Dow 250 Dosing Cabinet	SA267407	NM 41 0000OO 055 011
IBC-Container Aerophine	SA267407	NM 41 0000OO 055 011
IBC-Container Dow	SA267407	NM 41 0000OO 055 011
Mill discharge chute	SA267411	NM 41 0000OO 055 013
Mounting details	SA267389	NM 41 0000OO 055 010

Figure 17.9 – Grinding Area Section C – C View

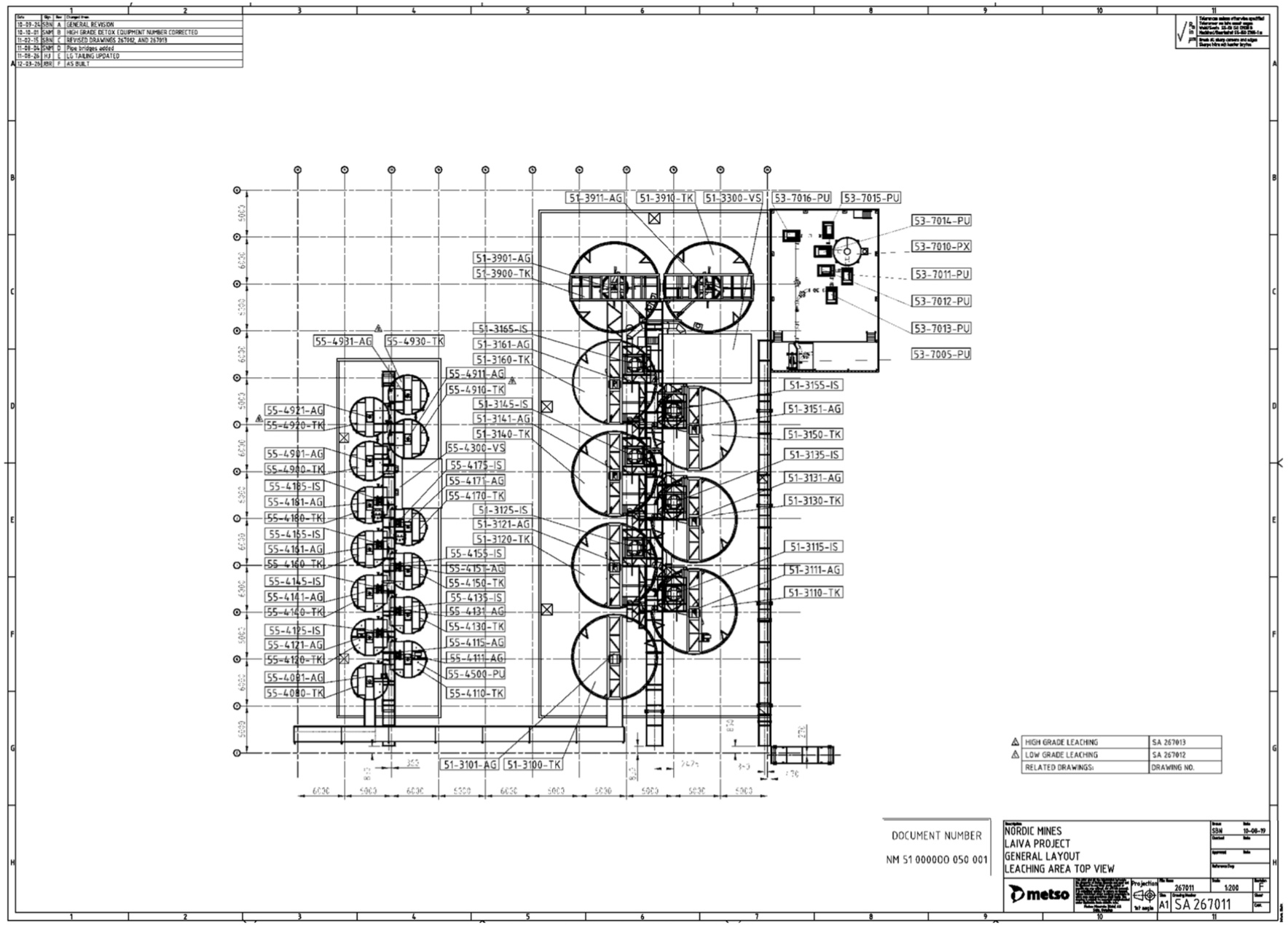

Figure 17.10 – Leach Area Plan View

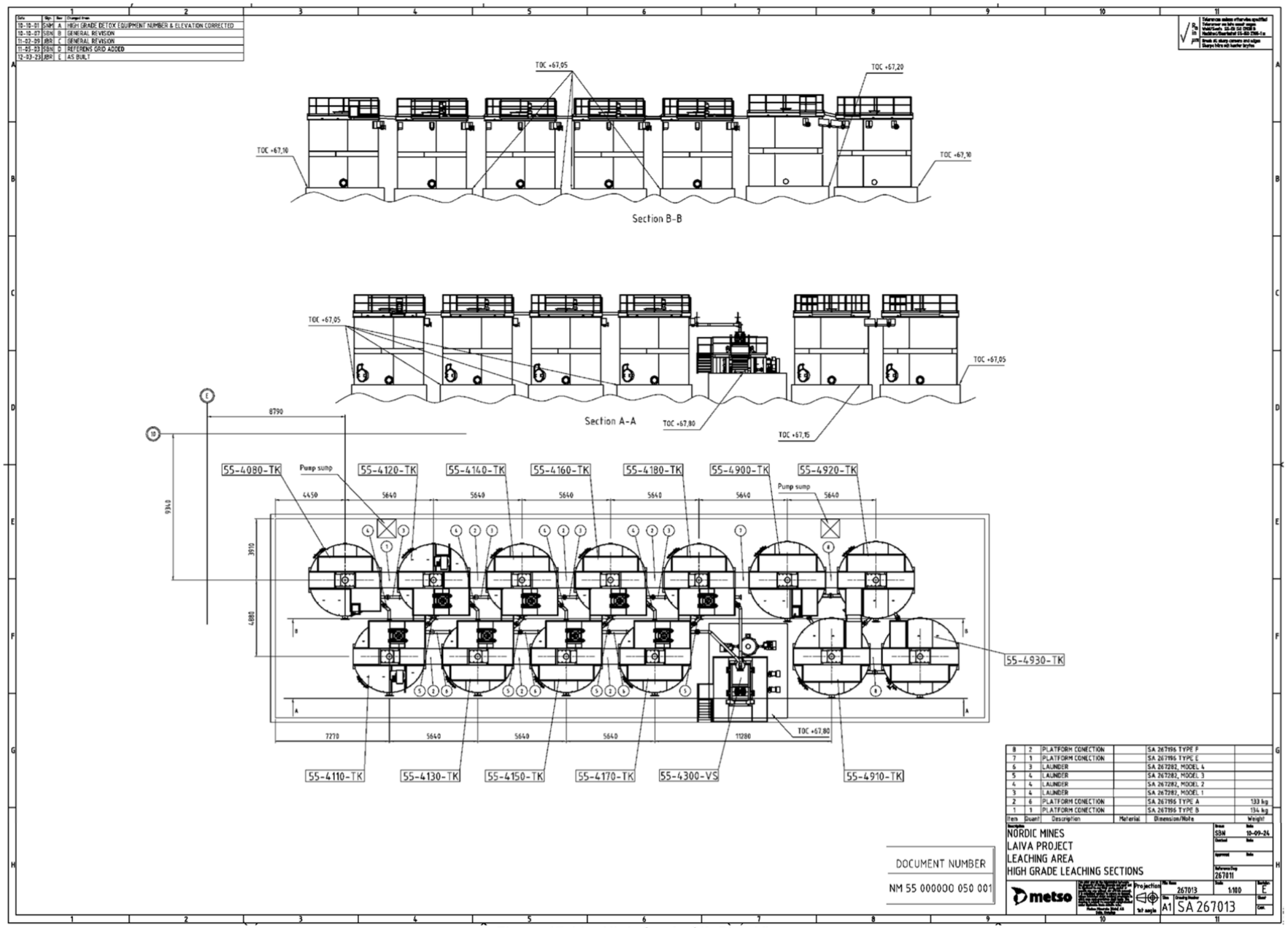

Figure 17.11 – High Grade CIL Plan View

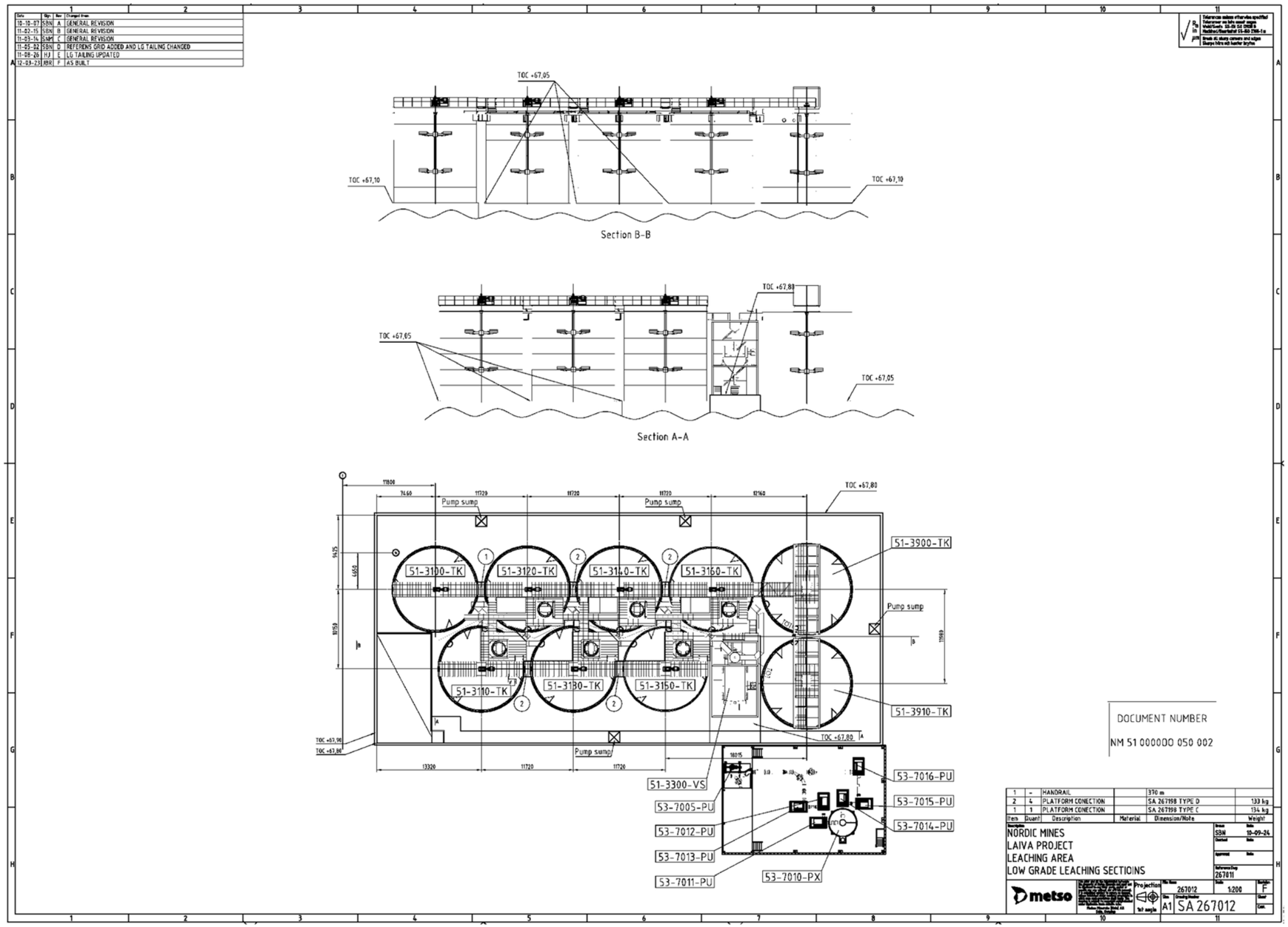

Figure 17.12 – Low Grade CIL Plan View

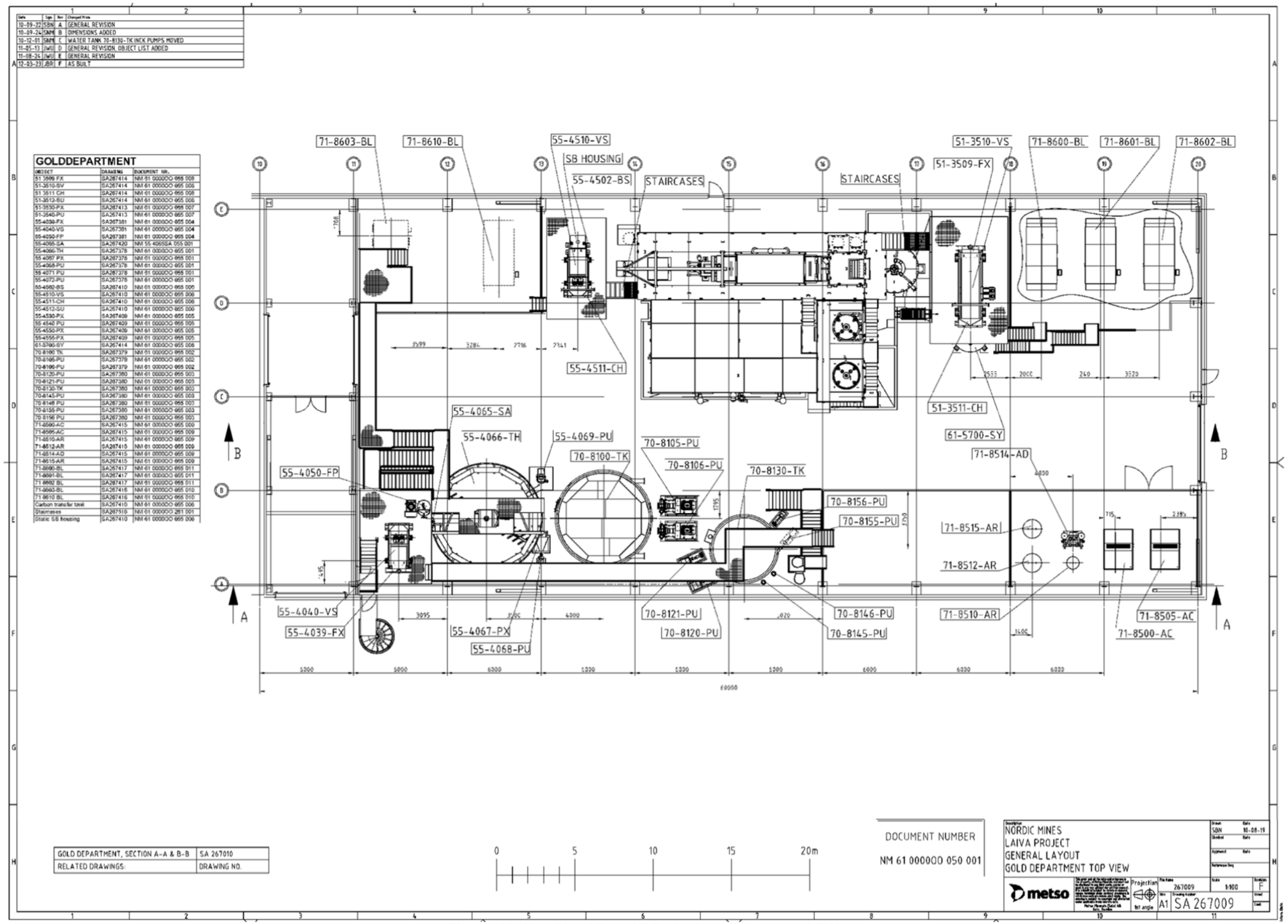

Figure 17.13 – Plan View ADR, Carbon Regen, and EW

A list of key equipment quantity and general specifications can be found in Table 17.1, on the following pages.

Table 17.1 – Equipment List

Equipment List
Nordic Mines Oy
Laiva Gold Project

Customer:
Project:

Project No: SAL00017 A
Date: **2010-04-29**
Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	Primary Crushing				
1	Feed Hopper	Crusher Feed Hopper			
	Dust suppression	Feed Hopper Dust Spray			
1	Feeder	Vibrating feeder	Metso VF661-2V	22.0	
1	Chute	Feeder Discharge Chute			
1	Chute	Feeder Underpan			
1	Rock Breaker		Liebein, Sandvik or equal. Chisel from Atlas Copco or similar	22.8	
1	Crusher	Jaw Crusher	Metso C160	250.0	
1	Chute	Crusher Discharge Chute			
1	Feeder	Vibrating feeder	Metso VMOU 30/16-5075	15.0	
1	Chute	Feeder Discharge Chute			
1	Conveyor	Belt Conveyor	Metso NB SC 1200 mm x 123 m	132.0	
1	Chute	Conveyor Discharge Chute			
1	Dust extraction	Dust extraction		18.5	
	Stockpile				
1	Tunnel	Corrugated Steel Tunnel	Lenght approx. 75 m, BxH=5,9x6,55, t=7mm		
1	Feeder	Vibrating feeder	Metso VMOU 25/10-5060/B	11.0	Speed control
1	Chute		Metso Standard		Mild steel with wearplate
1	Feeder	Vibrating feeder	Metso VMOU 25/10-5060/B	11.0	Speed control
1	Chute		Metso Standard		Mild steel with wearplate
1	Feeder	Vibrating feeder	Metso VMOU 25/10-5060/B	11.0	Speed control
1	Chute		Metso Standard		Mild steel with wearplate
1	Feeder	Vibrating feeder	Metso VMOU 25/10-5060/B	11.0	Speed control
1	Chute		Metso Standard		Mild steel with wearplate
1	Conveyor	Belt Conveyor	Metso NB SC 1200 mm x 113 m	60.0	
1	Chute	Conveyor Discharge Chute			
1	Dust extraction	Dust extraction		7.5	

metso

Equipment List

Nordic Mines Oy

Customer:

Laiva Gold Project

Project:

Project No: SAL00017 A

Date: **2010-04-29**

Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	Grinding				
1	Chute	Mill Feed Chute	Upper part		
1	Chute	Mill Feed Chute			Purchaser Supply
1	Mill			3000.0	Purchaser Supply
1	Screen	Trommel Screen			Purchaser Supply
1	Chute	Mill Discharge Chute	Underflow from Trommel screen		Mild steel with wearplate
1	Splitter		Pebble splitter		Mild steel with wearplate
1	Chute	Mill Feed Chute			Purchaser Supply
1	Mill			3000.0	Purchaser Supply
1	Chute	Mill Discharge Chute			Purchaser Supply
1	Chute	Mill Discharge Chute	Lower part		
1	Distribution box				Mild steel rubber lined with automatic dart valves
1	Pump Box		8 m3		Mild steel rubber lined
1	Pump Box		8 m3		Mild steel rubber lined
1	Pump		Metso, XR300 FNR-S C5HC	425.0	Speed control, ABB motor or equal

Table 17.1 – Equipment List – Continued

metso

Customer:
Project:

Equipment List
Nordic Mines Oy
Laiva Gold Project

Project No: SAL00017 A
Date: **2010-04-29**
Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
1	Pump		Metso, XR300 FNR-S C5HC	425.0	Speed control, ABB motor or equal
1	Screen	Two deck screen	Metso, MF 3,0 x 8,5 Multi Flow. Fitted with one deck.	110.0	
1	Pump Box		8m3		Mild steel rubber lined
1	Pump		Metso, VASA HD 7010-200	45.0	Speed control, WEG motor
1	Distribution box				Mild steel rubber lined with automatic dart valves
1	Centrifuge	Knelson centrifuge	Knelson CVD64 -040	150.0	
	Spiral concentrator	Spirals	Multotec or equal, - Spiral bank, containing 6 off triple start spiral assemblies.		
1	Pump Box				Mild steel rubber lined
1	Pump		Metso, VASA HD 507-150	110.0	Speed control, WEG motor
1	Flotation	Flot. RCS	Metso, Unit Cell RCS 30	110.0	WEG motor
1	Distribution box				Mild steel rubber lined with automatic dart valves
1	Pump Box		8 m3		Mild steel rubber lined
1	Pump Box		8 m3		Mild steel rubber lined
1	Pump		Metso, XR300 FNR-S C5HC	355.0	Speed control, ABB motor or equal
1	Pump		Metso, XR300 FNR-S C5HC	355.0	Speed control, ABB motor or equal
1	Cyclone battery		Multotec or equal, 14 unit array. 350 mm cyclones. 9 in operation. 3 stand-by. 2 blanked off position.		
1	Pump Box				
1	Pump		Metso, VASA HD 7010-200	160.0	Speed control, WEG motor
1	Pump		Metso, VASA HD 7010-200	160.0	Speed control, WEG motor
1	Cyclone battery		Multotec or equal, 34 unit array. 100mm diameter cyclones. 28 operating. 6 stand-by.		
1	Distribution box				Mild steel rubber lined
1	Pump Box				
1	Pump		Metso, VASA HD 7010-200	37.0	Speed control, WEG motor
1	Pump		Metso, VASA HD 7010-200	37.0	Speed control, WEG motor

Table 17.1 – Equipment List – Continued

metso

Equipment List
Nordic Mines Oy
Laiva Gold Project

Customer:
Project:

Project No: SAL00017 A
Date: **2010-04-29**
Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	Pebble Crushing				
1	Conveyor	Belt Conveyor	Metso NB SC 650 mm x 30 m	15.0	
1	Chute	Chute 2-way	Metso Standard		Mild steel with wearplate
1	Bin		Metso Standard, approx 30m3		Mild steel with wear rails
1	Feeder	Vibrating feeder	Metso VMO 20/8-4027	4.6	Speed control, 2x2.3 kW motors
1	Chute	Feeder Dischrge Chute			
1	Chute	Feed Chute	Metso Standard		Mild steel with wearplate
1	Crusher	Cone Crusher	Metso HP200 std medium, incl. Feed and disch chutes and Subframes, Basic automation	132.0	
1	Chute	Crusher Discharge Chute			
1	Dust Extraction	Dust Extraction		7.5	
1	Conveyor	Belt Conveyor	Metso NB SC 650 mm x 60 m	15.0	
1	Chute	Screen Feed Chute			
1	Screen	Double Deck Screen	Metso KS2 PP 5000/1600, incl.dust capsulation	15.0	
1	Chute	Screen Underpan			
1	Chute	Screen discharge Chute			
	Chute	Chute 2-way	Metso Standard		Mild steel with wearplate
1	Bin		Metso Standard, approx 18m3		Mild steel
1	Feeder	Vibrating feeder	Metso VMO 20/8-4027	4.6	Speed control, 2x2.3 kW motors
1	Chute	Feeder Discharge Chute			
1	Chute	Feed Chute	Metso Standard		Mild steel with wearplate
1	Crusher	Cone Crusher	Metso HP100 std medium, incl. Feed and disch chutes and Subframes, Basic automation	90.0	
1	Chute	Crusher Discharge Chute			
1	Dust Extraction	Dust Extraction		7.5	
1	Conveyor	Belt Conveyor	Metso NB SC 650 mm x 39 m	15.0	
1	Chute	Chute 2-way	Metso Standard	1.0	Mild steel with wearplate
1	Ball Feeder				Mild steel
1	Chute	Mill Feed Chute			Mild Steel - partly rubberlined
1	Mill		Metso, SRR Ball Mill 1,8 x 4,0m	160.0	WEG motor
1	Chute	Mill Discharge Chute			Mild Steel - partly rubberlined

metso

Equipment List

Nordic Mines Oy

Laiva Gold Project

Customer:

Project:

Project No: SAL00017 A

Date: **2010-04-29**

Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
1	Distribution box				Mild steel rubber lined with automatic dart valves
1	Pump Box		1 m3		Mild steel rubber lined
1	Pump Box		1 m3		Mild steel rubber lined
1	Pump		Metso, HR75 FNR-S C4HC	7.5	Speed control, WEG motor
1	Pump		Metso, HR75 FNR-S C4HC	7.5	Speed control, WEG motor
1	Cyclone battery		Multotec or equal, 165mm cyclone. 1 operating. 1 stand-by.		
1	Pump Box		1,5 m3		Mild steel
1	Pump		Metso, HR75 FNR-S C4	3.0	Speed control, WEG motor
1	Pump		Metso, HR75 FNR-S C4	3.0	Speed control, WEG motor
1	Pump	Spillage Pump	Metso, VS80 L150 O3A	22.0	Speed control, WEG motor
1	Pump	Spillage Pump	Metso, VS80 L150 O3A	22.0	Speed control, WEG motor

Table 17.1 – Equipment List – Continued

metso		Customer: Project:	**Equipment List** Nordic Mines Oy Laiva Gold Project	Project No: SAL00017 A Date: **2010-04-29** Rev: **2 A**	
Qty.	**Description**	**Equipment**	**Specification**	**Installed Power (kW)**	**Remarks**
	Low Grade Leach				
1	Screen	Single deck screen	Metso, KS H1P 5000/2500 Linear Motion Single Deck Dewatering Screen	30.0	2 x 15kW
1	Bin				Mild steel
1	Sampler		Metso Dual Sampler. Primary Rotary Sampler 350/400 and Secondary Sampler.	1.0	
1	Tank		907 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	75.0	CIL Low Grade, speed control
1	Tank		907 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	75.0	CIL Low Grade, speed control
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	11.0	CIL Low Grade
1	Tank		907 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	75.0	CIL Low Grade, speed control
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	11.0	CIL Low Grade
1	Tank		907 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	75.0	CIL Low Grade, speed control
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	11.0	CIL Low Grade
1	Tank		907 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	75.0	CIL Low Grade, speed control
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	11.0	CIL Low Grade
1	Tank		907 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	75.0	CIL Low Grade, speed control
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	11.0	CIL Low Grade
1	Tank		907 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	75.0	CIL Low Grade, speed control
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	11.0	CIL Low Grade
1	Screen	Single deck screen	Metso, KS H1P 5000/2500 Linear Motion Single Deck Dewatering Screen	30.0	2 x 15kW
1	Collection box				Mild steel
1	Bin				Mild steel
1	Sampler		Metso Dual Sampler. Primary Rotary Sampler 350/400 and Secondary Sampler.	1.0	

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Equipment List

Nordic Mines Oy

Laiva Gold Project

Customer:

Project:

Project No: SAL00017 A

Date: **2010-04-29**

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Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
1	Pump Box		1,5 m3		Mild steel
1	Pump		Metso, VASA HD 7010-200	45.0	Speed control, WEG motor
1	Pump		Metso, VASA HD 7010-200	45.0	Speed control, WEG motor
1	Pump		Metso, VSHM75 L120 W9	5.5	Fully recessed, WEG motor
1	Screen	Single deck screen	Metso, KS H1E 4000/1200 Linear Motion Single Deck Dewatering Screen	11.0	2 x 5,5 kW
1	Bin				Mild steel
1	Vacuum/Pressure tank		For 6 ton carbon - supply by Como		
1	Pump Box		1,5 m3		Mild steel, was 3495
1	Pump		Metso, HR150 FNR-S C4	18.5	was 3500, WEG motor
1	Pump	Spillage Pump	Metso, VS50 L150 O5	5.5	Speed control, WEG motor
1	Tank		1020 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	355.0	Detox Low Grade, speed control
1	Tank		1020 m3		Mild steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	355.0	Detox Low Grade, speed control

Table 17.1 – Equipment List – Continued

metso		Customer: Project:	**Equipment List** Nordic Mines Oy Laiva Gold Project	Project No: SAL00017 A Date: **2010-04-29** Rev: **2 A**	
Qty.	**Description**	**Equipment**	**Specification**	**Installed Power (kW)**	**Remarks**
	High Grade Leaching				
1	Screen	Single deck screen	Metso, KS H1P 2000/1000 Linear Motion Single Deck Dewatering Screen	7.5	2 x 3,74 kW
1	Bin				Mild steel
1	Sampler		Metso Rotary Sampler 80/100	1.0	
1	Thickener		Metso, SFL 30-7. Diameter 7m. Volume: approx. 100m3	2.6	2,2 + 0,37kW
1	Pump Box		1 m3		Mild steel
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump		Metso, HR50 FNR-S O4	5.5	Speed control, WEG motor
1	Pump		Metso, HR50 FNR-S O4	5.5	Speed control, WEG motor
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	15.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade

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Equipment List

Nordic Mines Oy

Laiva Gold Project

Customer:

Project:

Project No: SAL00017 A

Date: **2010-04-29**

Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Screen	Sieve Bend	Multotec or equal. Sieve Bend, 975 mm wide x 800 mm arc length at 45 deg. 700µm continuous slots.		
1	Screen	Single deck screen	Metso, KS H1P 2000/1000 Linear Motion Single Deck Dewatering Screen	7.5	2 x 3,74 kW
1	Collection box				Stainless steel
1	Bin				Stainless steel
1	Sampler		Metso Rotary Sampler 80/100	1.0	
1	Pump Box		1,5 m3		Stainless steel
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump		Metso, VSHM75 L120 W9	5.5	Fully recessed, WEG motor
1	Screen	Sieve Bend	Multotec or equal. Sieve Bend, 975 mm wide x 800 mm arc length at 45 deg. 700µm continuous slots.		
1	Screen	Single deck screen	Metso, KS H1P 2000/1000 Linear Motion Single Deck Dewatering Screen	7.5	2 x 3,74 kW
1	Bin				
1	Vacuum/Pressure tank		For 6 ton carbon - supply by Como		
1	Pump Box		1,5 m3		Stainless steel
1	Pump		Metso, HR75 LNR-S C4	4.0	WEG motor
1	Pump	Spillage Pump	Metso, VS50 L150 O5	5.5	WEG motor
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	55.0	Detox High Grade
1	Tank		67 m3		Stainless steel
1	Agitator		Ekato, Lightnin or equal	55.0	Detox High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	55.0	Detox High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	55.0	Detox High Grade

Table 17.1 – Equipment List – Continued

metso		Customer: Project:	**Equipment List** Nordic Mines Oy Laiva Gold Project	Project No: SAL00017 A Date: **2010-04-29** Rev: **2 A**	
Qty.	**Description**	**Equipment**	**Specification**	**Installed Power (kW)**	**Remarks**
	High Grade Leaching				
1	Screen	Single deck screen	Metso, KS H1P 2000/1000 Linear Motion Single Deck Dewatering Screen	7.5	2 x 3,74 kW
1	Bin				Mild steel
1	Sampler		Metso Rotary Sampler 80/100	1.0	
1	Thickener		Metso, SFL 30-7. Diameter 7m. Volume: approx. 100m3	2.6	2,2 + 0,37kW
1	Pump Box		1 m3		Mild steel
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump		Metso, HR50 FNR-S O4	5.5	Speed control, WEG motor
1	Pump		Metso, HR50 FNR-S O4	5.5	Speed control, WEG motor
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	15.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade

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Customer:

Project:

Equipment List

Nordic Mines Oy

Laiva Gold Project

Project No: SAL00017 A

Date: **2010-04-29**

Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Tank		67 m3		Stainless steel, Purchaser Supply
1	Agitator		Ekato, Lightnin or equal	11.0	CIL High Grade
1	Screen	Interstage screen	Kemix or equal (Pumping unit)	3.0	CIL High Grade
1	Screen	Sieve Bend	Multotec or equal. Sieve Bend, 975 mm wide x 800 mm arc length at 45 deg. 700µm continuous slots.		
1	Screen	Single deck screen	Metso, KS H1P 2000/1000 Linear Motion Single Deck Dewatering Screen	7.5	2 x 3,74 kW
1	Collection box				Stainless steel
1	Bin				Stainless steel
1	Sampler		Metso Rotary Sampler 80/100	1.0	
1	Pump Box		1,5 m3		Stainless steel
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump		Metso, VSHM75 L120 W9	5.5	Fully recessed, WEG motor

Table 17.1 – Equipment List – Continued

metso

Equipment List
Nordic Mines Oy
Laiva Gold Project

Customer:
Project:

Project No: SAL00017 A
Date: **2010-04-29**
Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	HCL Storage and Distribution				
1	Tank		5 m3		Glassfiber reinforced plastic or similar
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump	Spillage Pump		1.0	
	Elution, Carbon Regeneration, Electrowinning				
1	Elution System		6 ton pressure stripping plant	400.0	Supplied by Como
1	Carbon Regeneration System		300 kg/hr carbon regeneration kiln		Supplied by Como
1	Electrowinning System				Supplied by Como
1	Gold Room		barring furnace, sludge recovery, and anncillaries		Supplied by Como
1	Pump	Sump Pump	Metso, VS50 L120 O4	7.5	WEG motor
1	Screen		Metso, KS H1E 2000/600	4.5	
1	Scrubber		Regeneration kiln exhaust scrubber	7.0	
1	Pump	Sump Pump	Metso, VS50 L120 O4	7.5	WEG motor
1	Pump	Sump Pump	Metso, VS50 L120 O4	7.5	WEG motor
1	Scrubber		Electrowinning cell scrubber	7.0	
	Caustic Mixing and Distribution				
1	Tank	Heated tank	6 m3		Stainless steel
1	Heater			3.0	
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Pump	Spillage Pump		1.0	
	Cyanide Mixing and Distribution				
1	Tank		28 m3		Stainless steel
1	Agitator			4.0	
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Tank		30 m3		Stainless steel
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor

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Customer:

Project:

Equipment List

Nordic Mines Oy

Laiva Gold Project

Project No: SAL00017 A

Date: **2010-04-29**

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Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Pump	Spillage Pump		1.0	

Table 17.1 – Equipment List – Continued

metso

Equipment List
Nordic Mines Oy
Laiva Gold Project

Customer:
Project:

Project No: SAL00017 A
Date: **2010-04-29**
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Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	Propane Distribution				
1	Propane Distribution System		LPG Rental package	50.0	Purchaser Supply 40 + 10 kW
	CuSO4 Mixing and Distribution				
1	Tank		12 m3		Stainless steel
1	Agitator			2.5	
1	Tank		5 m3		Stainless steel
1	Pump		Metso, HR50 FNR-S O4	3.0	Speed control, WEG motor
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Pump	Spillage Pump		1.0	
	SO2 Distribution				
1	SO2 System		SO2 Rental Package	2.0	Purchaser Supply
	Lime Mixing and Distribution				
1	Lime Slaking System		45 m3 silo, screw feeders, mixing tank, agitator	2.0	
1	Tank		75 m3		
1	Agitator			5.0	
1	Pump		Metso, HR75 FNR-S C4	7.5	WEG motor
1	Pump		Metso, HR75 FNR-S C4	7.5	WEG motor
1	Pump	Spillage Pump		1.0	
	Lead Nitrate Distribution				
1	Bin				
1	Feeder	Screw feeder/conveyor		0.4	
	PAX Mixing and Distribution				
1	Tank		4 m3	0.4	Stainless steel
1	Agitator			0.4	
1	Tank		7 m3	0.4	Stainless steel
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Pump	Spillage Pump		1.0	

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Equipment List

Nordic Mines Oy

Customer:

Laiva Gold Project

Project:

Project No: SAL00017 A

Date: **2010-04-29**

Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	Dow 250 Mixing and Distribution				
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Tank		7 m3	0.4	Stainless steel
1	Pump		Metso, HR50 FNR-S O4	3.0	WEG motor
1	Pump	Spillage Pump		1.0	
	Oxygen Distribution				
1	Oxygen System		Oxygen Rental Package	2.0	Purchaser Supply

Table 17.1 – Equipment List – Continued

metso			**Equipment List** Nordic Mines Oy Laiva Gold Project	Project No: SAL00017 A Date: **2010-04-29** Rev: **2 A**	
		Customer:			
		Project:			
Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	Low Grade Tailings				
1	Pump	Spillage Pump	Metso, VS50 L150 O5	5.5	WEG motor
1	Pump Box		20 m3		
1	Pump		Metso, VASA HD507-150	132.0	Speed control, WEG motor
1	Pump		Metso, VASA HD507-150	132.0	Speed control, WEG motor
1	Pump		Metso, VASA HD507-150	132.0	Speed control, WEG motor
1	Pump		Metso, VASA HD507-150	132.0	Speed control, WEG motor
1	Pump		Metso, VASA HD507-150	132.0	Speed control, WEG motor
1	Pump		Metso, VASA HD507-150	132.0	Speed control, WEG motor
	High Grade Tailings				
1	Pump		Metso, HR75 FNR-S C4	11.0	Speed control, WEG motor
1	Pump		Metso, HR75 FNR-S C4	11.0	Speed control, WEG motor
1	Pump Box		1m3		
1	Pump		Metso, HR50 FNR-S O4	3.0	Sized according to feasibility study. WEG motor
1	Thickener				Purchaser Supply
1	Pump	Pump	Metso, MM150, FHC-S 03	30.0	Speed control. Sized according to feasibility study. WEG motor
1	Pump	Pump	Metso, MM150, FHC-S 03	30.0	Speed control. Sized according to feasibility study. WEG motor
1	Pump	Pump	Metso, MM150, FHC-S 03	11.0	Speed control. Sized according to feasibility study. WEG motor
1	Pump	Pump	Metso, MM150, FHC-S 03	11.0	Speed control. Sized according to feasibility study. WEG motor
1	Pump	Gland water Pump			Speed control. Sized according to feasibility study. WEG motor
1	Pump	Return water Pump	Metso, MM200 EHC-S C5	18.5	Speed control. Sized according to feasibility study. WEG motor
1	Pump	Return water Pump	Metso, MM200 EHC-S C5	18.5	Speed control. Sized according to feasibility study. WEG motor

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Customer:
Project:

Equipment List
Nordic Mines Oy
Laiva Gold Project

Project No: SAL00017 A
Date: **2010-04-29**
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Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
	Water Distribution				
1	Tank				
1	Pump		Metso, MM200 EHC-D C5	55.0	WEG motor
1	Pump		Metso, MM200 EHC-D C5	55.0	WEG motor
1	Pump		Metso, MM200 EHC-D C5	55.0	WEG motor
1	Pump		Metso, MM200 EHC-D C5	55.0	WEG motor
1	Pump		Metso, MM200 EHC-D C5	55.0	Sized according to feasibility study. WEG motor
1	Pump		Metso, MM200 EHC-D C5	55.0	Sized according to feasibility study. WEG motor
1	Pump			15.0	
1	Pump			15.0	
1	Pump			15.0	
1	Vacuum/Pressure tank				
1	Pump			15.0	
1	Pump			15.0	
1	Vacuum/Pressure tank				
1	Pump			15.0	
1	Pump			15.0	
1	Vacuum/Pressure tank				
	Air System				
1	Air Compressor		Atlas Copco, or equal, oil free and water cooled	75.0	Instrumental air and plant air,
1	Air Compressor		Atlas Copco, or equal, oil free and water cooled	75.0	Instrumental air and plant air,
1	Air Receiver		5 m3		
1	Air Drier		Drier		
1	Air Receiver		2 m3		
1	Air Drier		Drier		
1	Air Receiver		2 m3		
1	Air Blower		Aerzen, Kaeser or equal, - 6500 m3/h blower air	355.0	LG Detox aeration air,
1	Air Blower		Aerzen, Kaeser or equal, - 6500 m3/h blower air	355.0	LG Detox aeration air,
1	Air Blower		Aerzen, Kaeser or equal, - 6500 m3/h blower air	355.0	LG Leach aeration air,
1	Air Blower		Aerzen, Kaeser or equal, - 6500 m3/h blower air	355.0	LG Detox/Leach aeration air, standby
1	Air Blower		Aerzen, Kaeser or equal, - 4250 m3/h air blower	132.0	HG Detox aeration air,

metso

Equipment List

Nordic Mines Oy

Laiva Gold Project

Customer:

Project:

Project No: SAL00017 A

Date: **2010-04-29**

Rev: **2 A**

Qty.	Description	Equipment	Specification	Installed Power (kW)	Remarks
1	Air Blower		Aerzen, Kaeser or equal, - 4250 m3/h air blower	132.0	HG Detox aeration air, Standby
1	Air Blower		Aerzen, Kaeser or equal, - 900 m3/h blower air	45.0	Flotation air,
1	Air Blower		Aerzen, Kaeser or equal, - 900 m3/h blower air	45.0	Flotation air - stand by
	General				
	Piping and valves		Process, water, air piping, hand valves		Appendix No. 11
	Steel		Structural steel, platework, platforms, staircases.		Appendix No. 11
	Power Supply				
	Electrical and Automation System		Transformers, MCC, Control system, instrumentation.		Appendix No. 7
	CN Monitoring				
3	Cyanide monitoring system				For 6 streams

17.3 Plant Operations as Currently Configured

17.3.1 Run of Mine (ROM) Stockpile and Crusher Feed

Ore from active pits will be transported to a ROM stockpile area to be stacked in accordance with estimated grade and rock type to accommodate optimized blending of plant feed. Stockpiled ore in appropriate proportionate blend will be delivered to the primary feed hopper after segregation of oversize. BOYD recommends a Caterpillar 966K or equivalent to segregate oversize, perform blending, and supply the primary feed hopper.

17.3.2 Primary Crusher and Feed Arrangement

The primary feed hopper with a capacity of approximately 80 cubic meters, having grizzly bars spaced at nominally 1 m X 1 m, feeds directly onto a Metso VF661-2V vibrating feeder. This unit feeds directly into a Nordberg/Metso C160 jaw crusher.

The jaw crusher typically has a closed side setting of 200 mm. This unit has the capacity to deliver approximately 800 mtph at a closed side setting of 200 mm, which is more than ample for feed to the process plant.

Crusher discharge is via a chute to a Metso variable speed VMOU 25/10-5060/B vibrating feeder for belt loading.

17.3.3 Stacker Conveyor System

Primary crushed ore is delivered by a 1,200 mm wide inclined conveyor to a single point discharge under a covered semi-live stockpile with an estimated live capacity of 7,500 to 10,000 metric tonnes.

17.3.4 AG Mill Feed System

Beneath the stockpile are four discreet feed ports spaced at approximately 15 meters apart. Each feed port is equipped with a Metso VMOU 25/10-5060/B vibrating feeder independently controlled, which feeds onto a single belt in the tunnel beneath the stockpile for delivery to the primary autogenous grinding mill.

17.3.5 Primary Autogenous Grinding Mill

A mechanical scale on the mill feed conveyor controls the feeders under the stockpile to the Outotec Autogenous Mill (AG). The installed power is 3,000 Kw. The inside diameter of the AG-Mill is 6.5 m and the inside length is 8.5 m.

Water is added to the AG-Mill in order to reach 65 weight-% solids in the finest (-8 mm) discharge stream.

A double drum trommel screen after the AG -Mill separates the discharge into three sized products:

1. Fines -8 mm
2. Chips 8 - 50 mm
3. Pebbles 50 - 150 mm

The fines material, together with the discharge from the Pebble Mill, report to the classification sieve or can be diverted directly to the Pebble Mill feed if required.

The two largest product sizes of the AG-Mill can circulate back to the AG-Mill for regrind if needed through the Pebble Crusher System. Usually the coarsest material (pebbles) from the AG are used as grinding media in the Pebble Mill when pebbles are needed, and the power intake of the Pebble Mill enables it. The excess pebbles and chips (8-150 mm) are transported to the pebble crush system as required by conveyor belt.

17.3.6 Pebble Crusher System

To increase the Mill throughput, two cone crushers in series re-crush the excess critical size (8-150 mm) from the AG-Mill discharge material. The feed to the cone crushers is designed to be 87 tph (35% of AG Mill feed). The critical size is re-crushed down to -20 mm before being returned back to primary milling.

The conveyor belt under the primary crusher transports the crushed product to a double deck screen (40 and 9 mm), on top of the secondary cone crusher. The screen separates three products:

1. Fines –9 mm
2. Mid-size 9 - 40 mm
3. Oversize +40 mm

Mid-size product is crushed in the secondary cone crusher, Fines (-9 mm) and oversize (+40 mm) are bypassed.

If the primary cone crusher cannot be used, the secondary cone crusher still can operate if the oversized material, which is too large for the secondary crusher to handle, is screened off.

17.3.7 Secondary Grinding Pebble Mill

Underflows from the primary cyclone cluster are ground in closed circuit in the Pebble Mill. The grinding media are 50-150 mm size pebbles from the AG-Mill. The power draw from the Pebble Mill is controlled by the pebble material flow from the AG-Mill. The installed power is 3,000 kW. The inside diameter of the Pebble Mill is 6.5 m and the inside length is 8.5 m.

17.3.8 Classification

Fines discharged from the AG-mill and Pebble Mill are pumped to a two-deck classifying sieve with 3 mm opening on the upper deck and no screen included on the bottom deck. Material +3 mm reports back to the AG-Mill and -1.5 mm material is pumped to the primary classifying cyclone and on to Flash Flotation and gravity separator circuits.

A cyclone battery, total of 12 units, is used to classify particles ready for low grade leaching. The cyclone underflow with the coarse particles reports back to the Pebble Mill for regrind. The cyclone overflow is the feed to the low grade leaching circuit. The targeted particle size of cyclone overflow currently is P_{80} ~100 µm.

17.3.9 Flash Flotation and Gravity Circuits

From the sieve, the undersized flows by gravity to a sump and is pumped to the flash flotation circuit. The Flash Flotation unit (Metso Unit Cell RCS 30) produces high grade concentrate of gold-bearing sulphide minerals including arsenopyrite and pyrite. Copper sulphate (CuSO4) is added to activate arsenopyrite and PAX (Potassium Amyl Xanthate) and Aerophine 3418A are used as collectors. The produced concentrate is directed to the concentrate regrind circuit for onward direction to the high grade CIL circuit.

The tailings from flash flotation are circulated back to the primary cyclone cluster for re-classification. Underflow from the primary cyclone cluster is returned to the Pebble Mill for further grinding.

Overflow from the primary cyclone bank is forwarded to gravity concentration. The gravity concentration circuit feed is split by a cyclone circuit with cyclone underflow fed to the spirals (6 ea Multotec triple starts) circuit and the cyclone overflow fed to the Knelson concentrator (Knelson CVD64-D40).

The target split particle size for the gravity concentration circuit is 105 µm. The particle size fed to the spiral circuit is approximately P_{80} = 68 µm. The particle sizefed to the Knelson concentrator is approximately P_{80} = 150 µm.

The two gravity concentrates are directed to the concentrate regrind circuit for additional grinding before subsequent high grade CIL.

The target particle size for the high grade concentrate regrind circuit is P_{80} = 45 µm.

17.3.10 Leaching

17.3.10.1 Overview

The cyanide leach facility incorporates standard Carbon in Leach methodology. The average nominal annual mill throughput of 197 mtph was utilized for mass balance purposes. It has been assumed that the plant will operate 8,000 hours per year (assuming 90% overall availability). The feed to the cyanidation plant is comprised of two streams, including concentrate from the gravity circuits and flash flotation to high grade CIL and the tailings from each to low grade CIL.

Design of the low grade leach circuit, carbon adsorption, elution, and gold recovery circuit, with respect to retention time and gold production, is based on a nominal throughput of 187 mtph containing 0.434 g /t Au, 0.004 % As, and 0.022% S.

Design of the high grade leach circuit capacity, carbon adsorption, elution, and gold recovery circuit, as well as retention time, gold production, etc., is based on a nominal throughput of 10 tonnes per hour containing 51.4 g/t Au, 22.7 g/t Ag, 1.2 % As, and 5.9% S.

The upgrade concentrate regrind circuit will have the capacity to produce up to 13 mtph at P_{80} = 45 µm product. The increased high grade grinding capacity will ensurethat all gravity and flotation concentrates will be reground and report to the high grade leach circuit. During previous operations, the undersized regrind mill resulted in a portionof the concentrates reporting to the low-grade leaching circuit, thereby substantially lowering overall gold recovery.

The assumptions made for the initial plant modification upon restart are based on the results of a pilot scale test made by GTK in Finland; “Pilot Plant on Laivakangas Ores” dated 12 Feb 2008.

Pumps, piping, screens, cyclones, thickener, and other items which required sizing and designing depend on actual pulp flow variations, and are based on an instantaneous flow rate corresponding to an annual throughput of nominal 2 M mtpy. Reagent dosing systems and the cyanide detoxification system are also designed for higher pulp flow rate.

Mass flows for the leach circuit with increased high grade circuit throughput will be completed as the plant unit operations are optimized following restart.

A simplified block diagram of the process flow is shown in Figure 17.14 below.

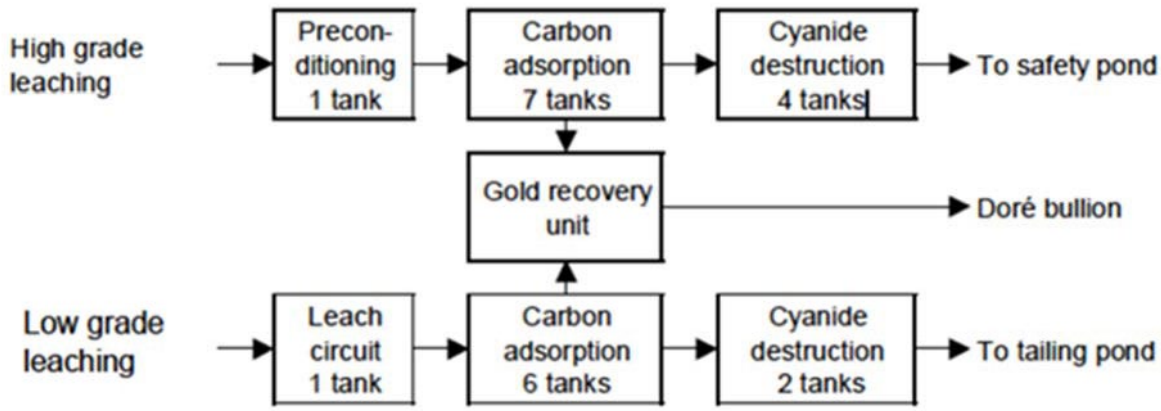

Figure 17.14 – Block Diagram for Process Flow

17.3.10.2 CIL Circuits

Leaching of gold and silver from previous processing carried out by means of CIL (Carbon-In-Leach). The CIL process is based on dissolution of gold and silver in an aerated pulp containing cyanide, followed by adsorption of the dissolved Au and Ag values on coarse granules of activated carbon.

The pulp flow is divided into two streams – high grade leach feed consists of concentrates from flash flotation and gravity concentration. The low grade leach feed consists of cyclone overflow from the grinding circuit, as well as flash flotation and gravity concentration tails. The principal reasons for separating the pulp into separate streams is to direct tailings high in Arsenic (arsenopyrite – higher gold content and Sulphur) into special lined cells and to maximize Au recovery in the high-grade circuit. Tailings from the low-µgrade circuit which are relatively low in reactive sulfides are thendeposited into an unlined TSF as free-draining paste. In addition, due to design

operating parameters, the low grade CIL circuit is less effective in recovery of Au from arsenopyrite.

Since the high-grade material has shown significant CN consumption characteristics during leach tests, it is likely that installation of a CN recovery system will be cost beneficial in the future, though neither the cost nor the benefits are included in this Technical Report analysis. A pre-conditioning tank will however be included as a first stage where lead nitrate and oxygen is added. In the low grade circuit, the first tank is designed as a pure leach tank with no carbon included. The following six tanks are traditional CIL tanks.

The feed will be first pumped to the trash screens, one for each stream, at an approximate pulp density of 1.4 mt/ m3. The trash screen will remove any coarse material that may settle in the tanks or block the interstage screens in the CIL tanks.

Two vibrating horizontal trash screens, provided with water sprays, as in the current circuit are mounted in parallel over the leach tank and have polyurethane decks with an aperture of 600 µm. The trash screens oversize is conducted from the screen directly to a bin for disposal. The trash screen undersize moves through the leach feed automatic sampler to the first tank in the leaching circuit.

The leach circuit is comprised of seven stirred tanks in series (eight tanks in the case of the high-grade leach circuit). The pulp moves through the leach by gravity via down-comers and interstage launders. Bypass provisions are installed for each individual stage.

Cyanide is added to the first two CIL stages as a 20% solution (w/v). Air is sparged into each leach tank to maintain the dissolved oxygen content above 6 mg/l. Lower oxygen content, if experienced, will slow the overall leaching rate, resulting in solid gold losses. Pure oxygen is used on the high-grade circuit since oxygen uptake rate tests made by GTK indicate that the addition of air is not sufficient for oxygen requirements for effective dissolution of gold.

Slaked lime is added as 20% slurry (w/v) to the circuit from a main ring in order to maintain the pH at 10-10.5. The lime addition rate is strictly controlled by online measurement of pH by means of electrodes to avoid generation of hydrogen cyanide gas (HCN). The acid constant for hydrogen cyanide, the pKa-value, is about 9.2. At pH 9.2, cyanide in the solution occurs as cyanide ions and hydrogen cyanide at a ratio of 50-50. At pH <9.2, the acid form (HCN) dominates and is released from the solution to

the atmosphere as a poisonous gas. A strict control of the pH-value ensures that most of the added cyanide will remain in solution.

The leach and CIL tanks are located outdoors. Leach tanks are contained within containment basins designed to hold a minimum one tank volume to prevent any leakage to the environment.

The adsorption is carried out as a multistage counter current process to obtain high gold and silver solution loading at high recoveries. The coarse granules of carbon are retained by interstage screens – which are submersed in the pulp.

The stainless-steel wedge-wire screens have an aperture of about 0.7 mm which allows the pulp to pass through but retains the carbon. Carbon is harvested and transferred upstream through the adsorption circuit, counter-current to the slurry flow by airlifts. The airlifts are operated in a sequence mode to minimize the effect of back mixing on the adsorption profile and to maintain an even distribution of the total carbon inventory over the entire circuit.

Re-activated and new carbon is added to the last CIL tank in order to restore and maintain the total required level of carbon.

Low-Grade Circuit

Loaded carbon is harvested from the first CIL tank by gravity flow and pumped to the loaded carbon screen. The loaded carbon is washed on the screen by water sprays and the carbon flows by gravity to the elution column. The loaded carbon screen U/F is returned to the first CIL tank. A level probe switches off the loaded carbon pump when the column is fully loaded as well as the loaded carbon screen after a short delay.

High-Grade Circuit

The loaded carbon flows by gravity along with slurry to the loaded carbon screen. The slurry (undersize flow) is pumped back to the CIL-tank and the loaded carbon (oversize flow) is washed into a carbon flush tank. The tank is pressurized by the main water system, and the carbon is hydraulically transferred to the elution plant.

The barren slurry from the last adsorption tank is screened on a horizontal vibrating carbon safety screen to detect and collect any carbon losses. The polyurethane deck has an aperture of 600 µm. The screen oversize flows by gravity directly into a bin for disposal. The screen undersize is discharged through a final residue sampler to the CIL tailings sump to be pumped to the cyanide detoxification circuit.

To optimize the usage of cyanide and reduce the costs, an automatic control system is currently installed. The automated control system was not utilized during previous operations, thereby contributing to less efficient operation and higher costs.

The cyanide control system utilizes semi-continuous potentiometric cyanide analysis which makes it possible to analyze the free cyanide levels in the leach solutions without any interference from metal-complexed cyanide, which is not available for cyanidation. The continuous data for free cyanide is used to automatically control the dosage of NaCN solution.

17.3.10.3 Cyanide Detoxification

Tailings from each CIL circuit are treated separately. Each stream reports to its own Detox Plant and subsequently to its own tailings storage facility.

Cyanide ions in the discharged pulp are decomposed by the INCO SO2/air process. The process is well proven and is able to achieve the targeted level of <10 mg CN total per liter solution in the discharge.

The cyanide detoxification is accomplished in two reactors in series, providing three hours residence time on a continuous basis. The reactors are located outdoors to mitigate environmental issues.

17.3.10.4 Loaded Carbon Processing

Pressure Zadra Elution

Loaded carbon is transferred to the acid column over a period of time as required. The carbon is allowed to drain off excess water until the column is filled with loaded carbon. Once full, the drain valve is shut and a mixture of raw water and hydrochloric acid (to a concentration of 3% HCl) is pumped up through the column before discharging to the Low Grade Detox tank.

When the appropriate volume of diluted acid is pumped through the column (normally one bed volume ~ 7m3), the carbon is then flushed with four bed volumes (28m3) of raw water to remove residual acid and to increase pH. The rinse solution is also sent to the Detox Tank.

Once the acid washing and rinsing is completed, the column is then pressurized and the carbon moved to the elution column. The elution column is then drained of excess water. Finally, the elution column is pressurized, put in a closed loop with the eluate tank, heater, a heat exchanger, and electrowinning cells.

Gold is stripped from carbon by high-temperature, low-cyanide concentration and high pH eluate solution and eluted into eluate solution that circulates through the carbon bed. Loaded gold solution then circulates to electrowinning cells where gold is transferred onto steel cathodes and precipitated into the bottom of the cell.

A caustic/cyanide solution is pumped from the eluate tank, heated up to around 90ºC by the reclaim heat exchanger (recovering heat from the solution exiting the elution column by a plate and frame heat exchanger) and then the solution is heated up to 140ºC in a direct fired heater. The eluted solution is kept under pressure to keep it in a liquid state, hence the term Pressure Zadra.

The hot pressurized solution is then pumped through the elution column via screens at the base. This change in temperature and pH of the eluate solution causes the gold and silver to re-enter solution as a cyanide complex. This is an equilibrium reaction. At low temperatures and cyanide concentrations gold will migrate from solution onto the carbon, but at elevated temperatures and cyanide concentration the process is reversed. The solution then exits the column at the top, again via tube screens and flows through the other side of the reclaim heat exchanger mentioned earlier and then into a flash pot to allow any entrained steam to vent to the atmosphere.

The gold and silver bearing solution then flows to the electrowinning cells (two operating in parallel), where the precious metals are plated onto cathodes. The solution is then transferred back to the eluate tank, thus completing the circuit.

Once the requisite time has elapsed (design 12 hours, but will depend upon metal loading on the carbon, i.e., gold, silver, copper, etc.), the solution exiting the column should be quite low in metal content (approximately 5 ppm or less) and the circuit will then be switched off. The elution column is de-pressurized with fresh water and the now eluted, barren carbon is transferred to the regeneration kiln feed hopper to be regenerated and used again in leaching. A dewatering screen is mounted above the feed hopper to remove excess water.

Carbon Regeneration

Once the feed hopper is full, the kiln can be started. Carbon is fed into the kiln at the prescribed rate (250 kg/hr) via a screen feeder which will also act to moderately dewater the carbon further. Water is needed in the kiln to create a reducing atmosphere, hence if the carbon has been allowed to stand for some time, it should be re-wetted before being fed to the kiln.

Carbon will be heated to 750ºC in the regeneration kiln, which is a horizontal, rotary, indirectly fired furnace. This removes volatiles (diesel, oils, grease, flotation reagents, etc.) and regenerates the surfaces of the carbon to return the carbon to near new adsorption capability. It is very important that the carbon is washed thoroughly with clean water prior to regeneration.

Any soluble material can permanently cement onto the carbon if it is not adequately washed. This is true for Silica which under certain conditions will dissolve into the eluate, and if it is not removed from the carbon, will revert to silica completely destroying the carbon.

Once regenerated, the carbon emits from the end of the kiln, and passes over a carbon sizing screen into the quench vessel where it is cooled. Once the batch is complete, the quench vessel is closed and pressurized, transferring the carbon hydraulically to the appropriate tank. Water from the above mentioned dewatering screens reports to the carbon safety screen.

17.3.10.5 Electrowinning and Smelting

Gold is electro-won onto steel wool cathodes in the electrowinning cells and further precipitated onto the bottom of the cell by direct current. Gold from the screens and precipitated as sludge is removed and placed into trays. The gold sludge material is then oxidized in the calcine oven.

Once calcination is complete, the iron is oxidized to iron oxides, which are taken up in the slag during the smelting process. This occurs when calcined gold/silver product is mixed with fluxes borax, silica, etc., and are placed in the barring furnace and heated to ~1200ºC.

When the contents of the barring furnace are fully molten the contents should have separated into two phases: metal, and slag. The molten contents are then poured into molds, the heavier metal (dore) remains in the bottom of the molds and the slag flows over the lip.

Finally, when the doré is solidified, the molds are emptied and gold/silver bars are easily parted from remaining slag by chipping. Doré bars are then drill-sampled, stamped, and placed in the safe for delivery to the buyer. Residual slag should be crushed and returned to the high grade regrind circuit for reprocessing.

17.4 Proposed Modifications to Existing Process Plant

17.4.1 Crushing

OTSO has rented primary and secondary diesel-powered crushing units to provide finer feed (nominal 25 mm) to the AG-Mill, blended with 200 mm feed produced by the existing C160 jaw crusher. Though the ratio may vary, roughly 20% of the feed will be 200 mm particles and 80% will be 25 mm particles. The coarse fraction is intended to serve as grinding media in the AG-Mill.

While the rented crusher units are included in this Feasibility analysis, BOYD strongly recommends replacing these rented units with a permanent electrically powered secondary cone crusher coupled with appropriate split of feed diversion from the existing primary crusher (Norberg/Metso C160), which has more than ample capacity at any closed side setting to supply total feed to the process plant.

That said, the rented diesel-powered units engaged by Otso include a track-mounted jaw crusher (Metso LOKOTRAK LT 106) and a track mounted cone crusher (METSO LOKOTRACK LT 2OOHP) to bridge the essential requirement for blended particle feed size to the AG-Mill, which is critical to achieve maximum plant throughput. A schematic of the parallel crusher line, including the portable jaw and cone crushers is shown in Figure 17.15, following this page.

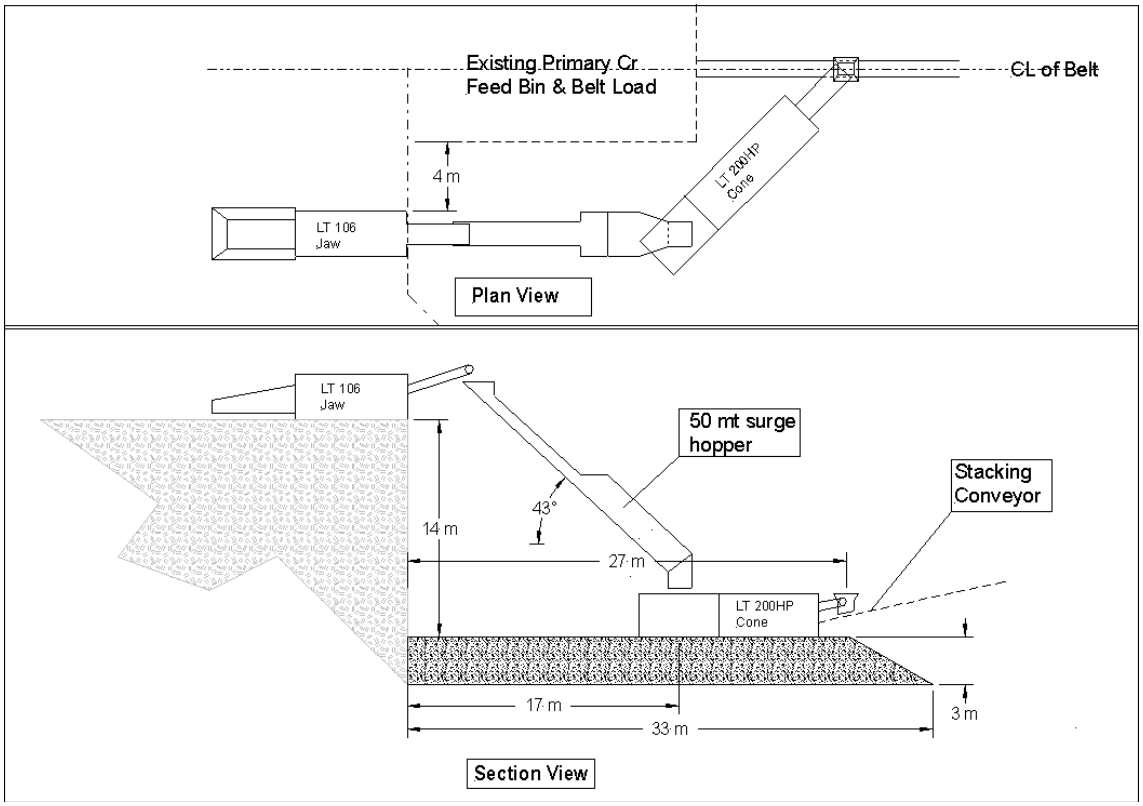

Figure 17.15 – Schematic of Portable Crusher Arrangement

17.4.2 Flash Flotation, Gravity, and Concentrate Regrind

BOYD recommends several minor modifications to the flash flotation and gravity concentration circuit, which include only minor piping changes and installation of one cyclone cluster for particle size separation to narrow the range of particle size processed by the Knelson centrifugal concentrator and the spirals. Also, OTSO is completing installation of a previously existing, but uninstalled Koppers KW-200 tower mill for regrind of high grade flash float and gravity concentrates prior to high grade leaching. A schematic of the revised pre-leach flow sheet is shown in Figure 17.16, following this page.

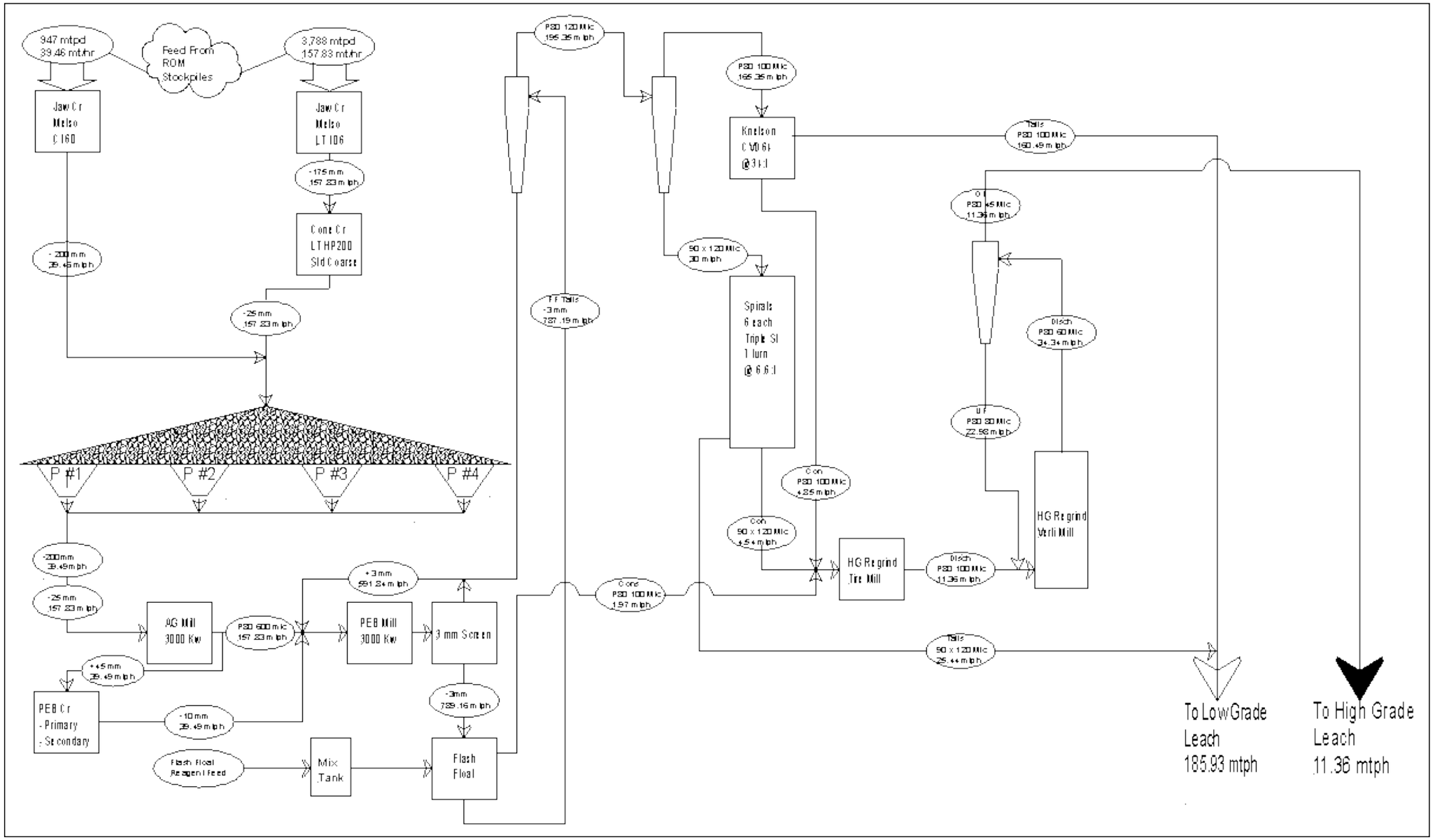

Figure 17.16 – Revisions to Pre-Leach Flow Sheet

The objective of these minor modifications is to:

1. Expose all mill feed to flash flotation and gravity concentration.
2. Divide flows to the Knelson concentrator and spirals to narrow the particle size seen by each and to apply the Knelson centrifugal concentrator to the finer fraction and spirals to the coarser fraction.
3. Maximize concentrate feed to the high grade leach circuit by adjusting the mass pull from flash flotation and the gravity concentrators for regrind in the tower mill.

In BOYD's opinion, the foregoing minor modifications should result in maximization of overall recovery and can easily be adjusted to hone in on optimum results and changing feed conditions.

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18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

Project Infrastructure in this instance can generally be construed to include site access, power supply, water supply, waste dumps, TSF facilities, and site drainage. As an existing project including mine, mill, and infrastructure which was in production as recently as early 2018, all infrastructure is in place and remains in a near-new state.

18.2 Logistics and Access

The project site is located approximately 25 km by road from the port city of Raahe, Finland. Much of the distance from Raahe is on paved two-lane road suitable for heavy transport of equipment and materials. The last 10 km is improved all-weather gravel road, also suitable for heavy transport of equipment and materials. Logging is prevalent in the area and log trucks frequently transit the roads used for site access to the Laiva Project.

Raahe has a small port which can receive bulk freight and heavy equipment. In the alternative, the City of Oulu, located approximately 90 km north of Raahe along the major north south highway along the west coast of Finland has a much more substantial port. Both ports generally remain open throughout the winter months.

Raahe, a town of approximately 25,000 has vendors for many routine operating supplies and services, because located nearby is a large steel mill as well as several other heavy industrial installations. Oulu, a city of approximately 200,000 is the largest city and industrial hub in the north of Finland. All supplies, services, and equipment essential to mining and processing are available in Oulu.

18.3 Electrical Power

Ample grid power is available to the site. An overhead line supplies 3-phase power at 120 kv ,50 hz to a 20,000 kva substation located on site. Grid service is generally highly reliable notwithstanding moderately harsh winter weather in the north of Finland. Power is moderately expensive at approximately $0.10 per kwh inclusive of demand and all other charges currently. No significant on-site standby generation capacity is present at Laiva.

18.4 Water Supply

Water is plentiful in northern Finland. Process makeup water not otherwise available from the freshwater collection and treatment system from tailings water recycle, pit pumping, and site run-off is expected to be sufficient to provide process water. Typical past consumption has been about 35,000 cu meters (9.25 million us gallons) per month. Under current operating conditions, purchased makeup water is not expected to be required except in periods of exceptional drought (rare in this region).

Treated potable water is supplied by the local municipality via pipeline to the Laiva site. Cost for treated potable municipal water is relatively high at €1.02 per cubic meter delivered or US$0.005 per US gallon. Only site domestic water consumption at nominally 1,500 cu m/year is considered as a component of infrastructure cost in this Technical Report.

18.5 Waste Dumps

Mining waste is divided into potentially acid generating waste (PAG) and benign, non-acid generating waste (NAG). The waste rock is differentiated based on sulfide content, with waste containing higher levels of sulfide material reporting to the PAG dump area and waste with lower content reporting to the NAG waste storage area.

The PAG waste must be deposited on impermeable underliner of HDPE or locally available processed peat. The underliner is placed on a prepared surface of crushed gravel topped with sand and protected from above by a layer of sand overlain by crushed gravel. Drainage water from the PAG waste is collected and pumped to a collection pond where it is mixed with other site drainage before being pumped to the sea.

Cost for development of an expected additional 125,000 sq meters of prepared PAG waste disposal area at US $6.50/sq meter all-in cost is expensed and included in “mine services” cost in Section 21.0.

No special preparation is required for NAG waste stockpiles. See locations of Waste storage in Figure 18.1, following this page.

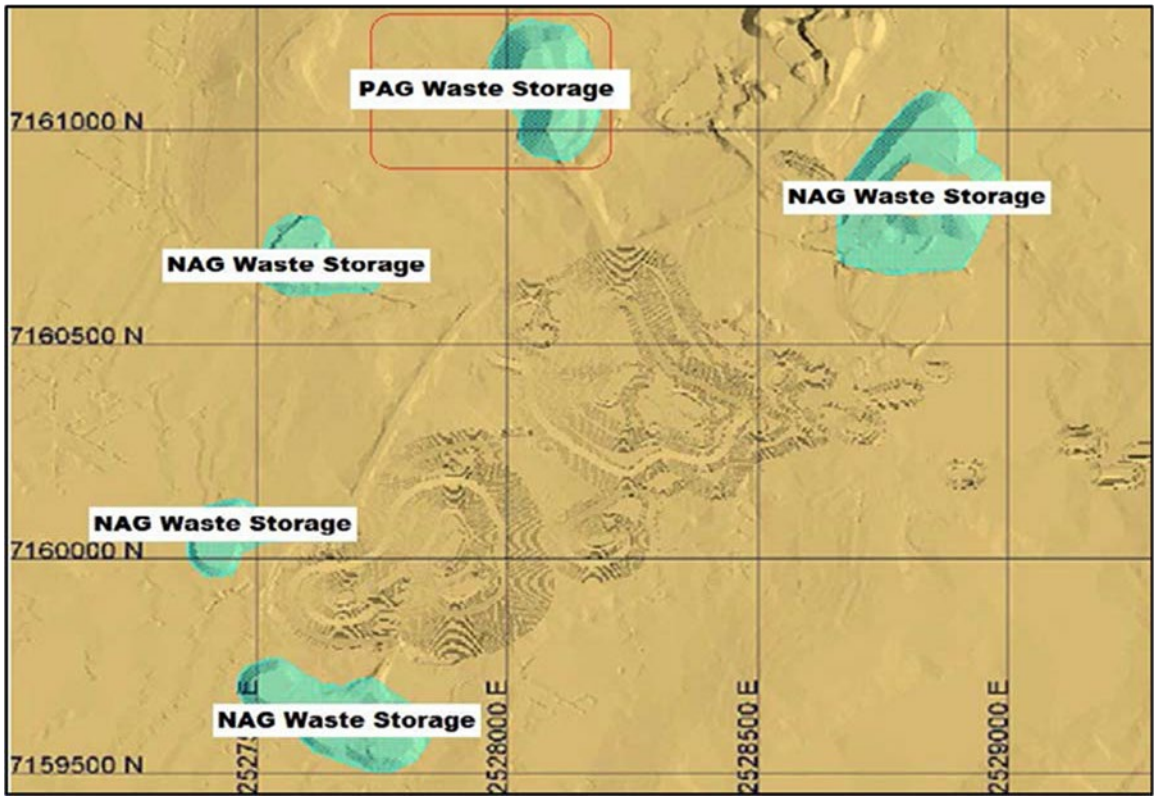

Figure 18.1 – Waste Storage Areas by Type

18.6 Tailings Storage Facilities

Similar to the mine waste dumps, the tailings storage facilities (TSF) are divided between high sulfide and low sulfide TSFs. High sulfide tailings from the High Grade leach circuit are delivered to HDPE lined cells. Tailings from the High Grade CIL circuit also contain elevated levels of arsenic resulting from the concentrated presence of arsenopyrite.

18.6.1 High Grade TSF

Following exit from the High Grade CIL circuit, the tailings pass through the Inco air/SO2 for cyanide destruction, before being pumped to the isolation cell. Each cell has a nominal capacity of 117,000 dry metric tons of High Grade tailings. A second cell was completed shortly before shutdown of operations by the previous operator. The remaining life of HG tailings cell #2 (already constructed) is estimated at approximately 30 months at planned operating levels.

The project had previously permitted a total of five HG tailings cells. Accordingly, a third HG tailings cell will need to be constructed for use beginning mid-year 2023, followed by a fourth cell estimated for use beginning early in 2024, and finally a fifth cell for use beginning mid-year 2025. Based on the current minable reserves. Cost for this work is capitalized as Sustaining Capital. Production beyond that will require amendment of the permit for HG tailings deposition.

18.6.2 Low Grade TSF

The relatively benign Low Grade tailings which account for the bulk of the tailings disposal (240 tph LG vs 10 tph HG) are pumped via a 7.39 km pipeline to a dewatering plant before being deposited as paste at approximately 55% solids. The elevation gain from the mill to the paste plant is approximately 60 meters. Low Grade tailings are pumped at approximately 35% solids through a 300 mm id steel pipeline. Pumping over the entire distance is accomplished by three Metso VASA HD507-150 pumps arranged in series for a total input of 396 kW. The pumping system has 100% on-line spare redundancy, which can be instantly activated with automatic valves via a “Y” arrangement, see Figure 18.2 below.

Figure 18.2 – Paste Tailings System

The steel pipeline is a less than ideal system due to abrasion wear and higher C factor than HDPE. In addition, the interior surface of the steel pipeline has accumulated a build-up of scale thought to have resulted from sulfur reducing bacteria (Paterson & Cooke, October 2014). FTR, a previous operator, planned to remove much of the accumulated scale by scraping the inside of the pipeline with a de-scaling pig service provided by an outside contractor prior to restart of operations. However, that work was never completed. OTSO plans to restart operations without descaling to empirically determine operating parameters absent pipeline descaling to determine best future alternatives vis the existing pipeline.

BOYD does not disagree with this strategy, although believes it constitutes some potential risk to achievement of performance parameters modelled.

The dewatering system located near the paste repository consists of a 35 m diameter Outotec High Compression Supa Flow conventional thickener with appropriate flocculent feed system and underflow pump to deliver paste to the repository. While thickener underflow was calculated by Outotec to be approximately 70% solids by weight, the centrifugal style underflow pump proved, not surprisingly, to be incapable of handling such a high percentage of solids. This was somewhat fortuitous, as the flocculent used, as well as the floc feed system were only able to deliver approximately 55% solids to the underflow for paste deposition.

The low grade paste repository was initially based on the notion that with 70% solids, the paste would not require containment. At 55% solids actually achieved, containment proved to be required. While no engineering designs are known to remain from the previous operator, if they ever existed, the previous operator sought to prevent spread of the thinner than expected paste by erecting a permeable rock berm to contain the spread of the paste. A short distance behind the permeable rock berm, the previous operator erected an impermeable earthen dike to direct free draining water rendered from the paste as it drained to a large collection pond that also collects the thickener overflow from the paste plant.

While perhaps unintended by the previous operator, the remaining structures form the foundation for a system based on a sound, well proven method for effectively managing free draining paste tailings. The following generalized schematic shown on Figure 18.3, following this page, provides the paste containment concept developed by the previous operator.

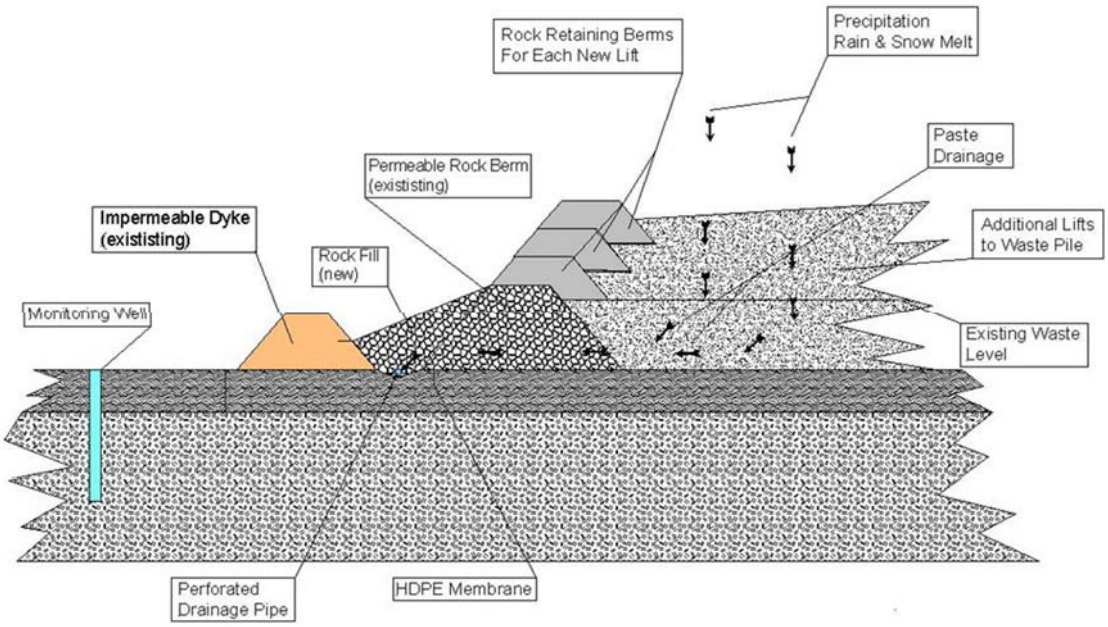

Figure 18.3 – Past Confinement Conceptual Diagram (not scale)

As the design grind for the low grade tailings is relatively course (P80 ~100 µm), and the pulp is virtually free from slimes, the paste product has been demonstrated to be free-draining, thereby eliminating development of a hydrostatic head requiring containment.

Rather, the paste deposited at ± 55% solids simply needs confinement as it dewaters discharging through the permeable rock berm. Similarly rain and snowmelt move through the deposited paste and through the permeable berm and are directed to the containment pond.

Prepared NAG waste rock from the mine will be utilized to continue with subsequent lifts to increase the height of the paste “stack” as production continues. Sufficient volume within the current paste deposition area remains to accommodate 12.7 million tons of low grade tailings within the elevation limit provided under the current permit. Deposition of paste above this elevation will require a permit amendment to continue to raise the elevation or alternatively to commission a new paste deposition area.

While OTSO is continuing to investigate most cost-effective approaches to paste containment in a manner acceptable to regulatory authorities, which undertaking is not yet complete, the foregoing containment concept has been used for cost analysis in this Technical Report.

18.7 Site Drainage

All site drainage, including site run-off, run-off from PAG waste, mine dewatering, paste thickener overflow, and drainage from deposited paste is collected and pumped to a temporary storage basin with a maximum capacity of approximately 700,000 cubic meters. Excess combined clear water effluent from the storage basin not recycled as process water is pumped through a 17.42 km pipeline to an offshore submerged discharge point. Maximum discharge rate with existing pumps and pipeline is approximately 240 cubic meters per hour (1,056 usgpm).

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19.0 MARKET STUDIES AND CONTRACTS

19.1 Introduction

Gold and minor silver in the form of doré bullion is expected to be delivered for sale to a recognized refinery for further processing with the resulting gold and silver being credited to OTSO's metal account for on-sale. As BOYD is not aware of any hedging agreements pertinent to OTSO, it is assumed that sales will be made at then current spot prices for gold and silver sold through the refiner.

Currently, OTSO is in continuing negotiations with MKS (Switzerland) S.A. for the refining contract.

19.2 Draft Refining Contract (under negotiation)

Key provisions of the refining contract still under negotiation and as advised by OTSO, which are contained in Section 22.0, Economic Analysis, in this Technical Report, are as follows:

19.2.1 Gold Dore Bullion

Quantity: Min. 30 kg per shipment

Quality: Min. 70% Au, balance non-deleterious

Rates: In USD per Net Doré ounce received

Gold Contents	Min. 70% Au	**Min. 80% Au**	Min. 90% Au
Treatment Charges (T/C)*	$0.34	**$0.32**	$0.28
Returnable Gold	99.90%	**99.92%**	99.95%
Returnable Silver**	99.00%		
OUTTURNS Gold/Silver***	Five business days after delivery at PAMP		Four business days after delivery at PAMP.

*Minimum T/C USD 450.00 per lot.
**Minimum 1% Ag to apply for recovery.
***or upon processing start with Customer's appointed representative.

Please note such outturn timing will only apply under normal processing capacity circumstances.

19.2.2 Provisional Payment

Based on the mine site weight and assay list, Customer can apply for provisional payment of up to 95% of the contained gold one working day after delivery of the doré at PAMP.

In case Customer asks for a provisional payment, the metals will need to be sold prior to the provisional transfer; MKS will charge an interest based on the Metal Lease rate (1M) + 1.75% from the value date of provisional payment to final outturn date. Such rate may be revised on a monthly basis and any revision will be communicated to the Customer by MKS.

The Customer may also request a provisional payment without pricing the metal prior to the provisional payment, in which case the Customer shall request it on a case-by-case basis to MKS.

Splitting Limits:
Au ±0.05%, Ag ±0.1%

Pricing:
The fine credited Au and Ag may be sold to MKS either SPOT or on the London LBMA gold AM or PM auction price or by any mutually accepted pricing method. Pricing should take place prior to provisional payment to Customer. Minimum amounts priced to be agreed upon between buyer and seller.

Payment:
Upon fixing of the metal and deduction of the prevailing charges, the USD will be transferred to the Customer's designated Currency account for value outturn date.

Invoice:
Prevailing charges will be invoiced on day of final settlement.

MKS' bank for payment US$:

JP MORGAN CHASE BANK, NEW YORK (CHASUS33)
Chips ABA 0002
Fav. Acct No 9492603460, UID 290644 OF MKS (Switzerland) S.A., Geneva

Termination Date:

This Schedule I, together with the Agreement, remain valid until 31 December 2022, and will be automatically renewed for a 12 (twelve) month period unless terminated by either Party as per Clause 15 of the Agreement.

19.2.3 Specifications

- PAMP weights and assays are to be considered as final and binding for settlements unless they are out of the splitting limits.
- The doré is melted in approx. 30 kg – 100 kg batches.
- We will cause the Insurers to inform Customer 30 days prior notice to any change in any insurance coverage.
- As per market practice, bank charges of USD 30 will be charged for each transfer.
- In case of metal transfers, a swap fee may be charged; market conditions at moment of transfer of metals will apply.
- When applicable, any changes in these charges and/or the introduction of new charges will be passed on to the customer at cost.
- Above rates apply for the recovery of Au and Ag only.

Delivery Point (DP) FOB Laiva Project site (as agreed)

19.2.4 Deleterious Elements Schedule of Charges

Element	Symbol	Free Upper Limit %	Increment %	Maximum Level %	Increment Charge USD per MT
Arsenic	As	0.01	0.01	1.0	20
Antimony	Sb	2.0	1.00	10.0	50
Bismuth	Bi	0.01	0.01	1.0	15
Cadmium	Cd	0.01	0.01	1.0	30
Mercury	Hg	0.01	0.01	0.05	300
Selenium	Se	0.1	0.05	1.0	50
Tellurium	Te	0.1	0.05	1.0	50
Thallium	Ti	0.1	0.05	1.0	50
Tin	Sn	2.0	1.0	10.0	30
Nickel	Ni	0.1	1.0	10.0	50
Zinc	Zn	5.0	1.0	15.0	20
Lead	Pb	1.0	1.0	5.0	50

Remarks:

a. The Refinery reserves the right to refuse material containing deleterious elements in excess of the Upper Limits.

b. Notwithstanding the above, Customer shall give PAMP adequate information about the physical and chemical nature of the material and indicate in writing any possible hazards that may arise during treatment.

c. The Refinery will not accept material which is radioactive or contains Iodine or Beryllium.

d. The Refinery will not, except by special arrangement, accept material which contains organo-metallic compounds or deleterious or penalty elements other than specified in the specific refining terms of the contract.

CALCULATION EXAMPLE OF PENALTY CHARGE:

Using 100 kg material at 0.4% Selenium.

For 100 kg material containing 0.4% Selenium the penalty charge would be $30.00 worked out as follows:

Selenium level 0.4% less “free” content of 0.1%= 0.3% divided by 0.05 (penalty increments) = 6 penalty units (p.u.)

6 p.u. @ $50.00 per MT ie (6 x $50 x 0.1MT) = $30.00

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The Laiva Mine and process plant were fully permitted for operation and were fully operated for the period February 2012 through December 2013 and the process plant only continued to operate, processing low-grade stockpile material until March 2014. Since then, the operation was on active care and maintenance, until restarted by FTR in August 2018 (processing November 2018). The site was placed back to care and maintenance in April 2019. All environmental permits are active and have legal validity. No permit requirements existing at this stage.

20.2 Environmental Studies Completed

In 2018, several studies on the LG TSF area structures were completed. These studies included stability assessment for the existing walls, tailings quality assessment, and danger risk assessment. Based on the information received from these studies, the LG TSF area structures have been re-assessed by Geobotnia and Knight Piesold in 2021. The latest studies have included new design for perimeter dams, tailings deposition model and detailed discharge plan for the first 12 months of production. All works recommended in these studies are currently being completed. Designs have been conditionally approved by applicable regulatory authorities for continued operation.

Additional studies have been completed regarding closure technique of HG1 pond (2018), LOM extensions of PAF storage including acid drainage management (2019), and waste rock geochemical analysis as part of the mine waste management program (2021).

20.3 Permitting

All permits are valid, and no permit related issues are existing. Permit compliance has been evaluated annually by related authorities, the most recent evaluation in October 2021. See Table 20.1, following this page for permits required and current status.

Table 20.1 – Environmental Permits

Description		Applicant	Authority	Date of Submission of Application	Current Status	Date of Permit Approval	Reference
Environmental and water resources management permit for Laiva Mine. (This permit controls pollution of soil, air and waterways and interference of Laiva Mine.)		Nordic Mines AB	Regional State Administrative Agency of Northern Finland	15/11/2007	Granted	24/11/2009	Permit decision no 84/09/2 Psy-2009-y-160
Permit change	Permit to change waste rock areas bottom layers structure from plastic.	Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	24/05/2010	Granted	29/10/2010	Permit decision no 100/10/1 PSAVI/196/04.08/2010
Permit change	Permit to change construction of LG TSF area.	Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	15/11/2013	Granted	1/6/2016	Permit decision no 76/2016/1 PSAVI/126/04.08/2013
Permit change	Application to take foreign origin concentrates into the process.	Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	08/06/2016	Granted	11/5/2017	Permit decision no 33/2017/1 PSAVI/1539/2016
Permit Change	Permit decision to approve detailed studies relating LG and HG TSF areas.	Nordic Gold Oy	Regional State Administrative Agency of Northern Finland	31/07/2018	Granted	13/6/2019	Permit decision no 80/2019 PSAVI/3048/2018
Permit for building process water pipeline under Pattijoki river and Haapajoki river.		Nordic Mines AB	Regional State Administrative Agency of Northern Finland	05/03/2009	Granted	24/11/2009	Permit decision no 85/09/2 Psy-2009-y-27
Permit for building wastewater discharge pipeline under Hörskönjoki river, Haapajoki river and Kuljunlahti canal, and to bottom of Perämeri and Kuljunlahti.		Nordic Mines AB	Regional State Administrative Agency of Northern Finland	05/03/2009	Granted	24/11/2009	Permit decision no 86/09/2 Psy-2009-y-28
Energy efficiency plan of Laiva Mine for operation.		Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	18/01/2011	Granted	12/06/2012	Permit decision no 80/12/1 PSAVI/10/04.08/2011
Permit to build dam around Vaarainjärvi sedimentation pond.		Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	31/01/2011	Granted	05/12/2011	Permit decision no 75/11/2 PSAVI/5/04.09/2011
The mine waste management plan.		Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	29/04/2011	Granted	30/06/2014	Permit decision no 65/2014/1 PSAVI/74/04.08/2011
Environmental permit of Nordic Mines Oy waste water discharge pipeline outlet place and permit for commissioning the construction work.		Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	13/07/2011	Granted	12/06/2012	Permit decision no 54/12/1 PSAVI/98/04.08/2011
Permit change	Permit to pump discharge water to the sea on quality-based limitation.	Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	24/04/2013	Granted	7/1/2016	Permit decision no 4/2016/1 PSAVI/34/04.08/2013
Waste water discharge pipeline under Hörskönjoki river and Haapajoki river, and permit for commissioning the construction work.		Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	14/12/2011	Granted	23/02/2012	Permit decision no 15/12/2 PSAVI/70/04.09/2011
Laiva Mine rehabilitation and closure plan.		Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	28/12/2011	Granted	30/06/2014	Permit decision no 66/2014/1 PSAVI/110/04.08/2011
Environmental permit of Laiva Mine fuel station.		Nordic Mines Oy	Regional State Administrative Agency of Northern Finland	27/02/2012	Granted	15/03/2013	Permit decision no 25/2013/1 PSAVI/23/04.08/2012

20.4 Environmental Monitoring Requirements Under Mining Permit

Environmental monitoring programs have been updated for the 2021 restart program. Final approval of the programs is expected momentarily and is expected to be pro forma, and as such are incorporated in this Technical Report. Environmental monitoring is currently being performed in accordance with the requirements during mine production. These monitoring programs were approved in 2018 with no change. Monitoring includes internal water quality monitoring, monitoring of discharge water quality, monitoring of surrounding creeks and rivers, groundwater monitoring, dust and air quality monitoring, monitoring for noise and vibration, monitoring of mine waste (waste rock and tailings), biological monitoring including sediment load, and monitoring of discharge to the sea including fishery impacts.

During both operational and care and maintenance periods, site environmental monitoring was completed based on separate monitoring programs for each requirement. Monitoring data is annually reported to regulatory authorities by an external consultant.

Monitoring requirements include monitoring of environment related structures (e.g., pipelines and pumpstations) and dam safety monitoring. Monitoring performance is annually reported to relevant regulatory authorities by the then current operator. See monitoring approvals in Table 20.2, following this page.

Table 20.2 – Environmental Monitoring Approvals

Description	Authority	Current Status	Date of Approval	Reference	Current Date	Comment
Approval of Laiva Mine environmental monitoring program.	Nordic Mines Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	27/09/2011	Granted	09/11/2011	POPELY/100/07.00/2010
Approval of Laiva Mine environmental monitoring program (lightened).	Nordic Mines Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	16/05/2014	Granted	22/07/2014	POPELY/100/07.00/2010
Approval of Laiva Mine environmental monitoring program for “care and maintenance” period.	Nordic Mines Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	01/04/2016	Granted	01/06/2016	POPELY/2663/2015
Approval for Laiva Mine environmental monitoring program.	Nordic Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	01/06/2018	Granted	14/08/2018	POPELY/2663/2015
Approval for Laiva Mine environmental monitoring for care and maintenance period.	Nordic Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	15/03/2019	Granted	21/03/2019	POPELY/2663/2015
Approval for Laiva Mine environmental monitoring program.	Otso Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	09/09/2021	Pending		
Approval for Laiva Mine biological monitoring.	Nordic Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	17/04/2018	Granted	26/06/2018	POPELY/2663/2015
Approval of Biological monitoring.	Otso Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	25/08/2021	Pending		
Approval of mutual coastal monitoring at Raahe sea areas.	Nordic Mines Oy, SSAB Oy, Raahen vesi Oy, Raahen Voima Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	27/10/2016	Granted	01/02/2017	POPELY/2462/2016, LAPELY 1459/5723-2016
Approval of monitoring program for internal water quality.	Otso Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	25/08/2021	Pending		
Approval of operational monitoring.	Nordic Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	29/1/2018	Granted	14/3/2018	POPELY/2663/2015
Approval of operational monitoring.	Otso Gold Oy	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	25/8/2021	Pending		

20.5 Social and Community Impacts

All mining activities are located within the mine lease area. The area has existing and a valid “town plan” in place approvals. The closest communities are located over 5 km away from the mine site. Due to current major forestry activities in the surrounding area, dust is limited to mine site areas, thereby having no impact on forestry designated zones

or local communities (verified by continuous monitoring under previous and current operator).

Excess site water, net of process recycle is discharged via pipeline to Bothnia Sea, as described in Section 18.0, above. The water quality for necessary discharge to the sea is also monitored monthly. Given the quality of the water discharged to the sea from operations to date, no issues from future operations can be reasonably predicted. The relatively minor adjustments to the process circuit discussed previously may actually reduce volume of discharge and attendant water quality.

20.6 Closure Plan and Cost

The project closure plan, including the mine waste management plan and mine waste control procedure, is currently being updated by an external consultant. The content requirements for these documents are described in the Mine Waste Act 190/2013; and the requirements to update the documents are stated in the permit decisions no 65/2014/1 PSAVI/74/04.08/2011 and no 66/2014/1 PSAVI/110/04.08/2011. By the Act 190/2013, the closure costs will be evaluated during the update process. The updates are due by 31 December 2021, and in each year thereafter. That said, the closure plan and attendant costs remain indeterminate, as it is expected that mine life will be significantly increased based on on-going resource development drilling. Accordingly, no closure costs are included in this Technical Study; instead, the closure costs are to be determined at a future date once the ultimate minable resource is determined and they can be accurately estimated.

The current closure cost bond in place is 4,290,950 € including existing mine waste areas (PAF 2.50 €/m2, clean waste rock 1€/m2, LG tailings 1.70 €/m2 and HG tailings 50 €/m2), post closure monitoring (0.5 M€) and tax (24%), which BOYD believes are adequate for the current minable resource as contained in this Technical Report. These will be updated based on further information as it becomes available. Disturbed areas and attendant costs for closure are reviewed in January of each year and will be updated as required.

21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

As previously described, the LGP is fully constructed with an operational mine, process plant, and all necessary infrastructure. The project commenced operation in 2012 and operated sporadically through early 2019 under two previous operators who were unsuccessful largely due to ineffective mining practices that lead to excessive mine dilution coupled with undercapitalization in both instances.

Through a business arrangement with FTR's (the immediate predecessor in interest) lender, OTSO has taken over the LGP effective in December 2020. Since that time, OTSO has worked to perform geophysical surveys, additional resource development drilling, engage a mining contractor, clean up and repair mill components and infrastructure in preparation of a re-start of operations ramping up from September 2021. The collective cost, including all of the foregoing plus negotiating agreements with lenders and trade creditors left by FTR, totals US $8.812 million all-in preproduction capital, which OTSO has elected to treat as sunk cost for purposes of forward-looking economic analysis of the LGP.

21.2 Summary

The basis for the capital and operating costs summarized in Tables 21.1 and 21.2 below, considers mining and processing all current Proven and Probable reserves as determined herein, over a project life (LOM) of 5.3 years based on annual process capacity of the existing plant. As the deposit remains open in all directions and resource development drilling is planned to continue, it is expected that project life will be increased by some amount which is not currently determinate. The cost of the additional resource development will be considered deferred capital to be amortized over the additional gold ounces discovered. Accordingly, this potential additional resource development is not included in the Technical Report.

Table 21.1 – Summary of Estimated Capital Costs (US$)

Item	Pre -Production	Sustaining	Total
Mine		400,000	400,000
Processing		8,085,800	8,085,800
Infra. & Indir.		7,680,000	7,680,000
Contingency		1,131,600	1,131,600
Total	**8,812,000***	**17,297,400**	**26,109,400**

* Not considered in economic analysis.

Table 21.2 – Summary of Estimated Operating Costs (US$)

Section	Cost (US$) ($/t Ore)
Mining	18.39
Processing	11.02
Infrastructure	0.49
Indirects	0.20
Tech Services	0.47
Safety	0.19
Environmental	0.56
G&A	1.73
Total Operating Cost	33.05

21.3 Capital Costs

21.3.1 Mine Capital Costs

The pits at the LGP are fully developed, including haul roads, sumps and pumping equipment, fully lined PAG waste dumps, and NAG waste dumps. The mining contractor supplies all mining and ancillary equipment on an operated and maintained basis. Accordingly, only relatively minor sustaining capital is expected for mine related light vehicles (OTSO mine supervision and technical services) and dewatering system maintenance. See estimated mine capital requirements for the LOM in Table 21.3 below.

Table 21.3 – Summary of Estimated Mine Capital Costs (US$)

Year	Mine Sustaining Capex
2021	25,000
2022	75,000
2023	75,000
2024	75,000
2025	75,000
2026	75,000
Contingency	28,000
LOM Total	**428,000**

21.3.2 Process Capital Costs

The process plant and related features are fully constructed and operational. Other than sunk capital not considered in this economic analysis, only sustaining capital is included prospectively. The sustaining capital includes all wear metal replacement for crushing equipment mill liners, pumps and impellers and other capitalized repairs. Sustaining capital estimates are based on historic component life and current replacement costs. See estimated process related sustaining capital for LOM in Table 21.4, following this page.

Table 21.4 – Summary of Estimated Process Capital Costs (US$)

Year	Process Sustaining Capex
2021	25,000
2022	3,942,900
2023	75,000
2024	3,892,900
2025	75,000
2026	75.000
Contingency	566,000
LOM Total	**8,651,800**

21.3.3 TSF and PAG Waste Dump Expansion

Three additional high grade tailings cells will need to be constructed over the LOM. Additionally, the construction of a further estimated 125,000 square meters of PAG waste storage will need to be constructed for an estimated total sustaining cost over the LOM of $7.2 million plus contingency. See estimated costs by year provided in Table 21.5 below.

Table 21.5 – Summary of Estimated TSF and PAG Waste Capital Costs (US$)

Year	TSF Exp	PAG Waste Exp	Total
2021	2,400,000	-	2,400,000
2022	4,112,500	687,500	4,800,000
2023	-	-	-
2024	-	-	-
2025	-	-	-
2026	-	-	-
Contingency	455,900	48,125	504,000
Total	**6,968,400**	**735,625**	**7,704,000**

21.3.4 Other Infrastructure and Indirect

Over the current LOM, estimated cost for sustaining capex by year to include utility equipment and vehicles as well as capital repairs to facilities is shown in Table 21.6 below.

Table 21.6 – Summary of Estimated Other Infrastructure and Indirect Capital Costs (US$)

Year	Infra. & Indir. Sustaining
2021	30,000
2022	90,000
2023	90,000
2024	90,000
2025	90,000
2026	90,000
Contingency	33,600
LOM Total	**513,600**

21.4 Operating Costs

21.4.1 Mine Operating Costs

Estimated Mine operating costs by year are shown in Table 21.7 below.

Table 21.7 – Estimated Mine Operating Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
Ore	1,508,648	7,262,211	3,769,298	932,328	3,400,976	3,531,939	**20,405,400**
Waste	10,566,502	30,636,878	26,384,999	15,528,123	16,160,664	10,723,010	**110,000,176**
Grade Control	499,539	2,408,444	1,256,953	302,147	1,101,310	1,142,137	**6,710,528**
Mine Services	865,700	2,636,933	2,164,733	1,247,633	1,415,693	1,004,586	**9,335,280**
Total Mine Operating Cost	**13,440,388**	**42,944,466**	**33,575,983**	**18,010,231**	**22,078,643**	**16,401,672**	**146,451,384**
Mine Operating Cost $/mt Ore	17.21	11.40	17.08	38.12	12.82	9.18	**13.96**
Mine Operating Cost $/Troy Oz Au Equ prod.	896.01	599.85	564.17	564.17	373.06	269.78	**485.36**

21.4.2 Process and Tailings Operating Costs

Estimated Process and Tailings costs are shown by year in Table 21.8 below.

Table 21.8 – Estimated Process and Tailings Operating Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
ROM Stockpile	61,341	307,479	306,939	308,456	308,456	308,456	**1,601,127**
Crushing	268,724	1,347,003	1,344,638	1,351,282	1,351,282	1,351,282	**7,014,210**
Grinding	950,178	4,762,858	4,754,496	4,777,988	4,777,988	4,777,988	**24,801,497**
Flash Float, Gravity, and Regrind	232,358	1,164,719	1,162,674	1,168,419	1,168,419	1,168,419	**6,065,008**
Leach	600,325	3,009,188	3,003,905	3,018,748	3,018,748	3,018,748	**15,669,661**
Leach Reagent Storage, Mixing, Distr	22,853	114,551	114,350	114,915	114,915	114,915	**596,498**
Elution, Carbon Regen, and EW	108,181	542,268	541,316	543,990	543,990	543,990	**2,823,735**
Misc Distribution Systems	175,674	880,584	879,038	883,381	883,381	883,381	**4,585,438**
Tailings Disposal	357,566	1,792,331	1,789,185	1,798,025	1,798,025	1,798,025	**9,333,157**
Process Instrumentation and Controls	38,825	194,613	194,271	195,231	195,231	195,231	**1,013,403**
Other Process Costs	545,993	2,736,841	2,732,036	2,745,535	2,745,535	2,745,535	**14,251,476**
Total Process & Tailings Op Cost	**3,362,018**	**16,852,434**	**16,822,849**	**16,905,970**	**16,905,970**	**16,905,970**	**87,755,212**
Process & Tails Operating Cost $/mt Ore	11.02	11.02	11.02	11.02	11.02	11.02	**11.02**
Process & Tails Op Cost $/Troy Oz Au Equ prod.	224.13	235.39	265.99	529.58	285.66	278.08	**290.83**

21.4.3 Infrastructure and Indirect Operating Costs

Estimated Infrastructure and Indirect operating costs are shown in Table 21.9, following this page.

Table 21.9 – Estimated Infrastructure and Indirect Operating Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
Site Drainage	39,320	117,960	117,960	117,960	117,960	117,960	**629,120**
Contract Guard Service	164,000	492,000	492,000	492,000	492,000	492,000	**2,624,000**
General Site Power	34,164	102,492	102,492	102,492	102,492	102,492	**546,624**
Domestic Water	7,200	21,600	21,600	21,600	21,600	21,600	**115,200**
Fuel Cost & Lube Total	12,006	36,018	36,018	36,018	36,018	36,018	**192,096**
General Labor	53,520	160,560	160,560	160,560	160,560	160,560	**856,320**
Misc Supplies	32,000	96,000	96,000	96,000	96,000	96,000	**512,000**
Total Infrastructure & Indirect Op Cost	**342,210**	**1,026,630**	**1,026,630**	**1,026,630**	**1,026,630**	**1,026,630**	**5,475,360**
Infra. and Indir. Operating Cost $/mt Ore	1.12	0.67	0.67	0.67	0.67	0.67	**0.69**
Infra. and Indir. Op Cost $/Troy Oz Au Equ prod.	22.81	14.34	16.23	32.16	17.35	16.89	**18.15**

21.4.4 Site Services Operating Costs

Estimated Site services costs, including Technical Services, Safety, and Environmental departments' operating costs are shown in Table 21.10 below.

Table 21.10 – Estimated Site Services Operating Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
Technical Services Department	61,910	742,925	742,925	742,925	742,925	742,925	**3,776,537**
Safety Department	24,593	295,112	295,112	295,112	295,112	295,112	**1,500,151**
Environmental Department	212,045	848,180	848,180	848,180	848,180	848,180	**4,452,945**
Total Site Services Op Cost	**298,548**	**1,886,216**	**1,886,216**	**1,886,216**	**1,886,216**	**1,886,216**	**9,729,633**
Site Services Operating Cost $/mt Ore	0.98	1.23	1.24	1.23	1.23	1.23	**1.22**
Site Services Operating Cost $/Troy Oz Au Equ prod.	19.90	26.35	29.82	59.09	31.87	31.03	**32.25**

21.4.5 G&A Operating Costs

Estimated G&A operating costs are shown in Table 21.11 below.

Table 21.11 – Estimated G&A Operating Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
Lionsbridge Management Fee	450,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	**9,450,000**
Labor	103,680	414,720	414,720	414,720	414,720	414,720	**2,177,280**
Supplies	71,997	287,988	287,988	287,988	287,988	287,988	**1,511,937**
Insurance	16,749	66,996	66,996	66,996	66,996	66,996	**351,729**
Property Tax	14,250	57,000	57,000	57,000	57,000	57,000	**299,250**
Total Site Services Op Cost	**656,676**	**2,626,704**	**2,626,704**	**2,626,704**	**2,626,704**	**2,626,704**	**13,790,196**
Site Services Operating Cost $/mt Ore	0.44	0.36	0.70	2.82	0.77	0.74	**1.73**
Site Services Operating Cost $/Troy Oz Au Equ prod.	43.82	36.73	41.57	82.36	44.43	43.25	**45.75**

21.4.6 Other Operating Costs

Other operating costs not included above, which include land tenure costs and royalty payable to PFL Raahe Holdings LP, are included in Table 21.12 below.

Table 21.12 – Other Estimated Operating Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
Land Tenure Cost (0.15% gold sales)	35,965	171,650	151,636	76,539	141,897	145,765	**723,453**
PFL Raahe Holdings Royalty (2.5% NSR)	-	2,944,259	3,019,349	1,452,814	2,248,158	3,245,171	**12,909,750**
Total Other Operating Cost	**35,965**	**3,115,909**	**3,170,985**	**1,529,353**	**2,390,055**	**3,390,936**	**13,633,203**
Other Operating Cost $/mt Ore	0.12	2.04	2.08	1.00	1.56	2.21	**1.71**
Other Operating Cost $/Troy Oz Au Equ prod.	2.40	43.52	50.14	47.91	40.39	55.78	**45.18**

21.4.7 All-in Operating Cost Summary

A summary of estimated all-in operating costs is presented below in Table 21.13 below.

Table 21.13 – Estimated All-in Operating Cost Summary (US$)

	2021	2022	2023	2024	2025	2026	Totals
Mining Cost Total	13,440,388	42,944,466	33,575,983	18,010,231	22,078,643	16,401,672	**146,451,384**
Processing Cost Total	3,362,018	16,852,434	16,822,849	16,905,970	16,905,970	16,905,970	**87,755,212**
Infrastructure and Indirect Total	342,210	1,026,630	1,026,630	1,026,630	1,026,630	1,026,630	**5,475,360**
Site Services Total	298,548	1,886,216	1,886,216	1,886,216	1,886,216	1,886,216	**9,729,633**
G&A Cost Total	656,676	2,626,704	2,626,704	2,626,704	2,626,704	2,626,704	**13,790,196**
Other Cost Total	35,965	3,115,909	3,170,985	1,529,353	2,390,055	3,390,936	**13,633,203**
Total All-in Operating Cost	**18,135,806**	**68,452,361**	**59,109,368**	**41,985,106**	**46,914,219**	**42,238,129**	**276,834,988**
All-in Operating Cost $/mt Ore	23.22	18.18	30.08	88.87	27.25	23.65	**34.76**
All-in Operating Cost $/Troy Oz Au Equ prod.	1,210.24	957.10	935.54	1,316.51	793.49	695.44	**918.38**

21.5 All-in Sustaining Costs

Estimated All-in sustaining costs (AISC) including capex and opex by year are included in Table 21.14 below.

Table 21.14 – Estimated All-in Sustaining Costs (US$)

	2021	2022	2023	2024	2025	2026	Totals
Capex Total	2,653,600	9,531,458	256,800	4,341,958	256,800	256,800	**17,297,416**
Opex Total	18,135,806	68,452,361	59,109,368	41,985,106	46,914,219	42,238,129	**276,834,988**
Total AISC	**21,338,489**	**77,983,819**	**59,366,168**	**46,327,064**	**47,171,019**	**42,494,929**	**294,681,487**
AISC Cost $/mt Ore	69.94	50.99	38.89	30.20	30.75	27.70	**37.00**
AISC Cost $/Troy Oz Au Equ produced	1,422.57	1,089.27	938.69	1,451.24	797.06	698.99	**976.63**

22.0 ECONOMIC ANALYSIS

22.1 Basis of Analysis

As previously noted, the LGP is a fully developed with existing pits, process plant, and all infrastructure. Accordingly, this Feasibility Study Technical Report does not contain preliminary facilities design work as is more typical. Additionally, the LGP has reached steady state operation under previous ownership/management, from which key operating parameters and costs can be drawn.

Updated mine design based on additional resource drilling and estimation as well as revised detailed mining plans have been developed. A mining contractor to perform all mining functions on a unit price basis has been engaged by OTSO. Based on the foregoing, BOYD has estimated from first principles mine capital and operating costs based on the contractor unit price schedule.

Process equipment remains generally unchanged with the addition of a parallel rental crusher line to produce finer feed to the primary milling circuit plus other minor flow sheet adjustments for optimized performance. The rental crusher units are under fixed monthly contract rates, which have been incorporated into the plant operating costs. Based on results of recent metallurgical testing, as discussed in Section 13.0 herein, BOYD has estimated from first principles plant capital and operating costs.

Infrastructure facilities are unchanged from historic. BOYD has estimated from first principles project infrastructure and indirect capital and operating costs based on historic figures updated to current pricing.

G&A and other costs have been provided to BOYD by OTSO. BOYD has reviewed these costs in detail and determined that they are reasonable for use in this Technical Report.

Pre-production capital for each of the following tasks and amounts have been provided by OTSO, as booked. BOYD has reviewed these costs and has no reason to doubt the accuracy thereof.

Pre-production Category	Amount ($US)
Environmental Bond	500,000
Security Deposit for Power	500,000
Pre-stripping	2,000,000
Laboratory	312,000
Misc. Consultants	950,000
TSF, Water Impoundment, and PAG Dump Repairs	800,000
Misc. Maintenance and Repairs	900,000
Pre-production Corporate Costs	1,200,000
Recruiting	600,000
Valmet Upgrade	150,000
Initial Chemical Inventory	450,000
Repair and Installation of Tower Mill	300,000
Pit Dewatering	150,000
Total Pre-production Capex	**8,812,000**

BOYD has analyzed pre and post-tax cash flows both ignoring sunk capital and including it to analyze the two alternatives.

22.2 Economic Assumptions

22.2.1 Gold Price

Gold price was taken at a fixed rate of $1,600/tr oz, conservatively supported by the 200-day trailing average London PM Fix quoted by Kitco at US $1,750 per tr oz Au, as well as the three-year volume weighted trailing average gold price also at approximately US $1,750 as quoted by Kitco based on London PM Fix.

Gold price was held constant at $1,600 per tr oz Au for the current LOM of 5.3 years.

22.2.2 Exchange Rate and Inflation

The USD to Euro exchange in September 2021 was approximately US $1.18 to 1.00 € and has been trending lower since that time. That said, most forecasts expect the ratio to continue in a relatively narrow trading range at least through 2023. BOYD examined the difference in the pre-tax NPV at 5% discount rate over the range of 1.15:1 and 1.21:1 exchange rate and determined that a variance of approximately 2% to the pre-tax NPV at 5% occurs. Given the relative insensitivity to exchange rate within forecast parameters, BOYD has elected to fix the exchange rate at US $1.18 to 1.00 € for economic analysis in this Technical Report.

Finnish inflation is expected to be relatively modest at “around 1.2% in 2022 and 1.5% in 2023”, according to trading economics.com. However, even modest inflation generally manifests as higher gold prices at fixed currency exchange rates, so BOYD has included no inflation given its decision to maintain a fixed gold price for the duration of the current LOM.

22.2.3 Corporate Taxation

Finnish corporate tax is currently 20% of adjusted income. Permitted adjustments to income include standard depreciation, tax loss carried forward (subject to certain expiry dates), and amortization of capitalized advance expenses. There is currently no standard depletion allowance in Finland.

BOYD has included both pre and post-tax calculations of key economic parameters in its analysis.

22.2.4 Royalties

OTSO is required to pay 0.15% of gold sales revenue to the holders of surface rights for occupation and use of surface not directly held by OTSO. In addition, OTSO is required to pay a 2.5% NSR to PFL Raahe Holdings LP as part of a partial settlement of a previous FTR debt obligation.

22.3 Technical Parameters Applied

22.3.1 Mine Production Schedule

Mine production is ramped up from September 2021 to achieve an average production of nominally 50,000 metric tonnes per day of combined ore and waste over the current life of the project.

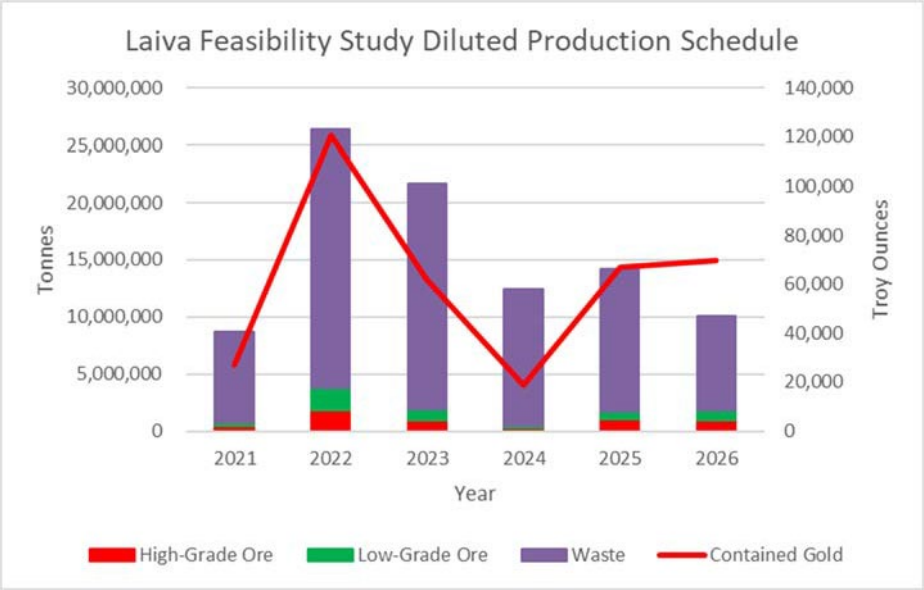

Figure 22.1 – LGP Mine Production Schedule

Table 22.1 – Mine Production Schedule

Mine Production	2021	2022	2023	2024	2025	2026	Total
Total Diluted Ore Tonnes to ROM Pad	781,042	3,765,669	1,965,280	472,415	1,721,928	1,785,762	**10,492,096**
North Pit High and Low Grade (mt)	607,547	3,101,587	1,930,802	70,193	215,504	150,236	**6,075,869**
South Pit High and Low Grade (mt)	173,495	664,081	34,479	402,222	1,506,424	1,635,526	**4,416,227**
Diluted Ore Grade to ROM Pad (g/mt)	1.071	0.996	0.984	1.239	1.214	1.217	**1.083**
Total Waste (mt)	7,875,958	22,603,664	19,682,053	12,003,918	12,435,004	8,260,102	**82,860,700**
North Pit (mt)	5,487,367	19,953,313	13,433,032	580,512	1,612,415	910,323	**41,976,962**
South Pit (mt)	2,388,591	2,650,352	6,249,021	11,423,407	10,822,590	7,349,779	**40,883,739**
Total Mining (mt)	8,657,000	26,369,333	21,647,333	12,476,333	14,156,932	10,045,864	**93,352,796**
Strip Ratio (waste:ore)	10.1	6.0	10.0	25.4	7.2	4.6	**7.9**

22.3.2 Process Production Schedule

Mined ore will be deposited on the ROM ore pad and stockpiled in accordance with grade and ore type for plant feed blending. Processing rate and schedule are independent of the mining schedule. The process production schedule is shown in Table 22.2, following this page.

The existing LGP process plant is limited to approximately 1.5 - 2.0 million metric tonnes per year. It's based on primary grinding capacity as supported by test work and historical throughput that achieved output particle size suitable for reasonable recovery for down stream process steps in aggregate. Adoption of additional crushing equipment (rented and included in Opex) to reduce the 80% of the particles size to P_{80} 25 mm for feed to

the existing primary grinding circuit (3,000 Kw AG mill) was deemed essential by BOYD to achieve this annual production target.

It may be possible to feed up to 2 million metric tonnes per year without compromising overall gold recovery based on recent test work. However, BOYD recommends the production rates shown in Table 22.2 below, based on the optimized mine plan andthe need for further plant optimization during ramp up. Actual operating experience and availability of increased minable high grade as resource development continues could suggest increasing mill throughput closer to 2 million metric tonnes per year.

Table 22.2 – Mill Production Schedule

Plant Production	2021	2022	2023	2024	2025	2026	Total
Plant Feed							
High Grade							
Tonnes (mt)	305,085	1,529,269	1,382,039	286,650	988,050	966,700	**5,457,793**
Grade (g/mt)	1.659	1.579	1.551	1.732	1.763	1.847	**1.651**
Low Grade							
Tonnes (mt)	-	-	144,546	1,247,477	546,077	567,427	**2,505,527**
Grade (g/mt)	-	-	0.465	0.465	0.467	0.468	**0.466**
Total							
Tonnes (mt)	305,085	1,529,269	1,526,584	1,534,127	1,534,127	1,534,127	**7,963,320**
Grade (g/mt)	1.659	1.579	1.398	0.702	1.301	1.337	**1.278**

22.4 Project Economic Analysis

22.4.1 Capital Costs

Given that the mine, plant, and infrastructure are fully constructed, the mine will be produced by a mining contractor who will provide all equipment, and the relatively short current LOM, only modest sustaining capital is required in this economic analysis.

Sustaining capital as required in total is estimated in the summary Table 22.3, following this page. Sustaining costs by period have been applied in the economic analysis.

Table 22.3 – Summary of Estimated Capital Costs (US$)

Item	Pre -Production	Sustaining	Total
Mine		400,000	400,000
Processing		8,085,800	8,085,800
Infra. & Indir.		7,680,000	7,680,000
Contingency		1,131,600	1,131,600
Total	**8,812,000***	**17,297,400**	**26,109,400**

* Not considered in economic analysis

22.4.2 Operating Costs

Operating costs have been estimated based on the schedule of rates for the mining contractor, recent historical parameters under previous management, and from first principles by BOYD. A summary of unit operating costs estimated on a total project basis is presented in the summary Table 22.4 below. Actual detailed operating costs have been determined by applicable period.

Table 22.4 – Summary of Estimated Average LOM Operating Costs (US$)

Section	Cost (US$) ($/t Ore)
Mining	18.39
Processing	11.02
Infrastructure	0.49
Indirects	0.20
Tech Services	0.47
Safety	0.19
Environmental	0.56
G&A	1.73
Total Operating Cost	**33.05**

22.4.3 Cash Flow Projection

Table 22.5, following this page, summarizes the LOM cash flows for the project, while Table 22.6, following this page, presents the annual production and cash flow schedule.

Table 22.5 – LOM Cash Flow Summary (US$)

Pre-Tax	
Net Operating Revenue	**472,300,097**
Less All-in Operating Costs	(275,717,047)
Operating Margin (EBITDA)	196,583,050
Less Sustaining Capital	(17,297,400)
Undisc Pre-Tax CF ex sunk	**178,736,551**
Post-Tax	
EBITDA	196,583,050
Depreciation Adjustment	(40,739,818)
Loss Carried Forward Adjustment*	(111,355,978)
Net Taxable Income	44,601,470
Tax Payable	(8,920,294)
After Tax Income (ex-sunk)	187,662,756
Undisc After Tax CF ex sunk**	**169,816,257**

* Note, this includes existing NOL expirations
** Note, sunk capital is included in NOL carried forward

Table 22.6 – LOM Production and Cash Flow Summary

Parameter	2021	2022	2023	2024	2025	2026	Total
Mining							
Ore Mined (mt)	781,042	3,765,669	1,965,280	472,415	1,721,928	1,785,762	**10,492,096**
Waste Mined (mt)	7,875,958	22,603,664	19,682,053	12,003,918	12,435,004	8,260,102	**82,860,700**
Total Mined (mt)	8,657,000	26,369,333	21,647,333	12,476,333	14,156,932	10,045,864	**93,352,796**
Strip Ratio (W:O)	10.1	6.0	10.0	25.4	7.2	4.6	**7.9**
Processing							
Ore Processed (mt)	305,085	1,529,269	1,526,584	1,534,127	1,534,127	1,534,127	**7,963,320**
Grade Processed (gr/mt)	1.659	1.579	1.398	0.702	1.301	1.337	**1.278**
Gold Recovery (%)	90.2%	90.2%	90.2%	90.2%	90.2%	90.2%	**90.2%**
Net Payable Gold (tr oz)	14,675	70,038	61,872	31,230	57,898	59,476	**295,188**
Financial							
Net Revenue (USD)	23,479,310	112,060,403	98,994,429	49,967,959	92,636,228	95,161,769	**472,300,097**
Operating Cost (USD)	18,135,025	68,384,771	59,040,583	41,951,931	46,862,374	42,164,573	**276,539,257**
Sustaining Capital (USD)	2,653,600	9,531,458	256,800	4,341,958	256,800	256,800	**17,297,416**
Undiscounted Pre-tax CF	2,141,602	34,300,755	39,898,719	3,771,108	45,667,216	52,957,152	**178,736,551**
Net Tax	-	-	-	-	-	8,920,294	**8,920,294**
Undiscounted Post-tax CF	2,141,602	34,300,755	39,898,719	3,771,108	45,667,216	44,036,858	**169,816,257**

22.4.4 Discount Rate and Resulting NPV Analysis

BOYD determined discounted cash flow results on a pre and post-tax basis over a range of discount rates including 5.0%, 6.2%, 8.0%, and 15.0%.

As a discount rate of 5.0% appears to be the most common rate used in NI 43-101 Technical Reports filed to date in 2021, this rate was chosen as the base case to provide a direct comparison with other recently filed Technical Reports. While most used a 5% discount rate, a limited few used an 8% discount rate so BOYD included analysis at this discount rate as well.

The calculated discount rate by the Capital Asset Pricing formula using the three-month t-bill rate a/o 28 September 2021 quoted at 1.50% for the risk free rate, a market beta of 0.90 based on a basket of 86 gold equities for 2021 to date, and a risk premium of 5.30%, again based on a basket of Canadian gold equities with international operations, yielded an indicated discount rate of 6.27% based on the CAPM formula, a standard metric.

This compares very favorably with the reported weighted average cost of capital (WACC) for a basket of ten Canadian gold producers at 6.17%. Accordingly, BOYD also analyzed the NPV at a discount rate of 6.20% for a comparison combining the two.

Finally, BOYD also analyzed the NPV using a discount rate of 15.0% at the request of a large shareholder of OTSO.

The resulting pre and post-tax NPV at the discount rates noted above is shown below in Table 22.7.

Table 22.7 – Pre and Post Tax NPV

Discount Rate (%)	Pre-Tax NPV (US$)	After-Tax NPV (US$)
5.0	152,932,950	146,030,015
6.2	147,642,666	141,139,869
8.0	140,260,307	134,307,193
15.0	116,747,228	$112,467,418

Note: Both pre and post-tax values do not consider sunk cost, except in so far as sunk costs serve as an adjustment to taxable income in the post-tax case.

22.5 Sensitivity Analysis

22.5.1 Variance of Key Parameters

Cash flows for the LOM at multiple discount rates and varied by key parameters are shown in Tables 22.8 through 22.15, on the following pages.

Table 22.8 – Gold Price Variance

Parameter	Excluding Sunk Costs				
	90%	95%	Base Case	105%	110%
Pre - Tax Undicounted Cash Flow	132,758,068	155,747,310	178,736,551	201,725,793	224,715,035
Pre - Tax 5.0% discounted Cash Flow	112,691,285	132,812,118	152,932,950	173,053,782	193,174,615
Pre - Tax 6.2% discounted Cash Flow	108,588,561	128,115,613	147,642,666	167,169,718	186,696,771
Pre - Tax 8.0% discounted Cash Flow	102,870,793	121,565,550	140,260,307	158,955,063	177,649,820
Pre - Tax 15% discounted Cash Flow	84,725,973	100,736,601	116,747,228	132,757,855	148,768,482
Post - Tax 5.0% discounted Cash Flow	110,195,266	126,758,072	146,030,015	163,978,804	180,399,329
Post - Tax 6.2% discounted Cash Flow	106,237,227	122,412,498	141,139,869	158,605,069	174,614,139
Post - Tax 8.0% discounted Cash Flow	100,718,218	116,344,520	134,307,193	151,092,807	166,522,783
Post - Tax 15.0% discounted Cash Flow	83,178,445	96,983,100	112,467,418	127,045,193	140,585,207

Table 22.9 – Sustaining Capital Variance

Parameter	Excluding Sunk Costs				
	90%	95%	Base Case	105%	110%
Pre - Tax Undicounted Cash Flow	180,466,293	179,601,422	178,736,551	177,871,681	177,006,810
Pre - Tax 5.0% discounted Cash Flow	154,546,427	153,739,688	152,932,950	152,126,211	151,319,473
Pre - Tax 6.2% discounted Cash Flow	149,231,074	148,436,870	147,642,666	146,848,462	146,054,257
Pre - Tax 8.0% discounted Cash Flow	141,812,920	141,036,613	140,260,307	139,484,000	138,707,693
Pre - Tax 15% discounted Cash Flow	118,178,398	117,462,813	116,747,228	116,031,642	115,316,057
Post - Tax 5.0% discounted Cash Flow	147,643,492	146,836,754	146,030,015	145,223,277	144,416,538
Post - Tax 6.2% discounted Cash Flow	142,728,277	141,934,073	141,139,869	140,345,665	139,551,460
Post - Tax 8.0% discounted Cash Flow	135,859,806	135,083,499	134,307,193	133,530,886	132,754,579
Post - Tax 15.0% discounted Cash Flow	113,898,588	113,183,003	112,467,418	111,751,832	111,036,247

Table 22.10 – High Grade Recovery Variance

Parameter	Excluding Sunk Costs				
	90%	95%	100%	100%	100%
Pre - Tax Undicounted Cash Flow	149,069,725	163,903,138	178,736,551	178,736,551	178,736,551
Pre - Tax 5.0% discounted Cash Flow	126,967,707	139,950,329	152,932,950	152,932,950	152,932,950
Pre - Tax 6.2% discounted Cash Flow	122,443,676	135,043,171	147,642,666	147,642,666	147,642,666
Pre - Tax 8.0% discounted Cash Flow	116,135,365	128,197,836	140,260,307	140,260,307	140,260,307
Pre - Tax 15% discounted Cash Flow	96,086,064	106,416,646	116,747,228	116,747,228	116,747,228
Post - Tax 5.0% discounted Cash Flow	121,947,146	133,507,065	146,030,015	146,030,015	146,030,015
Post - Tax 6.2% discounted Cash Flow	117,714,138	128,973,400	141,139,869	141,139,869	141,139,869
Post - Tax 8.0% discounted Cash Flow	111,805,616	122,641,144	134,307,193	134,307,193	134,307,193
Post - Tax 15.0% discounted Cash Flow	92,973,322	102,421,831	112,467,418	112,467,418	112,467,418

Table 22.11 – Low Grade Recovery Variance

Parameter	Excluding Sunk Costs				
	90%	95%	100%	105%	105%
Pre - Tax Undicounted Cash Flow	162,424,894	170,580,723	178,736,551	186,892,380	186,892,380
Pre - Tax 5.0% discounted Cash Flow	138,656,528	145,794,739	152,932,950	160,071,161	160,071,161
Pre - Tax 6.2% discounted Cash Flow	133,787,551	140,715,109	147,642,666	154,570,223	154,570,223
Pre - Tax 8.0% discounted Cash Flow	126,995,734	133,628,020	140,260,307	146,892,593	146,892,593
Pre - Tax 15% discounted Cash Flow	105,387,138	111,067,183	116,747,228	122,427,273	122,427,273
Post - Tax 5.0% discounted Cash Flow	132,259,074	139,144,545	146,030,015	152,915,486	152,915,486
Post - Tax 6.2% discounted Cash Flow	127,760,934	134,450,401	141,139,869	147,829,337	147,829,337
Post - Tax 8.0% discounted Cash Flow	121,478,548	127,892,870	134,307,193	140,721,515	140,721,515
Post - Tax 15.0% discounted Cash Flow	101,420,725	106,944,071	112,467,418	117,990,764	117,990,764

Table 22.12 – Operating Cost Variance

Parameter	Excluding Sunk Costs				
	90%	95%	Base Case	105%	110%
Pre - Tax Undicounted Cash Flow	206,363,164	192,549,858	178,736,551	164,923,245	151,109,938
Pre - Tax 5.0% discounted Cash Flow	177,352,245	165,142,597	152,932,950	140,723,302	128,513,655
Pre - Tax 6.2% discounted Cash Flow	171,394,000	159,518,333	147,642,666	135,766,999	123,891,332
Pre - Tax 8.0% discounted Cash Flow	163,072,787	151,666,547	140,260,307	128,854,067	117,447,827
Pre - Tax 15% discounted Cash Flow	136,509,014	126,628,121	116,747,228	106,866,335	96,985,442
Post - Tax 5.0% discounted Cash Flow	167,529,422	157,543,851	146,030,015	134,144,979	123,168,833
Post - Tax 6.2% discounted Cash Flow	162,116,124	152,356,061	141,139,869	129,569,997	118,856,330
Post - Tax 8.0% discounted Cash Flow	154,545,602	145,104,216	134,307,193	123,180,899	112,838,434
Post - Tax 15.0% discounted Cash Flow	130,284,800	121,894,990	112,467,418	102,787,783	93,671,660

Table 22.13 – Strip Ratio Variance

Parameter	Excluding Sunk Costs				
	90%	95%	Base Case	105%	110%
Pre - Tax Undicounted Cash Flow	189,736,569	184,236,560	178,736,551	173,236,543	167,736,534
Pre - Tax 5.0% discounted Cash Flow	162,812,277	157,872,613	152,932,950	147,993,286	143,053,623
Pre - Tax 6.2% discounted Cash Flow	157,286,498	152,464,582	147,642,666	142,820,750	137,998,834
Pre - Tax 8.0% discounted Cash Flow	149,571,772	144,916,039	140,260,307	135,604,574	130,948,841
Pre - Tax 15% discounted Cash Flow	124,966,270	120,856,749	116,747,228	112,637,707	108,528,185
Post - Tax 5.0% discounted Cash Flow	155,658,317	150,886,699	146,030,015	141,173,332	136,316,648
Post - Tax 6.2% discounted Cash Flow	150,546,311	145,883,615	141,139,869	136,396,122	131,652,376
Post - Tax 8.0% discounted Cash Flow	143,400,077	138,891,364	134,307,193	129,723,022	125,138,851
Post - Tax 15.0% discounted Cash Flow	120,525,802	116,525,492	112,467,418	108,409,344	104,351,270

Table 22.14 – Dilution Percentage Variance

Parameter	Excluding Sunk Costs				
	90%	95%	Base Case	105%	110%
Pre - Tax Undicounted Cash Flow	179,355,133	179,044,986	178,736,551	178,429,818	178,124,774
Pre - Tax 5.0% discounted Cash Flow	153,506,939	153,219,161	152,932,950	152,648,295	152,365,186
Pre - Tax 6.2% discounted Cash Flow	148,207,620	147,924,374	147,642,666	147,362,485	147,083,819
Pre - Tax 8.0% discounted Cash Flow	140,812,712	140,535,761	140,260,307	139,986,338	139,713,844
Pre - Tax 15% discounted Cash Flow	117,260,073	117,002,968	116,747,228	116,492,842	116,239,801
Post - Tax 5.0% discounted Cash Flow	146,585,958	146,307,229	146,030,015	145,754,308	145,480,095
Post - Tax 6.2% discounted Cash Flow	141,687,823	141,413,101	141,139,869	140,868,116	140,597,831
Post - Tax 8.0% discounted Cash Flow	134,844,035	134,574,888	134,307,193	134,040,939	133,776,118
Post - Tax 15.0% discounted Cash Flow	112,969,074	112,717,579	112,467,418	112,218,579	111,971,054

Table 22.15 – Dilution Grade Variance

Parameter	Excluding Sunk Costs				
	90%	95%	Base Case	105%	110%
Pre - Tax Undicounted Cash Flow	178,507,803	178,622,177	178,736,551	178,850,926	178,965,300
Pre - Tax 5.0% discounted Cash Flow	152,732,743	152,832,846	152,932,950	153,033,054	153,133,157
Pre - Tax 6.2% discounted Cash Flow	147,448,367	147,545,516	147,642,666	147,739,815	147,836,965
Pre - Tax 8.0% discounted Cash Flow	140,074,289	140,167,298	140,260,307	140,353,315	140,446,324
Pre - Tax 15% discounted Cash Flow	116,587,918	116,667,573	116,747,228	116,826,883	116,906,537
Post - Tax 5.0% discounted Cash Flow	145,836,897	145,933,456	146,030,015	146,126,575	146,223,134
Post - Tax 6.2% discounted Cash Flow	140,952,248	141,046,058	141,139,869	141,233,680	141,327,490
Post - Tax 8.0% discounted Cash Flow	134,127,288	134,217,240	134,307,193	134,397,145	134,487,097
Post - Tax 15.0% discounted Cash Flow	112,312,503	112,389,960	112,467,418	112,544,875	112,622,332

Figures 22.2 through 22.5, following this page, are graphical representations of the above data. As is typica, gold price has the greatest sensitivity to variance over the range from 90% to 110% of base case gold price taken at US $1,600 per tr oz. This is followed in rank by operating costs, high grade recovery percentage, then low grade recovery percentage. It should be noted that the high grade recovery base case of 97% was not permitted to increase beyond 100% of base case. Similarly, the low grade recovery is not permitted to increase beyond 105% of base case. In both instances, based on test work results received to date, BOYD considers this to be the practical limit for recoveries.

Waste tonnage (strip ratio) is moderately sensitive over the 90% to 110% range examined, which in BOYD's opinion encompasses the probable range in practice.

Other variables show very low sensitivity over the range examined. See Figures 22.2 and 22.3, following this page, for graphical representations of pre and post-tax at 5% discount rate.

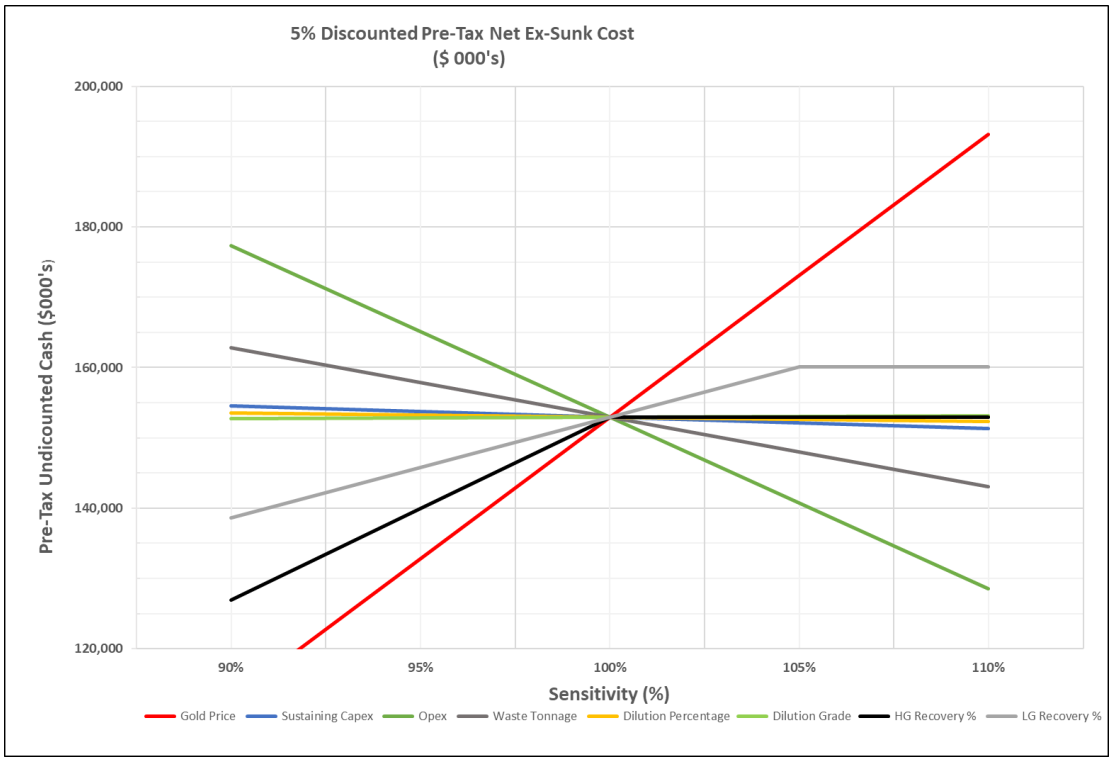

Figure 22.2 – Pre-Tax Sensitivity at 5% Discount Rate

5% Discounted Pre-Tax Net Cash Ex-Sunk Costs ($ 000's)

Sensitivity Parameter	90%	95%	100%	105%	110%
Gold Price	112,691	132,812	152,933	173,054	193,175
Sustaining Capex	154,546	153,740	152,933	152,126	151,319
Opex	177,352	165,143	152,933	140,723	128,514
Waste Tonnage	162,812	157,873	152,933	147,993	143,054
Dilution Percentage	153,507	153,219	152,933	152,648	152,365
Dilution Grade	152,733	152,833	152,933	153,033	153,133
HG Recovery %	126,968	139,950	152,933	152,933	152,933
LG Recovery %	138,657	145,795	152,933	160,071	160,071

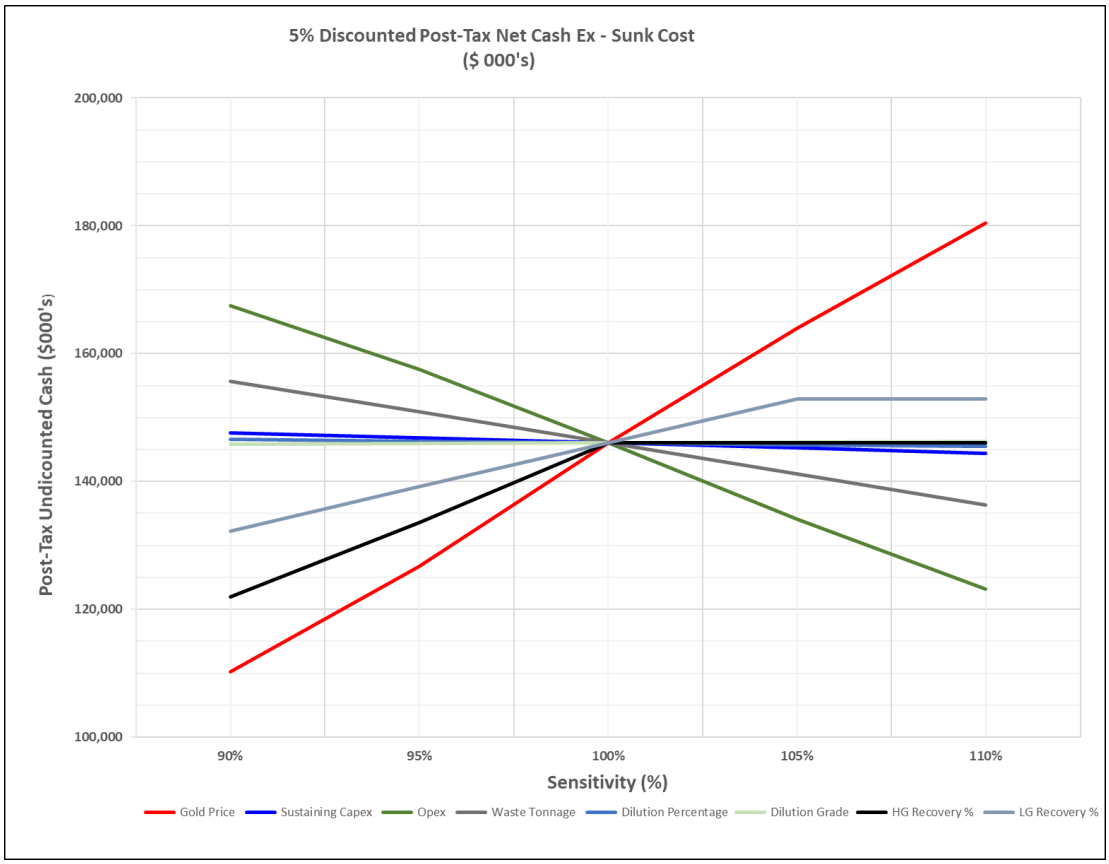

Figure 22.3 – After-Tax Sensitivity at 5% Discount Rate

5% Discounted Post -Tax Net Cash Ex-Sunk Costs ($ 000's)

Sensitivity Parameter	90%	95%	100%	105%	110%
Gold Price	110,195	126,758	146,030	163,979	180,399
Sustaining Capex	147,643	146,837	146,030	145,223	144,417
Opex	167,529	157,544	146,030	134,145	123,169
Waste Tonnage	155,658	150,887	146,030	141,173	136,317
Dilution Percentage	146,586	146,307	146,030	145,754	145,480
Dilution Grade	145,837	145,933	146,030	146,127	146,223
HG Recovery %	121,947	133,507	146,030	146,030	146,030
LG Recovery %	132,259	139,145	146,030	152,915	152,915

Graphical representations of pre and post-tax NPV are shown for the range of discount rates examined as Figures 22.4 and 22.5, following this page.

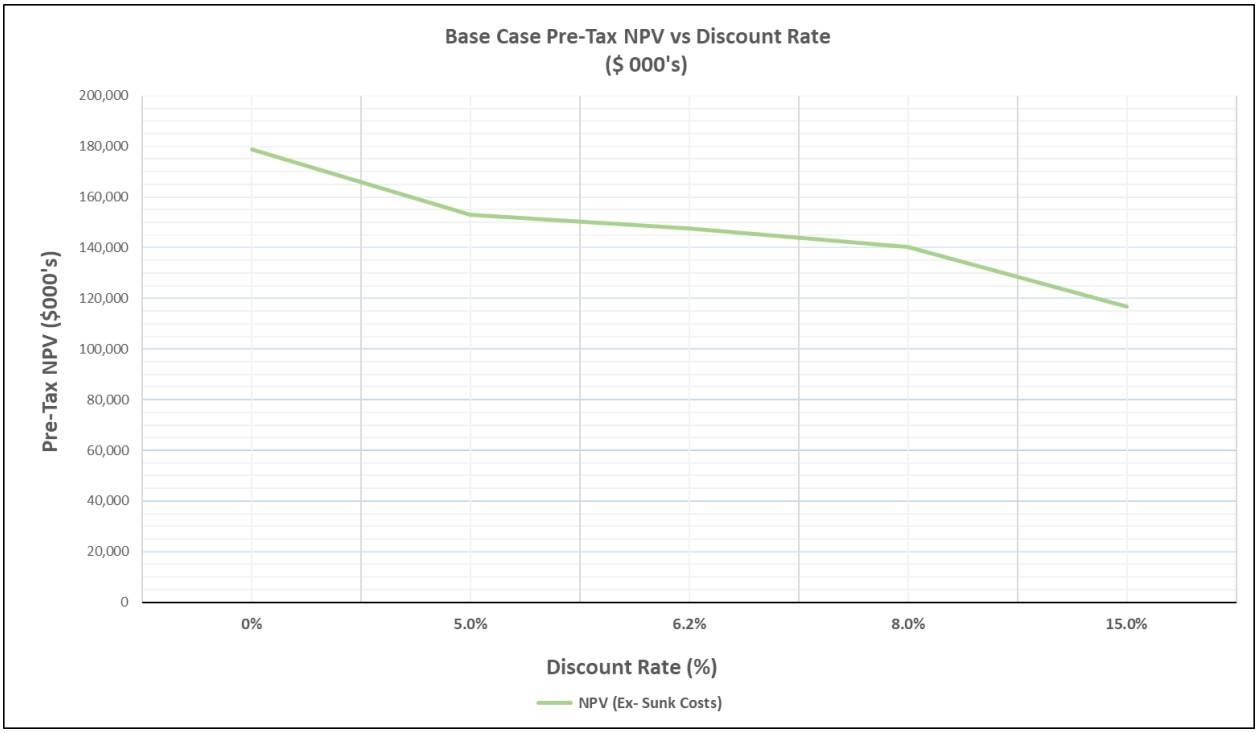

Figure 22.4 – Pre-Tax NPV at Multiple Discount Rates Analyzed

Base Case Pre-Tax NPV ($ 000's)					
Discount Rate (%)	**0%**	**5.0%**	**6.2%**	**8.0%**	**15.0%**
NPV (Ex- Sunk Costs)	178,737	152,933	147,643	140,260	116,747

Figure 22.5 – Post-Tax NPV at Multiple Discount Rates Analyzed

Base Case Post-Tax NPV ($ 000's)					
Discount Rate (%)	**0%**	**5.0%**	**6.2%**	**8.0%**	**15.0%**
NPV (Ex- Sunk Costs)	169,816	146,030	141,140	134,307	112,467

22.6 Economic Analysis Conclusions

From this forward-looking economic analysis, which is based on the LOM provided by the current Reserves determined herein, BOYD concludes that the project economics are generally favorable if sound mining and processing practices as recommended in this Technical Report are followed. The foregoing coupled with potentially significant resource development potential on untested portions of the LGP are encouraging for further project prosecution.

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23.0 ADJACENT PROPERTIES

There are no adjacent properties proximal to the LGP not currently held by the Issuer.

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24.0 OTHER RELEVANT DATA AND INFORMATION

All relevant data and information regarding the Laiva Gold Project is included in other sections of this Technical Report.

BOYD is not aware of any other data that would make a material difference to the quality of this Technical Report or make it more understandable, or without which the report would be incomplete or misleading.

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25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 Introduction

The recent management change to OTSO includes the advanced stage LGP and several satellite exploration targets at Mussuneva, Kaukainen, and Oltava. These prospects are considered to represent orogenic gold deposits, hosted in lower greenschist facies metamorphic rocks comprising mafic metavolcanic and quartz diorite, intruded by post- mineral granitoid and later dolerite stocks and dykes.

Mineralization is comprised of steeply dipping, generally east-west striking, planar, sheeted quartz-sulphide vein arrays hosted in mylonitic shear zones. Mineralized zones are typically meters to tens of meters wide, composed of multiple, sheeted, millimeter to centimeter scale quartz-sulphide veins. Individual veins are typically planar, but display local pinching, swelling, and minor strike-slip offset. Rare quartz lenses perpendicular to the main trend of mineralization are observed up to 10 cm wide and less than 1 m long.

The focus of this Technical Report is exclusively directed at the LGP, which includes two open pit mines as well as several nearby satellite pits contained within the Laiva Project area, in addition to a fully developed and operational process plant and all necessary infrastructure. It specifically does not include other project areas including Mussuneva, Kaukainen, and Oltava.

The LGP was managed and operated for a short period by two previous predecessors. Both previous efforts resulted in failure for a combination of reasons, including but not limited to in-effective geologic modelling in the early attempts, poor mining practice resulting in excessive dilution, and undercapitalization; all of which was exacerbated by periods of relatively low gold price.

OTSO has assumed management control of the project in its entirety and has proceeded with additional resource development drilling and other preparations for restart of the operation since assuming management responsibility for the project at nominal year end 2020. This has led to BOYD (an independent qualified consulting firm) being commissioned by OTSO to prepare an update to the MRE and a Feasibility Study in the form of NI 43-101 Technical Reports for filing as contained herein.

25.2 BOYD's Background with the Laiva Project

In addition to BOYD's engagement for completion of the MRE and Feasibility Study commissioned by OTSO, as noted above, BOYD was previously engaged by FTR to perform an initial NI 43-101 compliant MRE (Effective Date 9 August 2017) and subsequently an NI 43-101 compliant PEA (Effective Date 6 April 2018). Accordingly, BOYD personnel including qualified geologists, mining engineers, and process metallurgists have previously visited the property on behalf of FTR and OTSO in its role as independent consultant, including most recently 27 August – 7 September 2021.

Following BOYD's initial site visit, it was determined that the geological model should be revised to better reflect the steeply dipping mineralization observed in field exposures, drill core, and sectional interpretation. The revised interpretation allowed for better definition of the discrete mineralized zones and has reduced the effects of grade smearing observed in previous mineral resource estimates. BOYD's interpretation also included a review of the existing lithological wireframes prepared for NORDIC by SRK in 2014, which were not included in previous mineral resource estimates. This significantly improved the mineral resource estimate by allowing for a better estimation of the dilutive effects of post mineral granite and dolerite intrusions.

25.3 Key Interpretations and Conclusions

25.3.1 Drilling, Sampling, and QAQC Practices

Drilling practices have generally been to an acceptable standard in previous programs. However, drill core sampling methodology under previous programs required revision to terminate samples at lithological contacts, changes in alteration, and separate sampling of mineralized intervals for a minimum sample width of 20 cm and maximum width of 1 m. In addition, it was recommended that oriented drill core should be collected using a Reflex or similar tool which should become standard operating procedure. All of the foregoing has been implemented under OTSO's recent drilling program.

Owing to the complexity of the mineralized body at Laiva, BOYD recommended closely spaced short-hole RC drilling ahead of actual mining to define the mineralized zones more precisely to further minimize mining dilution. This too has been implemented by OTSO.

The QAQC program has previously relied upon the internal procedures employed by the independent laboratories used for sample preparation and analysis, with little evidence of continual assessment of control limits per sample batch or remedial action. While the QAQC results demonstrate good performance overall, implementation of a more

rigorous external QAQC program was recommended in future programs to better assess the potential for contamination during sample preparation as well as laboratory accuracy and precision. Selection of duplicate samples to better represent the mineralization at Laiva was also recommended. This too has been resolved under OTSO's recent and on-going program.

25.3.2 Metallurgical Testing

Previous metallurgical test work conducted on the Laiva deposit indicates that gold is typically fine-grained, free milling, and amenable to cyanide leach processing. Further test work was commissioned by OTSO to better characterize the gold grain size distribution, petrology, mineralogy, and optimum grind size. This work, though not yet fully complete has permitted a more detailed review of potential improvements to the existing mineral processing flow sheet as specifically recommended herein in Section 17.0, above.

25.3.3 Existing Process Plant as Amended

During the relatively stable 14 month period of previous operations, the Laiva process plant operated generally as expected given the characteristics and grade of feed supplied to the plant. However, plant performance was significantly impaired by mining practices which led to excessive mine dilution issues that were further exacerbated by reliance on the previous resource model, which lacked necessary definition of mineralized zones.

BOYD evaluated the historical operational information, results from the metallurgical test program completed in 2021, and results from the several phases of previous metallurgical test work to provide the basis for the estimated point forward gold recovery of 90% at a grinding circuit product size of P_{80} 100 µm.

The existing Laiva process facilities with the planned minor modifications should be able to achieve production rates for future operations that start at 1.5 M tonnes per year for year 1 ramping up to 2.0 M tonnes per year by year 3.

Potential to increase the plant production rate to over 2.0 M tonnes per year is likely dependent on the maximum feed rate that can be fed to the HG leaching while achieving plus 95% gold recovery. Therefore, to increase overall production rates above 2.0 M tonnes, if so desired, additional HG CIL tanks will likely be required.

The most promising option to substantially increase overall gold production from the processing would be the addition of a vat cyanide leach circuit for the treatment of the lower grade portions of the ore body. This addition would allow concurrent treatment of HG and LG ores.

Procedures previously utilized to monitor cyanide levels in the tailings discharge system included in BOYD's opinion, excessive sampling frequency with samples collected at the point of discharge from the process plant. Based on a review of the discharge permit, the sample point should actually be at the point of slurry discharge into the tailings facility and sample frequency should be significantly reduced.

In BOYD's overall opinion, the additional primary and secondary crusher units will improve primary grinding throughput, while maintaining suitable product particle size to maintain downline gold recovery.

Recommended minor piping modifications should maximize recovery from the gravity circuit, whereby the combined gravity circuit will see all tailings from the flash flotation unit, as compared to only part of the flow previously. Also, the particle size range will be narrowed for both the Knelson concentrator, focused on the finer fraction, and the spirals focused on the coarser fraction for better recovery from each.

Finally, installation of the regrind tower mill will permit maximum feed to the high grade leach circuit, which has demonstrated high recovery of gold at greater than 95% Au recovery.

25.3.4 Exploration Potential

In BOYD's opinion, the LGP likely holds favorable exploration potential. Near-mine exploration potential exists to the north of the North Pit, associated with a northeast trending structure defined by a gold-arsenic anomaly in bedrock sampling. Further stripping of ±3 m deep surficial till to allow channel sampling and mapping will be required to advance this area. In addition, the mineralization at the Laiva Gold Mine remains open at depth. The controls on high grade mineralization require better understanding in order to assess the down dip potential.

26.0 RECOMMENDATIONS

26.1 Introduction

BOYD's Preliminary Economic Assessment completed for FTR in July 2018 recommended completion of further infill drilling, metallurgical testing, and optimization trade-off studies, among other things. Almost none of the recommendations were followed by FTR likely due to undercapitalization and FTR's urgent desire to commence production. On the other hand, OTSO as a well-capitalized highly knowledgeable operator, has begun acting on these previous recommendations. This includes an ongoing in-fill drilling program, metallurgical testing, and minor, but important process plant modifications which have led to this Feasibility Technical Report.

In the context of this Technical Report, BOYD offers the additional recommendations contained in the sub-sections below.

26.2 Resources/Reserves and Mining

26.2.1 Continue With Program of Closely Spaced RC Definition Drilling

Given the complexity of the Laiva mineralized body, in BOYD's opinion, it is not cost effective to drill resource development core holes on a tighter spacing than is required by definition of measured and indicated resources. Very closely spaced short RC definition drilling will be more cost-effective in developing higher precision for the mineralized zone necessary to minimize mining dilution for attendant higher grade mill feed.

26.2.2 Implement Detailed Bench Mapping to Advance Structural Understanding of the Deposit

A better understanding of the structures, including sequence of events and implications for mineralization will improve overall understanding of the deposit both for continued resource development and effective mining strategies. Also, detailed mapping of intrusive bodies pre and post-mineralization, as well as intrusive impacts on primary mineralization should lead to a better understanding of potential zones of secondary enrichment through re-mobilization of gold. Mapping should be laid out on a 1 m X 1 m grid and surveyed for accurate geo-reference.

26.2.3 Continued Infill and Step Out Core or RC Drilling for Additional Resource Development for Priority Targets as Above

Based on good comparative results from recent diamond core drilling to RC drilling to the East of the existing North Pit, BOYD recommends instituting an RC drilling program for in-fill and step out resource development to be followed up with limited diamond core drilling to confirm accuracy of RC results, perhaps to include one twinned core hole per 10 RC holes.

It is essential that the RC drilling rig(s) be equipped with high pressure, high volume compressors to assure clearing of the holes with no lag in cutting returns given the potential for groundwater presence in the holes. The RC rigs should be equipped with auto samplers receiving the underflow from the velocity sink cyclone to ensure consistent sampling.

26.3 Processing

- Evaluate alternative crushing circuit options to eliminate rental crushers to optimize feed to the grinding circuit.
- Utilize results from the completed metallurgical test program to develop detailed optimization plans for all process unit operations.
- Develop metallurgical balance with adequate detail for all process related planning and reporting requirements following plant restart.
- Plan detailed sampling and analyses program for initial operating period to provide the results required to optimize the split between low-grade and high-grade leach circuits. The program should include:
    - Sampling procedures that ensure the collection of representative samples for the feed and discharge of each of the concentrate recovery systems.
    - Accurate determination of particle size distribution of all samples.
    - Quick turn-around analyses of gold and arsenic concentrations in solution and solids samples.
    - Quick turn-around determination of mineralogical composition of solids samples.
    - Bottle roll cyanide leach testing of samples, primarily for HG and LG CIL circuit samples.
- Maximize feed rate to HG circuit while achieving plus 95% gold recovery. Feed rate to the HG leach circuit is likely the limiting parameter to increase the overall plant feed rate.
- Evaluate modifications to the LG CIL circuit to reduce processing costs. LG CIL circuit potential modifications should include:
    - Reduction in leach solution cyanide concentration.
    - Reduction in air addition rate.
    - Reduction in number of CIL tanks being utilized.
    - Conversion to CIP.

- Implement a detailed carbon activity test program.
- Complete a detailed trade-off study comparing the economics of utilizing a new vat cyanidation circuit verses the existing CIL circuit for treatment of the LG zones of the ore body.
- Complete trade-off study of potential benefits for installation of a cyanide recovery circuit. A cyanide recovery circuit could potentially provide significant cost savings and environmental advantages.
- Complete leach solution cyanide analysis of samples from all locations related to the tailings discharge facility and prepare report to confirm the ability to monitor tailings discharge cyanide concentrations at the point of slurry discharge into the tailings storage facility.

26.4 Cost Estimate for Recommended Programs

26.4.1 Resources/Reserves and Mining

BOYD recommends the foregoing program be conducted for one year to be followed by reassessment of efficacy and continuing opportunities thereafter. The total estimated cost of the recommended programs for a one-year period is shown in Table 26.1 below.

Table 26.1 – Total Estimated Cost of the Recommended Resource/Reserves Programs

Item	Total Program Cost (US $/yr)	Comment
Definition Drilling	525,000	RC Drilling plus PAL Assay
Detailed Bench Mapping	25,200	Mapping and Surveying
Geochemical Surveys	100,000	Sample Collection and Analysis
Geophysical Surveys	180,000	Mag and ADR
RC Drilling	360,000	6,000m @ US $40/m drill + US $20/m assay
Diamond Core Drilling Verification Check	155,000	1,000m @ US $135/m drill + US $20/m assay
Total	**1,345,200**	Reassess after one year

26.4.2 Estimated Cost for Process Related Programs

Mush of the estimated process recommendations should be routinely included in operating costs, which are already accounted for in this Technical Report. The trade-off studies referenced above may require assistance from external consultants, but are not likely to exceed US $100,000. In the alternative, the trade-off studies may be undertaken by OTSO staff if time and expertise is available.

27.0 REFERENCES

Arthur, J., 2016. Nordic Mines, Laiva Project Mineral Resource Update, May 2016. Unpublished report prepared for Nordic.

Dowd, P.A., 2008. Letter report summarizing a mineral resource estimate at Laiva. Prepared for Nordic Mines.

Eilu, P., Sorjonen-Ward, P., Nurmi, P. and Niiranen, T., 2003. A review of gold mineralization styles in Finland. Economic Geology, v.98, p.1329-1353.

Groves, D.I., Goldfarb, R. J., Robert, F. and Hart, C. J. R., 2003. Gold deposits in metamorphic belts: Overview of current understanding, outstanding problems, future research, and exploration significance. Economic Geology, v.95, p.1-29.

Hanski, E., 2015. Synthesis of the geological evolution and metallogeny in Finland.p.39-71. In: Maier, W. D., Lahtinen, R. and O'Brien, H., (eds.), Mineral deposits of Finland.

Howarth, V., 2016. Terravision Radar Enhanced GPR – Laiva Mine. Unpublished report prepared for Nordic Mines AB.

Lehtonen, T., 2006. An assessment of the historical geophysical data at Laivakangas target in central Finland. Unpublished report prepared by Astrock Oy.

Luukas, J., Kousa, J., Nironen, M. and Vuollo, J., 2017. Major stratigraphic units in the bedrock of Finland, and an approach to tectonostratigraphic division, p.9-40. In: Nironen, M. (ed.), 2017. Bedrock of Finland at the scale 1:1 000 000 – Major stratigraphic units, metamorphism and tectonic evolution. Geological Survey of Finland, Special paper 60.

Nironen, 2017. Guide to the geological map of Finland – bedrock 1:1 000 000, p.41-76. In: Nironen, M. (ed.), 2017. Bedrock of Finland at the scale 1:1 000 000 – Major stratigraphic units, metamorphism and tectonic evolution. Geological Survey of Finland, Special paper 60.

O'Conner, M. and Reeves, C., 2012. Summary Report – mineral resource and ore reserve update, Laiva Gold Deposit, Finland. Summary technical report prepared for Nordic by CSA Global (UK) (R215.2012).

Orudña, J., 2016. Review of the regional exploration potential of the Laiva gold project, Finland. Unpublished report prepared by Rocklore Exploration Services Ltd.

Pratt, W., 2010. Controls on gold mineralization at Laiva mine, Finland. Unpublished report prepared by Specialized Geological Mapping Ltd.

SRK Consulting (UK) Ltd, 2013. A mineral resource update for the Laiva Gold Mine, Finland. Prepared for Nordic (UK5700).

SRK Consulting (UK) Ltd, 2015. The Laiva Gold Mine updated business plan 2014 – technical report. Unpublished report prepared for Nordic (UK6191).

Titley, M. 2008. Mineral resource summary report, Nordic Mining – Laiva Gold Deposit, Finland. Summary technical report prepared for Nordic by CSA Global (UK) (R254.2008).

Titley, M. 2009. Laiva Gold Project, Finland. NI 43-101 technical report prepared for Nordic by CSA Global (UK).

Titley, M. 2011. Summary report – mineral resource and ore reserve update, Laiva Gold Deposit, Finland. Summary technical report prepared for Nordic by CSA Global (UK).

Titley, M. 2012. Summary report – mineral resource and ore reserve update, Laiva Gold Deposit, Finland. Summary technical report prepared for Nordic by CSA Global (UK) (R451.2012).

BOYD 2017. NI 43-101 Technical Report, Mineral Resource Estimate, Laiva Gold Mine Project, Raahe, Finland.

BOYD 2018. NI 43-101 Technical Report, Preliminary Economic Assessment, Laiva Gold Mine Project, Raahe, Finland.

BOYD 2021. NI 43-101 Technical Report, Resource Estimate Update, Laiva Gold Mine Project, Raahe, Finland.

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28.0 CERTIFICATE OF AUTHOR QUALIFICATIONS

Sam J. Shoemaker, Jr.

As a co-author of this Technical Report entitled "NI 43-101 TECHNICAL REPORT FEASIBILITY STUDY LAIVA GOLD MINE PROJECT", Raahe Finland dated 1 November 2021 with an effective date of 4 October 2021, I Sam Shoemaker do hereby certify that:

1. I am engaged as a Project Manager – Metals for the John T. Boyd Company, with offices at 600 17th St. Suite 2800S, Denver, Colorado 80202-5404, telephone +1 (303) 293-8988, e-mail s-shoemaker@jtboyd.com.

2. This certificate applies to the aforementioned Technical Report.

3. I hold the following academic qualifications:

   B.S. Mining Engineering, Montana College of Mineral Science and Technology 1982

4. I am a Registered Member Society for Mining, Metallurgy, and Exploration Inc.

5. I have worked as a mining engineer in the minerals industry for 38 years.

6. I have read NI 43-101 and Form 43-101F1 and, by reason of education, experience, and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 10 years as a mining engineer with Cleveland Cliffs Inc., 18 years with other mining companies, and 10 years as a consultant where I have worked on mineral resource estimates in gold, silver, copper, nickel, iron, platinum, palladium, and aggregates. Additionally, I was responsible for completing geologic models, reserve estimates, economic analysis, slope designs, pit optimization, pit design, long term scheduling, short term scheduling and reserve validation in a variety of commodities including gold, silver, iron, platinum, palladium, lithium, uranium, aggregates, limestone, rare earths, frac sands, placer gold, lead, zinc, and talc.

7. I have read this Technical Report and believe it conforms in content and format with the requirements of NI 43-101f1.

8. I have not visited the Laiva Gold Project site.

9. With the exception of being a co-author of an NI 43-101 TECHNICAL REPORT MINERAL RESOURCE ESTIMATE LAIVA GOLD MINE PROJECT with an Effective Date of 9 August 2017 and a co-author of an NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LAIVA GOLD MINE PROJECT with an Effective Date of 6 April 2018 as an independent consultant to Fire Steel Resources Inc., a predecessor in interest to OTSO, I have no prior association with the Laiva Project, which is the subject of this Technical Report.

10. I am independent of Otso Gold Corp. and its subsidiaries. I hold no beneficial interest in any of the foregoing.

11. I am the principal author of Section 14. of this Technical Report.

12. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information required for disclosure to avoid making this Technical Report misleading.

Report dated this 1st day of November 2021 with an effective date of 4 October 2021.

Dated 1 November 2021
(Signed and Sealed)

Sam J. Shoemaker, Jr., B.S., SME Registered Member
Project Manager – Metals
John T. Boyd Company

Gregory B. Sparks

As a co-author of this Technical Report entitled "NI 43-101 TECHNICAL REPORT FEASIBILITY STUDY LAIVA GOLD MINE PROJECT", Raahe Finland dated 1 November 2021 with an effective date of 4 October 2021, I Gregory B. Sparks do hereby certify that:

1. I am engaged as Managing Director - Metals for the John T. Boyd Company, with offices at 600 17th St. Suite 2800S, Denver, Colorado 80202-5404, telephone +1 (303) 293-8988, e-mail g-sparks@jtboyd.com.

2. This certificate applies to the aforementioned Technical Report.

3. I hold the following academic qualifications:

   B.Sc. Mining Engineering, University of Missouri at Rolla (formerly Missouri School of Mines).

4. I am a Registered Professional Engineer in the State of Colorado, registration #22310. I am also a member in good standing with the following technical associations:

   - SME – Professional Legion of Honor Member #03042460.
   - CIM – Member # 97865.

5. I have worked as a mining engineer in the minerals industry continuously for 47 years.

6. I have read NI 43-101 and Form 43-101f1. By reason of education, experience, and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101.

   I have worked as a mining engineer for more than 47 years, including without limitation, 6 years as Manager Mining and Exploration for Tenneco Oil Company – Minerals Division, 4 years as Vice President Mining for Standard Metals Corp., 8 years as Development Group Vice President for Echo Bay Mines Ltd, 21 years as an independent mining consultant, and 5.75 years as Managing Director – Metals for the John T. Boyd Company. During the course of my career, I have been responsible for managing operating mines and process plants, directing resource and reserve estimates, detailed mine design for surface and underground mines, economic analysis of all aspects of mining projects, and due diligence reviews and valuations of mineral projects in 37 countries. My commodity experience includes gold, silver, copper, lead, zinc, manganese, nickel, uranium/thorium, talc, borate minerals, salt, and limestone.

7. I have read this Technical Report and believe it conforms in content and format with the requirements of form NI 43-101f1.

8. I have visited the Laiva Gold Project site on multiple occasions from 15 – 16 March 2017 with the most recent being the period from 27 August – 7 September 2021.

9. With the exception of being a co-author of an NI 43-101 TECHNICAL REPORT MINERAL RESOURCE ESTIMATE LAIVA GOLD MINE PROJECT with an Effective Date of 9 August 2017 and a co-author of an NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LAIVA GOLD MINE PROJECT with an Effective Date of 6 April 2018 as an independent consultant to Fire Steel Resources Inc., a predecessor in interest to OTSO, I have no prior association with the Laiva Project, which is the subject of this Technical Report.

10. I am independent of Otso Gold Corp. and its subsidiaries. I hold no beneficial interest in any of the foregoing.

11. I am the principal author of Sections 1.0 through 13.0 and 15.0 through 27.0 of this report. I have collaborated with my co-author on all relevant other sections of this Technical Report.

12. As of the effective date, 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 this Technical Report not misleading.

Report dated this 1st day of November 2021 with an effective date of 4 October 2021.

Dated 1 November 2021
(Signed and Sealed)

Gregory B. Sparks, B.Sc., Professional Engineer,
Managing Director – Metals
John T. Boyd Company

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29.0 DATE AND SIGNATURE PAGE

JOHN T. BOYD COMPANY
By:

J.R. Kelso,
(Not a QP under NI 43-101,
but +35 Years of Experience)
Chief Metallurgist

Report Date: 1 November 2021
Effective Date: 4 October 2021

SME
Society for Mining, Metallurgy & Exploration
Samuel J. Shoemaker Jr
SME Registered Member No. 2941320
Signature
Date Signed 10/13/2021
Expiration date 12/31/2021
Area of Competency Mining & Resources

Sam J. Shoemaker, SME Registered Member
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Project Manager - Metals

Report Date: 1 November 2021
Effective Date: 4 October 2021

COLORADO REGISTERED
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22310
PROFESSIONAL ENGINEER

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