NI 43-101 TECHNICAL REPORT
PRELIMINARY ECONOMIC ASSESSMENT
LAIVA GOLD MINE PROJECT

Raahe, Finland

Prepared For

FIRESTEEL RESOURCES INC.

Vancouver, B.C., Canada

By

John T. Boyd Company

Mining and Geological Consultants

Denver, Colorado, USA

Report No. 3832.001

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Firesteel Resources Inc.
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CANADA

Attention: Mr. Michael Hepworth, CEO

Subject: NI 43-101 Technical Report
Preliminary Economic Assessment
Laiva Gold Mine Project
Raahe, Finland

Dear Sirs:

Firesteel Resources Inc. (FTR) has requested that the John T. Boyd Company (BOYD) complete a preliminary economic assessment (PEA) on the Laiva Gold Project located near Raahe, Finland. BOYD is pleased to present the results of this study in the attached technical report that complies with Canadian National Instrument 43-101 (NI 43-101).

In August of 2017 (revised October 2017), BOYD completed our initial NI 43-101 Technical Report which provided our independent mineral resource estimate for the Laiva Project. This report is a continuation of our prior work.

The Laiva Project includes two existing shallow open pit mines located near an existing process plant. The process plant is configured as a crush, grind, CIL facility with attendant CN destruction and tailings storage facility. The name plate capacity is 250 metric tons per hour (tph). The process plant was of entirely new construction and was commissioned in February 2012. The plant operated through March 2014. Since that time, the plant has remained on care and maintenance on a well-maintained basis.

The project as originally configured by the previous operator proved to be sub-economic owing in part to financial issues incurred by the prior operator, but perhaps more significantly as the result of inaccurate previous resource modelling and attendant mining practices which resulted therefrom.

The recent operator sought to mine the resource by large scale open-pit bulk methods. This mining practice resulted in massive mining dilution of the actual mineralized zones, which manifested as significantly lower than expected head grade to the process plant.

The existing process plant designed by Metso remains in near-new condition and is generally well suited to processing the Laiva resource. The grinding circuit consists of an autogenous grinding mill followed by a pebble mill. However, mills of these types require a stable blend of feed to perform at design levels. Previous mine production was quite variable as to nature of feed (varying in proportion from un-mineralized granite [hard] to well mineralized metavolcanics [soft]). This feed variability caused the mills to significantly underperform design feed rates, except during brief periods. Also, total utilization figures for the process plant as reported by the former operator, achieved only 81.2%. This substandard plant performance was the direct consequence of the reoccurring processing upsets, resulting primarily from variable feed (rock-type). Insufficient routine maintenance brought about by financial distress of the previous operator also contributed to this exceptionally low plant utilization.

In preparing the current PEA, BOYD has developed annual operating plans with supporting cost estimates to allow the overall economic value of the project to be determined on a preliminary basis. The PEA technical report describes the results of this study, as well as providing recommendations for future work. BOYD is hereby providing the attached Technical Report – Preliminary Economic Assessment to comply in form and content with NI 43-101 standards.

Respectfully submitted,

JOHN T. BOYD COMPANY
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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 from 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 mineralized material 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 mineralized material 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 mineralized material in the mining process, thereby reducing the mined grade of the mineralized material.
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. As a verb, it refers to selective mining of the best mineralized material 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.
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.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

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.
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.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

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 mineralized material 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.
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.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

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 mineralized material grade.
SG	:	Specific Gravity.
SMV	:	Selective mining unit.
SRK	:	SRK Consulting, an independent international consulting group focused on advice and solutions to earth and water resource industries
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.

GLOSSARY OF ABBREVIATIONS AND DEFINITIONS - Continued

μm	:	Micron (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

FTR is planning on developing the Laiva Gold Project and bringing it into production. FTR is a Canadian public company listed on the TSX-V stock exchange, and the PEA of the Laiva Project is considered a material event. This is the second of our NI 43-101 Technical Reports on the Laiva Gold project. On (Revised) 17 October 2017, BOYD issued our first Technical Report: Resource Estimate, Laiva Gold Project, Raahe, Finland addressed to FTR. Where appropriate and for completeness of this report, sections of our October 2017 Technical Report are duplicated in this document.

FTR engaged BOYD to utilize our initial report as a basis to prepare a second NI 43-101 Technical Report providing our PEA of the Laiva Project. BOYD does not now have, nor has it previously had any beneficial interest in FTR nor any of its subsidiaries. This report is prepared on a fee-based arrangement in accordance with BOYD's current standard commercial rates. Payment of fees is in no way contingent upon the results of our resource estimate or other conclusions contained in this Technical Report.

Conclusions and recommendations contained in this report represent BOYD's independent professional opinions which are based on information available at the time of this Technical Report.

1.2 Property Description, Location, 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 south of Oulu, which is the closest regional center. Helsinki is located approximately 600 km to the south. (see Figure 1.1, Page 1-2).

Figure 1.1
Laiva Project Location (Source Laiva Gold Project)

The Laiva Gold Project is comprised of two license areas, Laiva and Oltava. The advanced stage Laiva Gold Mine and satellite exploration projects at Mussuneva and

Kaukainen (Figure 1.2) are located within the Laiva area. Oltava is located 12 km south of Laiva and is an early stage exploration property (Figure 1.2). Two shallow surface mines, a 6,000 t/day (tpd) process plant, tailings storage facility, and attendant infrastructure are situated within the license area.

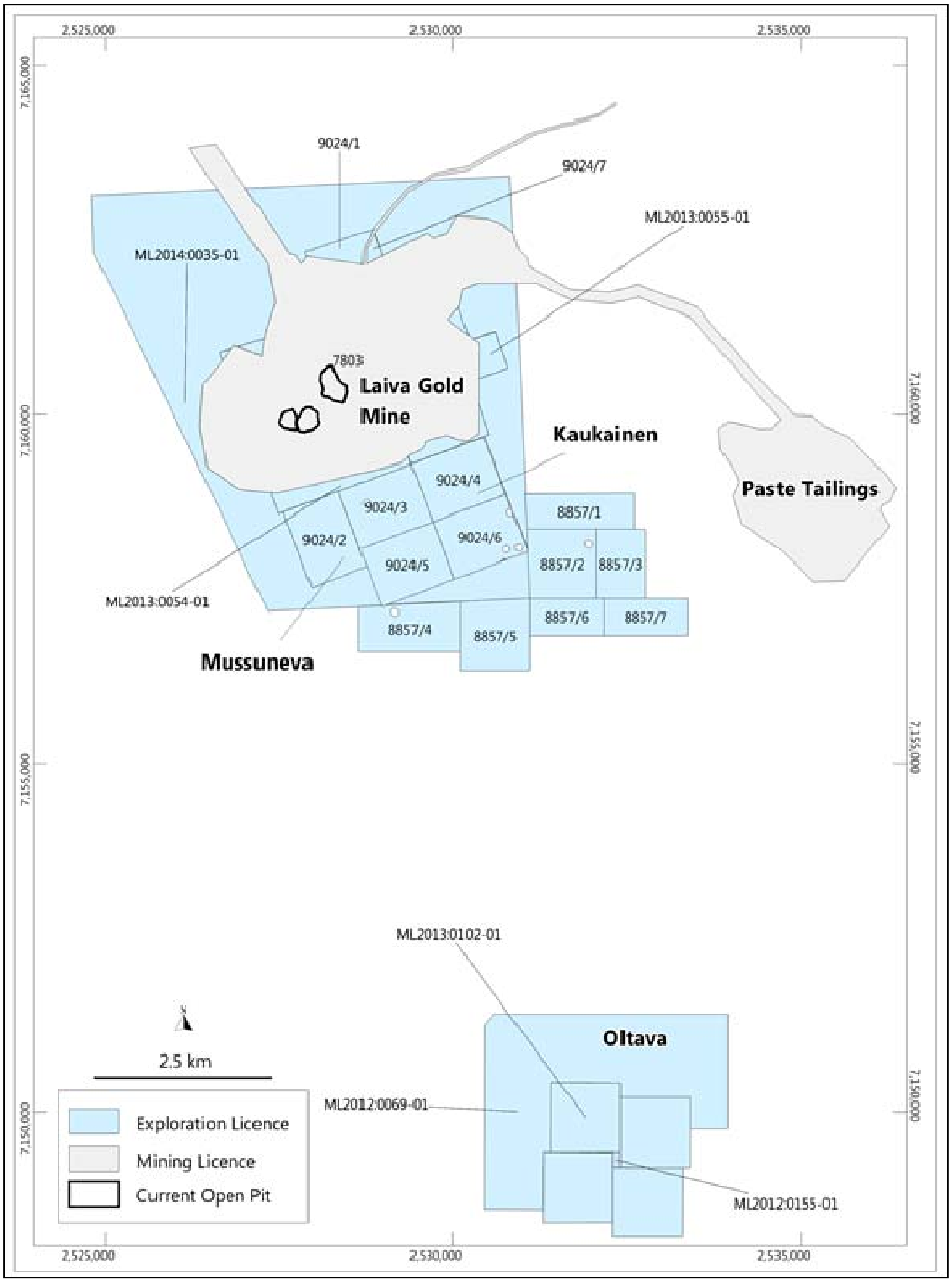

Figure 1.2

Laiva Gold Project Mineral Rights and Registration Numbers, Laiva and Oltava Properties

Source: Laiva Gold Project

The Laiva Gold Project is wholly owned by Nordic Mines Oy (NORDIC), a private Finnish company which is owned 100% by FTR.

1.3 Accessibility and Physiography

The Laiva Gold Project is readily accessed from the town of Raahe, which is located 20 km northwest of the project and has a population of approximately 20,000. Raahe 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 HelsinkiAlternatively, Raahe is easily accessed by rail or air from Helsinki to Kokkola and then by vehicle from Kokkola to Raahe.

From Raahe, the project is reached via public paved roads (Road 88) to the village of Kopsa (see Figure 1.3). A paved road from Kopsa leads to the entrance to a 4-km 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.

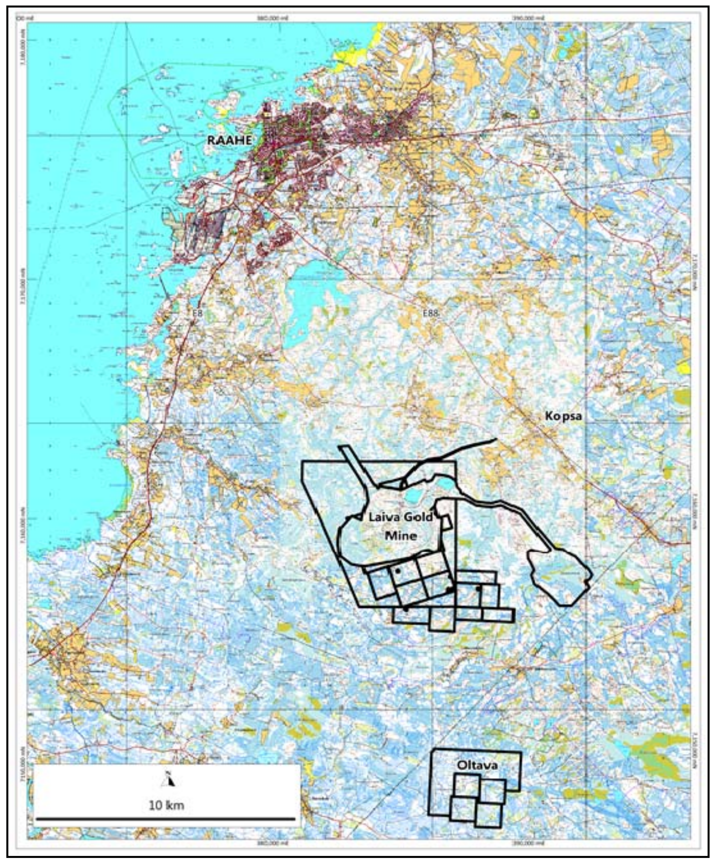

Figure 1.3
Laiva Gold Project License Areas Access
Source: Laiva Gold Project

The project area is generally flat lying with gentle topography within an area of Boreal Forest (Figure 1.4). Broad, shallow depressions contain peat bogs and the area is incised by shallow streams. Elevation of the land surface varies between 30 meters above sea level (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 clearings and boggy areas. An aerial photograph of the Laiva Gold Mine site showing physiography and infrastructure, is provided on Figure 1.4:

Figure 1.4
Aerial Photo of the Laiva Gold Mine Showing Physiography and Infrastructure
Source: NORDIC, 2013)

1.4 Project History

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. In 1999, the area was acquired by Endomines Oy (Endomines) who explored the project until it was acquired by NORDIC in 2005.

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.

NORDIC commissioned independent consultants to complete several mineral resource and reserve estimates between 2008 and 2015. NORDIC released two mineral resource estimates and a mineral reserve estimate prepared in accordance with JORC (2012) by SRK in 2013 and 2015, and by 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 Laiva Mine commenced production in January 2012; 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.

Mining was conducted from two open pits: North Pit and South Pit. The North Pit currently measures approximately 500 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 are currently flooded. An area measuring 260 by 240 m and located immediately west of the South Pit, has been stripped to approximately 5 m deep, but there has not been any production from this area to date.

Total gold production was 2,241 kg from 2.8 million tons of mineralized material processed. This equates to an average life-of-mine (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 tonnes of sulphide-rich waste, was produced. The resulting average strip ratio was 1:2-8 (tonne mineralized material: average number of tonnes of waste). The mine operated at an average all-in cost of approximately $1,760 per ounce gold.

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 in approximately 1900Ma. The contact between the Karelian Craton and the Svecofennian orogenic belt is marked by the deep, crustal scale, northwest trending Raahe-Ladoga suture. This geologic feature is located approximately 50 km north of the Laiva gold deposit and can be traced through Finland into Sweden.

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 dykes. These Svecofennian units are variably folded and faulted, and display lower greenschist through to amphibolite facies metamorphism. Younger mafic dykes 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 Laiva-Oltava project area comprises 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. 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 dykes, and later mafic (dolerite) dykes.

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.
The Laiva Gold Mine is comprised of mafic meta-volcanic (termed Uralite Porphyrite by NORDIC) intercalated with rare meta-volcanosedimentary horizons and intruded by quartz diorite plutons and dykes. Together, the metavolcanic and quartz diorite are the main host to mineralization. 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.

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 dykes, sills, and plutons; however, relatively minor narrow (<2 m thick) pre-mineral pegmatite dykes 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 to the northeast and truncates mineralization at depth.

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

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

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 dykes. Parasitic folding of granite dykes, 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 that offset mineralization.

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. Syn-mineral quartz-sulphide veins occupy mylonitic shear zones with a steep dip (75 degrees to 85 degrees) 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.

Mineralization comprises 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.

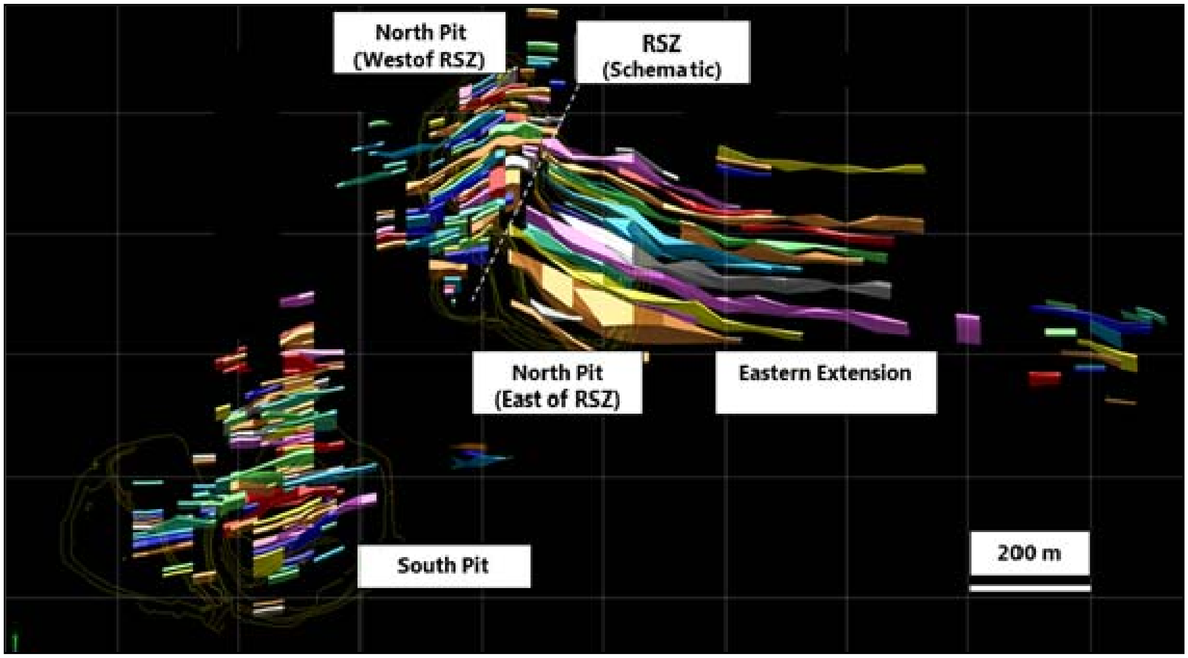

Figure 1.5
Plan View of Mineralized Shear Zones Forming the Laiva Gold Deposit.
RSZ – Red Shear Zone

Source: BOYD

1.6 Mineral Resource Estimate

The scope of the current technical report is to prepare a PEA of the Laiva Project. Mineral resources reported in our October 2017 Technical Report were used as the basis of the current mine plans used in this PEA. The October 2017 mineral resource is summarized in Table 1.1.

Table 1.1
Laiva Project Mineral Resources
Laiva Open-Pit Constrained Mineral Resource Estimate

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

Notes:

1. The mineral resources presented above are reported within an economic pit shell generated by the Whittle Pit Optimization software.

2. 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.
3. The mineral resources presented here were estimated using a block model with a block size of 9 m by 9 m by 9 m sub-blocked to a minimum of 3 m by 3 m by 3 m using ID^3 methods for grade estimation. Mineral resources are reported using an open-pit gold cut-off of 0.40 g/t Au.
4. 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 November 27, 2010.
5. The effective date for this mineral resource estimate is August 5, 2017.
6. The mineral resources presented herein are in-situ and other than an economic pit shell no attempt has been made to apply a mining dilution or a mining recovery factor to them.
7. This mineral resource estimate was completed by Sam J. Shoemaker, Jr. Registered Member SME and a Qualified Person under NI43-101.

1.7 Mineral Processing

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 autogenous grinding (AG) and ways of processing mineralization by gravimetric concentration followed by cyanide leaching, and possibly flotation. The objective was to find the process having the lowest costs with a reasonable gold recovery.

The 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.
3. Gravity separation.
4. Fine flotation.
5. Thickening the produced process products for further leaching tests.
6. Bench-scale cyanide leaching 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.

Results from the pilot scale test program confirmed the following:

- AG could be utilized to achieve the required mineralized material particle size.
- Flash flotation and gravity concentration could be utilized to concentrate the higher-grade minerals that could be treated in a separate, more aggressive cyanide leach circuit.
- Concentrate tailings were amenable to direct cyanidation under standard leaching conditions.

Test results from the pilot scale test program were utilized to design the Laiva Gold Mine Process facilities, which were constructed in 2011 and operated for a total of 17 months. During this period, the facility experienced 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 a few minor modifications to enhance productivity and improve key process parameter measurement and recording systems.

Laiva processing facilities start at the point where mineralized material from the open pit is transported by haul trucks to the crushing station and crushed by the primary crusher (Metso C 160 Jaw crusher) to - 200 mm. The capacity of crushing plant is 700 tph. Crushed mineralized material is stored at the stockpile. The capacity of the mineralized material stock pile is 25,000 tons.

From the stockpile, mineralized material is fed through vibrating feeders to a belt conveyor, for transport to primary grinding mill (AG Mill). Secondary grinding is done in a Pebble Mill. The discharges from the AG and Pebble mills are classified first by a scalping sieve and secondly by a cyclone cluster. A gravity separator and Flash Flotation unit produce a high-grade gold-concentrate, which is treated in a Carbon in Leach (CIL) plant (high grade). The gold in the cyclone overflow is recovered in a separate CIL plant (low grade).

A metallurgical test program has been initiated by the Laiva process staff under the direction of the BOYD project team. The objective of this program is to provide information to aid in the planning of plant commisioning and the initial period of operation following restart.The samples utilized for the test program are composites of drill cuttings collected during the recent drill program. These composites are considered to be representative of the expected plant feed for the initial years of the restart of mine operations.

The schedule to complete this test program will provide sufficient time to allow evaluation of the test results and implementation of any further identifed process plant improvments.

The test program inlcudes:

- Complete head analyses with gold assay by size.
- XRD, XRF, multiple-element ICP, and optical minerlaogy.
- Lab scale flotation concentration of each sample.
- Agitated reactor canide leaching of concentrates and tailings samples.
- Geochemical, mineralogical analyses and gold assay by size of chosen metallurgical products.

1.8 Mining

The Laiva Project is composed of three mining areas: the previously developed South pit area in the southwestern portion of the project; the previously developed North pit area in the northern portion of the project; and the East Pit area located east of the North and South pit areas. These three mining areas develop the same mineral resource and are shown in Figure 1.6.

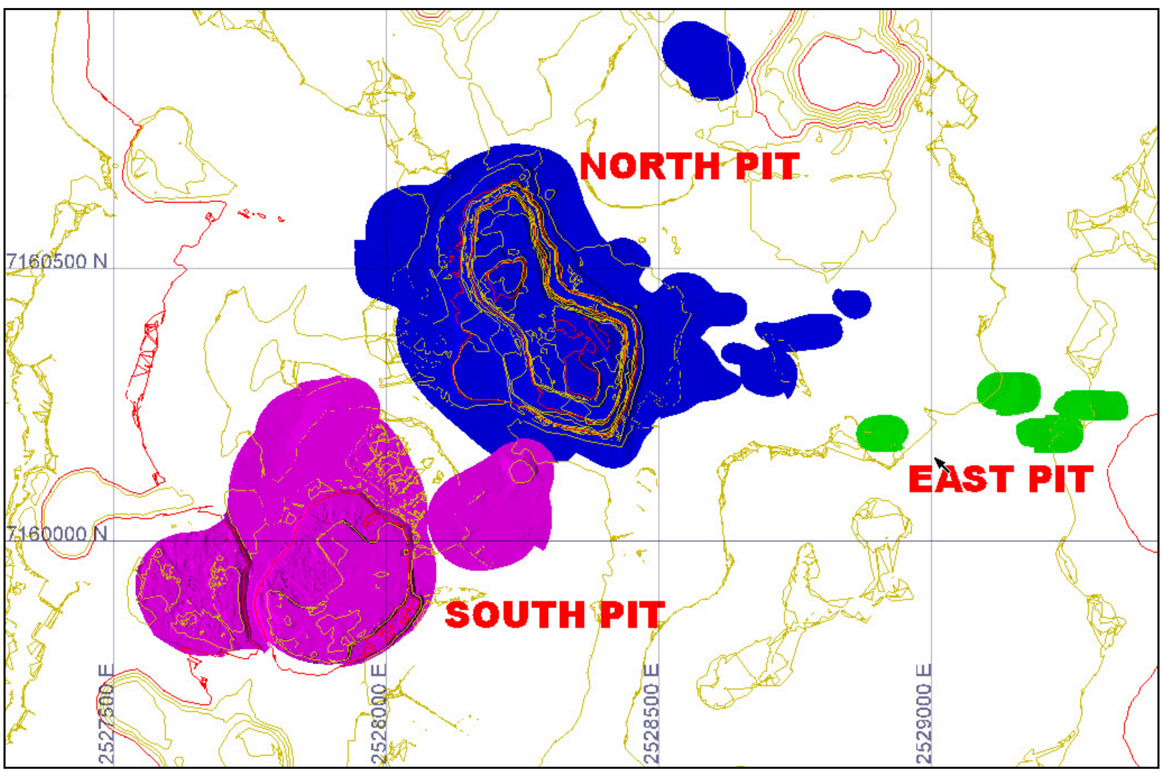

Figure 1.6
Laiva Project Mining Areas

Pit shells were determined for the Laiva deposit from a Whittle economic pit optimization program. The Whittle pit shells described above were used to construct individual pit phases at each of the three mining areas. The phases were designed to provide high-value early phases as well as balancing waste stripping over the entire life of each pit. All of the phased designs used the same geotechnical and design criteria previously described. Based on these initial pit optimization results, a pit shell was selected to design the phased pits that make up the three mining areas for the PEA.

A mine contractor will be used for most mining activities, including: site preparation, haul road construction and maintenance, bulk waste drilling and blasting, excavation and haulage of waste, management of waste dumps, oversize breakage, and pit dewatering. Nordic Mines will perform drilling and blasting, as well as mineralized material loading into haul trucks supplied by the mine contractor. The mine contractor will provide the open-pit equipment, with the exception of zone blast hole drilling and mineralized material loading equipment. Responsibilities of the contractor include providing 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, as well as for contract maintenance of Nordic equipment. Nordic will provide pit technical services including blast design, blasthole layout, mineralized material grade control, mine planning, and blasthole sampling. Specialized contractors will provide explosives storage on site. Explosives, blasting agents, fuel, and other consumables will be supplied by established third-party vendors.

A mine production schedule was prepared using Maptek's Chronos scheduling software. Mineralized material selection was based on the economic cutoff (0.471 g/t). Additionally, below cutoff, mill incremental material (0.401 g/t to 0.471 g/t) was also considered as millfeed in the production schedule. The production schedule has no pre-production period, but does assume a reduced millfeed requirement during the first two years of production (1,500,000 tonnes in year 1 and 1,750,000 tonnes in year 2). After year 2, the mill is assumed to operate at full capacity (2,000,000 tonnes annually). Other than a small amount of material at the mill, no stockpiling was used in the production schedule. Table 1.2 and Figure 1.7 (which follows) show the annual mine plan for the Laiva Project.

Table 1.2
Laiva LOM Annual Production Schedule

Project Year	1	2	3	4	5	6	Total
Mill Tonnes	1,500,000	1,750,000	2,000,000	2,000,000	2,000,000	1,586,000	10,836,000
Gold Grade (g/t)	1.593	1.391	1.422	1.343	1.471	1.515	1.449
Contained Troy Ounces	76,800	78,300	91,400	86,400	94,600	77,300	504,800
Recovered Troy Ounces	69,500	70,900	82,600	78,100	85,500	69,900	456,500
Waste Tonnes	11,306,000	12,266,000	15,000,000	15,000,000	14,905,000	4,522,000	72,999,000
Total Tonnes	12,806,000	14,016,000	17,000,000	17,000,000	16,905,000	6,108,000	83,835,000
Stripping Ratio	7.54	7.01	7.50	7.50	7.45	2.85	6.74

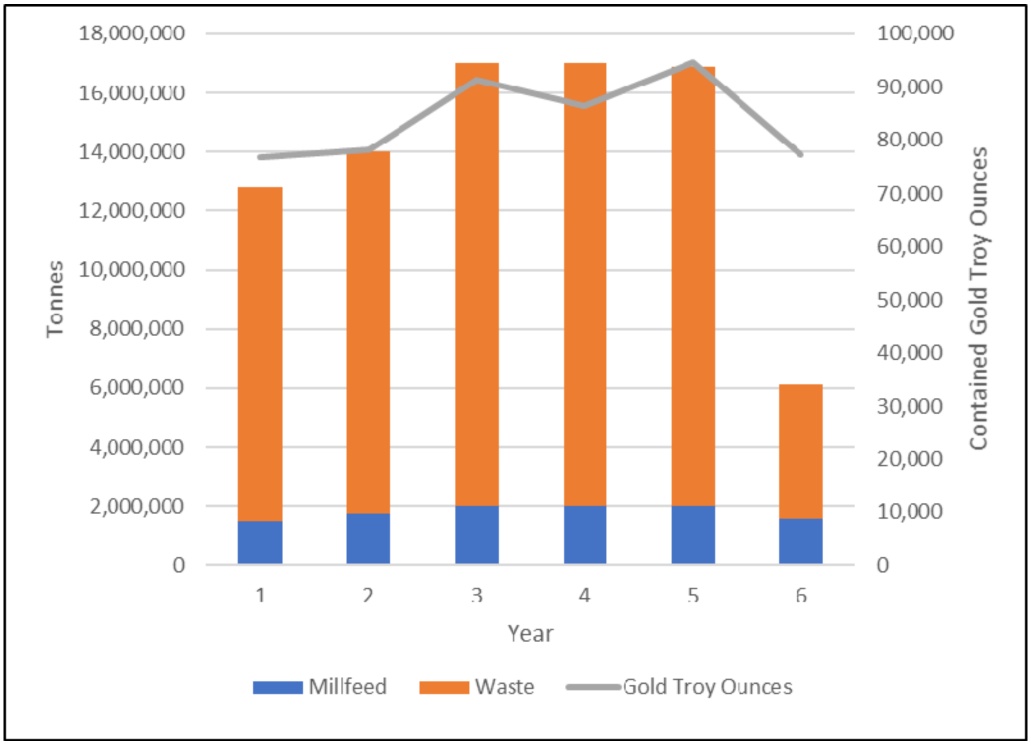

Figure 1.7
Laiva LOM Production Schedule

1.9 Economic Results

Three economic scenarios were considered in the economic analysis:

1. A pre-tax scenario to examine the economic results of the pre-tax cash flows.
2. An after tax scenario that assumes prior existing losses can be carried forward to offset income taxes.
3. An after tax scenario that assumes prior existing losses cannot be carried forward to offset income taxes.

The corporate tax in Finland is set at a rate of 20% of taxable income, after deducting depreciation depletion, and amortization. A carried forward loss of US$133,648,840 is available subject to approval from the taxing authority. BOYD's analysis assumes two different scenarios; one where the carried forward loss is available to offset the corporate income tax, while the other assumes that this loss is not allowed to offset the corporate income taxes.

LOM operating cost estimates are shown in Table 1.3.

Table 1.3
LOM Operating Costs

Section	Units	Cost (US$)
Mining	$/t Mineralized Material	22.69
Processing	$/t Mineralized Material	12.07
Infrastructure	$/t Mineralized Material	0.29
Indirects	$/t Mineralized Material	0.15
G&A	$/t Mineralized Material	1.94
Total Operating Cost	$/t Mineralized Material	37.14

Figure 1.8 (following) shows the cash operating costs on an annual basis over the LOM period. Operating costs gradually increase during the first two years during the plant ramp up, then remain steady during the full production years that follow.

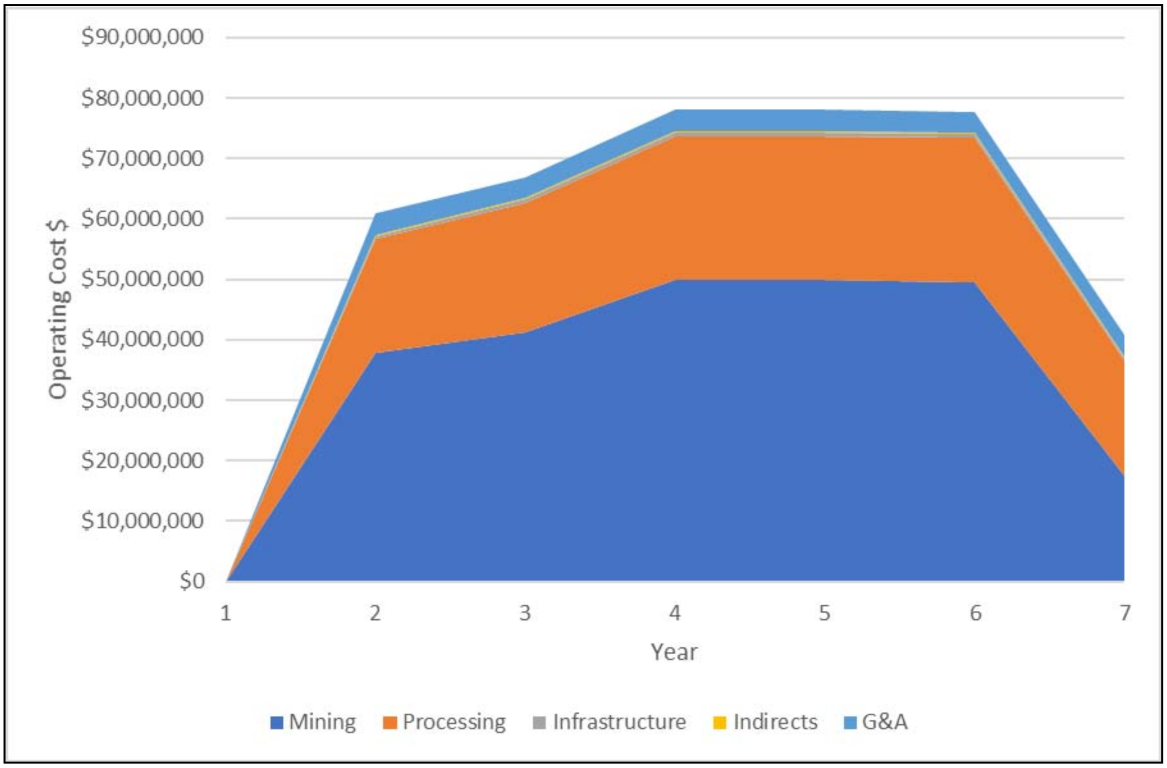

Figure 1.8
Annual Operating Costs

The project has minimal capital requirements during pre-production, as well as sustaining capital. Capital costs are shown in the Table 1.4.

Table 1.4
Capital Costs

Item	Pre Production (US$)	Sustaining (US$)	Total (US$)
Mine	1,068,000	2,062,000	3,130,000
Processing	2,923,000	-	2,923,000
Infrastructure	377,000	2,270,000	2,647,000
Indirects	130,000	-	130,000
G&A	2,171,000	-	2,171,000
Contingency	446,000	262,000	708,000
Total	7,115,000	4,594,000	11,709,000

Table 1.5 summarizes the life-of-mine cash flows for the project, while Table 1.6 presents the annual cash flow schedule.

Table 1.5
LOM Cash Flow Summary (US$)

Net Revenue Less Royalties	591,488,000
Operating Costs	402,467,000
Operating Margin	189,021,000
Capital Expenditure	11,711,000
Pre-Tax Cash Flow	177,310,000
Tax Payable	1,118,000
Net Cash Flow after Tax	176,192,000

Please note that the tax payable listed in Table 1.5 includes a significant carried forward loss of US$133,648,840 million.

Table 1.6
Annual Project Production and Cash Flow

Year:	-1	1	2	3	4	5	6	Total
Mineralized Material Tonnes	-	1,500,000	1,750,000	2,000,000	2,000,000	2,000,000	1,586,000	10,836,000
Waste Tonnes	-	11,306,000	12,266,000	15,000,000	15,000,000	14,905,000	4,522,000	72,999,000
Total Tonnes	-	12,806,000	14,016,000	17,000,000	17,000,000	16,905,000	6,108,000	83,835,000
Strip Ratio	-	7.54	7.01	7.50	7.50	7.45	2.85	6.74
Gold Grade (Au g/t)	-	1.59	1.39	1.42	1.34	1.47	1.52	1.45
Gold Recovery (%)	-	89.9	90.4	90.4	90.4	90.4	90.4	90.3
Recovered Gold (Troy Ounces)	-	68,900	70,600	82,500	77,900	85,300	69,700	454,900
Revenue	-	89,597,000	91,789,000	107,233,000	101,293,000	110,943,000	90,633,000	591,488,000
Operating Cost (US$)	-	60,849,000	66,979,000	78,047,000	78,043,000	77,784,000	40,765,000	402,467,000
Capital Cost (US$)	7,115,000	724,000	1,259,000	830,000	1,259,000	262,000	262,000	11,711,000
Pre-Tax Cash Flow (US$)	-7,115,000	28,024,000	23,551,000	28,356,000	21,991,000	32,897,000	49,606,000	177,310,000
Tax (US$)	-	-	-	-	-	-	1,118,000	1,118,000
After-Tax Cash Flow (US$)	-7,115,000	28,024,000	23,551,000	28,356,000	21,991,000	32,897,000	48,488,000	176,192,000

The undiscounted base case cash flow demonstrates that the project is projected to provide a very favorable operating margin of 32%.

The cash flow was then evaluated at a discount rate of 5% per year, with comparative results presented over the range of annual discount rates of between 5% and 10%, as shown in Table 1.7.

Table 1.7
Results of Evaluation

Discount Rate (%)	Pre-Tax NPV (US$)	After-Tax With Loss	After-Tax Without Loss
5	91,540,000	90,723,000	68,965,000
8	78,272,000	77,597,000	58,202,000
10	70,642,000	70,043,000	52,026,000

Internal rates of return (IRR) before, after tax with carried forward losses, and after tax without carried forward losses are 44.6%, 44.4%, and 36.5% respectively. At 5% per year, the discounted cash flow after tax with carried forward loss shows a payback period of 1.7 years. The after tax without carried forward losses shows a payback period of 2.1 years.

It should be noted that results of this PEA are preliminary in nature. For example, the LOM plan includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the conclusions of the PEA will be realized.

1.10 Sensitivity Analysis

Figures 1.9 and 1.10 (Page 1-18) shows the sensitivity of the project after-tax with carried forward losses and after-tax without carried forward losses for a cash flow discounted at 5% (Net Present Value $[NPV]_5$) to a variation over a range of 30% above and below the base case in: (1) gold prices, (2) operating costs, and (3) capital expenditure.

As might be expected, the project is most sensitive to changes in gold price and operating cost with gold prices less than US$1,037 per troy ounce generating negative NPV. Operating costs greater than 125% of the base case also show a negative NPV.

The project is less sensitive to capital costs. There is little to no impact varying the capital costs from 70% to 130% of the base case.

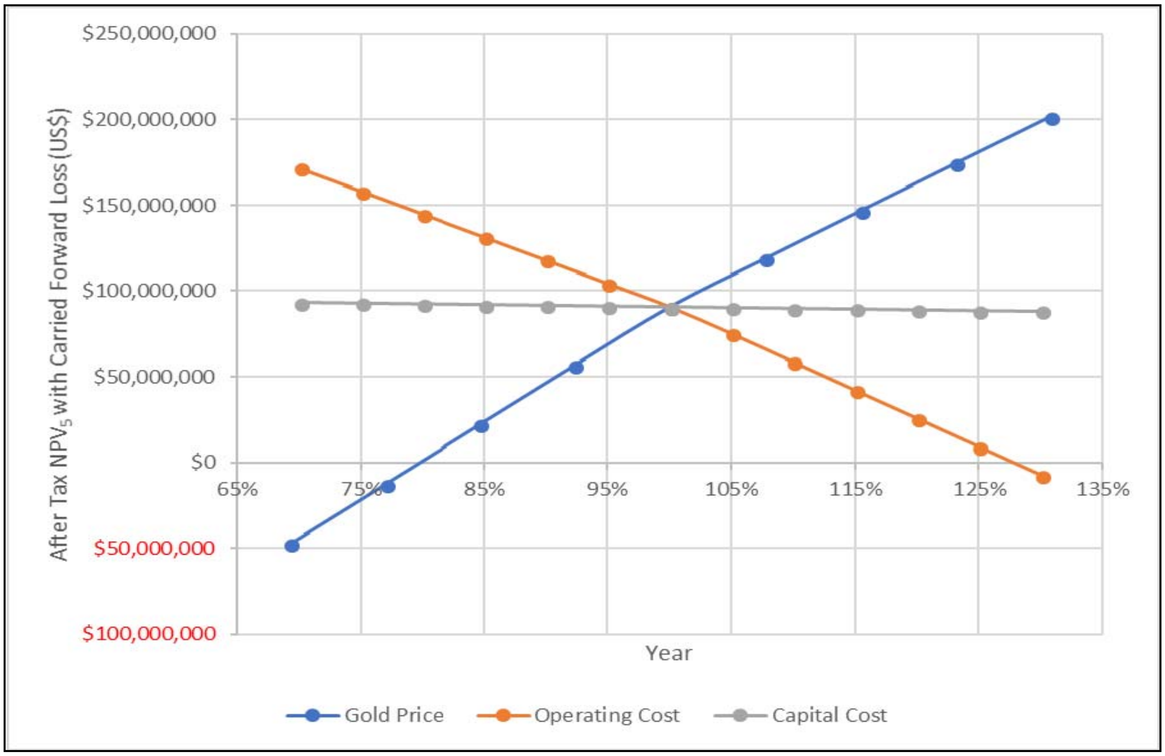

Figure 1.9
After Tax with Carried Forward Losses NPV Sensitivity Chart

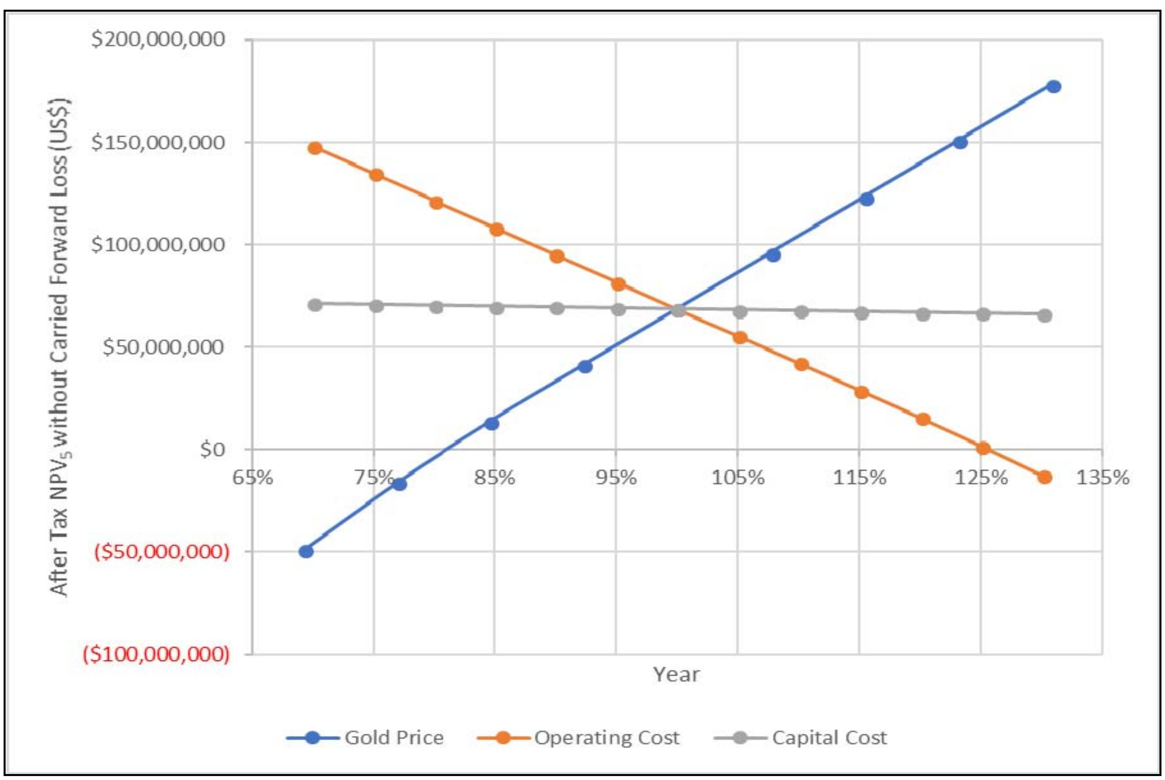

Figure 1.10
After Tax without Carried Forward Losses NPV Sensitivity Chart

1.11 Conclusions

The Laiva Gold Project includes the advanced stage Laiva Gold Mine 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, and intruded by post-mineral granitoid and later dolerite stocks and dykes.

Previous metallurgical test work conducted on the Laiva deposit, indicates that gold is typically fine-grained, free milling, and amenable to cyanide leach processing. The key findings of the PEA are:

- The plant operated generally as expected given the characteristics and grade of mineralized material feed supplied to the plant. However, plant performance was significantly impaired by mining practices, which resulted in excessive mine dilution issues that were further exacerbated by reliance on the previous resource model (which lacked necessary definition of mineralized zones).
- A detailed evaluation of several phases of metallurgical test work results and operating data from previous operations, was completed to provide the basis for the estimated point forward gold recovery of 90.3%.
- The existing Laiva process facilities with the planned minor modifications, will be capable of achieving the planned production rates for future operations which start at 1.5 million tonnes per year for year 1, ramping up to 2.0 million tonnes per year by year 3.
- As the process facilities are existing and detailed operating information is available from previous operations, the estimated capital and operating costs for process unit operations should be considered to be more accurate than a typical PEA.

Based on the Laiva PEA results, BOYD concludes the following on a preliminary basis:

- The project demonstrates good economic robustness producing an NPV_8 of US$77.6 million with an IRR of 44.4%.
- A minimal capital investment of US$11.7 million (pre development and sustaining capital) is required for the project.
- Cash operating costs over the life of the project average US$884.74 per troy ounce of gold.
- The project has very limited sensitivity to capital costs.
- The project shows significant sensitivity to gold price, as well as operating costs.

On the basis of this PEA, BOYD concludes that exploitation of the gold resources in the Laiva Project area could provide attractive economic returns, and that further development is warranted.

1.12 Recommendations

To advance the project, BOYD recommends that a series of infill drilling, metallurgical testing, and optimization trade-off studies are conducted in support of a Pre-Feasibility Study prior to recommencement of mining operations. In addition, further exploration work should be commenced in advance of or concurrent with recommencement of operations. The principal work recommendations are discussed below.

1.12.1 Infill Drilling

Infill diamond core drilling should be performed at a 25 m spacing, prioritized to upgrade near-mine Inferred mineral resources for conversion to Probable Reserves, if warranted, from the results of the infill drilling. The program should focus on areas beneath the existing North and South Pits, and east of the existing North Pit. Approximately 20,000 m of infill drilling is recommended.

Infill diamond drill holes should be utilized to collect oriented drill core. This will allow better understanding of the structural controls on, and rake of, mineralization.

Implementation of a company standard operating procedure is recommended to detail the methodologies used for drilling, logging, sampling, and QAQC. An improved QAQC program should be considered to include the following:

- Dispatch samples in batches of 20 samples, comprising 16 core samples, 1 certified reference material (CRM), 1 blank inserted after a suspected mineralized sample wherever possible, a field duplicate sample and a laboratory duplicate sample.
- The CRM and blank samples should be inserted blind to the laboratory.
- All QAQC results should be assessed at the time analytical results are returned, and each batch of samples should be passed or failed based on set control limits prior to entry into the master database.

BOYD understands that this program is underway.

1.12.2 Process

BOYD recommends the following in the process area:

- Utilize results from ongoing metallurgical test program to develop detailed commissioning plans for all process unit operations.
- Develop metallurgical balance with adequate detail for all process related planning and reporting requirements.
- Plan detailed sampling and analyses program for the initial operating period to provide results required to optimize the split between low-grade and high-grade leaching circuits.
- Plan and execute lab scale testing of loaded and stripped carbon samples to evaluate potential benefits of warm acid rinse and cold cyanide carbon strip operations.

1.12.3 Pre-Feasibility Study

Building on the results of this PEA report, a NI 43-101 compliant Technical Report – Pre-feasibility Study is recommended to:

- Support conversion of Mineral Resources to Mineral Reserves.
- Provide the formal structure for the trade-off studies previously recommended.
- Complete more detailed engineering of mining plans.
- Complete detailed engineering of flow sheet modifications, if any, determined by process trade-off studies.
- Compile capital and operating costs and analyze detailed project economics.

The recommended Technical Report is characterized as a Pre-Feasibility Study, notwithstanding the proposed detailed engineering of mining plans and flow sheet optimization, as capital estimating will be relatively minor. This is because mining, except for drilling and blasting and loading which are to be self-performed, is expected to be contracted on a unit cost basis, and thereby likely to require minimal mine capital equipment ascribable to the Joint venture, including FTR. Additionally, the process plant is already constructed and is expected to require only modest capital investment associated with optimization efforts.

2.0 INTRODUCTION

2.1 Client Name and Purpose

This Technical Report is prepared for FTR to provide preliminary guidance regarding the potential economic viability of the Laiva gold mining project for use by FTR with respect to further investment decision making with respect to the Laiva Project.

2.2 Terms of Reference

BOYD was commissioned by FTR to prepare a PEA in the form of a Technical Report conforming to NI 43-101 standards for filing with the TSX- V stock exchange. FTR recently acquired a 100% interest in the Laiva Gold Project, near Raahe, Finland. An NI 43-101 compliant Technical Report estimating resources for the Laiva Project was completed by BOYD with an Effective Date of 9 August 2017, a Report Date of 18 August 2017 and Revision Date of 17 October 2017 (October 2017 Report).

BOYD has been engaged on an ongoing basis to provide technical review and advice (recommendations) to FTR in connection with its plans for recommissioning production from the Laiva Gold Project including without limitation, the following aspects of the project:

- Additional in-fill drilling.
- Mining procedures to minimize mining dilution which was a significant factor in the previous operator’s failure.
- Revisions to process strategies and practices to increase mill capacity and improve metallurgical recovery.
- Organizational structure and staffing schedules in order to be consistent with similar operations of similar scale.
- Enhancement of the tailings transport and deposition.

In connection with the foregoing, Mr. Gregory B. Sparks, Managing Director – Metals, Mr. Jody R. Kelso, Associate Senior Metallurgist, and Mr. Christopher C. Wilson, Associate Senior Geologist visited the Laiva Project site on multiple occasions since August 2017 through April 2018. The engagement to provide these services has provided BOYD personnel with an in-depth understanding of the Laiva Gold Project in its role as an independent consultant.

BOYD is a leading international mining and geologic consultancy providing consulting services since 1943. The company has extensive experience in preparing Qualified Person and Competent Person Reports for filing with stock exchanges around the world. The authors of this Technical Report, including Dr. Christopher Wilson, MAusIMM (CP), MSEG, Sam Shoemaker, SME Registered Member, and Gregory B. Sparks, P. Eng., are Competent Persons for deposits of the nature of the Laiva Gold Project.

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3.0 RELIANCE ON OTHER EXPERTS

Information reviewed during preparation of this report included historic data for the project remaining from the previous operator, Nordic Mines Oy (NORDIC), as well as direct observations made by members of the BOYD project team. The data contained in the historic files remaining from NORDIC included reports prepared by others, as well as operating records and other information developed by NORDIC. While these data provided background and general context for the Laiva Project, there was no reliance on these data in BOYD's preparation of the primary subject of this Technical Report, except as hereinafter specifically cited as a reference listed in Section 27. This notwithstanding, BOYD did review the following key documents prepared by others:

- 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.

Much of the drill hole information upon which this and the previous Resource Estimate Technical Report are based was recent historic information. BOYD has reviewed and qualified to NI 43-101 standards, these data in preparation of the Resource Estimate completed by BOYD in our October 2017 Technical Report. The foregoing incorporation of historic information notwithstanding, BOYD conducted its own field investigations with the compilation of results used to prepare both our October 2017 and the current NI 43-101 compliant Technical Reports.

Other information provided by FTR is contained in this Technical Report, including without limitation, validity of mineral tenure, status of environmental permits and potential liabilities. These matters were not independently verified by BOYD, but appear to be reasonable representations that are suitable for inclusion in this Technical Report – Preliminary Economic Assessment.

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4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Description and Ownership

The Laiva Gold Project is located in Northern Ostrobothnia, western Finland. The centroid of the project area is at 2528500 mE, 7161000 mN (Finnish Coordinate System, Zone 2). The nearest town, Raahe, is 20 km to the northwest, and the study area is located 70 km directly south of Oulu, which is the closest regional center. See Figure 4.1, Location Map Showing Laiva Gold Mine Project (Page 4-3).

The Laiva Gold Project is comprised of two license areas: Laiva and Oltava. The advanced stage Laiva Gold Mine and satellite exploration projects at Mussuneva and Kaukainen, are located within the Laiva license area (Figure 4.2, Page 4-4). Oltava is located 12 km south of Laiva and is an early stage exploration property.

The Laiva Gold Project is wholly owned by NORDIC, a private Finnish company, which is now owned 100% by FTR.

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 and is responsible for administration of permitting.

The Laiva Gold Project comprises 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 registration numbers of the licenses which form the Laiva Gold Project, and the current status of licenses are shown in Table 4.1.

Table 4.1
License Status

Exploration License Status				
Project	Registration Number	Status	Granted	Expiry Date
Laiva	9024/1	Valid	28.4.2014	28.4.2019
	9024/2			
	9024/3			
	9024/4			
	9024/5			
	9024/6			
	8857/1	Valid	23.11.2012	23.11.2017
	8857/2			
	8857/3			
	8857/4			
	8857/5			
	8857/6			
	8857/7			
	ML2014:0035-01	Renewal/Applied Extension	31.5.2006	Pending
	ML2013:0054	Renewal/Applied Extension	27.9.2005	Pending
	ML2013:0055	Renewal/Applied Extension	27.10.2005	Pending
Oltava	ML2012:0155	Valid	19.8.2014	19.8.2017
	ML2013:0102	Valid	17.11.2008	29.7.2019
	ML2012:0069	Valid	28.7.2016	29.7.2020

Exploration licenses are valid for an initial period of 4 years and can be extended for 3-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 10 years, or until further notice in order to exploit the deposit. An annual fee of €50/ha is payable to landowners within the mining permit and a rehabilitation fee is payable to the mining authority at the following rates:

- High grade tailings pond: €7/m^2
- Sulphide rich waste dump: €2.5/m^2
- All other areas: €1/m^2

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

SCALE (km)
100 0 100 200

FIGURE 4.1
LOCATION MAP
Showing
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
FIRESTEEL RESOURCES INC.

John T. Boyd Company
July 2018
Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 4-1.DWG

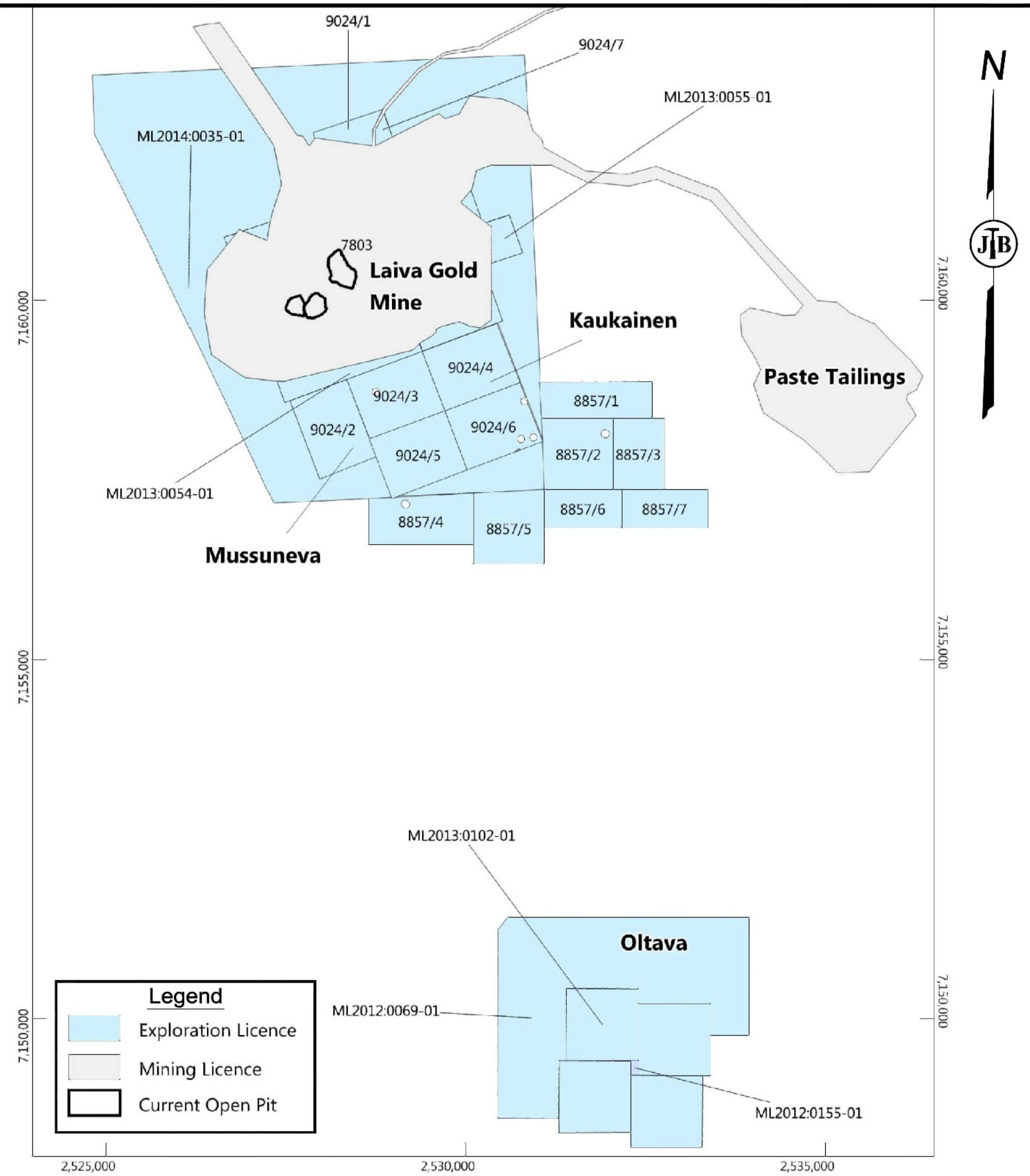


Figure Source : Nordic Mines Oy, with Additions by John T. Boyd Company.

SCALE (km)

1 0 1 2

FIGURE 4.2

MAP SHOWING
NORDIC MINES MINERAL RIGHTS
AND REGISTRATION NUMBERS
LAIVA AND OLTAVA PROPERTIES
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
FIRESTEEL RESOURCES INC.

John T. Boyd Company | July 2018 | Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 4-2.DWG

4.3 Surface Rights

NORDIC owns 900 ha of the surface area within the mining license area. The remainder of the surface rights at Laiva within the mining and exploration licenses are held by private landowners and forestry companies. The mining license permits NORDIC 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 NORDIC.

4.4 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.

It was reported to BOYD that the property is not subject to other royalties, back-in rights, payments or other agreements and encumbrances.

4.5 Comments

BOYD is not qualified to provide any opinion of a legal nature, but based on our review, we understand that NORDIC has sufficient rights and title in order to conduct exploration at the Laiva Gold Project and mining at the Laiva Gold Mine. Seven exploration licenses at Laiva and one exploration license at Oltava are due to expire in November and August 2017, respectively. License renewal approvals will be required from the mining authority in order to maintain these licenses in good standing. 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. Though BOYD has not performed detailed examination of rights and title held by NORDIC, it has no reason to doubt the validity of these mining rights.

Details regarding environmental liabilities and permits are discussed in Section 20.

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

5.1 Accessibility

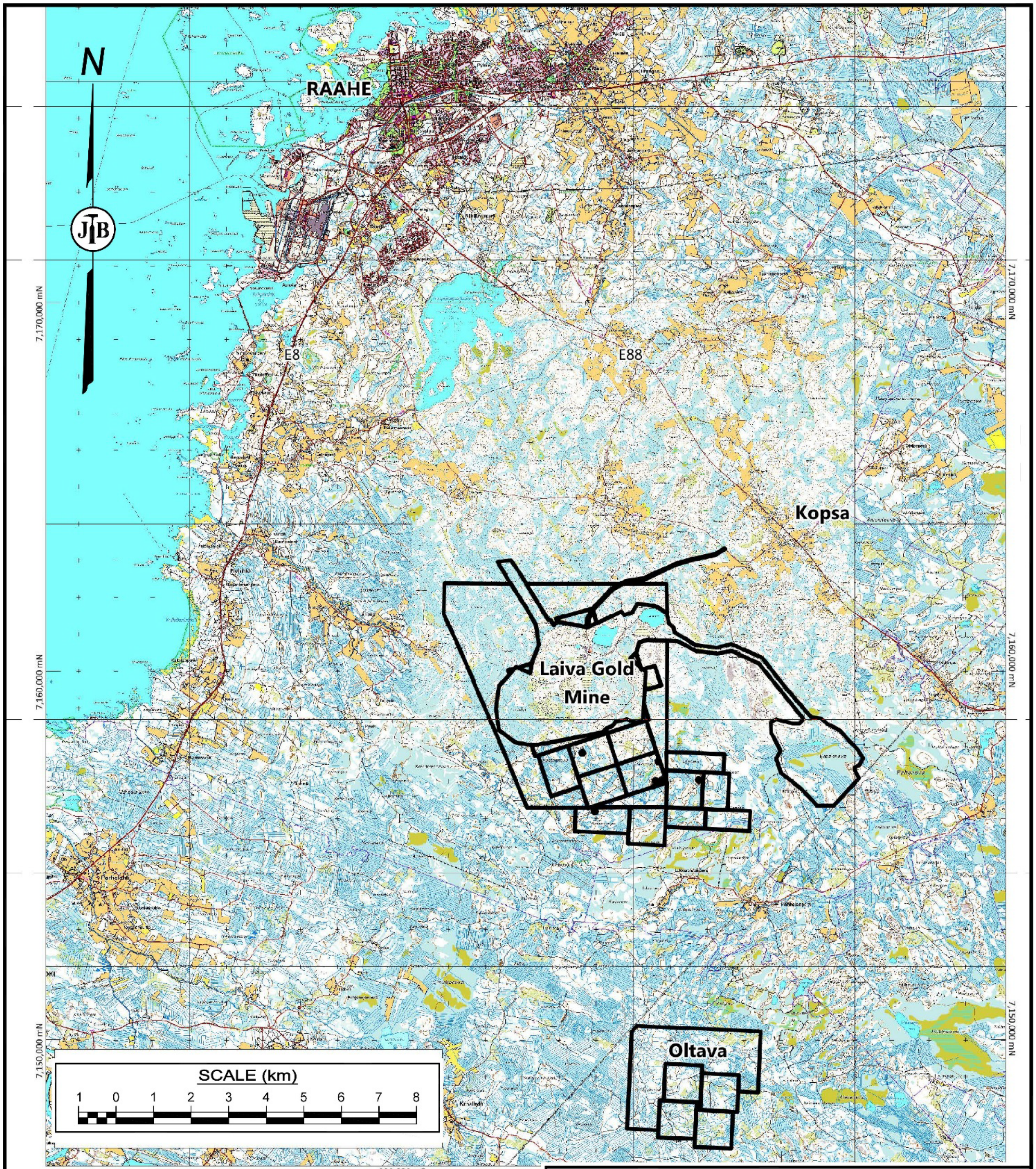
The Laiva Gold Project is readily accessed from the town of Raahe, North Ostrobothnia, located 20 km northwest of the project. Raahe has a population of approximately 20,000, and is located 600 km north of Helsinki and 80 km southwest of Oulu. A major coastal highway (E8) passes through the town. A port is located at Raahe, as well as a steel plant. The closest airport is located at Oulu which has direct flights to Helsinki (Figure 4.1). Alternatively, Raahe is accessed by rail or air from Helsinki to Kokkola and then by vehicle from Kokkola to Raahe.

From Raahe, the project is reached via public paved roads (Road 88) to the village of Kopsa. 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. See Figure 5.1, Map Showing License Areas, Laiva Gold Mine Project, following this page.

Access across the Laiva property are via gravel tracks constructed by NORDIC, as well as by 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 shallow streams. 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.



P:\CAD_GROUP-jfg\3832.001\FIGURE 5-1.DWG

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

FIGURE 5.1

MAP SHOWING LICENSE AREAS LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

Prepared For

FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018

Scale as Shown

Vegetation comprises pine, spruce, and birch trees with occasional grassed clearings and boggy areas. The following aerial photograph (supplied by NORDIC) of the Lavia Gold Mine (Figure 5.2) shows physiography and site infrastructure:

Figure 5.2
Aerial View of Project Site

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 affected the operating season for exploration and mining activities.

5.4 Infrastructure

Existing facilities at the Laiva Gold Project include an open-pit mine under care and maintenance. The mine consists of two pits and a 6,000-tonne-per-day mill and processing plant, including an autogenous mill, flash flotation concentrator, gravity circuit with spiral concentrator, and high and low grade cyanide leach circuits.

Tailings facilities include a lined high-grade tailings pond located adjacent to the cyanide plant, and a low-grade tailings paste deposit with slurry pipeline located 7 km east of the mine. Separate waste dumps for inert and sulphide-rich material are used and located proximal to the open-pit mines (see Figure 5.3, following this page).

P:\CAD_GROUP–jfg\3832.001\FIGURE 5–3.DWG

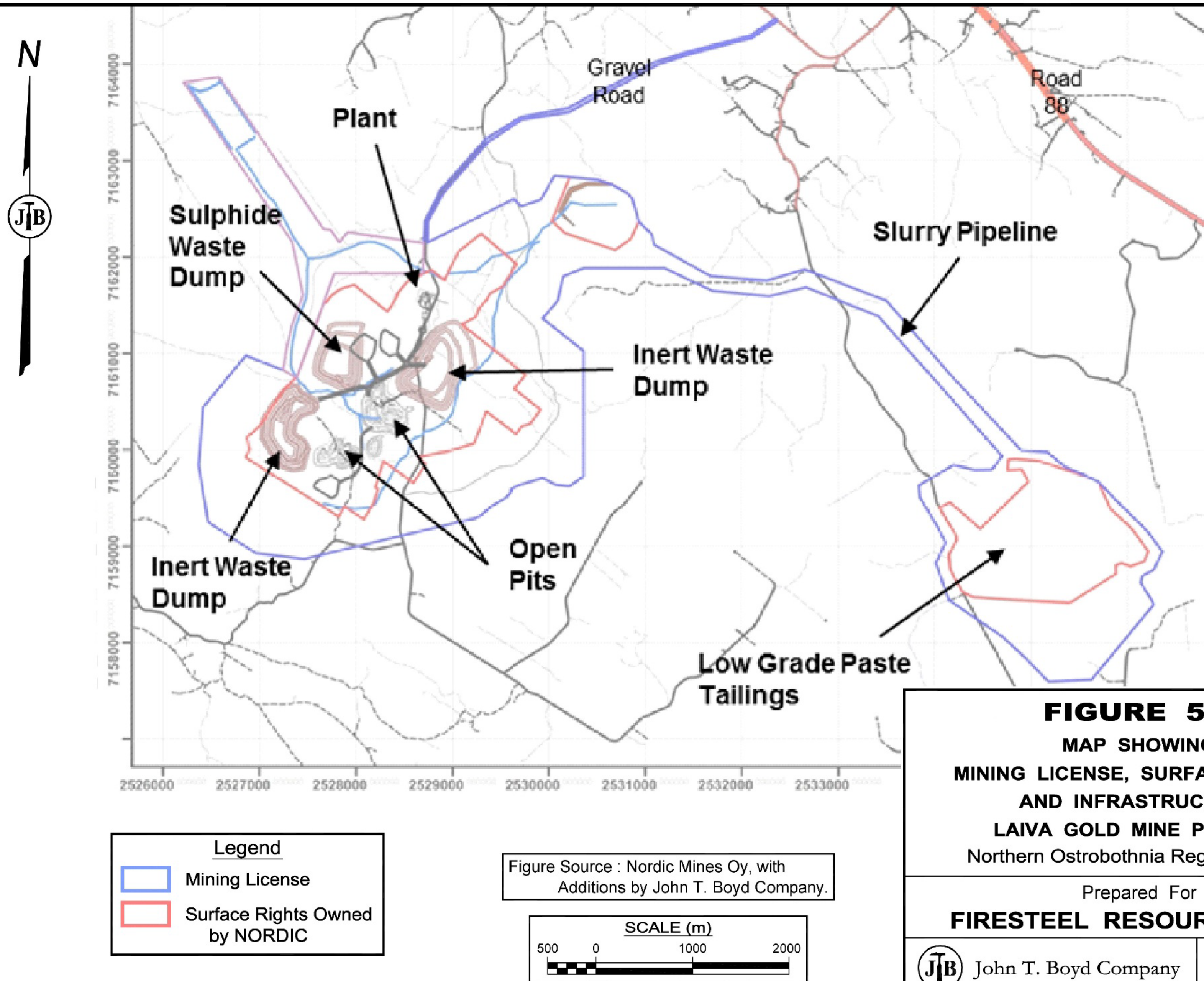


FIGURE 5.3

MAP SHOWING

MINING LICENSE, SURFACE RIGHTS AND INFRASTRUCTURE

LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

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NORDIC reported sufficient surface rights and water resources for mining and processing operations. Groundwater from the open pit and waste water filtered from the paste tailings are pumped and stored in a purpose built reservoir, which provides process water for the plant. Excess water in the dam is discharged through an 18 km long pipe into the Gulf of Bothnia.

The mine site has access to two lines of grid power; 110 kV 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 fuel to the mine fleet is located at the mine site.

The Laiva Mine operated between 2011 and 2013 with approximately 200 personnel and a minimal expatriate staff. It is anticipated that for any future recommencement of mining operations, the majority of mining personnel could be filled using Finnish nationals due to the prior engagement of largely Finnish personnel at the project and active mining industry in Finland. Some specialist technical and senior managerial positions may require expatriate staffing.

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6.0 HISTORY

6.1 Prior Ownership and Discovery

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 acquired by NORDIC in 2005.

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.

6.2 Historic Exploration

6.2.1 GTK

<u>6.2.1.1 Laiva</u>

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.

<u>6.2.1.2 Oltava</u>

GTK conducted regional soil sampling, ground geophysical surveys and diamond drilling at the Oltava project in the 1950s 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) the 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).

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.

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.

6.2.2.1 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 as follows:

Table 6.1
Outokumpu Historic Internal Mineral Resource Estimates

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

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 current mineral resources and the issuer is not treating the Outokumpu estimate as current mineral resources.

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 as follows:

Table 6.2
Endomines Mineral Resource Estimates

Date	Company	Deposit	Category	Tonnes	Gold (g/t)	Contained Gold (oz)	Methodology
2005	Endomines	Laiva North	Measured	-	-	-	Block model with a
Code:	Internal		Indicated	460,000	3.20	47,331	10x3x10 m block size
			Inferred	250,000	3.00	24,116	using a 2 g/t gold cut-off.
		Laiva South	Measured	-	-	-	Block model with a
			Indicated	290,000	4.10	38,232	5x3x5 m block size using
			Inferred	150,000	3.80	18,328	a 2 g/t gold cut-off.

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 current mineral resources and the issuer is not treating the Endomines estimate as current mineral resources.

6.3.3 NORDIC Mines Previous Mineral Resource and Reserve Estimates

NORDIC 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. The historical mineral resource estimates are superseded by the mineral resource estimate presented in Section 14, 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 current mineral resources and the issuer is not treating the NORDIC estimates as current 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 x20 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 25x10x20 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	-	-	-	

Table 6.3 – Continued

Date	Company/ Consultant	Deposit	Category	Tonnes (Mt)	Gold (g/t)	Contained Gold (koz)	Methodology
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	CSA Global (UK)		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	-	-	-	
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	-	-	-	
2011	Nordic	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.
11th July	CSA Global (UK)		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 Optimisation using a gold price of $535/oz, mining recovery of 90% and mining dilution of 10%.
			Probable Reserve	7.15	1.86	427	
2012	Nordic	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.
8th May	CSA Global (UK)		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	

Table 6.3 – Continued							
2012 8th May	Nordic CSA Global (UK)	Primary Resource	Measured Indicated	5.21 14.02	2.08 2.03	349 915	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.
Code:	JORC (2008)		Inferred	2.14	2.07	142	
		Additional Resource	Measured Indicated Inferred	1.83 5.30 -	0.64 0.64 -	38 109 -	
		Global Estimate	Proved Reserve Probable Reserve	6.41 8.78	1.57 1.60	324 452	Whittle Pit Optimization using a gold price of $750/oz, mining recovery of 90% and mining dilution of 10%.
Note: Numbers May Differ Between Reports Due to Rounding.							

6.3.3.2 NORDIC Recent 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 in Table 6.4. These mineral resource and reserve estimates are no longer considered relevant as the parameters used are considered inappropriate by BOYD. BOYD has produced a revised mineral resource estimate presented in Section 14. BOYD recommends that further work including infill drilling, reassessment of previous mining practices, revised detailed mine plans, metallurgical test work and optimization of mineral processing is conducted in support of a mineral reserve estimate in accordance with NI 43-101. A qualified person has not done sufficient work to classify the NORDIC estimates as current mineral resources and reserves and the issuer is not treating the NORDIC estimates as current mineral resources and reserves.

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	Nordic	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.
1st November	SRK Consulting (UK)		Indicated	15.42	1.49	739	
Code:	JORC (2012)		Inferred	3.15	2.28	231	
2015	Nordic	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.
1st January	SRK Consulting (UK)		Indicated	15.97	1.52	780	
Code:	JORC (2012)		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	Nordic	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.
May	SRK Consulting (UK) / Dr John Arthur		Indicated	24.32	1.13	885	
Code:	JORC (2012)		Inferred	4.37	1.64	231	

Note: Numbers May Differ Between Reports Due to Rounding.

6.4 Production History

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:

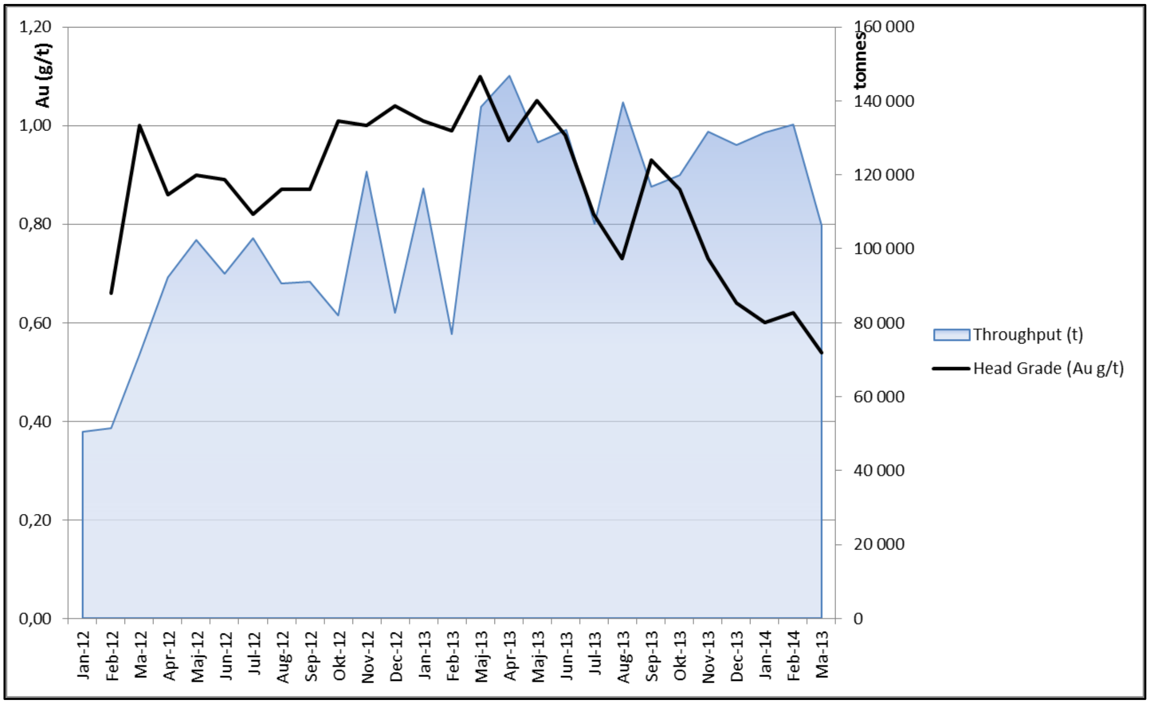


Figure 6.1
Previous Production from the Laiva Gold Mine

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 to10 m deep. Both pits are currently 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 was 2,241 kg from 2.8 million tonnes of mineralized material 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 tonnes of sulphide-rich waste, was produced with a strip ratio of 1:2.8. The mine operated at an average all-in sustaining cost of approximately US$1,760 per ounce gold.

7.0 GEOLOGICAL 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). 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 1900 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. 1770 Ma (Nironen, 2017). Several stages of Svecofennian granitic magmatism are recognized, comprising syn-orogenic (1930 to 1850 Ma) and late-orogenic (1850 to 1770 Ma) stages (Hanski, 2016).

Between 1860-1890 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).

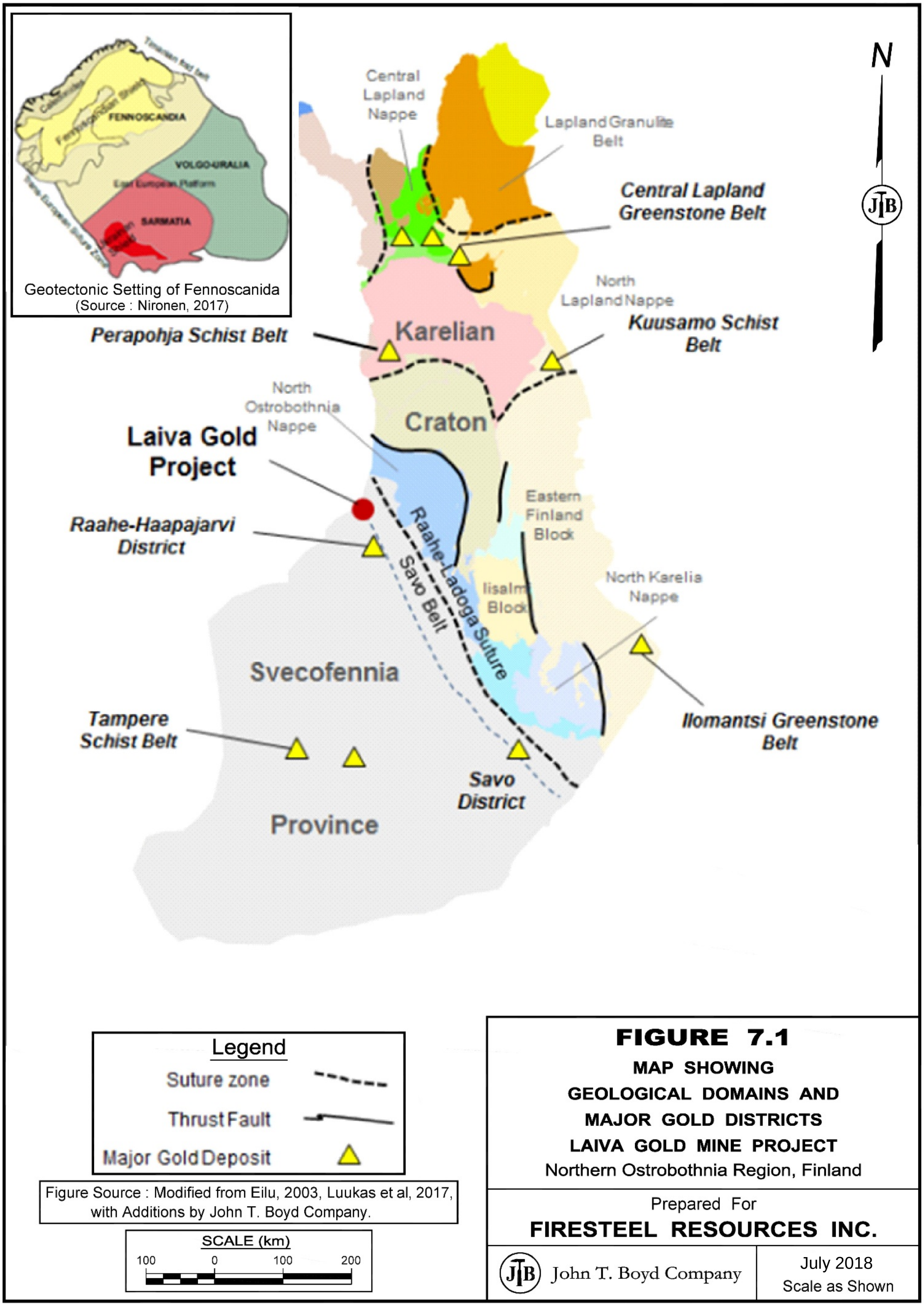


FIGURE 7.1

MAP SHOWING GEOLOGICAL DOMAINS AND MAJOR GOLD DISTRICTS LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

Prepared For

FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018

Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 7-1.DWG

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 dykes. These Svecofennian units are variably folded and faulted and display lower greenschist through to amphibolite facies metamorphism. Younger mafic dykes 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 comprises 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). 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 dykes and later mafic (dolerite) dykes (Figure 7.2 overleaf).

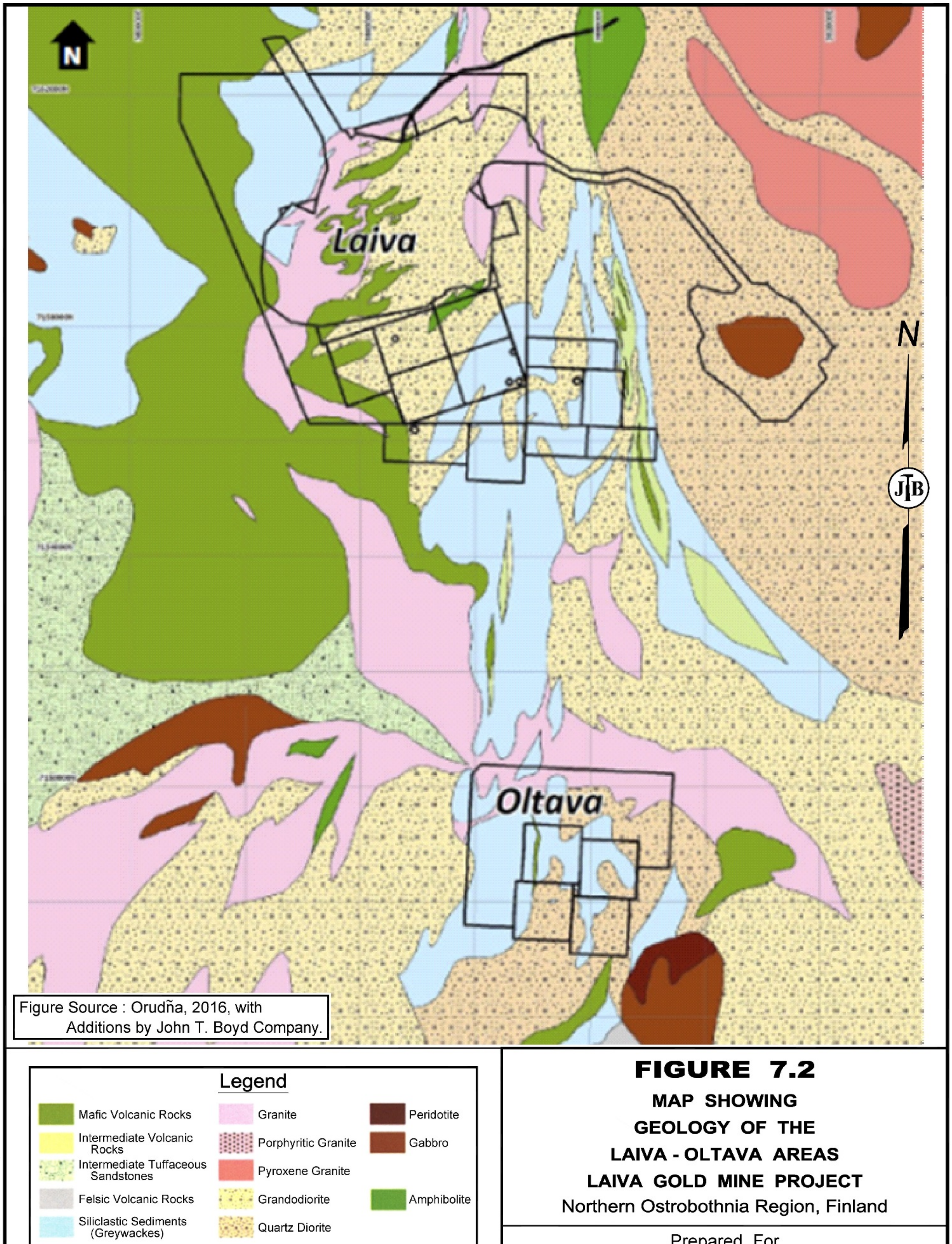


FIGURE 7.2

MAP SHOWING GEOLOGY OF THE LAIVA - OLTAVA AREAS LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

Prepared For

FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018

Scale as Shown

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

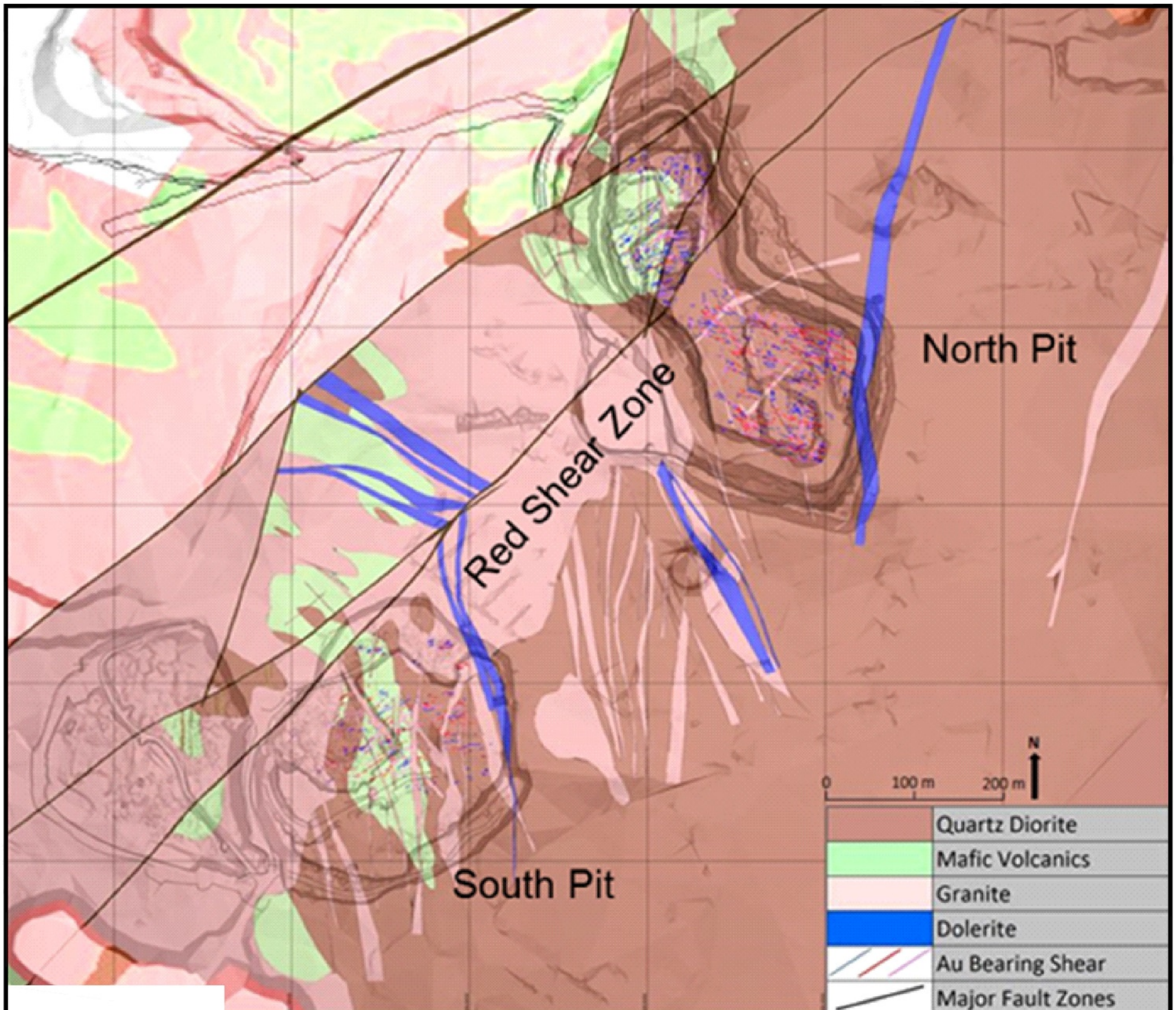


Figure Source : Nordic Mines Oy, with Additions by John T. Boyd Company.

SCALE (m)
100 0 100 200

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

Prepared For
FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018
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 Laiva Gold Mine

7.4.1 Lithostratigraphy

The Laiva Gold Mine comprises mafic meta-volcanic (termed Uralite Porphyrite by NORDIC) intercalated with rare meta-volcanosedimentary horizons, intruded by quartz diorite plutons and dykes. Together, the metavolcanic and quartz diorite are the main host to mineralization (see Figure 7.3). 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 (see Figure 7.3 overleaf) 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 dykes, sills and plutons; however, relatively minor narrow (<2 m thick) pre-mineral pegmatite dykes 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 dykes and sills typically occur proximal and sub-parallel or perpendicular to the contact between the granite pluton and metavolcanic rocks. The granitic dykes and sills vary in thickness between <1 m to >4 m and form a conjugate pattern. In the South Pit area, the granite dykes form a dense network, encapsulating xenoliths of mineralized metavolcanic rocks (see Figure 7.4 overleaf).

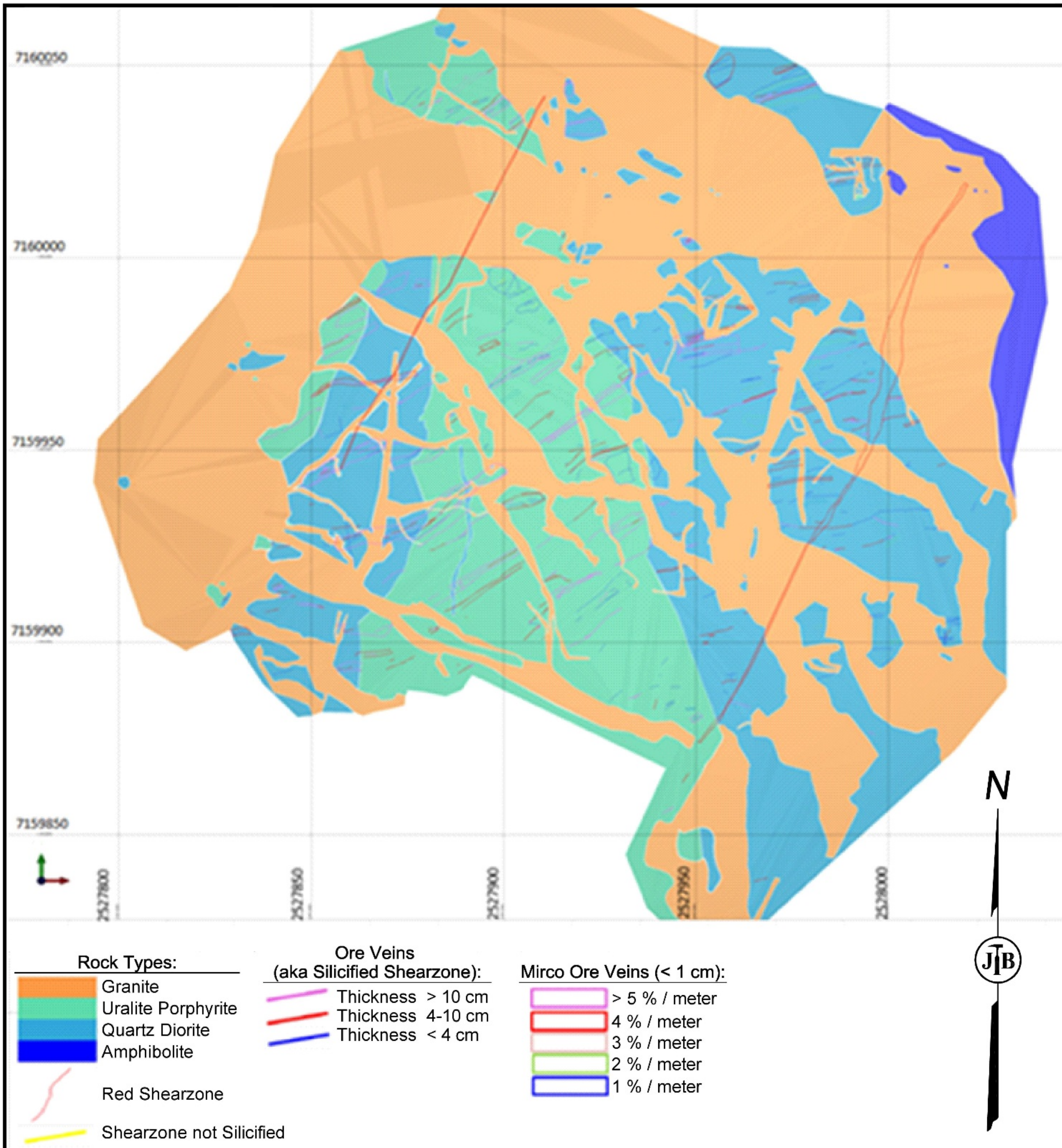


Figure Source : Nordic Mines Oy, with Additions by John T. Boyd Company.

SCALE (m)

25 0 25 50

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
FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018
Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 7-4.DWG

All units are intruded by dolerite sills and dykes 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

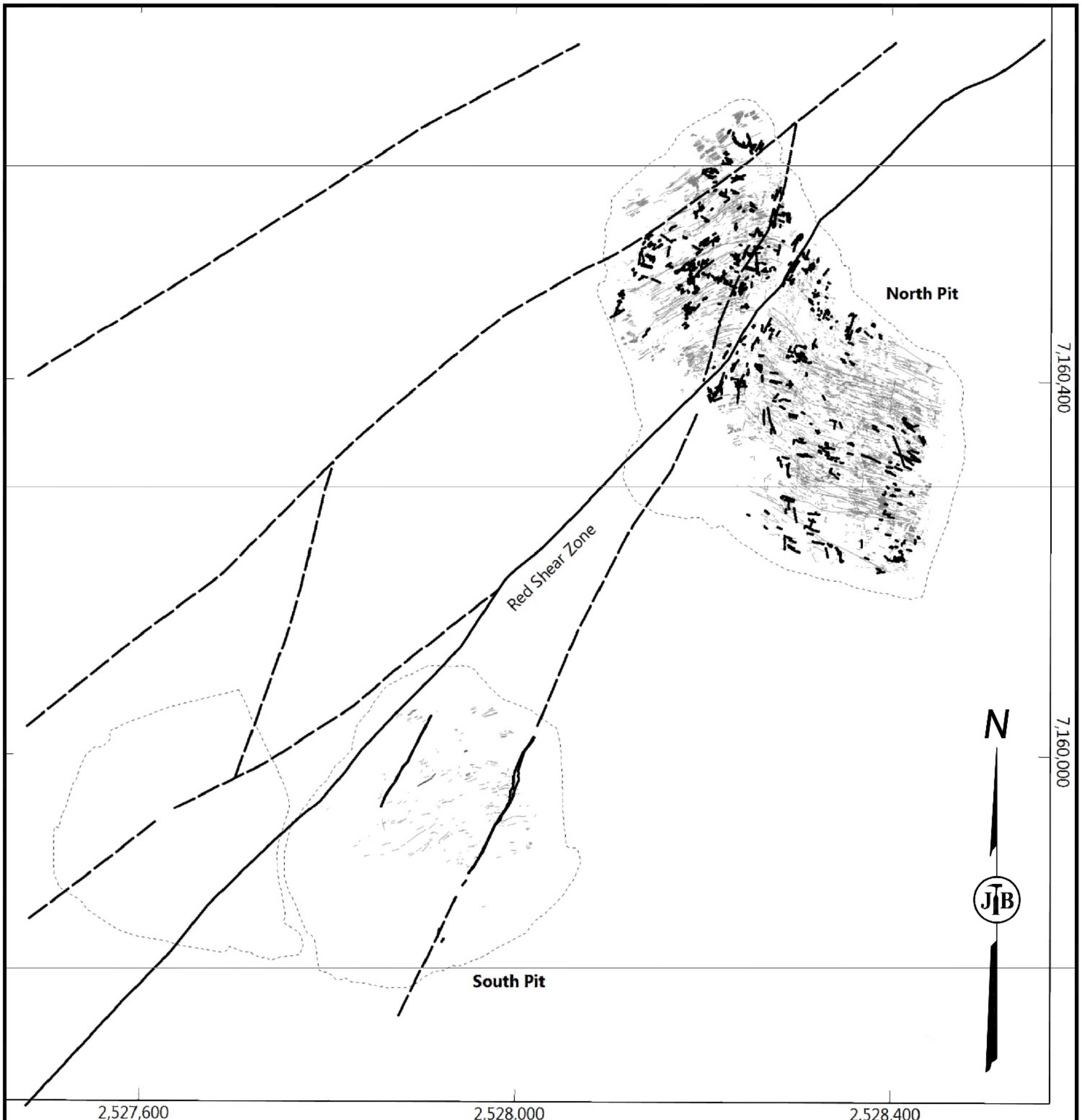
Structure has been mapped during grade control work and drill core logging, as well as inferred from ground magnetic data. Much of the structural understanding of the Laiva Gold Mine has been developed recently by Pratt (2010), SRK (2014), and BOYD (this study).

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 dykes. Parasitic folding of granite dykes, 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), but in general strike approximately east-west. The main orientations of mylonitic shear zones are shown in Figure 7.5. The mylonitic shear zones are hosted in quartz diorite and mafic volcanic rocks, post-date pegmatite dykes and are cross cut by granite and diorite dykes. 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 dykes has not been observed. The faults strike northeast and dip steeply southeast. The Red Shear Zone is the best defined of these faults (Figures 7.3, above and 7.5 overleaf), which displays a dextral shear sense, strikes northeast and dips steeply southeast. Lateral displacement in the order of 10 m is observed.



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

FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018

Scale as Shown

P:\CAD_GROUP–jfg\3832.001\FIGURE 7–5.DWG

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

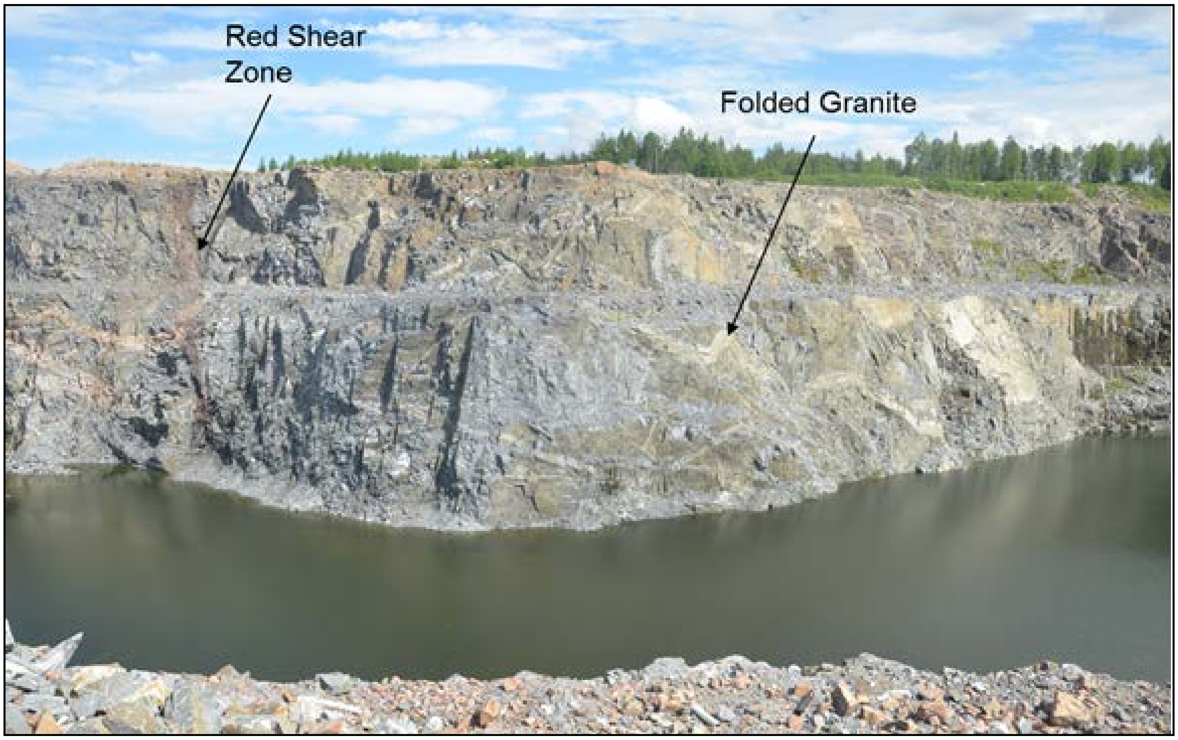


Figure 7.6
View Northeast Over the North Pit, Showing the Red Shear Zone and Late Granite Intrusions

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 below).

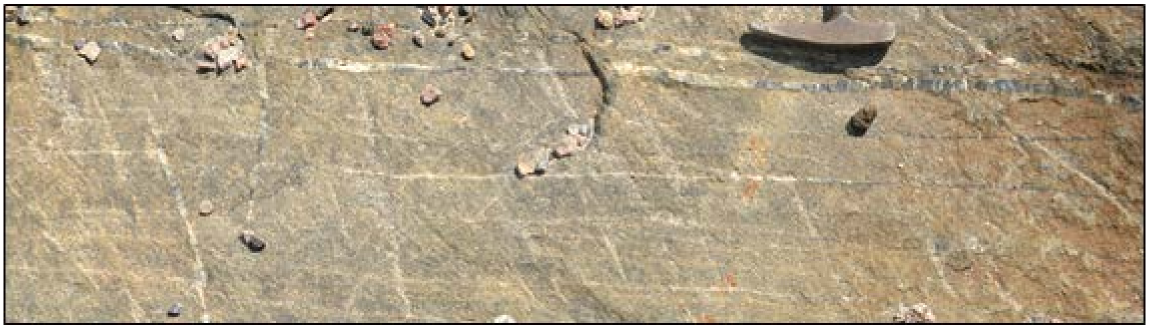


Figure 7.8
Planar, Sheeted Quartz-Sulphide Veins and High Angle, Short Scale Tension Gashes. South Pit. Hammer Head for Scale

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 Dykes

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

Granite dykes 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 dykes. Small scale fault drag of mineralized zones at the contacts with granite dykes is observed in the South Pit and provides some evidence that the mineralized zones are locally folded proximal to granite contacts (see below).

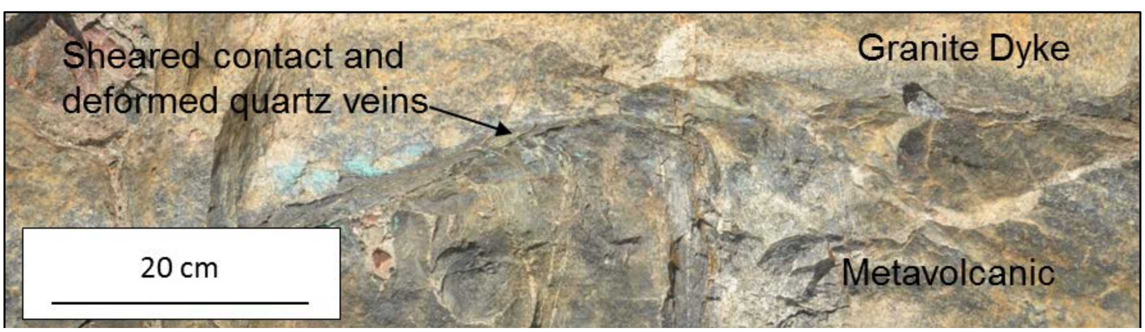


Figure 7.9
Sheared Contact Between Metavolcanic Hosted Mineralized Shear Zone and Granite Dyke

7.4.3 Alteration

Alteration at Laiva is weak and comprises a rare, narrow (mm-scale) sericite-pyrite selvedge to mineralized shear zones (see 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 comprises 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 below).

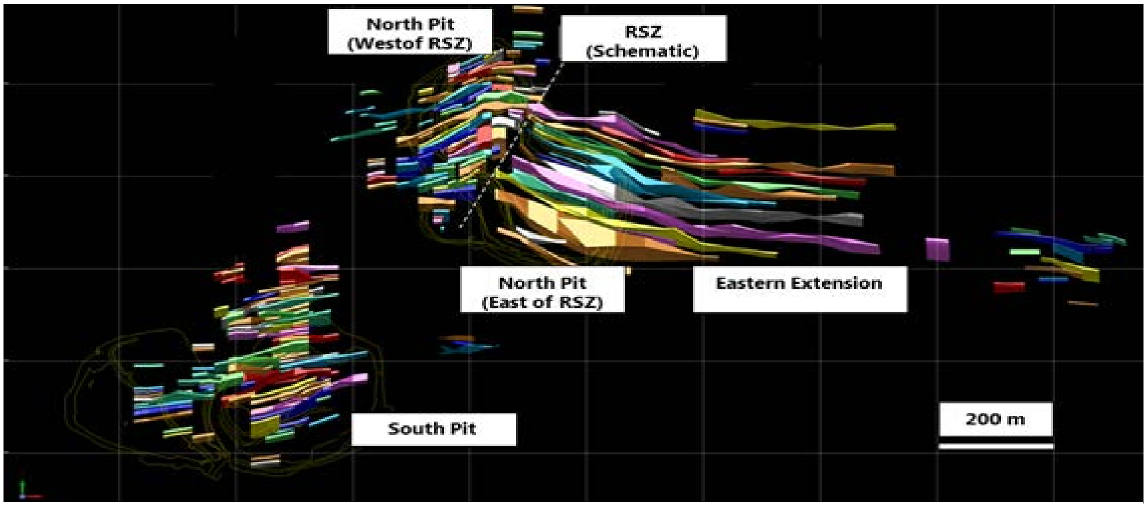


Figure 7.11
Plan View of Mineralized Shear Zones Forming the Laiva Gold Deposit. RSZ – Red

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 1300 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 below).

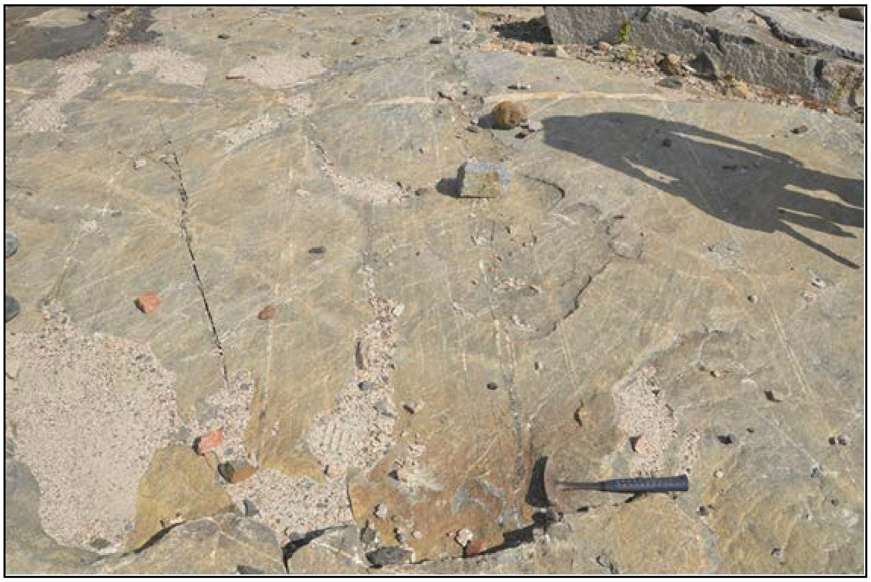


Figure 7.12
Exposed Mineralized Shear Zone in the South Pit

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 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. Whilst 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 and 10). 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:

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 cannot be relied upon. The results from historic drilling warrant follow up exploration.

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 >1km.

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 a mineralized material 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 as 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.

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9.0 EXPLORATION

9.1 Introduction

NORDIC commenced a phased regional geochemical sampling program in 2005, including an initial assessment of historic regional geochemical and geophysical data obtained from previous operators, followed up by NORDIC with 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 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.

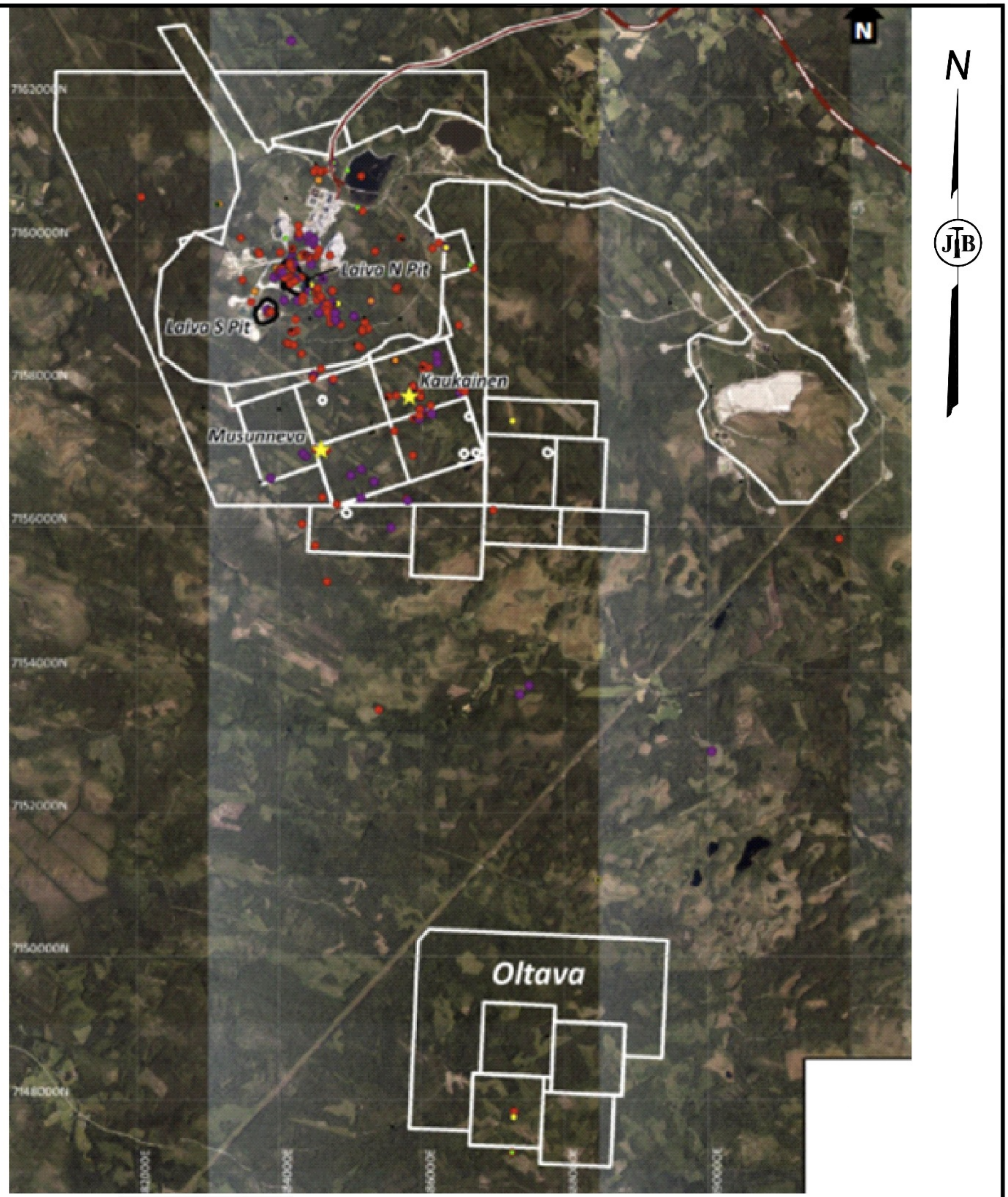
The issuer, FTR, has not conducted exploration work to date. The methodology and results of the exploration conducted by NORDIC and the previous operators are summarized below.

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, zinc. The gold geochemistry of boulder samples is shown in Figure 9.1, overleaf. 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.



Au (g/t)
- >10.00
- 1.00 - 10.00
- 0.50 - 1.00
- 0.25 - 0.50
- 0.10 - 0.25
- 0.01 - 0.10
- < 0.01

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

SCALE (km)

1 0 1 2

FIGURE 9.1

MAP SHOWING
HISTORIC BOULDER SAMPLE LOCATIONS
GOLD GEOCHEMISTRY
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018
Scale as Shown

P:\CAD_GROUP-jfg\3832.001\FIGURE 9-1.DWG

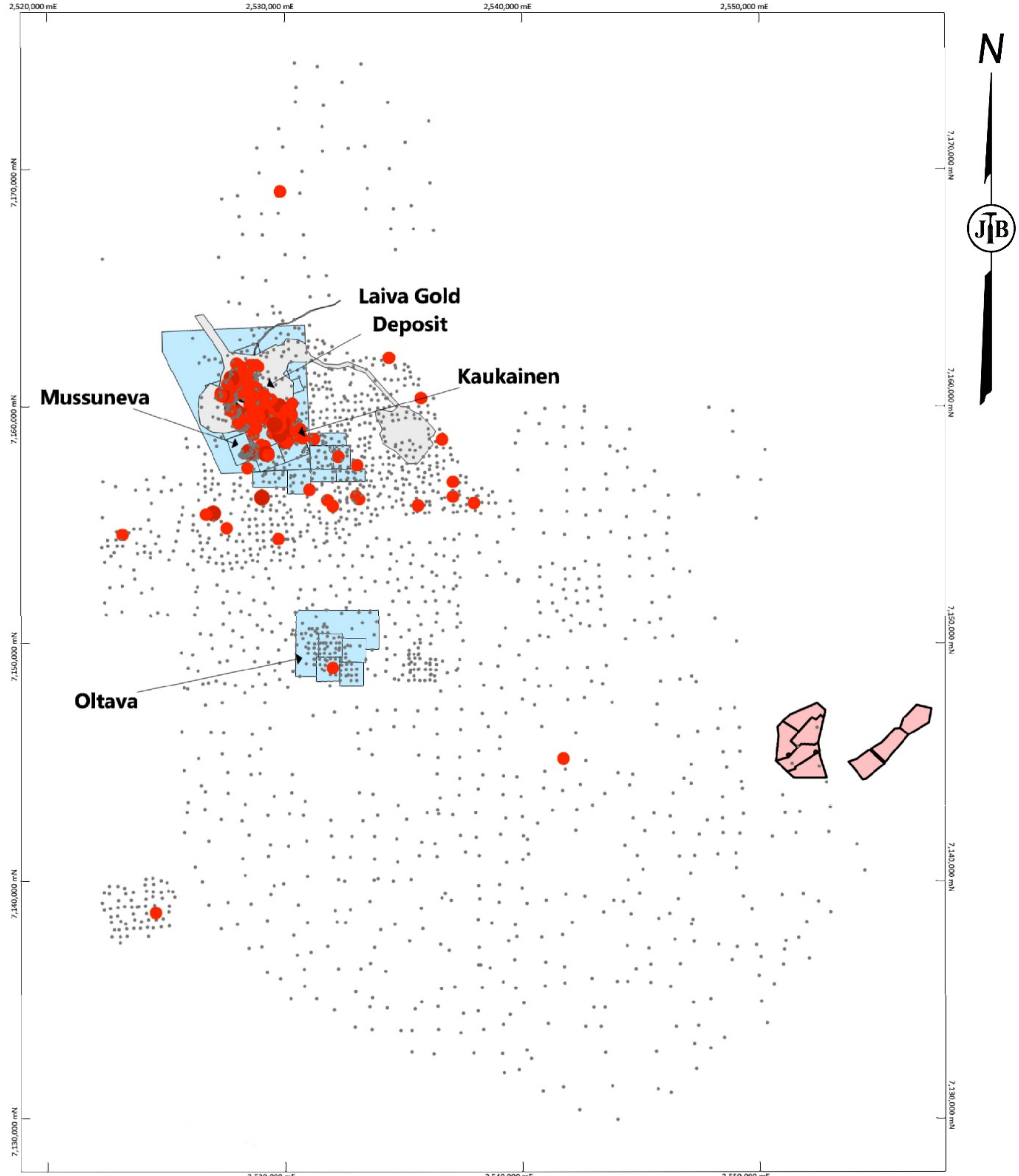
9.3 Surficial Till Sampling

NORDIC collected 2,841 surface till samples over an area of approximately 46 km by 34 km (Figure 9.2 overleaf, Page 9-4). 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 (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 overleaf).

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 overleaf). 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).



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

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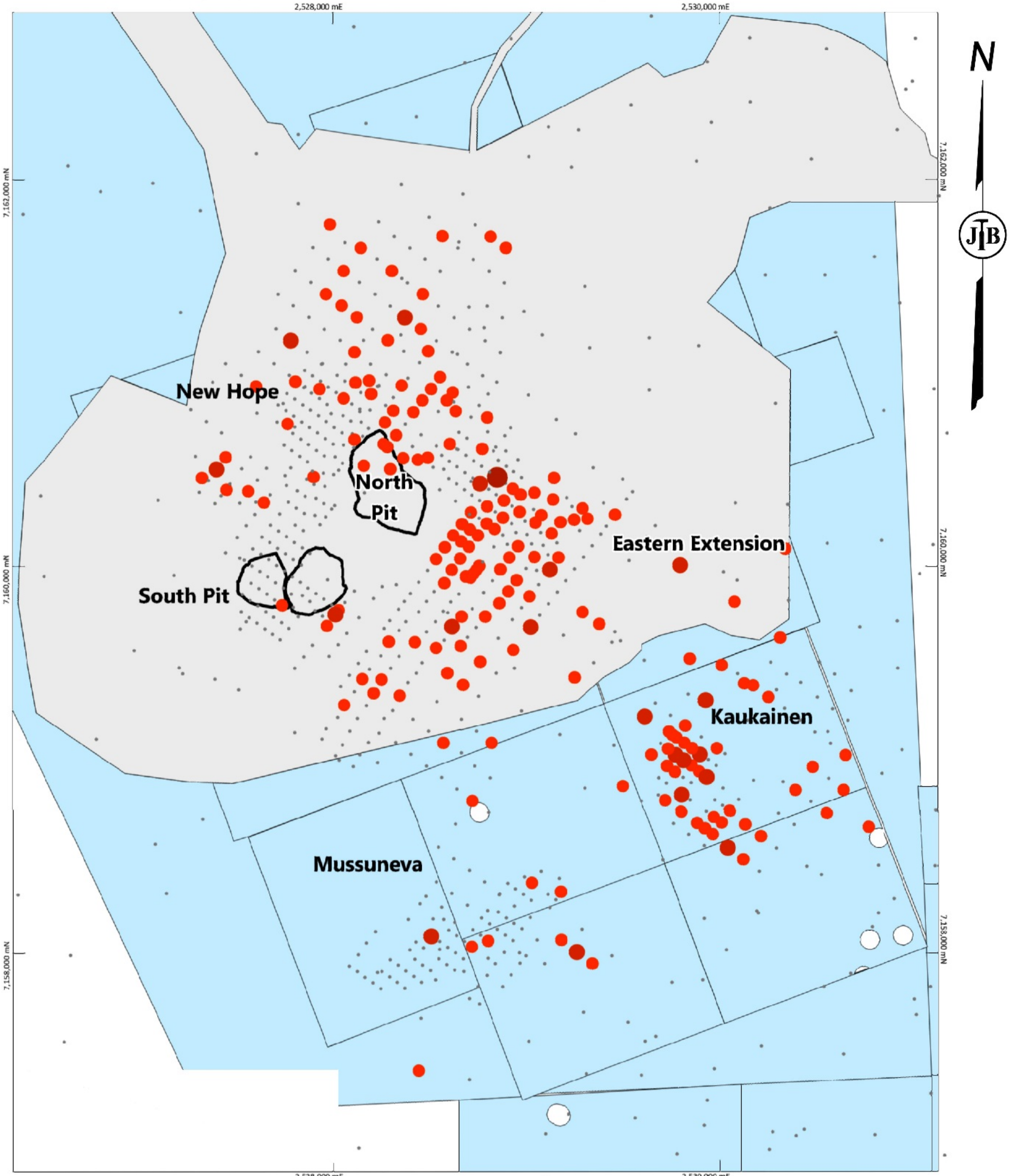
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-2.DWG



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

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July 2018
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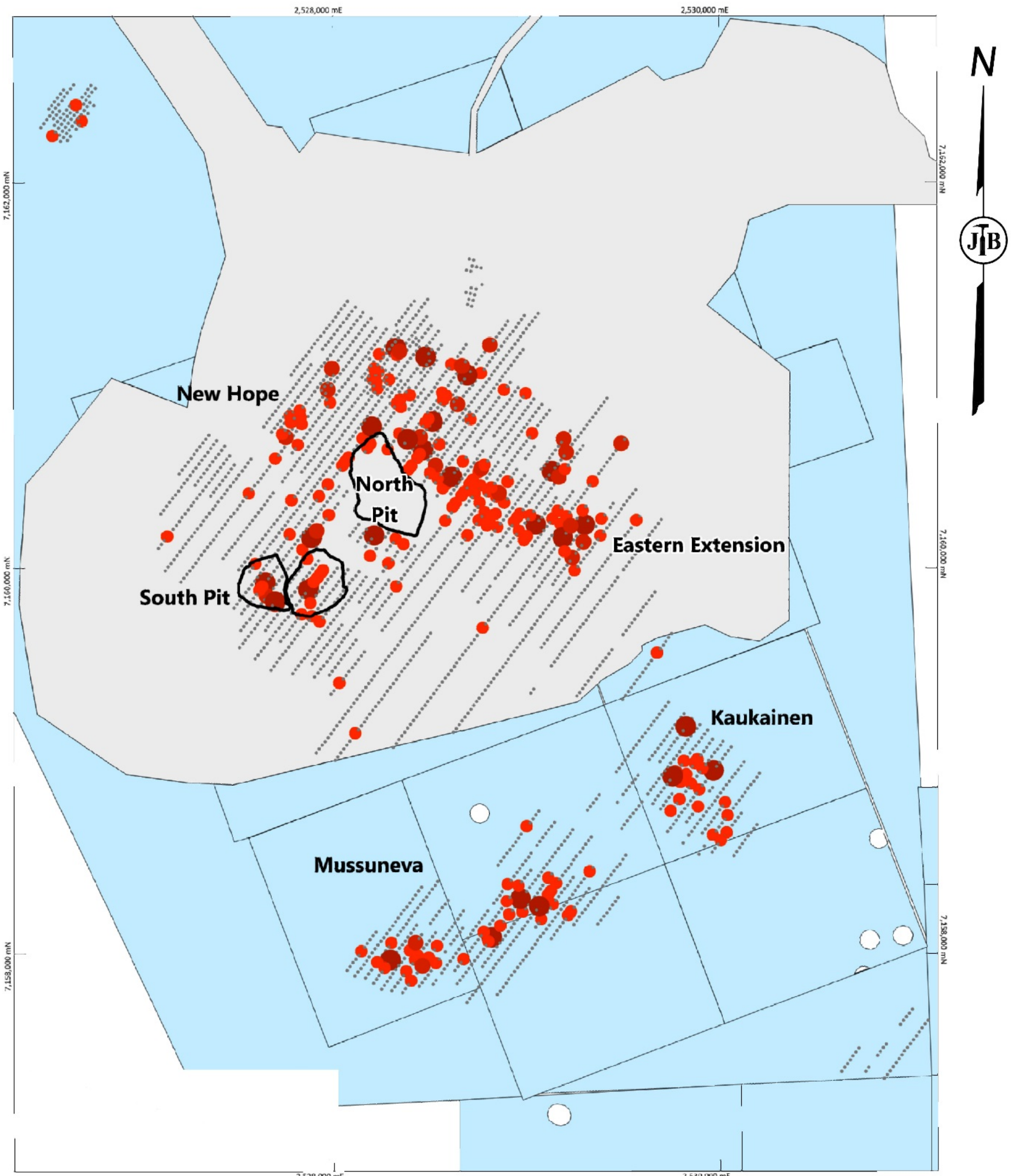
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-3.DWG

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 by 25 m, 100 m by 50 m, 50 m by 50 m and occasionally 25 m by 25 m spacing (Figures 9.4 to 9.7 below), 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 was recorded and the collar coordinate was surveyed using hand held GPS.



Legend

- Exploration Licence
- Mining Licence
- Current Open Pit

Bedrock Samples (Au ppm)

- 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

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P:\CAD_GROUP-jfg\3832.001\FIGURE 9-4.DWG

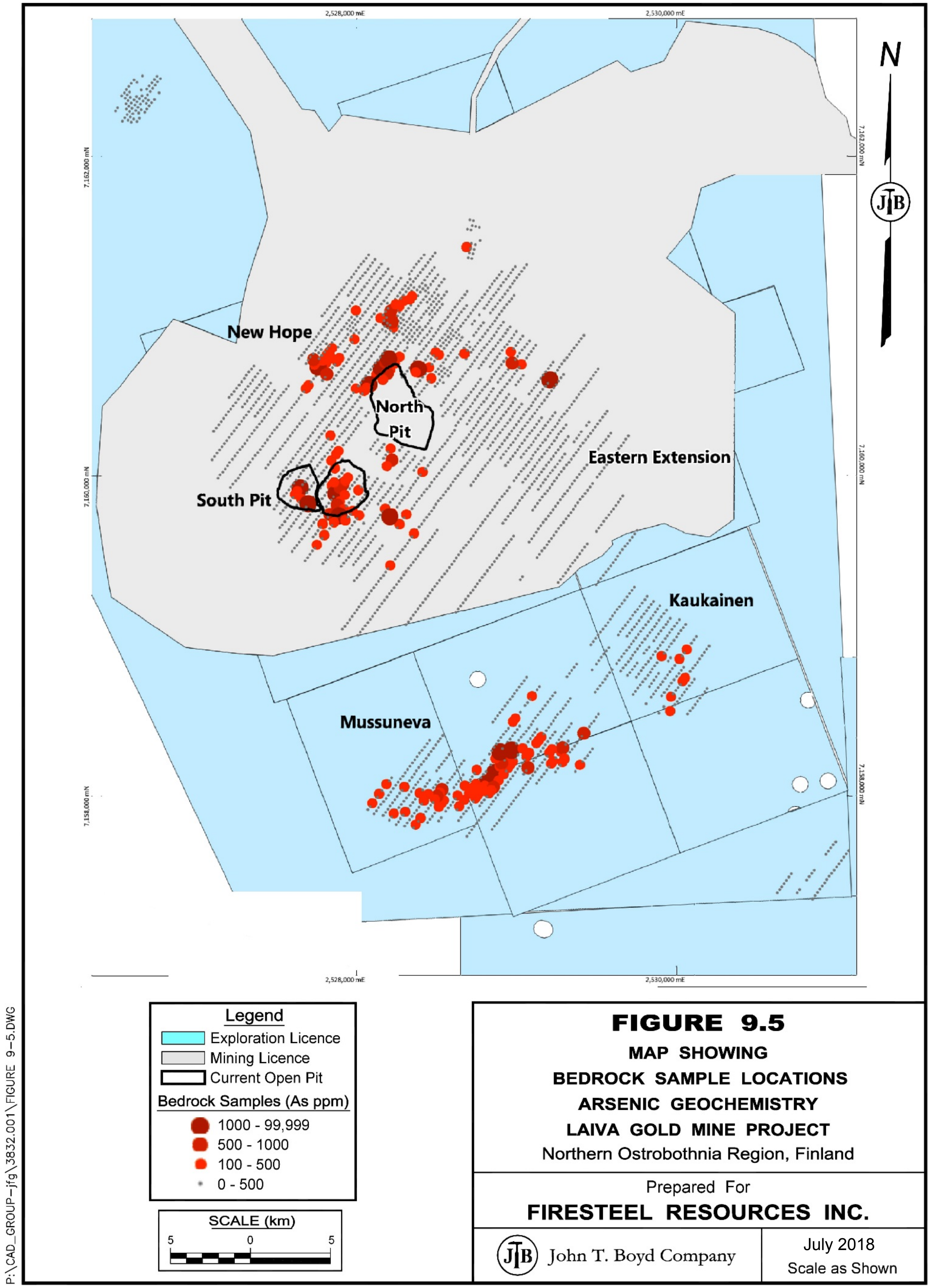

FIGURE 9.5

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

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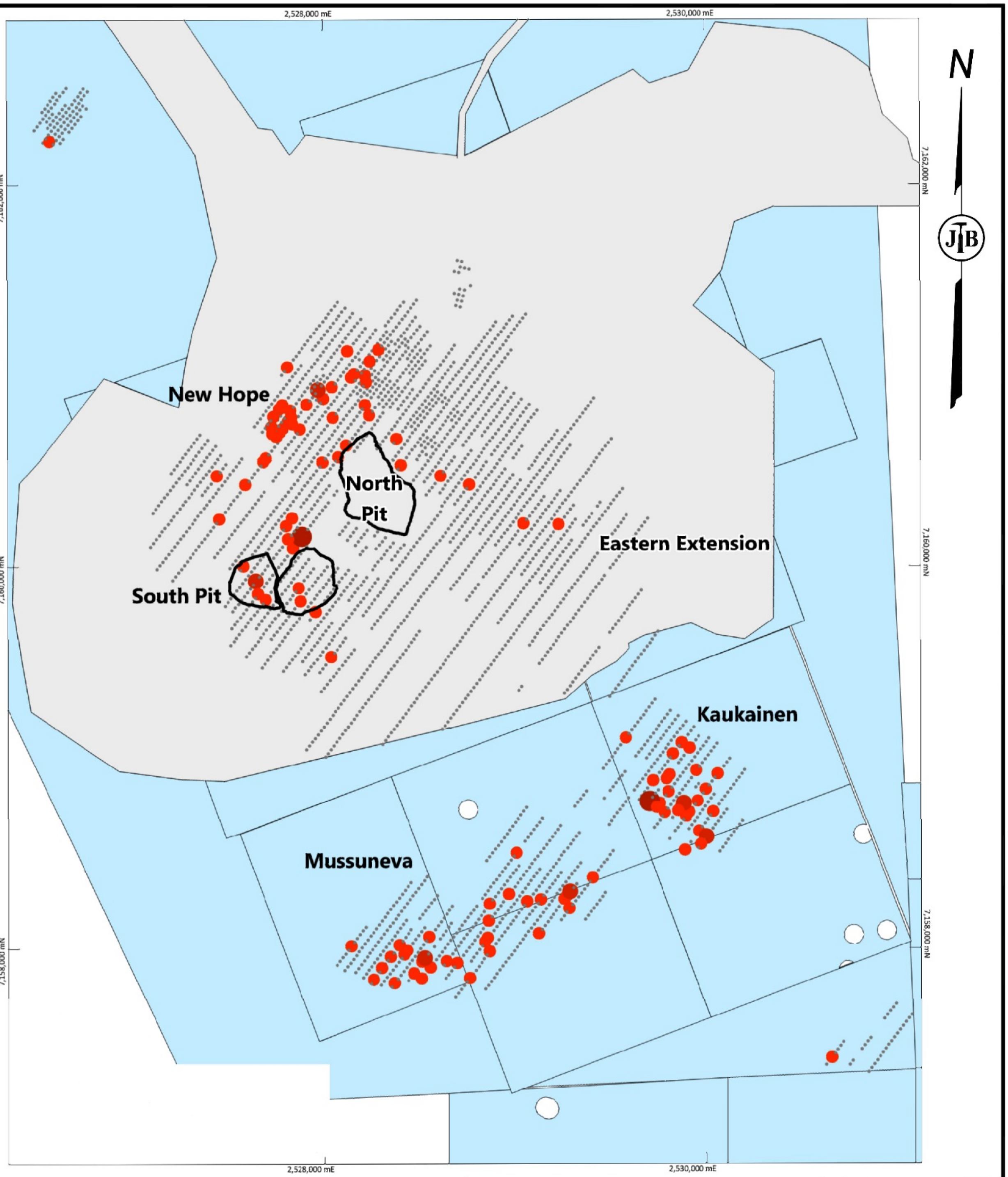

Legend
- Exploration Licence
- Mining Licence
- Current Open Pit

Bedrock Samples (Cu ppm)
- 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

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P:\CAD_GROUP-jfg\3832.001\FIGURE 9-6.DWG

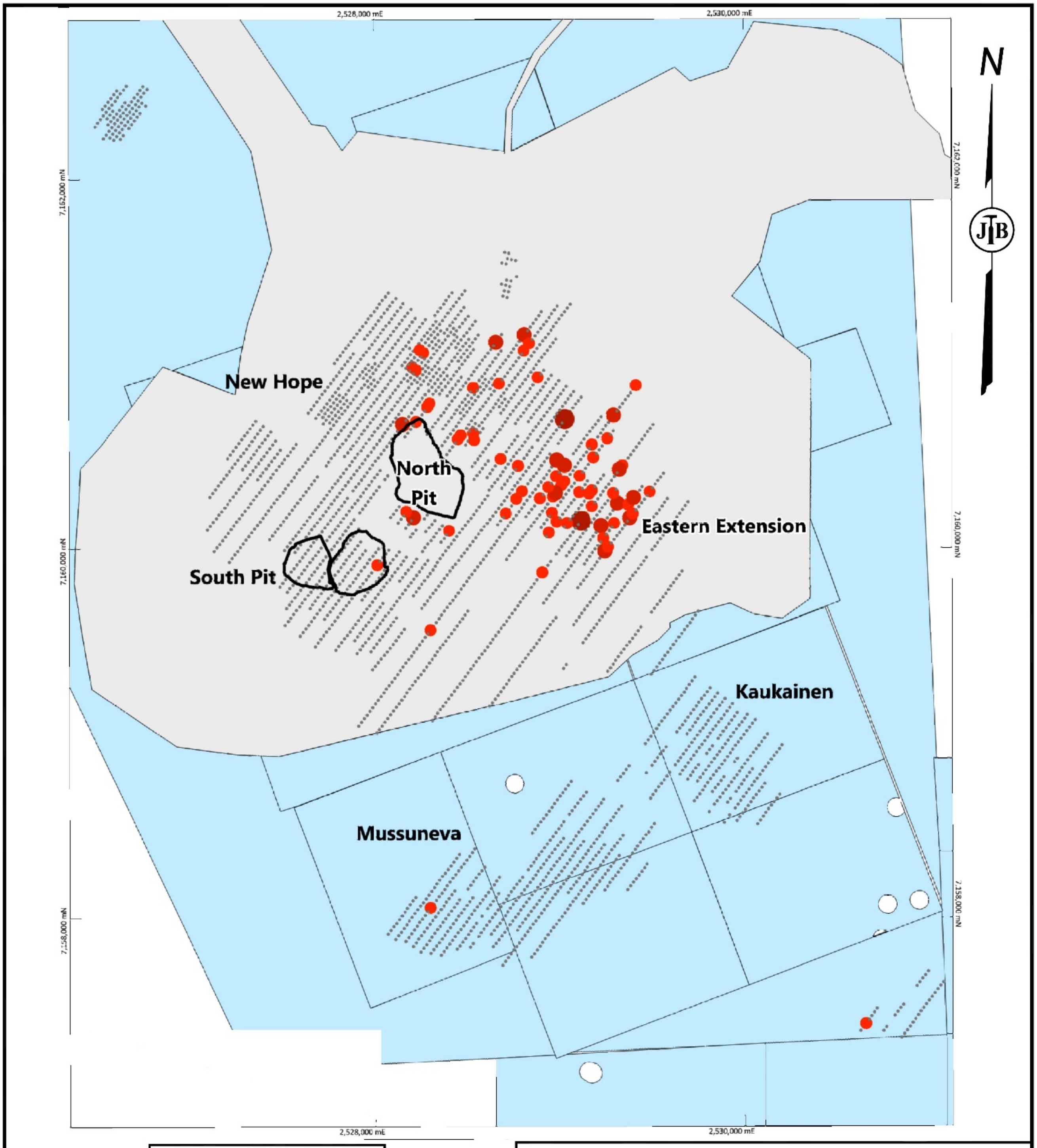

FIGURE 9.7

MAP SHOWING BEDROCK SAMPLE LOCATIONS MOLYBDENUM GEOCHEMISTRY LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

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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 (Figure 9.4, above). The western part of this zone displays elevated concentrations of arsenic and lesser copper (Figures 9.5 and 9.6, above). 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, above). Arsenic and copper are not associated with the mineralization. This area was followed up with extensive diamond drilling by NORDIC (see Section 10).

The New Hope prospect is located 300 m west of the current North Pit and comprises a 150 by 50 m gold-arsenic-copper anomaly. The prospect was followed up with two channel samples and three exploration drill holes (see Section 10). 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, above). 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

The 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 (Figures 9.8 to 9.11, below). 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 km^2) area of the Oltava project (total 21.5 km^2) 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.

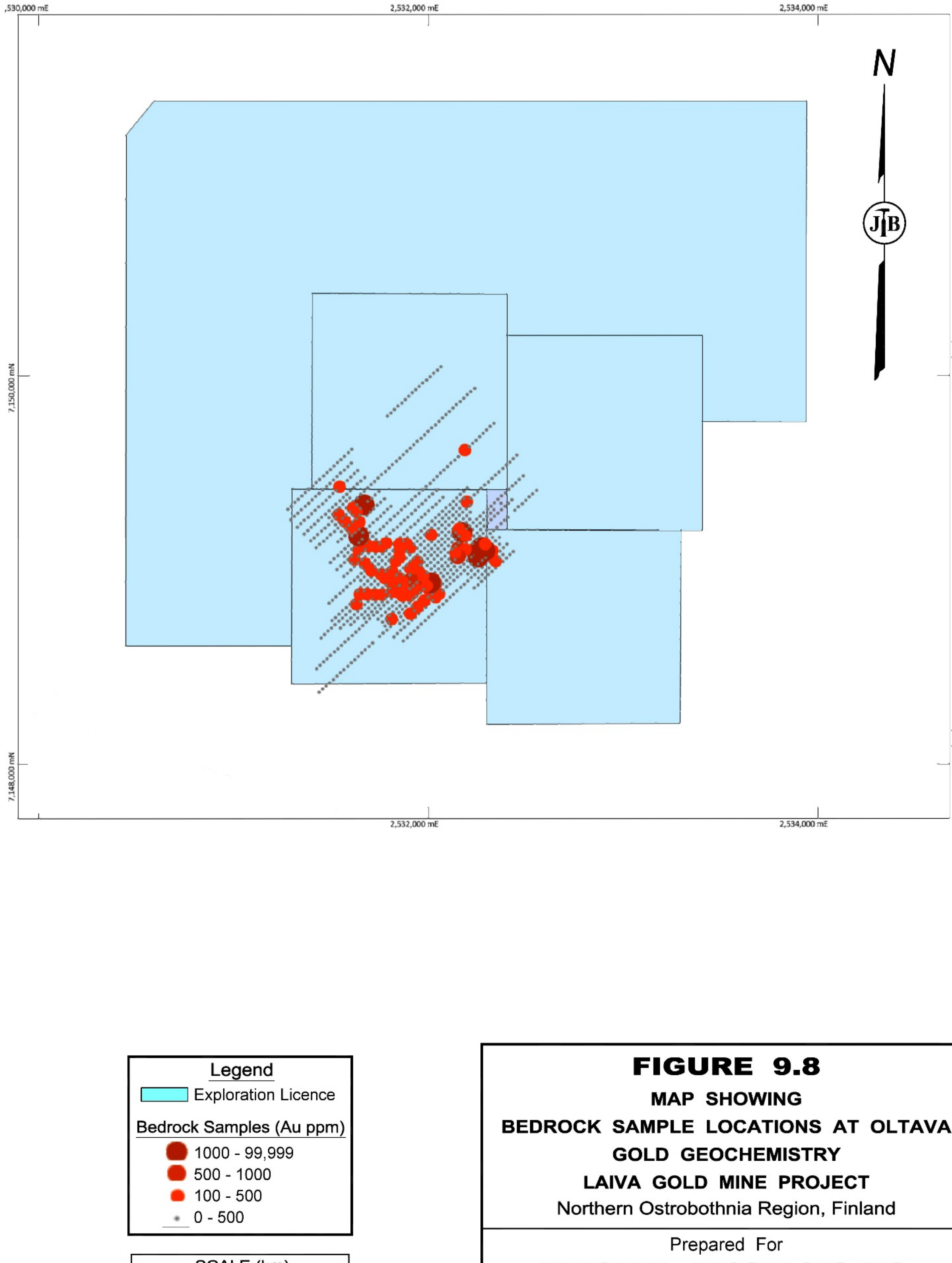

FIGURE 9.8

MAP SHOWING

BEDROCK SAMPLE LOCATIONS AT OLTAVA

GOLD GEOCHEMISTRY

LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

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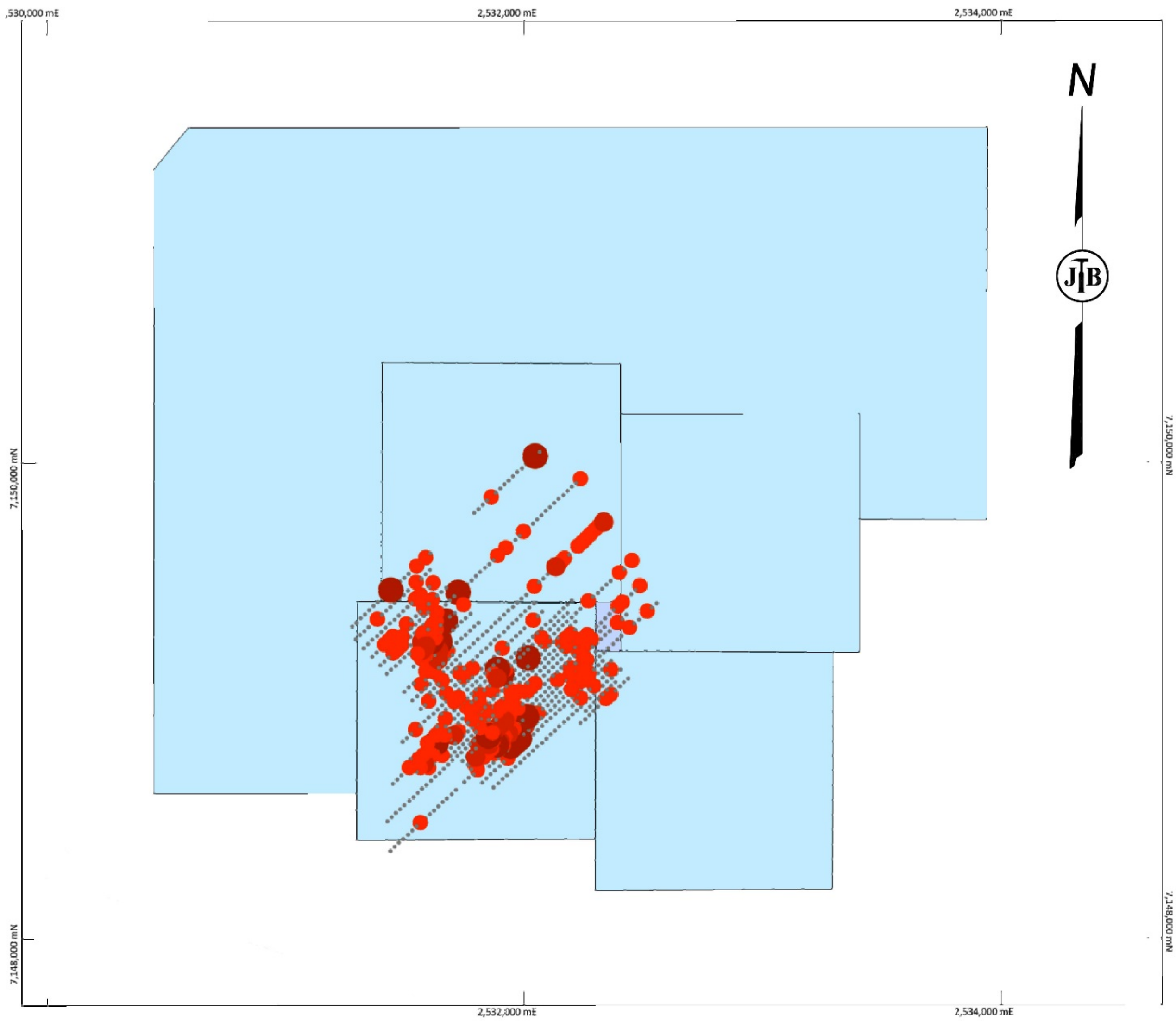
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-8.DWG



Legend

Exploration Licence

Bedrock Samples (As ppm)

- 1000 - 99,999
- 500 - 1000
- 100 - 500
- 0 - 500

SCALE (km)

5 0 5

FIGURE 9.9

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

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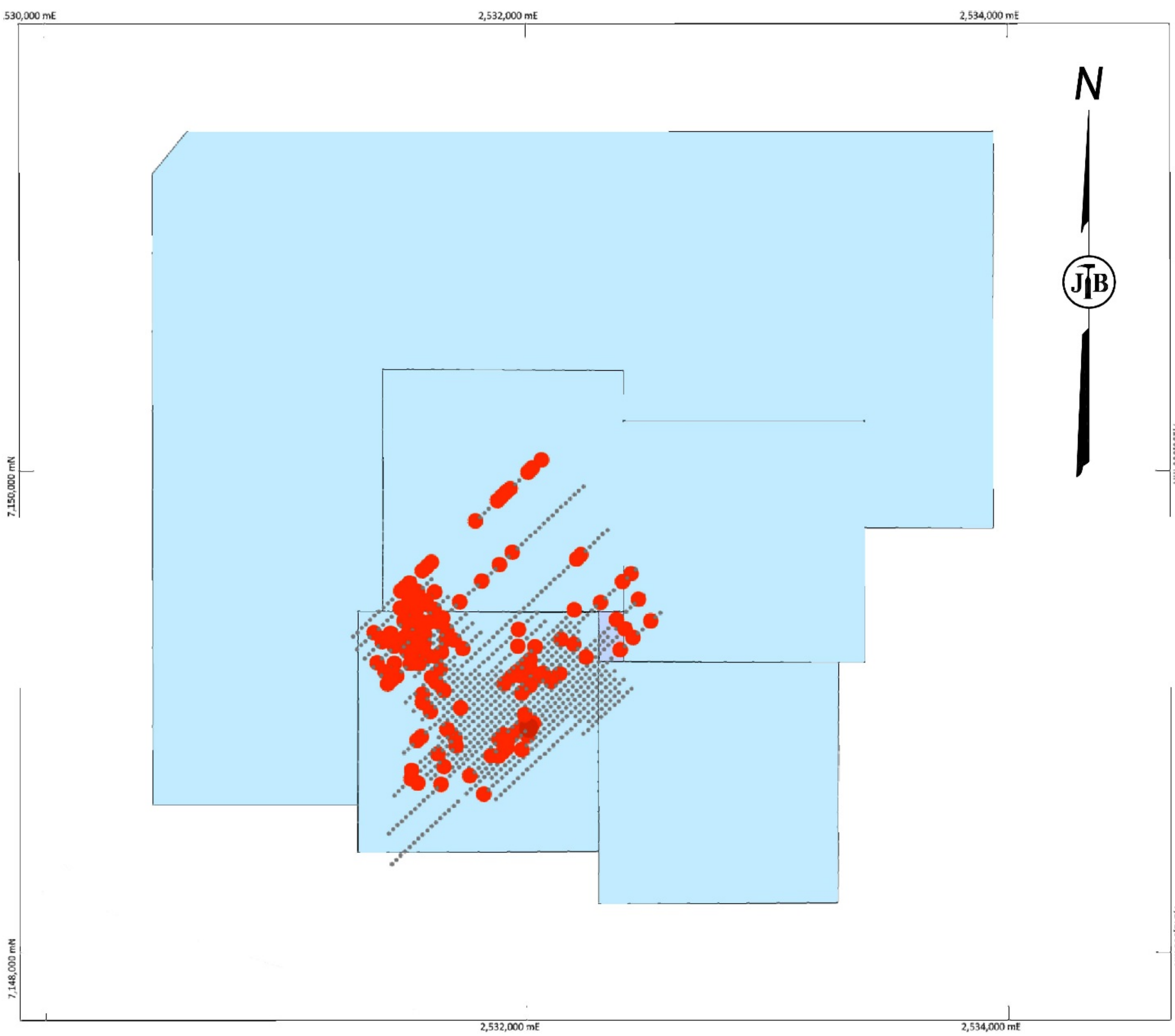
P:\CAD_GROUP-jfg\3832.001\FIGURE 9-9.DWG



Legend

Exploration Licence

Bedrock Samples (Cu ppm)

- 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

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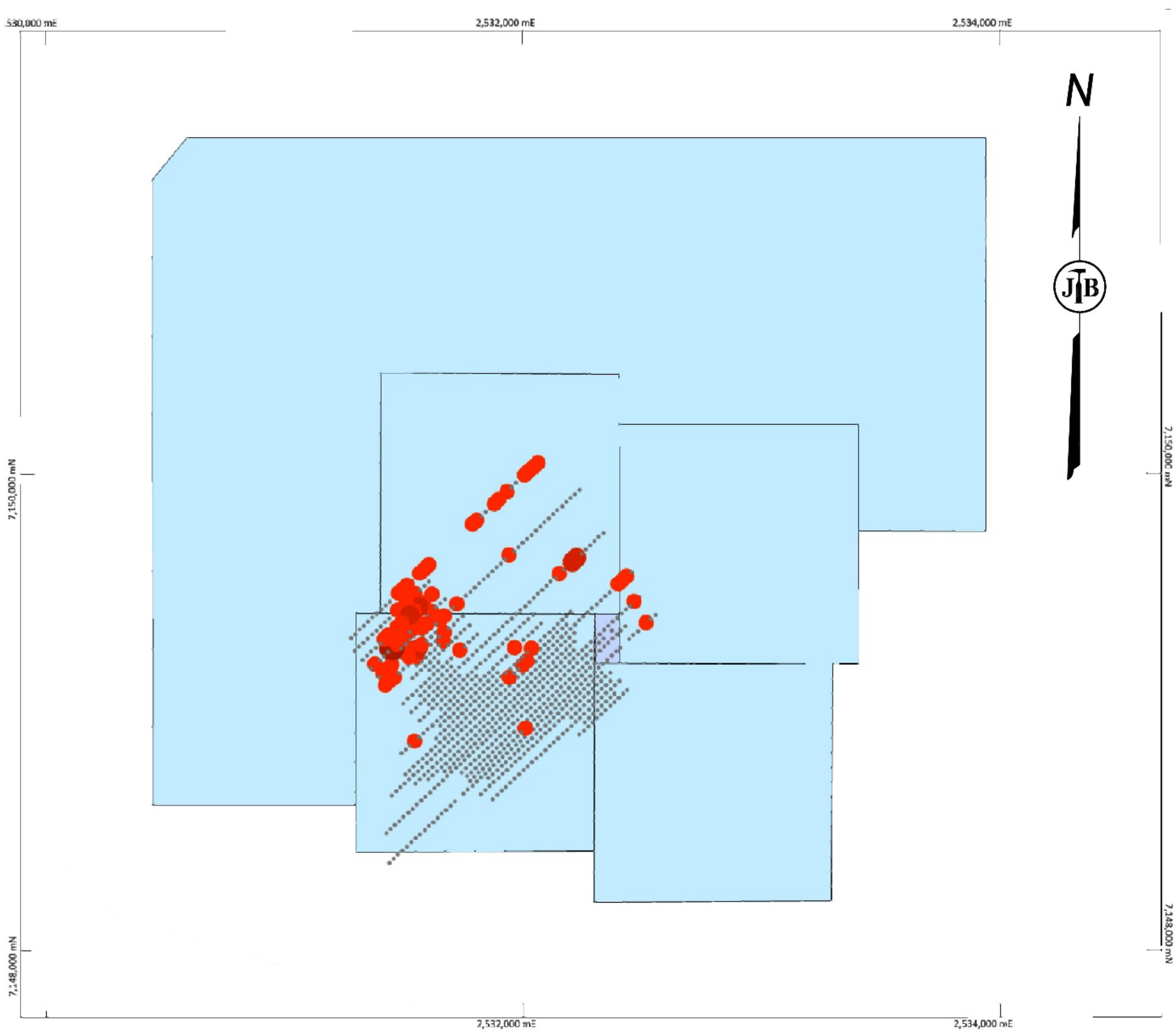
P:\CAD_GROUP–jfg\3832.001\FIGURE 9–10.DWG



Legend

Exploration Licence

Bedrock Samples (Mo ppm)

- 1000 - 99,999
- 500 - 1000
- 100 - 500
- 0 - 500

SCALE (km)

5 0 5

FIGURE 9.11

MAP SHOWING
BEDROCK SAMPLE LOCATIONS AT OLTAVA
MOLYBDENUM GEOCHEMISTRY
LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

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

9.5 Channel Sampling

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 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, overleaf. The results of the Astrock (2006) interpretation, 2010 GPR trial and BOYDs own observations are summarized below.

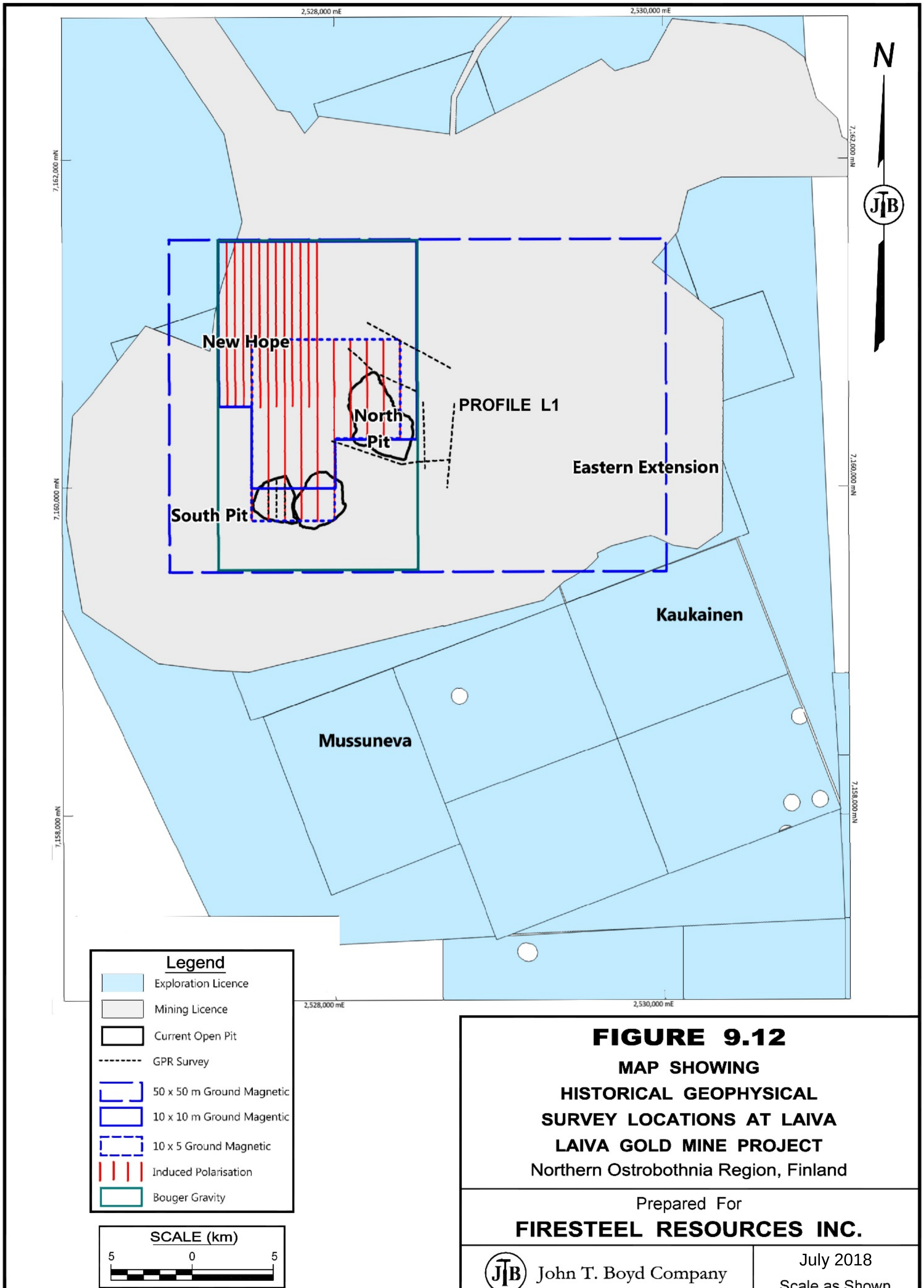

FIGURE 9.12

MAP SHOWING HISTORICAL GEOPHYSICAL SURVEY LOCATIONS AT LAIVA

LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

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July 2018

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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.

Laiva 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 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.

9.6.2 Bouger Gravity Data

Outokumpu commissioned a 2.4 km^2 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.
- 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. Results are inconclusive.

9.6.5 Ground Penetrating Radar Data

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 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 were provided with drill hole date 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 are currently conducting a ground magnetic survey at Laiva using a GSM 19 base station set to 15 second acquire and a GSM19W mobile until with inbuilt DGPS set at 0.2 second acquire. To date 270 line kilometres have been completed over the mine and areas immediately surrounding. Another 250 line kilometres are planned to include the Mussuneva and Oltava targets.

A nominal first pass line spacing of 50 m is being used. The program is designed to assist with structural interpretation and distinction of mafic-rich lithologies which typically host the highest grade mineralization.

9.7 Structural Geology Interpretation

In 2010 NORDIC commissioned Specialised 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 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 (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 whilst 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.

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10.0 DRILLING

10.1 Introduction

Several phases of drilling have been conducted at Laiva by Outokumpu, Endomines, and NORDIC between 1981 and 2014. The majority of drilling has been conducted at the Laiva gold deposit, with a limited number of historic exploratory holes drilled at Mussuneva, Kaukainen, and Oltava (Tables 10.1 and 10.2).

Table 10.1
Drilling Campaigns Conducted by Laiva

Operator	Period	Series	Type	No. of Holes	Meterage (m)
Outokumpu	1981-1987	LAI01-LAI95	Diamond	95	6,926
Endomines	1991-2005	LAD1-LAD12	Diamond	10	772
		Subtotal - Historic Drilling			7,698
Nordic Exploration					
	2005-2011	LAN501-LAN906	Diamond	209	42,670
	2007-2011	LAN703-LAN953	RC	92	13,350
	2010	1100001-1100040	Diamond	35	5,075
	2011	LAN11001-LAN11044	Diamond	30	8,801
	2012	LAN12011-LAN12053	Diamond	28	3,532
		Subtotal - Nordic Exploration			73,428
Nordic Geotechnical					
	2008	LAG1-LAG4	Diamond	4	607
	2010	AGMILL1-AGMILL11	Diamond	11	181
	2010	K200-K206	Diamond	6	93
	2010	PMILL22-PMILL24	Diamond	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	Diamond	92	3,010
	2013	LS1-LS119	Diamond	100	3,965
		Subtotal - Nordic Grade Control			68,596
Nordic Blasthole					
	2011-2012	712001-7100237	Blasthole	243	1,733
Total Nordic					144,687
Total Database					152,385

Table 10.2
Drilling Conducted by Oltava

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

For the purposes of mineral resource estimation, only the holes drilled by NORDIC 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 are, 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 and 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 at Laiva was largely inclined at -45 degrees 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 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 drilling was conducted using both diamond core holes (60,078 m) and RC holes (13,350 m).

Grade control drilling was conducted on a 12.5 m by 6.5 m grid pattern, inclined at -45 degrees and oriented north, and comprised 61,621 m of RC drilling and 6,975 m of diamond drilling.

Drilling statistics are shown in Tables 10.1 and 10.2, above (Source: NORDIC). Exploration and grade control hole collars and drill traces used in the BOYD mineral resource estimate are shown in Figure 10.1 below:

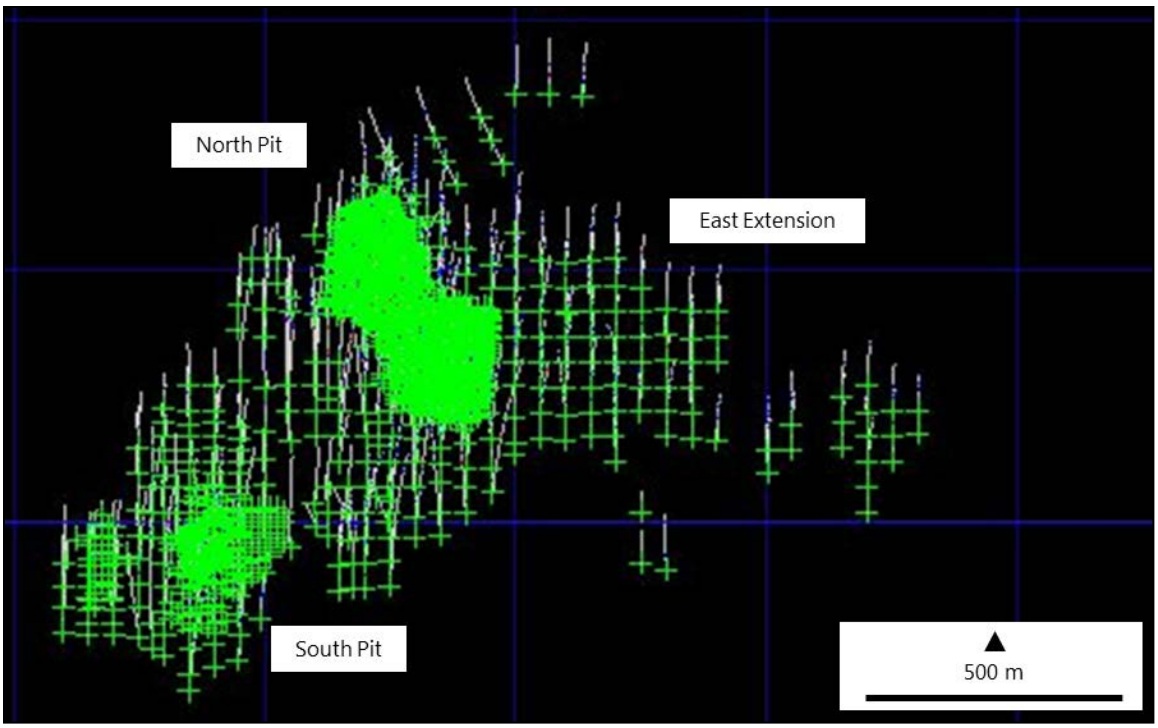


Figure 10.1
Laiva Drill Hole Collars and Traces

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 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 NORDIC'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.

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

NORDIC's drill data are 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 are 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 taken into account in the mineral resource estimation methodology and mineral resource classification (see Section 14).

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, 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 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

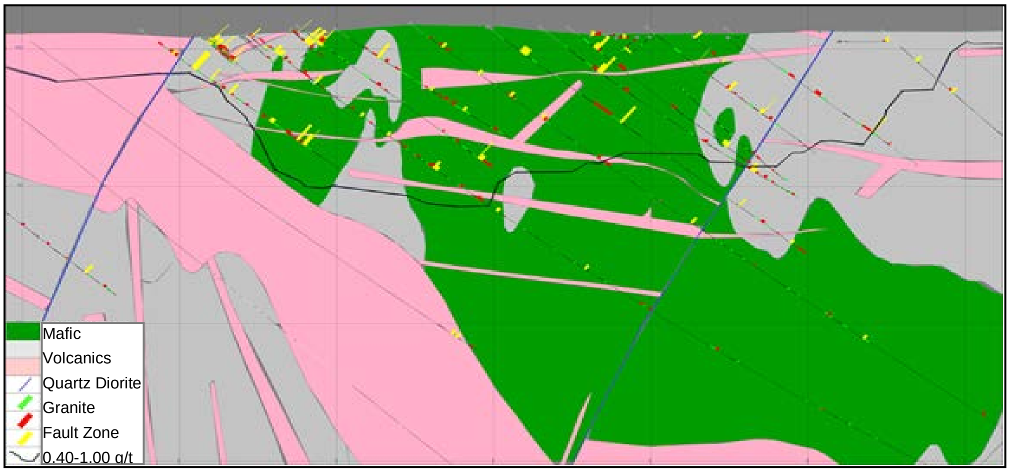
Drilling by NORDIC has defined gold mineralization at the Laiva Gold Mine over an area of 1,300 m by 600 m and to a maximum depth of 250 m below surface. The drilling and sampling procedures have been conducted to a good standard and the results of the drilling program are considered adequate for mineral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

10.14 Comments on Section 10

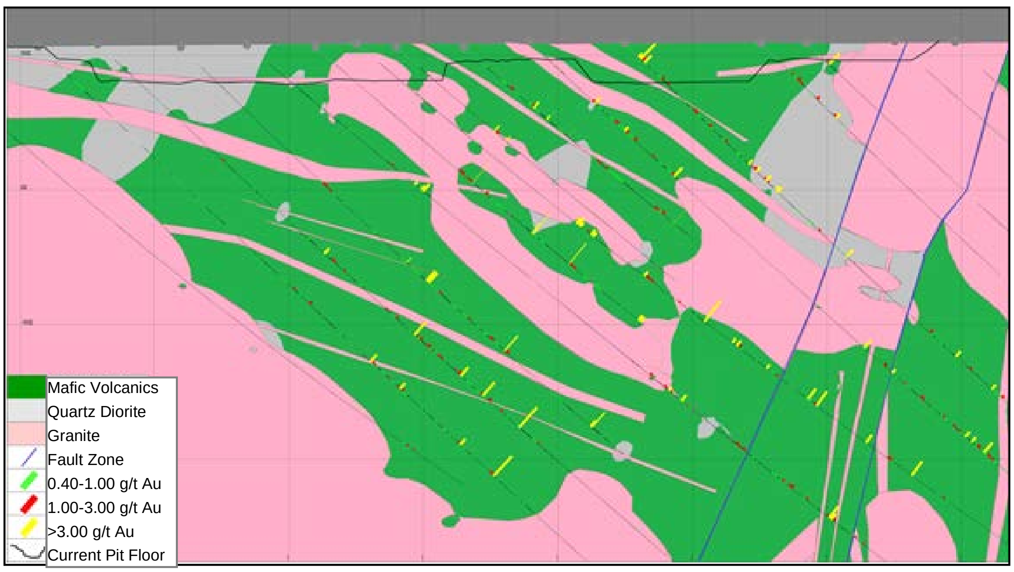
Overall the drilling, logging and sampling procedures as well as the database provided by NORDIC are considered reliable and adequate for mineral resource estimation purposes. BOYD strongly recommends that future sample intervals are determined by lithology and presence of mineralization. Whilst a nominal 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, dykes, and faults.

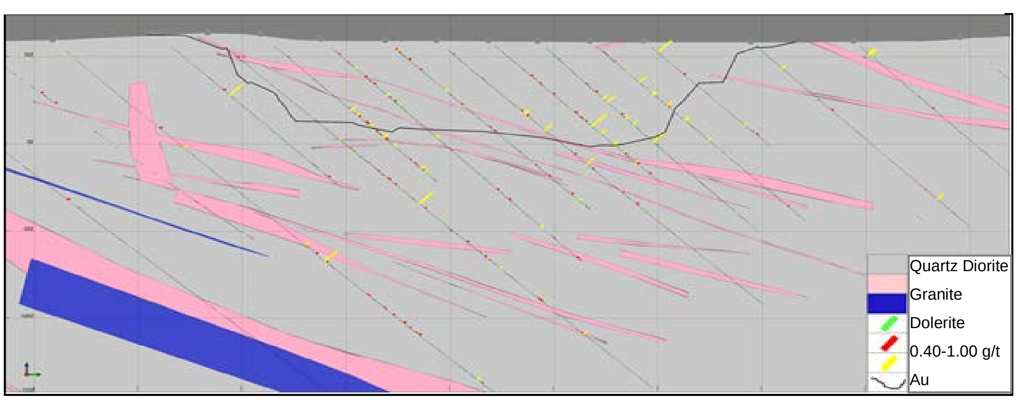
Representative cross sections showing the results of NORDIC's exploration drilling program are shown as follows:



Section 8200, Laiva North Pit West of the Red Shear Zone. Facing West.



Section 7900, Laiva South Pit. Facing West.



Section 8400, Laiva North Pit East of the Red Shear Zone. Facing West.

Figure 10.2
Laiva Cross Sections

10.15 Subsequent Infill Drilling

FTR recently completed a 2018 infill RC drill program immediately to the east of the North Pit and between the North and South Pits. A total of 47 inclined RC holes were drilled for a total of 2702 m. Holes were drilled normal to the strike of mineralization and were designed as infill holes in areas where previous drilling had intercepted mineralization. The deepest hole was drilled to a down-hole depth of 135 m.

All holes were cased with 6 in. steel pipe through the glacial till cover to depths of between 3 m to 9 m until seated in bedrock. Most sample intervals were dry and drilling practice and core recovery was good. Some assay results are pending.

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11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Introduction

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.

11.2 Chain of Custody

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 NORDIC by electronic mail confirming the sample numbers and safe receipt of samples.

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.

The above chain of custody maintained the validity and integrity of samples prior to analysis.

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 ALS are shown in in the following, respectively:

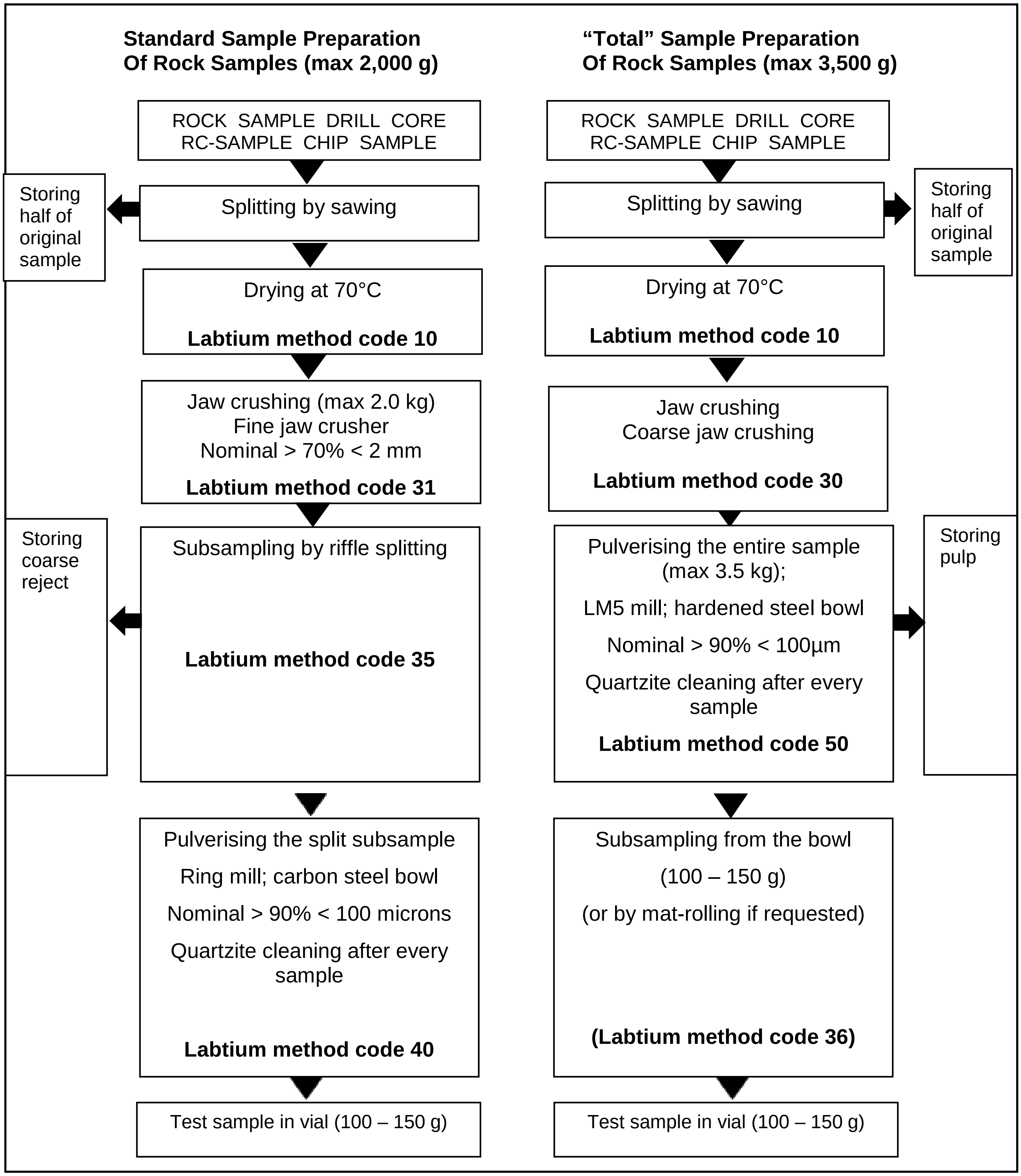


Figure 11.1
Sample Preparation Flowsheet (Labtium).

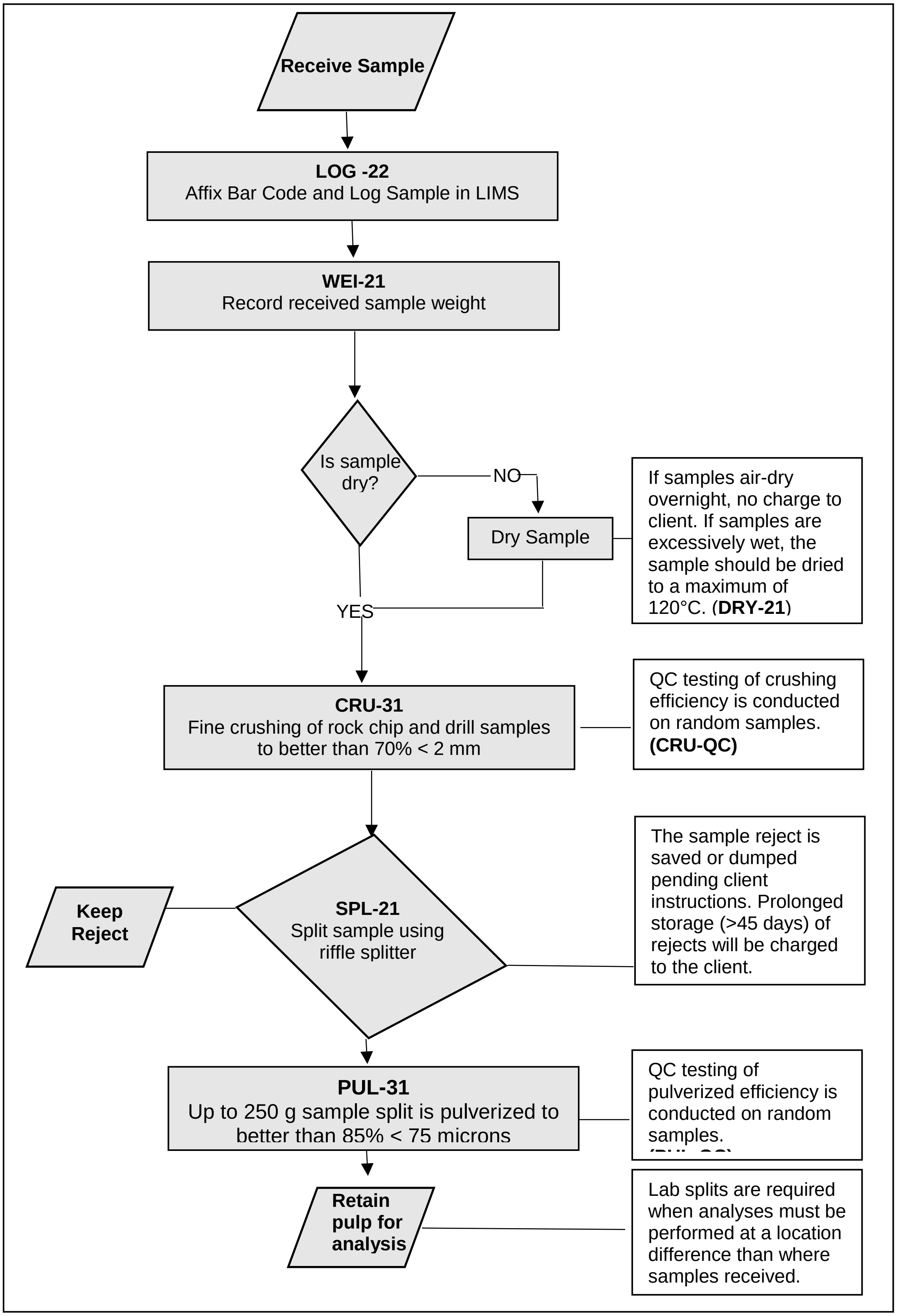


Figure 11.2
Sample Preparation Flowsheet (ALS Chemex).

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 as follows:

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
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 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.

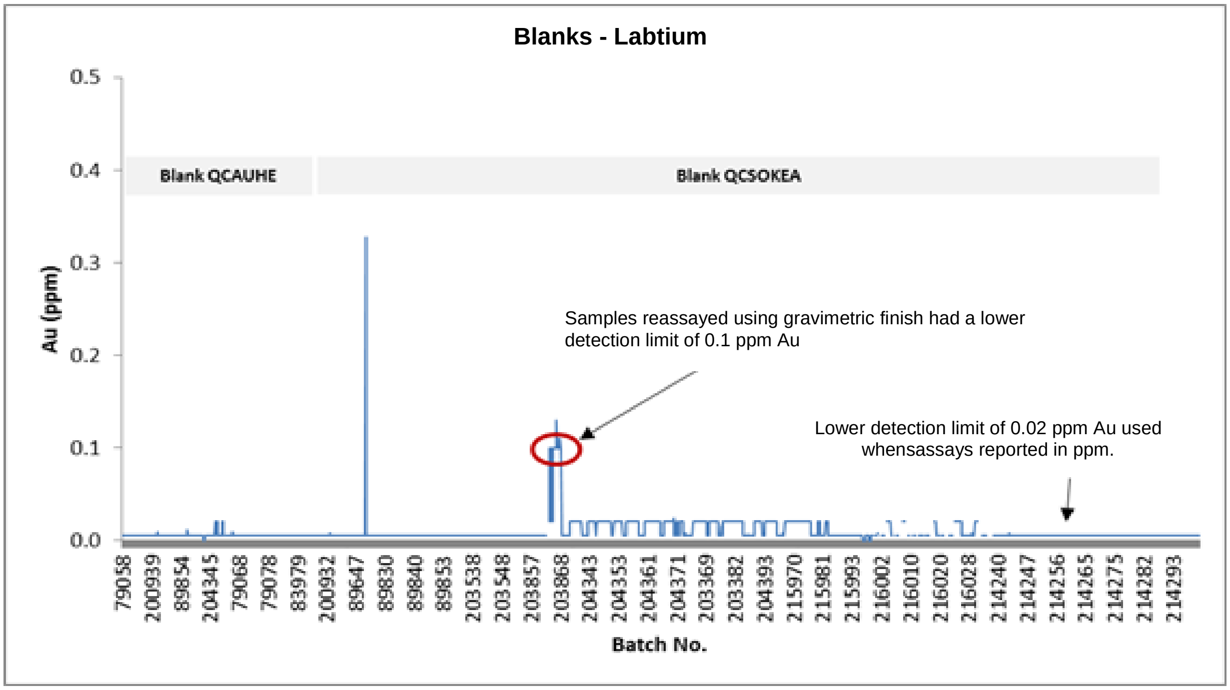


Figure 11.3
Labtium Blank Results

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.

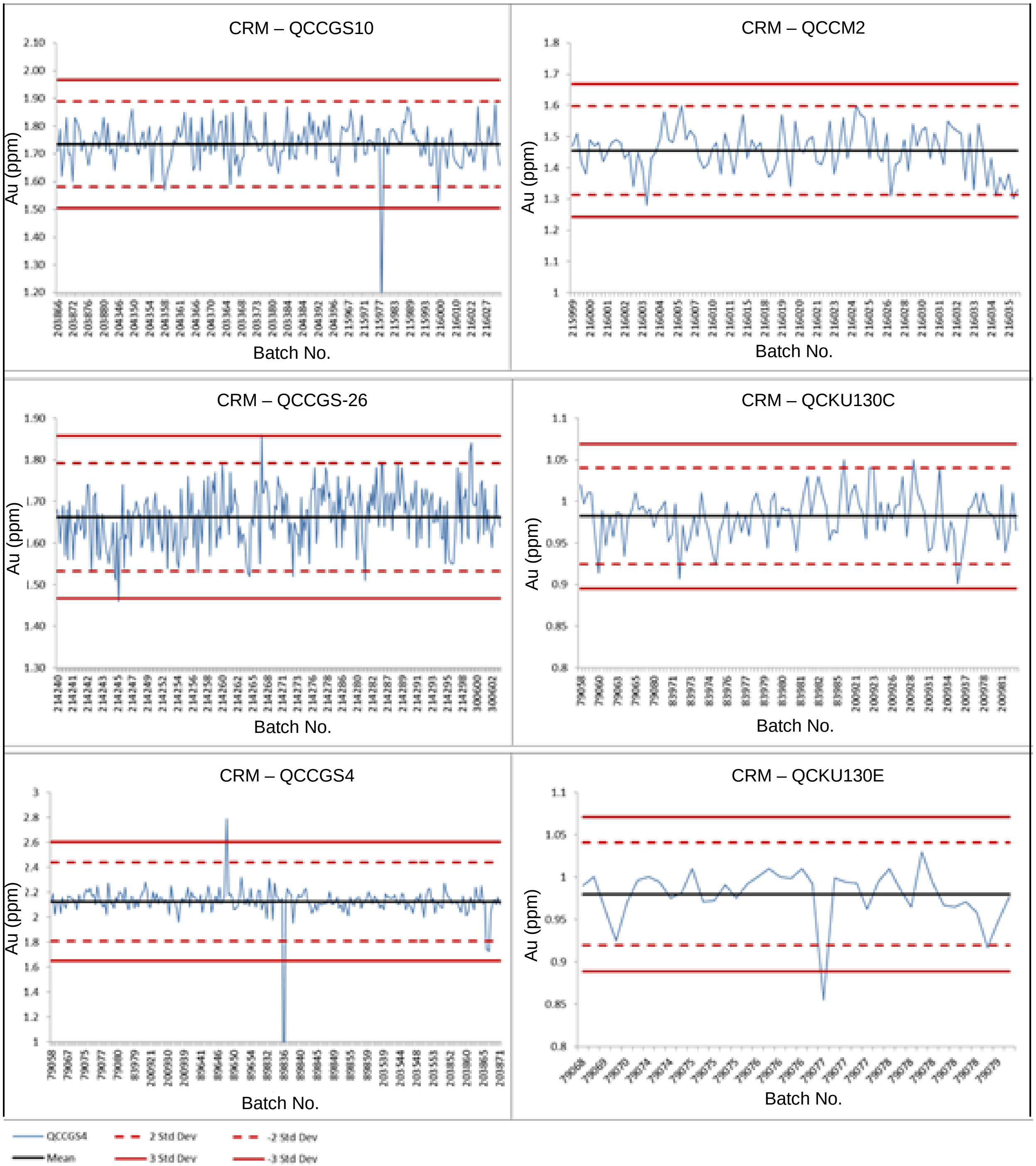
11.5.2.1 Labtium CRM

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. However, only the data mean and data standard deviations are reported for the CRM, and the certificate details of the CRM are unknown to the author. Without the certified mean and standard deviations of the CRM, it is only possible to accurately assess the precision of the Labtium assay results. The accuracy of the Labtium results must be assessed through assessment of the laboratory mean and other methods, such as umpire assays and verification sampling of sample pulps at an independent laboratory (see section 11.5.4). In addition, 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.



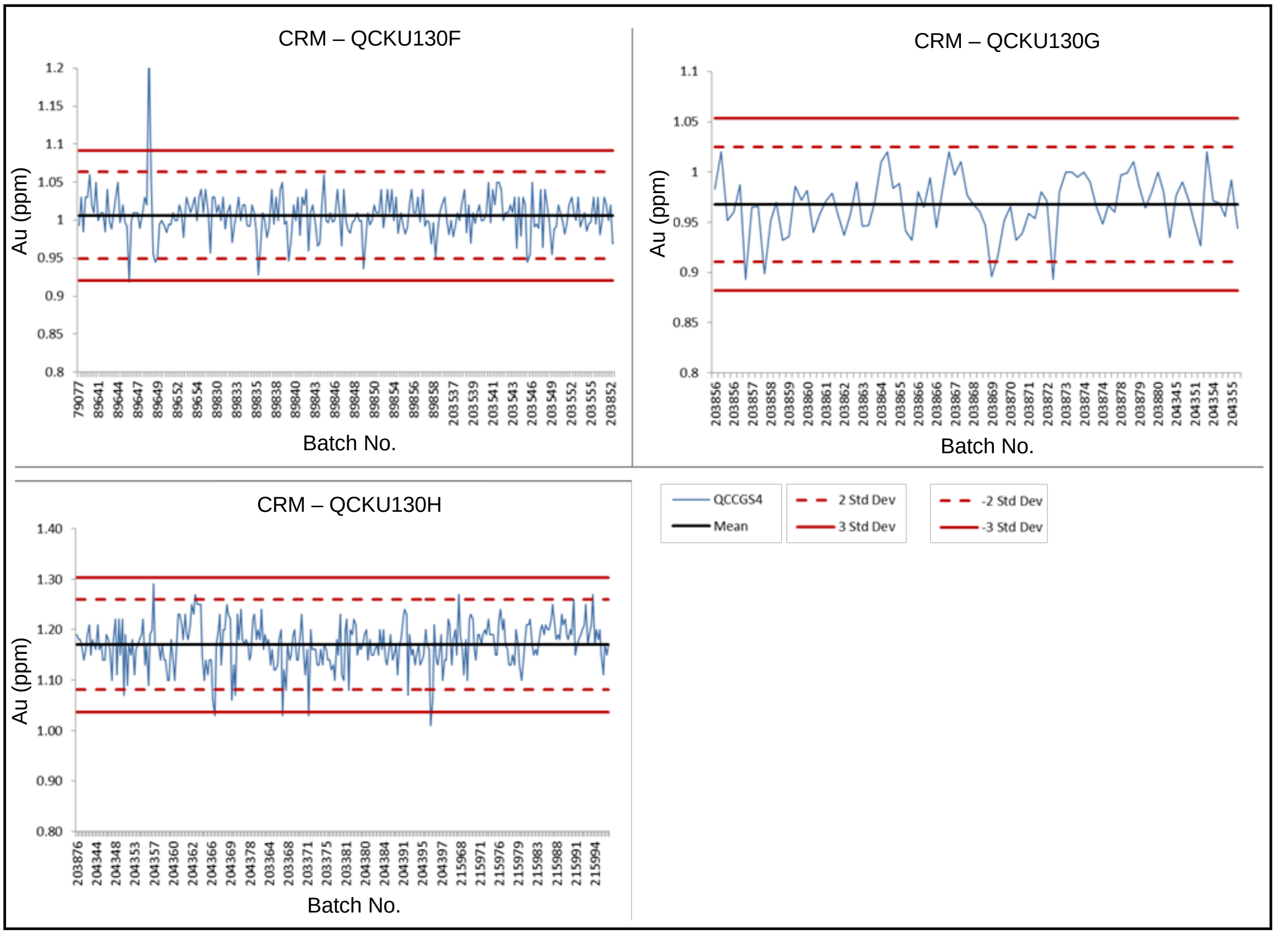


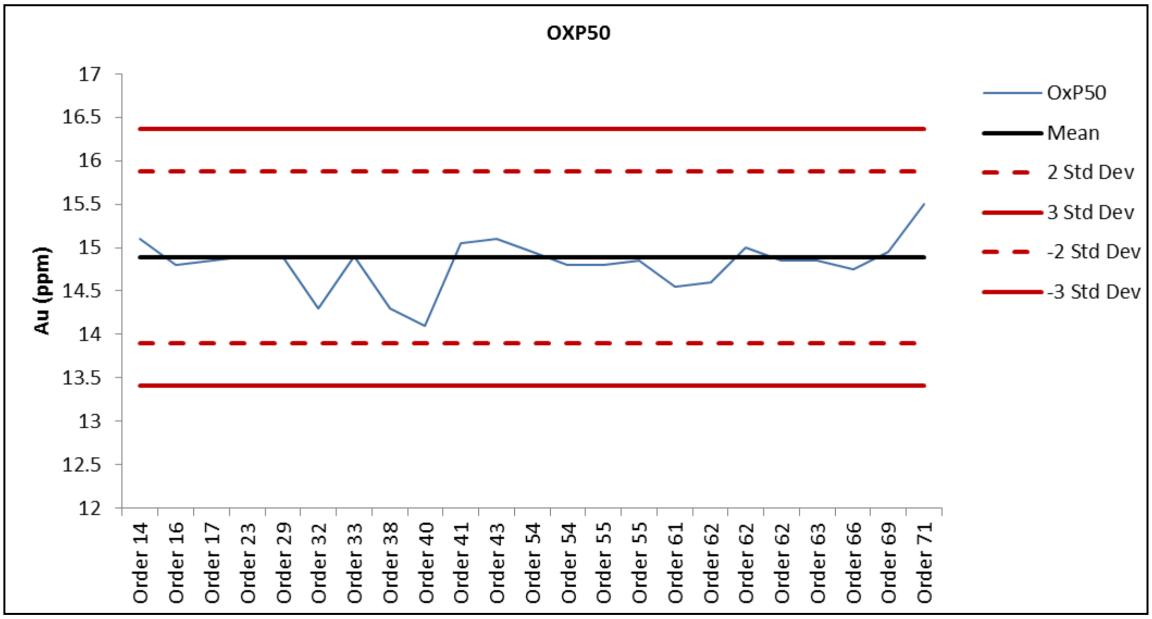
Figure 11.4
CRM Performance (Labtium)

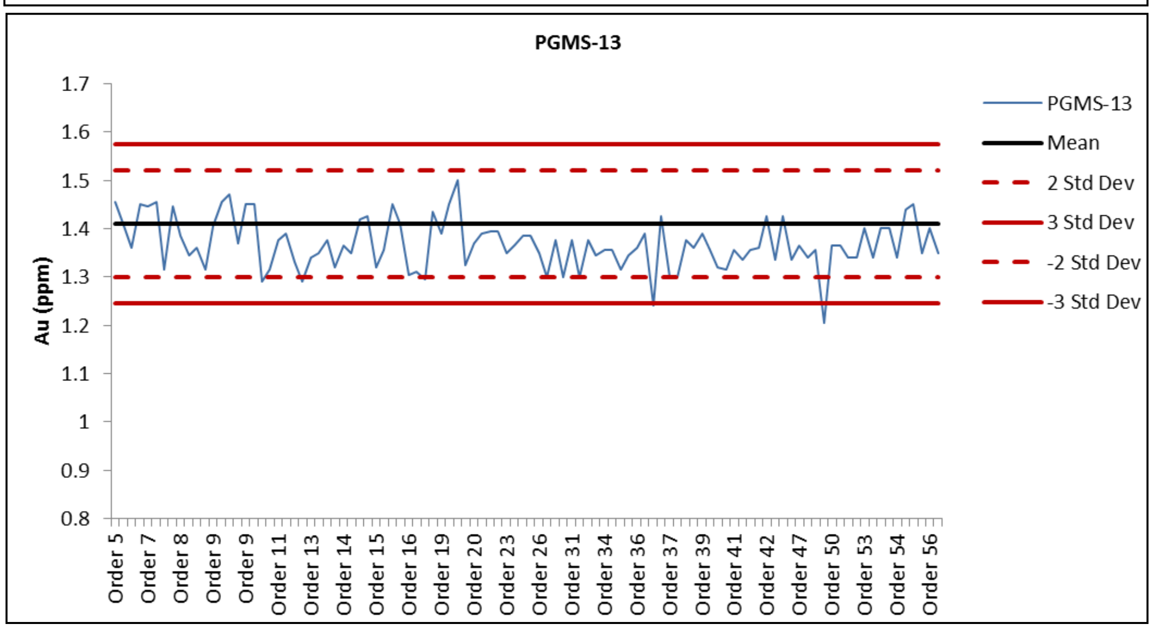
11.5.2.2 ALS CRM

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). 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.





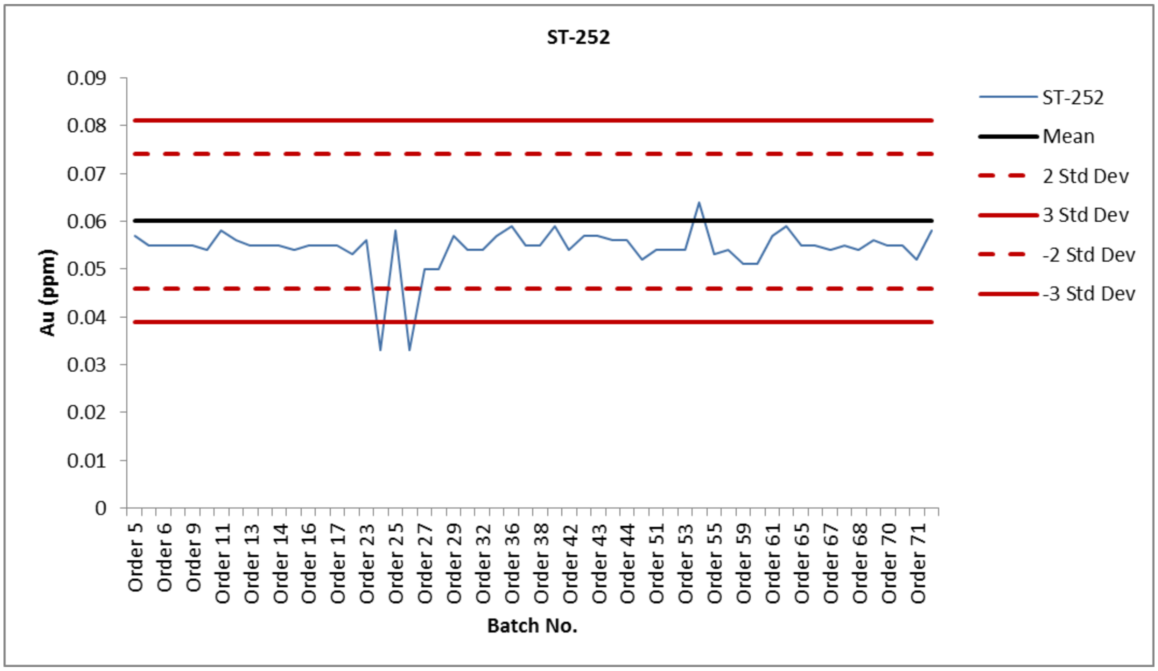


Figure 11.5
ALS CRS Results

<u>11.5.2.3 CRS CRM</u>

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 below).

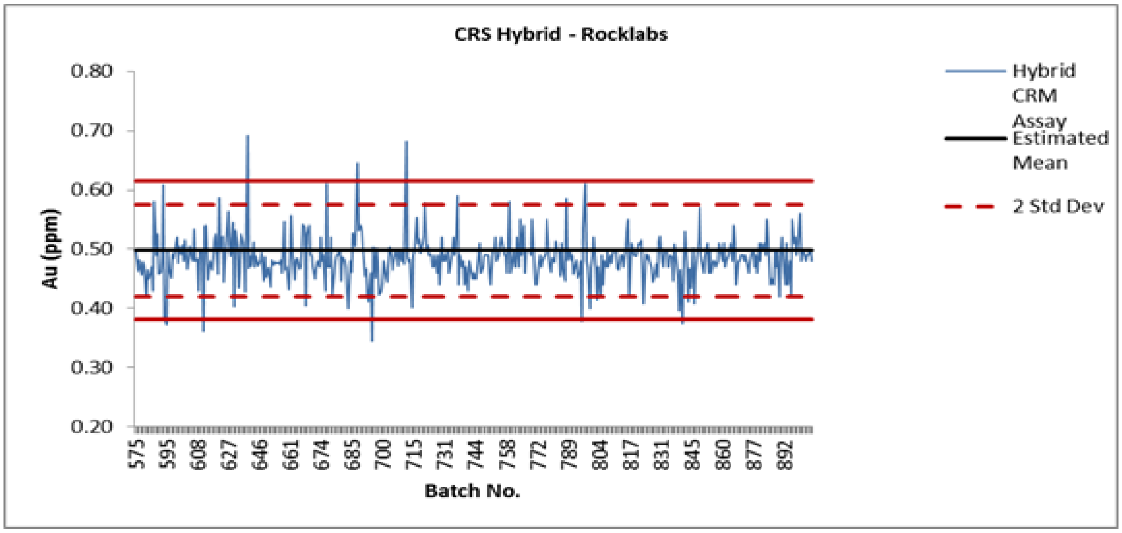


Figure 11.6
CRS Hybrid CRM Results, 0.50 ppm Gold Showing Data Standard (Deviation)

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 below).

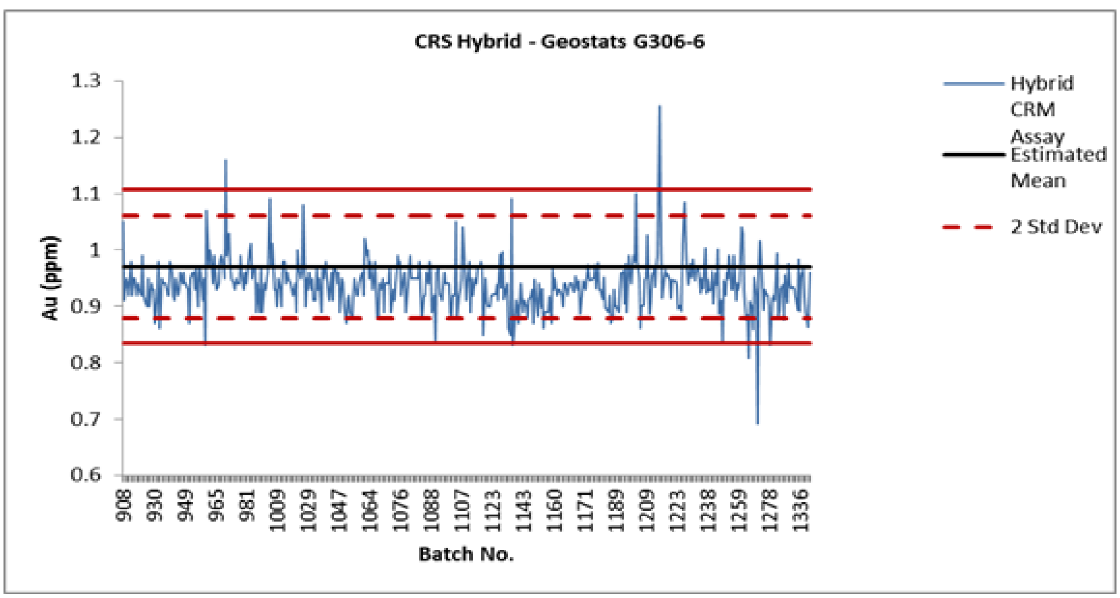


Figure 11.7
CRS Hybrid CRM Results, 0.97 ppm Gold
(Showing Standard Deviation Derived from the CRS Assay Results)

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 R^2 values of 0.9978, 0.9969, and 0.9945, respectively. This demonstrates good repeatability, as expected in a homogenized sample.

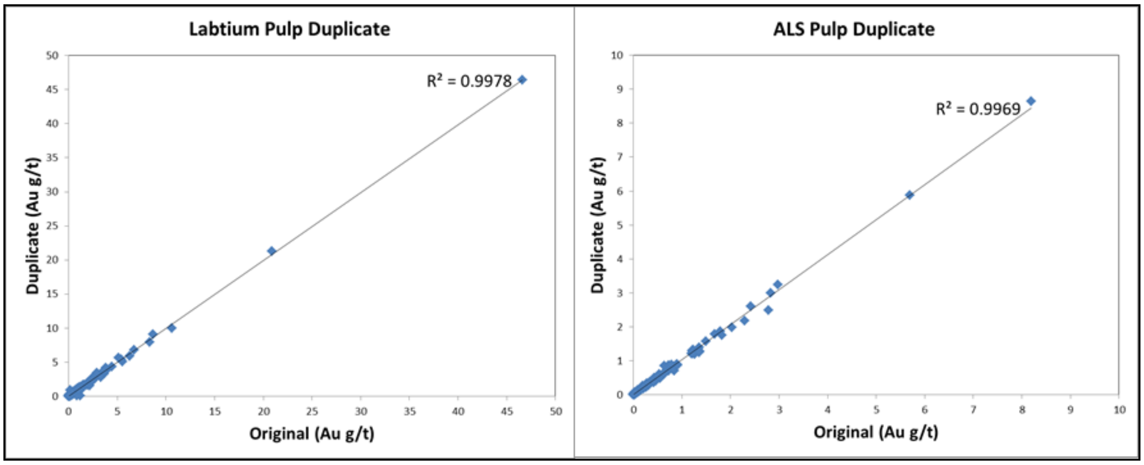


Figure 11.8
Laboratory Duplicate Results from Labtium and ALS

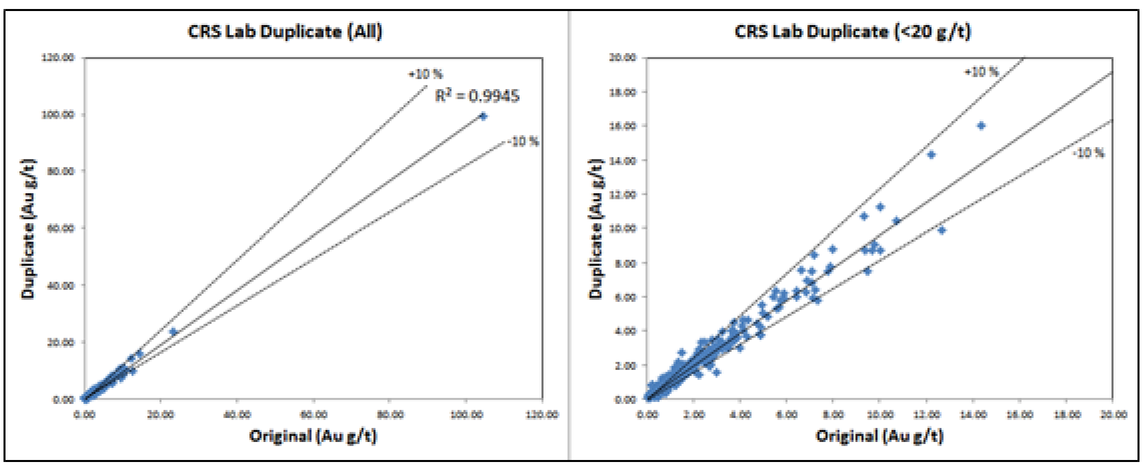


Figure 11.9
Laboratory Duplicate Results from CRS

11.5.3.2 Crush Reject Duplicate Samples

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 R^2 values of 0.9749 and 0.9654, respectively (as shown in Figure 11.10, below).

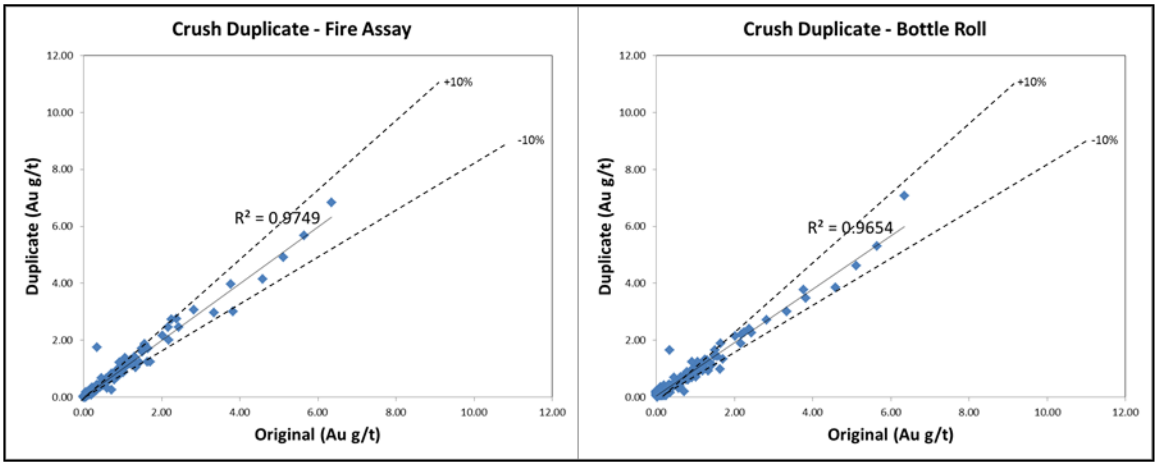


Figure 11.10
Crush Duplicate Results

11.5.3.3 Field Duplicate Samples

NORDIC conducted field duplicate analysis of 56 half cut drill core samples. Results show a moderate correlation with a R^2 value of 0.7916 (shown in Figure 11.11, overleaf). 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.

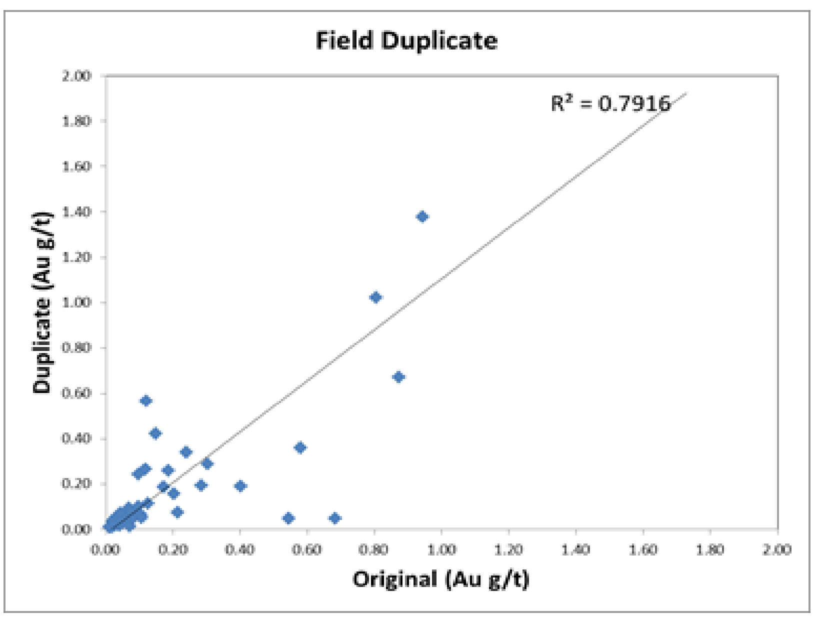


Figure 11.11
Field Duplicate Results from Half Core Samples

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).

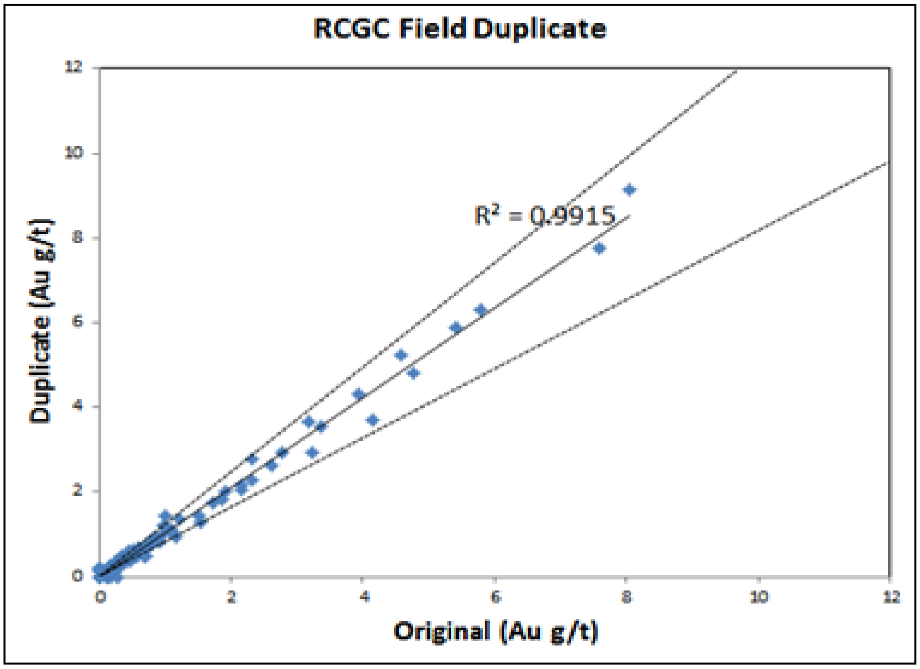


Figure 11.12
Field Duplicate Results from RC Grade Control Program

11.5.3.4 Check Assays

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 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 (five samples) (shown in Figure 11.13, below).

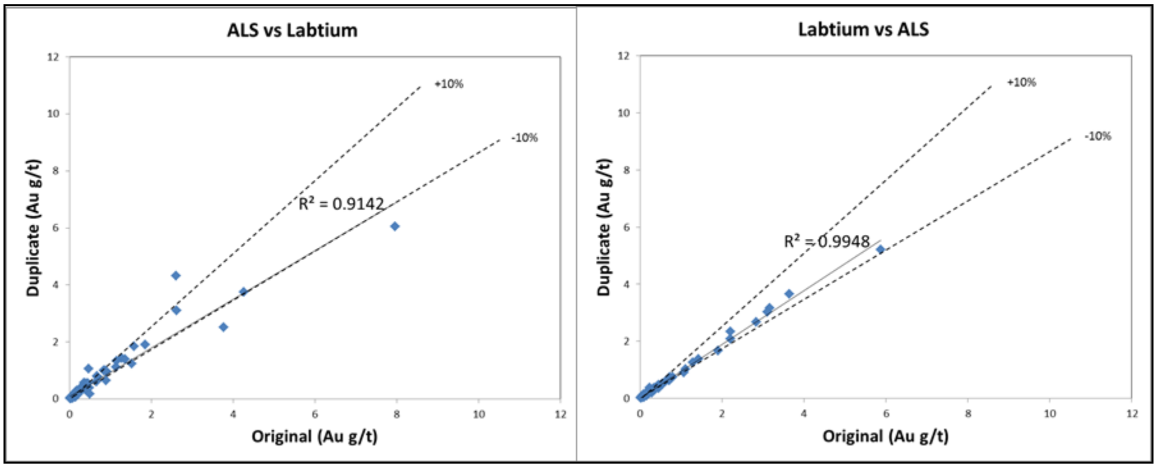


Figure 11.13
Check Assay Results

11.6 Conclusions

The sample preparation, security and analyses have in general been reported to be conducted to a high standard and the data are adequate for use in mineral resource estimation.

However, the QAQC program implemented by 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 following improvements to future QAQC programs are recommended:

- 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.

- The 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.
- More rigorous use of external CRM at gold grades between 2 and 5 g/t is recommended when using Labtium in the future, in order to better assess Labtium’s accuracy when reporting higher grade (>2 g/t gold) samples.
- CRS must be instructed to utilize a CRM which is certified for cyanide leach analysis with a mean grade appropriate or the mineralization observed at Laiva.

12.0 DATA VERIFICATION

The following measures were taken by BOYD to verify the data presented by NORDIC 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 more accurately represent the narrow, steeply dipping nature of mineralization and constrain the gold grade within the mineralized shear zones. 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 limitations or failure to conduct any of the data verification. As a result of the above data verification, BOYD is of the opinion that the data are adequate for use in mineral resource estimation which is the purpose of this Technical Report.

13.0 METALLURGICAL TEST WORK

13.1 Introduction

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

Gold grade in the sample tested was 7.7 g/t, arsenic content 0.26 wt% and copper content 0.02 wt%. 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 300-400 µm but the majority were less 10 µm. Minor gold occurred in silicates.

The mineralized material was found to be amenable to direct cyanidation with a gold extraction of 85% at a P80 = 75 microns 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 a lime consumption was 1.3 kg/t.

The Work Index figure in Mergan ball mill grinding was 15.3 kWh/h with the grind size of P80 = 60 µm.

Standard bulk flotation tests were completed on the mineralized material sample. The mineralized material 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 and ways of processing mineralization by gravimetric concentration with succeeding cyanide leaching, and possibly flotation. The objective was to find the most economic process and the lowest costs with a reasonable gold recovery.

The 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.
3. Gravity separation.
4. Fine flotation.
5. Thickening the produced process products for further leaching tests.
6. Bench-scale cyanide leaching 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.

13.2 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 (Figure 13.1, following this text), and were collected from blasted rock in the base of the pit. Sample descriptions are given in Table 13.1.

Table 13.1
Sample Descriptions

Sample ID	Host Rock	Weight (tonnes)	Head Grade 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 quartz diorite 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 mineralized material 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).

13.3 Pilot Scale Test Work Results

Results from the pilot scale tests showed that the mineralization could be ground autogenously to the desired fineness P80 0.075 to 0.085 µm with an energy consumption of 17 kWh/t to 20 kWh/t. In the tests with Type A mineralization, the energy consumption to a fineness of 0,088 µm was 16.9 kWh/t. In the tests with Type B mineralization, the respective figures were 0.076 µm and 19.7 kWh/t. The major part of the grinding tests was run with Type D mineralization, which had an average fineness of 0.087 kWh/t at an energy consumption of 19.0 kWh/t.

The principal flow sheet in gravity concentration consisted of a Falcon C400 centrifugal separator as the rougher stage, and a LG7 spiral as the 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 were reported to the primary classifying cyclone. The cyclone underflow was reported into a pebble mill and the overflow was 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 consisted of the 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 electro winning of the gold from the filtrate.
- Intensive CIL cyanidation followed by elution of the loaded carbon and electro winning.

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.

In the final test work, studies on re-grinding fineness, oxygen uptake rate, cyanide concentration, mixing and impeller selection, preg-robbing, and pulp density were carried out on the high grade concentrate.

The final results from high grade leach tests are presented in Table 13.2.

Table 13.2
Summarized 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.

In 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 13.2 presents the results in the final low grade leach tests; after 16-24 h 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 13.3
Results in 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 leach on the flotation tails were carried out. Figure 13.1 shows the leach tails versus flotation feed grade dependency:

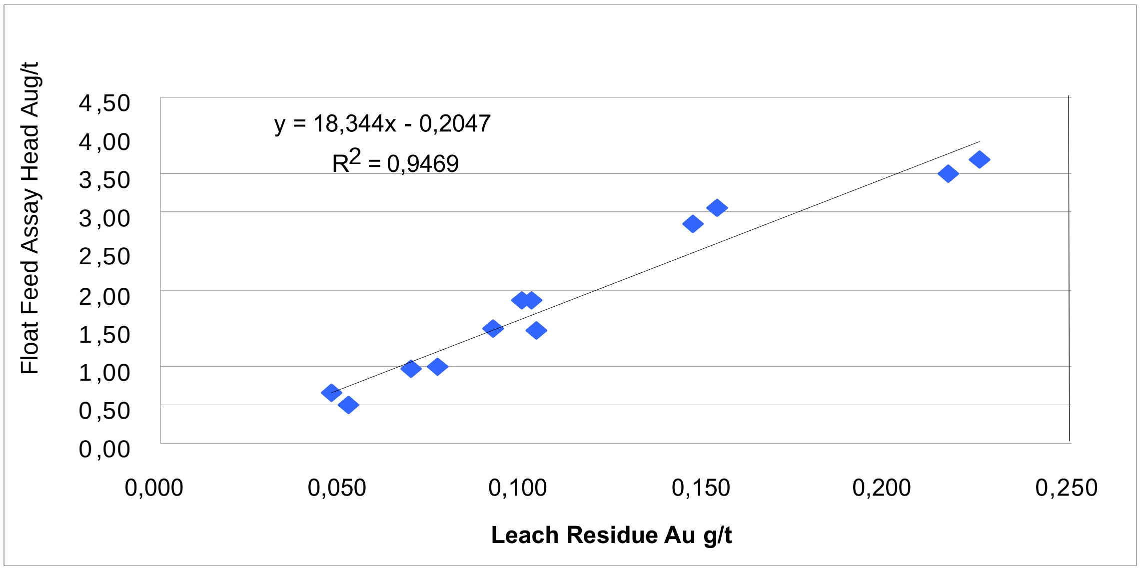


Figure 13.1
Leach Residue vs Float Feed Assay Head

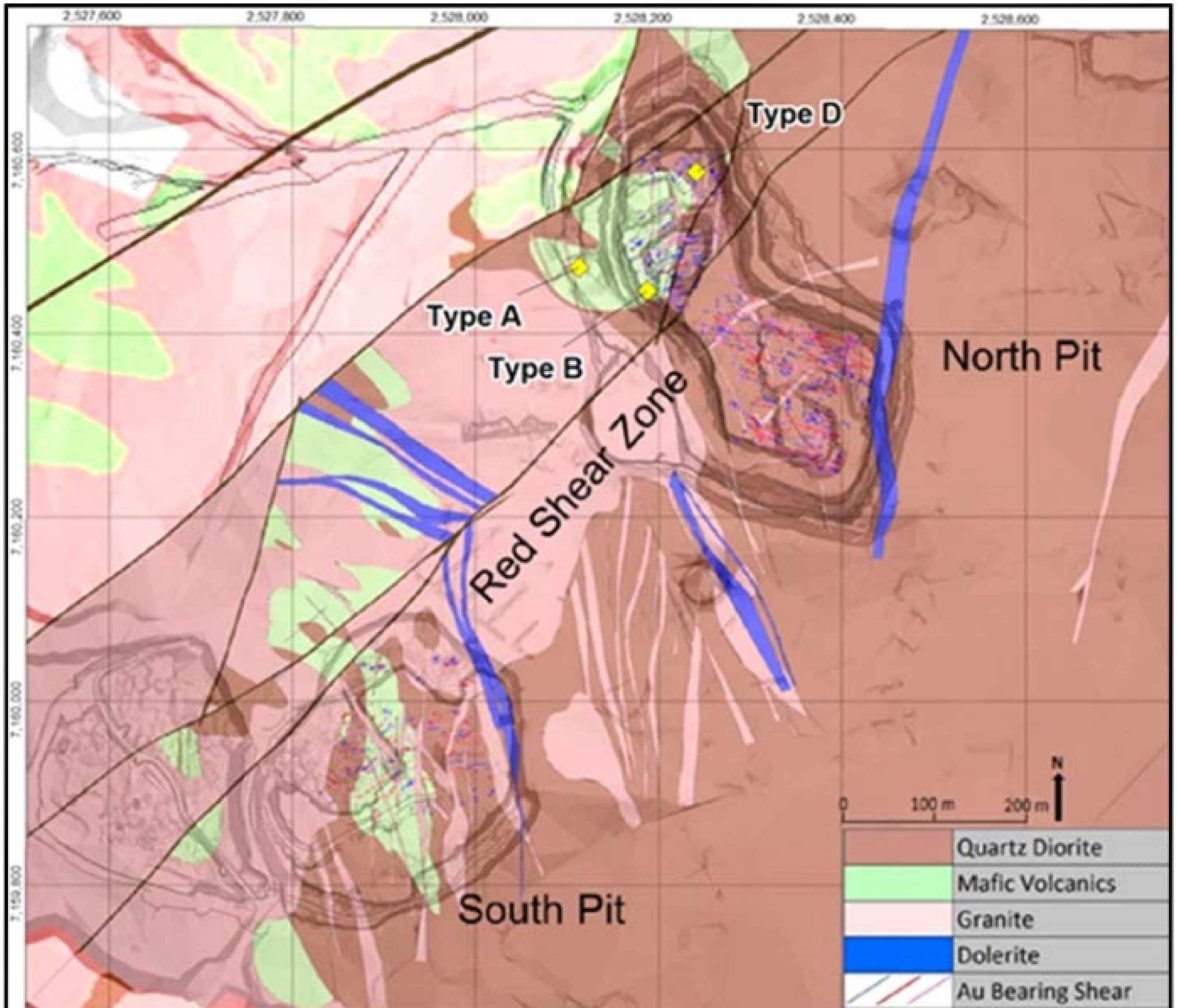
The additional metallurgical test program has been initiated by the Laiva process staff under the direction of the BOYD pocess group.The samples utilized for the test program are composites of drill cuttings collected during the recent drill program. The samples are considered to be representative of the expected plant feed for the initial years of the restart of mine operations.

The test program will be completed in a time frame to allow evlaution of the test results and implementation of any further identifed process plant improvments.

The test program inlcudes:

- Complete head analyses with gold assay by size.
- XRD, XRF, multiple-element ICP, and optical minerlaogy.
- Lab scale flotation concentration of each sample.
- Agitated reactor canide leaching of concentrates and tailings samples.
- Geochemical, mineralogical analyses and gold assay by size of chosen metallurgical products.

Following this page is Figure 13.2, Map Showing Location of Samples Collected in 2007 for Pilot Scale Test Work.



N

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

SCALE (m)
100 0 100 200

FIGURE 13.2

MAP SHOWING LOCATION OF SAMPLES COLLECTED IN 2007 FOR PILOT SCALE TEST WORK LAIVA GOLD MINE PROJECT

Northern Ostrobothnia Region, Finland

Prepared For

FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018

Scale as Shown

P:\CAD_GROUP-jfg\3832.000\FIGURE 13-1.DWG

14.0 MINERAL RESOURCE ESTIMATE

The mineral resource estimate described below has been reproduced from the October 17, 2017 BOYD Technical Report entitled "*Resource Estimate, Laiva Gold Project, Raahe, Finland*". It is included in this report as a reference to the previously described mineral resource estimate. There is no new or updated mineral resource estimated associated with this report.

14.1 Overview

BOYD was requested by FTR to provide a mineral resource estimate for its Laiva Gold deposit located near the town of Raahe, in the Northern Ostrobothnia Region of Western Finland. Although there have been several previous mineral resource estimates completed on the Laiva gold deposit, the mineral resource estimate requested would be the first mineral resource completed on the property in accordance with NI 43-101 and CIM reporting standards. This mineral resource estimate was undertaken by Sam J. Shoemaker, Jr. of BOYD, which is based in Canonsburg, Pennsylvania, USA, with a satellite office in Denver Colorado, USA, office. The effective date of this mineral resource estimate is 9 August 2017.

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, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.*

*An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops,* *trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*

An ***Inferred Mineral Resource*** *is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.*

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 deposit has had extensive exploration drilling before and during its operating period and much of those 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 and CIM standards.

14.2.1 Previous Mineral Resource Estimates

All of the previous mineral resource estimates completed on the property were not NI 43-101 compliant and as such are not presented within this section. These previous mineral resource estimates can be found in detail within Section 6.0 History.

14.2.2 Mineral Resource Estimation Procedure

The procedures used to estimate the mineral resource estimate on the Laiva gold deposit is outlined as follows:

1. Collect and validate the exploration drill hole dataset.
2. Convert the dataset into a format ready for loading into the Vulcan mine planning system.
3. Load the dataset into a Vulcan database.
4. Determine structural domains from mapped geology as well as exploration drill holes.
5. For each domain, run statistics on the raw drill hole data.
6. Determine a threshold gold grade to limit the influence of high-grade outliers from these raw statistics.
7. Determine a composite length from the raw statistics.
8. Construct the composite file using the length established in item 7 above.
9. Using Vulcan Implicit Modelling, develop a three-dimensional solid that represents both the numeric data as well as the geologic field observations.

10. Construct a block model flagging those blocks within the mineralized solid in each domain.
11. Develop variograms from the composite data in each domain to determine the search ellipsoid (major, semi-major, and minor directions).
12. Run the grade estimation only estimating gold grade from composites within each domain into blocks within that same domain.
13. Run the post estimation script including adding block density, topo adjusted density, and pit optimization flags.
14. Visually check the block model against the composites.
15. Complete statistical checks of the block model.
16. Prepare the block model for an economic pit optimization.
17. Run an economic pit optimization to determine a pit shell to use to calculate the estimated mineral resources.
18. Generate the open-pit constrained mineral resource estimate.

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 9 m by 9 m by 9 m sub-blocked to 3 m by 3 m by 3 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

The current resource estimate for the Laiva gold deposit is based on the drill holes whose assays were available by 29 June 2017: 398 diamond core drill holes totaling approximately 62,908 m and 3,129 RC drill holes totaling approximately 82,596 m for a total database of 3,527 drill holes with the total meters drilled being approximately 145,504 m. Figure 14.1, Page 14-4, shows the collars of these drill holes. All drill holes used were drilled starting in 2005 and later. The last drill holes were drilled in 2014. A number of blastholes taken during the previous operations were also available but were not used in the drill hole database. Additionally, a number of diamond core drill holes drilled prior to 2000 were available but were also not used in the drill hole database. In the case of the blastholes, in the opinion of BOYD, QA/QC of samples taken was insufficient to meet NI 43-101 requirements. As previously noted, drill data obtained prior to 2000 does not meet the requirements for qualification under NI 43-101.

14.2.4.2 Assays

Of the 119,378 gold assays available on 29 June 2017, all were used for the mineral resource estimation. For unsampled intervals, gold values were set to 0.000 g/t. Gold assays used were fire assays. Total assayed sample length is 119,547 m.

P:\CAD_GROUP-jfg\3832.001\FIGURE 14-1.DWG

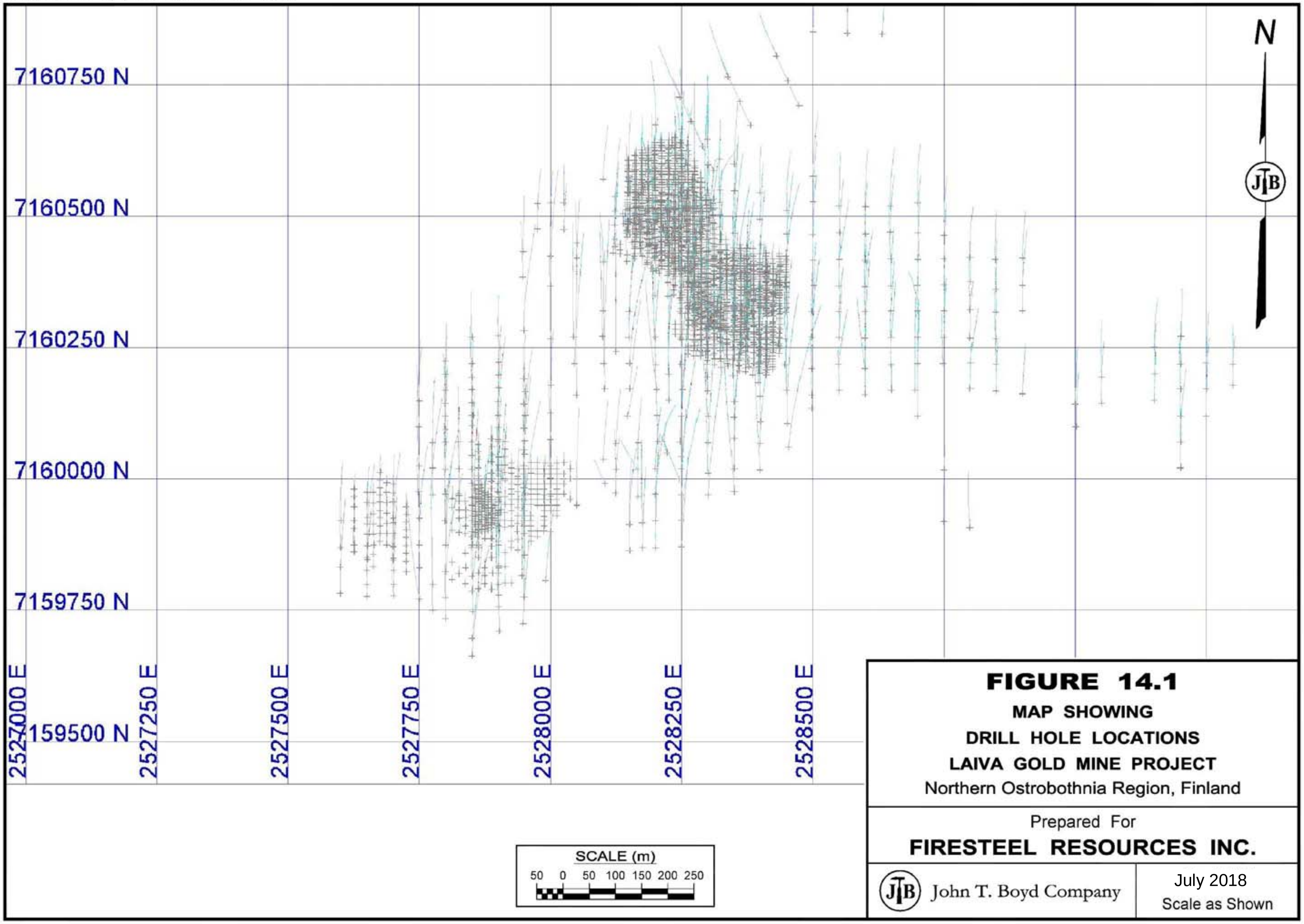


FIGURE 14.1

MAP SHOWING
DRILL HOLE LOCATIONS
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
FIRESTEEL RESOURCES INC.

John T. Boyd Company

July 2018
Scale as Shown

14.2.4.3 Density

There have been 5,772 density measurements completed on the Laiva Gold Deposit to date across 1,355 different drill holes. Based on these measurements, an average density for mineralized material of 2.83 t/m^3 are indicated for the deposit. For non-mineralized material, the density used is 2.70 t/m^3.

14.2.4.4 Topography

The topography of the area around the Laiva Gold Deposit is shown in Figure 14.2, Page 14-6. All contours are expressed in meters above sea level. Minor contours are every 5 m while major contours are every 25 m.

14.2.5 Laiva Gold Deposit Data Analysis

14.2.5.1 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.

P:\CAD_GROUP–jfg\3832.001\FIGURE 14–2.DWG

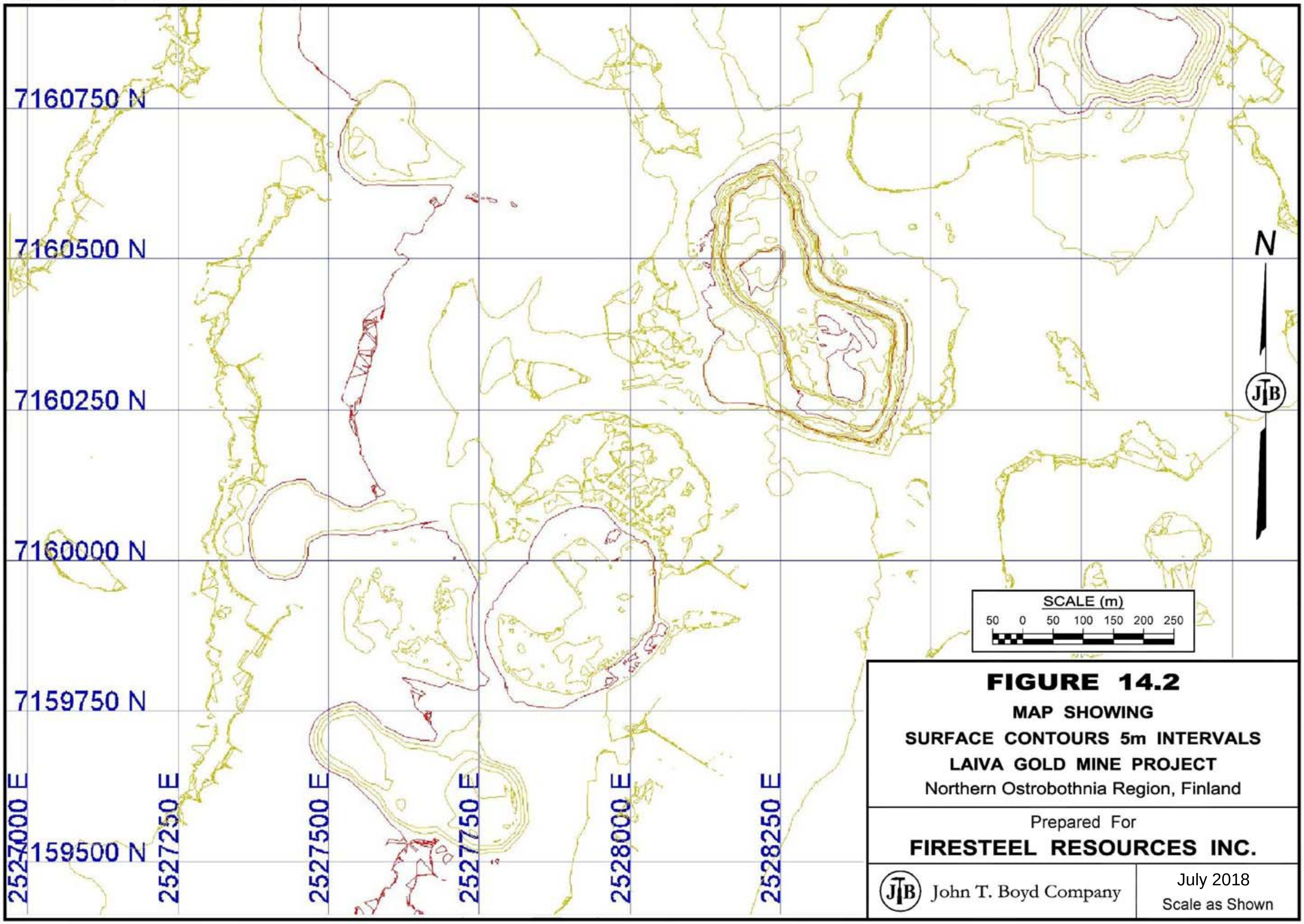


FIGURE 14.2

MAP SHOWING
SURFACE CONTOURS 5m INTERVALS
LAIVA GOLD MINE PROJECT
Northern Ostrobothnia Region, Finland

Prepared For
FIRESTEEL RESOURCES INC.

John T. Boyd Company | July 2018 | Scale as Shown

These four domains were used to flag the composites for later variography as well as statistics.

To produce the individual mineralized solids, a grade shell was generated using the observed strikes and dips 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 14.3 shows the interpreted structural domains as well as the mineralized solids.

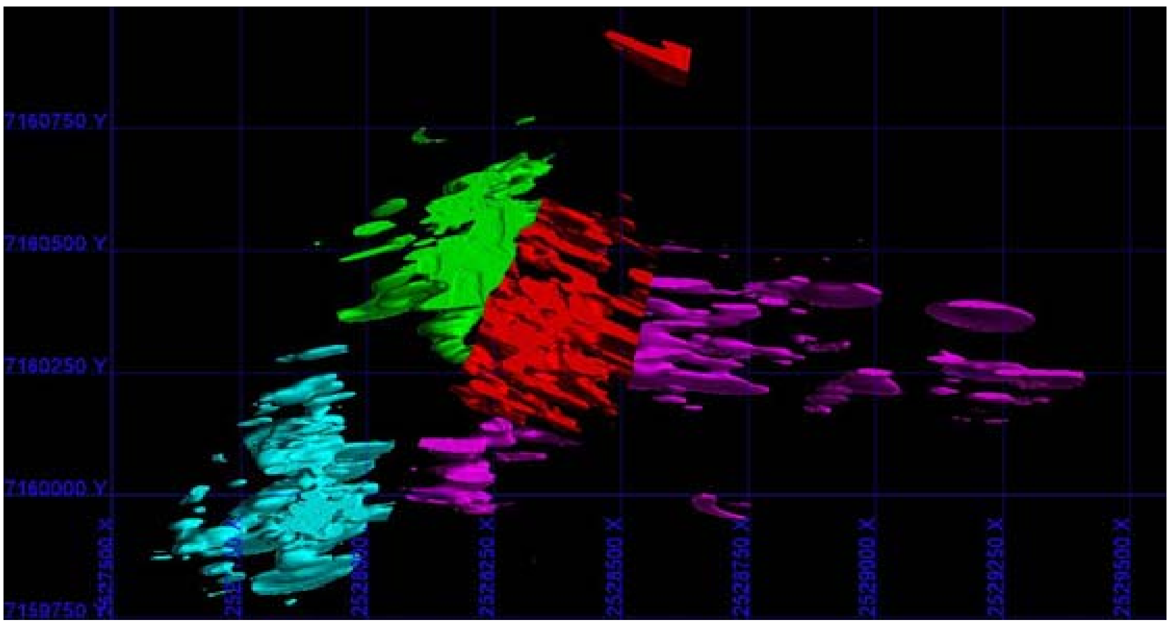


Figure 14.3

Laiva Structural Domain Orientation and Mineralized Solids

<u>14.2.5.2 Statistics</u>

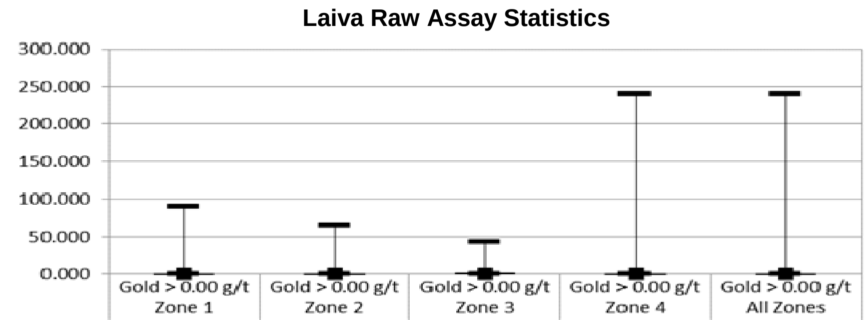
Statistics were run on both the individual domains as well as the overall exploration database. The results of this analysis are shown below:

Table 14.1

Domain Statistics

Domain:	All Domains	Domain 1	Domain 2	Domain 3	Domain 4
Number of samples:	80,833	4,398	19,847	5,477	51,111
Minimum:	0.010	0.010	0.010	0.010	0.010
Maximum:	240.000	91.000	65.100	43.500	240.000
Range:	239.990	90.990	65.090	43.490	239.990
Average:	0.642	0.798	0.770	0.929	0.548
Standard deviation:	1.930	2.460	1.900	1.840	1.890
Variance:	3.720	6.070	3.600	3.380	3.570

Laiva Box plot statistics for these results are shown in Figure 14.4:



	Gold > 0.00 g/t Zone 1	Gold > 0.00 g/t Zone 2	Gold > 0.00 g/t Zone 3	Gold > 0.00 g/t Zone 4	Gold > 0.00 g/t All Zones
Q1	0.100	0.070	0.190	0.050	0.060
– Min	0.010	0.010	0.010	0.010	0.010
■ Median	0.250	0.260	0.400	0.180	0.210
▲ Mean	0.798	0.770	0.929	0.548	0.642
– Max	91.000	65.100	43.500	240.000	240.000
Q3	0.630	0.700	0.940	0.440	0.540
NSamples	4398	19847	5477	51111	80833

Figure 14.4
Laiva Raw Assay Box Plot Statistics

14.2.5.3 High Value Gold Grade Limits

High, outlier gold 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 log-normal probability plot was generated for each domain.

Using these plots, the threshold gold grade was selected where the data start to break up or where there is a significant slope change in the plot. An example of this is shown in for all domains in Figure 14.5.

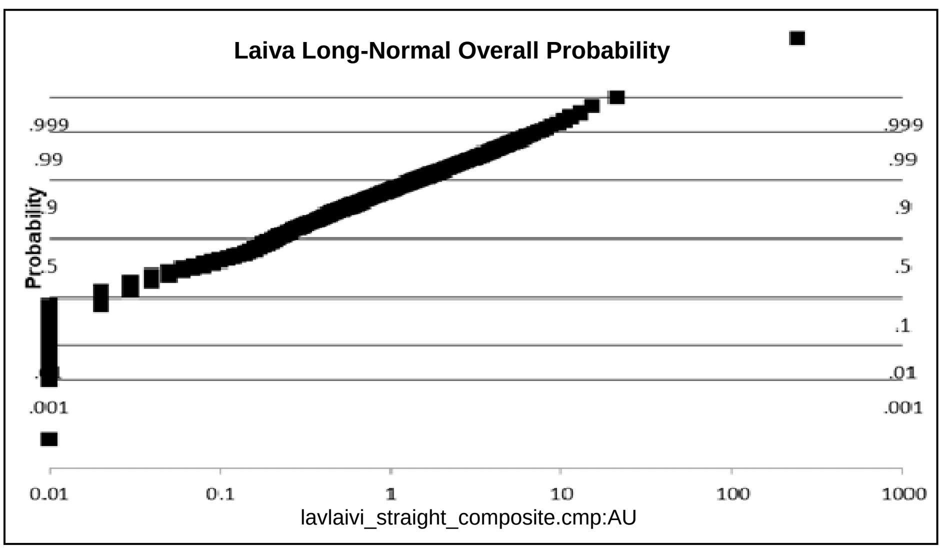


Figure 14.5
Laiva Straight Composite cmp: AU

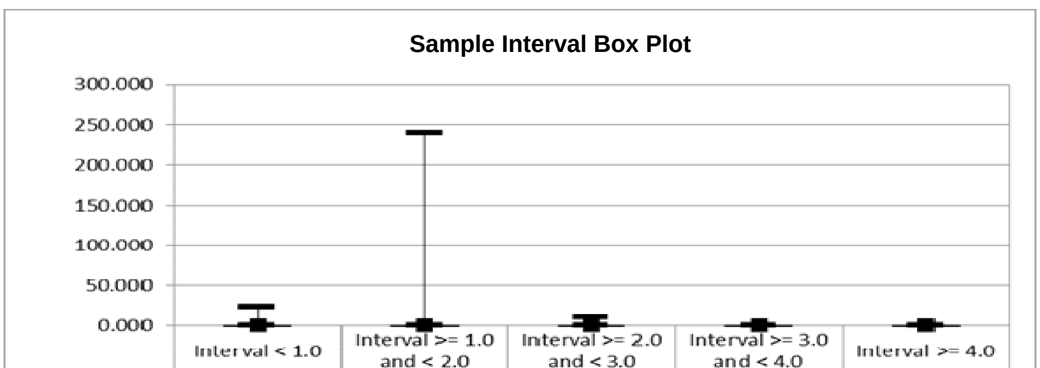
In this example, the threshold gold grade would be 21 g/t.

The threshold gold grade can be applied as a fixed cap or as having a limited area of influence. Several test estimations were running considering no grade capping, grade capping, or a limited area of influence. The best match to the underlying composites was no capping, but a limited area of influence was used since it closely matched no capping. The capped approach resulted in a poor correlation to the underlying composites.

To apply this threshold, during block estimation a limited area of influence was used that limited these samples to a maximum of 10 m in the major and semi-major search ellipsoid and 5 m in the minor ellipsoid. This threshold and the limited search ellipsoid limits the impact of high-grade outliers.

<u>14.2.5.4 Compositing</u>

Statistics were run on the assay database examining the number of samples for sample lengths in 1.0 m increments through a total length of 4.0 m. The purpose of this analysis was to determine what sample length was associated with the total number of samples. Figure 14.6 shows the results of this analysis:



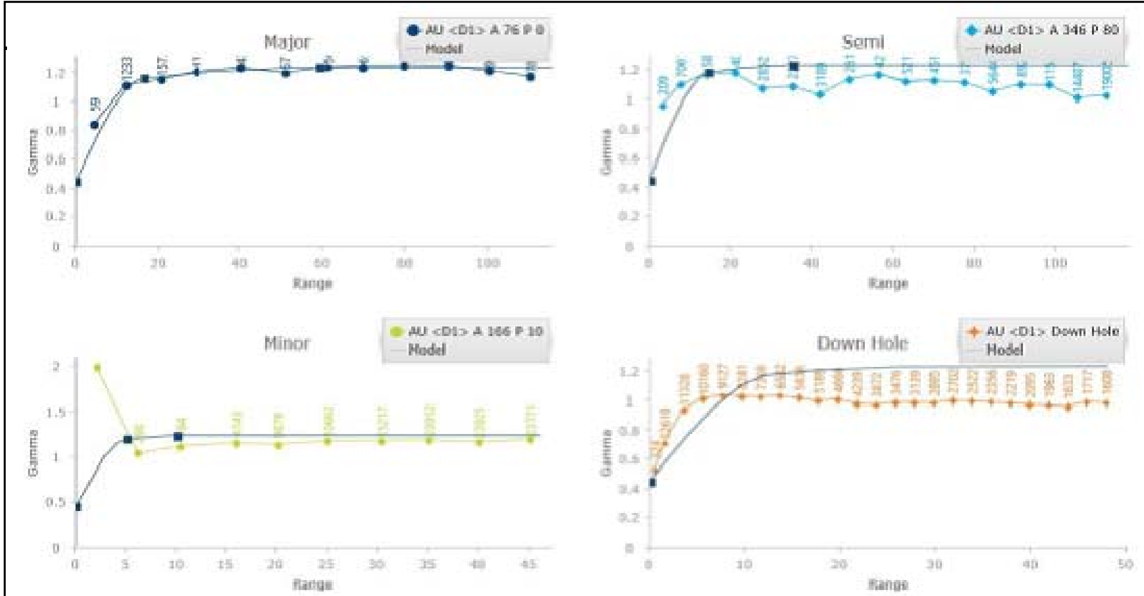
	Interval < 1.0	Interval >= 1.0 and < 2.0	Interval >= 2.0 and < 3.0	Interval >= 3.0 and < 4.0	Interval >= 4.0
Q1	0.000	0.000	0.000	0.000	0.000
▬ Min	0.000	0.000	0.000	0.000	0.000
■ Median	0.130	0.060	0.000	0.000	0.000
▲ Mean	0.765	0.430	0.151	0.007	0.000
▬ Max	23.050	240.000	10.700	0.470	0.250
Q3	0.580	0.330	0.010	0.000	0.000
NSamples	897	118926	158	91	1113

Figure 14.6
Sample Interval Box Plot

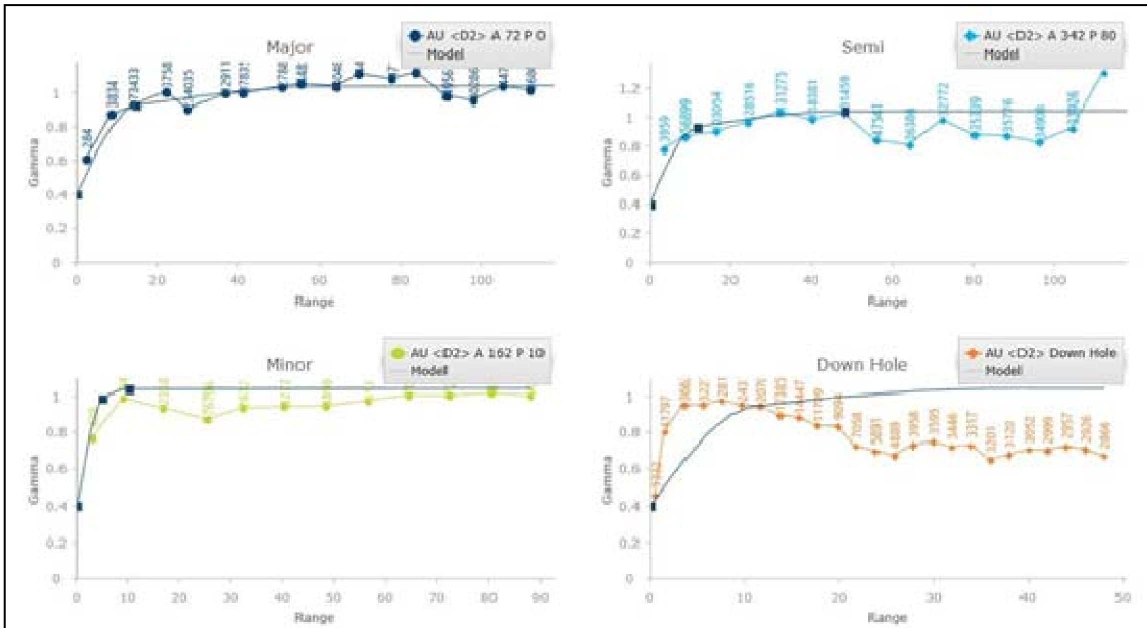
Most samples taken were at a length of 1.0 m or less. Based on this, a composite length of 1.0 m was selected, broken by the mineralized solid domains. Composites less than 1.0 m were divided by the run length (1.0 m). This composite length was selected to better reflect the actual breakdown of the mineralization in the individual drill holes within the mineralized zone.

14.2.6 Search Ellipsoid

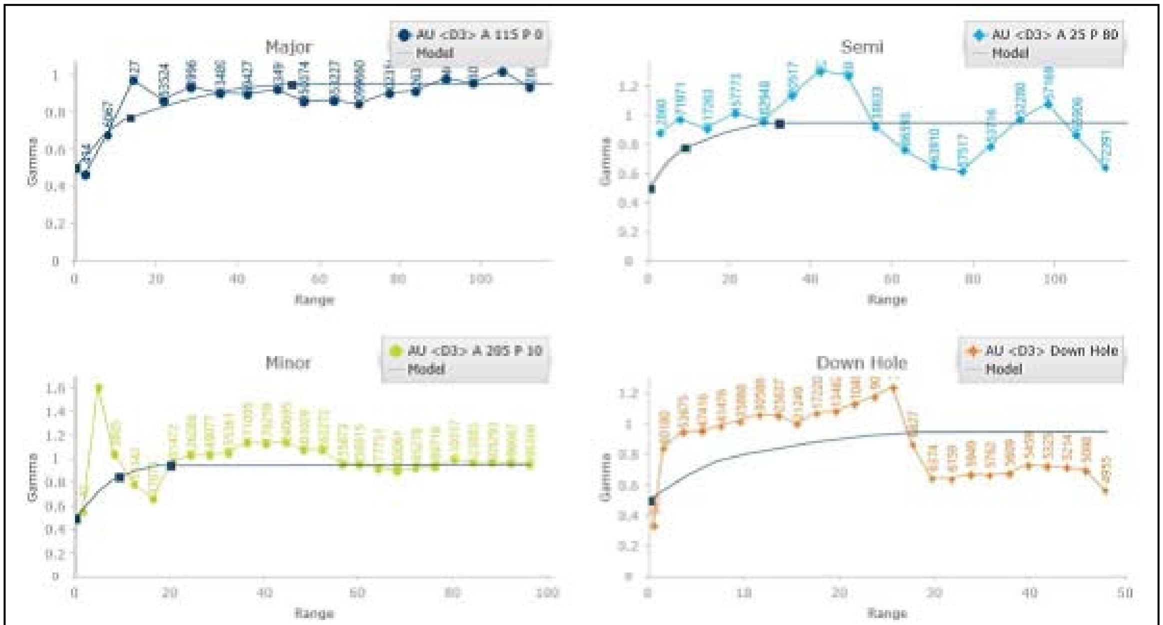
The search ellipsoids for the grade estimation were developed using variograms for each domain. Variograms were run in each domain for gold using the same structural orientations used to develop the mineralized solids. These variograms are shown in Figure 14.7.



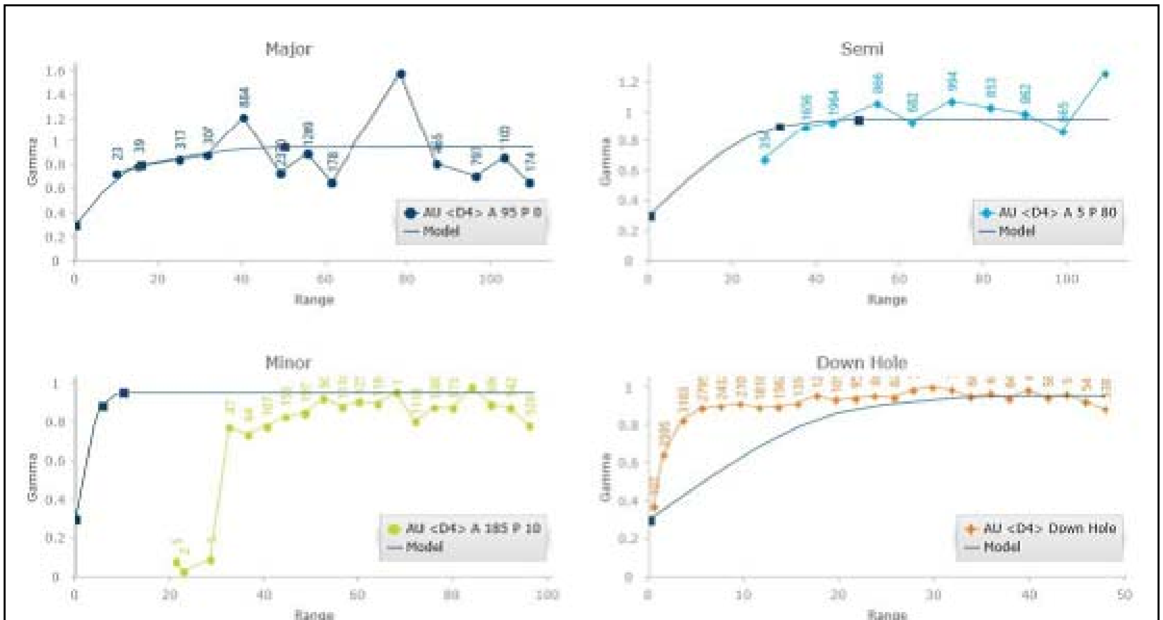
Domain 1



Domain 2



Domain 3



Domain 4

Figure 14.7
Variograms

14.2.7 Block Model

The Laiva block model used a parent block size of 9 m by 9 m by 9 m with minimum sub-blocking of 3 m by 3 m by 3 m. The small child block size was selected because it represented the SMU that would allow maximum mineralized material control selectivity during mining thereby minimizing dilution. Table 14.2 shows the block model extents.

Table 14.2
Block Model

Item	X	Y	Z
Origin	2,527,300	7,159,350	-300
Offset Minimum	-	-	-
Offset Maximum	2,205	1,800	504
Parent Block Size (m)	9.00	9.00	9.00
Child Block Size (m)	3.00	3.00	3.00
Bearing Dip/Plunge	90.00	-	-

The following table shows the block model variables:

Table 14.3
Block Model Variables

Variable	Default	Type	Description
id3_au	-99	double	Inverse Distance Cubed Gold g/t
id3_samp	-99	integer	Inverse Distance Cubed Number of Samples
id3_ddh	-99	integer	Inverse Distance Cubed Number of Drillholes
pass_id3	-99	integer	Inverse Distance Cubed Pass Number
id3_dist	-99	double	Inverse Distance Cubed Average Search Distance
nn_au	-99	double	Nearest Neighbor Gold g/t
pass_nn	-99	integer	Nearest Neighbor Pass Number
nn_dist	-99	integer	Nearest Neighbor Average Search Distance
density	-99	double	Whole Block Density
class	5	integer	Resource Classification
vtopo	100	integer	Vulcan Topo Variable (0=100% Air, 100=100% below Topo)
domain	0	integer	Mineralized Domain (default=0)
wht_rx	wast	name	Whittle Rock Code
rdensity	-99	double	Vulcan Topo Adjusted Density
au	-99	double	Vulcan Gold Grade g/t

14.2.8 Laiva Mineral Resource Estimation

A 3D block model was constructed in the Vulcan mine planning software that was constrained by the mineralized domains (described above). The block model uses a variable block size of 9 m by 9 m by 9 m (X, Y, Z) sub-blocked to a minimum of 3 m by 3 m by 3 m (X, Y, Z).

Bulk densities were assigned to each block following mineral resource estimation, with blocks having a gold grade of greater than 0.00 g/t being assigned a bulk density of 2.83 t/m^3. Other blocks were assigned a bulk density of 2.83 t/m^3. The current topographic surface was used to flag the topo 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 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 established for only that material within the mineralized domain for gold. Additionally, rock (waste) material outside of the mineralized domain was assumed to always be waste with no gold grade.

Using the Vulcan composite file (described above), interpolations were run in each mineralized domain for gold. Runs were completed using ID^3. Four passes were run to allow for use in resource classification. Only composites and blocks flagged as within the mineralized zone could be considered in the grade estimation. The block model interpolation parameters are shown in Tables 14.4 to 14.7.

Table 14.4
Laiva Domain 1 Estimation Parameters

Item	Pass 1	Pass 2	Pass 3	Pass 4
Search Parameters	0.2	0.5	1	1.5
Major Range (m)	12.0	30.0	60.0	90.0
Semi-Major Range (m)	7.0	17.5	35.0	52.5
Minor Range (m)	5.0	5.0	5.0	5.0
Azimuth (degrees)	76	76	76	76
Plunge (plunge of the azimuth in degrees)	-	-	-	-
Dip (degrees)	-80	-80	-80	-80
Search Ellipsoid				
Azimuth (degrees)	76	76	76	76
Plunge (plunge of the azimuth in degrees)	-	-	-	-
Dip (degrees)	-80	-80	-80	-80
Major (m)	12	30	60	90
Semi-Major (m)	7	18	35	53
Minor (m)	5	5	5	5
Estimation Parameters				
Minimum Number of Composites	8	4	2	2
Maximum Number of Composites	10	8	4	4
Maximum Composites Per Drill Hole	2	2	2	2

Table 14.5
Laiva Domain 2 Estimation Parameters

Item	Pass 1	Pass 2	Pass 3	Pass 4
Search Parameters	0.2	0.5	1	1.5
Major Range (m)	12.8	32.0	64.0	96.0
Semi-Major Range (m)	9.6	24.0	48.0	72.0
Minor Range (m)	5.0	5.0	5.0	5.0
Azimuth (degrees)	72	72	72	72
Plunge (plunge of the azimuth in degrees)	-	-	-	-
Dip (degrees)	-80	-80	-80	-80
Search Ellipsoid				
Azimuth (degrees)	72	72	72	72
Plunge (plunge of the azimuth in degrees)	-	-	-	-
Dip (degrees)	-80	-80	-80	-80
Major (m)	13	32	64	96
Semi-Major (m)	10	24	48	72
Minor (m)	5	5	5	5
Estimation Parameters				
Minimum Number of Composites	8	4	2	2
Maximum Number of Composites	10	8	4	4
Maximum Composites Per Drill Hole	2	2	2	2

Table 14.6
Laiva Domain 3 Estimation Parameters

Item	Pass 1	Pass 2	Pass 3	Pass 4
Search Parameters	0.2	0.5	1	1.5
Major Range (m)	10.6	26.5	53.0	79.5
Semi-Major Range (m)	6.4	16.0	32.0	48.0
Minor Range (m)	5.0	5.0	5.0	5.0
Azimuth (degrees)	115	115	115	115
Plunge (plunge of the azimuth in degrees)	-	-	-	-
Dip (degrees)	-80	-80	-80	-80
Search Ellipsoid				
Azimuth (degrees)	115	115	115	115
Plunge (plunge of the azimuth in degrees)	-	-	-	-
Dip (degrees)	-80	-80	-80	-80
Major (m)	11	27	53	80
Semi-Major (m)	6	16	32	48
Minor (m)	5	5	5	5
Estimation Parameters				
Minimum Number of Composites	8	4	2	2
Maximum Number of Composites	10	8	4	4
Maximum Composites Per Drill Hole	2	2	2	2

Table 14.7
Laiva Domain 4 Estimation Parameters

Item	Pass 1	Pass 2	Pass 3	Pass 4
Search Parameters	0.2	0.5	1	1.5
Major Range (m)	10.0	25.0	50.0	75.0
Semi-Major Range (m)	10.0	25.0	50.0	75.0
Minor Range (m)	5.0	5.0	5.0	5.0
Azimuth (degrees)	95	95	95	95
Plunge (plunge of the azimuth in degrees)	36	36	36	36
Dip (degrees)	-80	-80	-80	-80
Search Ellipsoid				
Azimuth (degrees)	95	95	95	95
Plunge (plunge of the azimuth in degrees)	36	36	36	36
Dip (degrees)	-80	-80	-80	-80
Major (m)	10	25	50	75
Semi-Major (m)	10	25	50	75
Minor (m)	5	5	5	5
Estimation Parameters				
Minimum Number of Composites	8	4	2	2
Maximum Number of Composites	10	8	4	4
Maximum Composites Per Drill Hole	2	2	2	2

14.2.9 Laiva Resource Classification

The mineral resource classification used on the Laiva Gold deposit is based on which pass generated a grade estimate. The resource classification used was:

- Measured – Blocks estimated in pass 1 are classified as measured.
- Indicated – Blocks estimated in pass 2 are classified as indicated.
- Inferred – Blocks estimated in pass 3 are classified as inferred.

Blocks flagged during pass 4 are not considered in the mineral resource estimate and were populated to provide future exploration guidance to FTR. Within the open-pit constrained mineral resource estimate, material flagged as 4 is considered waste material.

14.2.10 Checks on the Laiva Mineral Resource Estimate

Following grade estimation, the gold grade populated block model was checked to ensure the resource estimation correctly populated the block model. These checks included an overall review of the estimated gold values, QQ plots of the block model versus the composites, a section by section comparison between the ID^3 gold values and the underlying drill holes, and a statistical comparison of the raw assay values versus the composite values versus the block values.

The overall block gold grades were examined to ensure that the estimation had honored all the estimation parameters as well as staying within the mineralized zone. A visual check on a sectional basis showed this to be true. Each of the drill hole cross-sections were also reviewed and the underlying drill holes were checked to determine that the original gold grade closely matched the estimated block gold grade. Every cross-section on a 10-m spacing was examined and every assay interval agreed closely with the overlying estimated block model gold grade. A statistical comparison of the raw assay values versus the uncapped composite values versus the estimated block values was run and is shown in Table 14.8.

Table 14.8
Laiva Mineral Resource Estimation Model Statistics

Data	Raw Assays	Composites	Block Model Limited
Source:	Drill Hole Database	1 Meter Run Length	-
Capping:	No	No	No
Capping Grade (g/t):	-	-	Various
Limited HG Search:	No	No	Yes
Number of samples:	80,960	91,567	625,996
Minimum:	0.01	0.00	0.00
Maximum:	240.00	230.14	184.68
Range:	239.99	230.14	184.68
Average:	0.64	0.59	0.74
Standard deviation:	1.93	1.73	1.31
Variance:	3.71	2.98	1.71

The QQ plot of the block model estimated gold grades versus the composites are shown in the Figure 14.8.

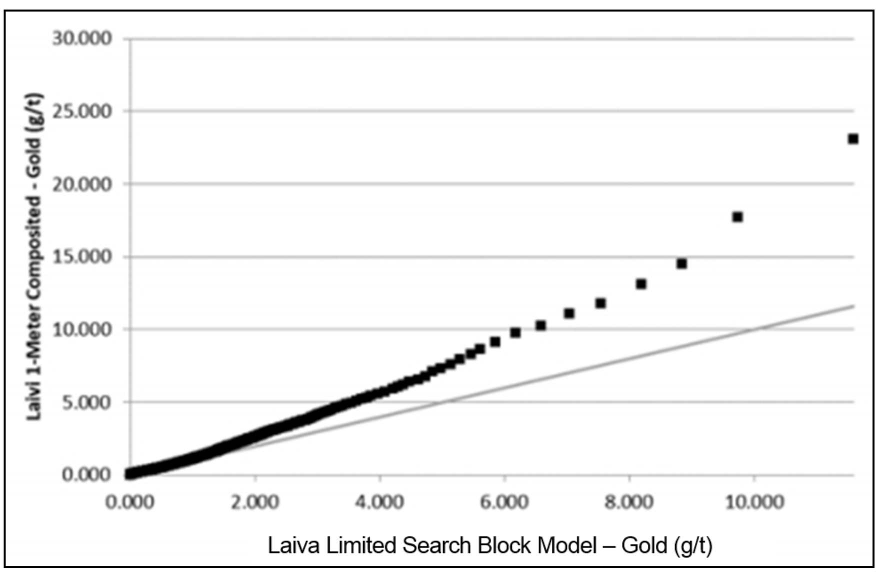


Figure 14.8
QQ Plot Inverse Distance³ Limited High-Grade Search

The block model checks indicate that the mineral resource estimate closely matches the underlying composites at lower gold grade values. At higher gold grades, the block model gold grades are under-estimated relative to the underlying composites.

14.3 Laiva Economic Pit-Shell

The mineral resources are envisioned to be extracted using an open-pit mining method. An open pit requires some sort of calculated economic pit shell in order for a mineral resource to be reported. For this mineral resource, the economic pit shell was calculated using the Whittle economic pit optimizer.

14.3.1 Economic Assumptions

The operating assumptions (economic and gold recovery) used for the Whittle economic pit optimization are shown in Table 14.9

Table 14.9
Laiva Economic Assumptions

Item	Value	Units
Overall Mineralized Material Mining Cost	3.19	US$/Mineralized Material Tonne
Overall Waste Mining Cost	2.59	US$/Waste Tonne
Processing Cost	11.98	US$/Mineralized Material Tonne
G&A Cost	2.00	US$/Mineralized Material Tonne
Selling Cost	0.75	US$/Troy Ounce
Overall Recovery	90.26	Percent
NSR Royalty	0.15	Percent
Gold Price	1,250.00	US$/Troy Ounce
Calculated Cutoff Grade	0.40	grams per tonne

For open-pit mineral resources, a cut-off grade of 0.40 g/t gold was used. There are no underground mineral resources reported in this estimate. Overall pit slopes were assumed to be 53 degrees.

Using these assumptions, a Whittle economic pit optimization was completed and an economic pit shell generated. This Whittle Pit Shell is illustrated in Figure 14.9.

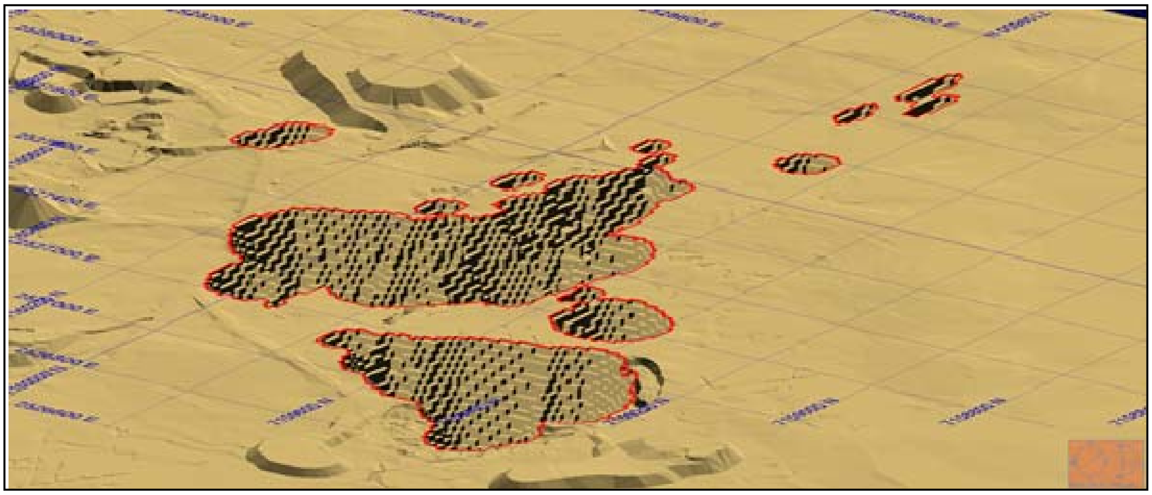


Figure 14.9
Whittle Pit Shell

14.3.2 Pit Constrained Mineral Resource Estimate

The estimated mineral resources for the Laiva gold deposit is shown in Table 14.10.

Table 14.10
Laiva Open-Pit Constrained Mineral Resource Estimate

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

1. The mineral resources presented above are reported within an economic pit shell generated by the Whittle Pit Optimization software.
2. 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.
3. The mineral resources presented here were estimated using a block model with a block size of 9 m by 9 m by 9 m sub-blocked to a minimum of 3 m by 3 m by 3 m using ID^3 methods for grade estimation. All mineral resources are reported using an open-pit gold cut-off of 0.40 g/t Au.
4. The mineral resources presented here were estimated using the CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council 27 November 2010.

5. The effective date for this mineral resource estimate is 9 August 2017.
6. The mineral resources presented herein are in-situ and other than an economic pit shell no attempt has been made to apply a mining dilution or a mining recovery factor to them.
7. This mineral resource estimate was completed by Sam J. Shoemaker, Jr. Registered Member SME and a Qualified Person under NI43-101.

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15.0 MINERAL RESERVE ESTIMATES

Not applicable to this Technical Report.

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16.0 MINING METHODS

The Laiva Project is composed of a single project with three mining areas: the previously developed South Pit area in the southwestern portion of the project; the previously developed North Pit area in the northern portion of the project; and the East Pit area located east of the North and South pit areas. These three mining areas develop the same mineral resource and are shown below in Figure 16.1.

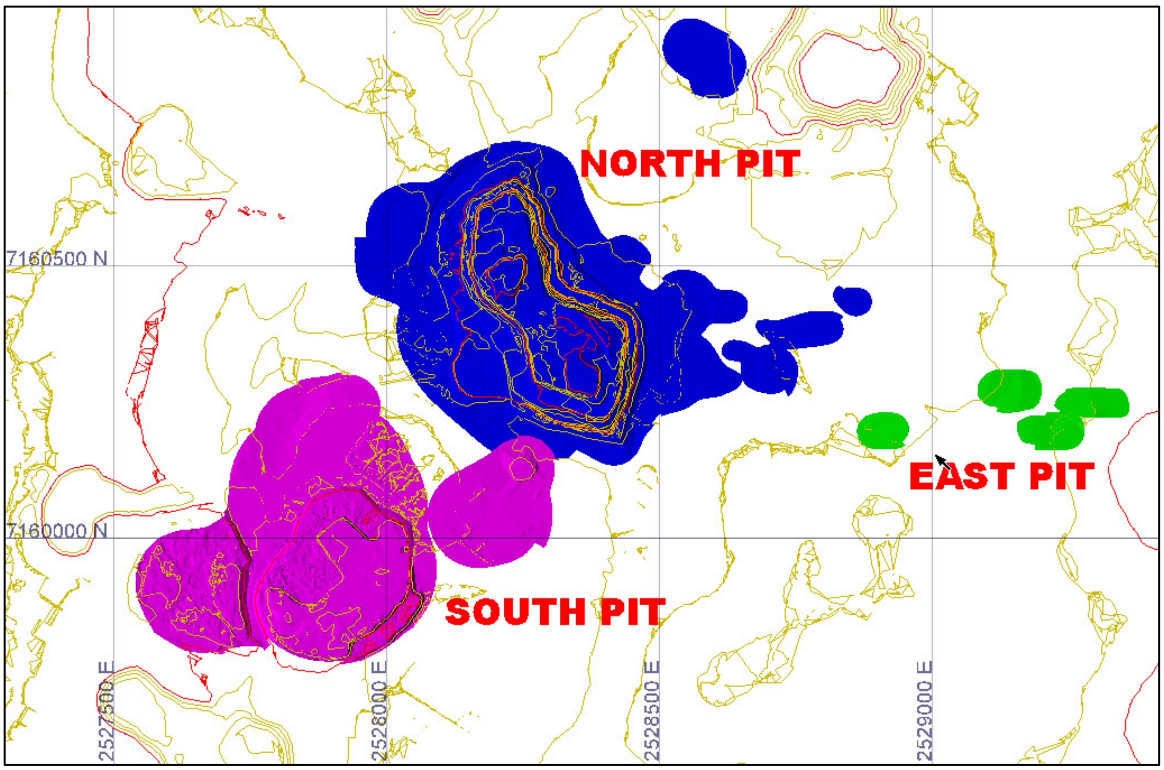

Figure 16.1
Laiva Project Mining Areas

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 three mining areas for the PEA.

Standard mining technology has been utilized to create an open pit at each of the three mining areas. The South pit has approximate dimensions of 520 m south to north by 550 m east to west and a maximum depth of 190 m below current ground level. The South is composed of six mining phases with a single standalone phase adjoining the main South pit to the northeast. The North pit has approximate dimensions of 580 m south to north by 620 m east to west and a maximum depth of 170 m below current ground level. The North pit is composed of seven mining phases with several phases being small standalone pits slightly to the north and east of the main North pit. The East pit has approximate dimensions of 110 m south to north by 190 m east to west and a maximum depth of 27 m below current ground level. The East pit is composed of four mining phases with two small pits located to the west of the main East pit. All of the pit shells were selected from the Whittle open-pit optimization discussed below and were used to guide the design of three mining areas.

A mine contractor will be used for most mining activities including site preparation, haul road construction and maintenance, bulk waste drilling and blasting, excavation and haulage of waste, management of waste dumps, oversize breakage, and pit dewatering. Nordic Mines will perform drilling and blasting as well as mineralized material loading into haul trucks supplied by the mine contractor. The mine contractor will provide the open-pit equipment with the exception of mineralized material zone blast hole drilling and mineralized material loading 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 contractors fleet as well as contract maintenance of Nordic equipment. Nordic will provide pit technical services including blast design, blasthole layout, mineralized material grade control, mine planning, and blasthole sampling. Specialized contractors will provide explosives storage on-site. Explosives, blasting agents, fuel and other consumables will be provided by established suppliers.

16.1 Mine Design

The ultimate pit limits selected for the three mining areas for the Laiva project were selected based on Whittle open-pit economic optimizations. The 3 mining areas will be developed using 17 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. A mine production schedule was prepared by BOYD using Maptek's Chronos scheduling software.

16.1.1 Mining Dilution

The Laiva resource model uses a fixed 3 m by 3 m by 3 m block size 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 PEA mine planning activities, as mineralized material zones are gradational defined by cut-off grade boundaries.

16.1.2 Whittle Pit Optimization

In order to design the various required phased designs for the Laiva project, BOYD completed a 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 and a final conceptual pit shell was determined. This pit shells were then used for PEA-level detailed phased pit designs and production scheduling.

Mine planning geotechnical parameters were determined by BOYD and are shown below in Table 16.1.

Table 16.1
Laiva Geotechnical Parameters

Item	Value
Inter-ramp Slope (degrees)	53.0
Face Angle (degrees)	80.0
Safety Bench (m)	5.2
Benching	Single
Bench Height (m)	9.0
Sub-bench Height (m)	3.0

Initial PEA level operating costs were calculated and used to provide the economic basis for the various Whittle pit optimization runs that were completed. All runs included measured, indicated, and inferred material. The PEA economics used for the Whittle optimization is shown below in Table 16.2.

Table 16.2
Laiva PEA Whittle Economics

Item	Value	Units
Overall Mineralized Material Mining Cost	4.05	US$/Mineralized Material Tonne
Overall Waste Mining Cost	2.65	US$/Waste Tonne
Processing Cost	11.27	US$/Mineralized Material Tonne
G&A Cost	2.41	US$/Mineralized Material Tonne
Selling Cost	0.75	US$/Troy Ounce
Overall Recovery	90.26	Percent
NSR Royalty	0.15	Percent
Annual Capacity	2,000,000	Tonnes
Gold Price	1,300.00	US$/Troy Ounce
Calculated Cutoff Grade	0.471	grams per tonne

Whittle optimizations were completed on the Laiva mineral resource model using the economic parameters in Table 16.2 as well as the geotechnical parameters presented in Table 16.1. 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 45 degrees.

The Whittle runs using the ramp reduced slopes as well as the economics presented in Table 16.2 were used to generate the final pit shells that were then used for the phased and ultimate pit designs on each deposit. Figures 16.2 show the resulting Whittle pit shells used for design.

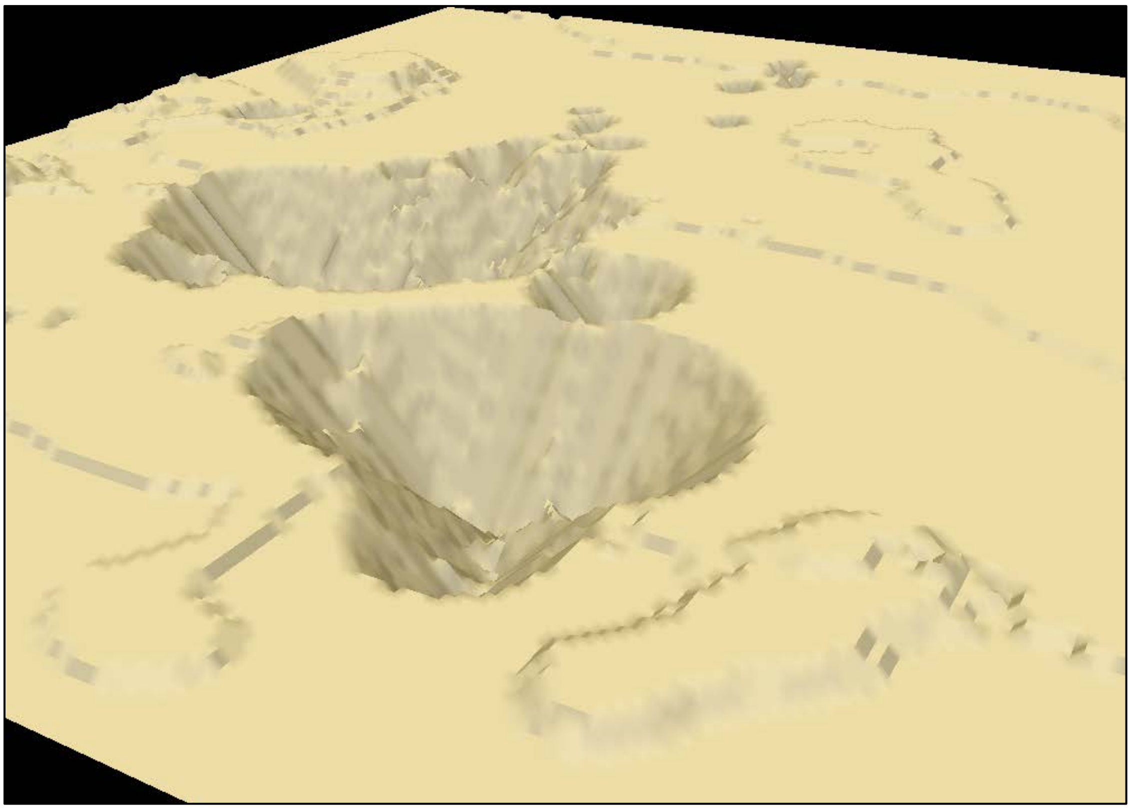

Figure 16.2
Laiva PEA Whittle Pit Shell

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$390/troy ounce) up to a revenue factor of 1.5 (gold price of US$1,950/troy ounce). The revenue factor 1 pit shell (gold price of US$1,300/troy ounce) was selected for the detailed PEA phased designs. The Whittle pit by pit graph of the results is shown below in Figure 16.3.

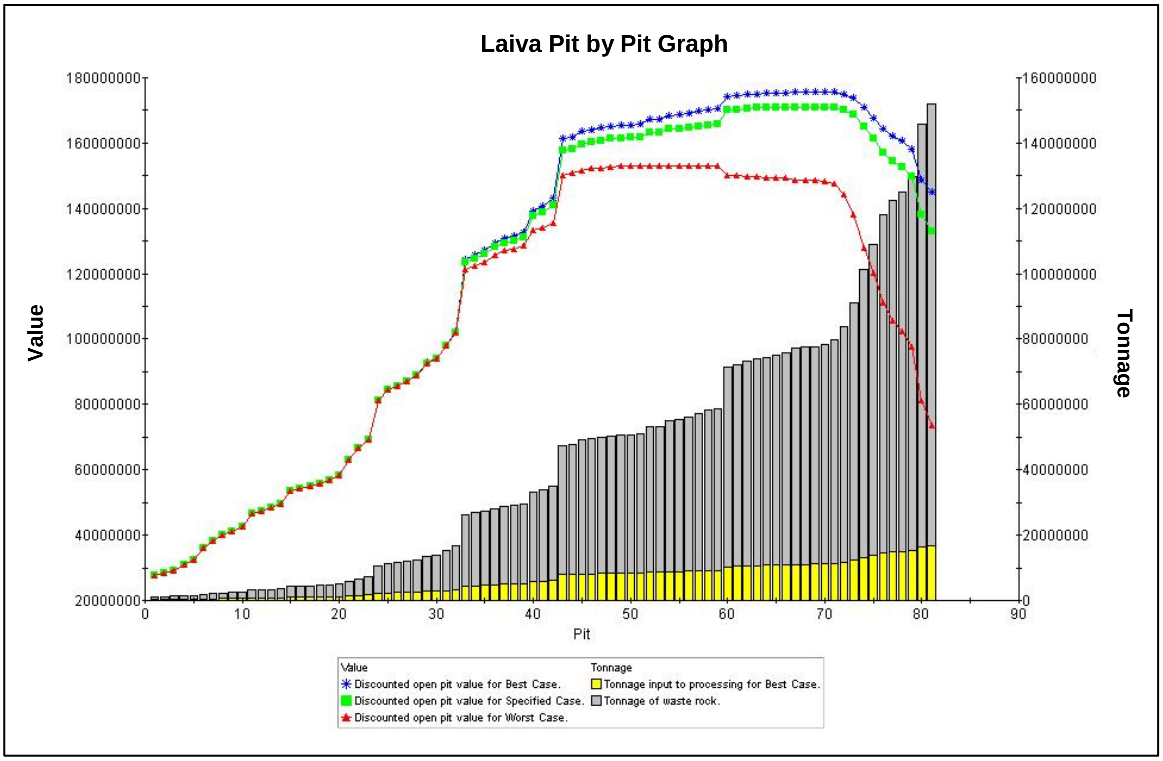

Figure 16.3
Laiva Whittle Pit by Pit Results

16.1.3 Underground Mine Design

No underground operations were considered for the Laiva PEA.

16.1.4 Design Criteria

Different mining phases were designed in accordance with the recommended bench configurations provided as tabulated above in Table 16.1. Single benching of 9 m production benches was determined to be feasible in all geologic units. Catch or safety benches with a width of 5.2 m are used in all designed phases. These safety benches are applied on every bench (9 m vertically). Mineralized material mining is planned on 3 m sub-benches while waste mining is planned on 9 m full height benches.

Two-way haul roads, 20-m wide at a 12% grade, were designed in most cases where higher traffic may require extra width for safe and efficient passing of trucks. To maximize mineralized material recovery at depth, the final benches of each pit floor were designed with single-lane access (9-m width). Safety berms were designed in accordance with the recommendations provided by BOYD.

16.1.5 Phased Designs

The Whittle pit shells described above were used to construct individual pit phases at each of the three mining areas. The phases were designed to provide high-value early phases as well as balancing waste stripping over the entire life of each pit. All of the phased designs used the same geotechnical and design criteria described above.

16.1.5.1 South Pit Phased Designs

The Laiva South pit consists of six phases. Phase 1 is the starter pit while phases 2, 3, and 5 expands these starter pits. Phase 6 is the final pit expansion for the South pit. Phase 4 is a standalone remote pit located just to the northeast of the main South pit. Table 16.3 describes the in-pit mineral resources for South pit while Figures 16.4 through 16.10 show the individual phase layouts.

Table 16.3
Laiva South Pit Mineral Resources

Class	Au (g/t)	Tonnes	Au (t.ozs)
Measured	1.213	72,000	2,800
Indicated	1.523	870,000	42,600
M+I	***1.499***	***942,000***	***45,400***
Inferred	1.680	3,322,000	179,500
Waste	-	37,246,000	-
Total Tonnes	-	41,510,000	-
Strip Ratio	-	8.73	-

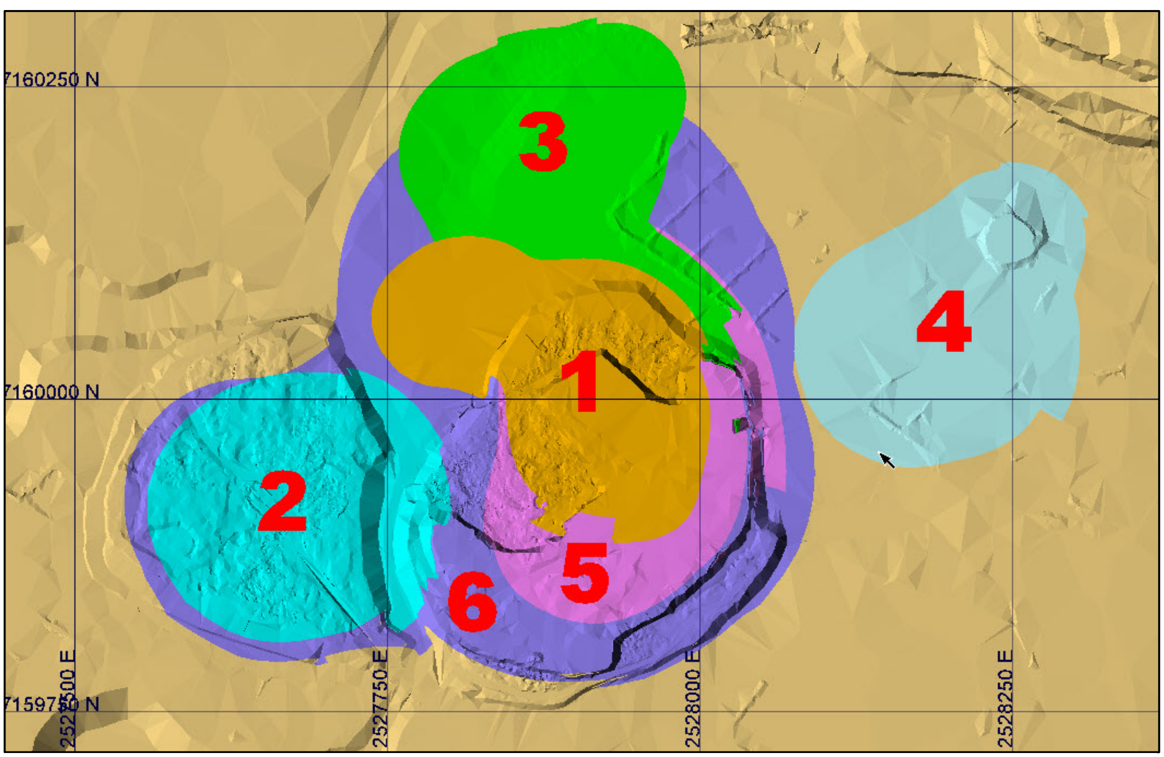

Figure 16.4
Overall Layout of the Laiva South Pit Phases

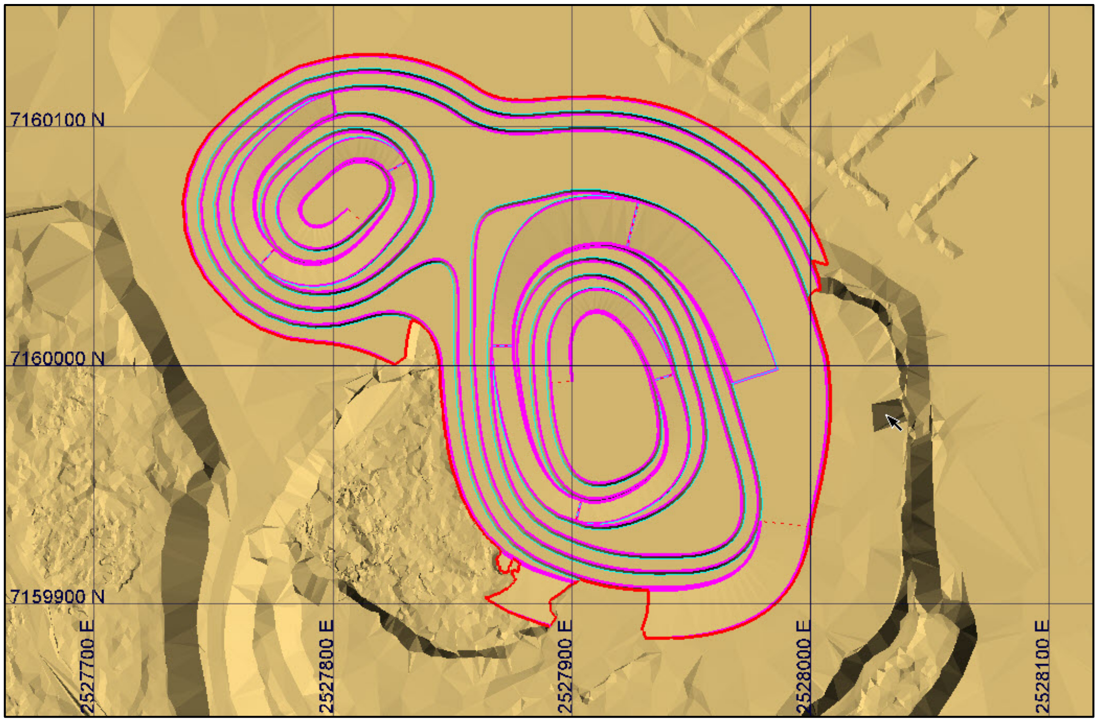

Figure 16.5
Laiva South Pit Phase 1 Design

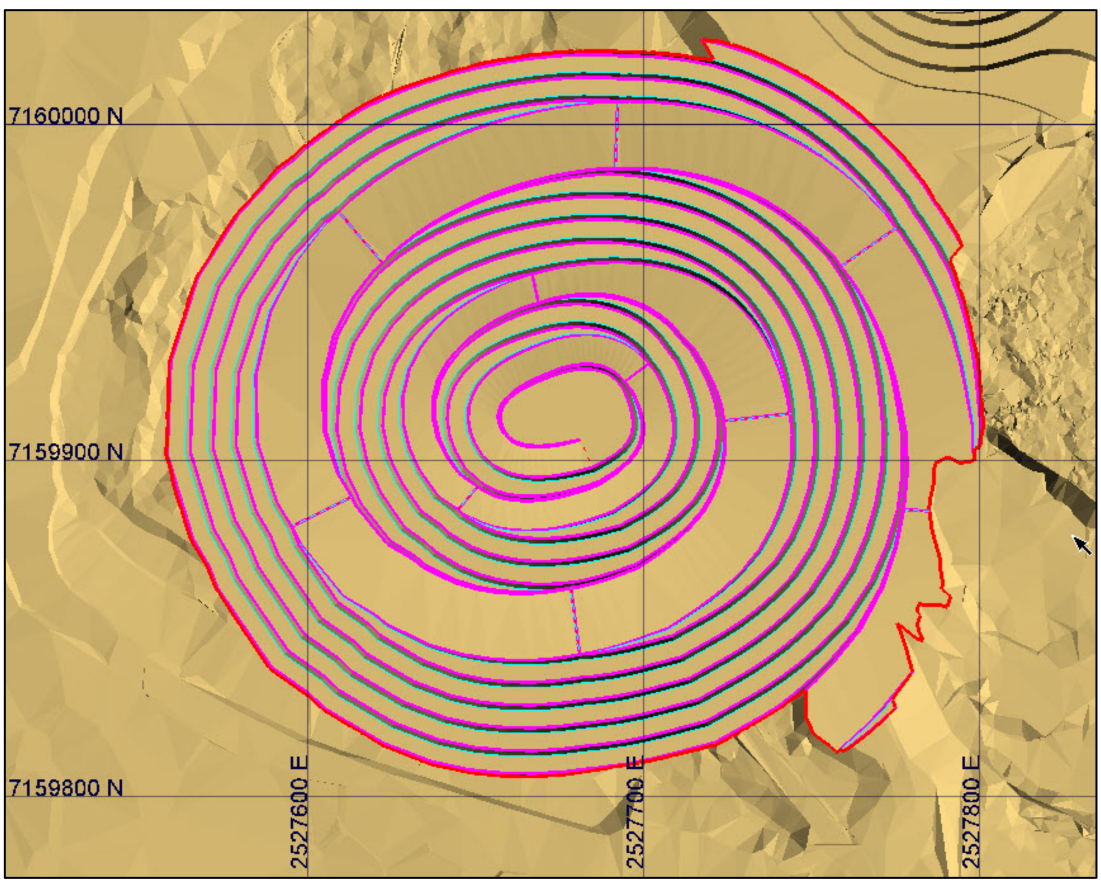

Figure 16.6
Laiva South Pit Phase 2 Design

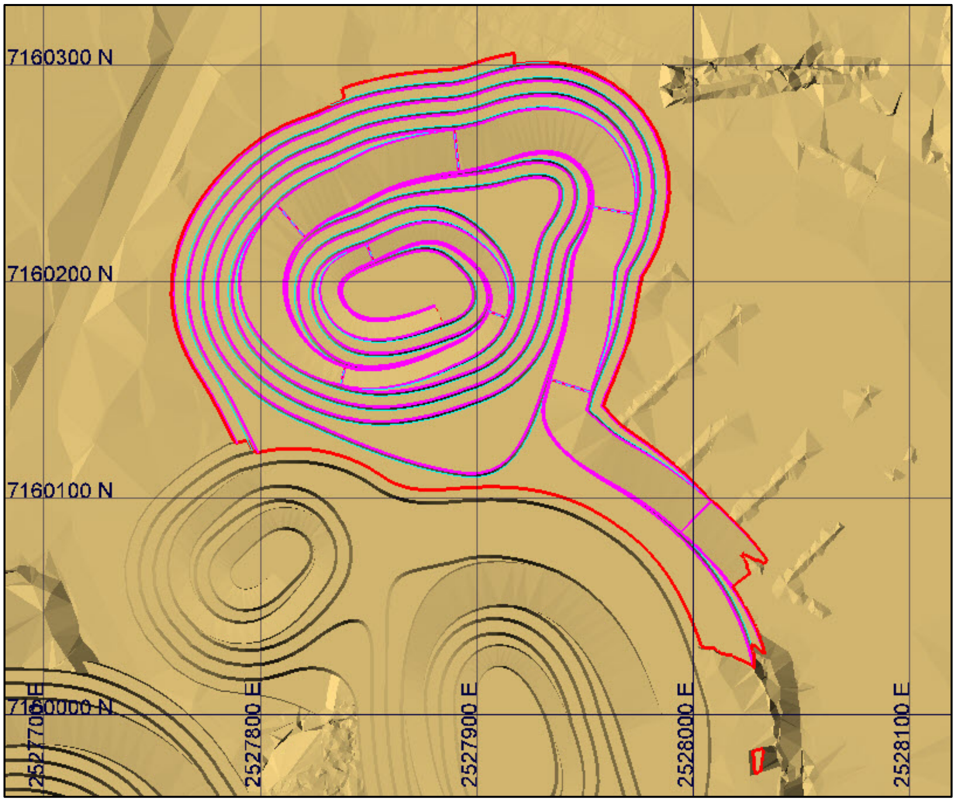

Figure 16.7
Laiva South Pit Phase 3 Design

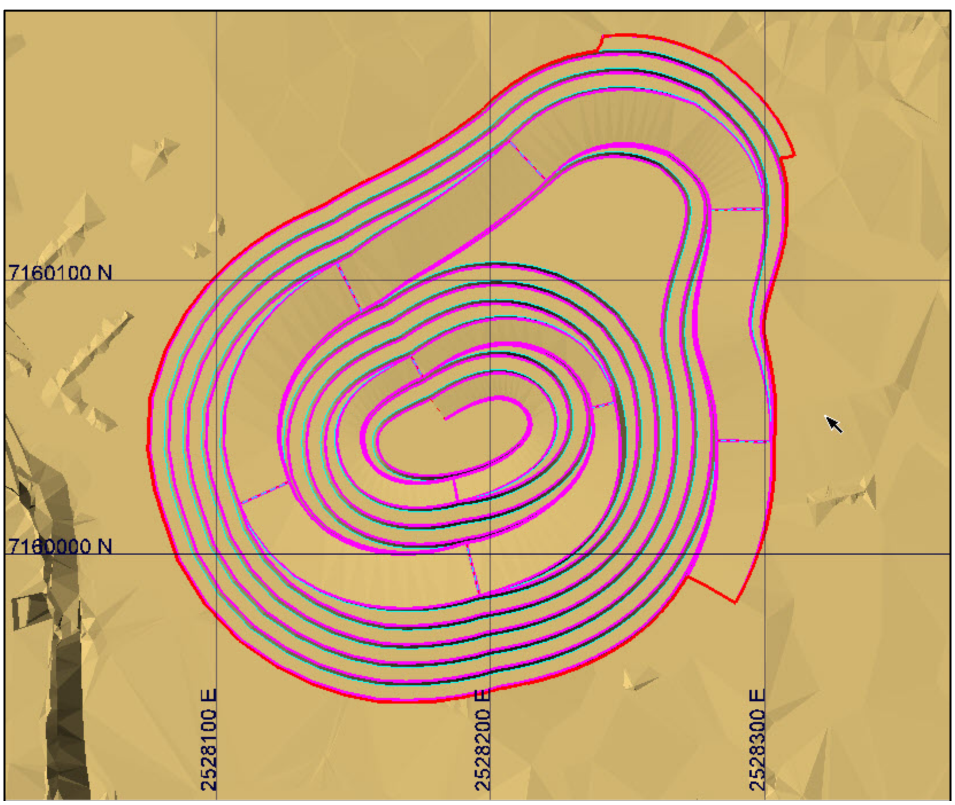

Figure 16.8
Laiva South Pit Phase 4 Design

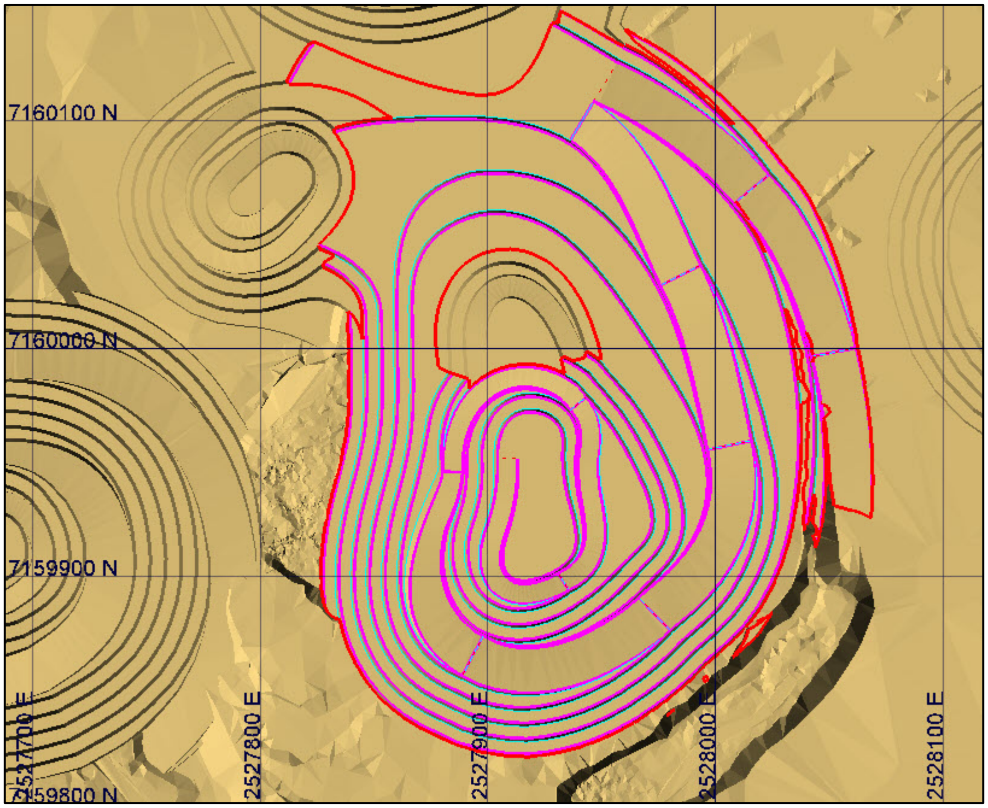

Figure 16.9
Laiva South Pit Phase 5 Design

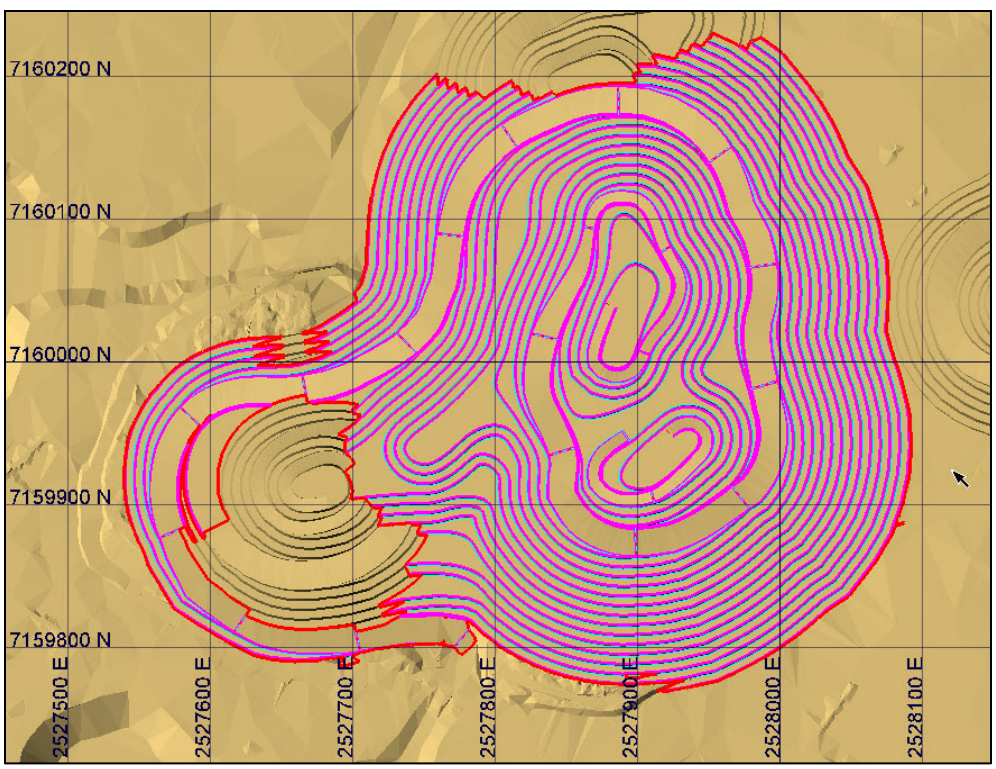

Figure 16.10
Laiva South Pit Phase 6 Design

16.1.5.2 North Pit Phased Designs

The North pit consists of seven phases. Phase 1 is the starter pit with phases 2 and 3 being expansions of that phase. Phases 4, 5, 6, and 7 are small standalone pits to the north and east of the main North pit. Table 16.4 describes the in-pit mineral resources for the North pit while Figures 16.11 through 16.18 show the individual phase layouts.

Table 16.4
Laiva North Pit Mineral Resources

Class	Au (g/t)	Tonnes	Au (t.ozs)
Measured	1.119	280,000	10,100
Indicated	1.147	2,415,000	89,100
M+I	***1.145***	***2,695,000***	***99,200***
Inferred	1.457	3,768,000	176,400
Waste	---	34,825,000	---
Total Tonnes	---	41,288,000	---
Strip Ratio	---	5.39	---

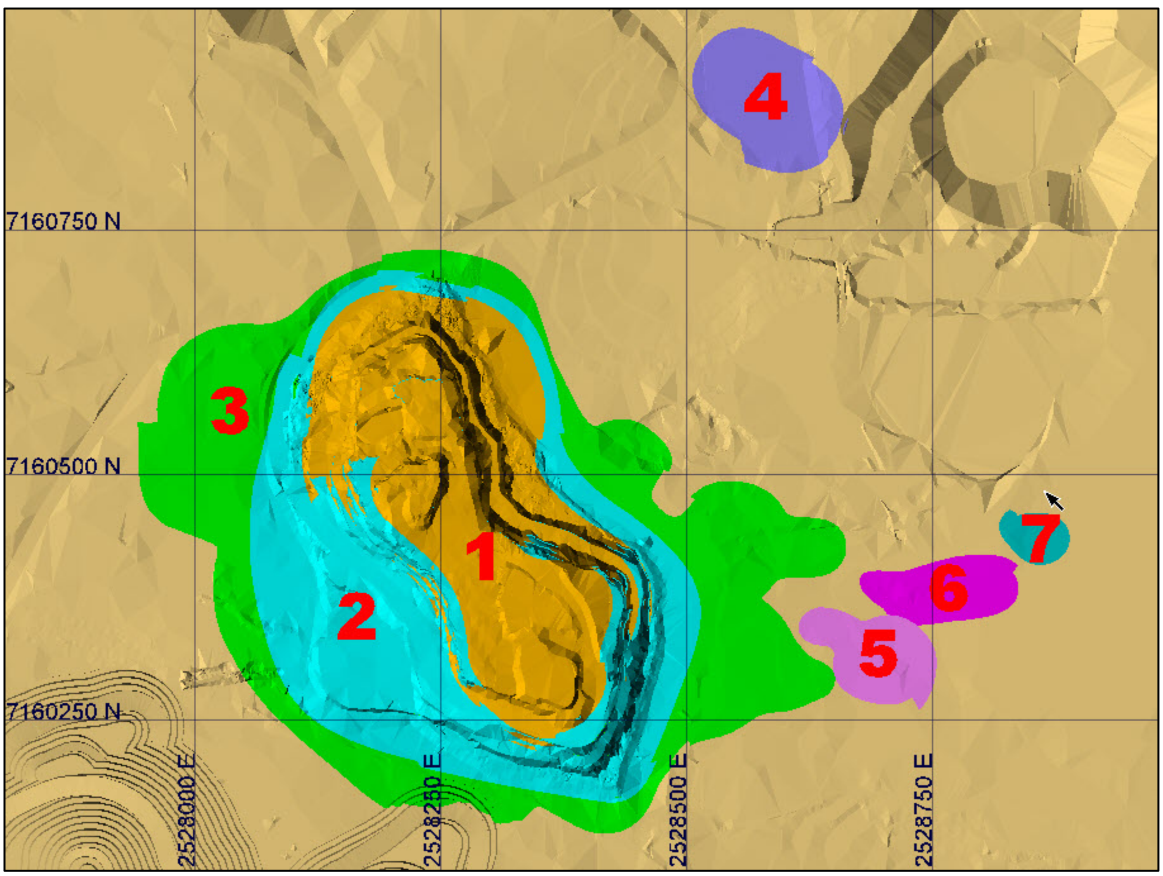

Figure 16.11
Overall Layout of the Laiva North Pit Phases

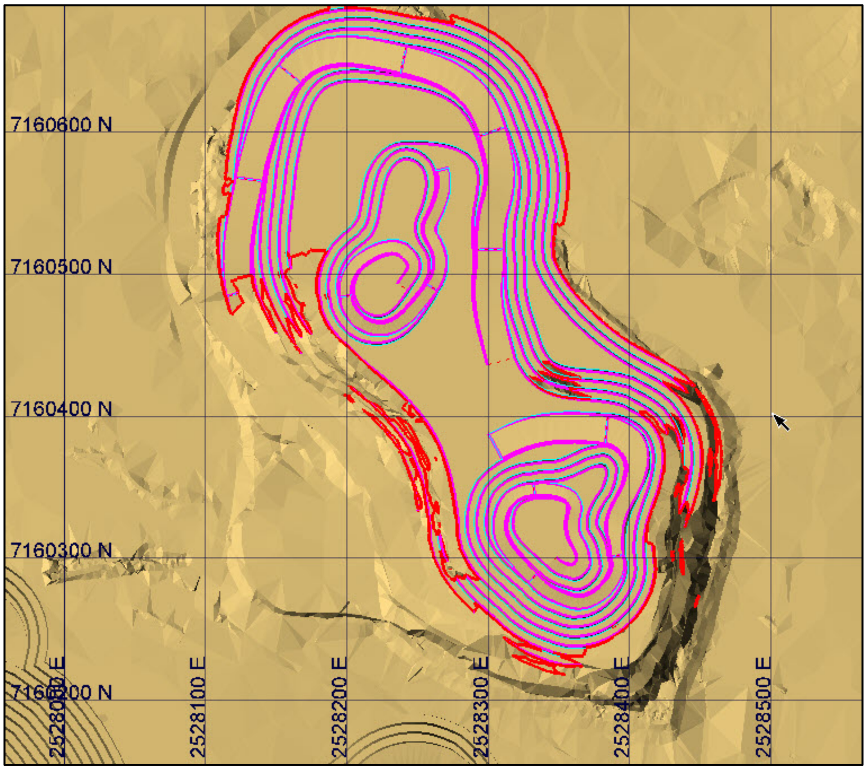

Figure 16.12
Laiva North Pit Phase 1 Design

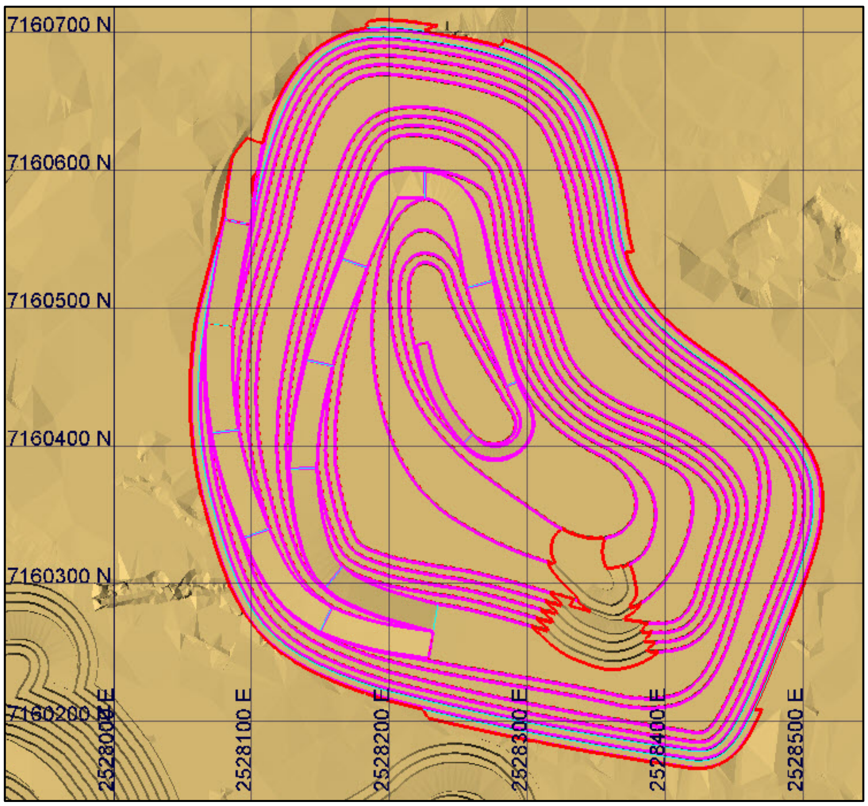

Figure 16.13
Laiva North Pit Phase 2 Design

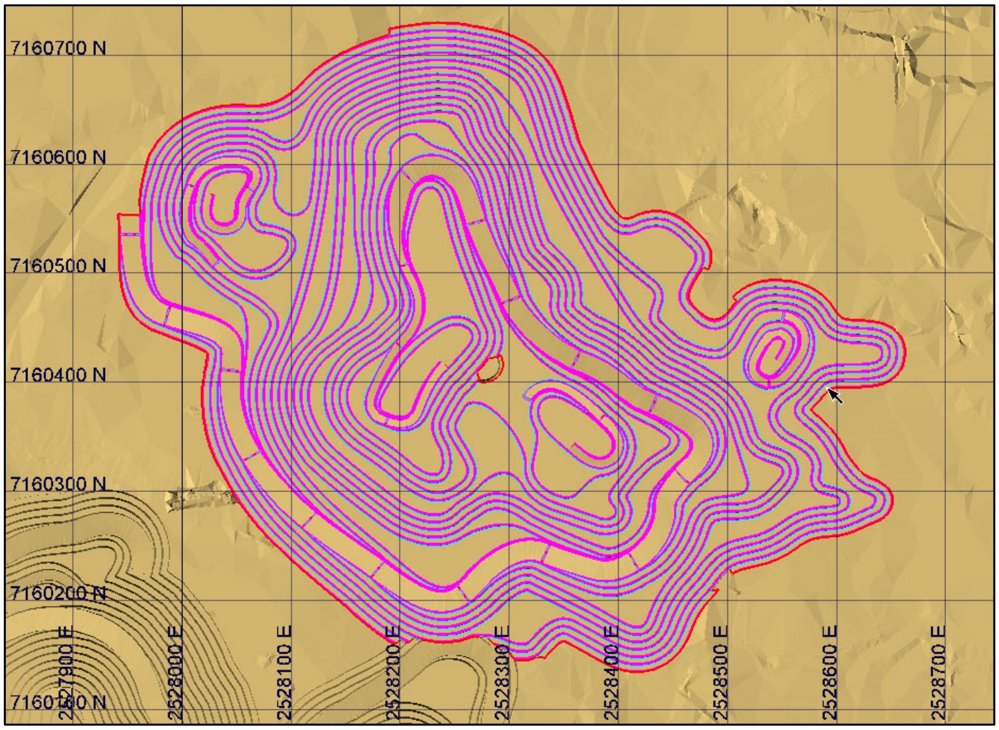

Figure 16.14
Laiva North Pit Phase 3 Design

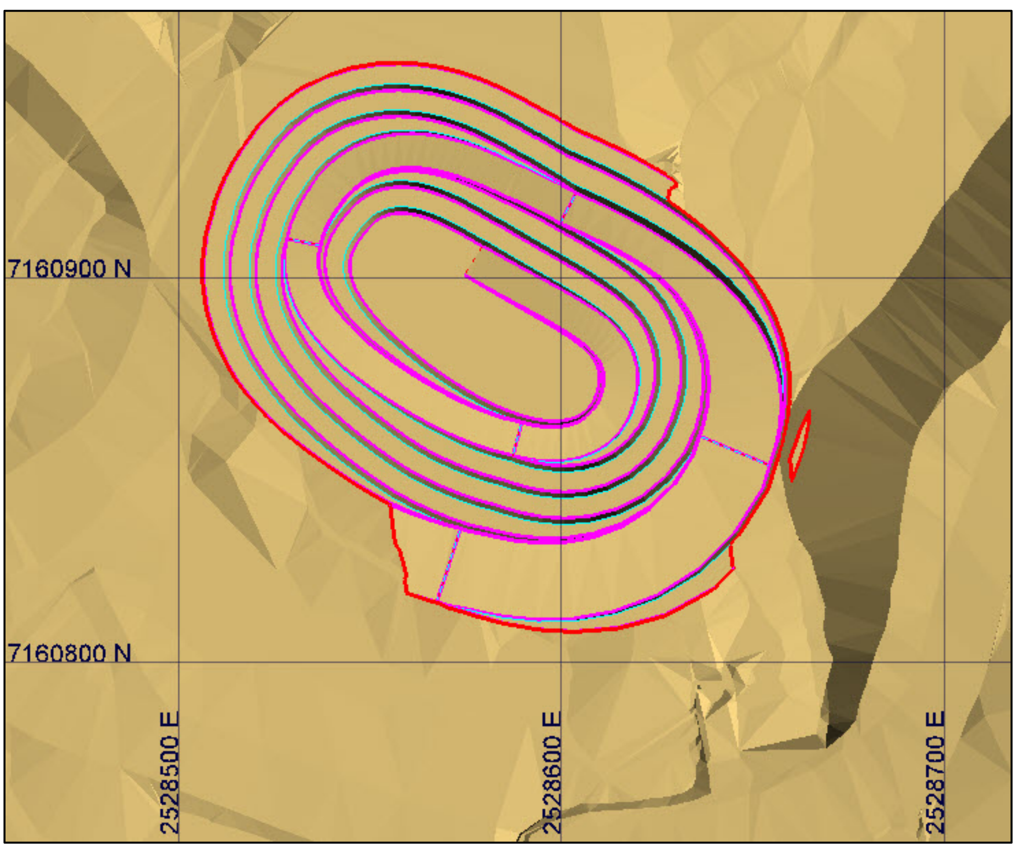

Figure 16.15
Laiva North Pit Phase 4 Design

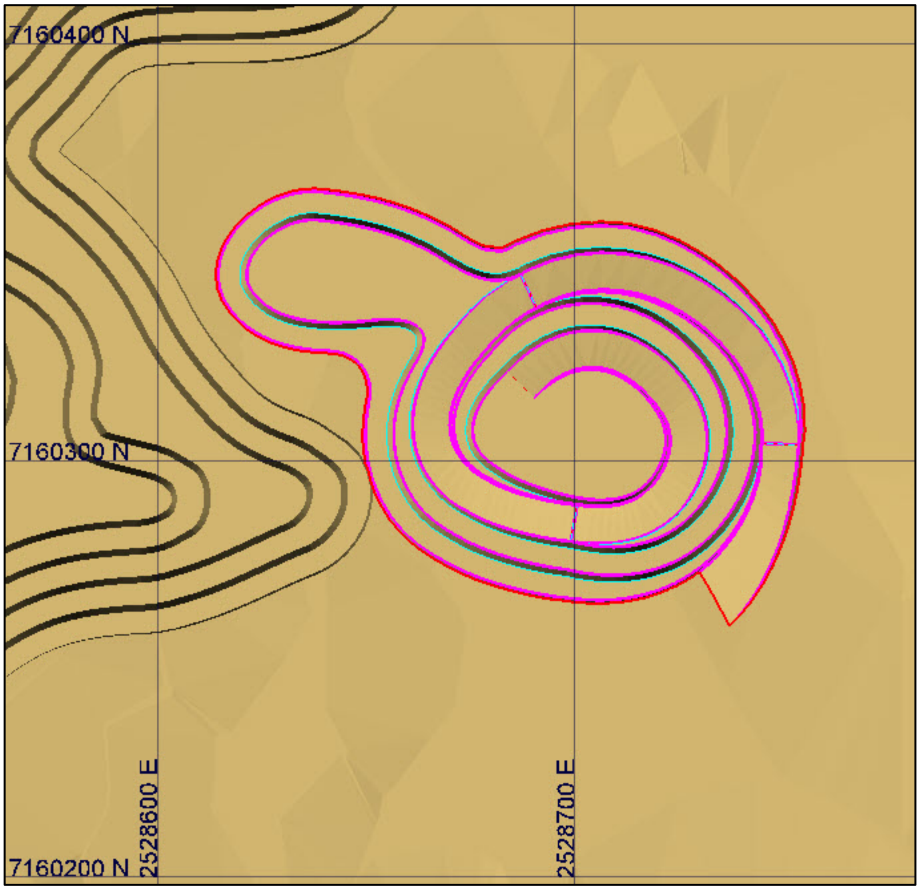

Figure 16.16
Laiva North Pit Phase 5 Design

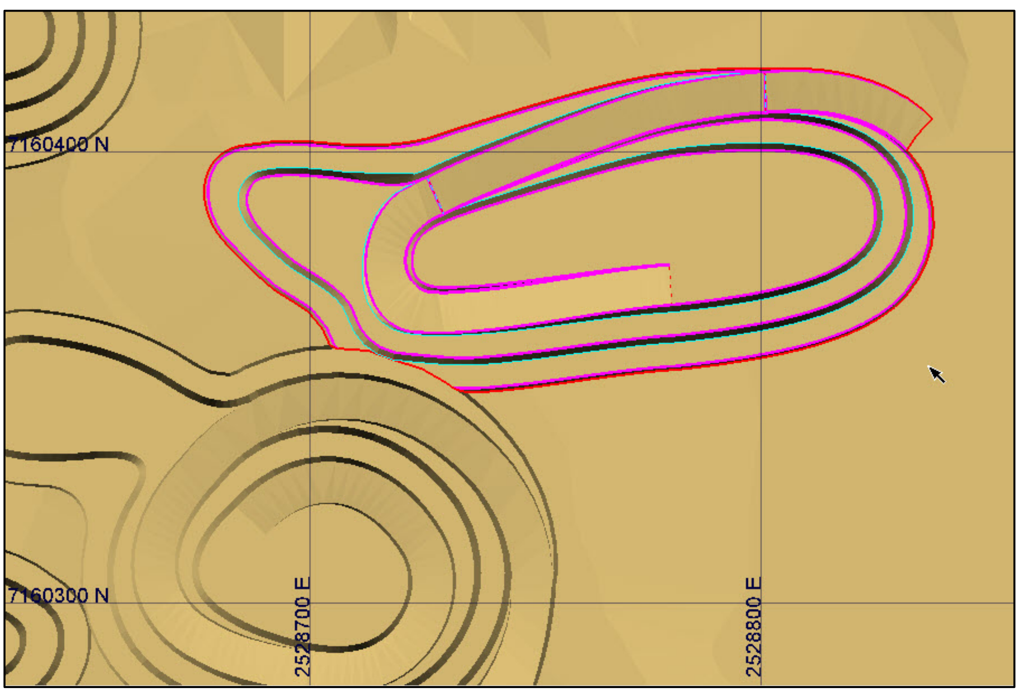

Figure 16.17
Laiva North Pit Phase 6 Design

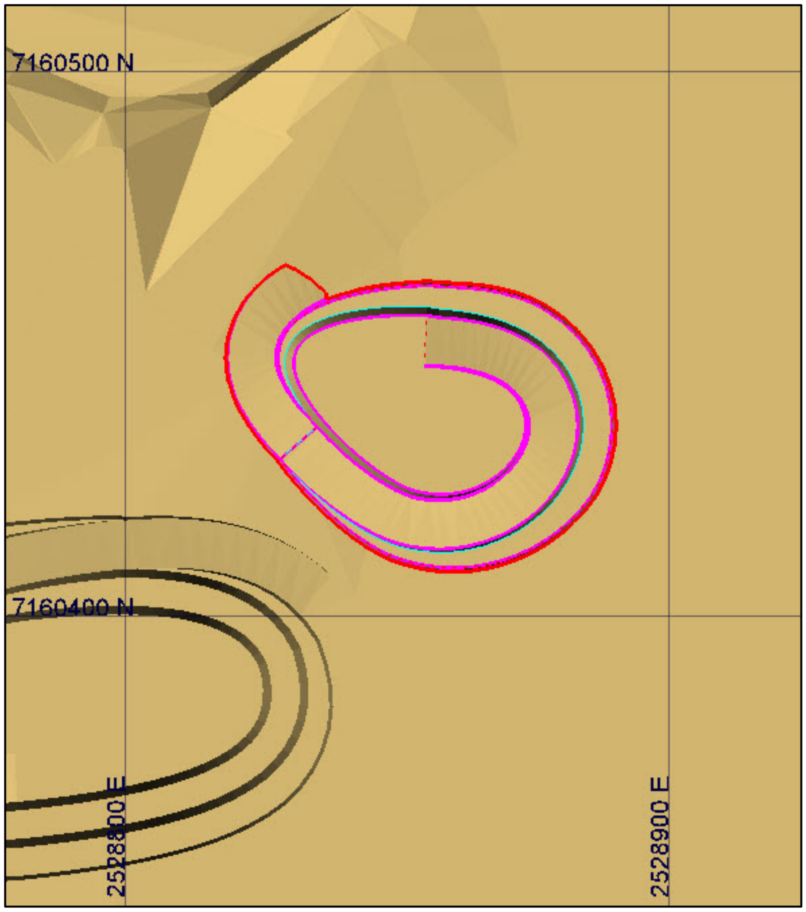

Figure 16.18
Laiva North Pit Phase 7 Design

16.1.5.3 East Pit Phased Designs

The East pit consists of four phases. Phases 1 and 2 are starter pits while Phase 3 expands these starter pits. Phase 4 is the final pit expansion for the East pit. Table 16.5 describes the in-pit mineral resources for each of the East pit phases while Figures 16.19 through 16.23 show the individual phase layouts.

Table 16.5
Laiva East Pit Mineral Resources

Class	Au (g/t)	Tonnes	Au (t.ozs)
Measured	0.000	0	0
Indicated	0.000	0	0
M+I	***0.000***	***0***	***0***
Inferred	1.399	89,000	4,000
Waste	---	946,000	---
Total Tonnes	---	1,035,000	---
Strip Ratio	---	10.63	---

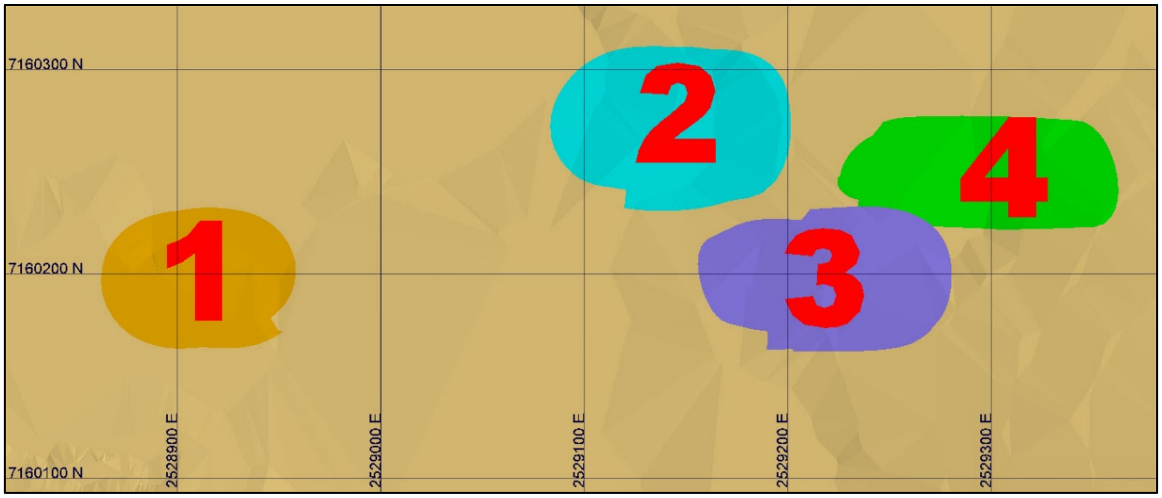

Figure 16.19
Overall Layout of the Laiva East Pit Phases

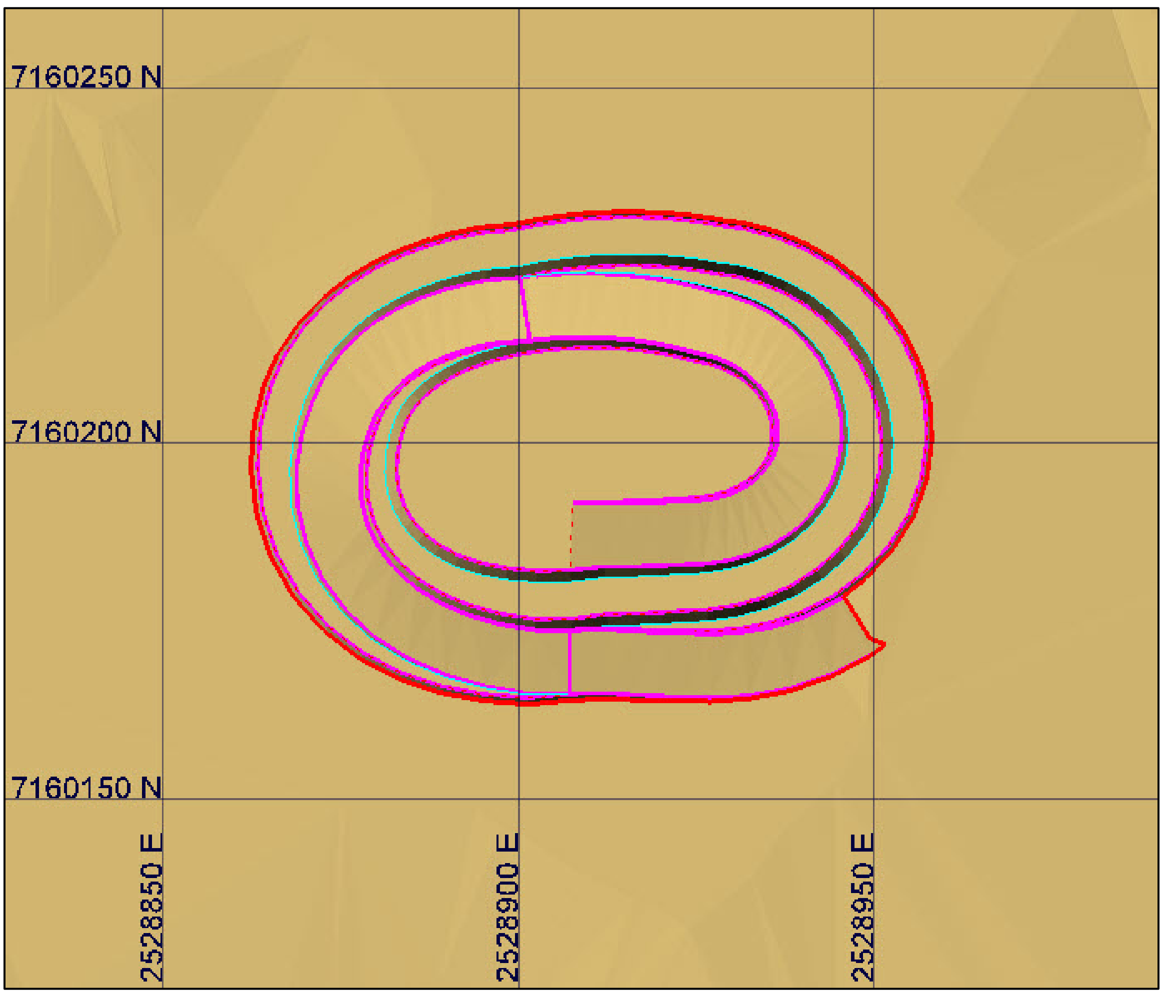

Figure 16.20
Laiva East Pit Phase 1 Design

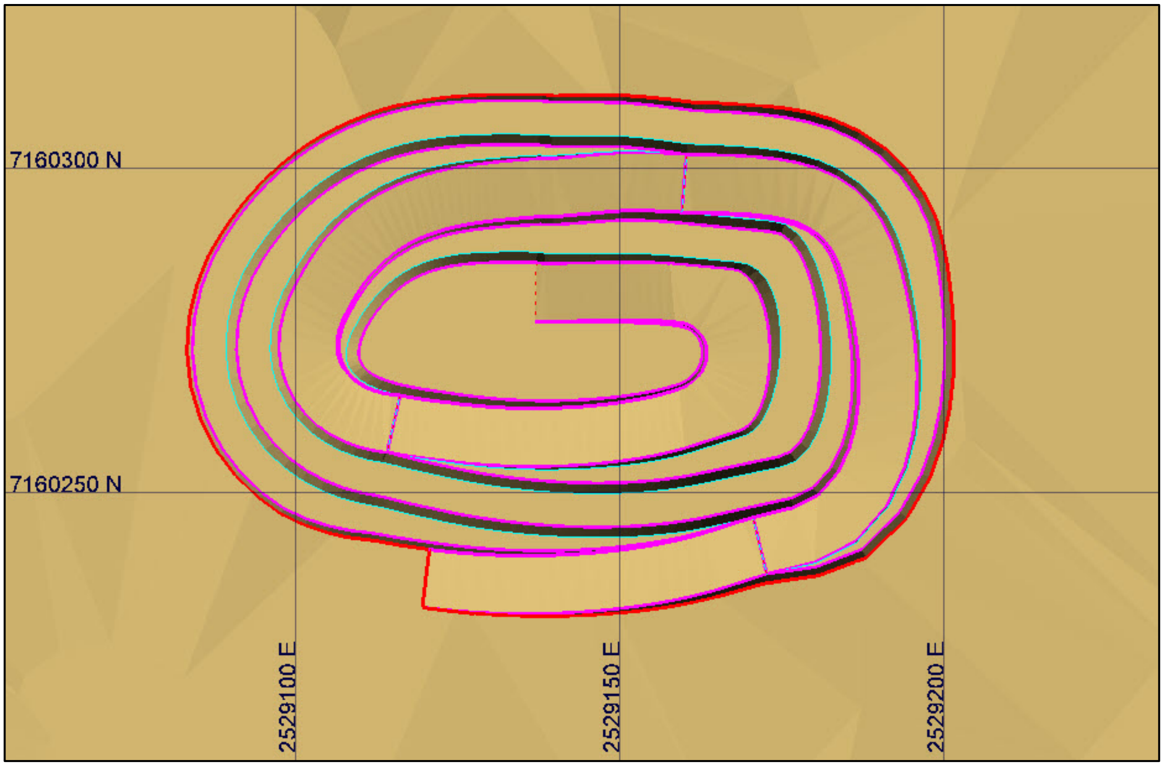

Figure 16.21
Laiva East Pit Phase 2 Design

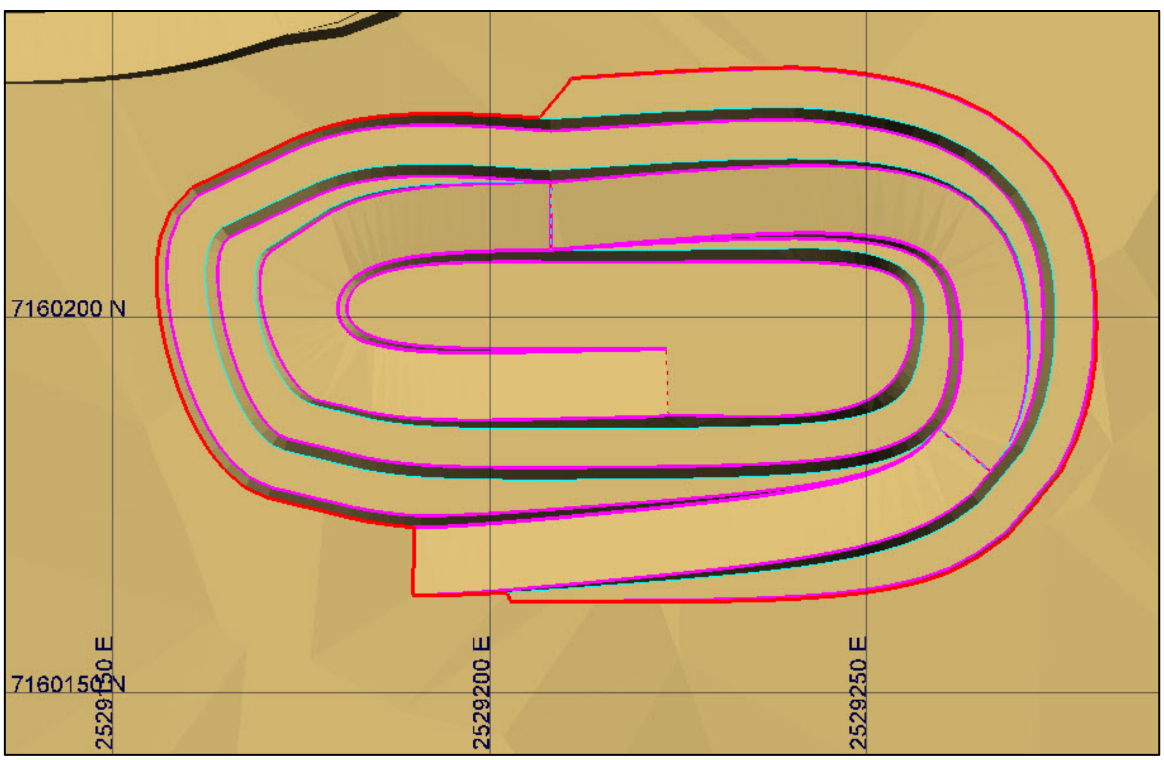

Figure 16.22
Laiva East Pit Phase 3 Design

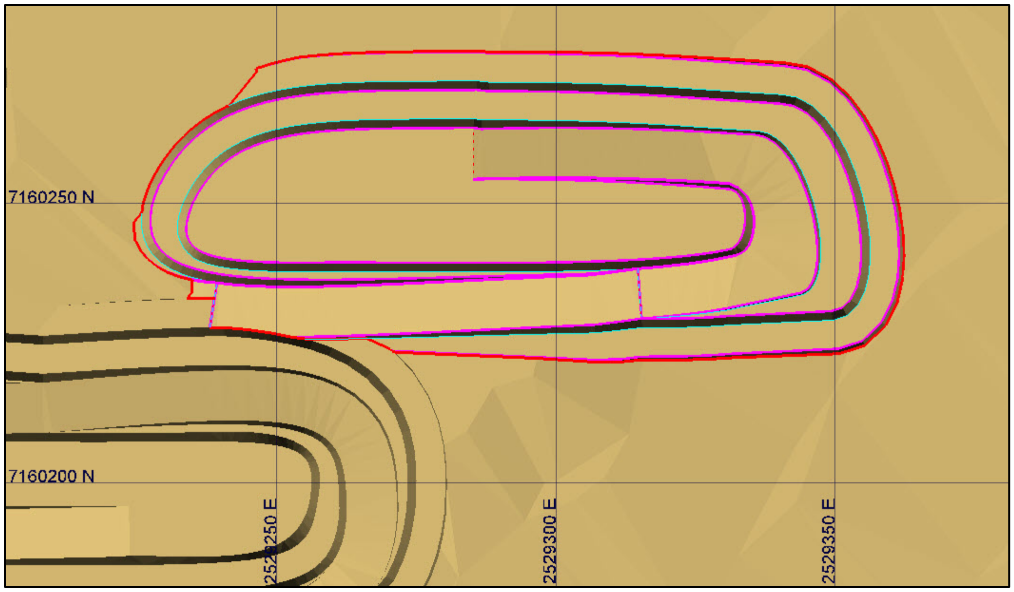

Figure 16.23
Laiva East Pit Phase 4 Design

16.1.6 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 overleaf in Figure 16.24.

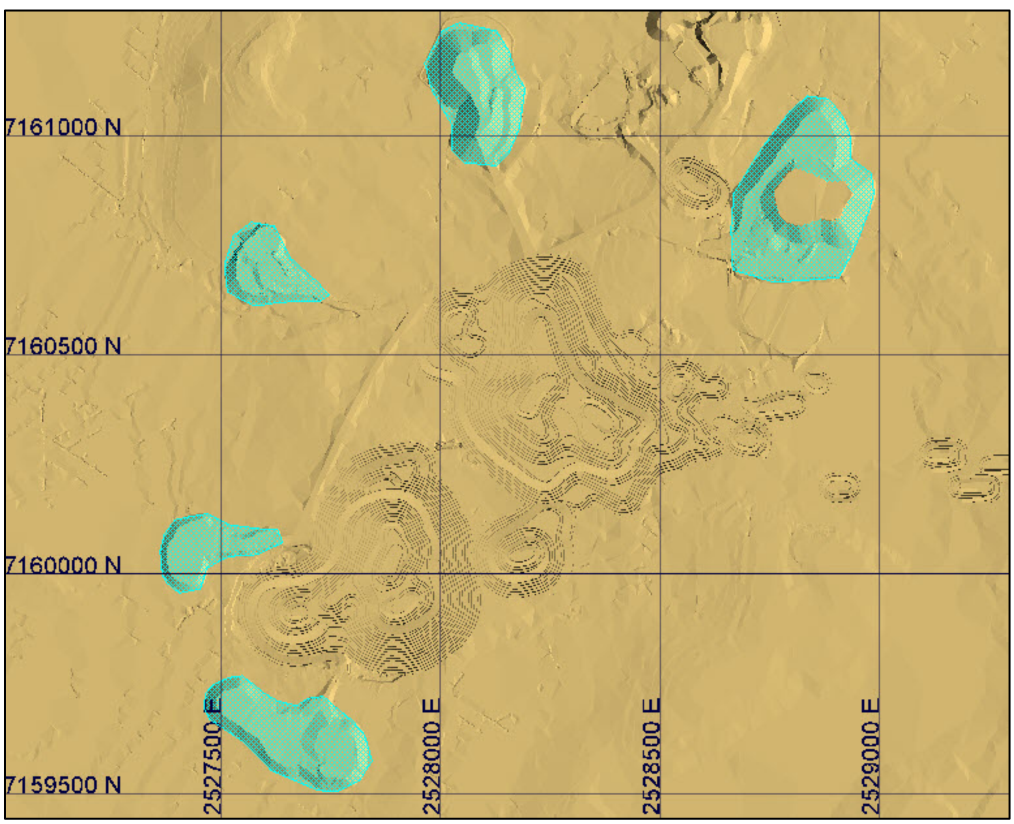

Figure 16.24
Laiva Existing Waste Disposal Areas (shown in blue)

16.2 Mine Production Schedule

A mine production schedule was prepared using Maptek's Chronos scheduling software. Mineralized material selection was based on the economic cutoff described in Table 16.2 above (0.471 g/t). Additionally, below cutoff, mill incremental material (0.401 g/t to 0.471 g/t) was also considered as millfeed in the production schedule. The production schedule has no pre-production period as well as a reduced millfeed requirement during the first two years of production (1,500,000 tonnes in year 1 and 1,750,000 tonnes in year 2). After year 2, the mill is assumed to be at full capacity (2,000,000 tonnes annually). Other than a small amount of material at the mill, no stockpiling was used in the production schedule. Table 16.6 and Figure 16.25 shows the LOM production schedule on an annual basis.

Table 16.6
Laiva LOM Annual Production Schedule

Project Year	1	2	3	4	5	6	Total
Mill Tonnes	1,500,000	1,750,000	2,000,000	2,000,000	2,000,000	1,586,000	10,836,000
Gold Grade (g/t)	1.593	1.391	1.422	1.343	1.471	1.515	1.449
Contained Troy Ounces	76,800	78,300	91,400	86,400	94,600	77,300	504,800
Recovered Troy Ounces	69,500	70,900	82,600	78,100	85,500	69,900	456,500
Waste Tonnes	11,306,000	12,266,000	15,000,000	15,000,000	14,905,000	4,522,000	72,999,000
Total Tonnes	12,806,000	14,016,000	17,000,000	17,000,000	16,905,000	6,108,000	83,835,000
Stripping Ratio	7.54	7.01	7.50	7.50	7.45	2.85	6.74

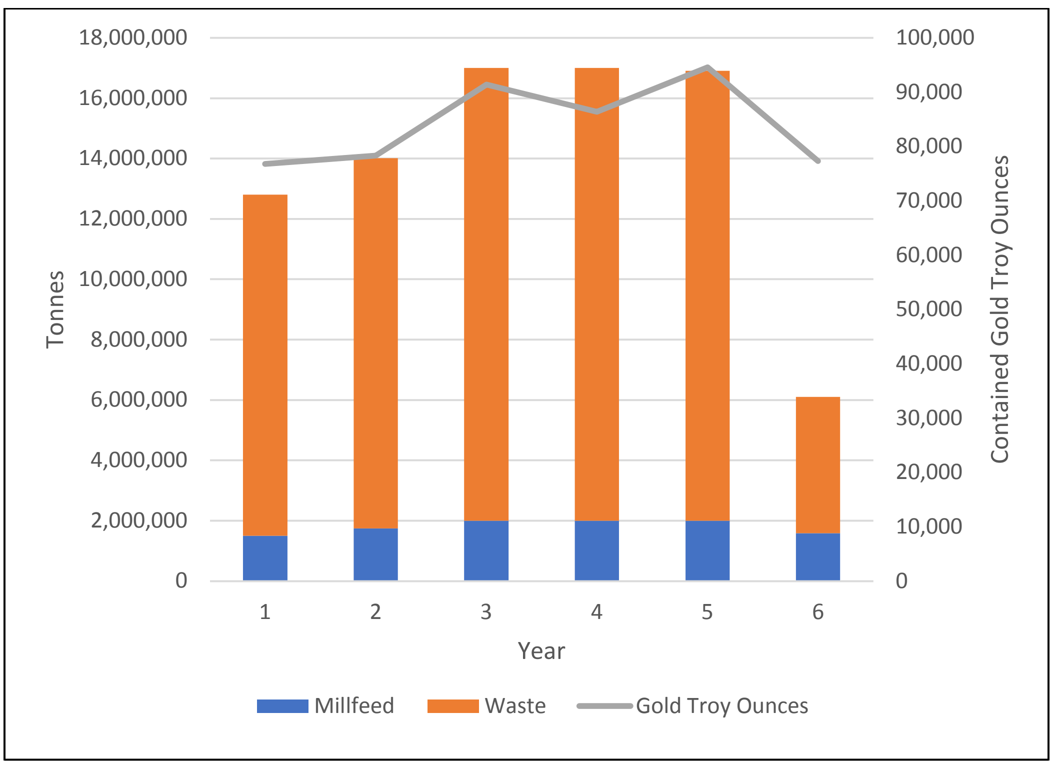

Figure 16.25
Laiva LOM Production Schedule

16.3 Mine Operations

16.3.1 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 16.7 below. The unit costs were taken from the 2016 estimated operating budget for NORDIC under the previous operator based on services proposed by Tallqvist Oy, a medium sized mining and general contractor based in Kokkola, Finland. The 2016 costs were escalated by 8% to arrive at expected unit cost for mine contracting commencing in 2018.

Table 16.7
Mine Contracting Costs

Mine Contracting Costs	Unit Cost (US $/mt)	Ave Annual Cost at Full Production ($M)
Bulk Waste Removal Total	2.70	2.759
Mineralized Material Haulage	1.01	2.014
Mine Support Services ($/total ton mined)	0.10	1.440
Mine Contracting Total		6.249

16.3.2 Mining Activities by NORDIC

Mineralized material drilling and blasting will be performed by NORDIC as well as mineralized material loading (self-performed to maximize grade control) into contractor supplied trucks, the following tables summarize productivity calculations and equipment units required for mineralized material drilling and mineralized material loading functions units at full production to be operated by NORDIC.

Table 16.8
Average Mine Production Statistics

Mineralized Material Tons to Mill Including Mining Dilution (mt/yr)	2,000,000
Mineralized Material Grade to Mill Including Mining Dilution (g/t)	1.45
Total Waste (mt/yr)	13,480,000
Total Mining (mt/yr)	15,480,000
Strip Ratio (waste:mineralized material)	6.74
Nominal Mining Days per Year	348
Mineralized Material Tons per Day (mt/d)	5,747
Waste Tons per Day (mt/d)	38,736
Total Tons per Day (mt/d)	44,483

16.3.3 Mineralized Material Drilling and Blasting

Mineralized material and near-waste drilling is planned to be performed by NORDIC crews with NORDIC equipment. Mineralized material and near-waste will be mined on 3 m sub-sub-benches to minimize inherent dilution at the hanging and foot wall contacts of a dipping dipping mineralized material zone (+/- 80 degrees). Mineralized material zones will be defined by closely spaced definition drilling discussed below.

Based on definition drilling updates to the resource block model to tightly define suitable zones for extraction of mineralized material. Three dimensional coordinates for mineralized material zone outlines will be derived from the model and will be surveyed to define the projected mineralized material zones on each bench for drilling by NORDIC crews. A buffer of near-waste amounting to approximately 15% will be included within the area defined for “mineralized material” drilling to provide for imprecision in mineralized material zone definition resulting from the foregoing procedure. Cuttings produced from each blast hole will be sampled to more precisely determine mineralized material/waste interface and will be mapped according prior to blasting.

Hanging wall waste will be blasted and mucked prior to executing mineralized material blasting. Blast patterns and delay periods will be laid out as necessary to minimize mineralized material and waste mixing and in the case of mineralized material shots to minimize heave.

NORDIC engineers will design and layout blasts. Loading and shooting is expected to be performed by the supplier of blasting agents and accessories.

Mineralized material and near-blast hole drilling productivities and costs are presented overleaf in Tables 16.9 and 16.10. See generalized mining schematic in Figure 16.26.

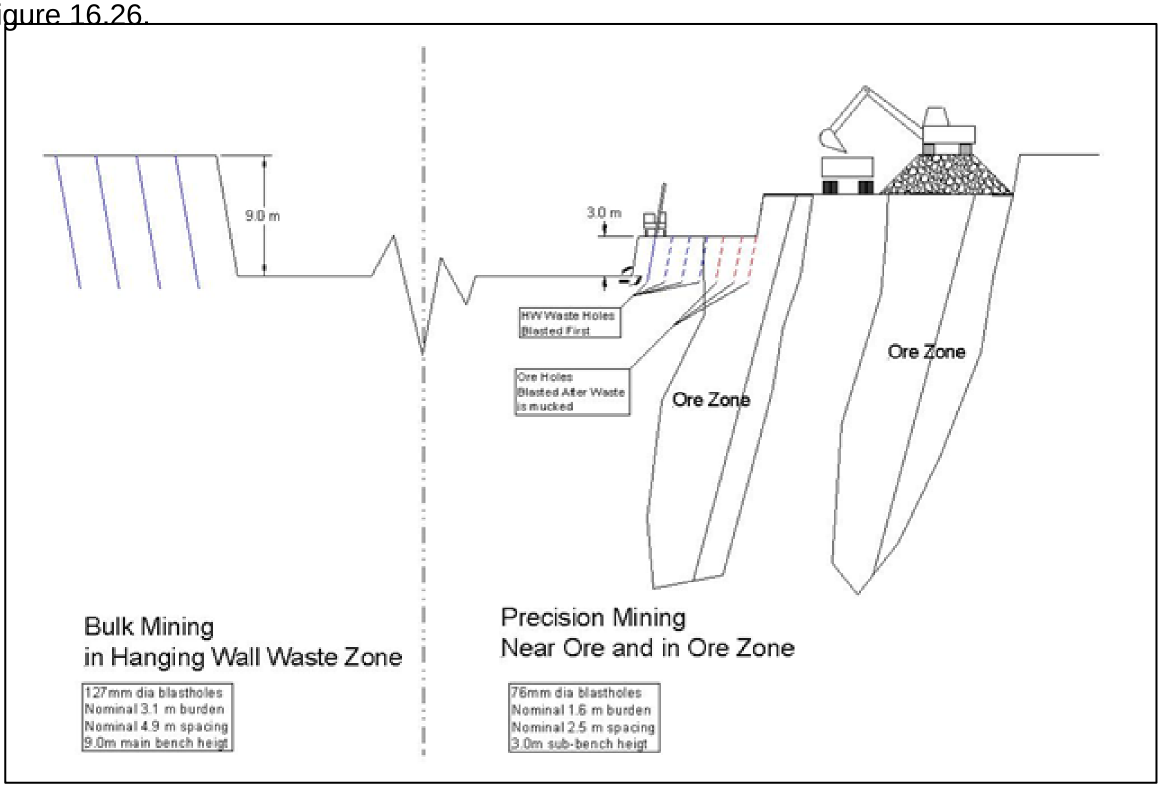

Figure 16.26
Generalized Mining Schematic

Table 16.9

Drill Productivity and Units Mineralized Material Only

Burden (m)	1.6
Spacing (m)	2.5
Hole depth (m)	3.05
Stemming (m)	1.20
Subdrilling (m)	0.5
Volume / meter of drilling (cu m)	4.00
Density (mt/cu m)	2.55
Tonnage per meter drilled (mt)	10.20
Near mineralized material waste factor included w drilling (%)	15
Meters drilling required per day (ex subdrilling)	256.92
Meter per blasthole less subdrilling	2.52
Blast holes required per day	107.00
Total drilling required per day (m)	326.35
Productivity	
Penetration rate (m/min)	1.25
Total drilling time required per day (min)	261.08
Hole flush and condition (min/hole)	2
Total flush and condition time/day (min)	214.00
Moves and set-up time (min/hole)	3
Total moves and setup time/day (min)	321.00
Total drill working time required per day (min)	796.08
Drilling shifts per day (2 drills Shift A and 1 drill Shift B)	1.5
Effective working hours per working shift	7
Effective working minutes per effective hour	50
Total effective drill minutes per day per drill	525
Drills required for mineralized material production	1.6
Drills in Fleet (Atlas Copco Power ROC T35 or equal)	2

Table 16.10

Mineralized Material Drilling Costs

Fuel and Lube		
Liters per working hour consumption	17.03	
Fuel cost per liter (US $/litre)	0.79	
Fuel cost	13.46	65.563
Lube cost	20%	13,113
Fuel and Lube Cost per ton mineralized material (US $/t)	16.15	0.04
Bits Unit cost US $/ea	140.00	
Bit life (m)	200.00	
Bit cost ($/mt)	0.70	79,499
Bit cost per ton mineralized material		0.04
Drill rods unit cost US $/ea	375.00	
Rod life (m)	1,000	
Rod cost ($/m) $/year)	0.38	42,589
Rod cost per ton mineralized material		0.02
Drill maintenance and repair cost ($/hr) ($/yr)	31.50	153,468
Drill maintenance and repair cost per ton Operator cost		0.08
per period including burdens ($/hr) ($/yr) Operator cost	30.51	254,820
per ton including burdens (US$/hr)		0.13
Total Drilling Cost ($/Yr)		609,051
Drilling Cost Per Ton Mineralized Material		0.30

16.3.4 Mineralized Material Loading

Mineralized material is planned to be loaded by relatively small 36 metric ton class hydraulic excavators utilizing 1.5 m width buckets with heaped capacity of 1.9 cubic meters. Mineralized material will be loaded into nominal 50 mt trucks supplied and operated by the mine contractor. The purpose for using smaller class loading and hauling equipment is to maximize selectivity. Each loading operation would be directed by a grade control technician to actively distinguish and direct separation of mineralized material and waste. See Tables 16.11 and 16.12, below for mineralized material loading productivity and cost.

Table 16.11
Loading Productivity and Units Mineralized Material Only

Productivity	
Bucket capacity (cu m heaped)	1.9
Broken muck density (mt/cu m)	1.65
Tons per pass	3.135
Passes required per day	2,109
Truck capacity (cu m)	33
Truck capacity (loose tons)	54.45
Trucks per day	122
Cycle time per truck load (sec)	
Bucket fill	10
Swing	5
Dump	3
Return swing	7
Total Cycle (sec)	25
Cycles per truck	18
Loading time per truck (min)	7.50
Truck spot (min)	0.30
Total cycle time per truck (min)	7.80
Total loading time per day (min)	951.60
Loading shifts per day	2
Hours available per shift	7.00
Minutes available per hour	50.00
Total effective loading minutes per day per unit	700.00
Loading units required for mineralized material production	1.4
Loading Units in Fleet (Caterpillar 336E or Equal)	2

Table 16.12
Mineralized Material Drilling Costs

Fuel and Lube		
Liters per working hour consumption	35	
Fuel cost per liter (US $/litre)	0.79	
Fuel cost	27.65	314,325
Lube cost	15%	47,149
Fuel and Lube Cost per ton mineralized material		0.18
Loading maintenance and repair cost ($/hr) ($/yr)	33.60	381,965
Loading maintenance and repair cost per ton		0.19
Operator cost per period including burdens ($/hr) ($/yr)	30.51	339,759
Operator cost per ton including burdens (US$/hr)		0.17
Total Drilling Cost (US$/Yr)		1,083,198
Total Loading Cost (US$/t)		0.54

16.3.5 Definition Drilling

As previously above, mineralized material zones are intended to be defined in greater detail by closely spaced definition drilling using percussion or RC drilling. Angle holes would be drilled from the hanging wall side of the mineralized material zone at -45 degree dip. Lines of definition holes would be set at 10 m along the strike of the mineralized mateiral zone. Holes would be spaced on 5 m centers along each line. Holes would be drilled to a depth of 12.73 m to cover the vertical extent of three sub-benches of 3 m each. Samples would be collected from each 1 m drill interval for assay. Drilling costs per meter are assumed to be similar to blast hole drilling. The total cost of definition drilling per average ton of mineralized material mined, including PAL assay is estimated at $0.17 per ton of mineralized material produced.

<u>16.3.5.1 Blast Hole Grade Control Sampling and Direction at the Face While Mucking</u> Also, as noted above, blast hole cuttings are proposed to be sampled, assayed, and mapped prior to blasting to minimize dilution by selective blasting mineralized material and waste separately. Blast hole cuttings would be sampled as a representative sample per blast hole for quick analysis by PAL methods. Resulting grades would be mapped prior to blasting to ensure separate blasting of mineralized material and waste. In addition, after blasting, grade control technicians would be assigned to each location during mucking to guide distinction between mineralized material and waste and direct to the proper location accordingly. The total cost per average ton of mineralized material produced, including both blasthole cuttings sampling, analysis by PAL procedures and all grade control personnel is estimated at an average of $0.39 per ton of mineralized material produced.

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17.0 RECOVERY METHODS

The Laiva Gold Mine Process facilities were constructed in 2011 and operated for a total of 17 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 a few minor modifications to enhance productivity and improve key process parameter measurement and recording systems.

The process facilities modifications include:

- Elimination of bottlenecks with crushing circuit for pebble mill.
- Installation of instrumentation throughout the plant to provide required information for an accurate metallurgical balance.
- Installation of additional concentrate re-grind capacity.
- Installation of automation system for flash flotation circuit.

During previous operations, the process plant production rate peaked at 216 tph, compared to a design production rate of 250 tph. Plant production rates were limited by the following primary issues:

- Inconsistent plant feed with respect to the ratio of hard rock to soft rock.
- Inadequate budget to install identified fixes for several materials handling issues.
- Inadequate budget to instigate a proper preventative maintenance program.
- Inadequate process plant instrumentation and automation.

The planned production rate following restart of operations is 1.5 million tons per year for year one, ramping up to 2.0 million tons per year by year three.

The lower than design production rate for the initial two years of operation provides ample opportunity to optimize the plant throughput and recovery processes.

17.1 Processing Plant and Facilities

The mineral processing plant is located approximately 1 km northeast from the South and North open pits in the Laivakangas area. Recovery method is direct leaching by CIL method. Processing of mineralized material consists of three primary circuit components: grinding circuits, pre-concentration by flash flotation and gravity concentration, followed by High and Low grade leaching circuits. Design throughput rate is 250 tph (6,000 tpd, 2 Mta). The process flowsheets are shown in Figures 17.1 to 17.3, overleaf.

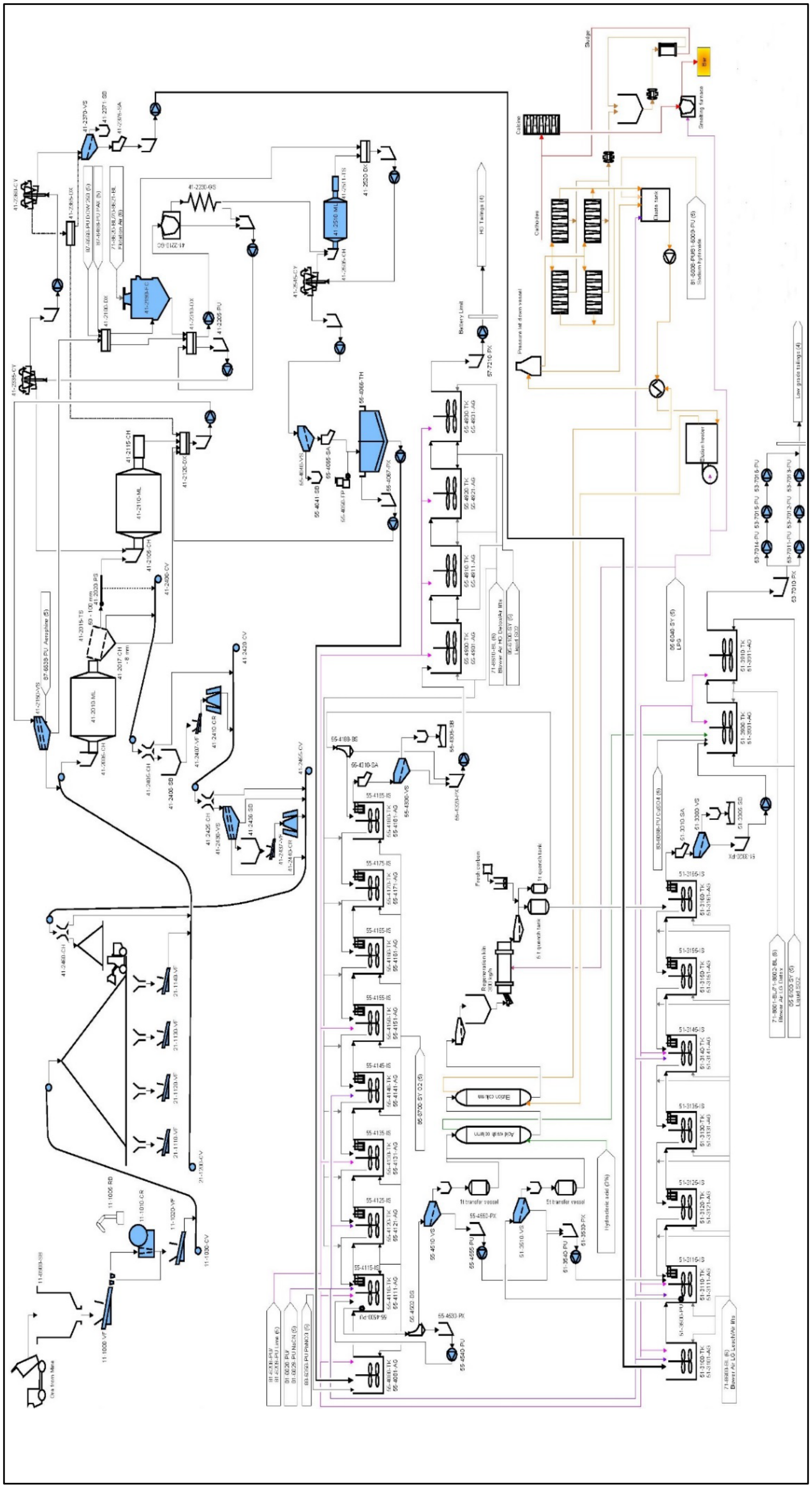

Figure 17.1
Process Flow Sheet Overview (Source: Metso)

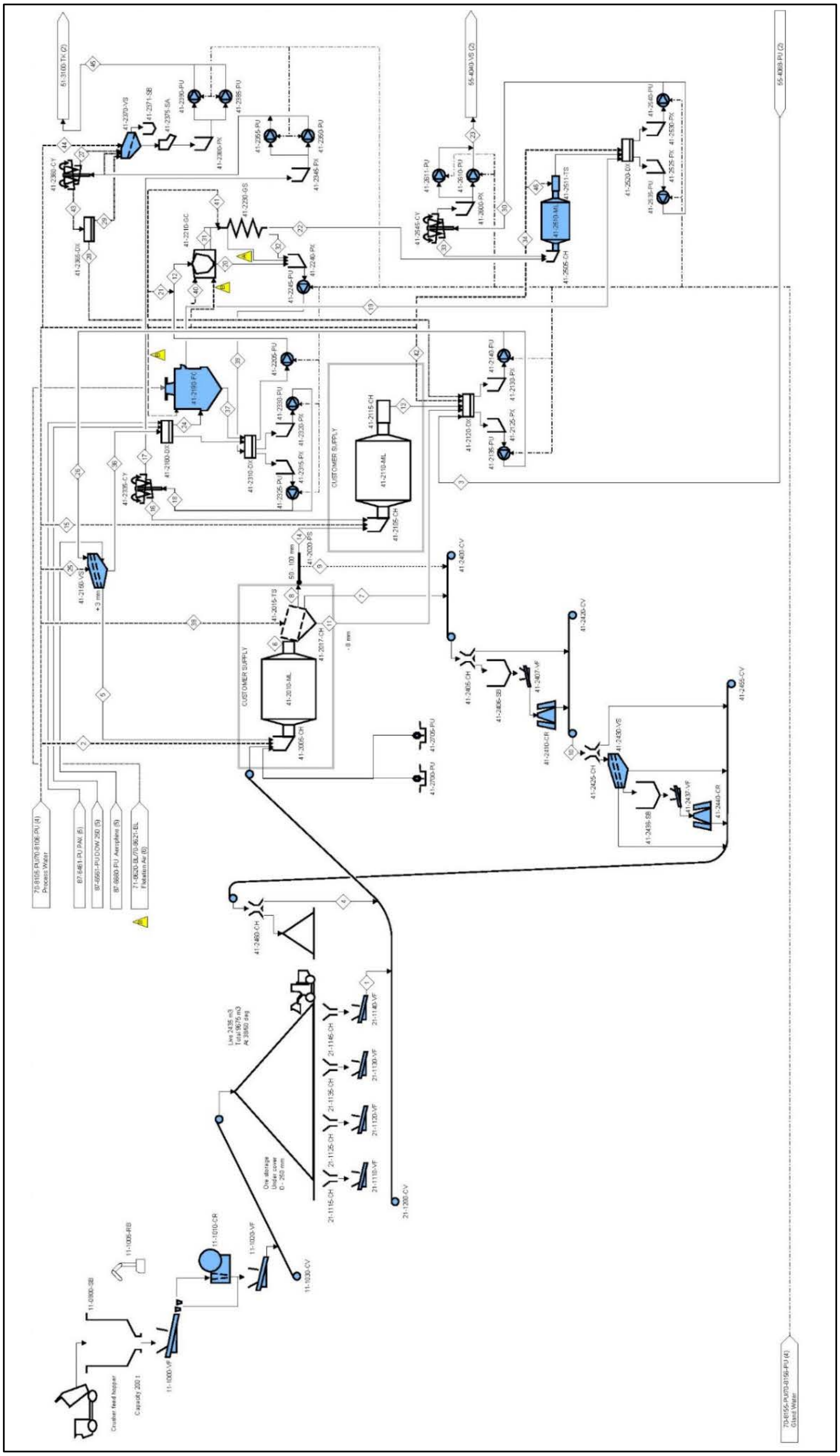

Figure 17.2
Crushing, Grinding, and Pre-concentration Circuit (Source: Metso)

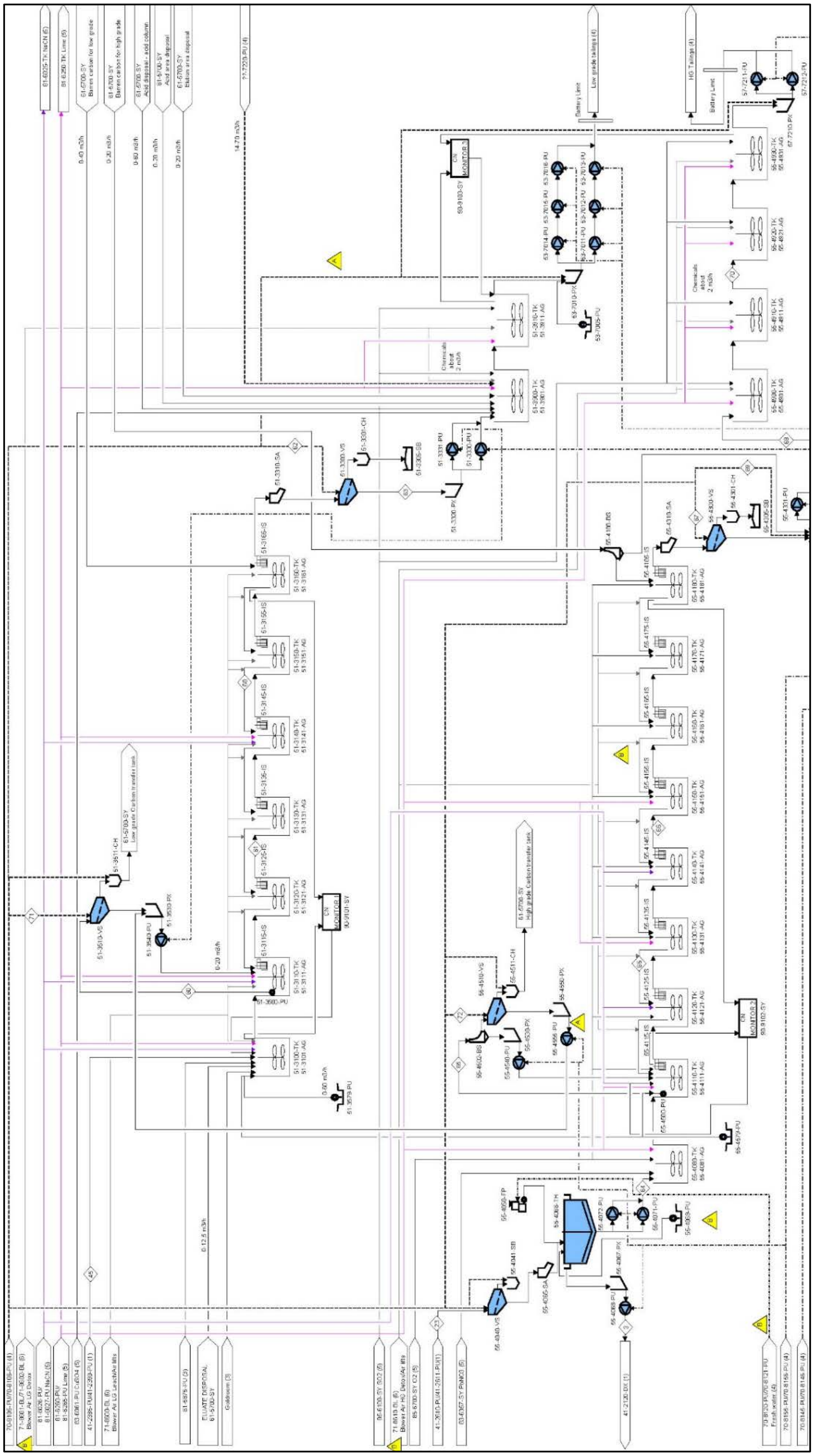

Figure 17.3
Leach Circuit (Source: Metso)

17.2 Crushing and Grinding

17.2.1 General

Mineralized material from the open pit is transported to the crushing station by haul trucks and crushed with primary crusher (Metso C 160 Jaw crusher) to - 200 mm. Crushed mineralized material is stored at the stockpile. The capacity of crushing plant is 700 tph. The capacity of the mineralized material stock pile is 25,000 tons.

From the stockpile, mineralized material is fed through vibrating feeders to the belt conveyor, which leads to the primary grinding mill, the AG Mill. Secondary grinding is done in a Pebble Mill. The discharges from the AG and Pebble mills are classified first by a scalping sieve and secondly by a cyclone cluster. A gravity separator and Flash Flotation unit produce a high grade gold-concentrate, which is treated in a CIL plant (high grade). The gold in the cyclone overflow is recovered in a separate CIL plant (low grade).

17.2.2 Crusher and Stock Pile

Mineralized material from the open pit is trucked to the crusher feed hopper on two or three shifts. The storage capacity of the hopper is 200 t. The crusher capacity is 500 – 800 tph and the maximum feed size to crusher is 1,000 mm. Crushing reduces the material size to P80 150 mm. The crushed mineralized material is transported to the stockpile by conveyor belt. The stockpile capacity is 25k ton mineralized material, corresponding to 40 hours, live capacity from that is 10k ton mineralized material, consumption of the Mill. Four vibrating feeders under the stockpile feed the mineralized material to a conveyor belt which leads to AG Mill. A wheel loader can access the mineralized material stockpile through doors and can be used to increase mill throughput by assisting more mineralized material to the feeders.

17.2.3 Primary Autogenous Mill

The mechanical scale in mill feed conveyor controls the feeders under the stockpile and the targeted feed is 250 tonnes per hour to the Outotec Autogenous Mill (AG). In an AG Mill the grinding media is the mineralized material itself. The installed power is 3,500 Kw, giving a specific grinding energy of 9.5 kWh/t, corresponding to 50% of the grinding energy. The inside diameter of the AG Mill is 6.5 m and the inside length 8.5 m.

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

A double drum trammel 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 Pebble Mill, report to the classification sieve or can be diverted directly to Pebble Mill feed if desired.

The two largest product sizes of the AG Mill can circulate back to AG Mill for regrind if needed through 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 primary cone crusher by conveyor belt.

17.2.4 Pebble Crushers System

To increase the Mill throughput, two cone crushers in series re-crush the excess amount of 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 away.

17.2.5 Secondary Pebble Mill

Underflows from the primary cyclone cluster are ground in close circuit in the Pebble Mill. The grinding media are 50-150 mm sizes pebbles from the AG Mill. The power draw of the Pebble Mill is controlled by the pebble material flow from the AG Mill. The installed power is 3,500 kW, giving a specific grinding energy of 9.5 kWh/t, corresponding to 50% of grinding energy. The inside diameter of the Pebble-Mill is 6.5 m and the inside length 8.5 m.

17.2.6 Classifying Circuit by Sieve and Cyclones

Fines discharges from the AG Mill and Pebble Mill are pumped to a two deck classifying sieve with 3 mm opening in the upper deck and 1.5 mm opening in the bottom deck. Material +1.5 mm reports back to the AG Mill and material -1.5 mm goes to the primary classifying cyclone via Flash Flotation and gravity separator circuits.

A cyclone battery, total 12 pcs, is used to classify particles ready for tails 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 O/F is P80 ~100 µm.

17.2.7 Gravity Separator and Flash Flotation

From the sieve, the undersized flows by gravity to a split box with three exits:

1. Cyclone feed pump.
2. Flash flotation.
3. Gravity separator.

The Flash Flotation unit produces high grade concentrate of gold-bearing sulphides such as 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 regrind in a small Ball Mill. The tails from flash flotation is circulated back to primary cyclone cluster for re-classifying.

In presence of coarse gold in the mineralized material, a gravity separator can treat a part of the flow from the split box. Gravity separators separate particles by density. The gravity concentration is directed to the small Ball Mill for additional grinding before subsequent cyanide leaching

The underflow from the dewatering cone is reground in the small Ball Mill to optimize grind size distribution for HG leaching and to increase feed rate to leaching. The pulp density is adjusted to 1.4 t/m3 by adding water.

17.3 Leaching

17.3.1 General

The cyanide leach facility incorporates the standard Carbon in Leach method. The initial nominal annual mill throughput of 2 million tonnes was utilized for mass balance purposes. It has been assumed that the plant will operate 8,000 hours per year (given availability 91%). The feed to the cyanidation plant is comprised of two streams; cyclone overflow and the concentrate from the Knelson gravity separator and flash flotation circuits.

Design of the low grade leach circuit, carbon adsorption, elution and gold recovery circuit, with respect to retention time, gold production etc., is based on a nominal throughput of 245 tph, containing 0.77 g /t Au, 0.004 % As, and 0.022% S, has been assumed.

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 5 tonnes per hour, containing 68.2 g/t Au, 22.7 g/t Ag, 1.2 % As, and 5.9% S has been assumed.

Additional concentrate regrind capacity will be installed prior to restart of operations. The planned throughput for the high grade leaching circuit will be increased to approximately 10 tonnes per hour. 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.

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” – 2008-02-12.

Pumps, piping, screens, cyclones, thickener and other items which required sizing and designing, depend on actual pulp flow variations, are based on an instantaneous flow rate corresponding to an annual throughput of 2 million tonnes. Reagent dosing systems and 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 prior to plant restart.

A simplified block diagram of the process flow is shown in Figure 17.4.

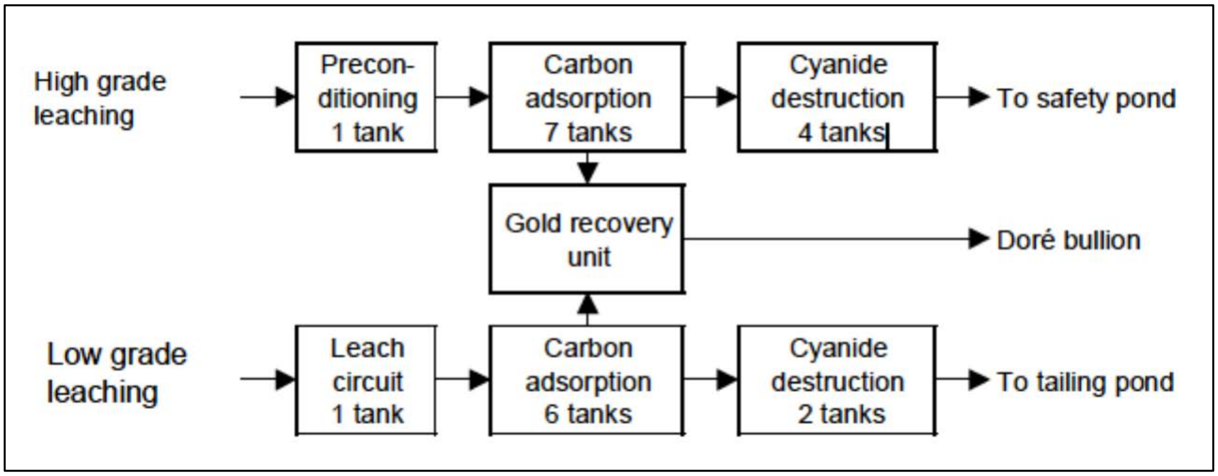

Figure 17.4
Simplified Process Block Diagram

17.3.2 Gold Leach and CIL Circuits

The leaching of gold and silver from the pulp is carried out by means of CIL. The CIL process is based on dissolution of gold and silver in an aerated pulp containing cyanide and followed by adsorption of the dissolved valuables 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 grade circuit which are relatively low in reactive sulfides are then deposited 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 or the benefits are included in this PEA analysis. A pre-conditioning tank will however be present as a first stage where lead nitrate and oxygen is added. In the low grade circuit, the first tank is a pure leach tank with no carbon in it. The following six tanks are CIL tanks.

The feed is first pumped to the trash screens, one for each stream, at an approximate pulp density of 1.4 t/m^3. 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 spray, 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 high-grade leaching). 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 will slow down 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 addition of air is not sufficient for the oxygen consumption.

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 (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.

17.3.2.1 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.

17.3.2.2 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 costs, an automatic control system is currently installed. The automated control system but 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.

17.3.3 Cyanide Detoxification

The tails from each CIL circuit is treated separately. Each stream reports to its own Detox Plant and subsequently to its own tailings pond.

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 CNtot per litre solution in the discharge.

The cyanide detoxification is accomplished in two reactors in series, providing three hours residence time on continuous basis. The reactors are located outdoors.

17.3.4 Activated Carbon Handling

<u>17.3.4.1 Pressure Zadra Elution</u>

Loaded carbon is transferred to the acid column over a period of time. 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% HCI) 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 ~ 7m^3), the carbon is then flushed with four bed volumes (28m^3) of raw water to remove residual acid and to increase pH. The rinse solution is also sent to the Detox Tank.

Having completed the acid washing and rinsing, 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 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 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 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 then 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.

17.3.4.2 Carbon Regeneration

Once the 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 further moderately dewater the carbon. 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 is 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, it will revert to silica completely destroying the carbon.

Once regenerated, the carbon falls out of the end of the kiln, via 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.

17.3.5 Electrowinning and Smelting

Gold is electro-won onto steel wool cathodes in the electrowinning cells and furthermore precipitated onto 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 for several hours.

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 is fully molten the contents separate into two phases, metal and slag. The molten contents are then poured into molds, the heavier metal remains in the bottom of the molds and the slag flows over the lip of the mold.

Finally, when the doré is solidified, the molds are emptied and gold/silver bars are then cleaned of remaining slag, drilled stamped and placed in the safe ready for shipment. Residual slag is crushed and returned to the mill.

18.0 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 2013, all infrastructure is in place and remains in a near-new state.

18.1 Logistics and Access

The project site is located approximately a 30-minute drive time 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 are 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.

As noted, 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.

Raahe, a town of approximately 25,000 population has vendors for many routine operating supplies and services, as 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.

18.2 Electrical Power

Ample grid power is available to the site. An overhead line supplies 3-phase power at 120k at 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 quite inexpensive at approximately $0.046 per kwh. No significant on-site standby generation capacity is present at Laiva.

18.3 Industrial Water Supply

Water is plentiful in northern Finland. Process make-up water is supplied by the local municipality via pipeline to the Laiva site. Typical past consumption has been about 35,000 cu meters (9.25 million us gallons) per month. Cost for treated potable municipal water is relatively high at €1.02 per cubic meter delivered or US$0.005 per us gallon.

Plans are being developed to install a simple treatment plant to process recycled water and site drainage water for industrial use as process make-up water and as pump gland water, which will significantly reduce the use and attendant cost of municipal water, by perhaps as much as much as US$300,000 per year or US$0.15 per ton of mineralized material processed.

18.4 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.

No special preparation is required for NAG waste stockpiles. See locations of Waste storage in Figure 18.1, overleaf.

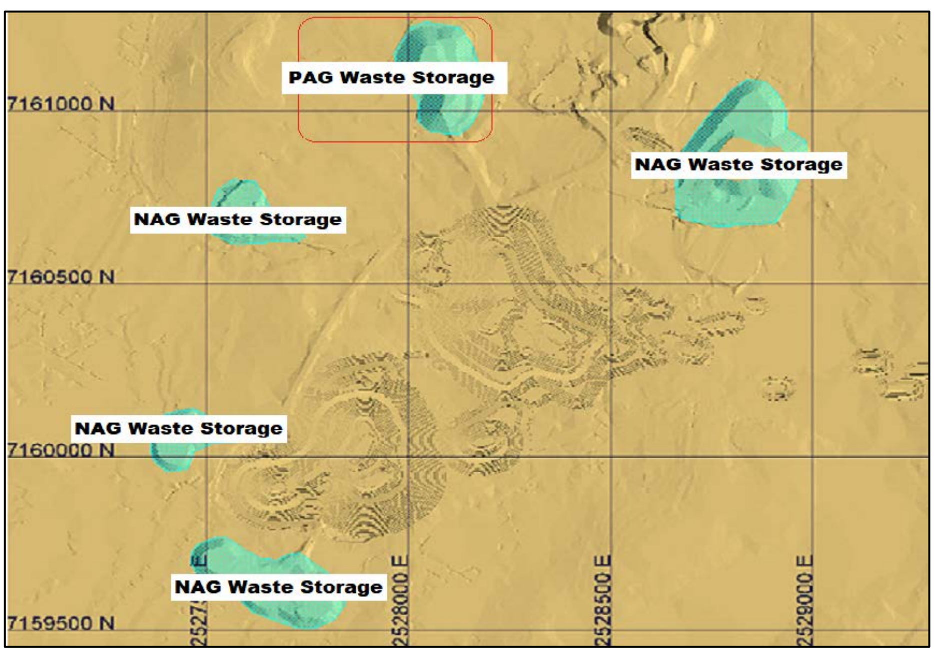

Figure 18.1
Waste Storage Areas by Type

18.5 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.

18.5.1 High Grade TSF

Following exit from the High Grade CIL circuit, the tailings pass through the Inco air/SO_2 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 NORDIC under the previous operator. The remaining life of HG tailings cell #2 (already constructed) is estimated at approximately 33 months at planned operating levels.

Laiva has previously permitted a total of five HG tailings cells. Accordingly, a third HG tailings cell will need to be constructed for use beginning in the year 2020, followed by a third cell estimated for use commencing in 2022. Based on the current minable resource, including all resource categories, two additional cells will need to be constructed upon restart of operations at planned throughput rates for the HG CIL leach circuit. Additional minable resources, if confirmed can be contained in permitted cells four and five, as required for up to an additional five years of operation, assuming similar tenor of mill feed, or nearly double the current resource estimate. Production beyond that will require amendment of the permit for HG tailings deposition.

18.5.2 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 300mm id steel pipeline. Pumping over the entire distance is accomplished by 3 each 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.

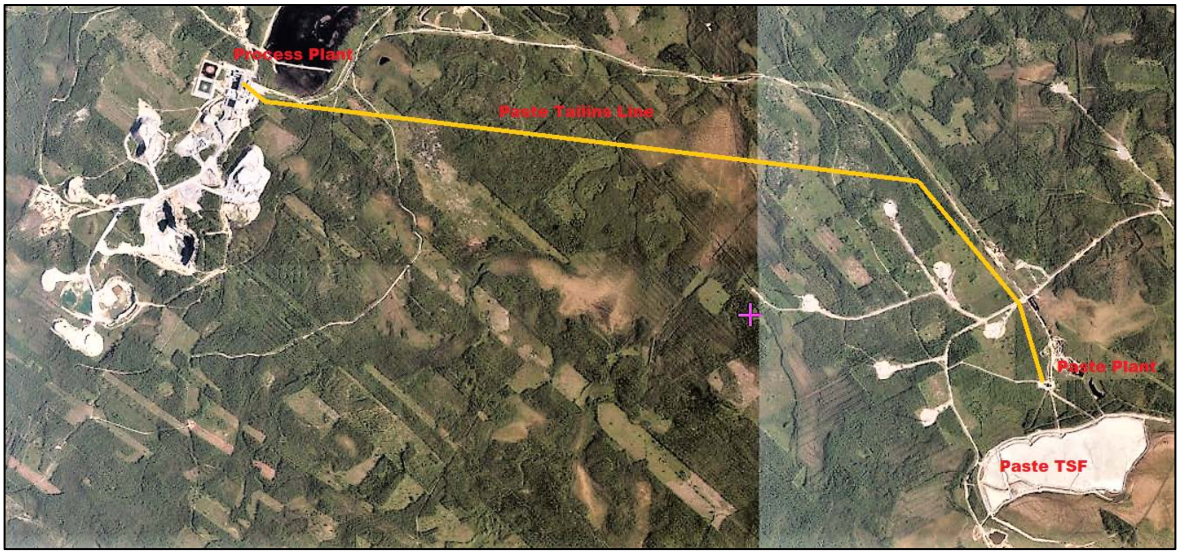

Figure 18.2
Paste Tailings System

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 have accumulated a build-up of scale thought to have resulted from sulfur reducing bacteria (Paterson & Cooke, October 2014). NORDIC is planning 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.

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 proved to be 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 dyke to direct free draining water rendered from the paste as it drained to a large collection pond which 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 as shown on Figure 18.3, provides the paste containment concept being developed by NORDIC currently.

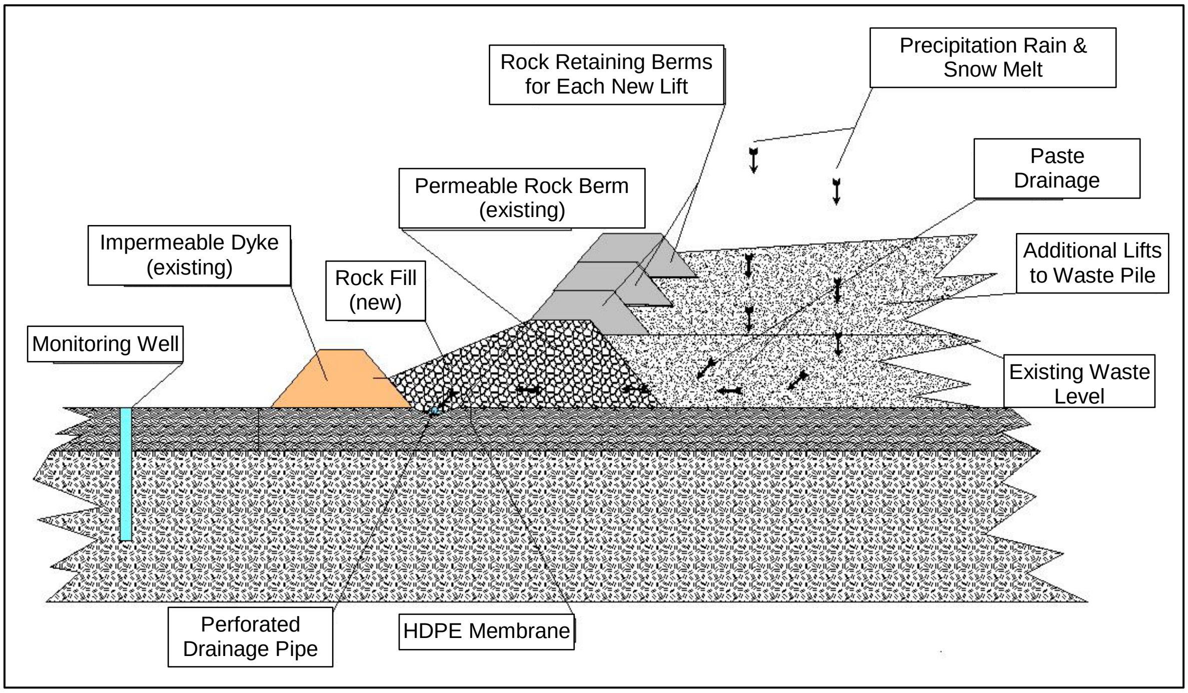

Figure 18.3
Past Confinement Conceptual Diagram (note scale)

As the design grind for the low grade tailings is relatively course (P_{80} ~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.

18.6 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. Combined clear water effluent from the storage basin 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

Gold in the form of dore bullion is expected to be delivered for sale to a recognized refinery for further processing with the resulting gold and silver being credited to FTR/NORDIC's metal account. From the metal accounts gold is expected to be delivered to the gold forward sale contract as described in Section 19.2, below, with the balance available for sale at spot price on the open market. No hedging contracts are considered for the open ounces, i.e., not a part of the forward sale contract in the financial analysis presented in Section 22.

19.1 Typical Gold/Silver Refinery Contract Terms

It is expected that FTR/NORDIC will enter into a refining agreement for dore bullion produced which generally embodies the typical terms outlined below.

- Material to be refined means gold/silver dore in the form of bullion, which is suitable for direct melting and sampling. Customer expects, but does not represent or warrant, that the bullion will contain 60% to 80% gold and 15% to 35% silver with a minimum of 90% precious metal. The dore will be shipped in the form of bars weighing approximately 20 to 23 kgs.
- Refiner warrants that metals delivered to Customer shall be of the purity required for London Good Delivery.
- The Customer shall deliver the Material to the Refiner at the Refinery. The Material shall be delivered to the Refinery at the Customer's cost in containers suitable for road and airfreight and each box shall be sealed, on the outside, with a unique numbered seal, (the "Shipment"). The unique number of each seal is to be detailed on the packing list that accompanies each delivery.
- No later than the date of shipment of the Material from the Mine to the Refiner ("Actual Date of Shipment"), the Customer shall furnish to Refiner shipping documents with information with respect to the Shipment.
- Risk of loss and damage to the Material shall pass from the Customer to the Refiner upon receipt of the Material at the Refinery.
- Title to the Customer's Material (as defined above) and precious minerals therein shall at all times remain with the Customer. Refiner is a fiduciary agent and bailee and shall not and does not take title to or have any rights (whether ownership or otherwise) to Customer's Material, unless agreed in writing by the parties.
- The Customer may ask for a physical return of the gold and silver in its shipments and can request that the ingots be shipped by Refiner to the Customer or to such

persons as the Customer shall designate from time to time and Refiner shall have no interest therein or to the amounts paid or payable from such disposition of the ingots. A mutually agreed additional cost will be applied to the ounces of gold and silver delivered under this clause.

- The Refiner shall recover and credit the Customer with the following percentages of the final agreed assayed gold and silver contents of refined Material from each Shipment:

Gold Assaying	Metal Recovery
20% - less than 50%	99.80% of the agreed assayed content
50% - less than 80%	99.85% of the agreed assayed content
80% and above	99.90% of the agreed assayed content
Silver – all levels	98.00% of the agreed assayed content

- The Customer may instruct the Refiner to credit the contents to the Customer's unallocated metal account with the Refiner and to await further instructions from the Customer.
- The Customer shall pay the Refiner's charges as follows: US$0.60 per troy ounce of Material received. (Minimum charge of US$1000.00 per shipment).
- Deleterious Element Charges:

Element	Range (PPM)	Penalty (US$ per kilo net weight received)
Bismuth	0 – less than 50	Free
	50 – less than 1,000	2.00
	1,000 – less than 2,000	3.00
	2,000 and above	Not acceptable
Mercury	0 – less than 100	Free
	100 – less than 500	4.00
	500 and above	Not acceptable
Tellurium	0 – less than 100	Free
	100 – less than 500	2.00
	500 and above	To be reviewed
Cadmiu	0 – less than 5,000	Free
	5,000 – less than 30,000	2.00
	30,000 and above	To be reviewed
Tin	0 – less than 2,000	Free
	2,000 – less than 30,000	2.00
	30,000 and above	To be reviewed
Arsenic	0 – less than 2,000	Free
	2,000 – less than 10,000	2.00
	10,000 – less than 20,000	3.00
	20,000 and above	To be reviewed
Lead	0 – less than 2,000	Free
	2,000 – less than 10,000	2.00
	10,000 – less than 20,000	3.00
	20,000 and above	To be reviewed

Element	Range (PPM)	Penalty (US$ per kilo net weight received)
Selenium	0 – less than 100	Free
	100 – less than 20,000	2.00
	20,000 – less than 30,000	3.00
	30,000 and above	To be reviewed
Copper	0 – less than 50,000	Free
	50,000 – less than 75,000	2.00
	75,000 – less than 100,000	3.00
	100,000 and above	To be reviewed
Antimony	0 – less than 10,000	Free
	10,000 – less than 50,000	3.00
	50,000 and above	Cannot process

19.2 Gold Forward Sale Contract

In November 2017 FTR announced that it had entered into a prepaid gold forward purchase contract with PFL Raahe Holdings LP, an Ontario Limited Partnership, controlled by Pandion Mine Finance LP based in New York, New York, for a portion of the production from the Laiva Project. The forward sale contract provides for the advance payment of US$20.6 million to FTR/NORDIC on the sale of 67,155 troy oz gold to be delivered over a period of 60 months following 17 months from closing of the forward sale contract, as well as other provisions

This represents approximately 14.5% of the total recovered gold at an estimated 91.7% average recovery rate from the mineable resource including gold contained in Measured, Indicated, and Inferred categories included in this PEA. This PEA incorporates resources presented in the NI 43-101 Technical Report – Resource Estimate with an effective date of 9 August 2017 and filed in final form with the TSX Venture Exchange on 17 October 2017 as adjusted by application of the mining plans presented in section 16.0 herein. The forward sale delivery obligation when based only upon Measured and Indicated categories for mineable resources as above is 50.6%.

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 indeed operated for the period February 2012 through March 2014. Since then the operation has been on active care and maintenance, including maintenance of requirements specified under permits issued to NORDIC for the Laiva Project. Accordingly a brief discussion of environmental matters is believed to be appropriate, notwithstanding the subject of this Technical Report being devoted to resource estimation for the Laiva Project. All necessary permits for operation are reported by NORDIC to remain active and in good standing.

20.2 Permitting

The holder of an exploration license is permitted to conduct exploration activities such as mapping, channel sampling, and drilling without the need for additional permits.

The mining license granted to NORDIC permits the operator to extract 2 million tons of mineralized material per year and to conduct related mining activities. An environmental permit has been granted to NORDIC by the Environmental Permit Authority (GPA), which applies to open-pit excavation, mineral processing, waste rock, tailings, and waste water treatment and management. The environmental permits granted to NORDIC are summarized in Table 20.1.

Table 20.1
Environmental Permits

Description		Authority	Current Status	Date of Approval	Reference	Comment
Environmental and water resources management permit for Laiva mine.		Regional State Administrative Agency of Northern Finland	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	Regional State Administrative Agency of Northern Finland	Granted	29.10.2010	Permit decision no 100/10/1 PSAVI/196/04.08/2010	

Table 20.1
Environmental Permits

Description		Authority	Current Status	Date of Approval	Reference	Comment
Permit change	Permit to change construction of paste-area		Granted	1.6.2016	Permit decision no 76/2016/1 PSAVI/126/04.08/2013	Includes obligation to update risk assessment, dam safety documents, overflow water management plan, closing plan and liabilities by 31/12/2017 (decision no 7/2017/1, PSAVI/2982/2016)
Permit change	Application to take foreign origin concentrates into the process		Granted	11.5.2017	Permit decision no 33/2017/1 PSAVI/1539/2016	
Permit for building process water pipeline under Pattijoki river and Haapajoki river		Regional State Administrative Agency of Northern Finland	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		Regional State Administrative Agency of Northern Finland	Granted	24.11.2009	Permit decision no 86/09/2 Psy-2009-y-28	
Energy efficiency plan of Laiva mine for operation		Regional State Administrative Agency of Northern Finland	Granted	12.6.2012	Permit decision no 80/12/1 PSAVI/10/04.08/2011	
Permit to build dam around Vaarainjärvi sedimentation pond		Regional State Administrative Agency of Northern Finland	Granted	5.12.2011	Permit decision no 75/11/2 PSAVI/5/04.09/2011	
The waste management plan for mining		Regional State Administrative Agency of Northern Finland	Granted	30.6.2014	Permit decision no 65/2014/1 PSAVI/74/04.08/2011	Needs to be updated within 6 months from operational start (POPELY/2663/2015, 28/12/2016)
Environmental permit of Nordic Mines Oy waste water discharge pipeline outlet place and permit for commissioning the construction work		Regional State Administrative Agency of Northern Finland	Granted	12.6.2012	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		Granted	7.1.2016	Permit decision no 4/2016/1 PSAVI/34/04.08/2013	

Table 20.1
Environmental Permits

Description	Authority	Current Status	Date of Approval	Reference	Comment
Waste water discharge pipeline under Hörskönjoki river and Haapajoki river, and permit for commissioning the construction work	Regional State Administrative Agency of Northern Finland	Granted	23.2.2012	Permit decision no 15/12/2 PSAVI/70/04.09/2011	
Laiva mine rehabilitation plan	Regional State Administrative Agency of Northern Finland	Granted	30.6.2014	Permit decision no 66/2014/1 PSAVI/110/04.08/2011	Needs to be updated within 6 months from operational start (POPELY/2663/2015, 28/12/2016)
Environmental permit of Laiva mine fuel station	Regional State Administrative Agency of Northern Finland	Granted	15.3.2013	Permit decision no 25/2013/1 PSAVI/23/04.08/2012	

NORDIC has also applied for a permit to store sulphide-rich waste rock on peat lined dumps. The application for this permit was made on 2 December 2014 and a decision by the EPA is pending.

20.3 Environmental Monitoring Requirements Under Mining Permit

Under the terms of the mining permit, an environmental monitoring program is required and must be conducted regardless of whether the mine is in operation. During operations, the monitoring program includes surface and ground water sampling, as well as biodiversity, dust, noise, and seismicity monitoring. A sea water monitoring program at the outflow pipe is conducted jointly with SSAB and Raahe Water.

A revised monitoring program for the current period of care and maintenance was approved by the Centre for Economic Development, Transport and the Environment of North Ostrobothnia on 22 July 2014 and revised on 1 June 2016. Prior to recommencing operations, approval of the environmental monitoring program must be sought from the Centre for Economic Development, Transport and the Environment of North Ostrobothnia.

A mine rehabilitation plan was approved by the EPA on 30 June 2014; however, the rehabilitation plan will need to be updated and approved by the EPA within six months of re-commencing operations.

The monitoring programs at Laiva are summarized in Table 20.2, overleaf.

Table 20.2
Environmental Monitoring Approvals

Description	Authority	Current Status	Date of Permit Approval	Reference	Comment
Approval of Laiva Mine environmental monitoring program	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	Granted	9.11.2011	POPELY/ 100/07.00/2010	
Approval of Laiva Mine environmental monitoring program (lightened)	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	Granted	22.7.2014	POPELY/ 100/07.00/2010	This lightened monitoring program has been in use during the period of care and maintenance
Approval of Laiva Mine environmental monitoring program for "care and maintenance" period	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	Granted	1.6.2016	POPELY/ 2663/2015	Further on lightened monitoring program for care and maintenance period. Monitoring plan for operation must be accepted by authorities before production starts again. (POPELY/2663/2015, 28/12/2016)
Approval of mutual coastal monitoring at Raahe Sea areas	Centre for Economic Development, Transport and the Environment of North Ostrobothnia	Granted	1.2.2017	POPELY/2462/2016, LAPELY 1459/5723-2016	Mutual monitoring program for sea water, fishery and environmental impact monitoring around discharge area in sea. Mutual monitoring is demanded in environmental permits and replays former company based monitoring.

20.4 Further Updates to Existing Permits and Monitoring Plans

Owing to the acquisition transaction which was on-going during 4Q 2017, NORDIC under previous ownership did not complete geotechnical testing and analysis of the existing permitted paste TSF (Permit decision no 76/2016/1PSAVI/126/04.08/201).

As a consequence, NORDIC was advised following acquisition by FTR in early 2018 that the geotechnical testing and analysis would need to be completed by the new owner, FTR, and may require completion and approval prior to restart of production. NORDIC is nearing completion of the geotechnical test work and will be completing the analysis thereupon over the next 60 days. BOYD believes that based on analysis to date together with a plan for remediation generally described in Chapter 18, no material risk is present with the existing paste TSF which would may updating the permit and restart of production in parallel with completing the remediation proposed.

In addition to the foregoing, several updates to current permits and monitoring plans are required as a matter of routine on an on-going basis.

20.5 Closure Cost

As provided under Finnish Statute, cost for project closure is paid into a sinking fund on a per square meter of disturbance requiring remediation upon final closure. The cost for each classification of disturbance varies based on the expected final cost of remediation. Table 20.3 shows the current annual cost per square meter for each classification as well as the total expected to be paid in to the sinking fund at completion of the project based on the current total minable resource in all categories.

Table 20.3
End of Project Sinking Fund Balance

Facility Classification	Current Rate (US $/ sq m)	Total End of Project
High Grade Tailings Cells	8.25	2,399,174
Paste TSF	1.18	8,790,825
PAG Mine Waste	2.95	3,169,556
NAG Mine Waste	1.18	8,451,929
Current Fund Balance		1,500,772
End of Project Total		24,312,256

It should be noted that annual rates paid in to the sinking fund for each facility classification are subject to periodic review by the agencies in cooperation with NORDIC. Accordingly, rates may increase if it is determined that the sinking fund balance is insufficient to cover the anticipated remediation cost at the end of the end of the project life. However, in BOYD’s opinion, the sinking fund balance projected at the end of the project life is generally sufficient to cover the remediation required under the closure plan.

20.6 Community and Social Impact

Mining and heavy industrial activities are common and well accepted in northern Finland. The Laiva Project is located in an area of forest and agricultural activity. The project site is located more than 5 km from the nearest community. Accordingly, no significant impacts to communities or material changes are believed to exist. The Laiva Project was operated in its current configuration previously with no apparent impacts.

21.0 CAPITAL AND OPERATING COSTS

21.1 Summary

A single production scenario was considered for the PEA. The estimated capital costs are shown in Table 21.1.

Table 21.1
Summary of Estimated Capital Costs (US$)

Item	Pre Production	Sustaining	Total
Mine	1,068,000	2,062,000	3,130,000
Processing	2,923,000	-	2,923,000
Infrastructure	377,000	2,270,000	2,647,000
Indirects	130,000	-	130,000
G&A	2,171,000	-	2,171,000
Contingency	446,000	262,000	708,000
Total	7,115,000	4,594,000	11,709,000

The estimated operating costs are shown in Table 21.2.

Table 21.2
Summary of Estimated Operating Costs
(US$, Unadjusted for Inflation)

Section	Units	Cost (US$)
Mining	$/t Mineralized Material	22.69
Processing	$/t Mineralized Material	12.07
Infrastructure	$/t Mineralized Material	0.29
Indirects	$/t Mineralized Material	0.15
G&A	$/t Mineralized Material	1.94
Total Operating Cost		37.14

21.2 Capital Costs

21.2.1 Mine Capital Costs

Mining for the Laiva PEA is envisioned to be an open-pit operation using a mine contractor. As such, there is minimal capital costs associated with the mine. Table 21.3 shows the estimated mine capital costs.

Table 21.3
Estimated Mine Capital Costs (US$)

Cost Center	Pre Production	Sustaining	Total
Pit Dewatering			
New pumps	94,000	-	94,000
Additional pipe & fittings	25,000	-	25,000
Electrical distribution	38,000	-	38,000
Power cost (US $/kWh) (connect)	31,000	-	31,000
Dewatering Labor (including burden)	8,000	-	8,000
In-fill Drilling	488,000	-	488,000
Definition Drilling (m) (US $/m)	48,000	-	48,000
Pit Area Magnetic Survey	17,000	-	17,000
Blast-hole Drill Capital Lease (2 each)	-	1,246,000	1,246,000
Excavator Capital Lease (2 each)	-	816,000	816,000
Mine Pick-ups	120,000	-	120,000
Mine Grade Control Lab			
PAL machine (FOB site)	85,000	-	85,000
Miscellaneous equipment	24,000	-	24,000
Structure	90,000	-	90,000
Total Mine Capital	1,068,000	2,062,000	3,130,000

21.2.2 Process Capital Costs

Process capitals costs are detailed below in Table 21.4.

Table 21.4
Estimated Process Capital Costs (US$)

Cost Center	Pre Production	Sustaining	Total
Primary Crusher Repairs and Upgrade			
Chain curtain	21,000	-	21,000
Dump feed retarders	21,000	-	21,000
Construct grizzly and repair pocket	30,000	-	30,000
Remount hydraulic breaker	24,000	-	24,000
Replace jaw and cheek plates	51,000	-	51,000
Replace edge rubbers on feeder	4,000	-	4,000
Repair dust collection primary cr	4,000	-	4,000
Conveyor repairs	2,000	-	2,000
Miscellaneous service	3,000	-	3,000
Mineralized Material Storage			
Spare 1200mm conveyor belt	39,000	-	39,000
Spare 1400mm conveyor belt	18,000	-	18,000
Spare 650mm conveyor belt	9,000	-	9,000
Safety wall for #5 feeder	15,000	-	15,000
Repair structures in reclaim tunnel	2,000	-	2,000

Table 21.4
Estimated Process Capital Costs (US$)

Cost Center	Pre Production	Sustaining	Total
Pebble Crush Circuit			
Repower cone crusher belts – 3 each	15,000	-	15,000
Repair dust hoods	7,000	-	7,000
Repair miscellaneous leaks in chutes	8,000	-	8,000
Grinding Mills			
AG Mill			
liner change	106,000	-	106,000
gear box maintenance	8,000	-	8,000
service drive motor	30,000	-	30,000
modify cover for hydraulic unit	6,000	-	6,000
Pebble Mill			
liner change	53,000	-	53,000
repair feed chute	12,000	-	12,000
service drive motor	30,000	-	30,000
Miscellaneous Service to Mills	10,000	-	10,000
New Regrind Tower Mill			
mill complete fob site	272,000	-	272,000
structure	129,000	-	129,000
pumps, piping, electrical	88,000	-	88,000
Miscellaneous Service to Mills	5,000	-	5,000
Grinding Hall General			
Inspect and repair piping	24,000	-	24,000
Install trash screens on sumps	1,000	-	1,000
Repairs to flotation compressor	4,000	-	4,000
Change magnetic valves	5,000	-	5,000
Valve modifications on dual pumps	15,000	-	15,000
New panels for banana screen	9,000	-	9,000
Knelson concentrator mtc	5,000	-	5,000
Knelson feed box modifications	47,000	-	47,000
Maintenance and spares for cyclones	2,000	-	2,000
CIL			
LG circuit			
containment basis pump purchase	30,000	-	30,000
slurry lines to containment pumps	12,000	-	12,000
mods to safety screen enclosures	7,000	-	7,000
repair worn agitators	177,000	-	177,000
Kemix screen LG-CIL 2	9,000	-	9,000
Kemix screen LG-CIL 3	10,000	-	10,000
Kemix screen LG-CIL 4	10,000	-	10,000
Kemix screen LG-CIL 5	10,000	-	10,000
safety screen bypass	20,000	-	20,000
safety screen vent hood	13,000	-	13,000
safety screen overflow modifications	6,000	-	6,000

Table 21.4
Estimated Process Capital Costs (US$)

Cost Center	Pre Production	Sustaining	Total
HG Circuit			
agitator repair	35,000	-	35,000
replace transfer channels and valves	16,000	-	16,000
inc capacity feed pump	3,000	-	3,000
inc capacity overflow pump	3,000	-	3,000
replace Kemix screen HG-CIL 1	4,000	-	4,000
replace Kemix screen HG-CIL 2	4,000	-	4,000
replace Kemix screen HG-CIL 3	4,000	-	4,000
replace Kemix screen HG-CIL 4	4,000	-	4,000
replace Kemix screen HG-CIL 5	4,000	-	4,000
replace Kemix screen HG-CIL 6	4,000	-	4,000
safety screen bypass	8,000	-	8,000
safety screen overflow modifications	4,000	-	4,000
HG detox tank 1 repairs	2,000	-	2,000
HG detox tank 2 repairs	2,000	-	2,000
HG detox tank 3 repairs	2,000	-	2,000
HG detox tank 4 repairs	3,000	-	3,000
Chemical Handling			
modification of water delivery system	5,000	-	5,000
repair HCL system	2,000	-	2,000
modify lime circuit piping	19,000	-	19,000
install new lime line	35,000	-	35,000
repair fresh water piping	4,000	-	4,000
service chemical dosing pumps	12,000	-	12,000
Blowers and Compressors			
service compressors	19,000	-	19,000
service blowers	6,000	-	6,000
Plant Electrical			
Ventilation system for elect room	35,000	-	35,000
Control cable primary cr	2,000	-	2,000
Main UPS	9,000	-	9,000
Automate seawater control valve	4,000	-	4,000
Automation system UPS service	6,000	-	6,000
Ops station for Metso DNA	5,000	-	5,000
Miscellaneous Buildings			
Vent system for comp & blower room	35,000	-	35,000
Cooling system plant control room	2,000	-	2,000
Construct mill warehouse and shop	550,000	-	550,000
Other Plant Upgrades			
Online slurry samplers	40,000	-	40,000
Mass flow meters for HG and LG circuits	60,000	-	60,000
Plant AA	20,000	-	20,000
Automated reagent feeders for flash float	20,000	-	20,000
Met Lab Set-up	70,000	-	70,000
Initial Supplies and Spares			
Chemicals, first order	293,000	-	293,000
Initial spare parts	130,000	-	130,000
Total Process Capital	2,923,000	-	2,923,000

21.2.3 Infrastructure Capital Costs

The majority of the infrastructure capital costs are associated with the tailings storage facility and are shown in Table 21.5.

Table 21.5
Estimated Infrastructure Capital Costs (US$)

Cost Center	Pre Production	Sustaining	Total
Recycle Process Water Treatment Plant	38,000	-	38,000
Expanded PAG Waste Dump ($/sq m)	-	1,223,000	1,223,000
TSF			
HG circuit			
construct new cell complete	-	852,000	852,000
LG Paste TSF			
geotech testwork	35,000	-	35,000
clean existing line of scale & sand	41,000	-	41,000
remediationto containment	73,000	145,000	218,000
upgrades to paste distribution	30,000	-	30,000
uprade to flocculant feed system	6,000	-	6,000
service thickener	1,000	-	1,000
modify well water delivery	18,000	-	18,000
automate flocculant feed	1,000	-	1,000
Site Drainage			
monitoring system upgrade	34,000	-	34,000
upgrade pump system	-	50,000	50,000
5t Utility Truck with Hydraulic Crane	48,000	-	48,000
Const Guard House (access control)	25,000	-	25,000
General Site Power Pre Start-up	26,000	-	26,000
Domestic Water Pre Start-up	1,000	-	1,000
Total Infrastructure Capital	377,000	2,270,000	2,647,000

21.2.4 Indirect Capital Costs

Table 21.6 shows the estimated indirect capital costs.

Table 21.6
Estimated Indirect Capital Costs (US$)

Cost Center	Pre Production	Sustaining	Total
Fuel Cost and Lube Total	10,000	-	10,000
General Labor	80,000	-	80,000
Miscellaneous Supplies	40,000	-	40,000
Total Indirect Capital	130,000	-	130,000

21.2.5 Capitalized G&A Costs

Table 21.7 shows the estimated capitalized G&A costs.

Table 21.7
Estimated Capitalized G&A Costs (US$)

Cost Center	Pre Production	Sustaining	Total
Management			
GM	63,000	-	63,000
Mine manager	83,000	-	83,000
Plant manager	90,000	-	90,000
Contract buy-out - Finnas	56,000	-	56,000
Contract buy-out - Kuiper	30,000	-	30,000
Technical			
Mining engineer	49,000	-	49,000
Chief mine Geologist	33,000	-	33,000
Geologist	72,000	-	72,000
Geologist	64,000	-	64,000
Process metallurgist	65,000	-	65,000
Admin			
Accountant	63,000	-	63,000
Accounting clerk	48,000	-	48,000
Accounting clerk	17,000	-	17,000
Purchasing and Warehouse			
Manager P&W	27,000	-	27,000
Buyer	52,000	-	52,000
Clerk	17,000	-	17,000
Warehouse clerk	12,000	-	12,000
Warehouse clerk	12,000	-	12,000
Warehouse clerk	6,000	-	6,000
Warehouse clerk	5,000	-	5,000
Safety director	48,000	-	48,000
Environmental			
Environment manager	64,000	-	64,000
Environmental technician	17,000	-	17,000
Other Staff			
Dry keeper	17,000	-	17,000
Janitor	25,000	-	25,000
Environmental Monitoring Contractor	18,000	-	18,000
Insurance	160,000	-	160,000
Land Tenure Costs	40,000	-	40,000
Consultants			
Start-up Assistance	484,000	-	484,000
Environmental Strategy	17,000	-	17,000
Other Miscellaneous	100,000	-	100,000
Property Tax	128,000	-	128,000
Corporate Cost	167,000	-	167,000
Supplies	22,000	-	22,000
Total G&A Capital	2,171,000	-	2,171,000

21.3 Operating Costs

21.3.1 Mine Operating Costs

Mine operating costs are shown in Table 21.8.

Table 21.8
Estimated Mine Operating Costs (US$)

Year:	-1	1	2	3	4	5	6	Totals
Mineralized Material Waste	-	3,494,000	4,062,000	4,549,000	4,549,000	4,549,000	3,603,000	24,806,000
Definition Drilling	-	32,146,000	34,882,000	42,657,000	42,657,000	42,387,000	12,860,000	207,589,000
Grade Control	-	378,000	316,000	332,000	332,000	332,000	-	1,690,000
Mining Indirects	-	678,000	764,000	781,000	781,000	781,000	618,000	4,403,000
Total Mine Operating Cost	-	1,147,000	1,245,000	1,522,000	1,522,000	1,513,000	459,000	7,408,000
Total Mine Operating Cost $/t Min. Mat.	-	37,843,000	41,269,000	49,841,000	49,841,000	49,562,000	17,540,000	245,896,000
Total Mine Operating Cost $/t All Material	-	25.23	23.58	24.92	24.92	24.78	11.06	22.69
Total Mine Operating Cost $/Troy Oz Au	-	2.96	2.94	2.93	2.93	2.93	2.87	2.93
	-	549.25	584.55	604.13	639.81	581.03	251.65	540.55

21.3.2 Process Operating Costs

Process operating costs are shown in Table 21.9.

Table 21.9
Estimated Processing Operating Costs (US$)

Year:	-1	1	2	3	4	5	6	Totals
Processing Costs (less labor)	-	14,722,000	17,147,000	19,537,000	19,573,000	19,573,000	15,556,000	106,144,000
Labor	-	4,060,000	4,288,000	4,288,000	4,288,000	4,288,000	3,402,000	24,614,000
Total Process Operating Cost $/t Min. Mat.	-	18,782,000	21,435,000	23,861,000	23,861,000	23,861,000	18,958,000	130,758,000
Total Process Operating Cost $/t Min. Mat.	-	12.52	12.25	11.93	11.93	11.93	11.95	12.07
Total Process Operating Cost $/Troy Oz Au	-	272.60	303.61	289.22	306.30	279.73	271.99	287.44

21.3.3 Infrastructure Operating Costs

Process operating costs are shown in Table 21.10.

Table 21.10
Estimated Infrastructure Operating Costs (US$)

Year:	-1	1	2	3	4	5	6	Totals
Processing Costs (less labor)	-	463,000	509,000	555,000	560,000	565,000	502,000	3,154,000
Total Infrastructure Operating Cost $/t Min. Mat.	-	0.31	0.29	0.28	0.28	0.28	0.32	0.29
Total Infrastructure Operating Cost $/Troy Oz Au	-	6.72	7.21	6.73	7.19	6.62	7.20	6.93

21.3.4 Indirect Operating Costs

Indirect operating costs are shown in Table 21.11.

Table 21.11
Estimated Indirect Operating Costs (US$)

Year:	-1	1	2	3	4	5	6	Totals
Indirect Operating Costs	-	275,000	277,000	278,000	278,000	278,000	278,000	1,664,000
Total Indirect Operating Cost $/t Min. Mat.	-	0.18	0.16	0.14	0.14	0.14	0.18	0.15
Total Indirect Operating Cost $/Troy Oz Au	-	3.99	3.92	3.37	3.57	3.26	3.99	3.66

21.3.5 G&A Operating Costs

G&A operating costs are shown in Table 21.12.

Table 21.12
Estimated G&A Operating Costs (US$)

Year:	-1	1	2	3	4	5	6	Totals
G&A Operating Costs	-	3,486,000	3,489,000	3,512,000	3,503,000	3,518,000	3,487,000	20,995,000
Total G&A Operating Cost $/t Min. Mat.	-	2.32	1.99	1.76	1.75	1.76	2.20	1.94
Total G&A Operating Cost $/Troy Oz Au	-	50.60	49.42	42.57	44.97	41.24	50.03	46.15

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22.0 ECONOMIC ANALYSIS

22.1 Basis of Valuation

BOYD has prepared its preliminary economic assessment of the Laiva Project based on a discounted cash flow (DCF) model, from which NPV, IRR, payback and other measures of project viability can be calculated. Use of DCF-NPV analysis is a standard method within the mining industry to assess the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to evaluate the economic potential for development of the Laiva Project as proposed in the PEA, and to examine the robustness of the returns to variation in key assumptions such as gold price, capital and operating costs. Results of the PEA are used to determine if the underlying mineral project merits further study and investment necessary to advance the project to the pre-feasibility stage (and ultimately the feasibility stage).

Three economic scenarios were considered in the economic analysis:

1. A pre-tax scenario to examine the economic results of the pre-tax cash flows.
2. An after-tax scenario that assumes prior existing losses can be carried forward to offset income taxes.
3. An after-tax scenario that assumes prior existing losses cannot be carried forward to offset income taxes.

22.2 Economic Assumptions

22.2.1 Expected Gold Price

BOYD based the economic evaluation on a three-year average for the gold price, as well as various selling costs as described in Section 19.0.

The gold price used in this PEA, is:

- Gold Price – US$1,300.00 per troy ounce.

22.2.2 Exchange Rate and Inflation

All results are expressed in United States dollars (US$). Cost estimates and other inputs to the cash flow model for the project have been prepared using constant, 2018 money terms, i.e., without provision for inflation.

22.2.3 Corporate Taxation

The corporate tax in Finland is set at a rate of 20% of taxable income, after deducting depreciation depletion, and amortization. A carried forward loss of US$133,648,840 is available subject to approval from the taxing authority. BOYD's analysis assumes two different scenarios; one where the carried forward loss is available to offset the corporate income tax, while the other assumes that this loss is not allowed to offset the corporate income taxes.

22.2.4 Royalty

BOYD understands that a government NSR royalty of 0.15% is applicable to the property. The royalty has been fully provided for in the cash flow.

22.3 Technical Assumptions

Table 22.1 summarizes the main technical assumptions for the project.

Table 22.1

Technical Assumptions

Item	Unit	Value
Resource Mined	tonnes	10,836,000
Waste Rock Mined	tonnes	72,999,000
Stripping Ratio	W/O	6.72
Min. Mat. Grade to Mill	g/t	1.45
Average Gold Recovery	%	90.3
Annual Mill Production Rate	t/y	2,000,000
Recovered Gold	Troy Ounces	454,900
Initial Capital Cost	US$	7,114,000
Sustaining Capital	US$	3,285,000
LOM Revenue (average, net of royalty)	US$/t Processed	57.26
LOM Cash Operating Cost	US$/troy ounce	863.00

22.3.1 Mine Production Schedule

The production schedule does not include a pre-production period, but does include a reduced millfeed requirement during the first two years of production (1,500,000 tonnes in Year 1 and 1,750,000 tonnes in Year 2). After Year 2, the mill is assumed to be at full capacity (2,000,000 tonnes annually). Other than a small amount of material at the mill, no stockpiling was used in the production schedule. Figure 22.1 shows the LOM production schedule on an annual basis.

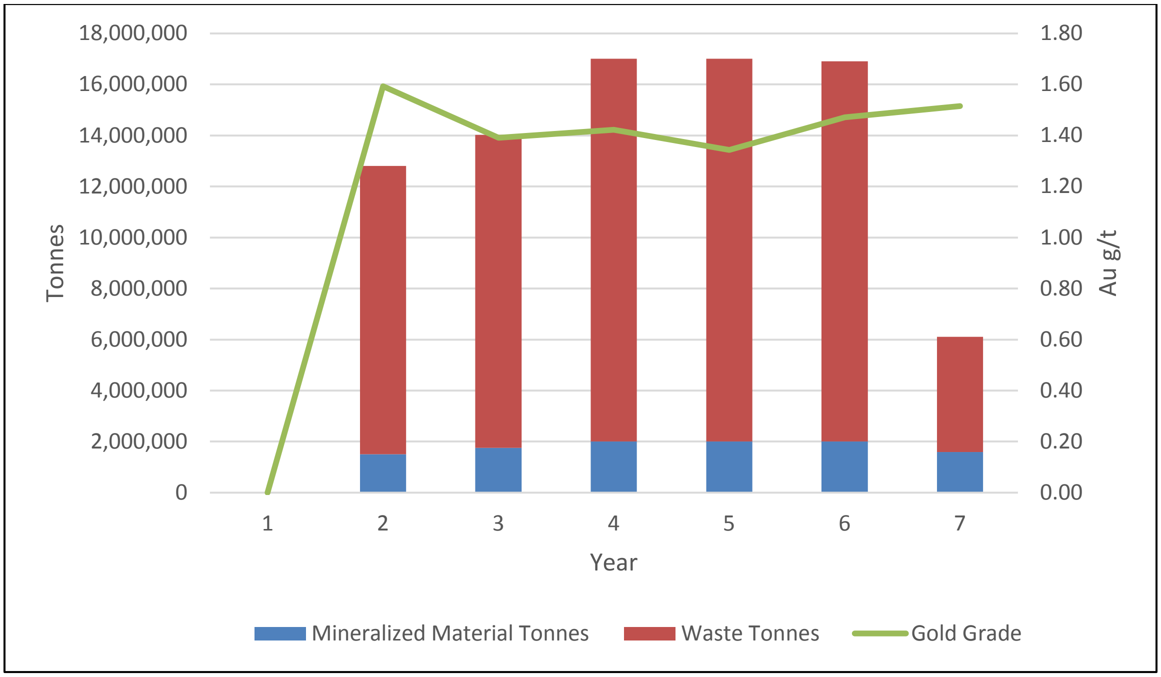

Figure 22.1
Laiva LOM Production Schedule

22.3.2 Processing Schedule

The production schedule does not include ongoing stockpiling and material mined from the open pit is assumed to be treated immediately. During years 1 and 2, the mill ramps up to reach its full annual capacity of 2.0 million tonnes in year 3. That rate continues throughout the remainder of the mine life.

22.4 Project Economics

22.4.1 Operating Costs

LOM estimates of total and unit operating costs are shown in Table 22.2.

Table 22.2
Summary of LOM Operating Costs (US$)

Section	Units	Cost (US$)
Mining	US$/t Min. Mat.	22.69
Processing	US$/t Min. Mat.	12.07
Infrastructure	US$/t Min. Mat.	0.29
Indirects	US$/t Min. Mat.	0.15
G & A	US$/t Min. Mat.	2.00
Total Operating Cost	US$/t Min. Mat.	37.14

Figure 22.2 shows the cash operating costs on an annual basis over the LOM period. Operating costs gradually increase during the first two years during the plant ramp up and remain steady during the full production years that follow.

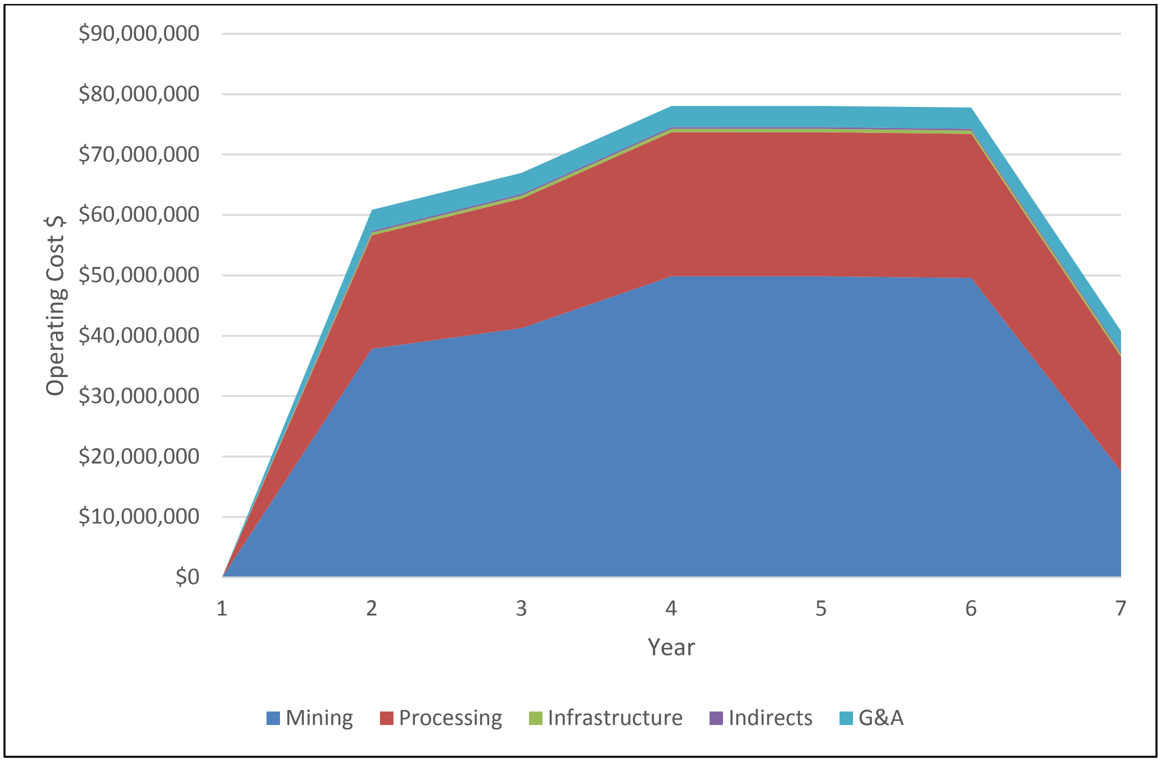

Figure 22.2

Annual Operating Costs (US$)

22.4.2 Capital Costs

The project has minimal capital requirements during pre-production, as well as relatively low sustaining capital. Capital costs are shown in Table 22.3.

Table 22.3

Capital Costs (US$)

Item	Pre Production	Sustaining	Total
Mine	1,068,000	2,062,000	3,130,000
Processing	2,923,000	-	2,923,000
Infrastructure	377,000	2,270,000	2,647,000
Indirects	130,000	-	130,000
G&A	2,171,000	-	2,171,000
Contingency	446,000	262,000	708,000
Total	7,115,000	3,285,000	11,709,000

22.4.3 Cash Flow Projection

Table 22.4 summarizes the LOM cash flows for the project, while Table 22.5 presents the annual cash flow schedule.

Table 22.4
LOM Cash Flow Summary (US$)

Net Revenue Less Royalties	591,488,000
Operating Costs	402,467,000
Operating Margin	189,021,000
Capital Expenditure	11,711,000
Pre-Tax Cash Flow	177,310,000
Tax Payable	1,118,000
Net Cash Flow after Tax	176,192,000

Table 22.5
Annual Project Production and Cash Flow

Year:	-1	1	2	3	4	5	6	Totals
Mineralized Material Tonnes	-	1,500,000	1,750,000	2,000,000	2,000,000	2,000,000	1,586,000	10,836,000
Waste Tonnes	-	11,306,000	12,266,000	15,000,000	15,000,000	14,905,000	4,522,000	72,999,000
Total Tonnes	-	12,806,000	14,016,000	17,000,000	17,000,000	16,905,000	6,108,000	83,835,000
Strip Ratio	-	7.54	7.01	7.50	7.50	7.45	2.85	6.74
Gold Grade (Au g/t)	-	1.59	1.39	1.42	1.34	1.47	1.52	1.45
Gold Recovery (%)	-	89.9	90.4	90.4	90.4	90.4	90.4	90.3
Recovered Gold (Troy Ozs)	-	68,900	70,600	82,500	77,900	85,300	69,700	454,900
Revenue	-	89,597,000	91,789,000	107,233,000	101,293,000	110,943,000	90,633,000	591,488,000
Operating Cost (US$)	-	60,849,000	66,979,000	78,047,000	78,043,000	77,784,000	40,765,000	402,467,000
Capital Cost (USR)	7,115,000	724,000	1,259,000	830,000	1,259,000	262,000	262,000	11,711,000
Pre-Tax Cash Flow (US$)	-7,115,000	28,024,000	23,551,000	28,356,000	21,991,000	32,897,000	49,606,000	177,310,000
Tax (USR)	-	-	-	-	-	-	1,118,000	1,118,000
After-Tax Cash Flow (US$)	-7,115,000	28,024,000	23,551,000	28,356,000	21,991,000	32,897,000	48,488,000	176,192,000

22.4.4 Evaluation

The undiscounted base case cash flow demonstrates that the project can provide a very favorable operating margin of 32%

The cash flow was then evaluated at a discount rate of 5% per year, with comparative results presented over the range of annual discount rates of between 5% and 10%, as shown in Table 22.6.

Table 22.6
Results of Evaluation

Discount Rate	Pre-Tax	After-Tax	
(%)	NPV (US$)	With Loss	Without Loss
5	91,540,000	90,723,000	68,965,000
8	78,272,000	77,597,000	58,202,000
10	70,642,000	70,043,000	52,026,000

IRR before, after tax with carried forward losses, and after tax without carried forward losses are 44.6%, 44.4%, and 36.5% respectively. At 5% per year, the DCF after tax with carried forward loss shows a payback period of 1.7 years. The after tax without carried forward losses shows a payback period of 2.1 years.

It should be noted that this PEA includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the conclusions of the PEA will be realized.

22.5 Sensitivity Analysis

22.5.1 Variation of Assumptions

Figures 22.3 and 22.4 show the sensitivity of the project after-tax with carried forward losses and after-tax without carried forward losses for a cash flow discounted at 5% (NPV_5) to a variation over a range of 30% above and below the base case in: (1) gold prices, (2) operating costs, and (3) capital expenditure.

As might be expected, the project is most sensitive to changes in gold price and operating cost with gold prices less than US$1,037 per troy ounce generating negative NPV. Operating costs greater than 125% of the base case also show a negative NPV.

The project is less sensitive to capital costs. There is little to no impact varying the capital costs from 70% to 130% of the base case.

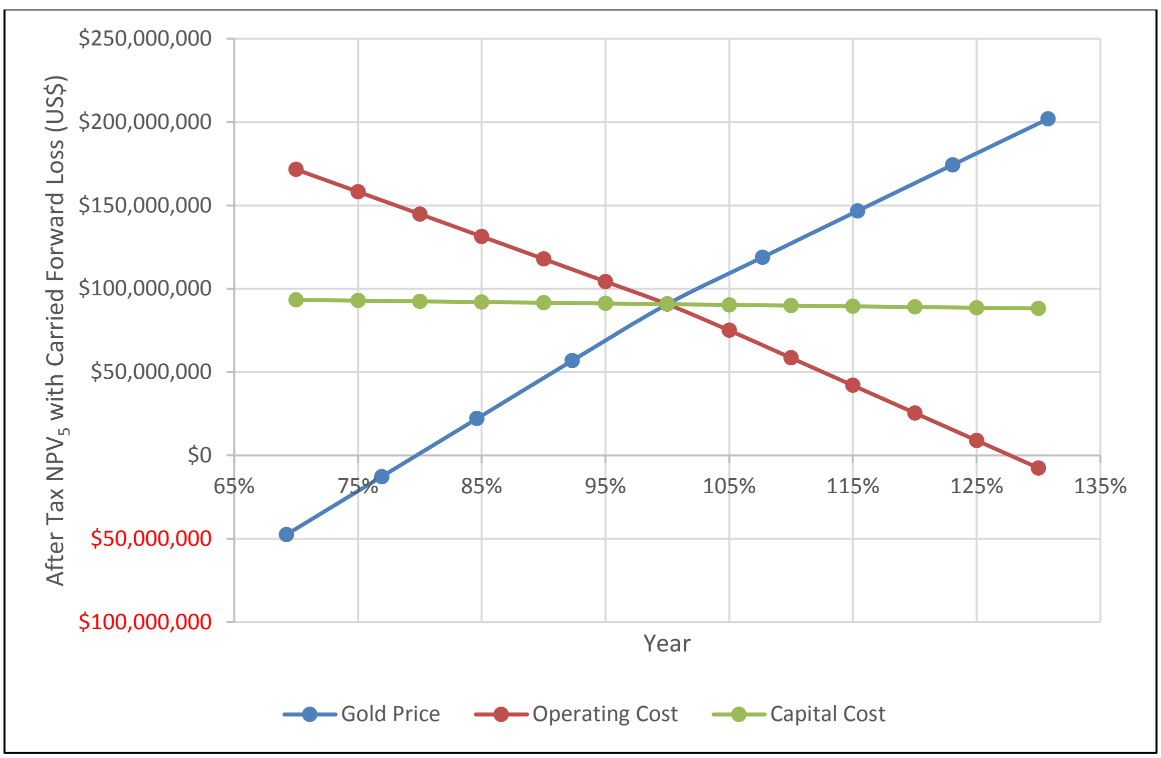

Figure 22.3
After-Tax with Carried Forward Losses NPV Sensitivity Chart

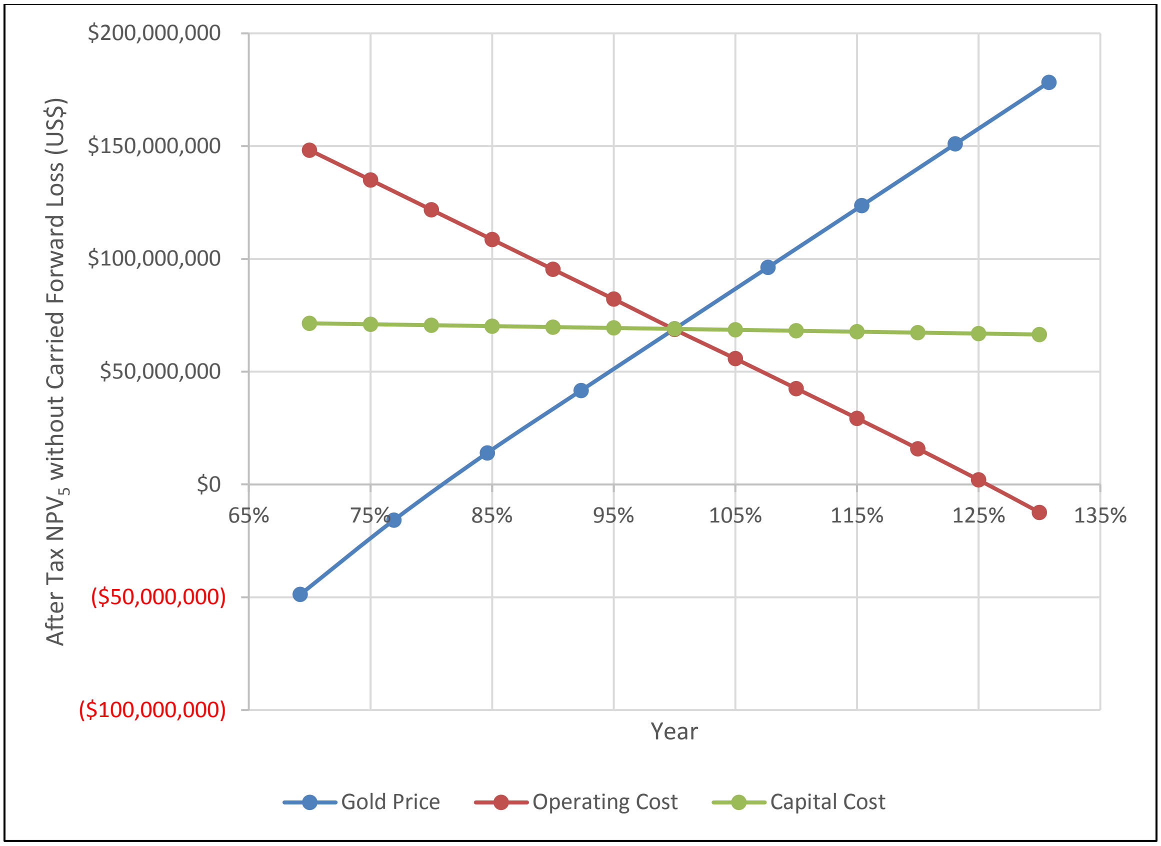

Figure 22.4
After-Tax without Carried Forward Losses NPV Sensitivity Chart

22.5.2 Gold Price Sensitivity

The sensitivity of the project economics to specific changes in gold price was further analyzed. The results are shown in Table 22.7, demonstrating positive returns with a gold price of US$1037/t oz.

Table 22.7
Sensitivity to Gold Price

Gold Price	Pre-Tax		After-Tax With Loss		After-Tax Without Loss	
US$/Troy Oz Au	NPV (US$)	IRR (%)	NPV (US$)	IRR (%)	NPV (US$)	IRR (%)
900	-47,444,000	-	47,444,000	-	-48,793,000	-
1,000	12,698,000	-2.4	12,698,000	-2.4	-15,822,000	-4.8
1,100	22,048,000	18.0	22,048,000	18.0	14,044,000	13.9
1,200	56,794,000	32.9	56,794,000	32.9	41,635,000	26.6
1,300	91,540,000	44.6	90,728,000	44.4	68,965,000	36.5
1,400	126,286,000	54.0	118,827,000	53.2	96,294,000	44.6
1,500	161,032,000	61.8	146,684,000	60.4	123,623,000	51.4
1,600	195,778,000	68.5	174,366,000	66.5	150,952,000	57.1
1,700	230,525,000	74.3	201,875,000	71.6	178,281,000	62.2

22.6 Conclusion

On the basis of this PEA of the project, BOYD concludes that exploitation of the gold resources in the Laiva Project area could provide attractive economic returns, and that further study of the project is warranted.

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23.0 ADJACENT PROPERTIES

Not applicable to this Technical Report.

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24.0 OTHER RELEVANT DATA AND INFORMATION

Firesteel requested that BOYD discuss a target for further exploration in respect of the mineralization at Laiva and surrounding exploration targets.

In keeping with National Instrument 43-101 Standards of Disclosure for Mineral Projects, the potential quantity and grade of a target for further exploration should be expressed as a range, bracketed by lower and upper potential target sizes and grades. The exploration potential described below was not considered or used in the economic analysis presented as part of the PEA.

24.1 Potential Quantity & Grade of Target for Further Exploration

Four areas have been identified as potential targets for further exploration: Near Mine, Eastern Extension, Musseneva and Oltava (Figures 7.2 and 9.2).

24.1.1 Near Mine Potential Target for Further Exploration

The potential target for Near Mine further exploration (zones below the current north and south open pits and strike extensions immediately proximal) is 3.2 to 5.1 Mt grading at 1.44 g/t Au.

The reader is reminded that a potential target for further exploration is not a mineral resource estimate, is conceptual in nature, and is used where there has been insufficient exploration to define the target as a mineral resource, and where it is uncertain if further exploration will result in the target being delineated as a mineral resource.

Basis for Determination of Near Mine Target for Further Exploration

Exploration drilling beneath the south and north pits has intercepted mineralization at depths of up to 250 m below surface (to RL -200 m) and is open down dip—apart from the western part of the North Pit where mineralization is truncated by a granite pluton. Further infill and step-out drilling is required to test the down-dip extensions of known mineralization beneath the existing open pits. Estimation of the lower bound and upper bound targets for further exploration is based on.

- <u>Width, Length and Vertical Extent of Mineralization</u>
  Mineralization in the north and south pits is well constrained by the resource outlined in this report. Mineralized shear zones in the north pit can be traced over a

strike length of up to 600 m and over a strike length of 400 m in the south pit. Mineralization occurs as multiple steeply dipping zones which vary in width between <1 to >10 m. Mineralization is open at depth.

On the basis that the Whittle Pit Shells extend to between -100 to -150 m RL, and that exploration drilling has intercepted mineralization at depths of up to -200 m RL, BOYD estimated a lower bound potential target for further exploration on the basis that further drilling could potentially extend the resource by 25% down dip (approximately to an RL of -125 to -175m). BOYD estimated an upper bound potential target for further exploration on the basis that further drilling could potentially extend the resource by 40% down dip (to approximately to an RL of -140 m to -200 m).

- Grade of Mineralization
  The upper and lower bound potential targets for further exploration was based on a grade of 1.44 g/t Au—the same weighted average grade of the measured, indicated and inferred resources reported by BOYD in this report.

Main Risks with Respect to Near Mine Potential Target for Further Exploration

The near mine potential target for further exploration is based on an extensive geological dataset including drilling beneath the existing resource—the style of mineralization, width and grade is reasonably well constrained. However, as with any exploration there are inherent risks and it is possible that infill and down-dip drilling will not confirm continuity of mineralization, width or grade.

24.1.2 Eastern Extension Potential Target for Further Exploration

The potential eastern extension target for further exploration (the eastern strike extension of north pit mineralization) is 0.85 to 3.2 Mt grading 1.25 mto 1.45 g/t Au.

The reader is reminded that a potential target for further exploration is not a mineral resource estimate, is conceptual in nature, and is used where there has been insufficient exploration to define the target as a mineral resource, and where it is uncertain if further exploration will result in the target being delineated as a mineral resource.

Basis for Determination of Eastern Extension Target for Further Exploration

Reconnaissance drilling and a recent RC infill drilling program (Section 10 of this report) indicates that mineralization extends up to 750 metres to the east of the north pit—as multiple, broadly east-west trending, steeply-dipping mineralized shear zones that display similar characteristics to mineralization in the north pit. The strike of mineralization is also confirmed by bedrock geochemistry (Figure 9.4).

Further infill, step-back and step-out drilling is required to test the down-dip and strike extensions of known mineralization of the east of the north pit. Estimation of the lower bound and upper bound targets for further exploration is based on.

- <u>Width, Length and Vertical Extent of Mineralization</u>
  Estimation of an upper bound potential target for further exploration was based on 5 mineralized zones of 300 m length, 75 m vertical extent, and 10 m width. A lower bound target was based on 3 mineralized zones, each of 200 m length, 50 m vertical extent and 10 m width. This is considered reasonable given that drilling has identified multiple mineralized zone up to 750 m from the existing north pity that extend to depths of at least 100 m. Widths are consistent with mineralization within the north pit. A specific gravity of 2.83 was used in keeping with the specific gravity used for the resource calculation.
- <u>Grade of Mineralization</u>
  The lower bound target for further exploration was based on an average grade of 1.25 g/t Au and the upper bound range on an average grade of 1.45 g/t Au. BOYD consider these grades to be realistic given the grades intercepted in reconnaissance drilling, and the average grade of the measured, indicated and inferred resource calculated by BOYD in Section 14.3.2 of this report.

24.1.2.1 Main Risks with Respect to Eastern Extension Potential Target for Further Exploration

The eastern extension potential target for further exploration is based on a number of angled exploration RC and Diamond drill holes which have intercepted multiple mineralized structures of similar style to mineralization in the north pit. Drilling, and bedrock geochemistry have defined mineralization over a strike length greater than that used in the estimation of the lower and upper bound potential exploration targets.

However, it is possible that further drilling will not confirm the continuity, width of grade of mineralization within the broader mineralized system.

24.1.3 Musseneva Potential Target for Further Exploration

The Musseneva potential target for further exploration (the eastern strike extension of north pit mineralization) is 0.2 to 1.1 Mt grading 1.25 to 1.45 g/t Au.

The reader is reminded that a potential target for further exploration is not a mineral resource estimate, is conceptual in nature, and is used where there has been insufficient exploration to define the target as a mineral resource, and where it is uncertain if further exploration will result in the target being delineated as a mineral resource.

24.1.3.1 Basis for Determination of Musseneva Target for Further Exploration

The Musseneva project is located approximately 2 km south of the north and south open pits and is defined by a broadly 080° trending zone of highly anomalous bedrock gold, arsenic and copper geochemistry (Figure 9.4), that extends over a strike length of approximately 1.2 km. Geochemistry suggests that the mineralized zone is between 200 to 300 m wide.

- Width, Length and Vertical Extent of Mineralization
  BOYD estimated the upper bound target at Musseneva on the basis that 3 mineralized zones are present—each 250 m long, 75 m in vertical extent and 10 m wide. The lower bound target was based on 3 mineralized zones—each 100 m long, 50 m in vertical extent and 5 m wide. BOYD consider this reasonable given the large size of the bedrock geochemical anomaly, and similar host rocks, style of mineralization and proximity to Laiva.
- Grade of Mineralization
  The upper bound target for further exploration was based on a grade of 1.45 g/t Au and the lower bound range on a grade of 1.25 g/t Au. BOYD consider these grades to be realistic given proximity to similar style of mineralization at Laiva. The grades fall within the range of a diamond drill hole completed by Outokumpo inn 1986—1.95 m at 1.20 g/t gold and 1 m at 1.68 g/t gold. Whilst the Outokumpo results are historic in nature, have not been verified by a qualified person, and cannot be relied upon, they provide an indication that the grades used by BOYD in establishing a potential target for further exploration are reasonable.

24.1.3.2 Main Risks with Respect to Eastern Extension Potential Target for Further Exploration

An extensive historic bedrock sampling program at Musseneva defined a large gold-arsenic-copper anomaly—a geochemical signature consistent with the mineralogy of the Laiva deposit. This anomaly indicates that Mussenenva is a valid exploration target of significant extent. However, with only one historic drill hole completed to date, the target for further exploration at Musseneva is based on a conceptual model that requires a sustained drilling campaign to evaluate. As with any conceptual model there are inherent risks—in this case BOYD cautions that drilling may not define mineralization of sufficient grade, or horizontal or vertical continuity, to define an economic resource. Moreover, the effects of post-mineralization dilution by granitic dykes and stocks, which are observed in the Laiva south pit, are unknown.

24.1.4 Oltava Potential Target for Further Exploration

The Oltava potential target for further exploration (the eastern strike extension of north pit mineralization) is 1.0 to 2.5 Mt grading 1.25 to 1.45 g/t Au.

The reader is reminded that a potential target for further exploration is not a mineral resource estimate, is conceptual in nature, and is used where there has been insufficient exploration to define the target as a mineral resource, and where it is uncertain if further exploration will result in the target being delineated as a mineral resource.

24.1.4.1 Basis for Determination of Musseneva Target for Further Exploration

The Oltava project is located approximately 10 km south of Laiva (Figure 5.1). Oltava is defined by a well-defined bedrock gold-arsenic anomaly with peripheral copper-molybdenum anomalies (Figures 9-8 to 9-11). Gold anomalism extends over a strike length of approximately 750 m and is up to 400 m wide.

GTK completed 3484 m of inclined diamond drilling between 1994 and 2002 and noted that gold mineralization is hosted in metavolcanic, mica gneiss, and quartz diorite units. Mineralization presents as east-west striking, sub-vertical, sheeted veins composed of quartz-arsenopyrite as is often the case at Laiva. However, the potential for development of mineralized shear zones comprising multiple veins is poorly understood and significantly more drilling is required.

- <u>Width, Length and Vertical Extent of Mineralization</u>
  BOYD estimated the upper bound target at Oltava on the basis that 4 mineralized zones are present—each 400 m long, 75 m in vertical extent and 10 m wide. The lower bound target was based on 3 mineralized zones—each 250 m long, 50 m in vertical extent and 5 m wide. BOYD consider this reasonable given the large size of the bedrock geochemical anomaly, and similar host rocks and style of mineralization to Laiva.

- <u>Grade of Mineralization</u>
  The upper bound target for further exploration was based on a grade of 1.5 g/t Au and the lower bound range on a grade of 1.25 g/t Au. BOYD consider these grades to be realistic given proximity to similar style of mineralization at Laiva. The grades are significantly lower than some the intercepts reported by GTK which ranged up to 1 m estimated true width at 13.0 g/t gold and 1.6 m estimated true width at 33.7 g/t Au. Whilst the GTK results are historic in nature, have not been verified by a qualified person, and cannot be relied upon, they provide an indication that the grades used by BOYD in establishing a potential target for further exploration are reasonable.

24.1.4.2 Main Risks with Respect to Oltava Potential Target for Further Exploration

The geochemical signature and style of mineralization intercepted in historic drilling is similar to the style of mineralization at Laiva. Despite almost 3500 m of historic drilling, target mineralization at Oltava is largely conceptual in nature, and will require significant additional drill testing. BOYD cautions that drilling may not define mineralization of sufficient grade, or horizontal or vertical continuity, to define an economic resource.

24.1.5 Summary of Potential Quantity and Grade of Exploration Targets

Based on the discussion and caveats presented above, the potential quantity and grade of the various exploration targets are shown below in Table 24.1.

Table 24.1
Potential Quantity and Grade of Laiva Exploration Targets

	Potential Tonnage Range (tonnes)		Potential Gold Grade Range (g/t)	
Deposit	Lower	Upper	Lower	Upper
Near Mine	3,200,000	5,100,000	1.44	1.44
Eastern	850,000	3,200,000	1.25	1.45
Musseneva	200,000	1,100,000	1.25	1.45
Oltava	1,000,000	2,500,000	1.25	1.45
Totals	5,250,000	11,900,000	1.25	1.45

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25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 Mineral Processing and Mining

The key findings of the PEA are:

- The 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.
- A detailed evaluation of several phases of metallurgical test work results and operating data from previous operations was completed to provide the basis for the estimated point forward gold recovery of 91.7%.
- The existing Laiva process facilities with the planned minor modifications will be capable of achieving the planned production rates for future operations which start at 1.5 million tonnes per year for year 1 ramping up to 2.0 million tonnes per year by year 3.
- As the process facilities are existing and detailed operating information is available from previous operations, the estimated capital and operating costs for process unit operations should be considered to be more accurate than a typical PEA.

26.0 RECOMMENDATIONS

26.1 Process

- Utilize results from ongoing metallurgical test program to develop detailed commissioning plans for all process unit operations.
- Develop metallurgical balance with adequate detail for all process related planning and reporting requirements.
- Plan detailed sampling and analyses program for initial operating period to provide results required to optimize split between low-grade and high-grade leaching circuits.
- Plan and execute lab scale testing of loaded and stripped carbon samples to evaluate potential benefits of warm acid rinse and cold cyanide carbon strip operations.

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28.0 CERTIFICATE OF AUTHOR QUALIFICATIONS

Chris C. Wilson

As a co-author of this Technical Report entitled “NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LAIVA GOLD MINE PROJECT”, Raahe Finland dated 3 July 2018 with an effective date of 6 April 2018, I Chris C. Wilson do hereby certify that:

1. I am engaged as an Associate Senior Geologist 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 c-wilson@jtboyd.com.
2. This certificate applies to the aforementioned Technical Report.
3. I graduated from the University of Aberystwyth with an honours degree in Geology in 1988 and from the Flinders University of South Australia with a PhD in Geology in 1994. I have practised my profession continuously since that time. This has included over 25 years of relevant experience in grass-roots exploration and advanced project management of gold and silver mineralized systems, including mesothermal vein type deposits.
4. I am a Chartered Professional Geologist and Fellow of the Australasian Institute of Mining and Metallurgy (No. 112316) and a Fellow of the Society of Economic Geologists (No. 868275)
5. I have worked, or carried out research, as a geologist continuously for over 25 years since my graduation from university.
6. I am familiar with NI 43-101 and by reason of education, experience with similar projects, and professional registration, I full fill the requirements as a Qualified Person as defined in NI 43 101.
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 visited the Laiva Gold Project site from 3 January – 23 January 2018 and again from 10 February – 26 March 2018.
9. I have no prior association with the Laiva Project.
10. I am independent of Firesteel Resources Inc. and its subsidiaries. I hold no beneficial interest in the foregoing.
11. I have collaborated with my co-authors on all relevant sections 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 3rd day of July 2018 with an effective date of 6 April 2018.

Dated 3 July 2018
(Signed and Sealed)

Chris C. Wilson, MGEOL, MAusIMM (CP)
Associate Senior Geologist - Metals

Sam J. Shoemaker, Jr.

As a co-author of this Technical Report entitled "NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LAIVA GOLD MINE PROJECT", Raahe Finland dated 3 July 2018 with an effective date of 6 April 2018, I Sam J. Shoemaker, Jr., 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 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 35 years.
6. I am familiar with NI 43-101 and by reason of education, experience with similar projects, and professional registration, I fulfill the requirements as a Qualified Person as defined in NI 43-101.
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. I was a co-author on the August 2017, mineral resource estimate NI43-101 Technical Report.
10. I am independent of Firesteel Resources Inc. and its subsidiaries. I hold no beneficial interest in the foregoing.
11. I am the principal author of section 14 and 16 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 3rd day of July 2018 with an effective date of 6 April 2018.

Dated 3 July 2018
(Signed and Sealed)

Sam J. Shoemaker, Jr., B.S., SME Registered Member
Project Manager - Metals

Gregory B. Sparks

As a co-author of this Technical Report entitled “NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LAIVA GOLD MINE PROJECT”, Raahe Finland dated 3 July 2018 with an effective date of 6 April 2018, 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 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 Member #03042460.
   - CIM – Member # 97865.
5. I have worked as a mining engineer in the minerals industry for 44 years
6. I am familiar with NI 43-101 and by reason of education, experience with similar projects, and professional registration, I fulfill the requirements as a Qualified Person as defined in NI 43-101.
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 visited the Laiva Gold Project site from 15–16 March 2017 and again from 27 July to 4 August 2017.
9. I was a co-author on the August 2017, mineral resource estimate NI43-101 Technical Report.
10. I am independent of Firesteel Resources Inc. and its subsidiaries. I hold no beneficial interest in the foregoing.
11. I have collaborated with my co-authors on all relevant sections 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 3rd day of July 2018 with an effective date of 6 April 2018.

Dated 3 July 2018
(Signed and Sealed)

Gregory B. Sparks

Gregory B. Sparks, B.Sc., Professional Engineer
Managing Director - Metals

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29.0 DATE AND SIGNATURE PAGE

JOHN T. BOYD COMPANY
By:

CHARTERED PROFESSIONAL
AusIMM
AUSTRALASIAN INSTITUTE OF MINING & METALLURGY
• GEOLOGY •
CHRISTOPHER C WILSON

Chris C. Wilson, MAusIMM (CP)
Qualified Person
Associate Senior Geologist

Report Date: 3 July 2018
Effective Date: 6 April 2018

Sam Shoemaker, SME Registered Member
Qualified Person
Project Manager

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Gregory B. Sparks, P. Eng.
Qualified Person
Managing Director – Metals

COLORADO REGISTERED PROFESSIONAL ENGINEER
GREGORY B. SPARKS
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